NI 43-101 Technical Report

Pre-feasibility Study

Brigus Gold Corp.

Goldfields Project, Saskatchewan, Canada

Prepared By

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In Cooperation with
Wardrop, DMA and EHA

Effective Date: October 6, 2011

Endorsed by Qualified Persons:

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Dan A. Mackie, P.Eng.
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Tim Maunula, P.Geo.
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1 SUMMARY

1.1 INTRODUCTION

Brigus Gold Corp. (Brigus) is a Canadian registered mining company, based in Halifax, Nova Scotia, and publicly listed on both the Toronto and New York stock exchanges. Brigus is a mid-tier mining company currently operating one gold mine and focusing on the development of other historical and newly delineated gold deposits.

This report provides technical information and a resource estimate update for the historical Box Mine (Box). The Box mine lies within the Goldfields Property (Goldfields) which also includes the Athona Deposit (Athona). Goldfields is located in northern Saskatchewan, approximately 1,000 km north of Regina, the provincial capital, and 13 km south of Uranium City. Brigus holds a 100% interest in Goldfields.

Brigus retained Wardrop, a Tetra Tech Company (Wardrop), to complete an updated resource estimate for Box. Wardrop was previously involved with Goldfields in 2006-2007 for the previous mineral rights holders, Greater Lenora Resources Inc. No additional exploration has been conducted on Athona since the latest National Instrument (NI) 43-101 resource estimate was issued by Wardrop in 2007 (Wardrop, 2007). The Athona resource data has been included in this updated estimate for information.

The following technical report conforms to the standards set out in National Instrument 43-101 (NI 43-101), Standards and Disclosure for Mineral Projects and is in compliance with Form 43-101F1 technical report.

The Qualified Person responsible for the Box resource estimate update is Paul Daigle, P. Geo., Senior Geologist with Wardrop. The site visit was conducted by Mr. Daigle on May 11 and 12, 2011. The Qualified Person responsible for the Athona resource estimate is Tim Maunula, P.Geo. with Wardrop.

Brigus retained March Consulting Associates Inc. (March Consulting) to complete a pre-feasibility study (PFS) for the purposes of developing a reserve estimate, capital cost estimate, operating cost estimate, and economic analysis for Goldfields.

The Qualified Person responsible for the mineral reserve estimate and mining methods is Cliff Lusby, P. Eng., Principal Mine Engineer Associate with March Consulting. The Qualified Person responsible for the capital cost estimate, operating cost estimate and economic analysis is Kyle Krushelniski, P. Eng., Senior Project Manager with March Consulting. Site visits were conducted by Mr. Lusby and Mr. Krushelniski on March 18th, 2010 and September 10th to 13th, 2010.

Brigus retained Dan Mackie Associates (DMA) and EHA Engineering Ltd. (EHA) for the development of the ore processing and process plant design. The qualified persons responsible for the process design are Al Hayden, P. Eng. and Dan Mackie, P. Eng.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Goldfields Claim Group is defined by the mineral rights to five mineral leases and 31 mineral claims, currently 100% held by 7153945 Canada Inc. a wholly owned subsidiary of Brigus. The Goldfields Claim Group covers a total area of 25,685 ha.

The Property is located:

- Within National Topographic System (NTS) 1:50,000 map sheets 74N07
- At approximately 59° 27' N latitude and 108° 31' W longitude, in northern Saskatchewan, Canada
- Approximately 1,000 km north of Regina, the provincial capital of Saskatchewanand approximately 850 km north of Saskatoon, Saskatchewan (SK)
- Approximately 13 km south of Uranium City, SK
- Approximately 25 km by road from Uranium City on Local Highway 962
- Approximately 60 km south of the border with the Northwest Territories, Canada
- In Northern Saskatchewan Administration District
- In Census Division No. 18 – La Ronge, SK
- The proposed Box mine is bounded to the south by the north shore of Neiman Bay in Lake Athabasca and to the north by the south shore of Vic Lake

Geologically, northern Saskatchewan is predominantly underlain by variably deformed and metamorphosed rocks of Archean age (3070 to 3014 Ma) to Helikian (1450 to 1350 Ma) age. In the northwest, the Archean to Aphebian crystalline basement, influenced by Lower Proterozoic thermotectonic events, is overlain by redbeds of the Martin Group (and immediately underlying Thluicho Lake and Ellis Bay Groups) which were probably deposited during and immediately following the main Hudsonian event (ca. 1900 to 1800 Ma). Immediately to the south, the metamorphic basement rocks are overlain by post-metamorphic sedimentary rocks of the Helikian Athabasca Group. Post-Hudsonian diabase dykes (ca. 1400 to 1100 Ma) are the youngest rocks in the Precambrian of northern Saskatchewan (Jensen, 2003).

1.3 GEOLOGY AND MINERALIZATION

1.3.1 BOX MINE

The geological setting at Box consists of a sequence of metasedimentary lithological units. The footwall sequence is represented by several series of alternating units of amphibolite and quartzite. These units exist from north of the Frontier Mine, an abandoned site, to the Box FW for approximately 1,000 m horizontally. At the footwall contact, a zone of metasediments consists of almost pure quartzite, feldspathic arkose, medium to coarse grained greywacke and sub-angular to rounded pebble conglomerates. Scattered along the footwall at irregular intervals are amphiboles, intrusive sills and/or homfelsed metasediments with some units exhibiting varying degrees of shearing which forms chlorite and hornblende schists.

The Box Mine Granite (BMG) unit is a depositional sequence of metasedimentary lithologies grading towards the southeast from a pebble to cobble size conglomerate to a coarse grained, then medium grained, greywacke, followed by feldspathic arkose. Due to the varying intensity of granitization or feldspathization and silicification of the clastic metasediments, it is difficult to determine if more than one sequence exists. The BMG has been moderately to intensely altered by hematitization, which indicates the contacts of the auriferous zone. The contacts vary from gradational to sharp.

BMG has a surface expression in excess of 750 m and an average width of 40 m with the central portion in excess of 60 m.

Gold mineralization is associated with fine grained pyrite in the range of 0.5% to 3.0% in the wallrock and quartz- carbonate veins. Some of the auriferous quartz veins trend N10°E and have associated sulphide mineralization in order of abundance as pyrite, galena, sphalerite, and chalcopyrite.

1.3.2 ATHONA DEPOSIT

The gold bearing zones at Athona are from west to east: the eastern portion of the West Mine Granite, the Athona Granite, the Pond Zone, and in a prominent en echelon and bouginage quartz vein system of the East Zone. The historic underground mine development was concentrated in the western portion of the Athona Mine Granite(AMG) and the eastern quartz vein systems (H, I, J, K veins) on the 125 and 250 foot levels. The Athona West Granite (AWG) is a medium to coarse grained, reddish hematitic altered granite, dipping moderately westwards, containing fracture filling, quartz veining within the footwall sheared contact or mylonite zone. The unit is underlain by the central gabbroic to amphibolitic intrusive which separates the AWG from the AMG.

1.4 EXPLORATION

Brigus initiated a site investigation program in 2010 for Goldfields. The program included the following:

- Performed a DC/Induced Polarization (IP) geophysical survey for identification of anomalies.
- Completed a piezometer installation program to characterize the shallow geologic and hydrogeologic conditions surrounding the Waste Rock Storage Area (WRSA) and provide hydrogeologic information surrounding the Vic Lake Tailings Management Facility (TMF).
- Completed the drilling program as summarized in Table 1-1:

The previous resource model was based on the historical drill data presented in Table 1-2. The resource model was updated with the data from the 2010 drill program.

Table 1-1: 2010 Exploration Drilling

Year(s)	Company	Type[a]	Location	Number of Holes	Drill Core Size[b]	Length (m)	Number of Samples
2010	Brigus	DDH	Surface (Box)	12	NQ	2825.4	562
2010	Brigus	DDH	Surface (Athona)	2	NQ	646.0	243
2010	Brigus	DDH	Surface (Piezometer)	19	NQ	497	44

Note:
a: DDH - diamond drill hole
b: EX, BQ, NQ – drill sizes

Brigus is currently implementing the 2011 program for the Goldfields to further update the resource estimates. The 2011 program has a multi-purpose approach and includes the following:

- Drilling to upgrade inferred resources and identify additional resources. This includes four holes that will be drilled using HQ sized core to be split for metallurgical testing and assaying.
- Drilling four geotechnical HQ holes that will be surveyed using an acoustic geophysical probe.
- Installation of piezometer wells at the TMF and WRSA to characterize the shallow geologic and hydrogeologic conditions.

Table 1-2: Historical Box Drill Hole Summary

Year(s)	Company[a]	Type[b]	Location	Number of Holes	Drill Core Size[c]	Length (m)	Number of Samples
1935-39	Cominco	channel	underground	32		6,548.65	4,385
1935-39	Cominco	DDH	surface	42	EX	4,576.12	1,708
1939	Cominco	DDH	underground	72	EX	4,594.98	2,959
1987-88	GLR	DDH	surface	52	BQ	6,383.73	2,628
1989	GLR	RCD	surface	47		3,168.60	2,715
1994	GLR	DDH	surface	52	BQ	6,705.77	2,443
1995	GLR	DDH	surface	100	BQ	18,825.00	3,469
2004	GLR	DDH	surface	15	NQ	1,007.67	577
2005	GLR	DDH	surface	22	NQ	3,299.15	782
2007	GLR	DDH	surface	13	-	3348.60	-
Totals:				434		58,458.27	21,611

Note:
a: GLR – Greater Lenora Resources
b: RCD – reverse circulation drill-holes
c: EX, BQ, NQ – drill sizes

1.5 MINERAL RESOURCE ESTIMATES

The resulting mineral resource estimates for Box from the Ordinary Kriging (OK) interpolation method, at 0.5 gram/tonne (g/t) Au cut-off grade (COG) are:

- Measured Resources of 858,000 tonnes at 2.05 g/t Au
- Indicated Resources of 12,966,000 tonnes at 1.63 g/t Au
- Inferred Resources of 3,158,000 tonnes at 1.74 g/t Au.

The OK resource estimates for Box were estimated for a range of gold Cut-off Grades (COGs) from 0.125 g/t Au to 4.0 g/t Au. Table 1-3 presents the resources for COGs between 0.25 and 2.0 g/t. The shaded line in the table indicates the relevant resource information at the COGs of 0.5 Au g/t. No recoveries have been applied to the interpolated estimates.

Table 1-4 shows Indicated and Inferred Resources for Athona. The base case is reported for a COG of 0.5 g/t Au.

Table 1-3: Resource Estimate Table for Box

Gold COG (g/t)	Measured			Indicated			Measured + Indicated			Inferred		
	Tonnes (x000 t)	Au (g/t)	Au oz (x1000 oz)	Tonnes (x1000 t)	Au (g/t)	Au oz (x1000 oz)	Tonnes (x1000 t)	Au (g/t)	Au oz (x1000 oz)	Tonnes (x1000 t)	Au (g/t)	Au oz (x1000 oz)
2.0	266	4.13	35	3,291	3.36	356	3,556	3.42	391	877	3.61	102
1.5	383	3.39	42	4,968	2.81	449	5,351	2.85	491	1,227	3.07	121
1.0	585	2.65	50	7,785	2.24	561	8,371	2.27	611	1,881	2.43	147
0.5	**858**	**2.04**	**56**	**12,966**	**1.63**	**681**	**13,824**	**1.66**	**737**	**3,158**	**1.74**	**176**
0.375	939	1.90	57	14,945	1.48	709	15,884	1.50	766	3,636	1.57	183
0.25	1,012	1.79	58	16,952	1.34	729	17,964	1.36	787	4,169	1.41	188

Table 1-4: Athona Indicated and Inferred Resources

Gold COG (g/t)	Measured			Indicated			Measured + Indicated			Inferred		
	Tonnes (x000 t)	Au (g/t)	Au oz (x1000 oz)	Tonnes (x1000 t)	Au (g/t)	Au oz (x1000 oz)	Tonnes (x1000 t)	Au (g/t)	Au oz (x1000 oz)	Tonnes (x1000 t)	Au (g/t)	Au oz (x1000 oz)
3.0	-	-	-	371.4	4.08	48.7	371.4	4.08	48.7	50.3	4.45	7.2
2.5	-	-	-	1,033.2	3.00	99.6	1,033.2	3.00	99.6	88.8	3.46	9.9
2.0	-	-	-	1,870.7	2.43	146.2	1,870.7	2.43	146.2	213.7	2.44	16.8
1.0	-	-	-	3,399.8	1.89	206.6	3,399.8	1.89	206.6	558.6	1.69	30.4
0.5	**-**	**-**	**-**	**7,036.4**	**1.28**	**289.6**	**7,036.4**	**1.28**	**289.6**	**1,406.4**	**1.10**	**49.7**

1.6 MINERAL RESERVE ESTIMATES

Table 1-5 shows the proven and probable reserves for Box and Athona. Goldfields has 22,333,045 tonnes of ore at an average grade of 1.420 g/t Au with 1,020,000 ounces of gold. Total waste generated is 81,651,910 tonnes for a life of mine (LOM) strip ratio of 4.56 at Box and 1.10 at Athona.

The mineable reserves were based on a COG of 0.72 g/t. The low grade (LG) ore is characterized as the ore below the COG of 0.72 g/t but above the marginal COG of 0.33 g/t. The marginal COG is the ore grade that allows for reasonable prospects of economic extraction. The LG ore will be used for filling existing stopes and stockpiled for future processing.

The mill feed is 1,825,000 tonnes per year (t/y) for a 5,000 tonnes per day (t/d) average. The LOM annual average mineable gold production is 82,156 oz/year, which includes processing the LG stockpile at the end of operations. During the active mine stages for Box and Athona the average gold production is 110,373 oz/year (process recovery has not been applied to the gold production numbers).

Table 1-5: Summary of Mineable Reserves for Box and Athona

Description	Ore (t)	Grade (g/t)	Gold (oz)	Waste (t)
Box				
Proven + Probable	16,502,247	1.508	800,000	75,228,132
Athona				
Proven + Probable	5,830,798	1.172	220,000	6,423,778
Total				
Proven + Probable	22,333,045	1.420	1,020,000	81,651,910

1.7 MINING

The mine pit design was completed based on the resource models from Wardrop and the pit slope angles from the Klohn Crippen 1995 preliminary report. A hanging wall (HW) angle of 55 degrees was used for both Box and Athona design. Klohn Crippen Berger Ltd. is currently updating the rock mechanics data for the project. This data will be used to optimize the pit design.

The mining equipment was selected based on the production schedule. It was determined that nine 90 tonne (100 ton) haul trucks will be required for initial operation with a peak of 10 trucks required starting in Year 4 of operations. Two 13 m^3 hydraulic shovels and one 11.5 m^3 wheel loader are required to maintain production rates. Two types of blast-hole drills were selected to provide for a wide variability of conditions.

1.8 MILL PROCESSING

The mill process plant is designed based on a traditional crushing, grinding, flotation, cyanidation and Merrill Crowe circuit. The rated capacity of the plant is 5,000 tpd of ore. The average grade from Box in the first seven years of operations is 1.97 g/t. The average grade will be reduced once the Athona ore and LG stockpiled ore are processed. The LOM average ore grade is 1.42 g/t. The plant is designed to operate 365 days per year with total annual availability of 94%. The target annual throughput is 1,825,000 tonne per year with an overall gold recovery of 91% for Box ore and 89% for Athona ore. The estimated project life is 13 years to process the ore from Box and Athona. This includes processing the LG stockpile after active mining has been completed.

The mill facility will be located in a natural valley northeast of Vic Lake. Site drainage from all mill facilities will report to Vic Lake. To minimize site preparation costs and to take advantage of the natural terrain, the mill facility is conceptually designed as three independent complexes including crushing, crushed ore storage, and grinding and leaching. Separating the mill facility into three complexes minimizes the building foundation fill requirements.

1.9 CAPITAL COST ESTIMATE

The capital cost includes mining equipment lease payments for six months, the project residence and office facility lease payments for one year. The remainders of the lease payments were accounted for during the operations period. All the dollars associated with the cost estimate and economic analysis are Canadian dollars unless specified otherwise. Table 1-6 summarizes the total capital cost at $159,235,000.

Table 1-6: Summary of Capital Cost

	Description	Total Capital ($000s)
Directs	Infrastructure	$44,535
	Mine	$12,956
	Mill	$44,838
	Subtotal	$102,329
Indirects	Construction Indirects	$27,379
	Freight Indirects	$5,249
	Owners Costs	$5,119
	Subtotal	$37,747
Contingency	Contingency	$19,159
	Subtotal	$19,159
	Total Capital Cost	**$159,235**

1.10 OPERATING COST ESTIMATE

The operating costs were established for Box. Box costs were adapted for Athona and the LG stockpile processing but adjusted to reflect changes in operating conditions due to reduced mining manpower requirements. Table 1-7 provides a summary of the project operating costs.

The operating cost for Box was estimated as follows:

- Milling
    - $10.70/t milled
- Mining
    - $14.47/t milled
    - $2.60/t mined
- General and administrative (G&A)
    - $4.99/t milled

The total operating cost for Box was $30.17/t milled. The mining costs were calculated using the LOM strip ratio for Box of 4.56.

Milling costs as calculated for Box were applied throughout the mill operation as they are not anticipated to vary significantly with different ore feeds. G&A costs for Athona are reduced due to the reduction of mining manpower, resulting in reduced support costs. The mining costs for Athona differed from Box due to longer cycle times, lower equipment requirements, and a lower strip ratio.

The operating cost for Athona was estimated as follows:

- Milling
  - $10.70/t milled
- Mining
  - $4.15/t milled
  - $1.97/t mined
- G&A
  - $4.70/t milled

The total operating cost for Athona was estimated at $19.55/t milled. The mining costs were calculated using the LOM strip ratio for Athona of 1.10.

Recovery of the LG stockpiled ore is scheduled to occur at the conclusion of mining of Athona. Operating costs were determined by reducing the equipment fleet and eliminating the costs associated with drilling and blasting.

- Stockpile recovery
  - $0.73/t milled
- Milling
  - $10.70/t milled
- G&A
  - $3.94/t milled

Total operating cost during the LG stockpile recovery period was estimated at $15.37/t milled.

Also included in the operating costs are the lease costs for the residence facility and mining equipment. The capital savings associated with leasing equipment are applied into the operations period affecting the total operating cost. Table 1-7 shows the increase in costs for Box and Athona. The total operating costs including equipment lease payments for Box are $34.24/t milled, while Athona increases to $19.96/t milled.

Table 1-7: Summary of Operating Costs for Box, Athona, and the LG Stockpile Recovery

Description	BOX		ATHONA		LG STOCKPILE RECOVERY	
	$/t milled	$/t mined	$/t milled	$/t mined	$/t milled	$/t mined
Mine	$14.47	$2.60	$4.15	$1.97	$0.73	$0.73
Mill	$10.70	-	$10.70	-	$10.70	-
G&A	$4.99	-	$4.70	-	$3.94	-
Total Operating Cost - Purchase	$30.17	-	$19.55	-	$15.37	-
Equipment / Facility Lease Costs	$4.07		$0.41			
Total Operating Cost - Lease	$34.24		$19.96		$15.51	

1.11 ECONOMIC ANALYSIS

An economic analysis was conducted to determine the net present value (NPV), internal rate of return (IRR), payback period, and cash cost per ounce. The analysis was completed for both the purchase option and the lease option. For the economic analysis, an average gold price of $1,250/troy oz was used. The economic indicators are presented in Table 1-8. The NPV at a 5% discount rate was $144,308,000 with an IRR of 19.6%. The cash cost per ounce of gold was $601. The payback period was five years.

Table 1-8: Summary of Economic Indicators

Variable	Location	Values
NPV @ 5%	**Project**	**$144,308,000**
	Box	$80,110,000
	Athona	$64,197,000
IRR	**Project**	**19.6%**
	Box	15.5%
	Athona	151.1%
Cash Cost ($/oz)	**Project**	**$601**
	Box	$605
	Athona	$585
Total Cost ($/oz)	**Project**	**$940**

A sensitivity analysis demonstrated that the project economics were most sensitive to the process recovery followed closely by the gold price. Operating cost and capital cost were less sensitive. Figure 1-1 shows the relative sensitivity of the variables. The steepest line is the most sensitive variable.

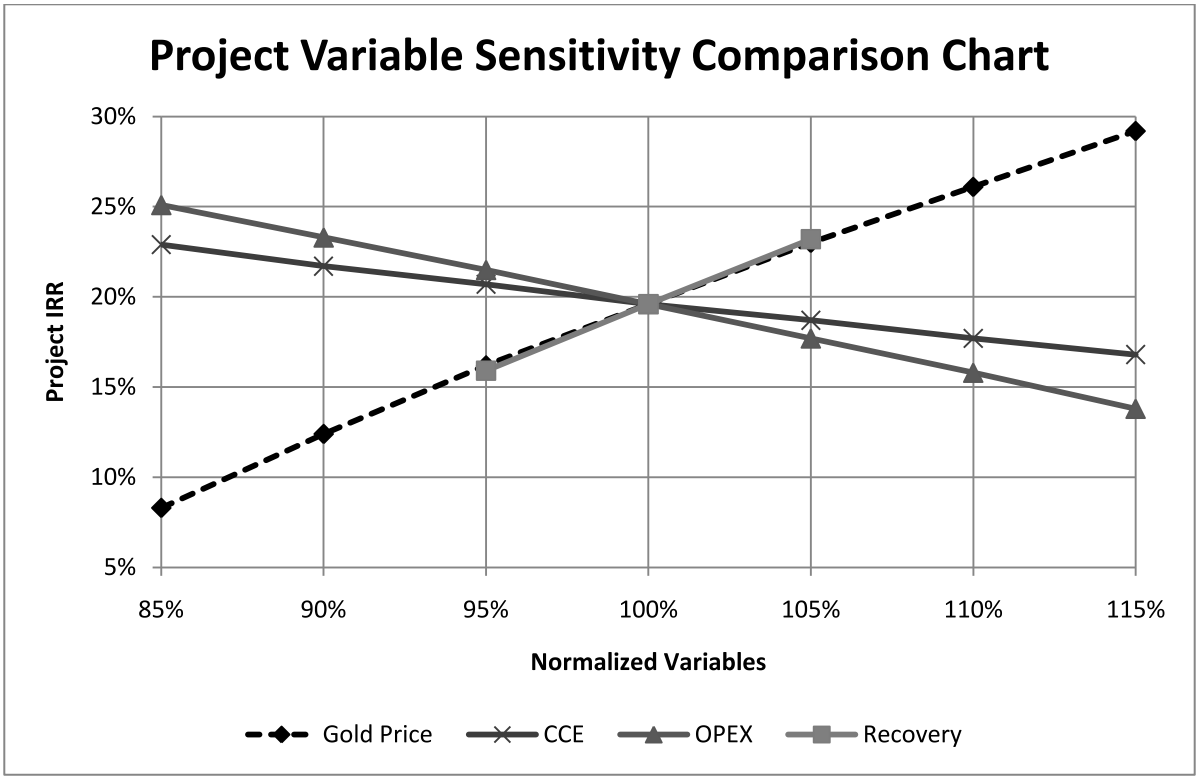

Figure 1-1: Plot of Sensitivity Analysis for Project Variables

1.12 ENVIRONMENTAL AND PERMITTING

The Goldfields Project is under the jurisdiction of both federal and provincial regulatory agencies. The Saskatchewan Ministry of Environment (MOE) regulates the operation of this project under the terms of "The Environmental Assessment Act". Due to the project's possible impact on aquatic habitat, the Department of Fisheries and Oceans (DFO), under the terms of the Aquatic Habitat Protection Permit, also maintains jurisdiction, as a regulatory agency. The water quality parameters, as defined by the "Metal Mining Effluent Regulations" (MMER) of the Federal Fisheries Act, and provincially, the Saskatchewan Surface Water Quality Objectives (SSWQO), define the discharge parameters for effluents generated by the future operations of Goldfields.

The MOE is designated as the Coordinating Regulatory Agency under the terms of Canada-Saskatchewan Agreement on Environmental Assessment Cooperation. In this role as Coordinator, the MOE has, in consultation with appropriate regulatory agencies, assessed potential impacts on the environment from the project as contained in the Environmental Impact Statement (EIS)(2008) and issued a ministerial approval dated May 28, 2008, for the project to proceed.

1.13 CONCLUSIONS

The PFS for the Goldfields Project has been completed and includes an estimate of the capital costs, operating costs, gold resources and reserves for the purposes of developing an economic model for the Goldfields Project. The results of the PFS are presented in this report.

1.14 RECOMMENDATIONS

Based on the results of the PFS the following opportunities were identified for the project:

- Continue exploration drilling in relevant areas of both deposits to enhance the resource estimate
- Conduct project specific process test work and optimize process recovery
- Complete the geotechnical assessment and update the ore reserve models to reflect the potential revised pit slopes
- Advance the project planning and design to minimize potential execution risks

2 INTRODUCTION AND TERMS OF REFERENCES

2.1 GENERAL

Brigus is a Canadian registered mining company, based in Halifax, Nova Scotia, and publicly listed on both the Toronto and New York AMEX stock exchanges. Brigus is a mid-tier mining company currently operating one gold mine and focusing on the development of other historically and newly delineated gold deposits.

This report provides technical information and a resource estimate update for the historical Box mine. The Box mine lies within Goldfields Property which also includes Athona. Brigus holds a 100% interest in Goldfields.

Goldfields is located in northern Saskatchewan, approximately 1,000 km north of Regina, the provincial capital, and 13 km south of Uranium City, SK. Uranium City is an isolated community and is accessible by regular scheduled flights from Saskatoon through Prince Albert, Points North, Stony Rapids and Fond-du-Lac.

2.2 PURPOSE AND TERMS OF REFERENCE

In order to make a production decision regarding Goldfields, Brigus requested that March Consulting Associates Inc. of Saskatoon, Saskatchewan an independent engineering consulting firm, provide capital and operating cost estimates to develop and operate the gold mine. A pre-feasibility economic analysis of the project was completed by March Consulting based on estimated capital and operating expenditures for the proposed mine operation. At the request of Brigus, March Consulting, with the participation of other specialized consultants, prepared this NI 43-101 technical report.

Brigus retained Wardropto complete an updated resource estimate for Box. Wardrop was involved with the property in 2006 and 2007 for the previous mineral rights holders, GLR Resources Inc.(GLR).

Athona is included in this report for completeness, although no additional work has been completed since the latest NI 43-101 technical report was issued in 2007 (Wardrop, 2007).

The purpose of this report is to present the updated results of the PFS for Goldfields and the findings of the economic evaluation. The PFS is intended for Brigus to further develop and advance Goldfields to production.

This Technical Report was prepared according to the guidelines set out in the NI 43-101 Standards of Disclosure for Mineral Projects and complies with Form 43-101F Technical Report effective June

30, 2011. March Consulting and Wardrop prepared this report to support the public disclosure of the mineral resource and reserve estimates as of May 31, 2011.

2.3 PROJECT TEAM, RESPONSIBILITIES AND PERSONAL INSPECTION

The Qualified Person responsible for the Box resource estimate is Paul Daigle, P.Geo., Senior Geologist with Wardrop. A site visit was conducted by Mr. Daigle on May 11 and 12, 2011. Mr. Daigle was accompanied by Mr. John Dixon, Exploration Manager and Mark McLaren, Project Geologist with Brigus, and Calvin Andreas, Civil Engineer-in-Training with March Consulting. The Qualified Person responsible for the Athona resource estimate is Tim Maunula, P.Geo. with Wardrop.

The Qualified Person responsible for the mineral reserve estimate and mining methods is Cliff Lusby, P. Eng., Principal Mine Engineer Associate with March Consulting. The Qualified Person responsible for the capital cost estimate, operating cost estimate and economic analysis is Kyle Krushelniski, P. Eng., Senior Project Manager with March Consulting. Site visits were conducted by Mr. Lusby and Mr. Krushelniski on March 18th, 2010 and September 10th to 13th, 2010.

Table 2-1 provides a detailed list of Qualified Persons who are responsible for this report. Table 2-2 shows the report sections that each Qualified Person is responsible for. The certificates are included on the Date and Signature Page.

Table 2-1: Qualified Persons

QPs	Designation	Company	Initials
Kyle Krushelniski	P. Eng.	March Consulting	KK
Cliff Lusby	P. Eng.	March Consulting	CL
Tim Maunula	P. Geo.	Wardrop	TM
Paul Daigle	P. Geo.	Wardrop	PD
Dan Mackie	P. Eng.	DMA	DM
Al Hayden	P. Eng.	EHA	AH

Table 2-2: Report Sections of Responsibility

Section	Title of Section	QP
1.0	Summary	KK
2.0	Introduction and Terms of Reference	KK
3.0	Reliance on Other Experts	KK
4.0	Property Description and Location	KK
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography	KK
6.0	History	
6.1 - 6.8	N/A	TM, PD
6.9 – 6.10	Ownership History	KK
7.0	Geological Setting	TM, PD
8.0	Deposit Types	TM, PD
9.0	Exploration	TM, PD
10.0	Drilling	TM, PD
11.0	Sample Preparation, Analyses and Security	TM, PD
12.0	Data Verification	TM, PD
13.0	Mineral Processing and Metallurgical Testing	DM, AH
14.0	Mineral Resource Estimates	TM, PD
15.0	Mineral Reserve Estimates	CL
16.0	Mining Methods	CL
17.0	Recovery Methods	DM, AH
18.0	Project Infrastructure	KK
19.0	Market Studies and Contracts	KK
20.0	Environmental Studies, Permitting, and Social or Community Impact	KK
21.0	Capital and Operating Costs	KK
22.0	Economic Analysis	KK
23.0	Adjacent Properties	KK
24.0	Other Relevant Data and Information	KK
25.0	Interpretation and Conclusions	KK
26.0	Recommendations	KK
27.0	References	KK, TM, PD
28.0	Illustrations	KK

2.4 SOURCE OF INFORMATION

The information presented in this technical report has been derived from the following sources:

1. Feasibility study report titled "Box Mine – Goldfields Project, Uranium City, SK, Canada" for Linear Gold Corporation (Linear), completed in September, 2009;
2. Feasibility study report titled "Greater Lenora Goldfields Project" for Greater Lenora Resources Corporation, completed in December 1996;
3. EIS 2007; and
4. Various studies and fieldwork done by Brigus and its consultants for the development of Goldfields.

3 RELIANCE ON OTHER EXPERTS

3.1 DISCLAIMER

It should be understood that the mineral resources and reserves presented in this technical report are estimates of the size and grade of the deposits based on a certain number of drill holes, samples, assumptions, and parameters available at the time of preparing this report. The level of confidence in the estimates depends upon a number of uncertainties. These uncertainties include, but are not limited to:

- Future changes in metal prices and/or production costs
- Differences in size and grade and recovery rates from those expected
- Changes in project parameters

In addition, there is no assurance that the project implementation will be realized.

The comments in this technical report reflect the best judgment of March Consulting in light of the information available at the time of preparation. March Consulting reserves the right, but is not obligated, to revise this technical report and conclusions if additional information becomes known to March Consulting subsequent to the date of this technical report.

3.2 RELIANCE ON OTHER EXPERTS

The authors are relying on reports, opinions, and statements from experts who are not Qualified Persons for information concerning legal, environmental, political, or other issues and factors relevant to the technical report.

The authors are relying on the following reports provided by Brigus:

AMEC, 2005f, Box Mine Project: Resource Estimate for Greater Lenora Resources. April 2005. 51 pages.

Jensen, K.A., 2003. Technical Report on the Goldfields Property for GLR Resources Inc., in the Beaverlodge Lake Area, NTS Map Sheets 74N-06, 74N-07, 74N-08, 74N-09, and 74N-10, Northern Mining District, Saskatchewan, Canada. 12 December 2003. 167 pages.

Bikerman, Box Mine-Goldfields Project, Uranium City, SK, Canada. September, 2009. 303 pages.

Information from third party sources is referenced under Section 27.0: References. The authors used information from these sources under the assumption that the information is accurate.

The authors compiled the information contained in this technical report from these sources. Review and verification beyond the PFS have not been made.

March Consulting has exercised reasonable diligence in using data supplied by Brigus, EHA, DMA, and other project participants and has no reason to believe that any data supplied are misleading or incorrect. However, March Consulting does not guarantee the accuracy of data supplied.

Specific items that March Consulting has accepted include:

1. Geological setting, deposit types, resource models, drill hole sampling and verification - March Consulting has relied upon the recent work performed by Wardrop.

2. Mineral processing and metallurgical testing and recovery methods - March Consulting has relied on the process design by DMA and EHA.

3. Milling operating costs – March Consulting has relied on the quantities and unit costs of the reagents and supplies developed by EHA.

4. Taxes – March Consulting did not include taxes in the preparation of the cash flows; tax is not included in the economic analysis.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION AND AREA

The Property is situated as shown in Figure 4-1 and Figure 4-2 in the northwest corner of Saskatchewan.

The property is located:

- within NTS 1:50,000 map sheets 74N07
- at approximately 59° 27' N latitude and 108° 31' W longitude, Zone 12, NAD83 Datum in northern Saskatchewan, Canada
- approximately 1,000 km north of Regina, the provincial capital of Saskatchewan and approximately 850 km north of Saskatoon, SK
- approximately 13 km south of Uranium City, SK
- approximately 25 km by road from Uranium City, SK on Local Highway 962
- Box is bounded to the southeast by Neiman Bay, Lake Athabasca and to the northwest by Vic Lake
- approximately 60 km south of the border with the Northwest Territories
- in the Northern Saskatchewan Administration District
- in the Census Division No. 18 – La Ronge, SK

The Goldfields Claim Group is defined by the mineral rights to five mineral leases and 31 mineral claims, currently 100% held by 7153945 Canada Inc., a wholly owned subsidiary of Brigus, and covering a total area of 25,685 ha.

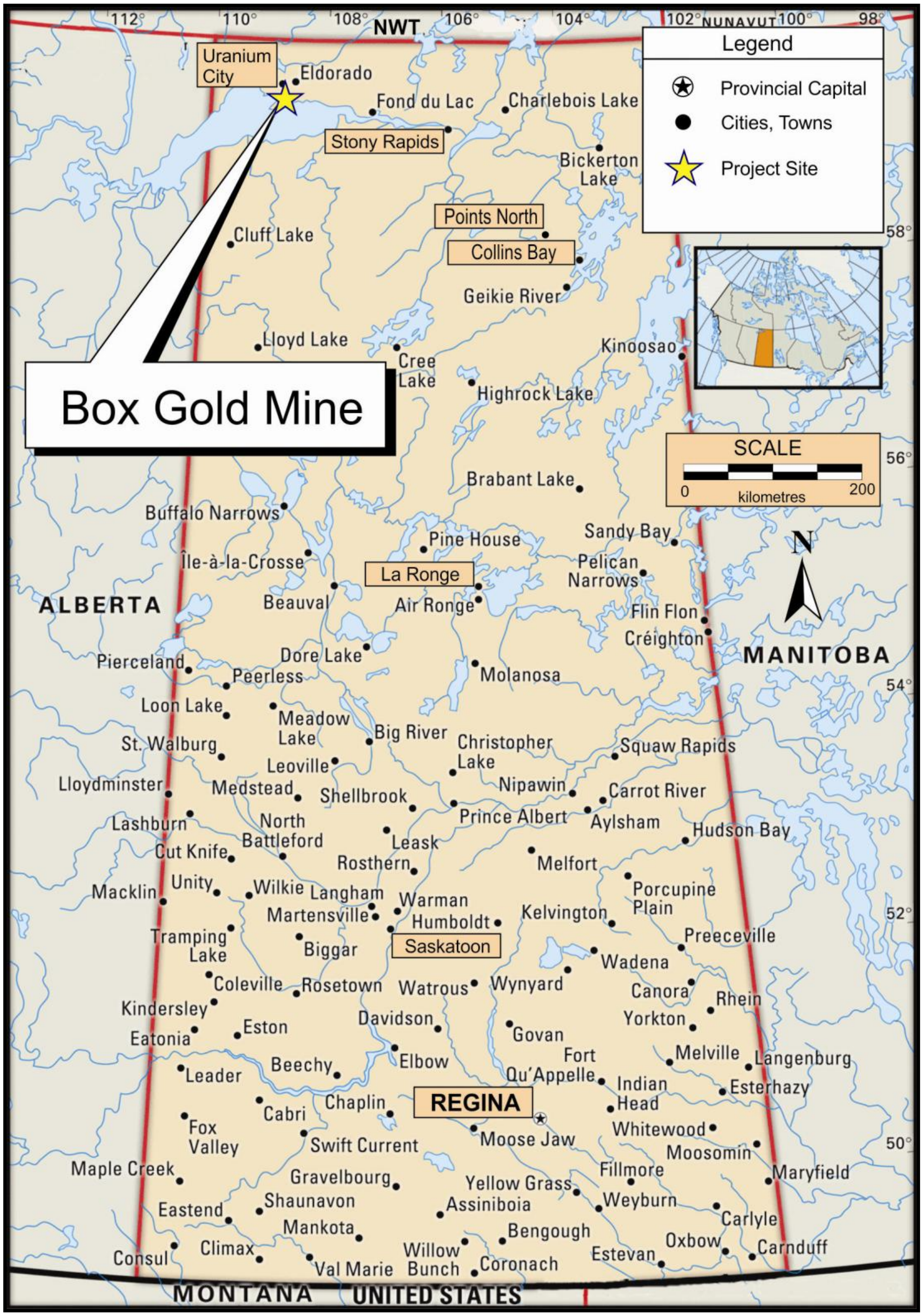

Figure 4-1: Goldfeilds Property Location Map (modified from Encyclopedia Britannica, 1999)

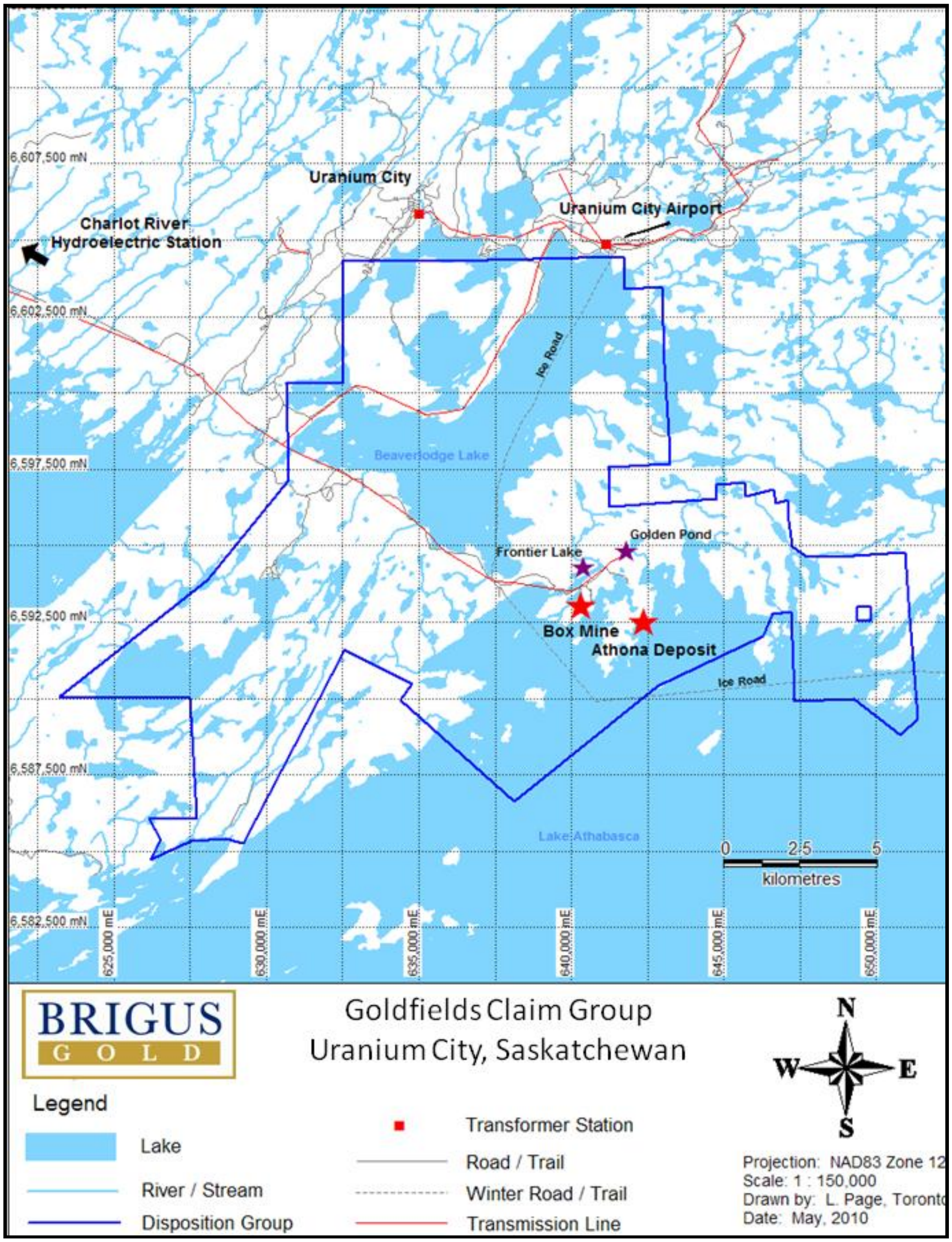

Figure 4-2: Goldfields Location Map (Brigus, 2010)

4.2 PROPERTY DESCRIPTION

Box is covered by one mineral lease, and surrounded by one mineral claim, of the Goldfields Claim Group, for a total of 4,617 ha, as summarized in Table 4-1 and illustrated in Figure 4-3.

Only the mineral lease covering Box and the mineral claim around the mineral lease are subject to this report. All other mineral rights are listed here for completeness. All claims are current and there are no outstanding issues with these claims.

Goldfields is controlled 100% by Brigus with Franco Nevada owning a 2% net smelter return (NSR) on an area of interest of 10 miles (16 km) from the external property boundaries of Box, Athona, Fish Hook Bay property and the Nicholson Bay property.

The Box project is also subject to a 1.5% NSR on all production beneath the 50 meters below mean sea level elevation that is on the original Cominco mining claims. This royalty does not apply to the current Box plan since it is above the minus 50 meters above sea level (ASL) elevation.

Table 4-1 - Summary of Box and Goldfields Mineral Claim Blocks

Mineral Claims	Number of Claims	Area (ha)
ML 5522	1	70
ML 5523	1	167
CBS 5664	1	4,547
Remaining Goldfields Claims	34	20,901
Total	36	25,685

Source: Government of Saskatchewan, Energy and Mines

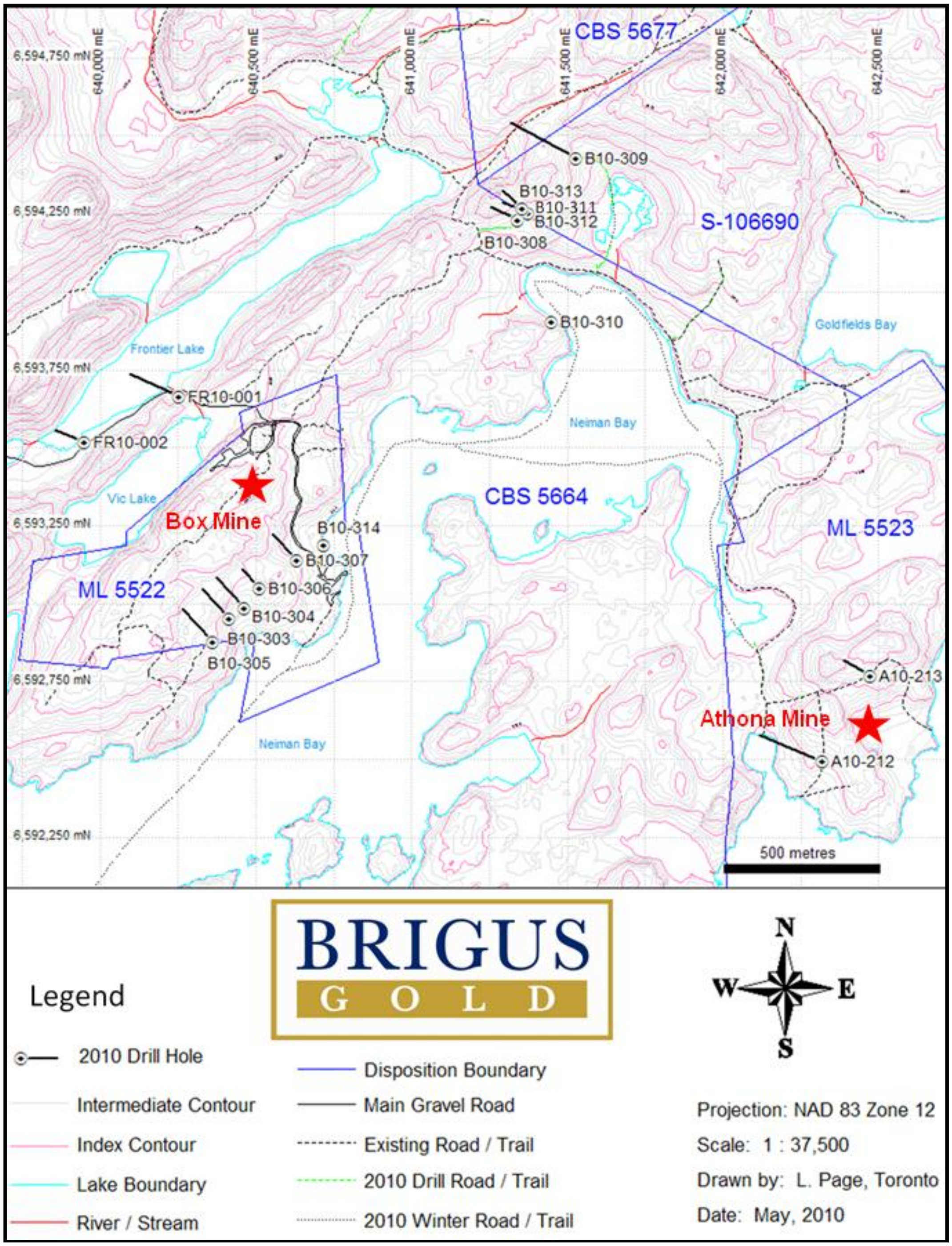

Figure 4-3: Goldfields Mineral Claim Map (Brigus, 2010)

4.3 MINERAL DISPOSITIONS

The Saskatchewan Provincial government is empowered by the Crown Minerals Act to grant mineral dispositions in the form of permits, claims and leases, within its jurisdiction, to qualified entities. The regulations covering these dispositions are contained in "The Mineral Disposition Regulations, 1986" otherwise known as Saskatchewan Regulation 30/86 and the pertinent subsequent amendments.

As defined by this Crown Minerals Act, those dispositions designated as Mining Claims do not require a legal survey, prior to registration; Mining Leases require a legal survey. The Claims indicated above have not been the subject of a legal survey; the Leases have been legally surveyed.

Under the terms and conditions as laid out in the Crown Minerals Act, all of the Claims and Leases pertaining to the Goldfields are designated as "Active" and are registered to Brigus.

These dispositions are subject to and must comply with the Assessment Work requirements as stipulated by the Crown Minerals Act. Brigus confirms that the assessment Work status of these Claims and Leases is current and that all Assessment Work pertaining to these dispositions has been both submitted and subsequently accepted by the Ministry of Industry and Resources.

4.4 EXISTING ENVIRONMENTAL LIABILITIES

As noted, Goldfields has been the object of previous mining operations. An Environmental Impact Statement; Box Starter Pit Mine, Goldfields Project, Northern Saskatchewan as submitted in January, 2007, prepared by UMA Engineering Ltd. for GLR has been completed and approved. The EIS has addressed these issues and provides appropriate remediation and mitigation measures. Brigus is committed to implementing these remediation and mitigation measures.

Brigus is currently negotiating a surface lease agreement that will define the extent of the existing environmental liabilities.

4.5 PERMITTING REQUIREMENTS

Brigus's 2011 exploration program is currently in progress. The current permits for the Goldfields Claim Group are in effect until October 2011.

The permitting process for mine development and construction involves the following as listed in Table 4-2.

Table 4-2: Permitting Process for Mine Development and Construction

Permit/Approval	Agency	Regulation/Act	Status
Environmental Permit (EIS Approval)	○ Ministry of Environment (Saskatchewan) as the leading Regulatory Agency	○ Environmental Assessment Act (Saskatchewan) ○ Canadian Environmental Assessment Act ○ Canada-Saskatchewan Agreement on Environmental Assessment Cooperation	Ministerial Approval Granted
Surface Land Lease Approval (Surface Lease Agreement)	○ Ministry of Industry and Resources (Saskatchewan) ○ Ministry of Environment (Saskatchewan) ○ Ministry of First Nations and Metis Relations (Saskatchewan)	○ Mineral Dispositions Regulations	Final Draft being negotiated
TMF - Effluent Discharge	○ Ministry of Environment (Saskatchewan), ○ DFO (Environment Canada)	○ MMER, Fisheries Act	
Approval to Construct	○ Ministry of Environment (Saskatchewan) ○ Ministry of Labour Relations and Workplace Safety	○ Environmental Assessment Act ○ Occupational Health and Safety Act ○ Saskatchewan Mining Regulations	Application to be submitted

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Property is situated roughly 1,000 km north of Regina, the provincial capital of Saskatchewan and 850 km north of Saskatoon. Access to the property is by a 25 km gravel road from Uranium City. Uranium City which is approximately 13 km from the property, is accessible by regular scheduled flights (twice a week during winter and spring months and three times a week during summer months) from Saskatoon through Prince Albert, Points North, Stony Rapids and Fond-du-Lac. Several charter aircraft companies operate in the area. Commercial flights to Uranium City from Saskatoon are typically four hours.

As there is no permanent road access to Uranium City, supplies are transported to Stony Rapids by road and then loaded onto barges and transported by water to communities around Lake Athabasca from June to October.

An ice road over Lake Athabasca is maintained from Stony Rapids to Uranium City and is open for six weeks in February and March. The Athabasca Basin Development Limited Partnership constructs and maintains the Lake Athabasca winter ice road annually with funding provided by the government of Saskatchewan.

Aircraft support for the project is readily available. Two airlines (TransWest Air and Pronto Airlines) currently provide passenger and freight service to Uranium City. The frequency of scheduled commercial flights will depend on project requirements. Personnel rotation requirements will be defined as the project advances.

The current airstrip is sufficiently long to handle C-130 STOL transport aircraft with freight capacities exceeding 30 tonnes.

5.2 CLIMATE

The following information was obtained from Environment Canada averages for Uranium City based on data from the past 58 years. The average temperatures during the winter months range from -8°C to -32°C and average temperatures during the summer months range from 11°C to 16°C. Average rainfall is 223.7 mm with the largest amount of rainfall recorded from May to October. The average snowfall is 215.1 mm and accumulates to 138.1 mm of precipitation. The lakes in this region generally freeze over by late September or early October and may remain frozen until late May.

5.2.1 PRECIPITATION AND EVAPORATION

The total average precipitation for the area is 361.8 mm per year, with February being the low month of 14.8 mm and August bringing 53.5 mm of precipitation. The greatest snowfall came on December 1, 1971 with 47.0 cm and the greatest rainfall was on June 27, 1962 with 46.7 mm. The year with the most snow on the ground was 1976 and on March 18 and 19, there was 97.0 cm.

Annual amount of evaporation has typically been an average of 300 mm, leaving a net precipitation of 60 mm.

5.2.2 TEMPERATURE

Average daily temperature ranges between -26.8°C in January and 16.2°C in July.

Average minimum temperatures recorded from 1953-1986 range from 11.0°C in July to -31.9°C in January. The extreme low temperature recorded from 1953-1986 was -48.9°C, occurring on January 15, 1974.

Average maximum temperature recorded from 1953-1986 ranges from 21.3°C in July to -21.8°C in January. The extreme high temperature recorded from 1953-1986 was 34.7°C, occurring on July 27, 1984.

5.2.3 WIND

Average wind speed measured from 1953 to 1986 is 11 km/hr, blowing from the east and northeast, 8% to 15% of the time.

5.3 TOPOGRAPHY

The elevation of Lake Athabasca is 211.5 meters (694 feet) ASL. The topographic relief of the area near the Beaverlodge - Goldfields area consists of moderately high hills with the highest being Beaverlodge Mountain at 419.8 meters (1,377 feet). North of the property, near Virgin Lake, the mean elevation of the area is about 450 meters (1,476 feet).

North of Goldfields, Contact Lake is approximately 427 meters (1,401 feet) ASL and the surrounding hills of conglomerate of the Martin Group, to the north of Contact Lake, are up to 445 meters (1,460 feet) ASL. Hills of basement rocks of the Tazin Group to the south are approximately 455 meters (1,493 feet) ASL. The relief decreases sharply to the north from approximately 445 meters (1,460 feet) to 350 meters (1,148 feet) in the sandstone and arkose units of the Martin Group located between Contact Lake and Cutler Lake.

The topographic relief is controlled by several factors generally related to the glacial erosion. Elongated ridges and valleys trend in the direction of the major fold for the Goldfields area and in the Fredette Lake basin. Many of the ridges in the western portion of the syncline are either composed of, or capped by amphibolite. The eastern portion have high hills consisting of dolomite, while in the Fredette Lake Basin have hills and ridges composed of either resistant conglomerate rocks or gneissic granitic basement rocks. Usually quartzite and granites which are jointed have been eroded and therefore are located in the valleys. Fault and shear zones are areas of weakness which are susceptible to erosion resulting in features such as talus slopes, linear deep valleys sometimes filled by lakes such as Vic Lake and Frontier Lake and straight topographic features. The eastern side of the Athona property is bordered by a relatively straight shoreline, along which are several small cliffs which continue below lake level. Additional erosional features of the area are caused by the foliation and jointing of intrusives as in the Macintosh granite.

Glacial rivers, discharge channels and alluvial fans are oriented in a southwesterly direction within the Beaverlodge area (Tremblay, 1972). These rivers and channels follow the topography of the area defined by elongated hills of rocks of the Martin Group and Tazin Group formed by erosion and locally fault-bounded. The thickness of glacial deposits over bedrock in the Beaverlodge area may vary from a few centimeters thick to about a few meters thick in low depression areas. A pit dug about 10 km south of Contact Lake near the Fay shaft shows that soil materials vary from reddish brown sand to yellowish clay, gravel, and fine sand over a few meters in thickness (Tremblay, 1972). Thicker overburden cover is present in areas lacking outcrops, around lakes and in valleys. Swampy areas in valleys are often composed of humus and peat layers that could measure a few meters to several tens of meters in thickness.

Soils present in areas of high relief and percentage of bedrock exposure are mainly composed of till, sand and silt of various colors (grey, beige, orange, brown and red). Their distribution is discontinuous and patchy. The thickness of till, sand and silty soils vary generally from a few centimeters to tens of centimeters near fractures, faulted bedrock and on some bedrock steps. Tills are rare, being generally of grey to pale brown color, and contain a few cobbles in a clayey matrix. Sand and silt soils are mainly beige, brown, and orange to red. Most of these soil types are transported materials mainly of glaciofluvial to lacustrine origin. However a component of the reddish and brown soils may be locally derived from the underlying bedrock based on soil particles observed near the contact with bedrock (Jensen, 2003).

5.4 LOCAL RESOURCES

Uranium City was once a major regional centre established during the uranium boom in the 1940's and 1950's. By 1982, Uranium City boasted a population of close to 5,000, however, in 1983, with the closure of the mines, Uranium City's economy collapsed. Uranium City is considered abandoned and maintains a population of less than 100.

There are few local resources available in Uranium City, although, gasoline, diesel and aviation fuel are available from the town and the airport. Most supplies must come from Stony Rapids or southern Saskatchewan.

The economy of northern Saskatchewan is based on mining, tourism and traditional hunting and gathering activities.

5.4.1 DEMOGRAPHICS

The primary impact area encompasses the Athabasca Basin between the northern settlement of Camsell Portage near the Alberta-Saskatchewan border extending east to include the communities of Hatchet Lake First Nation and the northern settlement of Wollaston Lake. Other communities within the Basin include the northern settlement of Uranium City, Fond du Lac First Nation, Black Lake First Nation and the northern hamlet of Stony Rapids. The communities of Wollaston Lake and Hatchet Lake First Nation are located within the Athabasca Basin. They are located approximately 300 km southeast of the project area and are less likely to be impacted either positively or negatively by the proposed development. In addition to these communities, there are outfitter camps located at various points throughout the basin.

The population of the region has fluctuated and shifted from earlier times when the focus was on Uranium City. With the closure of the Eldorado uranium mine in the early 1980s and a large depopulation of Uranium City as a result, the shift to other communities took place. Now larger populations are found in the east at Fond du Lac and Black Lake First Nations. The opening of the Goldfields mine will augment the revitalization of Uranium City and other communities in the basin.

In 1981, the Athabasca Basin had 4,536 registered residents in the region. Five years later in 1986, the population declined by over 50% to 2,058. The primary reason for the decline is the 1982 closure of Beaverlodge operations of Eldorado Nuclear at Uranium City.

Over the past 25 years the populations have grown to surpass the 1981 levels with the most noticeable increases in the First Nations communities and significant decreases in the settlements and hamlets in the basin.

Table 5-1 provides population information on the local and regional communities potentially influenced by the proposed Goldfields project.

Table 5-1: Human Populations of the Region

Community	1981[a]	1986[a]	1991[a]	1994[a]	1996[b]	2001[b]	2006
Black Lake First Nation	692	857	1029	1150	997	1054	1770[c]
Fond du Lac Fist Nation	16	20	31	41	751	682	1646[c]
Hatchet Lake First Nation	366	477	675	792	843	984	1428[c]
Wollaston Lake	138	187	121	106	1753	1546	129[d]
Uranium City	2479	237	171	175	-	-	85[d]
Camsell Portage	6	4	18	4	-	-	40
Eldorado	599	-	-	-	-	-	-
Stony Rapids	240	276	166	183	233	190	255[d]
Total	4536	2058	2211	2451	4367	4455	5353

Notes:
a: Historical data
b: Stats Canada, Community Profiles 2001. Results for Communities in Division No. 18, unorganized, are a total number for Wollaston Lake, Uranium City, Camsell Portage and other populations within this Division.
c: http://www.aboriginalcanada.gc.ca/acp/site.nsf/en/SK80001 .html
d: http://www.municipal .gov.sk.ca/apps/Pub/M DS/welcome.aspx

Uranium City

Uranium City is the closest community to the proposed Goldfields project. Uranium City is located approximately 25 km by road, northwest of the proposed mine site. The total population of the town and surrounding area is 85 according to the Saskatchewan Ministry of Health (2010). It is difficult to ascertain what percentage of this population is employable, because Stats Canada (2001) does not provide separate results for the community as listed under “Division Number 18 Unorganized”.

Fond du Lac

The population of Fond du Lac has increased dramatically in the past twenty-five years to a total of 1,646 members, with just over 900 individuals living on the first nation (Aboriginal Canada website, 2006). Thirty three percent of the population is in the 20 -44 year age bracket and is presumed to be the most employable. Of this group, about 13% have a college or trades certificate or a university certificate, degree or diploma. Fifty four percent of the population is under 20 years of age (Stats Canada, 2001). 190 people reported earnings in 2000; just under half of those people reported working year round and average earnings were $33,645 (Stats Canada, 2001).

Black Lake

The Black Lake First Nation population has more than doubled from 25 years ago to 1,770 members, with 1,402 individuals still residing on the Black Lake First Nation (Aboriginal Canada website, 2006). Of those members listed in the Stats Canada Statistics of 2001, 35% of the people in the age 20 -44 year bracket and are presumed to be the most employable. Of this group, about 18% have a college or trades certificate or a university certificate, degree or diploma. According to Stats Canada (2001), 195 people reported earnings in 2000. Of those 75 people worked year round with average earnings of $29,247. 53% of the population is under 20 years of age (Stats Canada, 2001).

Stony Rapids

The population in Stony Rapids as of 2006 was 255 (Stats Canada, 2006). Of this population 29.4% were between the ages of 0 to 14, 66.7% were between the ages of 15 to 64, and 3.9% were 65 and over. The population change from 2001 to 2006 was an increase by 34.2%.

Hatchet Lake

The Hatchet Lake First Nation is listed as Lac La Hache 220 on the Statistics Canada website. The Hatchet Lake First Nation population has more than doubled in twenty five years to 1,428 members. 1,106 people still reside on the reserve (Aboriginal Canada website, 2006). 56% of the population is under 20 years of age. 34% of the population is in the age 20 -44 year bracket and 10% of the population is aged 45 or older. Of the most employable age group (20 -44 years of age), 8.5% have a college or trades certificate or a university certificate, degree or diploma. Stats Canada (2001) reports that 240 people reported earnings in 2000 and 90 of those worked full time, year round with average earnings being $25,013.

Camsell Portage and Wollaston Lake

Camsell Portage has a population of approximately 40 people and there are 10 children in the local school (Athabasca Health Authority).

5.4.2 MINE LABOUR FORCE

Recognizing that the mine is located in a fairly remote section of the province, it will be necessary to have air transportation for at least some of the employees. Work crews will be brought to the site for a 14 day rotation. Air transport from local communities such as Fond du Lac and Stony rapids to Uranium City will be arranged. Other manpower resources would be transported from larger communities such as Saskatoon.

For the purposes of examining the availability of the labor force, the 20 - 44 years of age category are the most eligible for employment. Since 1986, the size of the younger labor force has declined in Uranium City and Stony Rapids, and has increased significantly in Fond du Lac and Black Lake First Nations. Statistics Canada (2001) groups information on the communities of Camsell Portage, Uranium City, Stony Rapids and Wollaston Lake under one category; Division No. 18 Unorganized. The population percentages in this grouping indicates that there is a fairly even three way split between children under 19 years, young adults up to the age of 44 and older adults up to and beyond 85 years of age. However, there is quite a different trend in the First Nation communities of Fond du Lac, Black Lake and Hatchet Lake. In these communities, people under 19 years of age make up 54%, 53% and 56% of the population while eligible workers up to 44 years of age make up on 33%, 35% and 34% of the populations in those respective communities. The older adult population is less than a tenth of the total population in those communities.

5.4.3 ECONOMY AND INFRASTRUCTURE

Uranium City, is serviced by a 1,500 m all weather runway with flights from Saskatoon. Charter air service by fixed wing including seasonal float and ski-equipped aircraft, and helicopter service are also available.

There is no permanent road to Uranium City. However, there is a gravel road from La Ronge, via Points North, and to Stony Rapids that allows for transportation of heavy equipment and supplies from the south to Stony Rapids. Heavy equipment and supplies may be transported by barge service operating from Stony Rapids, to the communities along the shores of Lake Athabasca from June to October. During winter, an ice road connects Stony Rapids to Uranium City.

Stony Rapids, situated approximately 150 km east of the property, is a logistics hub for northern Saskatchewan and hosts government administrative offices, banks, hospital facilities, hotels and food stores. Most consumable field supplies and equipment are readily available in Stony Rapids.

Electrical power for Uranium City and the region is supplied from the Charlotte River hydroelectric station operated by the provincial power authority, SaskPower. There are transmission lines connecting Box to the provincial grid. A power line has been constructed to connect this local grid with the provincial grid in the Collins Bay area.

Communities in the project area possess mixed economies characterized by both public and private sector investment. The local private sector economies are under-developed at most northern communities and are designed to service the residents' basic needs. Service stations, air landing strips, contracting services and small retail stores belong in this category. Recreational services are provided in all communities either by private enterprise or by the local administration. The public sector economy is represented by schools, policing services, government agencies such as health services and service groups. The spiritual aspects of the communities are addressed primarily by Roman Catholic missions and by other denominational groups. Table 5-2 provides a summary.

The Royal Canadian Mounted Police (RCMP) provides police service to the communities. Detachments exist in Fond du Lac, Stony Rapids and Wollaston Lake. The Stony Rapids detachment serves the community of Stony Rapids and the Black Lake First Nations Community. The Fond du Lac detachment serves the communities of Fond du Lac, Uranium City and Camsell Portage, while the Wollaston Lake detachment serves the community of Wollaston Lake and the Hatchet Lake First Nation. The proposed Goldfields development would be located within the area serviced by the Fond du Lac detachment. Impacts on policing, resulting from the proposed development, are expected to be negligible in Stony Rapids, Fond du Lac and Black Lake. However, there could be impacts in Uranium City where the development will result in an increase in the number of residents.

Basic community infrastructure includes water treatment plants, sewage lagoons and electrical power supply. Surface water from Black Lake provides water for the community of Black Lake, Fond du Lac River provides water for the community of Stony Rapids and the Fredette River provides water for the community of Uranium City. The community of Fond du Lac use wells for drinking water. Oil, wood burning stoves, and electric heat are the primary methods for heating homes and other buildings.

Housing shortages exist in the project area outside of Uranium City due to rapidly increasing populations in the area. New housing in the communities is limited. Hotel/motel facilities exist in Stony Rapids and Fond du Lac and Uranium City has a bed and breakfast facility. No impacts on housing demand are anticipated in any of the communities other than Uranium City.

Table 5-2: Business Infrastructure around Goldfields, 2006

Community	Business Category	Business
Black Lake/Stony Rapids	Retail/Commercial	Northern Stores
		Sarah's Store
		Scott's General Store
	Transportation	Billy Joe's Taxi
		Jimmy Sayazie's Taxi
		Pronto Airways
		R&JTaxi
		Robillard Taxi
		Transwest Air
	Other	Banner North Construction
		Camp Grayling Inc.
		E & L Enterprise
		Father Porte Memorial School
		Jagg Enterprises
		Medal Enterprises
		Stony Rapids Snowmobile Center
		Torson Contracting
		White Water Inn
	Organizations	Athabasca Denesuline Child & Family Services
		Athabasca Health Authority
		Athabasca Drug & Alcohol
		Black Lake First Nation Northern Hamlet of Stony Rapids
		Black Lake Gas Bar
		Black Lake Health
		Black Lake Education
		Black Lake Development Corporation
		NNADAP
		Independent Consultants in Food & Safety

Community	Business Category	Business
		SERM (Stony Rapids) Saskatchewan Highways (Stony Rapids) RCMP (Stony Rapids/Black Lake) Atmospheric Environment Service Stony Rapids Hospital Stony Rapids Home Care Services Stony Rapids School Roman Catholic Mission Visions North
Fond du Lac	Retail/Commercial	Northern Store P & M Gas Bar
	Transportation	Adam's Taxi Mom's Taxi Transwest Air Trapper's Taxi
	Other	A & C Arcade Trapper's Arcade
	Organizations	Fond du Lac RCMP ICFS Fond du Lac First Nation Father Gamache Memorial School Fond du Lac First Nation Fond du Lac Post Secondary Fond du Lac Health Outpatient Treatment & Prevention Center Fond du Lac Development Corporation Roman Catholic Mission
Hatchet Lake First Nation /Wollaston Lake	Retail/Commercial	Anne's Bakery Hatchet Lake First Nation Store Welcome Bay General Store

Community	Business Category	Business
	Transportation	Hector's Taxi Pronto Airways Transwest Air
	Accommodation	Hatchet Lake Lodge Welcome Bay Cabins Wollaston Co-op Hotel
	Other	D & D Camps Ltd Dene Enterprise Father Megret Elementary and High School Minor Bay Lodge & Outposts Ltd Pool Hall First Nation Recreation Wollaston Lake Lodge Ltd
	Organizations	Athabasca Economic Development & Training Corporation (includes Athabasca Basin Development Corporation, Pts Athabasca Contracting and PANS) Hatchet Lake Economic Development Corporation Hatchet Lake Education Hatchet Lake First Nation (Chief and Council) Hatchet Lake Health NNADAP Northern Settlement of Wollaston Lake Wollaston Lake RCMP Wollaston Lake SERM Wollaston Fishermen Co-Op
Uranium City	Retail, Commercial	Parkes General Store
	Transportation	Transwest Air Pronto
	Accommodation	Fish Hook Bay Lodge Urdel Ltd. (Bed and Breakfast)

Community	Business Category	Business
	Fuels	Uranium City Bulk Fuel
	Other	Camsell Portage Freighting Urdel Ltd. (Mechanical Shop) Uranium City Contracting - Construction Deconstruction GLR. - Exploration, gold Uranium City Resources - Exploration, Uranium
	Organizations /Schools	Athabasca Health Authority Beacon Baptist Church Ben McIntyre School Camsell Portage School Northern Settlement of Uranium City (includes Public Works) Saskatchewan Highways

5.4.4 HEALTH SERVICES

The Athabasca Health Authority provides medical services in the Athabasca Basin. In 2002, a new hospital was opened near Stony Rapids on Black Lake reserve land with 14 patient beds, four of them designated for long term care. There are four doctors and eight registered nurses employed at this facility; the doctors are on rotation from La Ronge and the nurses are also on a rotation from southern communities.

The Uranium City hospital was closed in May 2003 but a Primary Health Care Centre employing one nurse and one support worker remains and is located in the former RCMP barracks.

Both Black Lake and Fond du Lac have Primary Health Care Facilities. Black Lake has four and Fond du Lac has three primary care nurses and each has one home care nurse and support staff. They have regular clinic hours, Emergency Services 24/7, and other health care programs.

Turnaround time from the mine site to the hospital would be about 1.5 hours once the call is received. Transwest Air is contracted to provide medivac services and air ambulance services to Prince Albert and Saskatoon. The Athabasca Health Region also has access to a float plane and helicopter services which would decrease the turnaround time from the mine site to the hospital. A vehicle will be available at the site at all times for use in emergency transport to the Uranium City Airport.

5.5 EXISTING LAND USES

5.5.1 MINERAL EXPLORATION AND DEVELOPMENT

Mineral exploration and development in the Athabasca Basin has been primarily restricted to uranium-based mining activities in the last several years. These activities have been located south of Lake Athabasca at Cluff Lake Mine which is in the final stages of decommissioning and at various mines located in the Wollaston Lake area (Rabbit Lake, Key Lake, McArthur River, Cigar Lake, McClean Lake and Midwest Joint Venture). The development of the Goldfields Project would mean the first significant non-uranium related mining activity since the old Box mine and mill closed in 1942.

The development of the Goldfields Project will not impact negatively on mineral exploration and development activities but would be beneficial given that it could be used as a base for future work.

5.5.2 TRAPPING

From an economic point of view, trapping activity fluctuates with fur prices. Low fur prices in recent years have led to less trapping activity than has occurred when fur prices were high. However, marten and mink trapping may occur in the Goldfields area. Recently, there has been no long term trapping in the Goldfields area. One trapper from Fond du Lac traps along the north shore of Lake Athabasca but does not trap in the vicinity of the proposed development.

5.5.3 FISHING

Lake Athabasca is not heavily fished, given its remote location. Although a limited amount of commercial and subsidence fishing takes place, there are two fisheries cooperatives registered with the provincial government: one in Black Lake and the other in Fond du Lac. Goldfields is not expected to negatively impact their activities, since no fisheries-related discharge or habitat issues are anticipated.

A limited amount of recreational fishing and outfitter activity takes place in the area of the Project area, although it is primarily confined to Lake Athabasca. The main species of fish sought include northern pike, walleye, lake trout and lake white fish.

5.5.4 HUNTING

Residents of the Lake Athabasca Basin and recreational hunters as clients of outfitter camps, hunt primarily for moose and barren ground caribou. The Goldfields Project is located in Wildlife Management Zone 76.

5.5.5 TOURISM AND RECREATION

Limited tourism occurs in the area and is primarily associated with outfitter camps during the summer and fall months when fishing and hunting activities occur. Fishhook Bay Lodge is currently in operation with regular clientele on the north Shore of Beaverlodge Lake. The clients of this lodge fish primarily on Lake Athabasca. The proposed project will not negatively impact this activity.

5.5.6 FOREST AND WILD RICE PRODUCTION

Forests in the Taiga Shield are outside of the commercial forest zone and are in the reconnaissance zone. Commercial forest harvesting does not occur mainly due to the remoteness of the forests. Transportation issues and the small size of the trees makes them less marketable. Tazin Lake Upland is characterized by steep treeless slopes and closed stands of jack pine and black spruce on the sandy lower slopes.

Wild rice is an introduced grass species that was brought in for muskrat food and has proven to be a beneficial grain for human consumption. However, there are no wild rice crops grown within the project area as most of the production occurs in the central part of the province.

5.6 PHYSIOGRAPHY

The topographic relief of northern Saskatchewan is controlled by several factors related to glacial erosion. Predominant elongated, rounded and rolling valleys trend northeast/southwest in the general direction of glaciations and also follow the contours of a major fold in the Goldfields area.

The elevation of Lake Athabasca is 211.5 m ASL. The topography of the Beaverlodge-Goldfields area consists of hills of moderately high relief varying between 211 m and 420 m. The highest elevation on the property, from Lake Athabasca to Beaverlodge Mountain, is 419.7 m.

The vegetation in the property area is typical of boreal forests in northern Saskatchewan, consisting of small and scattered trees on hilly outcrop areas. The valleys are generally filled by glacial drift which supports a denser growth of larger trees. The major varieties of trees observed in the southern Goldfields area are black spruce, pine, white birch and poplar. The main tree species in the area are jack pine and white spruce over areas of bedrock and black spruce in valleys and swampy areas. The lower lying areas consist of muskeg swamps and bogs containing predominately black spruce, grass, shrubs, moss, and lichens.

Large portions of the Goldfields claim area have bedrock exposure as high as 50% with an overall bedrock exposure of approximately 10%.

5.7 INFRASTRUCTURE

The natural land contours will be used in the detailed design of the entire mine site to minimize capital cost and environment disturbance. The area has great potential to house all the required mine site areas including but not limited to the:

- Mill facility
- TMF
- WRSA
- Bulk Fuel Storage
- Bulk Explosives Facility
- Explosives and Cap Magazines
- Water Pump House
- Mine Service Complex
- Mine Offices and Dry Facility
- Electrical Substation
- Barge Loading Facility

More details on these areas are in Section 18 – Project Infrastructure.

6 HISTORY

6.1 PRE-1934

The earliest reported geological work in northern Saskatchewan commenced in 1880, when the Geological Survey of Canada conducted a topographic survey along the Saskatchewan River to Reindeer Lake and northwards to Lake Athabasca. From 1882 to 1893, J.B. Tyrrell surveyed many of the waterways and reported the presences of noritic rocks on the north shore of Pine Channel at the east end of Lake Athabasca.

6.2 DISCOVERY OF THE BOX GOLD DEPOSIT, 1934

Exploration activities in the Greater Beaverlodge (Uranium City) area dates back to the early 1930's. In August 1934, gold was discovered by Tom Box and Gus Nyman near the east shore of Vic Lake, adjacent to what is now the historic Box mine, where they staked a claim.

Consolidated Mining and Smelting Company of Canada Limited (Cominco) acquired the discovery by staking claims 1 to 17 and carried out surface and underground exploration. Trenching in two areas 45.7 m (150 ft) apart indicated 153.60 g/t (4.48 oz/st) gold (Au) over 0.85 m (2.8 ft) in the west pit and 3.43 g/t (0.1 oz/st) Au over 1.68 m (5.5 ft) in the east pit.

6.3 BOX MINE, 1935-1944

Development of a mine-mill complex was undertaken by Cominco from 1935 to 1942. Box operated from 1939 until 1942 when it was closed due to work force shortages brought about by World War II.

In 1935, a diamond drill program was completed consisting of 1,272.54 m (4,175 ft) in 13 drill holes. However, other sources indicated that 27 holes were completed for a total of 3,148 m (10,328 ft). Bench concentrate testing on a sample grading 80.57 g/t (2.35 oz/st) Au and 12.0 g/t (0.35 oz/st) Ag showed a recovery of 99.2%. In July 1935, the No.1 shaft, with dimension of 1.8 m by 3.7 m and inclined at 42° in the footwall close to the contact, was sunk to 76.2 m (250 ft). In September, the No.2 shaft, a three compartment production shaft, was collared 390 m (1280 ft) northeast of the No.1 shaft, and inclined at 45° at the footwall. Levels were established at 30 m (100 ft), 91 m (300 ft) and 152 m (500 ft) levels measured down dip. Drifts were driven near the footwall contact and horizontal holes were drilled across the ore body intersecting the main set of gold bearing quartz veins at the 300 level. Crosscuts were driven across the ore body at each shaft station and elsewhere along the drill holes to check results.

In 1936, development work began on three levels with plans for a 100 st/d mill. By April 1936, indications began to appear of some problems with the assaying carried out at the laboratory on the property. In May that same year, proposed plans to increase mill capacity to 500 st/d were approved but later rescinded.

In 1937, development continued from the No.1 and No.2 shafts on the 91.4 m (300 ft) level. Mill capacity was recommended to increase to 500 st/d. On June 29, 1937, an executive decision was made to proceed with a 1,000 st/d cyanide mill and the construction of a hydro plant. During this development the town of Goldfields was incorporated.

In 1938, development continued on three levels to a depth of 152.4 m (500 ft). An extensive program of underground drilling designed to intersect the main gold bearing stringers at right angles which amounted to 1,870 m (6,132 ft) of drilling. Hydro development completed and included 22 miles of transmission line, two tunnels and complete power house. An agreement was made between Cominco and Athona whereby the Box mill would process ore from the neighbouring Athona, 2 km to the east, and provisions were made for 3,000 st/d capacity.

By early 1939, 3,578 m (11,740 ft) of drifting and cross cutting as well as 8,967 m (29,419 ft) of drilling had been completed within the ore body. Box was identified as a very large tonnage and LG deposit. The first ore through the mine, on June 27, 1939, was initially extracted at 500 st/d. In August 1939, the first gold brick was poured with the mill capacity up to 1,000 st/d. Underground methods took advantage of the ore geometry and deposit size to develop large stopes through block caving. Recoveries were stated at 92%.

By August 1940, production was approximately 1,200 st/d. Drilling indicated a ‘reserve’ at less than $2.00 per ton with a large tonnage at less than $1.75 per ton (0.05 oz/st). The grade of mill feed was at an average of 1.707 g/t (0.0498 oz/st) (letter to Jewitt, 1945).

In June 1942, the decision was made to close Box due to a work force shortage with the outbreak of World War II. Estimated reserves at the time of the shutdown were given as 2.28 million tons at 1.714 g/t Au (0.050 oz/st). By this time, the mine had produced 67,899 ounces of gold from 1,417,520 tons grading 0.0479 oz/st. However, Coombe (1984) indicated that the production was 64,066 ounces of gold from 1,418,320 tons of ore at an average grade of 0.0452 oz/st Au.

6.4 ATHONA, 1935-1980

Athona is situated approximately 0.6 km southeast of Neiman Bay, and about 1.4 km south of the former town site of Goldfields. The mine was situated on the LUCKY-WILLY Group of 14 claims which were staked in 1934 by Great Bear Lake Mines Ltd., following the discovery of gold on the adjoining Box property of Cominco of Canada Ltd. Work consisted of extensive trenching and diamond drilling (7344.7 m or 24,097 ft) in 1935 which was successful in locating a number of gold occurrences.

The Willy Claims 1 and 2 and Lucky Claims 1 to 12 were staked in the fall of 1934 and spring of 1935 for Great Bear Lake Mines Limited.

Work between 1935 and 1938 consisted of extensive trenching and diamond drilling as summarized as follows:

- Work began on two shafts in 1935 which were sunk on the main zone as identified by N.W. Byrne. The No.1 shaft, a three compartment vertical shaft, was sunk to a depth of 85 m (278 ft) with levels at 38 m (125 ft) and 76 m (250 ft) and a westerly inclined 30 m (100 ft) winze from the 38 m level to the 106.68 m (350 ft) level. Lateral development consisted of 1016.20 m (3,334 ft) in the West Zone and 479.15 m (1,572 ft) in the East Vein on the 38 m level and 658.37 m (2,160 ft) in the West Zone and 676.05 m (2218 ft) in the East Vein on the 76 m level. Also, 13.72 m (45 ft) of raising was completed on the H vein system.
- The No.2 shaft, a two compartment -70° east inclined shaft located approximately 244 m south of the No.1 shaft, was sunk to a depth of 34 m (112 ft) with a level at 30 m (100 ft) with approximately 100.28 m (329 ft) of lateral development.
- In 1937 the company was reorganized, following the acquisition of Greenlee Mines Limited properties, and renamed Athona Mines (1937) Limited. Athona continued underground development indicating values around 0.22 oz/st Au.
- The annual report of 1938 indicated the ore reserves in the Main Zone as probable ore at 1,340,000 tons at 3.326 g/t (0.097 oz/st) Au (uncut) or 2.949 g/t (0.086 oz/st) Au (cut). The total "indicated ore" (including probable) was 3,485,000 tons at 2.949 g/t (0.086 oz/st) Au (uncut) or 2.743 g/t (0.080 oz/st) Au (cut). It was estimated that approximately 2,500,000 tons of ore was amenable to open pit mining based on a 1,500 ton per day operation. The 1939 annual report indicated the ore reserves in the Main Zone with 1,185,000 tons of probable ore at 2.949 g/t (0.086 oz/st) Au and in the East Zone as 30,000 tons of probable ore at 8.914 g/t (0.26 oz/st) Au and 50,000 tons of possible ore at 5.829 g/t (0.17 oz/st) Au.
- Athona closed not from the lack of ore or grade but for the lack of an agreement with Cominco on treating the ore and no satisfactory source of power for the operation of a mill.

These resource estimates are historical and do not meet the standard terminology criteria of the NI 43-101 or the CIM Standards on Mineral Resources and Reserves. Wardrop has not done the work necessary to verify this historical estimate. The estimate is no longer relevant as it is replaced by the current estimate prepared by Wardrop. The historical estimate should not be relied upon.

Between 1952 and 1953, Pole Star Mines Ltd. controlled the WILL 1 to 9 claims. Pole Star completed two holes close to SMDI 2163 that intersected minor scheelite (AF 74N08-0025).

By 1967, the showing was under Mokta CBS 305. Mokta completed an airborne scintillometer survey and ground follow-up mapping and prospecting (AF 74N08-0036).

In 1970, Norcan Mines completed two drill holes and radiometric prospecting on the CBS (AF 74N08-0071.0072).

In 1980, Pyx Exploration carried out a heap leaching test on three barrels of ore grade material.

6.5 DEJOUR MINES LTD., 1968

In 1968, Dejour Mines Ltd. completed detailed geological mapping and radiometric prospecting. The work discovered a number of radioactive fractures on Box VIC claim No. 2. In the same year, a small amount of diamond drilling was completed to test the fracture system. One hole intersected pitchblende filled fractures that assayed 14.0% triuranium oxide and 0.66% U_3O_8 over 150 mm.

6.6 GLR RESOURCES INC., 1987-2009

GLR, and its former companies, conducted exploration programs in the Beaverlodge area from 1987, when the Kasner Group of Companies optioned both Box and Athona properties from Cominco, until the sale of the leases and claims to Brigus and its former companies in 2009. Since that time, Mary Ellen Resources Ltd., Lenora Explorations Ltd. and AXR Resources Ltd. merged together in December of 1988 to form Greater Lenora Resources Corporation, which later became known as GLR . On July 24, 2001, Greater Lenora Resources Corporation completed a plan of arrangement pursuant to which GLR acquired all of the assets of Greater Lenora Resources Corporation.

The Kasner Group of Companies completed three diamond drill holes from September 2 to 23, 1988. These holes, LB-88-1 to LB-88-3, totalled 1,132.1 m. Drill holes LB-88-1 and LB-88-2 are located approximately 700 m northeast of Shaft No.2 and had an azimuth (AZ) of N 310°E, dipping 45°. LB-88-2 is approximately 50 m at N 130° E from LB-88-1 and these holes did not contain significant gold values. Hole LB-88-3 was drilled to test the down dip extension of Box under Neiman Bay and contained a number of anomalous gold values with the best intersection being 4.663 g/t Au (0.136 oz/st) over 3.0 m (Bowe and Petrie, 1988) (AF 74N07-0328). A 9,000 ton bulk sample from the Box trenches and a 4,000 ton bulk sample from the Athona trenches returned a reported grade of 1.88 g/t (0.055 oz/st) gold.

Also, a single drill hole (VI-88-1) was completed to test potential of the Vic Lake Fault zone. This drill hole was located at the boundary of the Box and Lodge Bay project boundaries. No significant gold assays were obtained.

During 1988, 52 drill holes were completed totalling 6,384 m. In the summer of that year, nine diamond drill holes were surveyed with a Mount Sopris down-hole radiometric logging instrument. The survey results indicated none of the holes contained uranium mineralization. A slight increase in radioactivity was detected in the BMG, foliated granite and HW gneisses, which may be caused from the decay of potassium in the feldspars and sericite.

In 1988, GLR completed a prefeasibility ore reserves calculation, using 1934 to 1988 data (AF 74N-0005).

The 1989 drilling program completed 47 reverse circulation (RC) drill holes with a total footage of 3,169 m. Samples were assayed by fire assay technique by early participants as well as several drill holes in the 1988 GLR drill program. The remainder of the 1988 drill holes and those of the 1989 RC drill program were assayed by the cyanide leach technique.

In 1989, GLR completed on-site metallurgical testing and bulk sampling.

In 1990, GLR announced new reserves for the deposit. The reserves were calculated using the Kriging, inverse distance weighting, and polygonal methods.

In 1992, GLR calculated the combined reserves for Box and Athona (Toronto Stock Exchange - January 20, 1992). RJK Mineral Corporation also listed the Box reserves.

After a period of evaluation and additional exploration activities, GLR completed 52 drill holes totalling 6,706 m in 1994. From 1994 onwards, the complete BQ drill core was assayed by the total metallic technique.

In 1994, they completed and released the results of drill holes B94-109 to -150. These infill drill holes were designed to test the deposit below the level of the existing mine workings (press release).

Between 1994 and 1995 delineation holes B95-151 to 250 and an environmental impact study was completed on the deposit (AF 74N-0006). The Box and Athona reserves were re-calculated at this time.

In 1995, 17 drill holes and an environmental impact study were completed on the Box-Athona Mines (AF 7408-0150). In October 1995, the combined reserves for Box-Athona were published. Box is open in all directions and Athona is open to the southwest.

A total of 18,825 m of BQ diamond drilling in 100 holes was completed during the 1995 summer program. The program was designed to fill in voids from previous drilling programs, define the strike limits and define the down dip extension of the mineralized zone below the level of the proposed open pit. The details of this program and the diamond drill sections were covered by McKay (1995).

The geological mapping of the trenches at Box was completed by F. Hurdy during 1995.

During 1995, the winter/spring diamond drill holes and the drill holes of the summer program were surveyed. At both Box and Athona, several of the original survey monuments and mine workings, such as vent raises, shaft collars, etc., were located and surveyed.

This accurate drill hole collar information was incorporated into the various data bases used for the reserve calculations. The surface trenches surveyed at Box were incorporated into the up-dated data base.

A feasibility study was completed by H.A.Simons Ltd. in 1995.

In 1996, GLR announced a new, lower combined reserve calculation for Box and Athona (labelled a resource calculation). This was published in The Northern Miner (13 May 1996, p.1-2).

In 1997, Behre Dolbear completed an ore reserve audit of published reserves for the combined Box-Athona. In 1997, Pearson, Hoffman & Associates completed a resource evaluation of Box. In the same year, GLR flew an airborne electromagnetic (EM) , resistivity, magnetic, and spectrometer survey over the property (AF 74N-0007).

In 1999, GLR announced intentions to proceed with a small-scale development and production plan which will focus on open pit mining a part of Box.

In 2001, GLR stated that an environmental impact study had been completed and Gekko of Australia tried simple gravity separation of Box ore and received excellent recoveries. GLR also concluded that the Fishhook and Nickolson were an unconformity type Au-PGE deposit that is similar to the Coronation Hill deposit in Australia.

In 2007 and 2008, GLR completed 13 drill holes over the Box mine totalling 3,348 m. Six of these 13 holes intersected the Box ore.

On August 21, 2009, GLR announced the sale of all its Goldfields assets, which included Box and Athona, to 7153945 Canada Inc. (7153945), a wholly owned subsidiary of Linear.

On March 17, 2011 GLR was renamed Mistango River Resources Inc.

6.7 LINEAR, 2009 – BRIGUS, PRESENT

Linear acquired Goldfields in August 2009. In early 2010, Linear undertook an infill drill program over the Box and Athona deposits. The drill program over the Box area consisted of twelve drill holes for a total of 2,858 m, where five of the twelve drill holes intersected the Box Mine gold deposit.

On June 24, 2010, Linear and Apollo Gold Corporation (Apollo) merged and changed their name to become Brigus Gold Corp.

6.8 DRILLING

The following was taken from Bikerman, 2009.

> *Initial surface and underground diamond drilling was conducted on the Box and the Athona properties during 1935 to 1939. Diamond drilling by GLR commenced in 1987-1988 on both properties. This was followed by reverse circulation drilling in 1989 and two additional*

programs of diamond drilling in 1994 and 1995. In total (1935 to 2005), 402 holes totalling 48,561 metres containing 17,226 assays has been completed on the Box property. The underground sampling of the drifts and crosscuts were compiled into 32 pseudo-drill holes totalling 6,548 m containing 4,385 assays. Table 6-1 summarizes the diamond drill and underground sampling at the Box Mine.

Table 6-1 - Box Drill Hole and Sample Summary

Year(s)	Company	Type	Location	Number of Holes	Drill Core Size	Length (m)	Length (ft)	Number of Samples
1935-39	Cominco	channel	underground	32		6,548.65	21,485.07	4,385
1935-39	Cominco	DDH	surface	42	EX	4,576.12	15,013.52	1,708
1939	Cominco	DDH	underground	72	EX	4,594.98	15,075.39	2,959
1987-88	GLR	DDH	surface	52	BQ	6,383.73	20,944.00	2,628
1989	GLR	RCD	surface	47		3,168.60	10,395.67	2,715
1994	GLR	DDH	surface	52	BQ	6,705.77	22,000.56	2,443
1995	GLR	DDH	surface	100	BQ	18,825.00	61,761.81	3,469
2004	GLR	DDH	surface	15	NQ	1,007.67	3,306.00	577
2005	GLR	DDH	surface	22	NQ	3,299.15	10,823.98	782
Totals:				434		55,109.67	180,806.00	21,611

"The core size of the 1935-1939 surface and underground drilling on both properties was EX size. Original historical records show gold values in ounces per ton (oz/st) for the Box Mine. There are no indications of any check samples. The average length of samples was 1.5 metres. Simons, 1995 reports that the Box "Assay results for the underground diamond drilling are up to 26% lower than the overall average value, and 19% lower than the average of total metallic assay values (1994 and 1995 drilling)…" The Simon's report states that for the surface drilling at Box in the 1930s the "Assay results for the old diamond drilling are up to 31% lower than the overall average assay value, and 25% lower than the average of total metallic assay values (1994 and 1995 drilling)…"

Diamond drilling at the Box deposit in 1988 was BQ size. The average sample length for the Box deposit was 1.0 m (Simons, 1995 reports 0.99 m). Simons (1995) states that the "Assay results from this drilling are up to 31% lower than the overall average assay, and 24% lower than the average of total metallic assay values (1994 and 1995 drilling)…".

In 1989, reverse circulation (RC) drilling at Box was conducted over a small area, 150 m of strike length. A total of 47 holes with a diameter of approximately 5.5 inches were drilled. Western Caissons, using a RC hammer drill, conducted the drilling. During the drilling the bit size was reduced every 25 m from 5.5", to 5.25", 5.125" and so on. Individual samples were taken at one-metre intervals. Samples weighed approximately 36 kg. Samples were assayed by total cyanide leaching by Casmyn Research. The main purpose of the drilling *program at Box was to verify and better define the grades of the orebody. "The initial results from the detailed RC drilling indicates that the Box Granite at this location is composed of one large massif containing two to three relatively higher grading pods and one "outlier" pod towards the footwall." (Carmicheal, S., 1990).*

Simons (1995) reports that "The average sample length from the 1989 drilling was 1.11 m. Assay results for this drilling are up to 43% lower than the overall average assay value, and 38% lower than the average of total metallic assay values (1994 and 1995 core drilling)…" Reasons given for this difference are "1) the gold is relatively coarse, and therefore heavy, and was not carried to the surface. 2) the samples were tested using cyanide leach assaying, which may not have extracted all of the gold."

In 1994 and 1995, a total of 152 holes, 25,531 m, were drilled at the Box deposit. The average sample length was 1.0 metre (Simons, 1995 reports states an average of 0.99 m). Simons, 1995 states that at Box the "Assay results for the 1994 diamond drilling are by the total metallic assay technique and are deemed to be the most reliable. The average assay values are 1% lower than the overall average…." The Simons reports go on to state that "The average for the 1995 holes is 21% lower than the overall average assay, and 14% lower than the total metallic assay values (1994 and 1995 core drilling)…."

The Box "Mine Granite" (BMG) is a fine- to medium-grained, pink, red, "brick red", pale to medium orange and white unit which consists of quartz and feldspar, with muscovite and sericite. The unit exhibits "porphyritic" texture consisting of pebble to cobble sized clasts and subrounded to rounded "fragments".

The BMG unit has a surface expression in excess of 750 m with an average width of about 40 m and reaches about 60 m width in the central portion of the zone. The lithological sequence that makes up the BMG has an overall strike of N47°E and dips to the southeast at from 30° to 45°. The BMG is lenticular and slightly arcuate in plan view. Deep drilling indicates the BMG extends over 200 metres below the surface. WGM, 1995 report states;" Taking all information into account, the drilling that was done on sections at right angles to the granite appears to be reasonably representative of the deposit."

Gold mineralization is associated with fine grained pyrite in the range of 0.5% to 3.0% in the wallrock and quartz-quartz-carbonate veins. Some of the auriferous quartz veins trend N10°E and have associated sulphide mineralization in order of abundance as pyrite, galena, sphalerite, and chalcopyrite.

Quartz veining in the larger "fragments" is truncated at the edges of these "fragments". The unit is hard to very hard and is non-magnetic. Alteration consists of weak to pervasive hematitization, sericitization, silicification, pyritization and occasional weak chloritization. Chlorite also occurs as minute seams and along fractures. Hematite locally occurs as fracture fillings. Quartz and quartz-carbonate occurs as irregular stringers, veinlets, *fracture fillings and irregular masses. Quartz veins vary in size from a few mm to several tens of centimetres. Veins are occasionally vuggy. Veins occur in several distinct vein sets that crosscut each other. Pyrite occurs as disseminated specks, minute irregular seams, irregular masses, crystal clusters and euhedral pyrite up to 1 cm across. Pyrite content varies from trace to 10%. Chalcopyrite, up to 2%, occurs as irregular masses and minute seams. Galena, up to 2-3%, occurs as irregular masses mixed with chalcopyrite and sphalerite and as scattered euhedral crystals up to a few millimetres wide. Sphalerite, locally up to 1-2%, occurs mixed with chalcopyrite and galena. Gold, as irregular blebs, minute seams and scattered specks, up to 2 mm across, occurs in quartz veins, at or near contacts, with pyrite, as minute seams and in inclusions of wall rock.*

The BMG often contains subunits, of varying thickness, of the various hanging wall units. These subunits are usually more altered and often exhibit stronger foliation than the corresponding hanging wall unit. These altered subunits rarely contain significant assay values.

It appears that a narrow unit of the BMG at or near the hanging wall contact exhibits brittle-ductile shearing parallel to the lithological contacts with the development of two generations of quartz veining represented by shear and extensional en echelon gash vein filling. The rotated veining probably indicates that the veining event occurred during progressive shearing. Several quartz stockwork breccia zones cross cut the BMG and may be the result of parallel fluid inflow into dilatant jogs in a dextral strike-slip fault system with steps to the right. Vuggy quartz and/or pink-carbonate breccia zones typically identify these late N45°W trending faults.

The diamond drilling program for 2004 and 2005 was suggested by AMEC to provide confirmation drill holes at or near previous drill holes from the various drilling campaigns since the historical Cominco drill core was not available for inspection and sampling, and the bulk of GLR's core sampling used the whole drill core for analytical purposes. The sampled portions of the drill core could not be inspected, and the assay method used was total pulp metallic analysis. The rejects and pulps could not be used to confirm the analytical results. Table 5 (not reproduced in this report) summarizes the collar coordinates of the confirmation 2004 and 2005 diamond drill holes and the previous drill holes that were being confirmed as part of the QA/QC for the Box Mine Project.

The current drilling would also provide the necessary requirements for a Quality Assurance and Quality Control (QA/QC) assay information. The on site supervision was conducted by John Dixon with indirect (off site) supervision by KAJ.

The 2004 diamond drilling was completed by Can-Drill of Saskatoon Saskatchewan. The drilling equipment consisted of a truck mounted Boyles 20 hydraulic diamond drill with a hydraulic bean supply pump. The initial phase of thirteen NQ size drill commenced on August 23, 2004 and was completed on September 21, 2004. The initial program was to *consist of 20 holes; however, all of the planned holes could not be completed utilizing the drill in a truck mount configuration. A second phase of drilling with the rig mounted on skids commenced on December 13, 2004. During the interim between the first and second phase an additional 22 holes were added to the program. Unfortunately, Can-Drill was unable to complete phase 2, and Hy-Tech Drilling of Smithers, British Columbia was awarded the contract to complete the project. Hy-Tech commenced on April 9th, 2005 and completed the drilling on May 9th, 2005. The Hy-Tech equipment consisted of a unitized rig pulled by a John Deere 640 skidder. The drill rig consists of a B-20 hydraulic drill with a bean hydraulic supply pump.*

Tri City Surveyors of Saskatoon surveyed the drill collar locations, checked two mine control stations and five 1988 drill collar locations and five piezometer holes by Global Positioning System (GPS). The Geodetic elevations were derived by a GPS survey (July 19, 2005) tied to Canadian Active Control System benchmarks in Yellowknife and Flin Flon. The geodetic elevations differed by +4.1810 m from Webb (S.L.S.) in 1988 and surveying completed in 1995. The 2004 and 2005 collar elevations were adjusted to the Box Mine survey control elevation.

The 2004 and 2005 diamond drilling program used NQ core size consisting of 15 and 22 drill holes respectively totalled 4,306.82 m. Of these drill holes, five where drilled specifically for piezometer testing. Piezometers were also installed in three of the previously drilled holes. Information gained from the piezometers will be used in modeling the water flow within the various rock units. All drill core was photographed wet prior to sampling for archive purposes. The 2004 and 2005 drill core boxes were labelled with metal tags and cross piled beside the core logging facilities in Uranium City.

6.9 OWNERSHIP HISTORY

The Goldfields takes its name from the former village of Goldfields, a gold mining settlement established in the 1930s. The larger area has attracted considerable interest over the last half century from a mineral extraction perspective.

From 1939 to 1942, Cominco operated an underground gold mine at the location of the Box deposit. In 1942, the mine was closed due to war time personnel shortages.

In 1987, Lenora Exploration Ltd. and Mary Ellen Resources Ltd. (later to become GLR) jointly optioned Box and Athona and commenced work to evaluate them as open pit operations.

In May 2009, Linear acquired the Box and Athona properties. In June 2010, Apollo and Linear merged into one combined entity named as Brigus. As a result, Box and Athona have been under the ownership of Brigus since June 2010.

6.10 PRODUCTION HISTORY

Gold was first discovered in Saskatchewan in the North Saskatchewan River near Prince Albert in 1859. Saskatchewan began producing gold in small quantities in the early 1900s and possibly earlier from panning and dredging operations on the North Saskatchewan River and its tributaries. Gold production in the province has been from placer operations on the North Saskatchewan River and from gold and base metal mines in the northern Shield. Of the total production in the province, approximately 90 percent originated as a by-product from the Hudson Bay Mining and Smelting Company's copper-zinc mine at Flin Flon.

In the period prior to the First World War, gold was discovered on the north shore of Lake Athabasca and in the Amisk Lake area near the present sites of Creighton and Flin Flon. Further prospecting in the 1920s and 1930s culminated in discoveries in the La Ronge volcanic belt, Flin Flon and Beaverlodge areas. By the late 1930s and early 1940s, gold was being produced in significant quantities at Box on the Crackingstone Peninsula and in minor amounts from the Prince Albert (Monarch/Pamon), Graham and Henning-Maloney mines near Flin Flon. Other deposits produced small amounts of gold during trial mill runs.

From 1939 to 1942, Cominco operated an underground gold mine at the location of Box. The mine processed approximately 1.29 million tonnes of ore having a calculated grade of 1.64 g/t, recovering 1,992,645 grams (65,066 ounces) of gold and 690,642 grams (62,205 ounces) of silver. Approximately 1.2 million tonnes of tailings were generated and deposited into the north end of Vic Lake.

The gold boom of the late 1980s resulted in the first significant gold exploration effort in the history of the province. Gold exploration figures reached their peak of $55 million in 1988. Large areas of high gold potential still remain unexplored. Five new gold mines have entered production in Saskatchewan since 1987 (Sask. Geol. Surv. 2006). Between 1987 and 1994, additional drilling was completed by GLR. The resource database to 1995 included over 26,800 m of core drilling and 3,169 m of RC drilling results. This information indicated ore reserves sufficient to warrant development of an open pit gold mining and milling operation.

7 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

From Jensen, 2003:

Northern Saskatchewan is predominantly underlain by variably deformed and metamorphosed rocks of Archean age (3070 to 3014 Ma.) to Helikian (1450 to 1350 Ma) age. In the northwest, the Archean to Aphebian crystalline basement, influenced by Lower Proterozoic thermitectonic events, is overlain by redbeds of the Martin Group (and immediately underlying Thluicho Lake and Ellis Bay Groups) which were probably deposited during and immediately following the main Hudsonian event (ca. 1900 to 1800 Ma.). Immediately to the south, the metamorphic basement rocks are overlain by post-metamorphic sedimentary rocks of the Helikian Athabasca Group. Post-Hudsonian diabase dykes (ca. 1400 to 1100 Ma.) are the youngest rocks in the Precambrian of northern Saskatchewan (Jensen, 2003). The general geology of the Beaverlodge area with the current property outline is illustrated in Figure 7.1.

The metamorphic crystalline basement is part of the Trans-Hudson Orogeny, a major Early Proterozoic orogenic belt that extended more than 5,000 km from the U.S.A. to Greenland (Lewry et al., 1985). In Saskatchewan the basement rock have been grouped into three broad crustal regions: the Western Craton, the Cree Lake Zone and the Reindeer Zone (e.g. Macdonald, 1987). The ensialic Western Craton and Cree Lake Zone lie west of a major crustal 'break' (the Needle Falls Shear Zone); the ensimatic Reindeer Zone lies to the east.

The Western Craton (Lewry and Sibbald, 1977; Macdonald, 1987) forms part of the more extensive Keewatin Zone (Hoffman, 1989), which underlies contiguous parts of the Northwest Territories. The Western Craton is separated from the easterly-lying Cree Lake Zone by the Virgin River-Black Lake Shear Zone. The later zone of mylonitic rocks form part of the 3,000 km long Snowbird Line that has been variously interpreted as an intracontinental activation structure or a crustal suture within the Trans-Hudson Orogeny.

The Western Craton comprises mainly Archean basement, Lower Proterozoic granite plutons and migmatites, and remnants of both Archean and Early Protoerzoic supracrustal belts. The Archean rocks have suffered upper amphibolite to granulite facies metamorphism during the Kenoran orogeny. These were later variably overprinted by Hudsonian greenschist to amphibolite fades metamorphism.

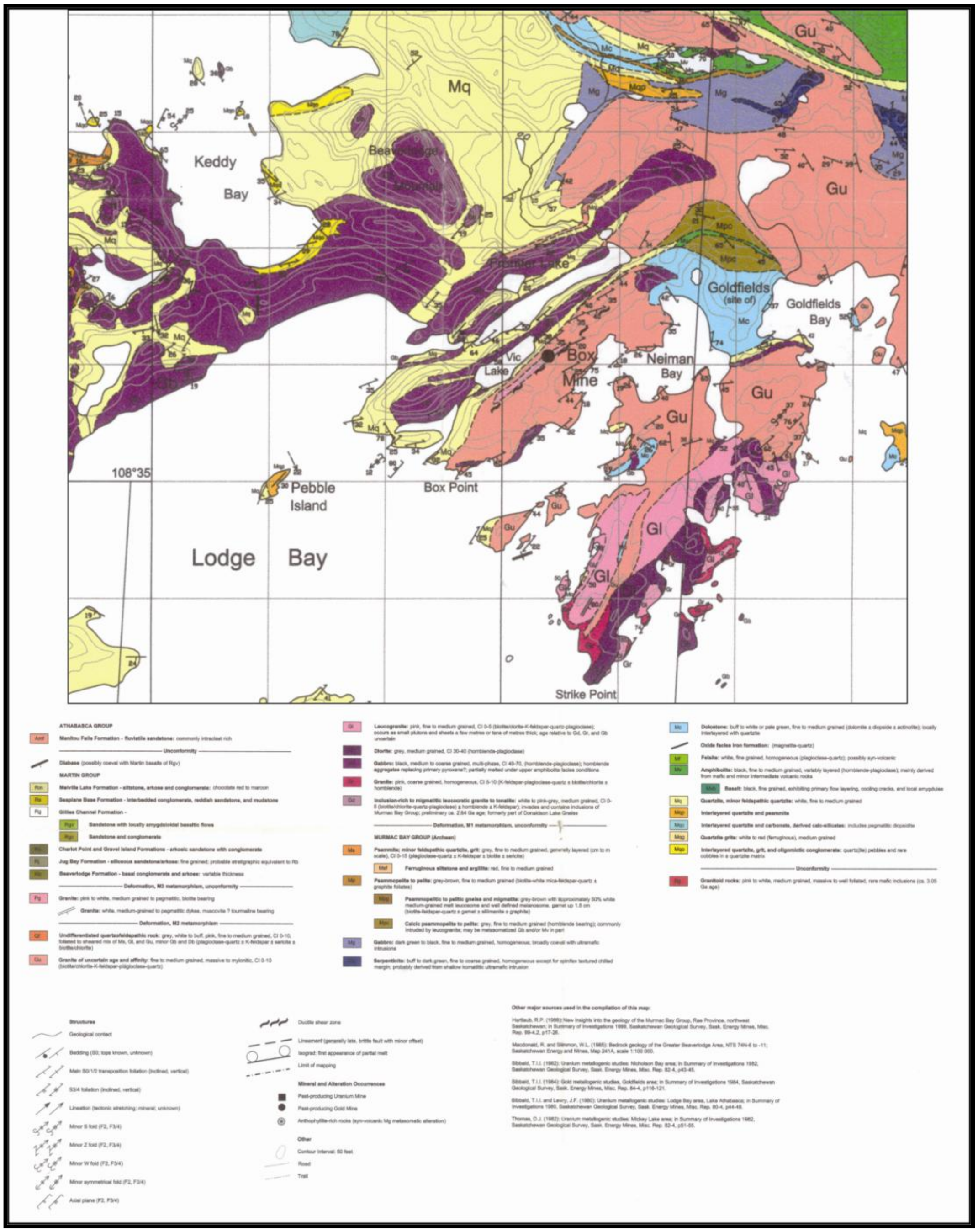

Figure 7-1: Regional Geology Map

North of the Athabasca Basin, the Western Craton is formed by several predominantly Archean cratonic blocks, bounded by mylonitic shear zones, and intervening terrains of reworked Archean and/or Lower Proterozoic retrogressed granulite facies rocks. The Western Craton is formed mainly by sediment and volcanic derived supracrustals. Alcock (1936b) termed these rocks the "Tazin Group". Later workers (e.g. Tremblay, 1972) also included derived felsic gneisses and granites within the Tazin Group. Linear zones of intense

Hudsonian reworking, manifested as zones of refoliation within this part of the craton (Beck, 1969).

The Goldfields Property of Brigus Gold Corp. is located within the past producing Beaverlodge mining district.

The Goldfields Property lies within the Western Craton Tectonic Zone of the Churchill Structural Province (Stockwell, 1961). The property is located in the southwestern portion of the Beaverlodge Domain (includes the former Black Bay Domain and the Nevins Lake Block, Morelli et al, 2001) of the Rae Province of the Canadian Shield which rests on the Churchill Platform which was sutured to the Superior Craton to the southeast during the Hudsonian Orogeny.

7.2 PROPERTY GEOLOGY

7.2.1 BOX MINE

From Jensen, 2003:

The geological setting at the Box Mine deposit consists of a sequence of metasedimentary lithological units. The footwall sequence is represented by several series of alternating units of amphibolite and quartzite. These units exist from north of the Frontier Mine to the Box Mine footwall, an approximate horizontal distance of 1,000 metres. At the footwall contact, a zone of metasediments consists of almost pure quartzite, feldspathic arkose, medium to coarse grained greywacke and sub-angular to rounded pebble conglomerates. Scattered along the footwall at irregular interval are amphiboles intrusive sills and/or homfelsed metasediments with some units exhibiting varying degrees of shearing which forms chlorite and homblende schists.

The BMG unit is a depositional sequence of metasedimentary lithologies grading towards the southeast from a pebble to cobble size conglomerate to a coarse grained greywacke then medium grained, followed by feldspathic arkose. Due to the varying intensity of granitization or feldspathization and silicification of the clastic metasediments, it is difficult to determine if more than one sequence exists. The BMG has been moderately to intensely altered by hematitization which indicates the contacts of the auriferous zone. The contacts vary from gradational to sharp.

The BMG has a surface expression in the excess of 750 metres and an average width of 40 metres with the central portion in the excess of 60 metres. The lithological sequence of the BMG has an overall strike of N047^{o}E with dips ranging from 30^o to 45^o with an average dip of 43^o to the southeast. The Box Mine deposit has a strike of the LOWER BMG zone and the *southwestern portion of the UPPER BMG zone of N047^{o}E with a 42^{o} southeast dip and the northeast portion of the UPPER BMG zone has a strike of N060^{o}E with a 42^{o} southeast dip.*

The immediate hanging wall of the BMG generally consists of medium to coarse grained greywacke with fine grained arkosic and amphibolites. The hanging wall contact zone is usually sheared near parallel to the strike of the lithologies and dipping approximately 43^{o} southeast.

Further southeast from the hanging wall, the lithologies are mapped and logged as gneisses and foliated older granites. Due to the strong degree of alteration, it is difficult to identify the intrusive baked contact metamorphism of the older granites.

Several hundred metres to the southeast or stratigraphically upwards, the lithological units appear to consists of slightly to strongly hematitized, locally pyritiferous, aphanitic to fine grained arkosic metasediments striking in the similar direction as the above units. The dip of these arkosic beds appears to be approximately 25^{o} to the southeast.

A topographic low is located to the north of these units and may represent an erosional or faulted unconformity which may be occupied by a metagabbroic intrusive.

The gold mineralization is associated with fine grained pyrite generally consisting from 0.5% to 3.0% in the wallrock and with the quartz and quartz carbonate veining. Some of the auriferous quartz veins trending at N010^{o}E have associated sulphide mineralization, in order of dominance as, pyrite, galena, sphalerite and chalcopyrite.

It appears that a narrow unit of the BMG at or near the hanging wall exhibits brittle-ductile shearing parallel or near parallel to the strike of the lithological unit with the development of two generations of quartz veining represented by shear and extensional en echelon gash veining. The rotated veining probably indicates the veining event was intruded during progressive shearing.

Several quartz stockwork breccia zones cross cuts the BMG and may be the result of parallel fluid inflow into dilatant jogs in a dextral strike-slip fault system with faults step to the right. These late N315^{o}E trending faults are typically identified by vuggy quartz and/or pink carbonated breccia zones. Breccia zones have been observed in Trench No. 6, southwest of Trench No.7 (southeast of the Shaft No.1), south of the adit located approximately halfway between Shaft 1 and Shaft 2, and approximately 50 metres to the northeast of the adit. These northwest trending fault breccia zones, cross cuts and probably displaces earlier suspected N010^{o}E trending step faults.

A relatively flat dipping quartz veining system is located near the southern extent of the adit. This vein is striking near parallel to the footwall contact of the BMG, dipping approximately 200 m northwest to north-northwest. The quartz vein contains pyrite and arsenopyrite.

7.2.2 ATHONA DEPOSIT

The gold bearing zones at the Athona Deposit are from west to east: the eastern portion of the West Mine Granite, the Athona Deposit Granite, the Pond Zone, and in a prominent en echelon and bouginage quartz vein system of the East Zone. The underground mine development was concentrated in the western portion of the Athona Mine Granite and the eastern quartz vein systems (H, I, J, K veins) on the 38 and 76 m (125 and 250 ft) levels. The Athona West Granite (AWG) is a medium to coarse grained, reddish hematitic altered granite, dipping moderately westwards, containing fracture filling, quartz veining within the footwall sheared contact or mylonite zone. The unit is underlain by the central gabbroic to amphibolitic intrusive which separates the AWG from the Athona Mine Granite (AMG).

The AMG is porphyroblastic with similar amounts of potassium feldspars and plagioclase. This unit may represent a complex multi-intrusive with variable composition or a metamorphosed sequence of sedimentary lithologies. The northern portion of the original mining claim has exposures of quartzite. The auriferous sulphides are contained within the AMG and in the fracture/shear zone filled with quartz veins trending N20℃E and dipping 80° west. The sulphides are less than 1% fine grained pyrite, trace amounts of galena and sphalerite with minor amounts of pyrrhotite. This zone appears to be shallow, dipping to the west and is underlain by a thin gabbroic sill which separates deeper, coarse grained granite locally termed "tombstone granite". The Pond Zone appears very similar to that of the AMG.

The major quartz vein systems containing the H, I, J and K veins are located at or near the eastern extent of the AMG. The veining has been traced from surface to below the second level for a strike length of approximately 100 meters. The vein set is a combination of en echelon and boudinage veins trending approximately N10℃E and dipping 72° east. Further to the southeast, a parallel to sub-parallel vein sets, L and M veins, appear to be contained within a north-northeast shear/fault zone which may extent towards the vicinity of Shaft 2.

The mineralization occurs within a grey to pink-red megacrystic leucogranite, the 'Athona granite'. The granite body is apparently conformable with surrounding rocks that strike N to NW and dip westerly. The granite is in two parts, separated by an amphibolite (the 'Upper gabbro'). The structurally lower part overlies a further amphibolite body (the 'Lower gabbro'). Underlying the `Lower gabbro' is porphyritic fine-grained granite. NW-SE trending diabase dykes cut the granite and amphibolite. On the east side of the peninsula is a NNE-trending, easterly dipping reverse fault.

The Main zone is approximately 24 m wide and appears in plan as a zone parallel to the granite (Upper gabbro) contact. It is characterized by numerous parallel quartz veins oblique to the strike of the contact. The veins are rarely more than 8 cm wide. They trend NNE and dip 80° west. The veins are very persistent and can be traced up to 120 m. Narrow quartz stringers commonly branch from the main veins in various directions. The veins occur almost exclusively in the granite. Where they intersect the granite (Upper gabbro) contact they pass into the overlying amphibolite, but pinch out within a few centimeters.

The total percentage of metallic minerals is low, possibly not more than one percent. Sphalerite, galena, pyrite, gold and rare chalcopyrite occur in the quartz veins and the granite. In the granite, pyrite is the only common sulphide. Gold in the granite is commonly associated with 'clots' of chlorite; more rarely it fills fractures in feldspar. In the quartz veins sphalerite and galena predominate over pyrite and chalcopyrite. They commonly occur as 'lumpy' aggregates.

Sphalerite and galena generally indicate high grade of local extent. Most of the gold is fairly coarse and occurs alone or with the sulphides in the quartz veins. There is a marked enrichment in gold along the amphibolite contact. The East zone comprises veins similar in attitude and mineralogy to those of the Main zone. In contrast to the Main zone, the East zone exhibits two additional sets of quartz stringers locally. They apparently do not contain gold. Pyrite, galena, sphalerite and gold occur in the main quartz veins. These minerals, particularly galena and gold, are frequently present as replacements in the wall rock along fractures adjacent to the main veins. The latter fractures are not quartz-filled. The East zone exhibits ore in two shoots separated by a low grade section. In the higher grade shoot the main quartz vein (up to 10 cm wide) carries more than 3 ounces per ton gold (Beavan, 1938).

7.2.3 GLACIAL SEDIMENTS

The thickness of glacial deposits over bedrock varies from a few centimetres thick to about a few metres thick in low depression areas. Thicker overburden cover is present in areas lacking outcrops, around lakes and in valleys. Swampy areas in valleys are often composed of humus and peat layers that could measure a few metres to several tens of metres in thickness.

Soils present in areas of high relief and percentage of bedrock exposure are mainly composed of till, sand and silt of various colours (grey, beige, orange, brown and red). Their distribution is discontinuous and patchy. The thickness of till, sand and silty soils vary generally from a few centimetres to tens of centimetres near fractures, faulted bedrock and on some bedrock steps. Tills are rare, being generally of grey to pale brown colour, and contain a few cobbles in a clayey matrix. Sand and silt soils are mainly beige, brown, and orange to red. Most of these soil types are transported materials mainly of glacio-fluvial to lacustrine origin. However, a component of the reddish and brown soils may be locally derived from the underlying bedrock based on soil particles observed near the contact with bedrock.

8 DEPOSIT TYPES

8.1 INTRODUCTION

The following is an excerpt from Jensen, 2003:

> *Kian Jensen, with the assistance of the geological personnel of Greater Lenora Resources, developed geological models based upon the known gold and gold-platinum group metal occurrences. These models will assist in the future exploration activities within the Goldfields-Beaverlodge area and to the Greater Beaverlodge area. The understanding of the mineralization and the controls that govern the mineralization will aid in the development of exploration targets and potential of the area.*
>
> *Mason (1987) has suggested that the Nicholson Bay - Fish Hook Bay areas are mineralogically similar to that of Coronation Hill deposit in Australia. This model has a central core of higher grade uranium bearing mineralization surrounded by a zonation of gold and gold-platinum group metals. To date, this is an excellent concept and valued exploration guide line. However, a great deal of data compilation will be required to define the limits of the mineralogical zonation.*
>
> *It has been suggested that the large area of the Martin Formation may have several similarities to the Olympic Dam type model. Unfortunately, this area has not received the exploration activity required to confirm this suggestion and future exploration activities are warranted.*
>
> *The genesis of the uranium deposits within or near the present limits of the Athabasca sandstone limits are well documented. It is worth noting that the current limits of the Athabasca sandstone near the north shore of Athabasca Lake has been moved northwards in the area around Nicholson Bay and Fish Hook Bay areas. The new locations of outliers of the sandstone explains the location and the depositional nature of the gold and platinum group metals associated with the pitchblende deposits of simple and complex mineralogy.*
>
> *The exploration activities and the compilation of the available information have resulted in the development of the geological environmental models for gold and gold-platinum group mineralization. It is apparent that some of the mineral occurrences may have similarities to more than one model type and indicates the complex geological setting involved and that several gold mineralization events exists for the Goldfields -Beaverlodge area.*

8.2 BOX MODEL

This model is based upon directed geological evidence that the mineralized zone is an altered, granitized coarse grained metasediments and limited granitic intrusives with structurally controlled quartz veining. At least three structural events have occurred to create the hydrothermal conduit system. High gold assays are generally associated with N315^{o}E and N010^{o}E trending quartz veining with sulphide mineralization of pyrite, galena and sphalerite. Along with the two high grade quartz vein orientations, at least three additional quartz vein orientations have been recognized at the Box Mine. This model represents a deposit on the flank of a synclinal fold with the primary faulting/shearing event parallel to the lithological contacts. Other similar type deposits may be located on either the west or east flanks of the Goldfields syncline with similar lithological units or in nearby areas with synclines with faulting parallel or near parallel to the fold axial plane.

8.3 ATHONA MODEL

It appears that the Athona Deposit is spatially related to zones of strong northerly trending faults that are sub parallel to fold axial planes. These faults may have been active during several events. Also, this represents gold deposition near the nose of the synclinal fold. The strongest shearing is located between the West Mine Granite and the Upper Gabbro. The dominant sulphide mineralization is pyrite not only within the quartz veining but also associated with the host rock. Associated sulphide mineralization is pyrrhotite, galena and sphalerite. It has been suggested by several authors and confirmed by Jensen that the Pond Zone is an altered and weakly granitized quartzite.

9 EXPLORATION

9.1 DC/IP GEOPHYSICAL SURVEY, 2010

In early 2010, Brigus, as Linear, retained Quantec Geoscience Ltd. (Quantec), of Toronto, Ontario to carry out a Titan24 DC/IP geophysical survey over Box, Athona and the surrounding area.

Grid lines were established by Durama Enterprises of La Ronge, SK, and consisted of 58.7 line kilometers across 14 lines oriented at 118° Az as shown in Figure 9-1.

Each line was surveyed using a dipole size of 100 m, with a spread length of 4 km. Each Titan24 spread was surveyed using a pole-dipole configuration. The IP survey was conducted over 803 grid stations at 50 m spacing.

The purpose of the DC/IP survey was to detect zones, and define structures, related to hydrothermal alteration conducive to the emplacement of gold mineralization. This survey is capable of detecting such zones and structures to depths of 750 m with DC resistivity and IP chargeability.

Results of the DC/IP survey, as reported by Quantec (2010), states that, “the ‘geo-electrical’ characterization of BMG and Athona shows that potential mineralization could be located close to or within strong high resistivity zones with moderate to high chargeability values.”

Analysis of the resistivity and chargeability results show several moderate to strong IP anomalies associated with high resistivity zones to a depth of 700 m from surface in all profiles (Quantec, 2010). Eight high resistivity anomalies appear to be related with felsic granitic intrusives or quartz veins that have been identified in the property. Quantec found that the correlation of results from line to line is generally good although the final location of interpreted anomalies require constrained inversion results. Current results present smooth models of resistivity and chargeability distribution.

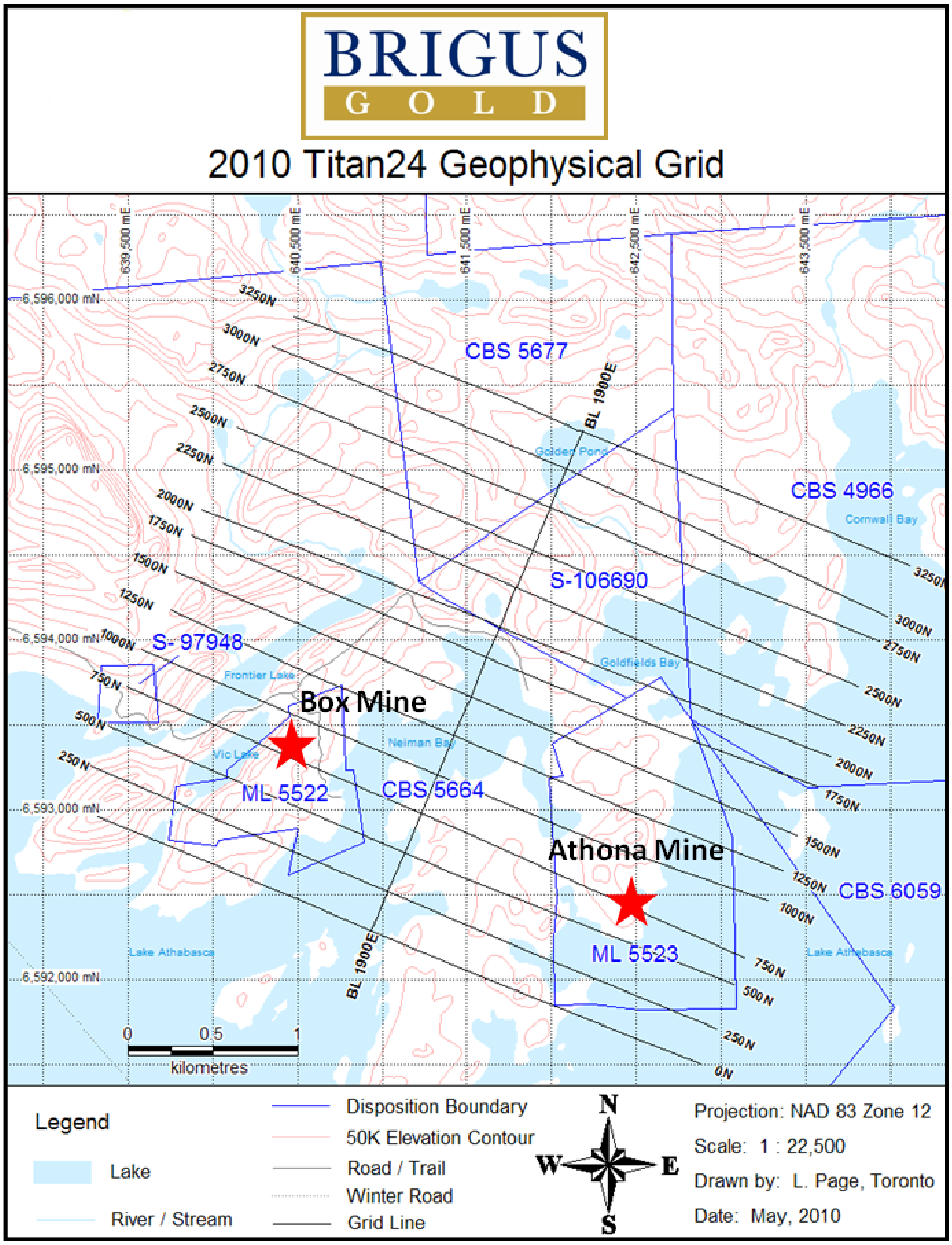

Figure 9-1: DC/IP Geophysical Survey, 2010

10 DRILLING

10.1 BRIGUS, 2010

Brigus, then as Linear, conducted a drill program in the Goldfields Claim Group from January to March 2010. The drill program was established to bring areas of lower drill sample data, at the base of Box, into the database for a future resource estimate update, and to test extension along strike of Box.

The drilling carried out by Silverton Drilling, a Uranium City based company, began on January 22, 2010 and was completed by March 27, 2010.

The 2010 drill program consisted of 12 NQ size diamond drill holes totalling 2,858 m. Five (5) of the 12 drill holes intersected the BMG, for a total of 1,333 m of drilling, and encountered several occurrences of elevated gold values. The drill holes are summarized in Table 10-1 and the drill hole location is shown in Figure 10-1.

Table 10-1: 2010 Drill Hole Summary

Drill Hole	Grid Northing	Grid Easting	Elevation (m)	Bearing (°Az)	Dip (°)	Length (m)
B10-303	-365.57	-317.61	227	0	-57	262.9
B10-304	-306.52	-326.12	224	0	-60	267.0
B10-305	-453.25	-338.49	235	0	-60	284.5
B10-306	-228.41	-314.09	231	0	-71	253.0
B10-307	-78.58	-327.84	214	0	-65	266.0
B10-308	1181.30	-12.43	251	339	-55	167.0
B10-309	1455.92	5.26	247	340	-55	413.0
B10-310	1040.15	-325.08	211	335	-84	189.0
B10-311	1222.60	-19.45	254	335	-60	176.0
B10-312	1222.60	-19.45	254	335	-75	129.0
B10-313	1219.66	2.40	254	338	-54	152.0
B10-314	17.19	-353.40	211	335	-89	299.0
Total						**2,858.4**

*Note: Highlighted drill holes intersect Box deposit.

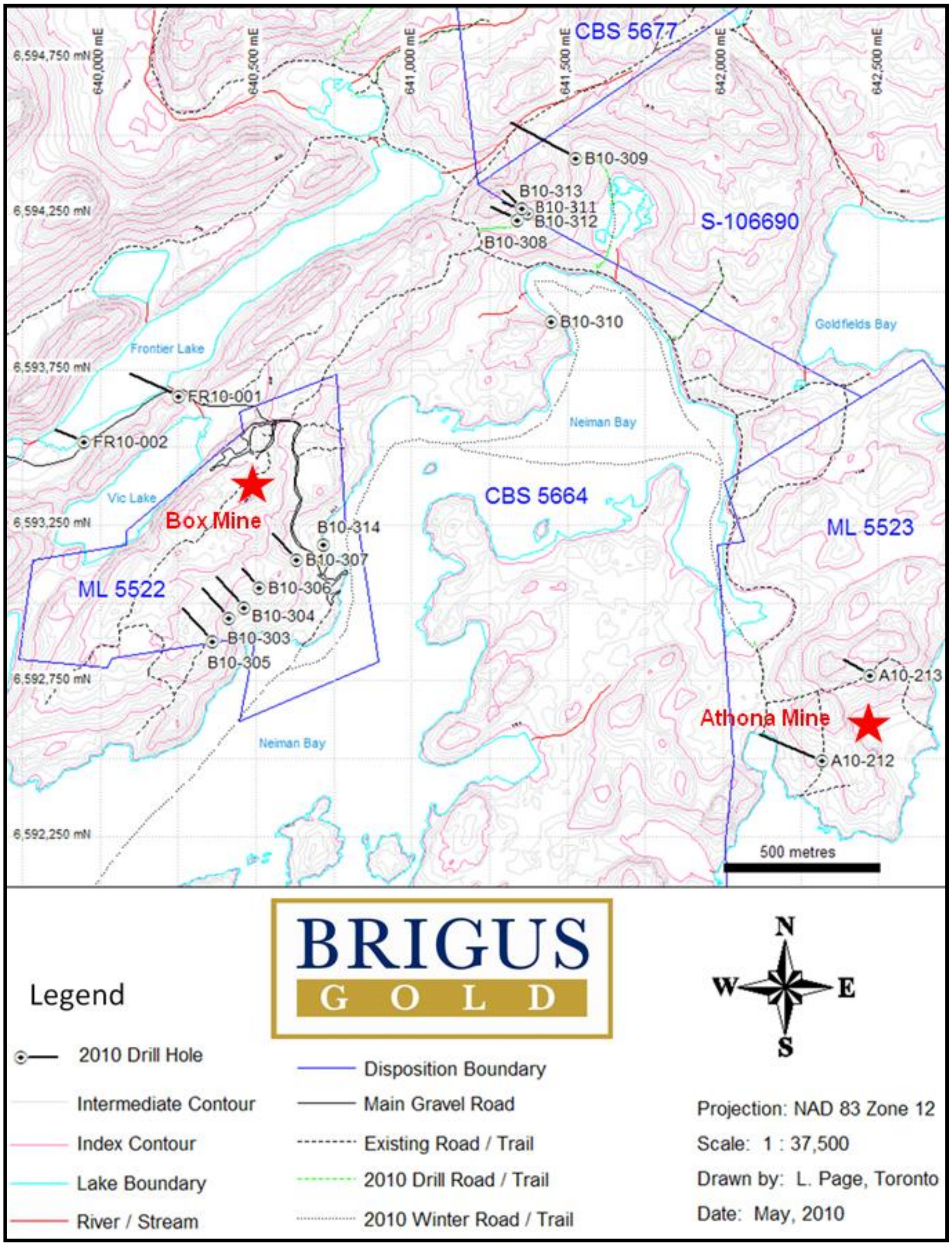

Figure 10-1: Box 2010 Drill Hole Location Map

10.2 PIEZOMETER MONITOR DRILL HOLES, 2010

The drilling of piezometer monitor holes began on April 6 and was completed on April 17, 2010. A total of 19 drill holes, HQ size, vertically drilled diamond drill holes were completed totalling 482.4 m. Table 10-2 summarizes the piezometer drill holes and drill hole locations are shown in Figure 10-2.

This phase of drilling was also carried out by Silverton Drilling of Uranium City.

Conditioning and sampling of these wells was conducted from April 18 to April 22, 2010. The water samples collected were shipped toSaskatchewan Research Council Environmental in Saskatoon for analysis.

Table 10-2: Summary of Piezometer Drill Holes, 2010

PZ_Hole_ID	East_83	North_83	Elev_m	Length_m
WM101A	640859	6594647	216	47.0
WM101B	640861	6594647	219	12.0
WM102	641495	6594593	253	13.5
WM103A	641247	6594860	238	42.0
WM103B	641244	6594860	238	37.5
WM104A	640932	6595000	255	27.0
WM104B	640993	6595168	261	12.0
WM104C	641004	6595165	263	42.0
WM105	641650	6595175	275	11.8
WM106A	641775	6594772	265	13.8
WM106B	641779	6594767	271	39.3
WM107	641929	6594913	247	14.8
WM108A	641994	6594083	241	24.5
WM108B	641989	6594071	238	14.3
WM109	641628	6594060	216	14.7
WM110A	641086	6594414	219	21.0
WM110B	641087	6594414	215	12.0
WM111	640776	6593564	225	14.2
WM112	639874	6593422	228	15.0
Total				428.4

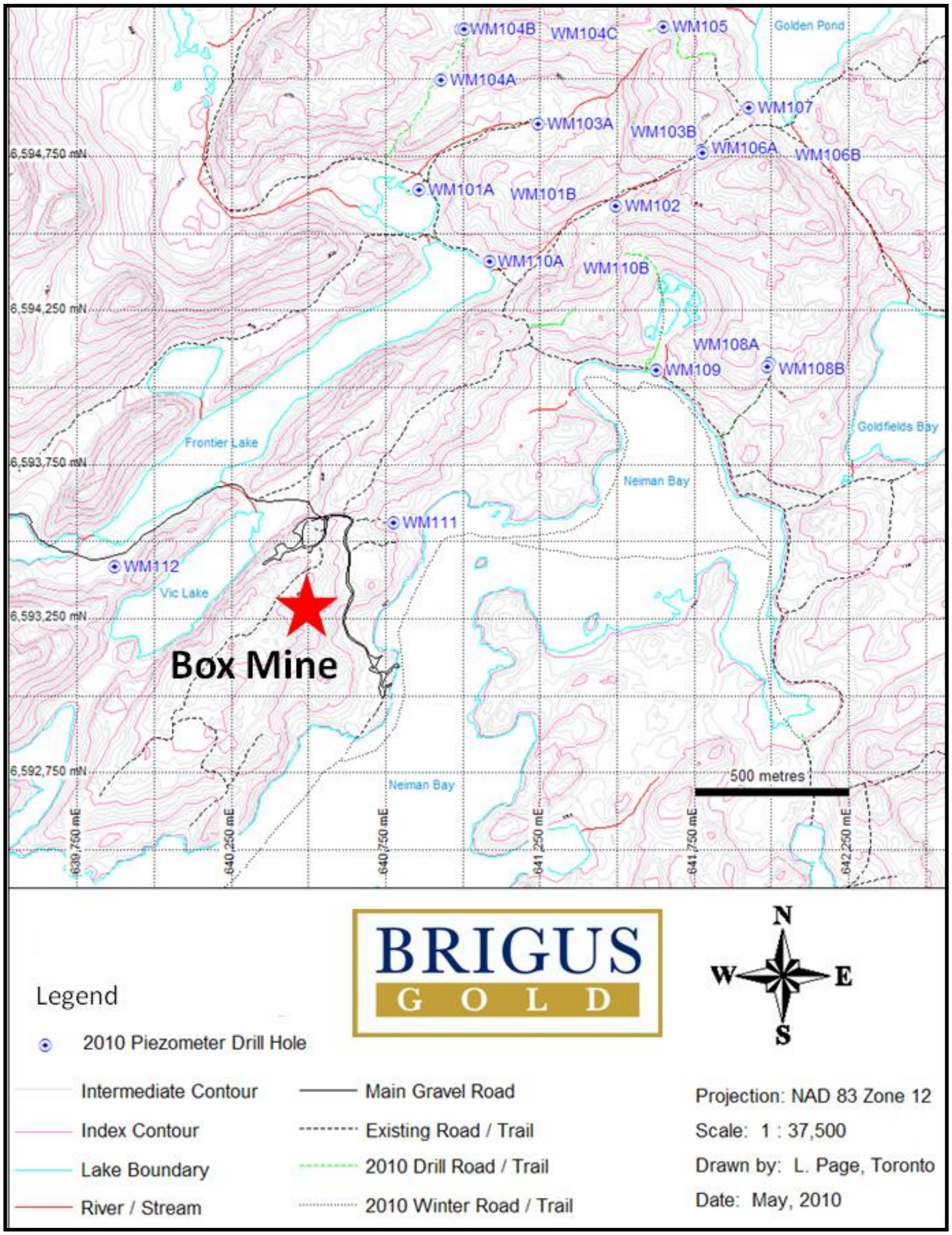

Figure 10-2: 2010 Piezometer Drill Hole Location Map (Brigus, 2010)

10.3 ATHONA 2006 DIAMOND DRILLING PROGRAM

No drilling has been completed by Brigus at Athona. The last drilling program was conducted by GLR totalling 1,592 m, on Mining Lease – ML 5523. The diamond drilling program consisted of 16 NQ core size drill holes.

The purpose of the 2006 summer drilling program was to bring Athona up to NI 43-101 standards. As outlined in the 1995 Simons Engineering Study, Athona contains 232,900 ounces of gold in 3.62 million tonnes at a grade of 1.99 g/t. These resource estimates are historical and do not meet the standard terminology criteria of the NI 43-101 or the CIM Standards on Mineral Resources and Reserves. Wardrop has not completed the work necessary to verify this historical estimate. The estimate is no longer relevant and has been replaced by the current estimate prepared by Wardrop outlined in Section 14 of this report. In Wardrop's opinion, the historical estimate should not be relied upon as it is superseded by the current NI 43-101 compliant resource in this report which incorporates the sixteen drill holes completed in 2006.

Wardrop selected 10 drill hole locations for a confirmation diamond drilling program. This program was performed in order to provide confirmation of grade, geology and spatial continuity of the Athona.

In addition, six (6) exploration drill holes were completed to test the extension of gold-bearing quartz veins located outside of the open pit area in order to expand the potential resource.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 BRIGUS, BOX DRILL PROGRAM 2010

For the 2010 drill program, the drill core was removed from the drill rig and placed in core boxes. The boxes were covered and sealed as soon as the run was completed. The drill core was trucked from the project site to Brigus' core logging/sampling facility/office in Uranium City twice daily by the crew on each drilling shift.

The core boxes were laid out on tables where the drill core was measured and marked by a geotechnician or geologist. The geotechnician performed the geotechnical logging which included: percent recovery, percent of fracture frequency, and rock quality designation.

The drill core was logged and described by the geologist that included: Lithology, alteration, mineralization, percent veins, and core and vein angles. All drill cores were sampled at 1 m intervals.

Sampling of the drill core was generally restricted to the BMG lithological unit and to the granitic hanging wall above the BMG. Sample numbers were written on the core box and a sample tag inserted at the start of the sample. The sample intervals were entered into an excel spreadsheet by the geologist after sampling of the hole was completed.

The drill core was photographed in a wet and dry state, by the geotechnician after the core was logged by the geologist and before the core was split.

The core was sawed in half by the geotechnician or geologist. Sample bags were pre-labelled with the sample number and half of the sample tag placed into the bag after the sample was cut. The other half of the sample tag was stapled in the core box. The sampled halves of the drill core were shipped to TSL Laboratories in Saskatoon and the remaining halves returned to the core box. The sample bags were sealed immediately after the half core and tags were inserted.

A blank sample, CDN-BL2 and grading less than 0.01 g/t Au, and; a standard sample, CDN-GS-2B and grading 2.03 ± 0.12 g/t Au, were both inserted within each 20 sample run. Both blank samples and standard samples were purchased from CDN Resource Laboratories Ltd., based in Delta, British Columbia.

Sample bag and sample tag numbers were double checked by geologist. Seven to eight individual samples bags were put into a larger rice bag, on which the sample numbers were identified with a series number, and sealed. A standard TSL Laboratories sample submittal form containing the sample numbers and assaying method was completed and inserted into the first rice bag of the shipment. This submittal form was also emailed to the lab on the day of shipment.

The rice bags were trucked by the geologist to the Pronto Airways office, in Uranium City, where they were weighed and driven to the Uranium City airport by a Pronto Airways official. This is the procedure for all cargo being flown out of Uranium City. The samples were flown out of Uranium City and transported to TSL Laboratories in Saskatoon on one of two weekly flights. Typically, the sample shipment preparation was done the morning of the flight to Saskatoon.

11.2 GLR, ATHONA DRILL PROGRAM 2006

See the Wardrop NI 43-101 compliant technical report for documentation of the sampling method and approach (Wardrop, 2007).

11.3 BRIGUS, BOX MINE

Samples taken at Box in the 2010 drill program were assayed by the Screen Metallic method. Since Box is a high nugget, or "nugget", deposit, the Screen Metallic method is known to be more effective for determining the gold value as both positive and negative fractions of the mesh are assayed. The Screen Metallic method eliminates the possibility of a "smearing" effect in the assay result where, due to the malleable nature of native gold, grains do not pass through the mesh and the sample is reported with a lower grade. The "smearing" effect does not render a representative sample result.

The Screen Metallic Assay includes:

- Crushing the entire sample
- Pulverizing the entire sample to 95% passing 150 mesh
- Screening the entire sample through 150 mesh
- Assaying the entire +150 mesh fraction
- Duplicating the -150 mesh fraction assay
- Determining the weighted average of gold for entire sample

Upon return of the assay certificates and results, the geologist inputs the assays into the Excel spreadsheet containing the sample intervals and a quality assurance check was made on the blank and standard samples.

The status of the sample shipment was updated on a separate excel spreadsheet. The drill hole logs were entered into GEMSlogger software program after the drill program was completed.

11.4 GLR, ATHONA

See the Wardrop NI 43-101 compliant technical report for details regarding the sampling preparation, analyses and security (Wardrop, 2007, now filed under "Mistango River Resources Inc.").

12 DATA VERIFICATION

12.1 BOX MINE

Prior to the resource estimate update, Wardrop verified the collar, survey and gold assay data from the 2010 drill program to determine the reliability of the data. The drill holes verified were: B10-303, -304, -305, -306 and -307.

Wardrop verified the assay data of the five 2010 drill holes from the original laboratory assay certificates from ActLabs. No errors were observed.

A site visit was conducted by Mr. Paul Daigle, on May 11 and 12, 2011. Mr. Daigle was accompanied on the site visit by Mr. John Dixon, Exploration Manager, and Mark McLaren, Project Geologist, with Brigus; and Calvin Andreas, Civil Engineer in Training with March Consulting.

The site visit included an inspection of Brigus' core storage/logging facility in Uranium City and Box gold project site, that is the former Box mine and the five drill sites from the 2010 drill program.

The core logging and sampling facility used by Brigus is located within Uranium City. The warehouse is well suited for the storage, logging and sampling of the drill core and is insulated for year round use.

The drill core is stored in wooden core boxes and the core boxes are stacked on well-built core racks in the facility. The core boxes present in the facility are from the 2010 drill program and that intersect the mineralized zone of Box. Drill core from Brigus' other drill programs were also stored here.

Selected drill core from 2010 drill program were inspected by Wardrop and compared to the drill hole database in the GEMS model. Sample tags are stapled at the end of each sample interval with the sample number imprinted and the drill hole number and the sample interval written in ink. Some sample tags were noted as having been wet where the drill hole number and sample interval were missing but where the sample number was still visible. Table 12-1 shows the drill core intervals that were verified.

Table 12-1: Summary of Drill Core Intervals Inspected by Wardrop

Drill Hole No.	Drill Core Box(es)	Interval		Sample No.	Grade (g/t) Au
		From (m)	To (m)		
B10-303	35	201.5	202.5	278665	31.04
B10-304	34 , 35	194.0	195.0	278521	12.99
		199.9	200.9	278528	37.20
		204.9	205.9	278533	31.41
B10-305	38, 43	217.7	218.7	278599	5.48
		220.7	221.7	278602	1.40
		245.7	246.7	278630	1.22
B10-306	37,38	217.0	218.0	278738	2.78
B10-307	41	223.0	224.0	278786	3.81
		224.0	225.0	278787	6.40

12.1.1 DRILL COLLAR COORDINATE ERRORS

At the Box project site, drill collars from the 2010 drill program were located and their coordinates recorded by a handheld GPS. Overlaying the five 2010 drill holes over the GEMS model found errors of up to 7 m (drill hole B10-304). As the area is tree covered, the GPS unit recorded the coordinates with an estimated error of ±8 m.

Figure 12-1 below shows a plan view of the 2007 (circles) and the 2010 drill holes (squares) superimposed over the corresponding drill holes in the GEMS database. The GPS coordinates taken during the site visit of the 2007 drill holes show a maximum error of approximately 15 m; the 2010 drill holes show a maximum error of 7 m.

The verified collar coordinates of the 2010 drill hole collars appear within the error margin of a handheld GPS unit. However, the drill hole collar coordinates of the 2007 drill holes do show a discrepancy in the drill hole survey.

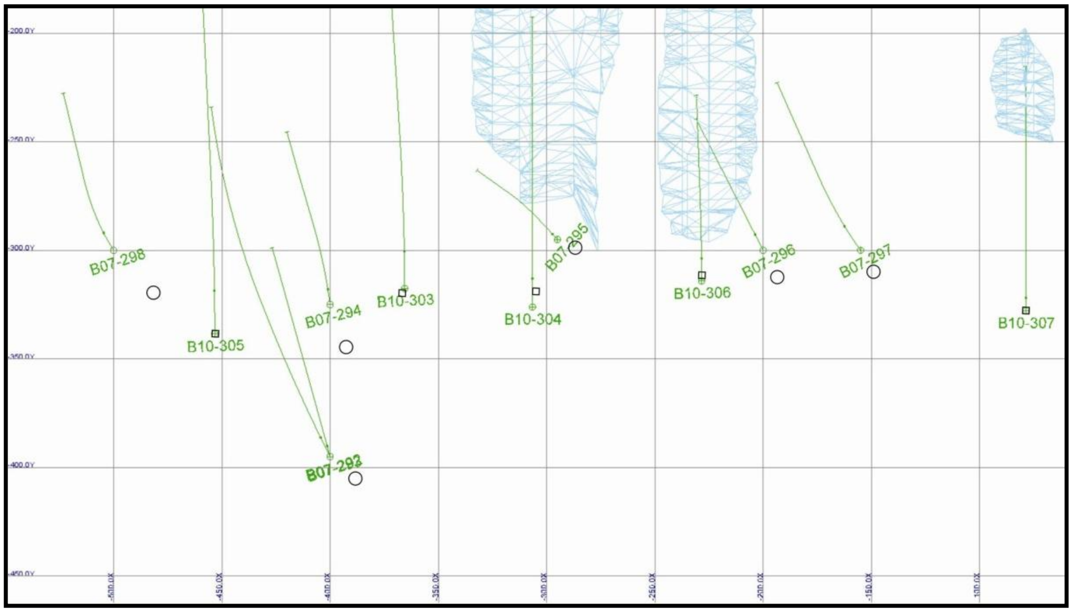

Figure 12-1: Plan View of 2007 and 2010 Drill Hole Collars Comparing GEMS Drill Hole Collar Coordinates (grid scale 50 m x 50 m)

In addition, March Consulting encountered an issue with 13 of the 2007 drill hole collar coordinates. These errors were found in relation to the surveyed Universal Transverse Mercator (UTM) coordinates and their mine grid coordinates. When March Consulting calculated the drill holes locations from one location, using the holes' mine grid and UTM coordinates, if there was a difference, then there would be a discrepancy between the mine grid and UTM coordinates. March Consulting calculated discrepancies of up to 75 m.

Five of the 13 errors noted occur outside of the Box pit. These drill holes showed the largest coordinate deviations between 42 m and 75 m. Wardrop did not identify these errors as they occur outside the Box pit and therefore do not affect the resource estimate.

March Consulting found discrepancies with the remaining eight drill holes' coordinates ranging between 0.9 m and 7.3 m. According to Brigus, the 2007 drill collars were originally surveyed by handheld GPS. March Consulting surveyed nine of the 2007 holes and the discrepancies of their UTM coordinates ranged from 0.55 m to 12.2 m. March Consulting had also surveyed two (2) of the 2010 holes and these holes had only differences of 0.581 m and 1.159 m. As noted in Wardrop's check of the coordinates by handheld GPS above, an error of ±8 m is possible in verifying UTM coordinates with a handheld GPS, particularly where the 2007 collars were not easily identifiable or, in some cases, where tree cover hampers a more accurate coordinate reading.

Wardrop located and recorded seven of the eight 2007 drill holes that affect the Box pit by GPS. Drill holes B07-292 and -293 were drilled from the same collar location. All seven coordinates showed some discrepancy with respect to the GEMS model (Figure 12-1) with the maximum error of approximately 15 m (B07-298).

One source of the error may be that the 2007 drill hole collars were not easily located on the ground as the collar landmarks were not present. Also noted was that five (5) of the 2007 drill holes appear to plot directly on mine grid lines. This may be indicative that the drill hole coordinates in the block model are those of the proposed drill hole locations.

In the GEMS block model, these seven 2007 drill holes intersect the base of the Box deposit but are located outside the open pit wireframe, produced by Brigus (Choquette, 2010). Figure 12-2 illustrates the points of intersection.

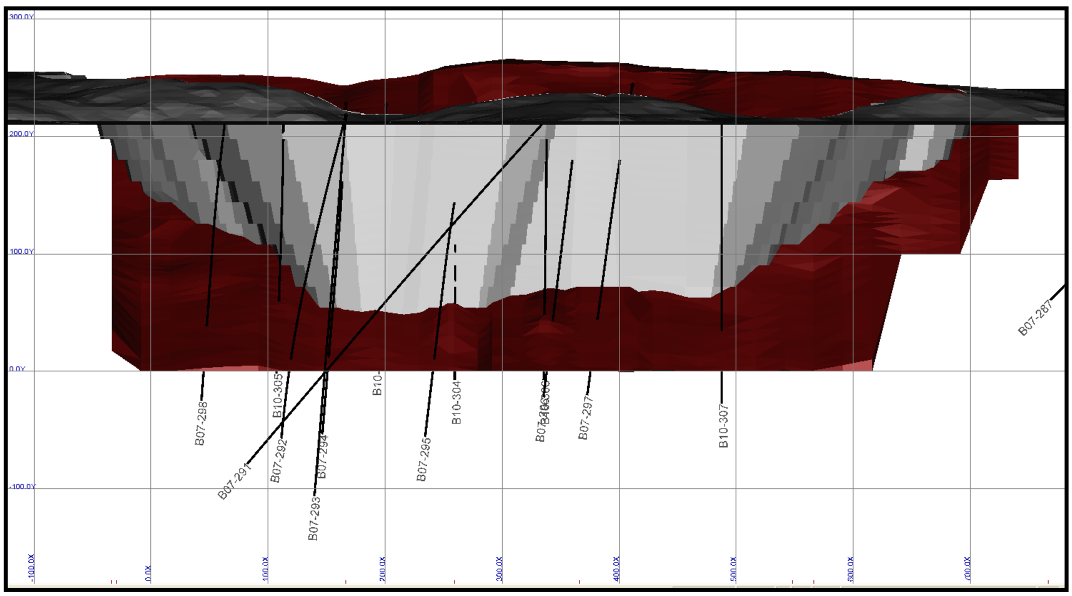

Figure 12-2: Box Longitudinal View

Wardrop is of the opinion that the corrected collar locations will have a minor impact on the overall resource estimate for Box. The effect of correcting the drill hole collars would occur on the block grades at the base of the open pit due to the change in sample support position and, thus, will change the weighting of the samples points used in the estimate. This will also produce a minor difference in the Indicated and Inferred Resources.

No block model update is recommended for the current phase of work. However, it is recommended that the discrepancies in the 2007 collar locations be corrected for any future resource estimate update.

12.2 ATHONA DEPOSIT

Wardrop completed a NI 43-101 compliant technical report in 2007. The data verification is included in Section 14 of that report (Wardrop, 2007).

Wardrop noted the following from the databases provided:

- *Various levels of precision were noted in the spatial coordinates, varying from 1 to 3 decimal places. The coordinates should be resolved in UTM coordinates with the appropriate level of precision.*
- *Multiple assay values were recorded as an average value. Wardrop recommends selecting an individual assay value based on an identified criterion and not to average multiple values.*
- *Gold assay values were noted in units of ppb, oz/st and g/t. Care should be taken moving forward to minimize errors arising from conversions between these units.*

In Wardrop's opinion, the data are useable for this resource estimation. However, more exhaustive diligence should be undertaken as this project moves forward to resolve these issues. If assay certificates are not found or core is not available for some of the historic drilling then additional confirmation drilling may be required.

Tim Maunula conducted a site visit to Athona from August 14 to 18, 2006.

Collar grid-UTM conversion issues were identified for Athona similar to those encountered at Box. Using Wardrop's field notes and GPS readings, a new grid conversion was developed.

The average distance differential between the mine grid and calculated UTM coordinates is -4.1 meters, ranging from -2 meters to -6.4 meters. As the block size is 5m x 5m x 10m, the amount of offset in the conversion should be within the block or immediately adjacent.

Wardrop's recommendation at this point is to use this grid conversion and before moving to feasibility study complete an updated survey of the mine grid to obtain better resolution.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Although some of the gold is associated with pyrite, in the ore from Box, test work has indicated that the recoverable values are finely disseminated with some nugget gold. There are no metallurgical or environmental hindrances associated with the mineralization in terms of recovery. Extensive test work dating back to 1936 has determined that the gold can be recovered by a variety of gravity and leaching methods. Fluctuation of gold prices from the mid 1990's has generated additional investigations, resulting in the current flow sheet that uses gravity, flotation, and concentrate leaching methods.

In order to confirm the current flowsheet and to move the project to detailed design stage, Brigus is conducting additional test work as outlined in Section 13.2.17 On Going Test Program

13.2 HISTORICAL TEST PROGRAMS

13.2.1 WHOLE ORE CYANIDATION 1930'S

Cominco originally conducted test work for the Box ore, when the plant was operated from 1939 to 1942 with underground ore. Whole ore cyanidation with 24-hours retention and Merrill-Crowe precipitation on zinc was employed from ore ground to 55% - 200 mesh. Recovery is reported as between 94 and 98%.

13.2.2 HEAP LEACHING VERSUS FLOTATION 1981

In 1981 in order to compare heap leaching with flotation for ore from Athona, Dawson Laboratories conducted leach tests on ore crushed to ¾" and flotation tests on ore ground to 50% - 200 mesh. Recoveries were about 20% for the heap leach samples and 92 to 97% for the flotation test.

13.2.3 GRAVITY CONCENTRATION 1988

In 1988 Casmyn Engineering conducted test work at ORTECH that showed that gravity concentration was effective in recovering gold.

13.2.4 VAT LEACHING 1988

In a scoping test conducted by INNOVAT Limited in 1988, ore crushed to ¼" yielded 69% recovery by cyanidation in a simulated vat leach.

13.2.5 1995 TEST PROGRAM FOR GREATER LENORA RESOURCES CORPORATION

Richard C. Swider Consulting Engineers Limited conducted a review of the metallurgical test programs carried out in 1936 by the former operator (Cominco) and in 1988 by Casmyn Engineering. The review showed that gravity concentration was effective in recovering and concentrating gold. Test data also indicated that a constant tailing loss might be expected over a relatively wide range of ore grades.

A new test program was recommended using a 40-tonne sample from the 1988 sampling trenches of Box and a 20-tonne bulk sample from the Athona stockpile.

Three drill core samples from the Box and two drill core samples from Athona were composited on a bench level basis to provide samples for metallurgical testing at depth and at grade.

Richard C. Swider Consulting Engineers Limited supervised the testing listed below and conducted at Lakefield Research of Ontario.

- Pebble grinding
- Preliminary demonstration of gravity concentration (tabling)
- Preliminary demonstration of the suitability of spirals for gravity concentration
- SAG mill pilot plant operation
- Continuous operation of spiral concentrations in conjunction with the SAG mill pilot plant
- Initial concentrate cyanidation

13.2.5.1 PEBBLE GRINDING TEST

The samples tested were obtained from surface trenches at Box and from the surface pile at Athona. The pebble grinding test work showed that both Box and Athona ores would make ideal pebbles for grinding.

13.2.5.2 TABLING OF BOX AND ATHONA ORES

The samples from the pebble grinding test were used for the preliminary tabling tests to determine the effective grind size and tailings losses.

For the Box ore, the suitable grind of 80% passing size of 350 microns was established and a tailing of 0.16 g/t was indicated for a single pass circuit. Overall data suggested that tailings of 0.16 g/t in a concentrate of less than 10% of feed weight were possible.

For the Athona ore, a similar grind was indicated although a slightly higher tailings of 0.24 g/t was indicated in a concentrate weight of less than 10%.

13.2.5.3 BATCH TESTING OF SPIRAL CONCENTRATOR

The tests using a mixture of Box and Athona ore as well as individual ores showed that acceptable tailings could be achieved with two stages of spirals or equivalent to roughing and scavenging.

13.2.5.4 SAG MILL PILOT PLANT

Approximately 40 tonnes of Box ore and 20 tonnes of Athona ore were treated through the SAG pilot plant. The ore samples, although not representative of overall ore grade were considered suitable for the SAG pilot plant.

Both ores were found to be suitable for single stage SAG milling to produce P80 of 350 microns. This particle size was previously found suitable for spiral gravity separation.

It was found that the use of two-stage grinding configuration was not necessary as previously determined.

This test work indicated the following Bond indices as summarized in Table 13-1.

Table 13-1: Bond Indices Summary

	Bond Ball Mill Index	Bond Rod Mill Index
Box ore		
Current	15.1	15.6
1988 Program	16.3	
Athona ore		
Current	16.4	16.3
1988 Program	16.4	

13.2.5.5 SPIRAL PILOT PLANT

A spiral circuit with a rougher, secondary, and scavenger stage for gold recovery and a cleaner stage for concentrate upgrading was found to be the preferred alternative for both ores. The results are summarized in Table 13-2.

Table 13-2: Spiral Pilot Plant Results

	Box Ore	Athona Ore
Tabling Tailings (g/t)	0.16	0.24
Spiral Tailings (g/t)	0.14	0.25
Spiral Concentrate Wt. Fraction		
Open Circuit	9.4	
Closed Circuit		3.8

13.2.5.6 FLOTATION GRAVITY TAILINGS

The tailings from the batch spiral runs and from the tabling of “at grade” samples were used for flotation tests. It was necessary to reproduce the spiral rougher tailings since the scavenger products had been dried for sampling purposes.

For these tests, standard flotation conditions were selected using un-ground spiral tailings. The results from the Box ore flotation scavenger tests indicated that tailings of 0.10 g/t could be produced from a head grade of 1.5 to 2.2 g/t with a flotation concentrate weight of less than 2% and with a total concentrate weight of approximately 10%.

The reagents used for flotation were potassium amyl xanthate and Aero Promoter 208 as collectors and Methylisobutyl carbine (MIBC) as frother at minimal concentrations. A test without R208 gave equally good results suggesting that reagent dosage optimization is possible during plant operations.

Similar tests conducted on Athona ore indicated tailings of 0.21 g/t from a head grade of 1.5 to 2.2 g/t.

13.2.5.7 CYANIDATION OF GRAVITY CONCENTRATE

Both Box and Athona gravity pilot plant concentrate and a mixture of gravity and flotation concentrate have been leached with cyanide. The concentrate was ground to approximately 80% passing 50 microns, and for 48 to 96 hours with and without pre-aeration.

Gravity recovery of free gold ahead of cyanidation recovered 50% of the free gold. Result from some tests show that gravity removal of free gold ahead of cyanidation is essential for this ore to ensure low cyanidation tailings. A Knelson concentrator may be used for this purpose. Pre-aeration reduced cyanide consumption by 50%.

13.2.5.8 PROJECTION OF OVERALL RECOVERY VS HEAD GRADE

Box ore

- Gravity & Flotation Tailings (g/t ore) Projected
- Free Gold Recovery (%) 50%
- Cyanidation Tailings (g/t conc) Projected
- Soluble Recovery (%) 99.5%

Table 13-3: 1995 Test Program Box Recovery and Grade Summary

HEAD GRADE	GRAVITY AND FLOTATION TAILS	RECOV	RECOV	FREE GOLD RECOV	CYANIDATION FEED	TAILS	EXTR'N	RECOV	RECOV	TOTAL RECOVERY	
g/t	g/t	g/t	%	g/t	g/t	g/t	g/t	g/t	%	g/t	%
1.50	0.10	1.41	94.0	0.71	0.71	0.14	0.69	0.69	97.5	1.39	92.8
1.60	0.10	1.51	94.4	0.76	0.76	0.14	0.74	0.74	97.6	1.49	93.3
1.70	0.10	1.61	94.7	0.80	0.80	0.14	0.79	0.79	97.7	1.59	93.6
1.80	0.10	1.71	95.0	0.86	0.86	0.15	0.84	0.84	97.8	1.69	94.0
1.90	0.10	1.81	95.3	0.90	0.90	0.15	0.89	0.89	97.9	1.79	94.2
2.00	0.10	1.91	95.5	0.96	0.96	0.15	0.94	0.94	97.9	1.89	94.5
2.10	0.10	2.01	95.7	1.01	1.01	0.15	0.99	0.98	98.0	1.99	94.7
2.20	0.10	2.11	95.9	1.06	1.06	0.16	1.04	1.03	98.0	20.9	95.0

Athona Ore

- Gravity & Flotation Tailings (g/t ore) Projected
- Free Gold Recovery (%) 35%
- Cyanidation Tailings (g/t conc) Projected
- Soluble Recovery (%) 99.5%

Table 13-4: 1995 Test Program Athona Recovery and Grade Summary

HEAD GRADE	GRAVITY AND FLOTATION TAILS	RECOV	RECOV	FREE GOLD RECOV	CYANIDATION FEED	TAILS	EXTR'N	RECOV	RECOV	TOTAL RECOVERY	
g/t	g/t	g/t	%	g/t	g/t	g/t	g/t	g/t	%	g/t	%
1.50	0.21	1.31	87.4	0.46	0.85	0.21	0.83	0.83	97.1	1.29	85.7
1.60	0.21	1.41	88.2	0.49	0.92	0.21	0.90	0.89	97.2	1.39	86.6
1.70	0.21	1.51	88.9	0.53	0.98	0.22	0.96	0.96	97.3	1.48	87.3
1.80	0.21	1.61	89.5	0.56	1.05	0.22	1.02	1.02	97.4	1.58	88.0
1.90	0.21	1.71	90.1	0.60	1.11	0.23	1.09	1.08	97.5	1.68	88.6
2.00	0.21	1.81	90.6	0.63	1.18	0.23	1.15	1.15	97.6	1.78	89.1
2.10	0.21	1.91	91.0	0.67	1.24	0.24	1.22	1.21	97.6	1.88	89.6
2.20	0.21	2.01	91.4	0.70	1.31	0.24	1.28	1.28	97.7	1.98	90.0

13.2.6 VAT LEACHING 1997

Under the supervision of INNOVAT Limited in 1997, leaching of spiral classifier tailings and whole ore was conducted at ORTECH and Lakefield. Both programs indicated economical recoveries on ore crushed to 10 mesh.

13.2.7 DEWATERING AND ENVIRONMENTAL STUDIES 1996

A program in 1995-6 was conducted at Lakefield Research Limited with input from Pocock Industrial Inc. to determine settling and filtration characteristics of the ore and tailings. Ore characterization studies were made on whole rock, gravity tailings, flotation tailings, and cyanidation tailings, including EPA acid-base accounting, EPA leachate extraction, and size distribution of residues.

13.2.8 GEKKO TEST PROGRAMS 1998 – 2004

Work began on the process design in late 1998 and continued through to 2005 at Gekko Systems in Australia and Lakefield in Canada, using the Gekko test protocol. Primary and cleaner gravity recovery in Gekko inline pressure jigs with scavenging in a Falcon concentrator gave recoveries in the mid 80's for ore ground to a P80 of 500 microns, followed by regrind and cyanidation of concentrates. Concentrates produced were tested for leaching in the Gekko inline leach reactor with gold values from solution extracted by direct electrowinning from the ILR. Cyanide destruction, using peroxide, was also tested by Gekko.

13.2.9 ONGOING TEST PROGRAMS

Acid-Base accounting and cyanide destruction tests are continuing to back up the EIS as well as further assurance that targets will be met. Additional test work involves using the INCO SO2/Air process for cyanide destruction to confirm results and reagent requirements.

To move the project ahead, Brigus is conducting a 2011 metallurgical testing program to confirm and optimize the selected flowsheet. The proposed test works which will be overseen by EHA Engineering Ltd. and conducted at SGS Canada Inc. are briefly described in Table 13-5.

Table 13-5: 2011 Test Program Summary

Type	Tests	Description
Head Characterization	Including: Au by pulp metallics, S, ICP-OES scan and whole-rock analysis (WRA)	Each of the Box, Athona and four variability samples will have a head characterization performed
Grindability Testing	Drop-weight test (DWT) or SMC test	Two tests will be performed, one each for Athona and Box.
	Bond Rod Mill Grindability Test	Determining the Bond rod mill work index (RWI). Two tests will be performed, one each for Athona and Box.
	Bond Ball Mill Grindability Test	Determining the Bond ball mill work index (BWI). Two tests will be performed, one each for Athona and Box.
	Bond Abrasion Test	Determining the abrasion index. Two tests will be performed, one each for Athona and Box.
	High-Pressure Grinding Rolls (HPGR) Evaluation	Optional
Gold Recovery Evaluation	Gravity Recovery	Conducted using an MD-3 Knelson concentrator and Mozley table. One test for 75, 150, and 250 µm size fractions each for Athona and Box samples.

Type	Tests	Description
		(Six tests total)
	Flotation	Limited to rougher flotation with the sole purpose to produce concentrate for Cyanidation. Two flotation tests at each grind size (Twelve tests total)
	Cyanidation Testing	Two cyanidation test on direct ore, gravity tailing, and bulk rougher concentrate at each grind size (Fifteen tests total)
Gold Recovery Optimization and Variability Testing	Optimization Testing Variability Testing	Optimization test will evaluate numerous variables for improving flotation/cyanidation. Four variability samples using the determined optimal conditions will be tested.
Carbon Modeling	CIL/CIP Testing	Predicts the "optimal" gold absorption circuit operating strategy. Testing will produce CIL/CIP plant operating parameters.
Cyanide Destruction	Whole ore sample testing Solution will be analyzed for: CNT, CNWAD, SCN, SO4, Hg, ICP MS scan.	A single batch test will be performed on select leach residue to demonstrate effectiveness and to establish reagent requirements.
Solid-Liquid Separation and Rheology Testing	Test for elements in MMER Cyanide Test for elements in WAD Cyanide Test pH Test TSS	Determine the characteristics of the tailings.
Tailings Physical Characterization	Including: Size distribution, settling rate, pore water quality, terminal settling density, permeability	Determine the physical characteristics of the tailings.
Acid Base Accounting Testing		Determine the acid-neutralizing potential (assets) and acid-generating potential (liabilities) of the waste rock samples. Testing will be performed both on fresh and previous rock samples to obtain an understanding of how the rock will behave over time.

13.3 GEOCHEMISTRY

Analysis of ore samples, taken from the Lakefield report provided the values as summarized in Table 13-6:

Table 13-6: Ore Geochemistry Analysis (Lakefield)

Whole Rock - Box Ore	%	Trace Elements	%
SiO2	78.8	As	0.003
Al2O2	11.1	Ba	0.025
Fe2O3	1.43	Be	<0.005
MgO	0.37	Cd	<0.002
CaO	0.09	Cu	0.01
Na2O	3.63	Co	<0.002
K2O	3.02	Hg	<0.00005
TiO2	0.12	La	<0.01
P2O5	0.04	Mo	<0.001
MnO	<0.01	Ni	0.032
Cr2O3	0.02	Pb	0.020
LOI	0.5	S	0.38
		Sr	0.0018
		ThO2	0.002
		V	<0.02
		Y	0.0029
		Zn	0.017
		Zr	0.014

14 MINERAL RESOURCE ESTIMATES

14.1 BOX MINE

14.1.1 INTRODUCTION

Wardrop completed a NI 43-101 compliant resource estimate update of Box. The deposit has been interpreted as a single main mineralized zone with an updated wireframe on the HW and foot wall (FW) of the deposit near the base of the main zone. The effective date of this resource estimate is January 7, 2011.

14.1.2 DATABASE

Brigus supplied all the digital data for the resource estimate. The updated dataset was imported into Gemcom GEMS™ 6.2.4 resource evaluation software package.

The dataset included information from 469 drill holes, underground chip/channel samples, and surface trench samples. From this dataset, 48 drill holes lie outside the interpreted deposit and, therefore, were not used for the resource estimation of the deposit.

Included in the dataset are 67 trench samples from 13 surface trenches, five of the 13 drill holes from the 2010 drill program, and six drill holes from 2007 GLR drill program; all of which were used for the block model estimation. Note that the trench samples were used previously, by Wardrop in 2008, to update the resource classification only. The original resource estimate was completed in MineSight software by AMEC in 2005.

14.1.2.1 SPECIFIC GRAVITY

Wardrop did not undertake an independent review of the specific gravity readings therefore the specific gravity for the deposit was taken from AMEC 2005. The average specific gravity used for the BMG wireframe was 2.64. This specific gravity was established by AMEC 2005 and accepted by Wardrop 2008. Table 14-1 shows the specific gravities used and corresponding rock codes used in the resource model.

Table 14-1: Specific Gravity and Rock Codes

Lithology	Specific Gravity	Rock Code
Air	0.00	0
BMG	2.64	100
BMG2	2.64	101

14.1.3 EXPLORATORY DATA ANALYSIS

Exploratory Data Analysis is the application of statistical tools to understand the characteristics of the values in the database. The tools used are descriptive statistics, histograms, probability plots and box plots.

The statistics on the raw assay data are presented in Table 14-2. Statistics on values greater than zero are shown.

Table 14-2: Raw Assay Statistics (No Zeroes)

	Length (m)	Au (g/t)
Count	21.146	21,146
Min	0.010	0.003
Max	6.700	3,197.140
Mean	1.154	1.437
Std Dev	0.367	23.501
Variance	0.135	552.310
CV	0.318	16.358

14.1.3.1 CAPPING

Cumulative probability plots and Parrish plots were used to determine whether capping was required. Through an independent review, Wardrop determined that the previous capping level of 60 g/t Au was acceptable and this capping level was applied to the updated dataset.

Typically, a change or a break in the probability curve (i.e. separation of data points) indicates distinct populations of data. Outlying high values occur in the highest percentiles of a cumulative probability plot where separations in the data are common.

AMEC 2005 determined that the historic underground chip sample assays were considered too high and a weighting factor of 0.66 was applied to these historic assay intervals. Wardrop reviewed the statistics of the historic assay values and determined that the underground assays do not overtly skew the statistics of the dataset and the weighting factor was not applied to these historic assay values in this resource update.

The cumulative frequency plots for the raw uncapped gold assay values for the Box Mine gold deposit are given in Figure 14-1.

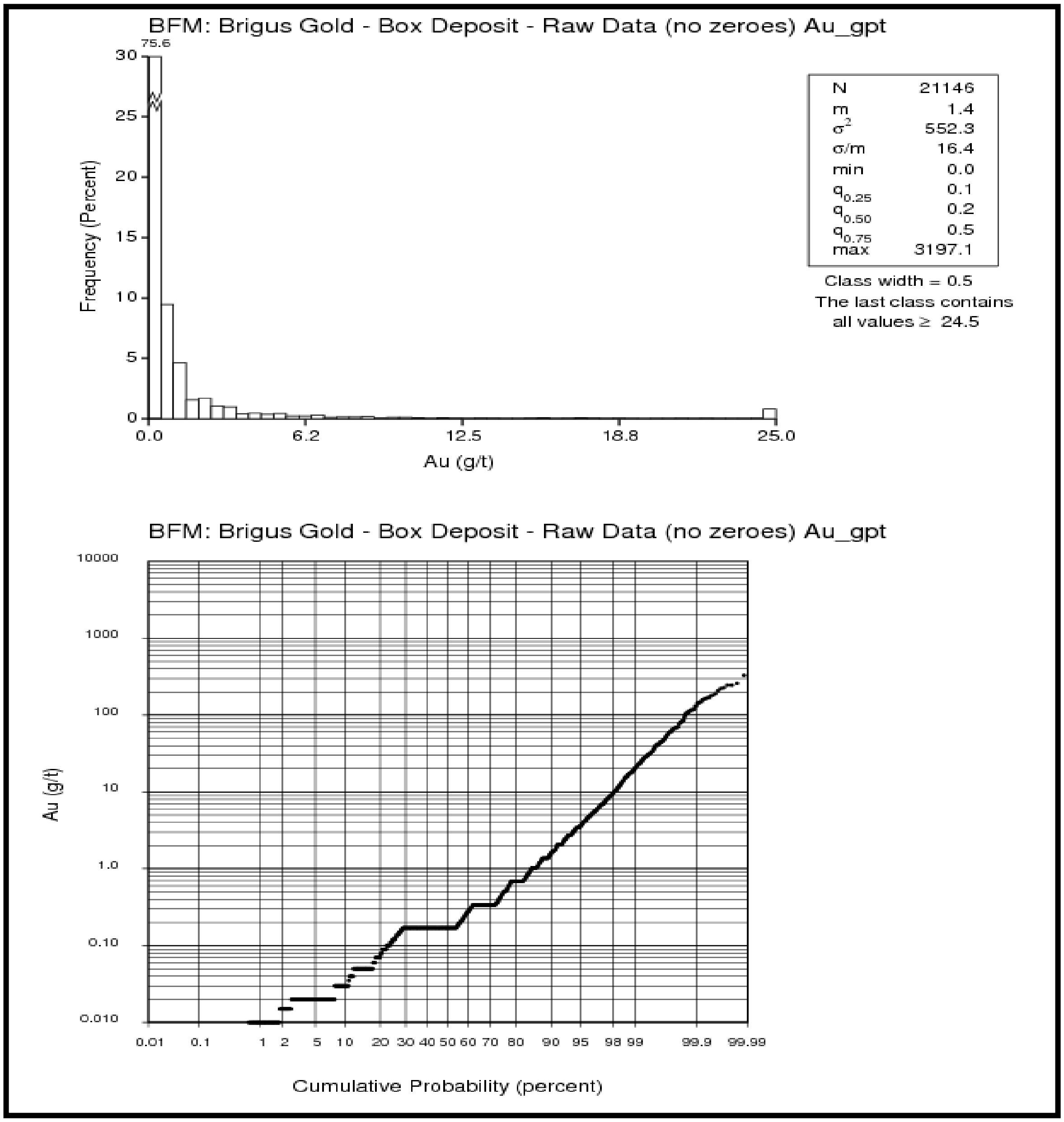

Figure 14-1: **Histogram and Cumulative Probability Plot for Au (uncapped)**

Table 14-3: Capping Levels Summary

Element	Capped Value (g/t Au)	Number of Samples Capped
Au (g/t)	60.00	65

Table 14-4: Capped Assay Statistics for Gold (No Zeroes)

	Au (g/t)
Count	21,146
Min	0.003
Max	17.000
Mean	0.858
Std Dev	2.384
Variance	5.683
CV	2.780

14.1.3.2 COMPOSITES

Composite lengths were created on 3.0 m intervals within the BMG wireframes (BMG and BMG2).

Table 14-5 presents the statistics for 3 m composites (no zeroes).

Table 14-5: 3 m Composite Statistics (No Zeroes)

	Au (g/t) Uncapped	Au (g/t) Capped
Count	6,421	6,421
Min	0.002	0.002
Max	568.640	60.000
Mean	1.701	1.391
Std Dev	9.339	3.676
Variance	87.210	13.513
CV	5.490	2.642

14.1.4 GEOLOGICAL INTERPRETATION

14.1.4.1 SOLID WIREFRAMES

The previous block model and resource estimate was completed using the interpreted BMG wireframe and served as the base of this resource update. Five (5) additional drill holes from the 2010 drill program allowed several small portions of the HW and FW to be incorporated at the base of the BMG wireframe. These areas were modelled as a separate wireframe labelled BMG2.

The BMG2 wireframe was created by digitizing 3D rings along cross sections at 12.5 m spacing and, where possible, grades of greater than 0.5 g/t Au were included from the five 2010 drill holes as a grade shell (see Figure 14-3). The 3D rings were joined together by tie lines and the BMG2 solid wireframe was created and validated. The BMG2 wireframe was clipped to the BMG wireframe to eliminate overlap of the two wireframes. Rock codes were assigned to the solid wireframes as shown in Table 14-6.

Table 14-6: Wireframe Solid Names and Explanation

Rock Code	Lithological Code	Zone
0	AIR	Air
90	WASTE	Country Rock
91	MINEDOUT	Historic Stopes and Workings
100	BMG	Main Mineralized Zone
101	BMG2	Addition to Main Mineralized Zone

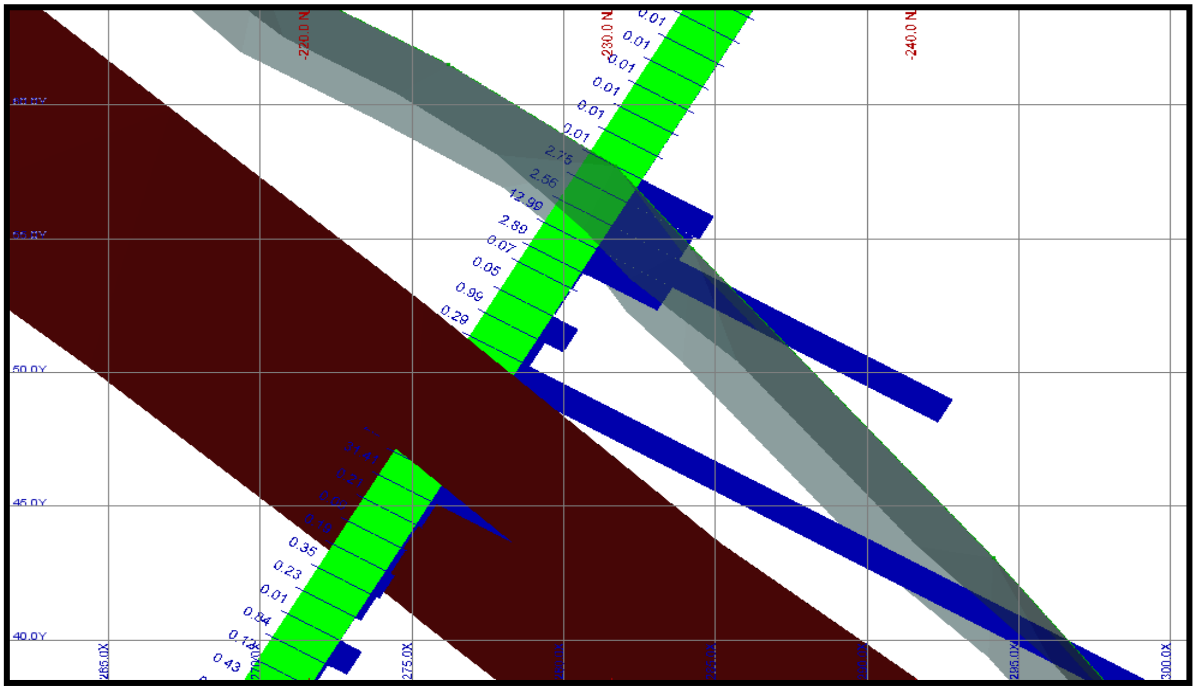

Figure 14-2: Box Mine Deposit Solids Showing BMG (red), BMG2 (blue-grey) and Drill HOles B10-304 with Au g/t (Looking Northeast)

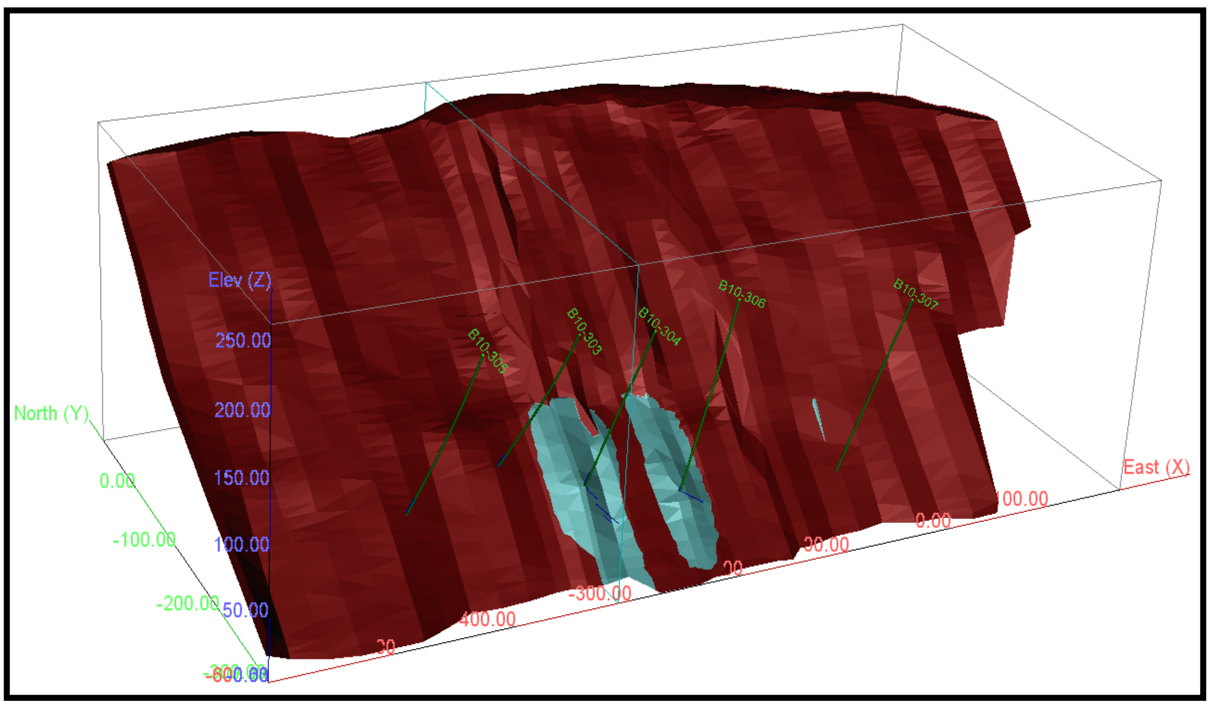

Figure 14-3: Box Solids (Showing BMG (red) and BMG2 (blue-grey) and 2010 Drill Holes - Looking North)

14.1.4.2 TOPOGRAPHIC SURFACE

A wireframe topographic surface was supplied to Wardrop by Brigus.

14.1.4.3 MINED OUT WIREFRAMES

The mined out stopes of Box were created from contour lines taken from the historic Box stope plan and section maps. The mined out stopes, from west to east, were: 309-311 (created as one stope), 508, 505, 302 and 301. A wireframe was also created to connect stopes 309-311 and 508.

Below the mined out stopes, ore passes for each of the four stopes were created from the base of the stopes to the ore pass at the lower drift level. Although these were not shown in the historic stope sections, these were assumed to be in place.

All wireframes were validated and found with no errors.

The wireframes created were based on Wardrop's older mined out wireframes. Stope 301 was clipped with the AMEC (2005) mined out wireframe as to avoid overlaps.

All five stopes and ore passes were exported and re-imported as a single wireframe and coded as MINEDOUT with rock code 91. The ore pass for stope 302 was absent from this single wireframe.

Figure 14-4 is a perspective view of the multiple solids used for the resource estimate looking towards the north. This figure presents the two mineralized zones and the mined out drifts, stopes and ore passes from the historic Box Mine.

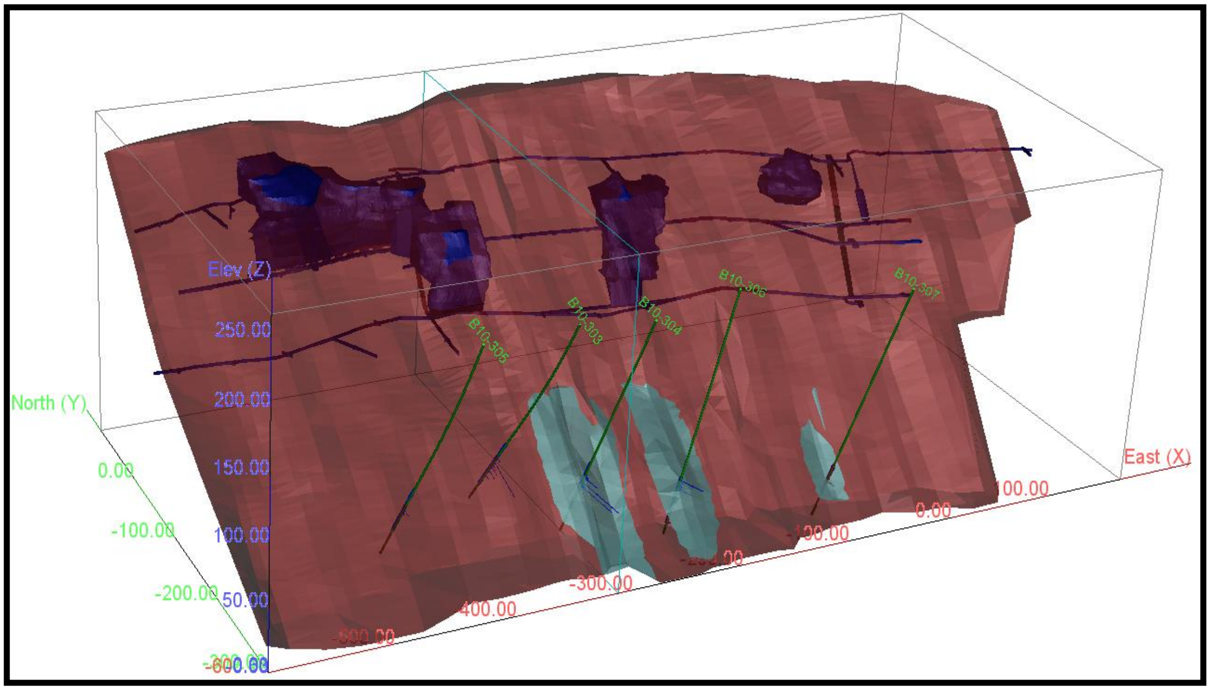

Figure 14-4: Box Solids (Showing BMG (red), BMG2 (blue-grey), and Mined-out (blue) and 2010 Drill Holes - Looking North)

14.1.5 BLOCK MODEL

A single block model was created to cover the BMG and BMG2 wireframes that make up the Box deposit. Table 14-7 shows the GEMS coordinates for the block model.

Table 14-7: Model Origin for Box

Element	Minimum	Maximum	
Easting (X)	-670	230	300 columns
Northing (Y)	-330	120	150 rows
Elevation (Z)	0	270	90 levels

A block size of 3 m x 3 m x 3 m was used to estimate the resources. These parameters were the same as those used in the original AMEC block model. A screen capture of the block model folders that were created for the block model is shown in Figure 14-5.

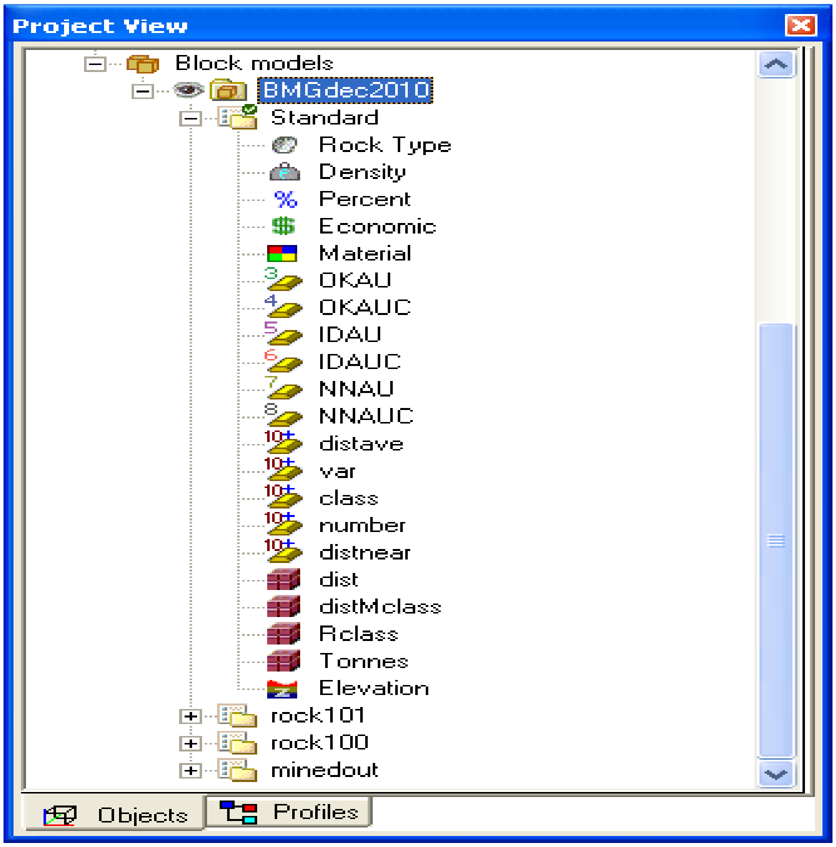

Figure 14-5: Block Model Folders for the Box Resource Estimate Update

14.1.5.1 INTERPOLATION AND SPATIAL ANALYSIS

The interpolation methods used for populating the block model and determining resource classification were OK, Inverse Distance Squared (ID2), and Nearest Neighbour (NN).

A single pass was used to interpolate blocks for the OK, ID2, and NN methods. The maximum number of samples per drill hole is limited to three. These parameters are summarized in Table 14-8.

Table 14-8: Number of Composites and Drill Holes used per Pass

Profile Name	No. Of Composites		Maximum No. of Samples per Drill Hole	No. Of Drill Holes	
	Min.	Max.		Min.	Max.
P1	2	10	3	1	4

A detailed list of parameters for the search ellipses for this resource **e**stimate is shown in Table 14-9.

Table 14-9: Search Ellipse Parameters

Profile Name	Rotation			Range			Search Type
	About Z	About Y	About Z	X (m)	Y (m)	Z (m)	
WEI_BMG	98	-40	0	50	50	6	Ellipsoidal

14.1.5.2 VARIOGRAPHY

With the new data added to the database, new variograms were created for the resource estimate update for Box. Samples used for variography are a function of geological interpretation. As there are two solid, or geological, wireframes for the same BMG Zone, only samples within the two solid wireframes are used, that is, 6,421 composite samples are used out of the total of 6,700. The variography was generated using SAGE 2001™ software.

Composited drill hole data was exported as a text file (.csv format) and imported directly into SAGE 2001™. Down hole variograms, using a lag distance equal to the composite length, were created for each of the separate domains. From the down-hole variograms, the nugget was estimated at 0.75.

The distance between drill holes ranges from less than 10 m up to 75 m therefore a 10 m lag distance was employed for variography. The number of lags used was 100, which was deemed sufficient to cover 1,250 m, that is, the length of the known deposit. All variograms utilized 22.5° bandwidths, 15° directional increments and 0.5 (50%) tolerance to optimize orientations.

Experimental variography was subsequently used to calculate best-fit modeled variography. If a lag contained less than 100 sample pairs in down-hole variography, or less than 350 pairs for spatial variography, they were ignored. Similarly, calculations were weighted by pairs. Two spherical structures were used for both down hole and spatial modelling and orientations used were customized to GEMS requirements.

Modeled variography results were recorded as a report file, and plot files for visual reference and are listed in Appendix G.

14.1.5.3 VARIOGRAPHY PARAMETERS

In GEMS, the convention used for variography parameters for Kriging profiles is right hand in the Z direction, right hand in the Y direction and right hand rotation in the Z direction. SAGE 2001™ software allows for the anisotropy results and rotation conventions to be output in GEMS format as described. Table 14-10 summarizes the variography parameters used for OK interpolation.

Table 14-10: Variography Parameters

Profile Name	Sill = 1	Search Anisotropy	Rotation About Z	Rotation About Y	Rotation About Z	X Range	Y Range	Z Range	Search Type
C0 (nugget)	0.750	-	-	-	-	-	-	-	-
C1	0.232	Rotation ZYZ	16	13	30	6.5	16.1	3.2	Spherical
C2	0.018	Rotation ZYZ	-4	49	36	142.5	162	44.9	Spherical

Figure 14-6 and Figure 14-7 show the resulting block model for the OK interpolation illustrating the general trend of the mineralization in perspective and cross section respectively.

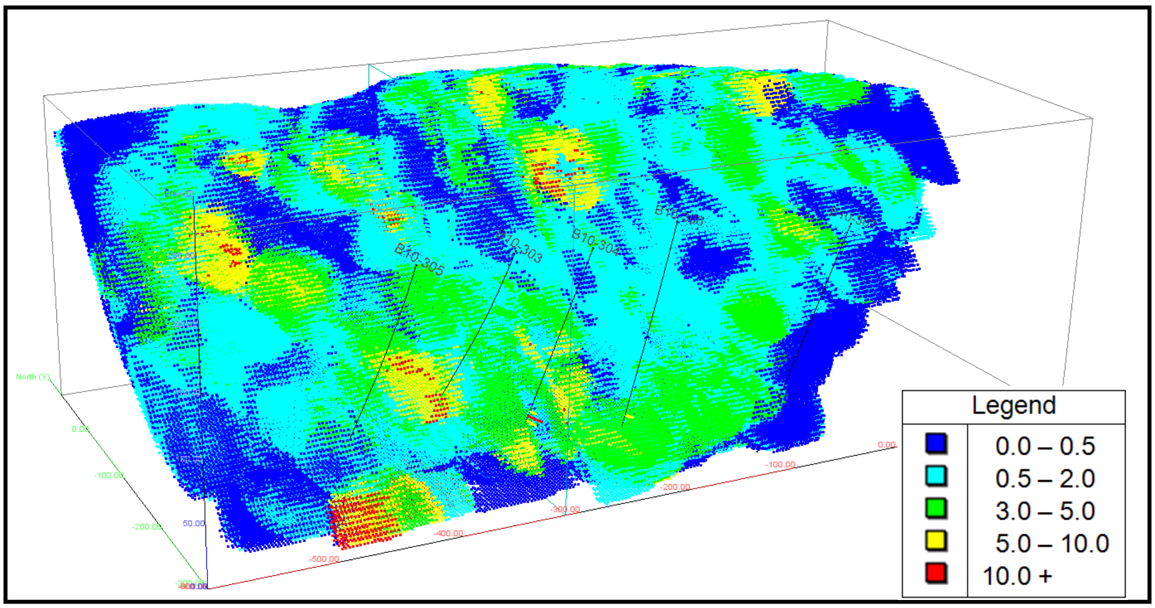

Figure 14-6: Block Model of Box; OK Interpolation

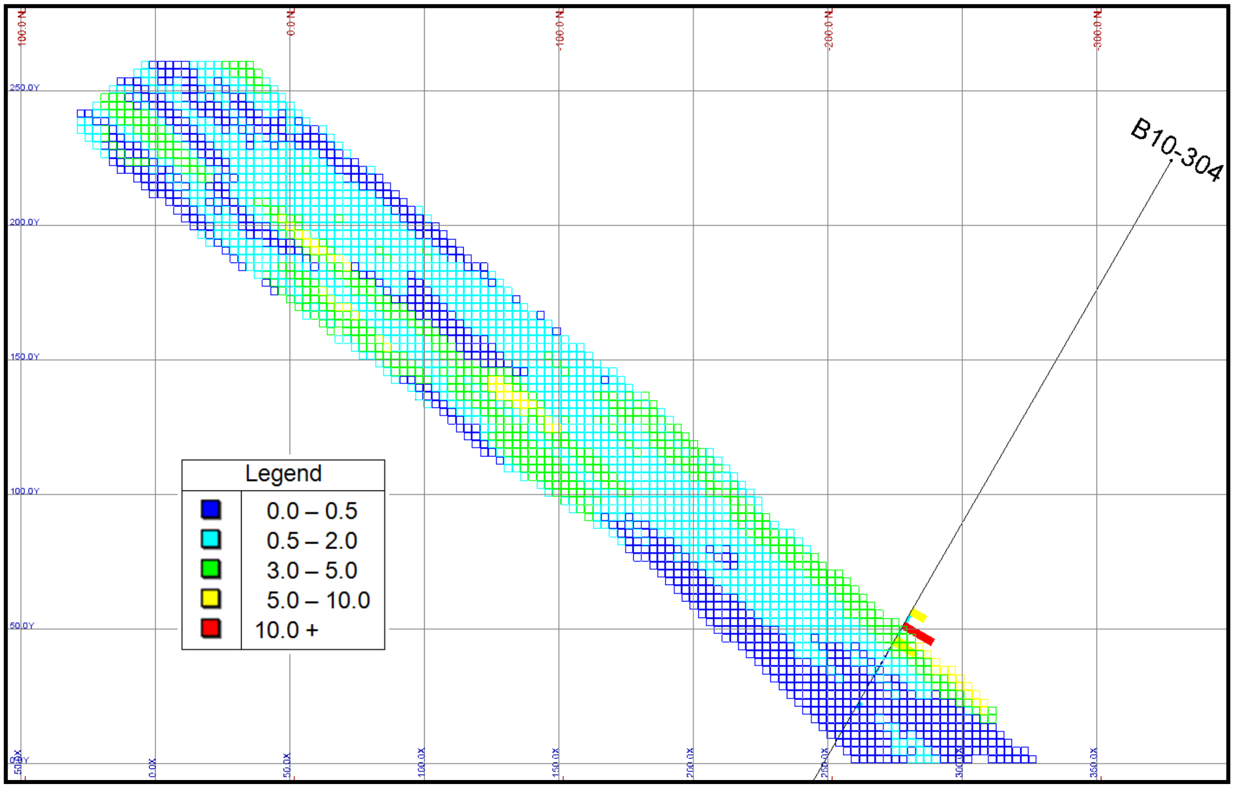

Figure 14-7: Block Model of Box; OK interpolation; Section 287.5E; looking east

14.1.6 MINERAL RESOURCE CLASSIFICATION

The mineral resource for Box is categorized into Measured, Indicated and Inferred Resources based on drill spacing and sample support.

- Measured Resources are those blocks that lie within 5 m of the historic workings, i.e. rock code 91, and surface trenches.
- Indicated Resources are those blocks that lie within a distance of 25 m from any one drill hole.
- Inferred Resources are those blocks that lie within an average distance of 35 m to a maximum of 40 m from any one drill hole.
- Any isolated Indicated Resource blocks at the edge of the deposit were converted to Inferred.
- Any remaining blocks were left as unclassified material.

The resulting mineral resource estimates from the OK interpolation method, at 0.5 g/t Au cut-off grade are:

- Measured Resources of 858,000 tonnes at 2.05 g/t Au
- Indicated Resources of 12,966,000 tonnes at 1.63 g/t Au
- Inferred Resources of 3,158,000 tonnes at 1.74 g/t Au.

The OK resource estimates for Box were estimated for a range of gold cut-off grades from 0.125 g/t Au to 4.0 g/t Au and are presented in Table 14-11. No recoveries have been applied to the interpolated estimates.

Grade tonnage curves for Measured and Indicated Resources are presented in Figure 14-8 and Grade tonnage curves for Inferred Resources are shown in Figure 14-9.

Table 14-11: Resource Estimate for Box

Gold Cut-off (g/t)	Measured			Indicated			Measured + Indicated			Inferred		
	Tonnes (x000 t)	Au (g/t)	Au oz (x000)	Tonnes (x000 t)	Au (g/t)	Au oz (x000)	Tonnes (x000 t)	Au (g/t)	Au oz (x000)	Tonnes (x000 t)	Au (g/t)	Au oz (x000)
4.0	101	6.26	20	726	5.52	129	827	5.61	149	228	5.99	44
3.5	125	5.79	23	1,072	4.95	171	1,197	5.03	194	315	5.37	54
3.0	161	5.21	27	1,502	4.45	215	1,664	4.53	242	430	4.80	66
2.5	207	4.67	31	2,258	3.88	281	2,464	3.94	312	634	4.13	84
2.0	266	4.13	35	3,291	3.36	356	3,556	3.42	391	877	3.61	102
1.5	383	3.39	42	4,968	2.81	449	5,351	2.85	491	1,227	3.07	121
1.0	585	2.65	50	7,785	2.24	561	8,371	2.27	611	1,881	2.43	147
0.5	**858**	**2.04**	**56**	**12,966**	**1.63**	**681**	**13,824**	**1.66**	**737**	**3,158**	**1.74**	**176**
0.375	939	1.90	57	14,945	1.48	709	15,884	1.50	766	3,636	1.57	183
0.25	1,012	1.79	58	16,952	1.34	729	17,964	1.36	787	4,169	1.41	188
0.125	1,046	1.74	58	18,164	1.26	737	19,210	1.29	795	4,709	1.27	192

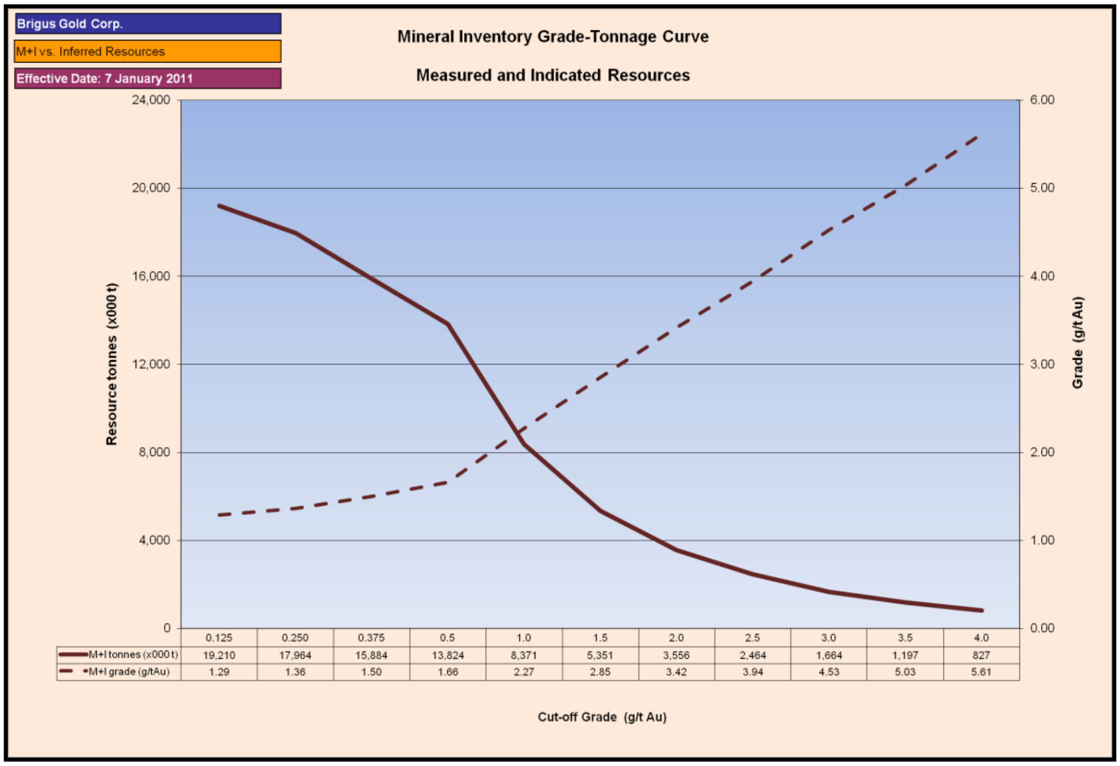

Figure 14-8: Grade Tonnage Curves for Measured and Indicated Resources

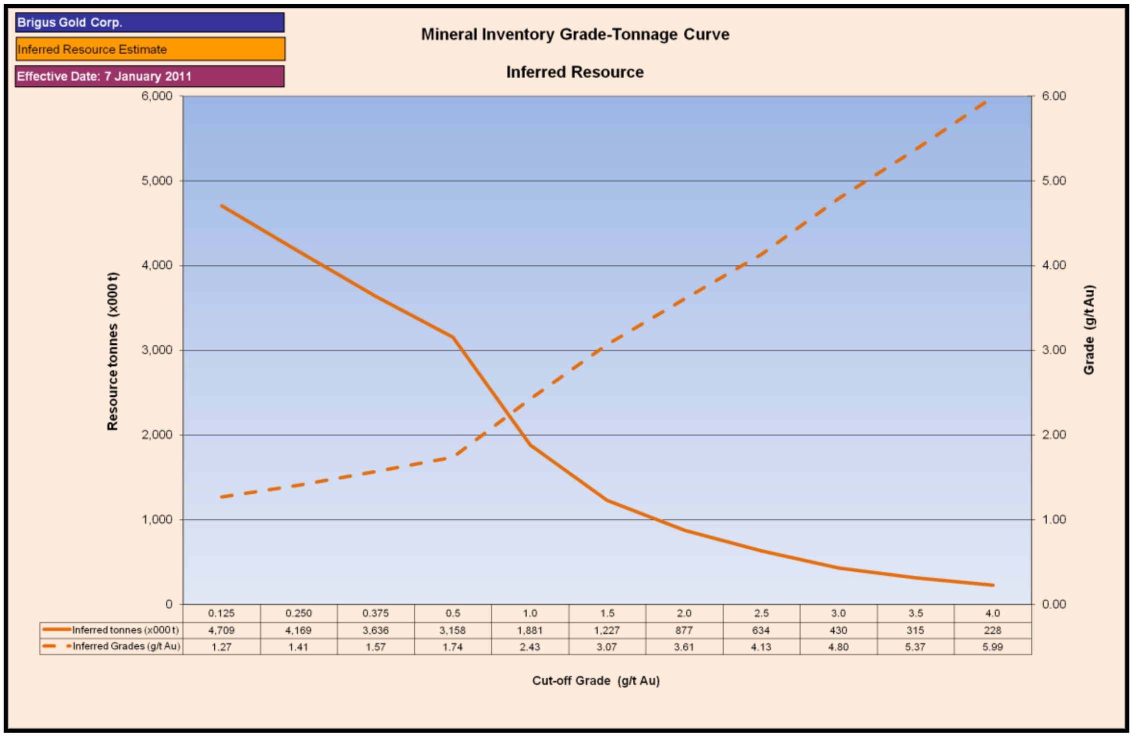

Figure 14-9: Grade tonnage Curves for Inferred Resources

14.1.7 VALIDATION

14.1.7.1 DEPOSIT VOLUME COMPARISON

The block model volumes were validated against the wireframe volumes and all differences were found to be within a tolerance of less than 0.10 %. The results of these comparisons are shown in Table 14-12.

Table 14-12: Volume Comparison between Wireframe Solid Models and Block Models

	Wireframe Total (m^3)	Block Model (m^3)	Difference (m3)	Difference (%)
BMG	10,368,456	10,377,578	9,122	0.09
BMG2	52,561	52,549	12	0.02

14.1.7.2 STATISTICS COMPARISON

A comparison was made between the composited values and those populated into the block model. The final resource estimate is based on the OK values, and both ID2 and NN methods were run as a validation method. The comparison of the mean grades for the capped gold shows no extraneous values. The results of the comparison are shown in Table 14-13:

Table 14-13: Comparison of Mean Gold Grades

	Au (g/t)
OK	1.322
ID2	1.346
NN	1.354
3 m Composites	1.391

14.1.7.3 VISUAL COMPARISON

A visual comparison was also made between estimated block grades and the surrounding composite grades used for estimation. This ascertains that estimated grades do not exceed the maximum grade of any one specific 3 m composite.

14.2 ATHONA DEPOSIT

14.2.1 INTRODUCTION

Wardrop completed a NI 43-101 compliant resource estimate update of Athona. The effective date of the resource estimate was May 2007. Wardrop determined the mineral resource for Athona using Gemcom™ version 6.04. The estimation was completed for gold at Athona using historic data and the 2006 drilling completed by GLR.

Reference the report (Wardrop, 2007) filed under Mistango River Resources Inc. for complete documentation of the resource estimation.

14.2.2 MINERAL RESOURCE CLASSIFICATION

On the basis of the criteria outlined in Table 14-14, approximately 85% of the blocks estimated in the Athona model are Indicated Resources and the balance is Inferred Resources.

Table 14-14: Resource Classification Criteria

Measured	Indicated	Inferred
• Not determined	• Minimum of two drill holes • Minimum of four composites • Distance to nearest composite less than 50 m	• Minimum of three composites • Distance to nearest composite less than 90 m

14.2.3 MINERAL RESOURCE TABULATION

Table 14-15 shows Indicated Resources for Athona . Table 14-16 summarizes the Inferred Resources for Athona. The base case is reported on a 0.5 g/t Au cut-off grade.

Table 14-15: Athona Indicated Resources

	Cut-off (g/t Au)	Tonnage (000's t)	Au (g/t)
Indicated	> 3.0	371.4	4.08
	> 2.5	1033.2	3.00
	> 2.0	1870.7	2.43
	> 1.0	3399.8	1.89
	> 0.5	7036.4	1.28

Table 14-16: Athona Inferred Resources

	Cut-off (g/t Au)	Tonnage (000's t)	Au (g/t)
Inferred	> 3.0	50.3	4.45
	> 2.5	88.8	3.46
	> 2.0	213.7	2.44
	> 1.0	558.6	1.69
	> 0.5	1406.4	1.10

14.2.4 BLOCK MODEL VALIDATION

The Athona grade models were validated by two methods:

1. Visual comparison of colour-coded block model grades with composite grades on section plots.
2. Comparisons of the global mean block grades for OK, NN and composites.

14.2.4.1 VISUAL COMPARISONS

The visual comparisons of block model grades with composite grades for each of the four zones show a reasonable correlation between the values. No significant discrepancies were apparent from the sections reviewed. Appendix B in the original report (Wardrop, 2007) includes representative Gemcom plots of the comparison between the block model and composite grades.

14.2.4.2 GLOBAL COMPARISONS

The grade statistics for the OK, inverse distance and NN models (approximated by an Inverse Distance to the fifth power) were compared. The contained metal in the IDW2 model is 0.1% higher than the OK model. The NN model contains 2.6% less metal than the kriged model. There is generally close agreement among all three methods (Table 14-17).

Kriged values are slightly higher than the NN, which is a reflection of sample density, with the higher kriged values corresponding to "smearing" of values in areas of lower drillhole density, particularly in the high grade (HG)gold veins along the lake.

Table 14-17: Block Model Validation

Estimation Model	Au (kg)	Relative Difference
NN	12,945.4	-2.6%
ID^2	13,296.1	0.1%
OK	13,281.0	

14.2.5 RECONCILIATION

No allowance has been made to subtract tonnage due to mine development drifts, crosscuts or shafts. The exact locations of some of these workings are not known. Previous mining can be visually confirmed at Athona. This mined-out material has been estimated between 10,000 and 15,000 tonnes (Bevan, 1995) which correspond to 0.1 to 0.2% of the Indicated Resource tonnage, respectively. Production was not achieved at Athona.

15 MINERAL RESERVE ESTIMATES

15.1 GENERAL

The mineable reserves were estimated by March Consulting utilizing updated mine pit designs. The updated pit designs were prepared for Box and Athona. The pit designs were based on Whittle pit shells as provided by Brigus and resource models as provided by Wardrop.

15.2 RESOURCE MODEL

The resource models for both Box and Athona were produced by Wardrop. The Box resource models used a 3 m x 3 m x 3 m block size for a block volume of 27 m^3 and approximate mass of 72 tonnes and the Athona resource model used a block size of 5 m x 5 m x 10 m for a block volume of 250 m^3 and approximate mass of 663 tonnes.

15.3 MINING RESERVES

15.3.1 CUT OFF GRADE

The cut off grade (COG) for Box was estimated using the equation in Figure 15-1. The strip ratio and costs used in the COG calculation are preliminary and have been revised throughout the project. Using the current strip ratio of 4.56 for Box would increase the COG but would not affect the marginal cut-off grade (MCOG), resulting in the same total reserves for the project. The updated project information will be used to refine the COG during the next phase of the project. The same COG was used for the analysis and pit design of Box and Athona. Due to the low strip ratio of 1.10 at Athona, a lower COG could be utilized for future revisions of the pit design. LG ore is defined as the ore with grades between the COG and the MCOG. The troy ounce is used in all calculations with a conversion of 31.10 g/oz.

Brigus Gold - Goldfields Project - Box Mine

Given:

Mining Cost ($/t) =	2.60	Stripping Ratio (waste:ore) =	4.04
Marginal Mining Cost ($/t) =	1.00	Mill Portion of G & A (%) =	33.0%
Milling Cost ($/t) =	10.70	gold value ($/oz) =	1250.00
G & A ($/t) =	5.00	gm / troy oz =	31.10

COG Calculation

$$\frac{\text{Gold Value (\$/oz) x COG (gm/t)}}{\text{31.10 (gm/oz)}} = \text{[Mining Cost (\$/t) x (1 + Stripping Ratio)] + Milling Cost (\$/t) + G \& A (\$/t)}$$

$$\text{COG (gm/t)} = \frac{\text{\{[Mining Cost (\$/t) x (1 + Stripping Ratio)] + Milling Cost (\$/t) + G \& A (\$/t)\} x 31.10 (gm/oz)}}{\text{Gold Value (\$/oz)}}$$

COG (gm/t) = 0.72

Marginal COG Calculation

$$\frac{\text{Gold Value (\$/oz) x Marginal COG (gm/t)}}{\text{31.10 (gm/oz)}} = \text{[Marginal Mining Cost (\$/t) + Milling Cost (\$/t) + (G \& A (\$/t) x Mill Portion of G \& A (\%))]}$$

$$\text{Marginal COG (gm/t)} = \frac{\text{[Marginal Mining Cost (\$/t) + Milling Cost (\$/t) + (G \& A (\$/t) x Mill Portion of G \& A (\%))] x 31.10 (gm/oz)}}{\text{Gold Value (\$/oz)}}$$

Marginal COG (gm/t) = 0.33

Figure 15-1: COG Calculation

15.3.2 ECONOMIC AND DESIGN PARAMETERS

Mineable reserves for Box are based upon the Measured and Indicated Resources in the computerized 3-D block model. Mineable pit shapes optimize the extraction of the mineral inventory given the economic and technical parameters determined for this study. The pit optimization procedures utilized to define the final pit design took into consideration the factors and assumptions in Table 15-1 as well as the following design criteria:

- Minimum pit bottom width of 20 m
- Total haul road width of 21 m with berms

The Box and Athona pits were designed using a two step process. The first step was to determine the economic mining limits using Gemcom's Whittle program. Whittle Lerchs-Grossman algorithms were utilized to delimit the economic pits for both Athona and Box. This algorithm provided a basic pit shape outline that served as the basis for final pit design. The routine floats an economic cone over all blocks in the 3-D block model to determine what mineralized material can be mined and processed given the economic parameters input. While the Whittle economic pits respect the wall slope angles they do not include the ramp width mining.

The base case Whittle economic pit design criteria for both the Box and Athona pits are shown in Table 15-1.

Table 15-1: Mine Pit Design Criteria

	Box	Athona
Inter-Ramp Wall Angle - hanging & end walls	55°	55°
Inter-Ramp Wall Angle - footwall	42°	55°[d]
COG (g/t)	0.72	0.50[c]
Marginal COG (g/t)	0.33	0.25[c]
Measured & Indicated Resources Block "standard class"	1[a] & 2[b]	1[b]
Bench height (m)	9	9
Crest to Toe Offset (m)	2	2
Bench configuration	Double	Double
Berm width (m)	8.6	8.6
Ramp width (m)	21	21
Ramp grade (%)	10	10
Mining ore and waste ($/t mined)	$2.60	$2.88[c]
Milling ($/t milled)	$10.70	$16.68[c]
G&A ($/t milled)	$5.00	$0.00[c]
Gold Value ($US/troy oz)	$1250	$1150[c]
Riparian zone restriction (m)	30	30
Gold recovery (%)	91	89

Notes:

a: Measured Resources

b: Indicated Resources

c: The values of COG, Marginal COG, gold value and operating costs used for the Athona Whittle analysis differed from Box since the Athona Whittle was completed much earlier in the design process. It was determined that the updated design criteria would have a minimal impact on the pit design at Athona due to the proximity of the deposit to Lake Athabasca and the low mine recovery that was calculated therefore the Whittle analysis was not updated.

d: Due to the configuration of the Athona pit there is no distinct FW formation so the pit was designed with a consistent 55° angle for hanging, foot, and end walls.

The second step required the use of Gemcom's Surpac program to design the mining pits, based on the Whittle economic pit shells, to include smoothing, ramp access, berms and mining benches for Athona and Box. Final mineable reserves were generated from these pit designs.

15.3.3 MINEABLE RESERVES

Box and Athona include 22,333,045 tonnes of proven and probable reserves at an average grade of 1.420 g/t containing 1,020,000 oz of gold. Table 15-2 provides a summary of the proven and probable reserves for Box and Athona.

Box has 16,502,247 tonnes of proven + probable reserves at an average grade of 1.508 g/t containing 800,000 oz of gold. Box has a strip ratio of 4.56 and a mine recovery of 97%. While the resources for Box were calculated using volume adjusted blocks, the reserves were calculated for full blocks. The rational for this is the degree of accuracy of the diamond drill holes, the accuracy of the drill collar surveys, and the mining unit size in relation to the 3 metre cubic resource model blocks does not justify the use of volume adjusted blocks at this level of study.

Athona has 5,830,798 tonnes of probable reserves at an average grade of 1.172 g/t containing 220,000 oz of gold. Athona has a strip ratio of 1.10 and a mine recovery of 61%. Part of the Athona resource is located in close proximity to the shore of Lake Athabasca. This ore cannot be economically extracted using underground mining techniques. Further study is required to determine the economic and environmental impacts of extending the pit towards Lake Athabasca to increase mine recovery.

Mining reserves by bench are shown for Box in Table 15-3 and Athona in Table 15-4. Benches are identified by meters ASL. Pit mining statistics for both Athona and Box are shown in Table 15-5.

Table 15-2: Mineable Reserves for Box and Athona

Description	Ore (t)	Grade (g/t)	Gold (oz)	Waste (t)
Box				
Proven	1,227,726	1.900	75,000	
Probable	15,274,521	1.477	725,000	
Proven + Probable	16,502,247	1.508	800,000	75,228,132
Athona				
Proven				
Probable	5,830,798	1.172	220,000	
Proven + Probable	5,830,798	1.172	220,000	6,423,778
Total				
Proven	1,227,726	1.900	75,000	
Probable	21,105,319	1.392	945,000	
Proven + Probable	22,333,045	1.420	1,020,000	81,651,910

Table 15-3: Box Mined Quantities by Bench – Proven & Probable Reserves

Bench	Total Volume	Total Waste		Development Waste		Production Waste		Total Ore			LG Ore (0.33 - 0.72 g/t)			HG Ore (0.72 + g/t)		
		Volume	Tonnes	Volume	Tonnes	Volume	Tonnes	Volume	Tonnes	g/t	Volume	Tonnes	g/t	Volume	Tonnes	g/t
261	14,417	5,480	14,248	-	-	5,480	14,248	8,937	22,881	2.420	972	2,495	0.474	7,965	20,386	2.658
252	570,165	496,752	1,291,555	53,671	139,545	443,081	1,152,011	73,413	189,748	2.083	15,849	40,487	0.527	57,564	149,261	2.506
243	1,602,964	1,432,648	3,724,885	461,046	1,198,720	971,602	2,526,165	170,316	443,004	1.836	51,975	134,648	0.513	118,341	308,356	2.414
234	2,209,703	1,966,352	5,112,515	873,563	2,271,264	1,092,789	2,841,251	243,351	635,533	1.472	95,553	249,551	0.507	147,798	385,982	2.095
225	2,666,444	2,374,439	6,173,541	1,292,295	3,359,967	1,082,144	2,813,574	292,005	764,621	1.431	114,129	299,020	0.525	177,876	465,601	2.013
216	2,714,880	2,396,928	6,232,013	1,513,112	3,934,091	883,816	2,297,922	317,952	832,907	1.488	109,701	287,472	0.517	208,251	545,435	2.000
207	2,667,569	2,303,933	5,990,226	1,559,088	4,053,629	744,845	1,936,597	363,636	950,377	1.525	117,909	307,930	0.506	245,727	642,447	2.014
198	2,422,660	2,048,413	5,325,874	1,551,546	4,034,020	496,867	1,291,854	374,247	980,242	1.572	123,066	322,328	0.507	251,181	657,914	2.094
189	2,339,942	2,026,067	5,267,774	657,184	1,708,678	1,368,883	3,559,096	313,875	820,291	1.562	97,416	254,969	0.508	216,459	565,322	2.038
180	2,099,547	1,882,008	4,893,221	653,096	1,698,050	1,228,912	3,195,171	217,539	569,028	1.571	68,202	178,485	0.506	149,337	390,543	2.058
171	2,019,455	1,755,773	4,565,010	664,041	1,726,507	1,091,732	2,838,503	263,682	690,775	1.412	96,444	252,545	0.498	167,238	438,230	1.939
162	1,795,533	1,458,222	3,791,377	664,707	1,728,238	793,515	2,063,139	337,311	882,303	1.330	138,456	362,815	0.502	198,855	519,488	1.909
153	1,720,523	1,377,056	3,580,346	677,106	1,760,476	699,950	1,819,870	343,467	898,199	1.430	139,563	364,455	0.505	203,904	533,744	2.061
144	1,502,162	1,247,525	3,243,565	678,653	1,764,498	568,872	1,479,067	254,637	665,683	1.429	99,846	260,528	0.507	154,791	405,155	2.021
135	1,427,249	1,193,429	3,102,915	694,459	1,805,593	498,970	1,297,322	233,820	613,221	1.467	80,136	209,848	0.512	153,684	403,373	1.964
126	1,231,818	901,257	2,343,268	173,581	451,311	727,676	1,891,958	330,561	864,627	1.484	93,420	244,063	0.518	237,141	620,564	1.864
117	1,166,114	820,622	2,133,617	172,812	449,311	647,810	1,684,306	345,492	903,118	1.479	88,263	230,306	0.527	257,229	672,812	1.805
108	982,206	682,344	1,774,094	168,721	438,675	513,623	1,335,420	299,862	784,366	1.607	68,985	179,911	0.522	230,877	604,455	1.930
99	916,697	660,197	1,716,512	167,935	436,631	492,262	1,279,881	256,500	669,819	1.686	58,590	152,112	0.528	197,910	517,707	2.027
90	717,871	478,840	1,244,984	163,356	424,726	315,484	820,258	239,031	624,911	1.607	67,662	176,988	0.528	171,369	447,923	2.033
81	659,837	436,169	1,134,039	162,602	422,765	273,567	711,274	223,668	584,495	1.468	69,876	182,904	0.523	153,792	401,591	1.899
72	510,336	313,938	816,239	161,388	419,609	152,550	396,630	196,398	513,858	1.454	61,506	161,948	0.507	134,892	351,910	1.889
63	454,625	274,886	714,704	-	-	274,886	714,704	179,739	472,586	1.484	53,757	141,918	0.494	125,982	330,668	1.909
54	317,837	171,146	444,980	-	-	171,146	444,980	146,691	384,556	1.525	43,146	113,264	0.513	103,545	271,292	1.947
45	262,952	133,649	347,487	-	-	133,649	347,487	129,303	339,863	1.407	47,007	123,457	0.519	82,296	216,406	1.913
36	145,628	53,261	138,479	-	-	53,261	138,479	92,367	242,922	1.355	39,798	104,995	0.518	52,569	137,927	1.992
27	102,746	42,563	110,664	-	-	42,563	110,664	60,183	158,313	1.207	30,807	81,330	0.506	29,376	76,983	1.948
Total	35,241,880	28,933,897	75,228,132	13,163,962	34,226,301	15,769,935	41,001,831	6,307,983	16,502,247	1.508	2,072,034	5,420,772	0.512	4,235,949	11,081,475	1.995

Table 15-4: Athona Mined Quantities by Bench – Proven & Probable Reserves

Bench	Total Volume	Waste		Total Ore			LG Ore (0.33 - 0.72 g/t)			HG Ore (0.72 + g/t)		
		Volume	Tonnes	Volume	Tonnes	gm/t	Volume	Tonnes	gm/t	Volume	Tonnes	gm/t
240	55,756	55,756	144,966	-	-		-	-	-	-	-	
231	261,342	236,817	615,724	24,525	62,606	1.365	4,725	12,521	0.581	19,800	50,085	1.561
222	512,989	405,664	1,054,726	107,325	277,256	1.337	33,325	85,131	0.520	74,000	192,125	1.699
213	699,687	426,612	1,109,191	273,075	704,304	1.244	101,300	259,303	0.522	171,775	445,001	1.665
204	688,438	344,413	895,474	344,025	904,114	1.240	124,650	326,745	0.507	219,375	577,369	1.654
195	636,694	270,644	703,674	366,050	970,033	1.199	140,000	371,000	0.517	226,050	599,033	1.622
186	523,325	166,875	433,875	356,450	944,593	1.104	150,100	397,765	0.515	206,350	546,828	1.532
177	479,055	187,680	487,968	291,375	772,144	0.993	133,500	353,775	0.499	157,875	418,369	1.412
168	345,992	152,617	396,804	193,375	512,444	0.973	94,650	250,823	0.491	98,725	261,621	1.435
159	289,962	136,962	356,101	153,000	405,451	1.242	68,300	180,995	0.464	84,700	224,456	1.870
150	156,289	64,489	167,671	91,800	243,271	1.486	29,700	78,705	0.502	62,100	164,566	1.957
141	35,205	22,155	57,603	13,050	34,582	1.191	2,925	7,751	0.485	10,125	26,831	1.395
Total	4,684,734	2,470,684	6,423,778	2,214,050	5,830,798	1.172	883,175	2,324,514	0.506	1,330,875	3,506,284	1.613

Table 15-5: Box and Athona Statistics

	Athona	Box
Stripping Ratio:	1.10	4.56
Recovery (%):	61.3%	97.0%
Gold (gms):	6,831,000	24,887,000
Gold (troy oz):	220,000	800,000

16 MINING METHODS

16.1 GENERAL

Ore has been classified into two categories – HG and LG. The HG ore, consisting of grades greater than or equal to the economic cut-off grade (COG) of 0.72 g/t, will be hauled directly to the mill. The LG ore, which will be stockpiled, consists of grades less than the COG and greater than or equal to the MCOG of 0.33 g/t. The stockpiled LG ore will be processed at the end of the mine life.

An ore grade control program will be necessary to delineate the waste from the two ore categories. Geological bench plans are to be supplemented with blasthole sampling, pit mapping and visual distinguishing of the ore bearing rock. A satisfactory grade control program will reduce mining losses and minimize ore dilution.

16.2 SLOPE STABILITY

The design parameters used for Box are based on the recommendations of the draft report by Klohn Crippen Ltd. entitled "Box Mine Project; Preliminary Open Pit Slope Design - June 1995". This draft report indicated that a HW inter-ramp angle of 55 degrees and a FW inter-ramp angle of 42 degrees would be a conservative estimate. Athona has no apparent footwall so 55 degrees was used as the inter-ramp slope angle on all sides of the pit.

A 9 m bench height was selected. The 9 m height is approximately 60% of the reach of the selected hydraulic shovel and should provide acceptable dilution control. Table 16-1 shows the pit wall design dimensions.

Table 16-1: **Box Wall Design Dimensions**

	Footwall	Hanging & End Walls
Inter-Ramp Slope Angle	42°	55°
Bench Height (m)	9	9
Intermediate Berm Width (m)	2	2
No. of Benches between Full Width Berms	2	2
Berm Width (m)	8.6	8.6
Inter-Berm Slope Angle	57.67°	77.47°

16.2.1 SELECTIVE MINING

The mining activities at Athona and Box will require the appropriate separation of ore and waste to minimize dilution. This requires proper identification of the ore and waste and efficient and effective loading.

As mineralization is generally limited to within the "BMG"; for mining purposes, the ore body can therefore be described as mineralized zones within a structurally controlled host rock. Additionally, the rock type (granite) hosting the mineralization is visually distinguishable from the rock types of the HW and FW. These physical attributes and visual characteristics will facilitate separation/segregation of the ore from the waste.

Additionally, sampling and analysis of blasthole drill cuttings will be used to confirm and define the zones of HG mineralization, LG mineralization and waste within the Box and Athona granite structures.

Loading equipment selection is generally predicated on the physical characteristics of the ore body, to be extracted. The selected loading unit must minimize ore dilution while achieving operating efficiency. The inherent physical characteristics of the zones of mineralization at the Goldfields Project are suited to front end loaders or hydraulic shovels. It was determined that a 13 m^3 bucket capacity should be used for ore and waste loading. Either of these loading units will allow for appropriate HG ore, LG ore and waste separation.

16.3 TREATMENT OF PREVIOUS UNDERGROUND EXTRACTION ZONES

The open pit design is predicated on the assumption that the existence of the previous underground stopes (areas of prior underground ore extraction) will not significantly disrupt the proposed mining schedule.

These stopes remain open and can measure up to 50 m x 50 m. The smaller underground workings such as the ventilation and access infrastructure (drifts, raises and winzes) will not impact open pit planning as the volume of material required to fill these structures is not significant; however, knowledge of the location of this infrastructure is required to avoid potential incidents.

To minimize the impact of these pre-existing stopes, the large voids will be filled with LG ore, prior to their interception by the open pit mining. Blasthole drills will be used to intercept and confirm the dimensions of the underground stopes. A pattern of drill holes will be used to develop a drop raise connecting these voids to the surface. The resulting surface depression will be backfilled with previously mined LG to permit uninterrupted mining of the current benches. It is anticipated that 80% of the LG used for backfilling will be separated during recovery and trucked to the LG stockpile. The balance of the LG backfill will blend with the ore within these areas and be fed with the HG to the mill. The use of LG, rather than waste, for backfilling the existing workings will minimize the impact of dilution during subsequent recovery of ore within these areas.

16.4 MINE PRODUCTION SCHEDULE

The mine production schedule is presented in Table 16-2. Due to the existing underground workings, the mine schedule accommodates the requirement to develop drop raises into the old stopes and backfill these with LG ore. It was assumed that 80% of the LG would be recovered to the LG stockpile and 20% would account for dilution of the HG ore.

The total mill feed is made up of four product streams:

- HG ore from Box
- HG ore from Athona
- LG ore dilution from Box stopes
- LG ore from stockpile

HG Ore from Box would be processed in Years 1 – 7. Year 1 is represented as a half year based on the current construction schedule that estimates production would commence at the end of the second quarter of the final year of construction. Athona ore would come on stream in the last half of Year 7 and continue to Year 9. Processing from the LG stockpile would take place from Years 9 – 13. Figure 16-1 indicates the contributions to total mill feed from the four sources of ore. The LG ore feed in years 2 – 5 represents the contribution from the LG ore dilution from mining the backfilled stopes. The LG feed from Years 9 – 13 represents the processing of the LG stockpile.

The annual gold production is represented in Figure 16-2. The annual average gold production is 81,965 oz/year, with an average of 108,840 oz/year in Years 1 – 9 of operation prior to processing the LG ore stockpile. Figure 16-2 also indicates the average gold grade per year of operation ranging from a high of 2.27 g/t in Year 1 to 0.51 g/t in Years 10 - 13 for a LOM average of 1.42 g/t.

Table 16-2: Mine Production Schedule

	Box		Athona		Total Production		Waste (t)			
	Ore (t)	Grade (g/t)	Ore (t)	Grade (g/t)	Ore (t)	Grade (g/t)	Box		Athona	Total
Year							Prod Waste	Development Waste		
1	900,000	2.27	-	-	900,000	2.27	7,254,675	3,163,279	-	10,417,953
2	1,825,000	1.99	-	-	1,825,000	1.99	6,927,093	9,130,960	-	16,058,054
3	1,825,000	1.95	-	-	1,825,000	1.95	7,402,835	7,553,860	-	14,956,696
4	1,825,000	1.94	-	-	1,825,000	1.94	8,816,037	4,473,685	-	13,289,721
5	1,825,000	1.86	-	-	1,825,000	1.86	5,191,412	7,762,114	-	12,953,526
6	1,825,000	1.97	-	-	1,825,000	1.97	3,599,833	1,722,797	-	5,322,629
7	1,269,651	1.93	555,349	1.67	1,825,000	1.85	1,809,946	419,609	2,515,416	4,744,971
8	-	-	1,825,000	1.61	1,825,000	1.61	-		2,442,214	2,442,214
9	699,065	0.51	1,125,935	1.59	1,825,000	1.18	-		1,466,147	1,466,147
10	1,825,000	0.51	-	-	1,825,000	0.51	-		-	-
11	1,825,000	0.51	-	-	1,825,000	0.51	-		-	-
12	858,531	0.51	966,469	0.51	1,825,000	0.51	-		-	-
13	-	-	1358,100	0.51	1,358,100	0.51	-		-	-
LOM Total	**16,502,247**	**1.51**	**5,830,853**	**1.17**	**22,333,100**	**1.42**	**41,001,831**	**34,226,304**	**6,423,777**	**81,651,911**

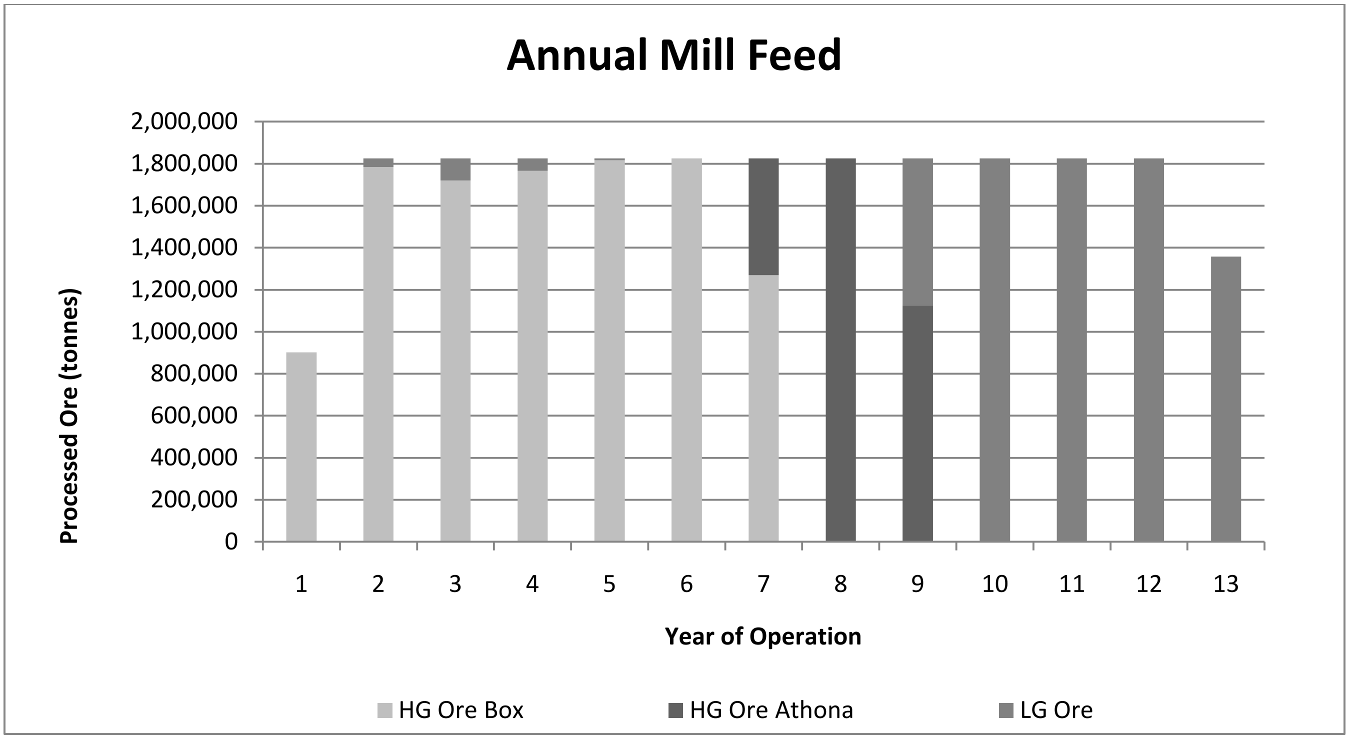

Figure 16-1: Annual Mill Feed Contribution from Box, Athona, and LG Stockpile

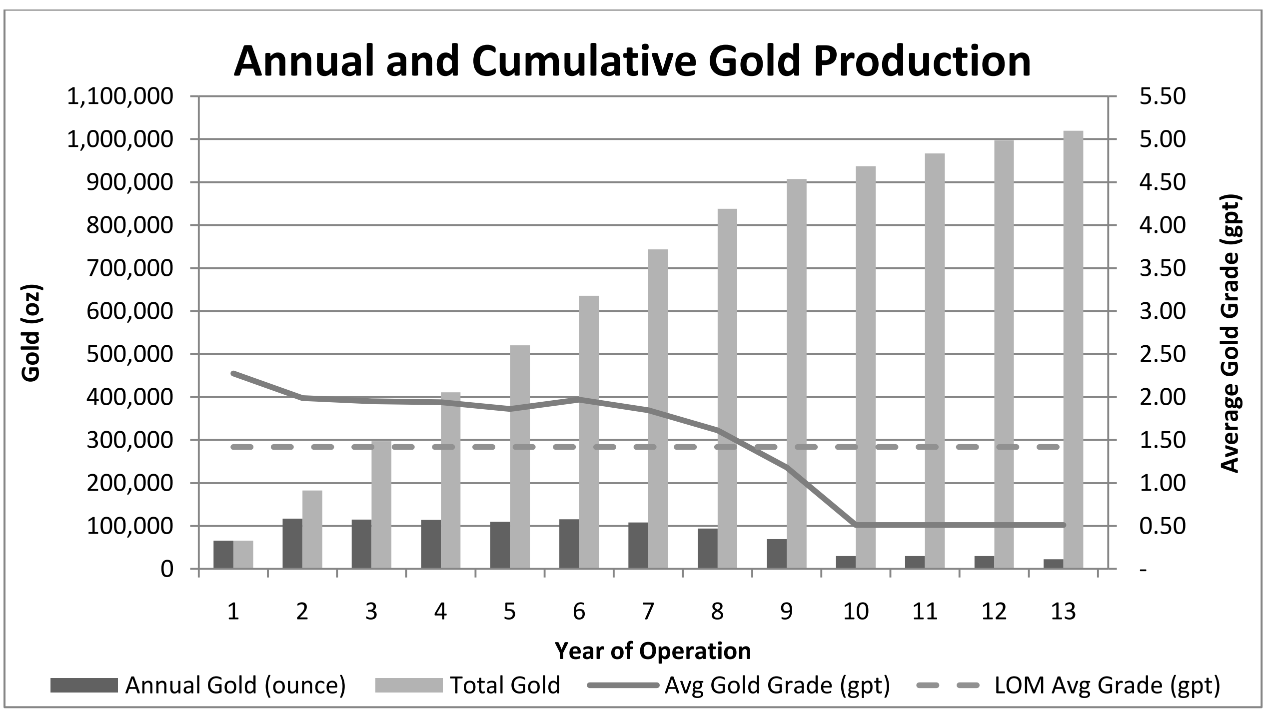

Figure 16-2: Annual and Cumulative Gold Production with Annual Gold Grade and LOM Average Gold Grade

The annual LG ore production and the LG stockpile balance is represented in Figure 16-3. The annual production of LG ore in Years 1 – 4 has been reduced by the quantity (1,065,824 t) required to backfill the existing stopes. The LG ore stockpile is estimated to reach approximately 7 million tonnes during Year 9 of operation.

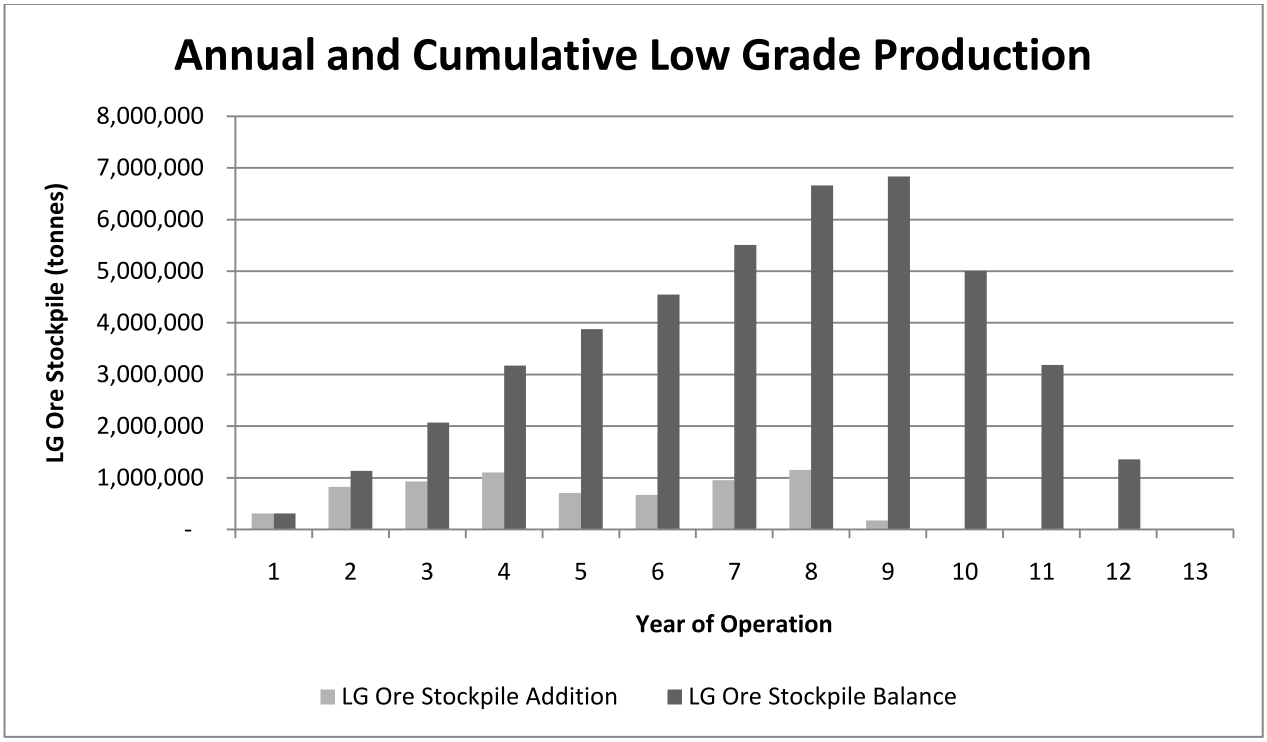

Figure 16-3: LG Ore Mined to Stockpile and Stockpile Balance

Box will generate 75,228,132 tonnes of waste for a LOM strip ratio of 4.56. The Box waste has been classified as waste associated with production and waste associated with expansion. Figure 16-4 illustrates a cross section of the Box pit showing four production phases and the relative proportion of production waste (Prod) to capitalized development (CapDev) waste. Total production waste is 41,001,831 tonnes and total capitalized development waste is 34,226,304 tonnes. The cross section also illustrates the ore body and its relation to the production phases.

Athona will generate 6,423,778 tonnes of waste for a LOM strip ratio of 1.10. Due to the low strip ratio at Athona, the waste was all classified as production waste.

Total waste production for the Goldfields project is 81,651,910 tonnes for a project strip ratio of 3.66. Figure 16-5 shows the annual waste production from Box and Athona. Years 10 – 13 of operations have no waste generation since the process feed is from the LG stockpile.

The annual pit plans for Box are provided in Figures 16-6 to 16- 12. Annual pit plans for Athona included in Figures 16-13 to 16-15.

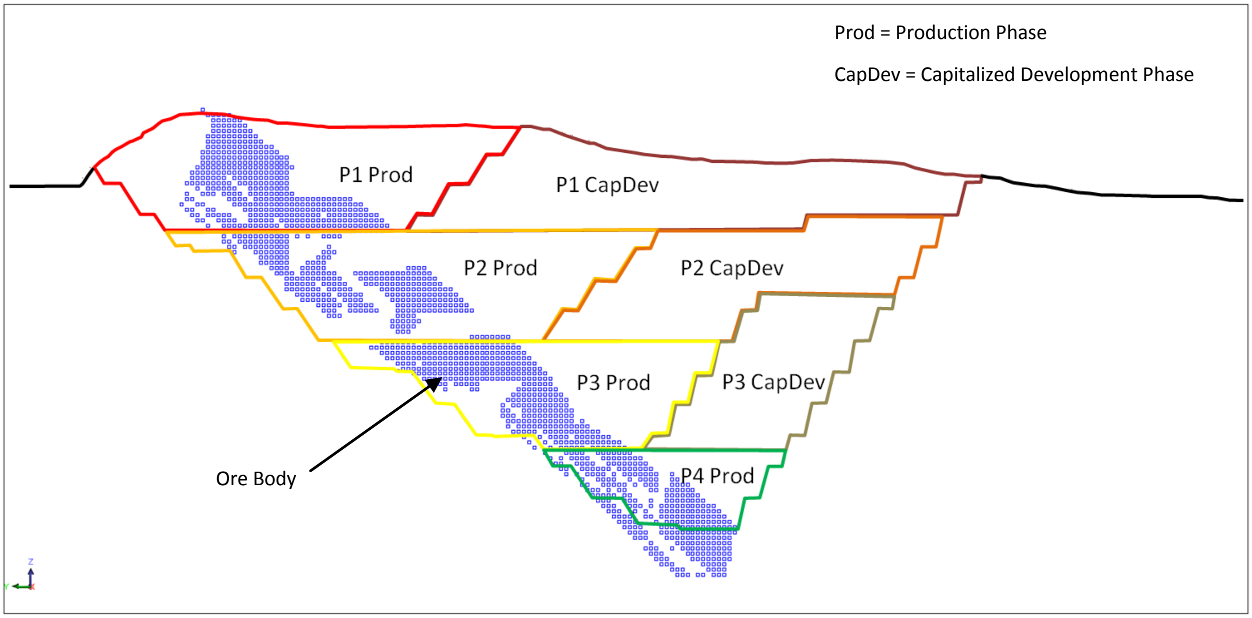

Figure 16-4: Box Pit Cross Section Showing Four Phase Production Strategy

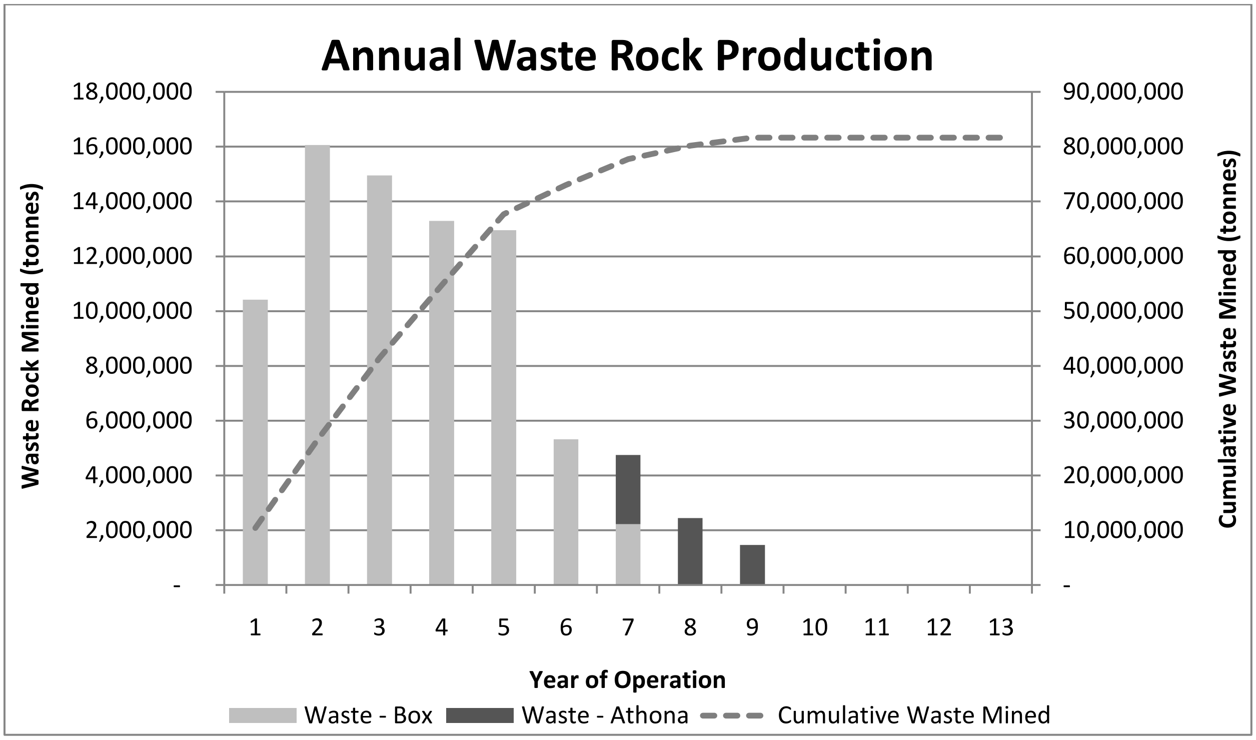

Figure 16-5: Annual Waste Production from Box and Athona

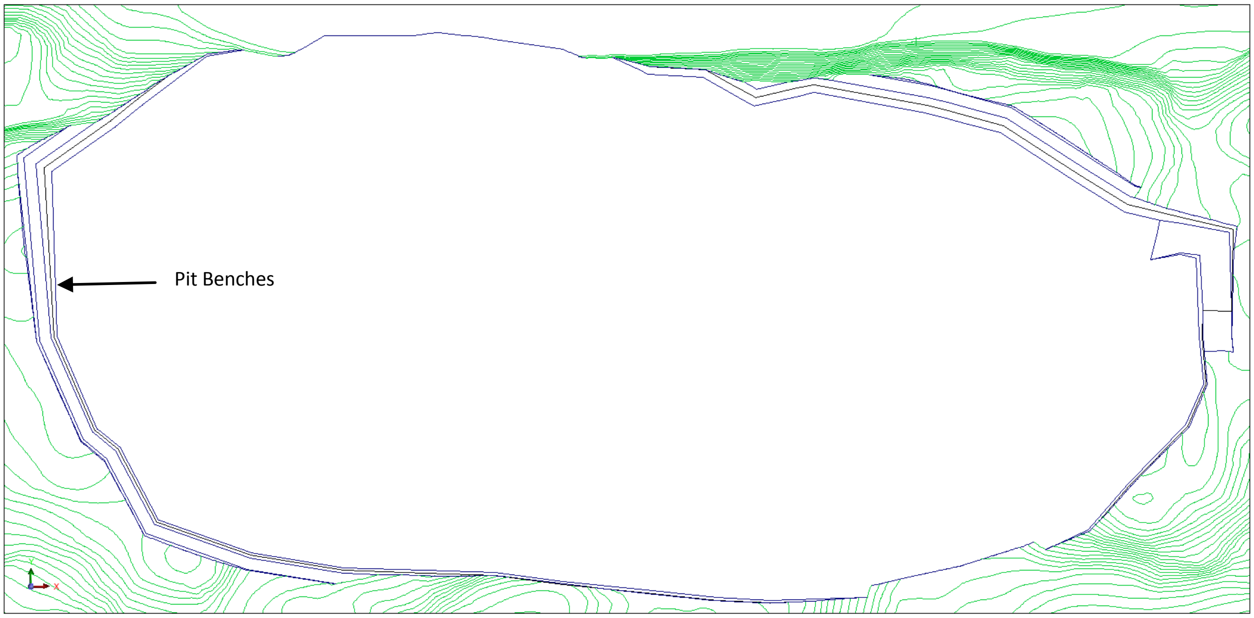

Figure 16-6: Year 1 Pit Plan for Box

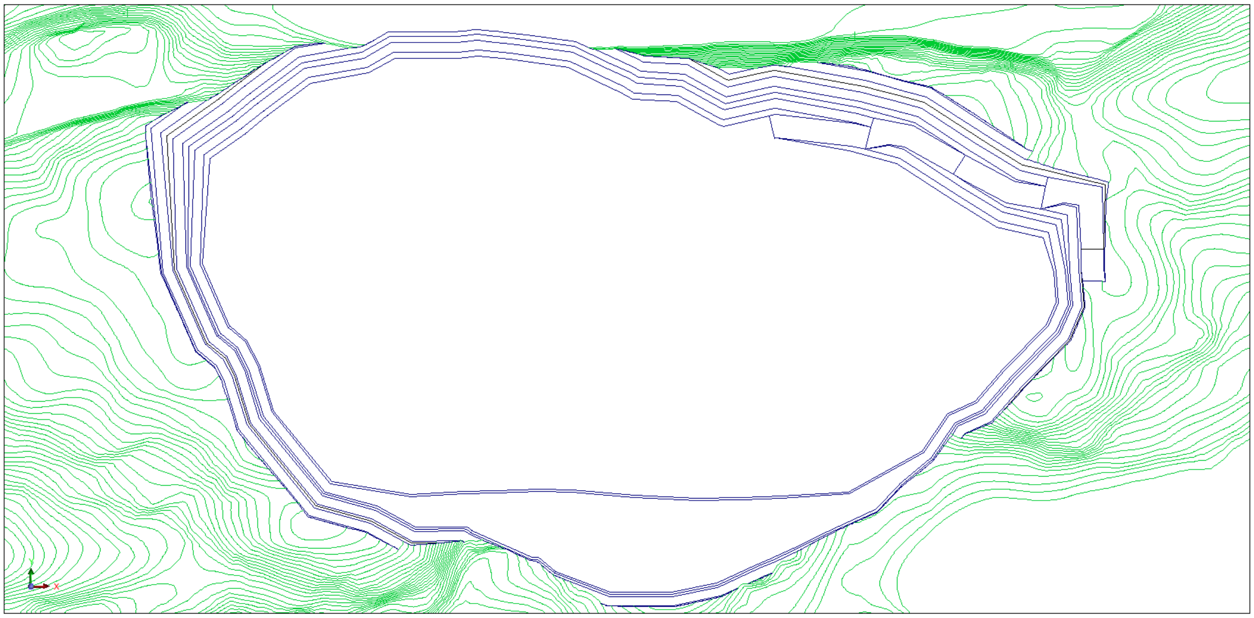

Figure 16-7: Year 2 Pit Plan for Box

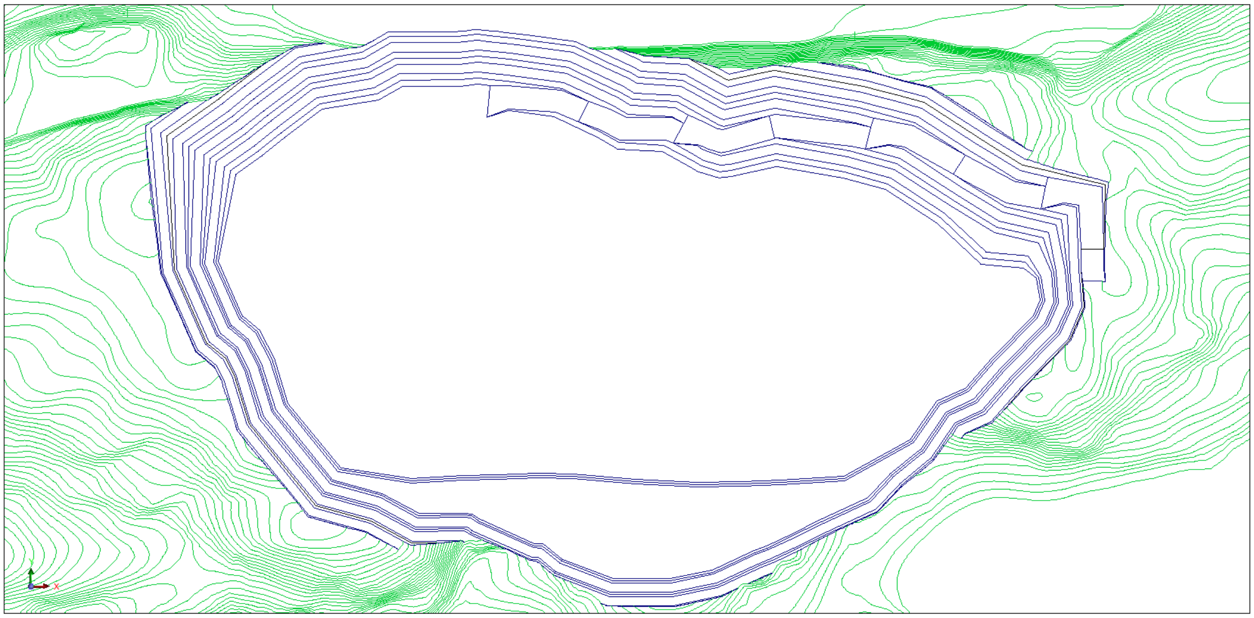

Figure 16-8: Year 3 Pit Plan for Box

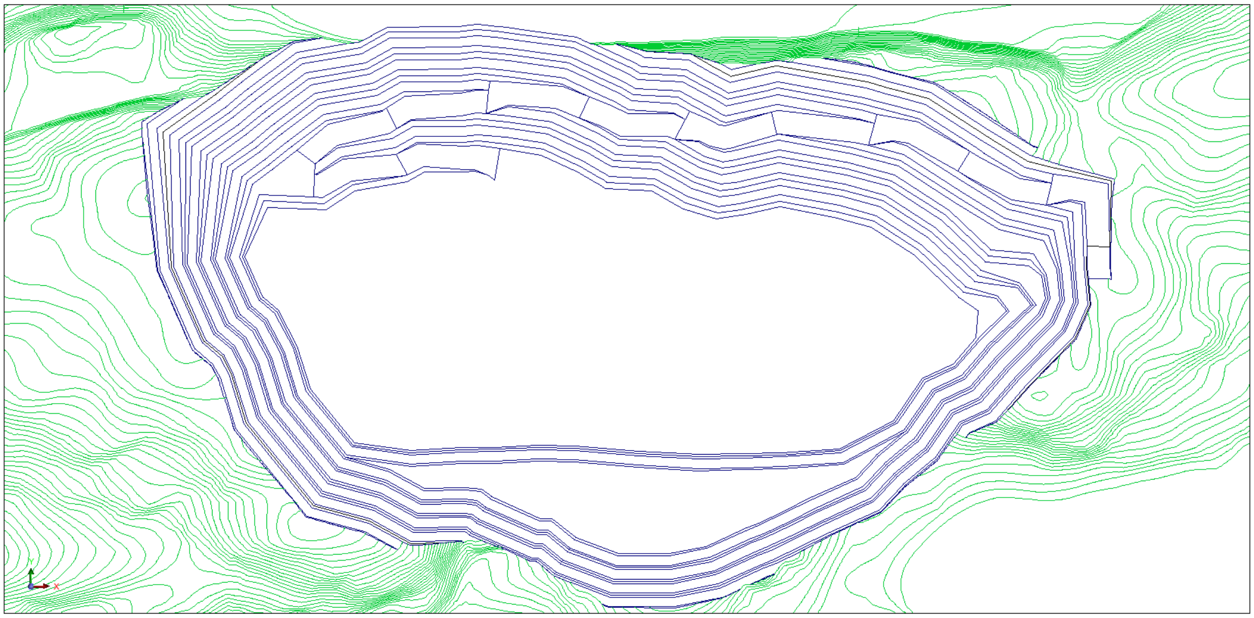

Figure 16-9: Year 4 Pit Plan for Box

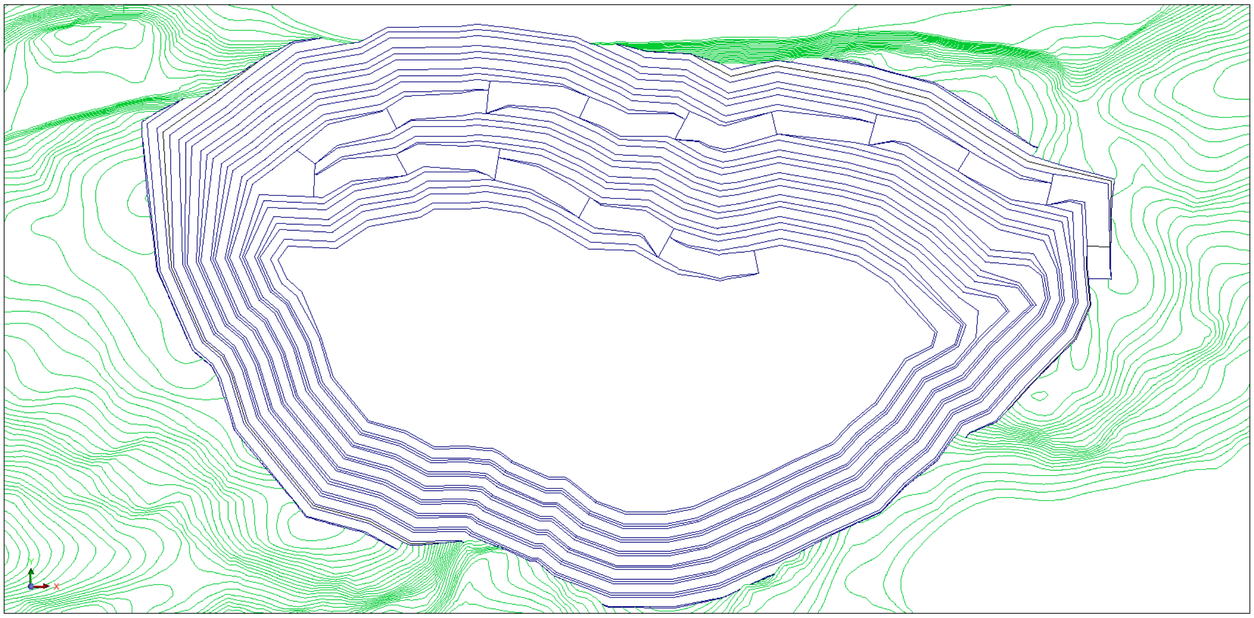

Figure 16-10: Year 5 Pit Plan for Box

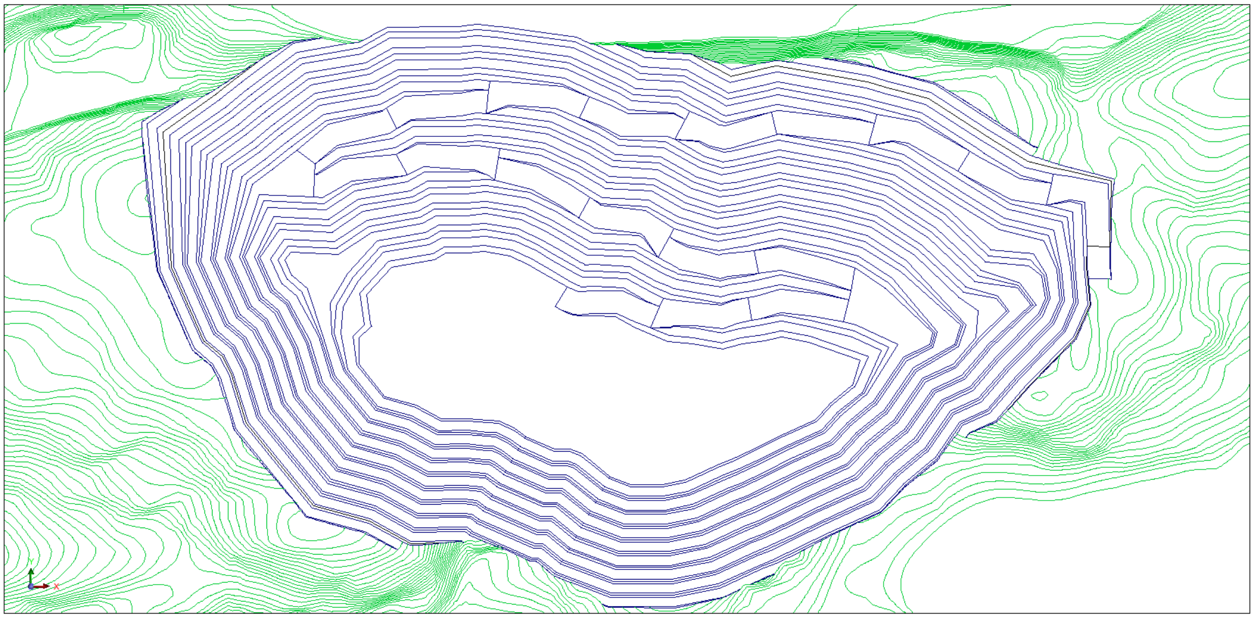

Figure 16-11: Year 6 Pit Plan for Box

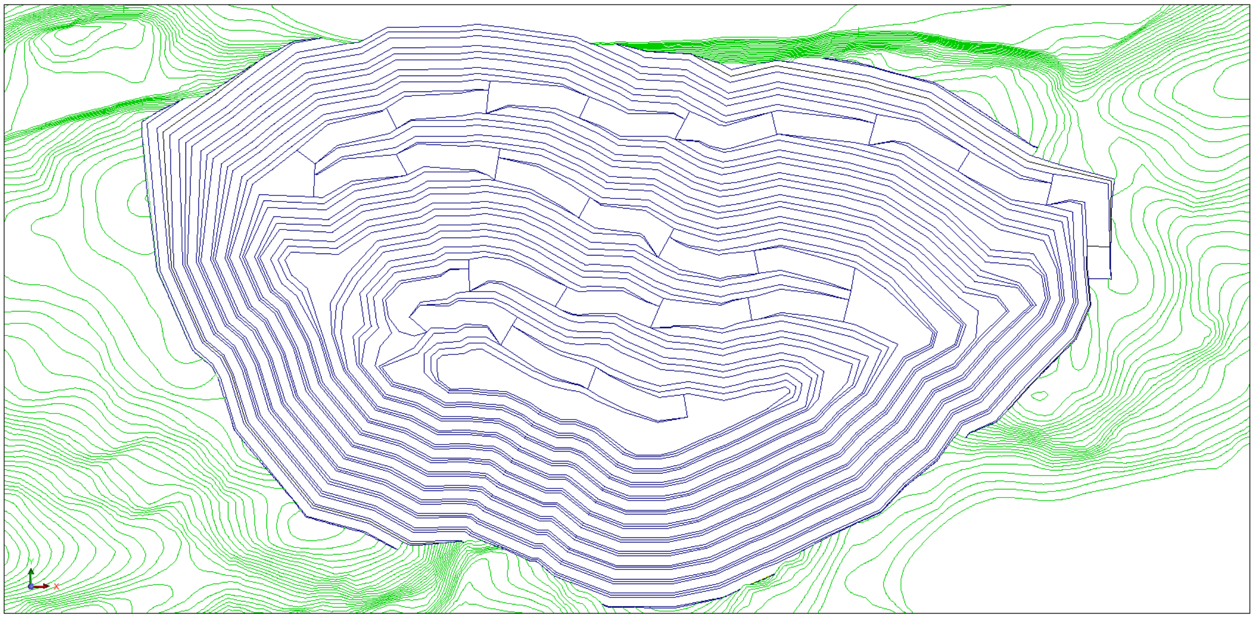

Figure 16-12: Year 7 (final) Pit Plan for Box

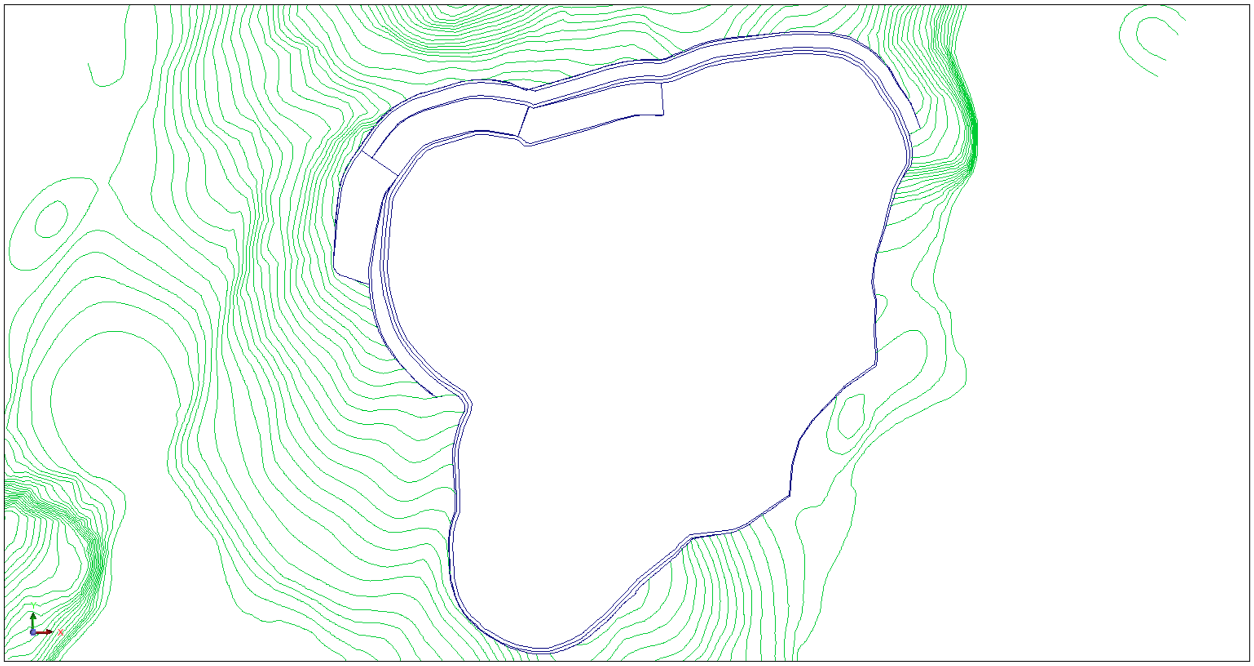

Figure 16-13: Year 1 Pit Plan for Athona

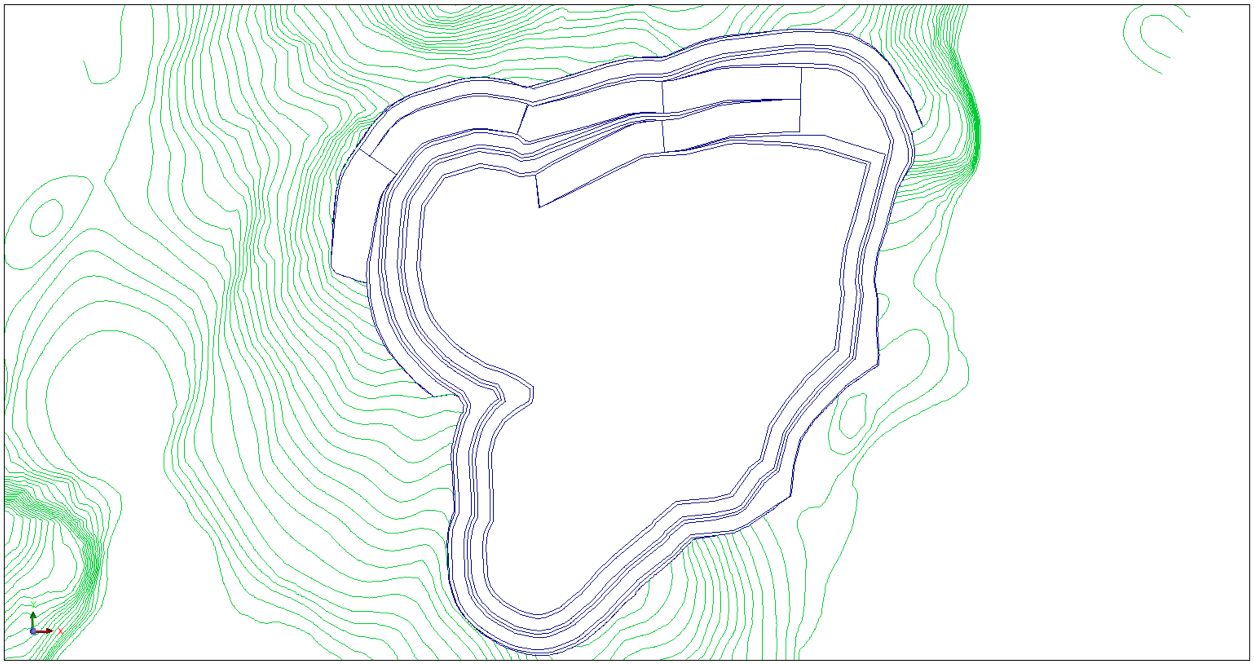

Figure 16-14: Year 2 Pit Plan for Athona

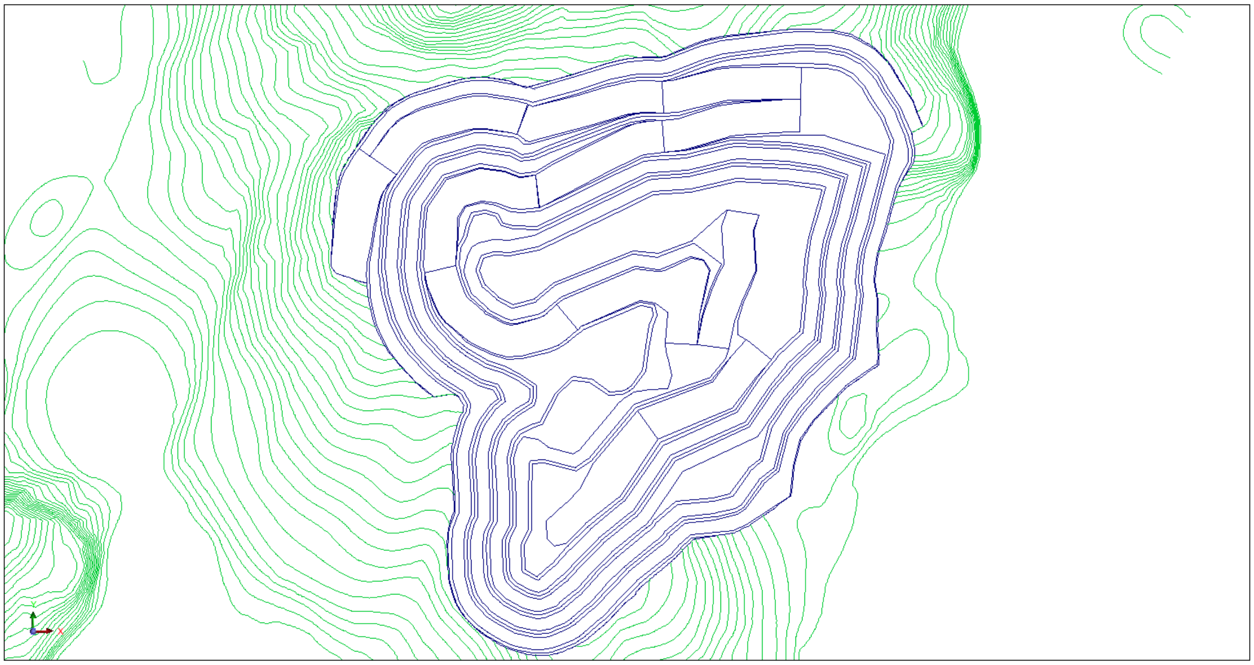

Figure 16-15: Year 3 Pit Plan for Athona

16.5 MINING EQUIPMENT

The mining equipment was selected based on the total daily open pit production of ore and waste. Table 16-3 shows pit production is highest in Year 1 with an average of 65,500 t/d reducing to 20,600 t/d in Year 7 and 9,500 t/d in Year 9. Mining equipment was selected to meet these production targets.

16.5.1 EQUIPMENT CYCLE TIMES

Equipment cycle times were calculated to assist with the selection of mining equipment. The cycle times include a fixed and a variable portion. Fixed cycle times include the loading and unloading activities that remain constant throughout the operation. The variable cycle time applies to the hauling portion which increases as the pit becomes deeper and the WRSA becomes higher.

In order to determine the fixed cycle times, it was assumed that the equipment for loading would require four passes to fill each truck. The required time per pass was determined from the manufacturer's equipment specifications. The truck unloading time is also included in the fixed cycle time. The fixed cycle time was assumed to be consistent for the LOM.

The variable cycle time is made up of three segments to determine the haul time: open pit ramp length, haul distance, and WRSA height. The time required to transit the haul road is consistent for LOM, but the other two sections are variable and increasing throughout. During initial operations the variable cycle time is at a minimum due to the shallow pit and minimal height of the WRSA. At the end of life the variable cycle time is at a maximum. Table 16-3 shows the average cycles times per year of operation for Box and Athona. A mechanical availability of 85% was used for determining the required fleet size.

Table 16-3: Haul Truck Fleet Simulation Summary

Year	Total Daily Open Pit Production (tonnes)	Estimated Average Cycle Time (min)	Number of Trucks Required
Box			
1	65,500	13.5	9
2	51,100	15.5	9
3	46,700	17.6	9
4	43,300	21.0	10
5	43,000	24.0	10
6	21,400	24.9	6
7	20,600	27.6	6
Athona			
1	14,900	18.5	3
2	9,500	20.0	3

16.5.2 EQUIPMENT SELECTION

Ninety tonne capacity haul trucks were selected based on production requirements and professional experience. Caterpillar's 777F haul truck was selected for this analysis and its selection dictated the specifications for the other mining equipment. To match the capacity of the haul trucks Hitachi's EX1900 hydraulic shovel, which has the capacity to meet the four pass loading requirement, was selected. Caterpillar's representatives recommended the 992K wheel loader for back-up to the main loading equipment, the D10T tracked dozer to handle the stockpile maintenance, and the 16M grader for haul road maintenance.

Haul profiles were generated for each material type, bench and destination or stockpile. Production cycle times, which were calculated for all of these, were used to determine the quantity of haul trucks and primary loading units required. Table 16-4 provides the summary of the hydraulic shovel sizing.

Table 16-4: Summary of Hydraulic Shovel Sizing

Description	Input / Result
Ore	
Loose ore bulk density	2.038
Percent swell	30 %
Load factor	0.77
Haul truck	
Payload	90.7 tonnes
Capacity	45 m^3
Shovel	
Estimated fixed time	35 sec
Truck positioning	51 sec
Dipper size	13 m^3
Bucket fill factor	0.90
Bucket capacity	11.7 m^3
Required passes per truck	4

Table 16-3 shows the number of trucks required for each year of the mine operation. The equipment required for the first year of operation includes nine 777F trucks, two EX1900 shovels, one 16M grader, one 844K wheel dozer, and one D10T tracked dozer. Due to the remote location of the project, redundancy for critical equipment is required. The operational redundancy will include:

- a CAT992K loader to provide primary loading and mill feed back-up
- a CAT 14M grader for light vehicle road maintenance and back up haul road maintenance
- a D10T tracked dozer for back-up at the WRSA and ore stockpiles

The blasthole drills were sized based on the drilling and bench requirements. Two types of blasthole drills were specified for the project to provide versatility:

- Single-pass blasthole drill (1) – Sandvik D55SP
- 100 – 200 mm DTH Crawler (2) – Sandvik QXR 1320 DTH

The major mining equipment required for the operation is listed in Table 16-5. As indicated in Table 16-3 an additional haul truck will have to be added to meet production requirements in Year 4.

Table 16-5: Major Mining Equipment

Equipment	Make & Model	Qty
Hydraulic Shovel (excavator)	Hitachi EX1900-6	2
Off-highway Haul Trucks	Cat 777F	9
Wheel Loader	Cat 992K	2[a]
Motor Grader	Cat 16M	1
Wheel Dozer	Cat 844H	1
Track Dozer	Cat D10T	2
Blasthole Drill DTH Crawler	Sandvik QXR 1320	2
DTH Single Pass Blasthole Drill	Sandvik D55SP	1

Note:
a: The 992K loader provides back-up for the mill feed loader and the primary loading equipment in the pit.

Table 16-6 provides a list of the auxiliary equipment required to support the operation. This includes the equipment service and site service equipment as well as equipment specified for use for environmental monitoring.

Table 16-6: List of auxiliary equipment.

Equipment	Make & Model	Qty
Motor grader	CAT 14M	1
Fuel & lube truck	-	1
Fuel truck	-	1
Mechanic's service truck	-	1
Tire manipulator	CWS	1
Water truck (5,000 gal)	-	1
Telehandler	CAT TL1055	1
Intergrated tool carrier	CAT IT62H	1
Forklift (warehouse)	CAT EP6000	1
Rough Terrain crane (65 – 75 t)	-	1
Fire truck	-	1
Container handler	Hyster Yardmaster II	1
Vacuum truck	-	1
Sanding/snow plow truck	-	1
Ambulance	-	1
Employee transport bus (55 passenger)	-	2
Pickup trucks	-	8
Light plants	-	8
Off-highway tractor	Western Star 4900SA	1
Lowboy trailer (90 – 100 t)	Aspen 100 ton	1
1" – 4" HDPE pipe fusing machine	-	1
4" – 12" HDPE pipe fusing machine	-	1
Environmental monitoring boats	-	2
Environmental monitoring ATVs	-	2

17 RECOVERY METHODS

17.1 MILL PROCESS PLANT

17.1.1 INTRODUCTION

The mill facility will be located in a natural valley northeast of Vic Lake. Site drainage from all mill facilities will report to Vic Lake. To minimize site preparation costs and to take advantage of the natural terrain, the mill facility is separated into three components: crushing, crushed ore storage, and grinding and leaching. Separating facilities minimizes the fill required for the building foundations.

The mill will be designed to operate 365 days per year with an annual availability of 94%. The mill will process gold ore from Box with an average grade of 1.97 grams of gold per tonne milled (g/t) up to Year 7 of operations. The Athona average grade is 1.61 g/t for Years 7 – 9. At the end of mine life for Athona, the LG stockpile with an average grade of 0.51 g/t will be processed. The target annual throughput is 1,825,000 tonne with overall gold recovery of 91% for Box and 89% for Athona. The simplified mill process flow diagram (PFD) is shown in Figure 17-1.

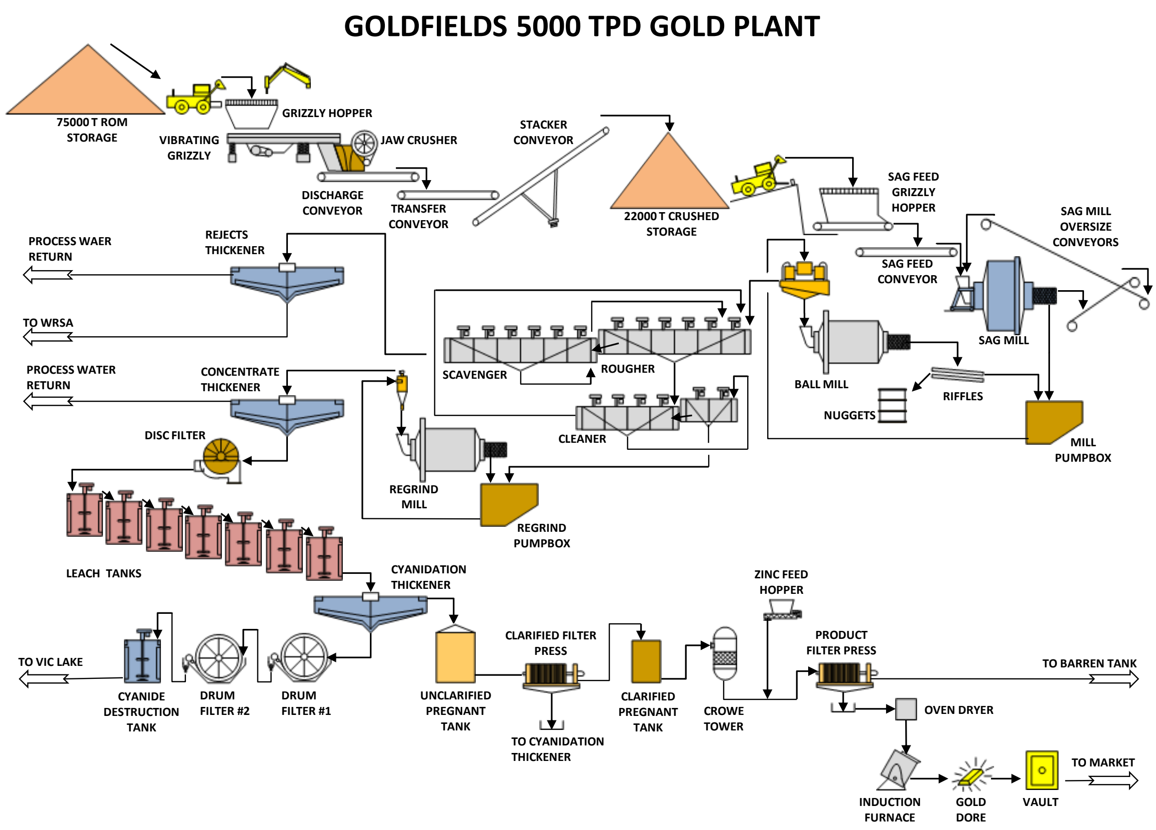

Figure 17-1: Simplified PFD.

17.1.2 SPECIFICATIONS

The general production specifications are summarized in Table 17-1:

Table 17-1: General Production Specifications

Area	Description	Units	Values
Production Criteria	Average tonnage	t/d	5,000
	Mill operating	d/y	365
	Mill mechanical availability	%	94
	LOM ore grade Au	g/t	1.42
	Gravity/flotation loss	%	8.0
	Cyanidation tailings loss	%	0.8
	Cyanidation soluble loss	%	0.3
	Total loss	%	9.1
	Total recovery	%	90.9

17.2 CRUSHING

17.2.1 DESCRIPTION

Ore from the open pit or stockpiles will be transported by haul truck to a static primary grizzly with a 914 mm opening. A hydraulic rock breaker (2,710 Nm class, 37 kW hydraulic) is stationed at the grizzly to disperse oversize lumps. A 1,219 mm x 4,877 mm vibrating grizzly feeder directs the coarse ore from the hopper to the jaw crusher (1,280 MM X 1,550 MM, 260 kW). The fines bypass the jaw crusher and discharge directly to the crusher discharge belt. The crusher has a capacity of 484 tonnes per hour (t/h) and discharges 150 mm (F80) product to the crusher discharge belt to the radial ore stacker. The stacker distributes the ore to the 20,000 tonne crushed ore stockpile. A tramp steel magnet and metal detector are installed to protect the belt conveyors.

17.2.2 SPECIFICATIONS

The specifications for Crushing are shown in Table 17-2.

Table 17-2: Crushing Area Specifications

Area	Variable	Units	Value
Crushing	Operating hours per day	h	11
	Operating days per week	d	7
	Throughput	t/h	484
	Bulk density	t/m^3	1.4
	Solids SG		2.67
	Moisture content	%	3
	Feed size distribution (P80)	mm	150
	Max feed lump size	mm	914
	Material repose angle	°	38

17.3 GRINDING

17.3.1 DESCRIPTION

Ore will be reclaimed from the crushed ore stockpile with a front-end loader and dumped into the 440 tonne apron feeder hopper. The apron feeder hopper holds approximately two hours of storage to guard against any potential loader downtime. The apron feeder (4.5 m) discharges into a transfer conveyor at a controlled rate of 220 tph. The transfer conveyor is equipped with a belt scale that measures the feed rate going to the SAG mill.

Process water is added to the SAG mill (6.0 m x 3.3 m, 1,700 kW) to discharge the ground material at 67% solids (w/w). The SAG mill uses 127 mm (5 inch) steel balls as a grinding media.

The ground material discharges through a trommel, the undersize material (U/S) drops to the pump-box and the oversize material (O/S) is returned to the SAG mill via a conveyor belt recycle loop.

The trommel U/S at 53% solids (w/w) along with ball mill discharge material is pumped to the cyclone cluster consisting of two 660 mm diameter units. The cyclone overflow (O/F) at 35% solids (w/w) and consisting of material with a P80 of 150 microns is directed to the agitated flotation conditioning tank.

The cyclone underflow (U/F) at 70% solids (w/w) feeds a ball mill (4.5 m x 7.3 m, 2,300 kW), which completes the grinding circuit. Since this material could contain coarse gold, the ball mill discharge is passed through a riffle box before it is directed to the pump-box and mixes with the SAG trommel U/S. Any coarse gold recovered in the riffle is placed in a drum for transport to the gold refining process. Process water is added to the ball mill to maintain 67% solids (w/w) discharge. The ball mill uses 38.1 mm (1.5 inch) steel balls as a grinding media.

17.3.2 SPECIFICATIONS

The specifications for Grinding are presented in Table 17-3.

Table 17-3: Grinding Area Specifications

Area	Variable	Units	Value
Grinding	Bond Work Index (Wi)	kWh/t	16.4
	SAG mill feed, F80	microns	150,000
	Cyclone overflow, P80	microns	150
	Feed rate	t/h	220
	SAG mill discharge	% solids (w/w)	67
	Ball mill discharge	% solids (w/w)	67
	Circulating load ball mill	%	200
	Primary cyclone overflow	% solids (w/w)	35
	Primary cyclone underflow	% solids (w/w)	70

17.4 FLOTATION / REGRIND

17.4.1 DESCRIPTION

The cyclone overflow from grinding flows by gravity to the agitated flotation conditioner tank where the promoter and xanthate reagents are added. The conditioner tank discharges into the rougher feed box connected to the first stage of the rougher flotation bank where MIBCfrother is added.

The rougher flotation stage consists of six 20 m^3 agitated tank style cells operating in series. Rougher flotation concentrate representing about 10% of the feed is pumped to cleaner flotation cells for further separation of the gangue and concentrate.

The tailings from the rougher flotation stage proceeds to six 20 m^3 scavenger cells for recovery of gold that did not float during the roughing stage. The discharge from the last scavenger tank is pumped to the rejects thickener. The concentrate from the scavenger flotation stage is pumped back to the first rougher flotation tank.

The cleaner flotation stage consists of two 5.1 m^3 re-cleaner DR style tank cells and four 5.1 m^3 cleaner DR tank cells. The concentrate from the rougher stage enters the intermediate box between the cleaner and the re-cleaner banks for polishing. The concentrate from the cleaner cells is fed to the re-cleaner cells for further polishing. The tailings from the cleaner cells are returned to the rougher flotation feed. The concentrate from the cleaner cells enters the re-cleaner bank through the feed box.

The concentrate from this bank is reground in the flotation regrind ball mill (2.2 m x 4.5 m, 224 kW) to ensure additional liberation of the gold prior to cyanidation. The tailings from the re-cleaner bank go to the cleaner bank as feed to the cleaning stage. The regrind mill discharge together with the re-cleaner concentrate is classified in a cyclone and the overflow is directed to the concentrate thickener. The cyclone underflow is returned to the regrind mill for further grinding.

17.4.2 SPECIFICATIONS

The specifications for Regrind are presented in Table 17-4. The specifications for Flotation are shown in Table 17-5.

Table 17-4: Regrind Area Specifications

Area	Variable	Units	Value
Regrind	Ball mill circulating load	%	200
	Regrind cyclone underflow	% solids (w/w)	70
	Regrind cyclone overflow, P80	microns	50

Table 17-5: Flotation Area Specifications

Area	Variable	Units	Value
Flotation	Concentrate solids specific gravity		4.0
	Rougher concentrate	wt % (of ore)	10
	Scavenger concentrate	wt % (of ore)	3
	First cleaner concentrate	wt % (of ore)	6
	Second cleaner concentrate	wt % (of ore)	2.5
	Concentrate launder	% solids (w/w)	15
	Rougher conditioner retention time	min	2
	Rougher retention time	min	15
	Rougher scavenger retention time	min	15
	Cleaner retention time	min	6
	Cleaner scavenger retention time	min	<1
	Secondary cleaner retention time	min	6
	Tertiary cleaner retention time	min	4

17.5 REJECTS THICKENER

17.5.1 DESCRIPTION

The tailings from the scavenger flotation are pumped to the rejects thickener (20 m, high rate, bolted) to recover water for recycling and to thicken the solids prior to pumping waste to the WRSA. The thickener overflow is returned to the process water tank for re-use in the mill. To expedite the settling of solids, flocculant is added to the thickener feed. The thickener underflow is pumped to the WRSA via the waste rock storage tank. The rejects thickener will be located outside the mill and will require freeze control measures like insulation, cladding, and covers to prevent freezing.

17.5.2 SPECIFICATIONS

The specifications for Reject Thickener are presented in Table 17-6.

Table 17-6: Reject Thickener Specifications

Area	Variable	Units	Value
Tails Thickening	Thickener underflow	% solids (w/w)	60
	Thickener overflow	% solids (w/w)	0.005
	Unit area	m^2/tpd	0.2
	Concentrate repulp tank retention time	h	4

17.6 CONCENTRATE THICKENING

17.6.1 DESCRIPTION

The feed to the concentrate thickener (7 m, high rate, bolted) consists of the flotation cyclone O/F, vacuum disk filter filtrate and flocculant. The clear O/F from the thickener is recycled to the process water tank and the U/F which contains about 70% solids (w/w) is pumped to the concentrate vacuum disk filter. The filtrate is returned to the concentrate thickener. The filter cake, at approximately 80% solids (w/w) is re-pulped to approximately 55% solids (w/w) and pumped to the pre-aeration tank in the leaching circuit.

17.6.2 SPECIFICATIONS

The specifications for Concentrate Thickener are presented in Table 17-7.

Table 17-7: Concentrate Thickener Specifications

Area	Variable	Units	Value
Concentrate L/S Separation	Thickener underflow	% solids (w/w)	70
	Thickener overflow	% solids (w/w)	0.005
	Unit area	m^2/tpd	0.15
	Filtrates	% solids (w/w)	0.2
	Disc filter cake	% solids (w/w)	80
	Disc filter unit area	kg/h m^2	942
	Process water retention time	h	2

17.7 CYANIDE LEACHING

17.7.1 DESCRIPTION

The re-pulped slurry is received in the pre-aeration mix tank (4.2 m x 4.9 m) where lime is added to adjust the pH prior to adding the sodium cyanide lixiviant. The slurry overflows the pre-aeration mix tank to six (6) mechanically agitated leach tanks (4.2 m x 4.9 m) in series. Air is added to each tank to maintain oxidizing condition in the tanks. Provisions are made for adding lime and sodium cyanide to the first and fourth tanks in the series.

The leached slurry discharges to the cyanidation thickener (7m, high rate, bolted) for separation of the pregnant solution and the leached residue. The thickener underflow which is approximately 65% solids (w/w) is pumped to the drum filter #1 surge tank. The solution overflow is pumped to the unclarified pregnant tank for further clarification of the solution.

17.7.2 SPECIFICATIONS

The specifications for Cyanide Thickening are presented in Table 17-8.

Table 17-8: Cyanide Thickener Specifications

Area	Variable	Units	Value
Cyanidation	Cyanidation feed density	% solids (w/w)	55
	Preaeration tank retention time	h	8
	Leach tank retention time	h	48
	Thickener underflow	% solids (w/w)	65
	Thickener overflow	% solids (w/w)	0.005
	Thickener unit area	m^2/tpd	0.3

17.8 LEACH FILTRATION

17.8.1 DESCRIPTION

The cyanidation thickener underflow is received in the drum filter #1 surge mix tank and fed to the cyanide tailings drum filter #1. Barren solution washes the gold bearing solution from the filter cake. This washing process is done twice to ensure maximum recovery of the dissolved gold in the solution before the residue is pumped to the cyanide destruction circuit. Water is substituted for barren solution as required to maintain a solution balance.

The filtrate from the first and second drum filters are combined and returned to the cyanidation thickener.

17.8.2 SPECIFICATIONS

The specifications for Leach Filtration are presented in Table 17-9.

Table 17-9: Leach Filtration Specifications

Area	Variable	Units	Value
Cyanidation	Drum filter feed tank surge capacity	h	8
	Drum filter cake solids	% solids (w/w)	80
	Filter 1 wash ratio (n)		1.1
	Filter 2 wash ratio (n)		1.1
	Filter R at n=1.1		0.2
	Filter rate	kg/h m^2	104.8
	Filter repulp density (stage1)	% solids (w/w)	55

17.9 CLARIFICATION

17.9.1 DESCRIPTION

The unclarified cyanidation thickener overflow is stored in the unclarified pregnant tank where it is fed to one of the two pre-coat filter presses. This step is essential to ensure the solution has minimum amount of suspended solids prior to the gold precipitation stage. Two pre-coat filter presses are required to provide continuous feed to the Merrill Crowe process as this filtration operation is a semi-continuous process. The clarified solution is stored in the clarified solution surge tank for feeding the Merrill Crowe de-aeration packed tower.

17.9.2 SPECIFICATIONS

The specifications for Clarification are presented in Table 17-10.

Table 17-10: Clarification Specifications

Area	Variable	Units	Value
Cyanidation	Precoat density	% solids (w/w)	20
	Clarifier sludge	% solids (w/w)	80
	Press sludge	% solids (w/w)	80
	Unclarified preg tank surge capacity	h	4
	Clarified preg tank surge capacity	h	4
	Filter press unit feed flowrate	m^3/h m^2	1.2

17.10 MERRILL CROWE PROCESS

17.10.1 DESCRIPTION

The clarified pregnant solution is passed through the Merrill Crowe deaeration tower to remove dissolved air which adversely affects the precipitation of gold. Zinc dust, added to the clarified, de-aerated pregnant solution, cements the gold.

The slurry containing the precipitate is filtered in two filter presses; one unit filters while the other is cleaned. The filter cake at 80% solids (w/w) is removed from the filter once the unit pressure rises to a pre-determined value. The cake drops to the product precipitate boat for transport to the gold room. The filtrate is directed to the barren tank for re-use in the process.

17.10.2 SPECIFICATIONS

The specifications for Merrill Crowe are presented in Table 17-11.

Table 17-11: Merrill Crowe General Specifications

Area	Variable	Units	Value
Merrill Crowe	Solids feed rate	t/h	0.001
	Slurry feed rate	t/h	11.3
	% Solids	% w/w	0.002

17.11 GOLD REFINING

17.11.1 DESCRIPTION

The filter cake from the filter press boat is dried in an oven. The dry filter cake is mixed with fluxes and fed to the crucible furnace (diesel fired) where separation of the gold from the impurities takes place. Doré bars are poured for shipment to a refinery. Slag will contain some gold and will be partially reprocessed by hand and returned to the process.

17.11.2 SPECIFICATIONS

The specifications for Gold Refining are presented in Table 17-12.

Table 17-12: Gold Refining Specifications

Area	Variable	Units	Value
Gold Refining	Charge	kg/week	200 – 250
	Operational frequency	-	Weekly
	Mould size	oz	1,000
	Power source type	-	Diesel

17.12 WATER REQUIREMENTS

17.12.1 DESCRIPTION

The preliminary mine-site water requirements are shown in Table 17-13. Water will be supplied from Lake Athabasca. A dedicated pump house will be installed to provide the site water requirements. Two water storage tanks will be located in close proximity to the mill.

17.12.2 SPECIFICATIONS

Table 17-13: Preliminary Mine Site Water (Fresh Water) Requirements

Location	Quantity (m^3/d)
Process Plant including cyanidation area	3,439
Road dust control	60
Equipment washing	20
Mine site domestic usage	50
Total daily water requirements	3,569

17.13 ENERGY USE

17.13.1 DESCRIPTION

Power will be supplied by a refurbished SaskPower distribution line that previously serviced the Goldfields town site. The supplied power is 115 kV that will be stepped down to 15 kV for site distribution at a dedicated site substation. Transformers will be installed to provide 600 V and 4,130 V to each facility as required by equipment demands. Table 17-14 summarizes the total project power requirements.

17.13.2 SPECIFICATIONS

Table 17-14: Estimated Energy Requirements

Item	Units	Value
Connected load	kW	8,917
Diversity/utilization	%	70
Average running load	kW	6,242
Energy consumption	kWh/t	29.96
Power factor corrected		0.95
Monthly demand	kVA	6,570

17.14 PROCESS MATERIALS

17.14.1 DESCRIPTION

The estimated consumables listed in Table 17-15 are required to support the operation of the process plant annually. Materials necessary to maintain the plant equipment in good operating order are not included in the list.

17.14.2 SPECIFICATIONS

Table 17-15: Estimated Consumables for the Operation of the Process Plant

Consumables	Tonnes/year
PAX	110
R208	37
MIBC	55
Cyanide	82
Lime	233
Flocculant	27
$CuSO_4$	5
Metabishulphite	296
Descalant	2
Precoat	456
5” balls	704
1.5” balls	1,261
Zinc	29
Lead nitrate	3
Flux	29

18 PROJECT INFRASTRUCTURE

18.1 SITE INFRASTRUCTURE LAYOUT

The preliminary site layout outlines the proposed locations of the WRSA, facilities, stockpiles, and pit locations as illustrated in Figure 18-1.

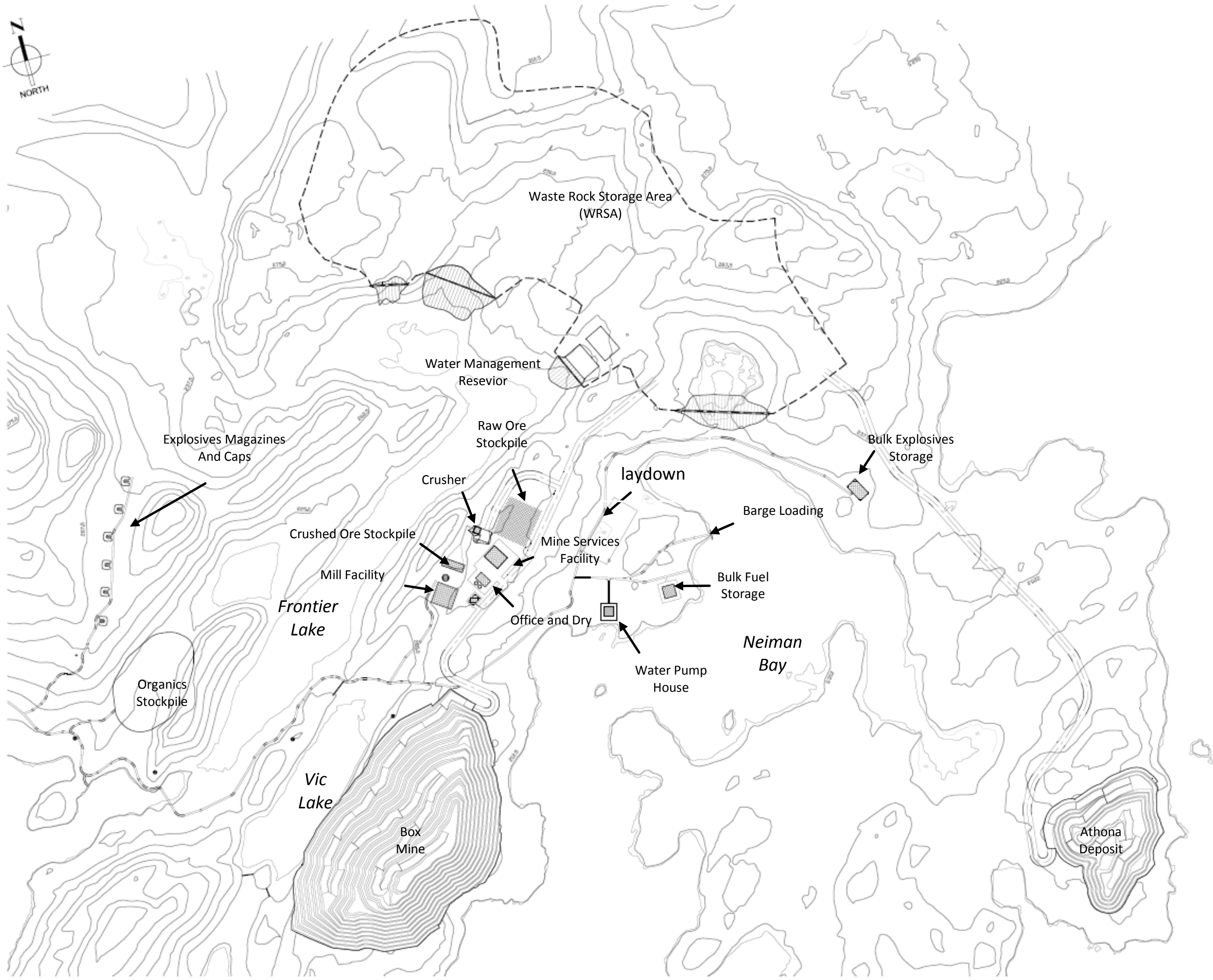

Figure 18-1: Infrastructure General Layout

18.2 MILL BUILDING

The mill building will incorporate the grinding, leaching, and processing areas and will be located in the valley northeast of Vic Lake. The building's foundation will be a concrete slab that is cast on backfill and bedrock. The backfill will be a blend of waste rock and valley overburden. The bedrock will come from blasting into the east valley slope and creating a flat surface for the concrete foundation. This area of the mill building will house the SAG and ball mills to minimize foundation shifting and concrete stresses and strains. The building will be a pre-engineered steel building with dimensions of 60 m x 60 m x 20 m. The building will have metal cladding with insulated walls suitable for northern climates. A preliminary mill facility layout is provided in Figure 18-2.

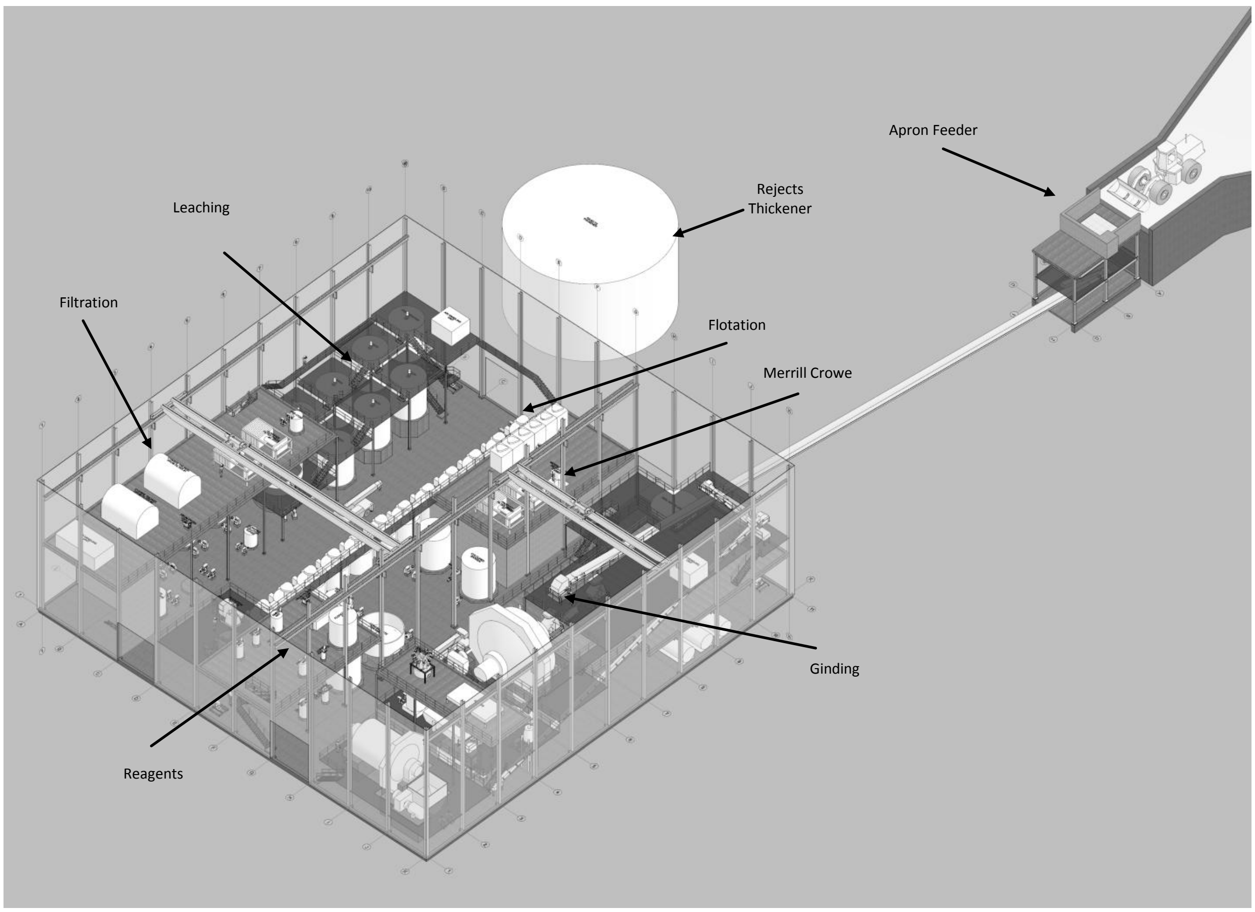

Figure 18-2: Isometric of the Preliminary Mill Plant Layout

The concrete for the foundation will be pre-mixed dry concrete shipped to site in tote bags (1 yd^3 or 0.8 m^3). This ensures quality concrete without the requirement to source local concrete aggregate. It also eliminates the requirement to set up a batch plant during the construction period. The required equipment for concrete preparation and delivery will be a ready-mix truck, front-end loader, water pump, and concrete pump.

All precipitation runoff from the mill building and crushed ore stockpile will go to the TMF. Facility heating will be provided by diesel fired boilers complete with heat recovery and air make-up units. The water system will be supplied from the fresh water storage tank. All waste water will be temporarily stored in above ground insulated and heat traced septic tanks and transferred to the TMF. Electrical power will come to the facility by overhead distribution.

18.3 MINE SERVICE COMPLEX

The mine service complex will include the maintenance shop and warehouse facilities and will be located on the ridge above the mill. The mine service complex will be a 55 m x 46 m x 14 m pre-engineered steel building. The building will have a concrete foundation that is cast on waste rock backfill and bedrock.

The mine service complex will include: repair bays (895 m^2), welding bay (99 m^2), office area (58 m^2), tool room (79 m^2), wash bay (254 m^2), maintenance shop (155 m^2), electrical and instrumentation shop (155 m^2), heated warehouse (625 m^2), parts office (20 m^2), and ambulance and fire truck garage (142 m^2). The second level will include: storage (753 m^2), mechanical room (79 m^2), and lunch room and washroom (58 m^2). A 20 tonne overhead hoist will span over the four repair bays. The building will have metal cladding with insulated walls suitable for northern climates. A preliminary mine services building layout may is provided in Figure 18-3.

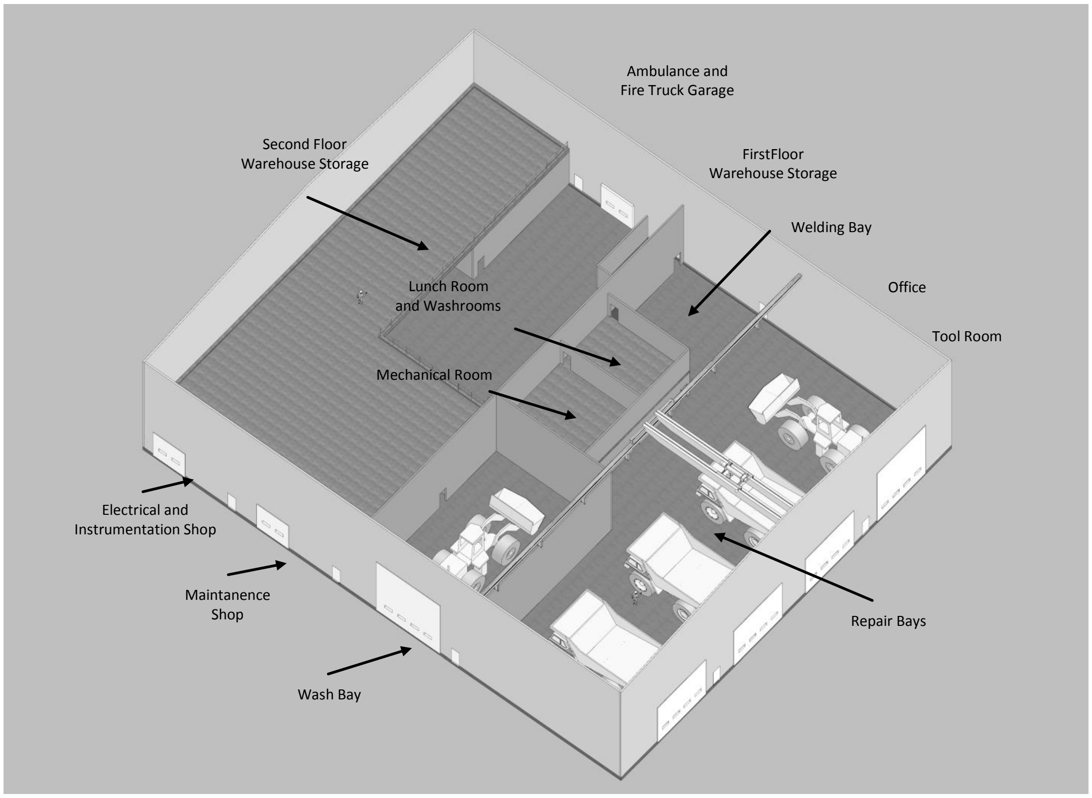

Figure 18-3: Isometric of the Preliminary Mine Service Complex Layout

Facility heating will be provided by diesel fired boilers complete with heat recovery and air make-up units. The water system will be supplied from the fresh water storage tank. All waste water will be temporarily stored in above ground insulated and heat traced septic tanks and transferred to the TMF. Electrical power will come to the facility by overhead distribution.

18.4 WASTE ROCK STORAGE AREA

The WRSA is located approximately 1,100 m from the Box pit. The WRSA will use engineered berms and the natural terrain to contain the waste rock, gravity rejects, and runoff from precipitation. The WRSA will cover approximately 105 hectares and will consist of four impermeable berms, three catch ponds for runoff control, and a water management reservoir with two settling ponds. There will be approximately 450,000 m^3 of waste rock material used to build the berms and settling ponds. The required waste rock will be generated during the construction and operations periods. The elevation of the WRSA is expected to reach a peak of 320 m ASL.

The waste rock will be placed in 10 m lifts with each subsequent lift inset from the previous lift. This inset interrupts surface runoff paths on the side slope of the stockpile and will help to retain suspended solids from the runoff. The overall stockpile side slopes will be 2H:1V to provide long term slope stability.

All water in the WRSA will report to the water management reservoir settling ponds where it will be monitored to verify compliance with the MMERrequirements prior to discharge. The storage ponds and settling ponds will be designed to handle a 100 year rain event in addition to the water contained in the gravity rejects from the mill facility. The settling ponds will have vertical baffles controlling the flow of water, allowing adequate retention time for suspended solids to settle and for the natural destruction of ammonia blasting residue if required. Water discharge from the water management reservoirs will be continuously discharged to Frontier Lake based on the results of upstream monitoring.

18.5 TAILINGS MANAGEMENT FACILITY

Tailings from the milling process will be deposited into the Vic Lake TMF through sub-aqueous deposition. Vic Lake is naturally contained by the local terrain with intermittent flow to Frontier Lake during periods of high runoff. Frontier Lake is located northwest of Vic Lake and is approximately 3 m lower in elevation than Vic Lake.

Two control structures are required to permit the operation of the Vic Lake TMF at an elevation of 220 m. All elevations are ASL. The north outlet structure will control the release of water to Frontier Lake. It will be constructed with waste rock material and lined with an impermeable liner system that is sealed to the bedrock. The outlet structure will have a top elevation of 221 m, to provide a 1 m free-board. The on-site light vehicle road will traverse the outlet structure as the main point of access to the mine site. A monitoring pond at the base of the outlet structure will permit sampling prior to release to Frontier Lake. At the south end of Vic Lake, a dyke will be constructed to prevent out flow to Box Bay, Lake

Athabasca. This dyke will be constructed using waste rock material and will be lined with an impermeable liner system that will be sealed to bedrock.

During the construction period it is proposed that the level of Vic Lake will be reduced to 215 m from its current elevation of 218 m. This would permit sealing the control structures to the underlying bedrock and grouting of fracture zones in the bedrock, if required, to limit seepage. It also provides the benefit of providing additional storage capacity to determine if process changes are required for the tailings to meet MMER discharge requirements. For example if cyanide levels in the tailings water are elevated above MMER requirements, contingency measures for the reduction of cyanide can be implemented. It is estimated that discharge to Frontier Lake would not be required until year three of operation.

Existing piezometers for the monitoring and characterization of ground water are installed between Vic Lake and Frontier Lake and between Vic Lake and Box Bay. Additional piezometers will be installed as required to monitor ground water quality.

18.6 PIPELINES

The raw water pipeline will run from Neiman Bay to two storage tanks located outside the mill. One storage tank will be used for process water and the other used for the water requirements of the other facilities. The raw water pipeline is 8” diameter high density polyethylene (HDPE) that will be insulated and heat traced. The raw water pipeline is 1,200 meters in length.

The tailings pipeline from the mill to the Vic Lake TMFwill be 2” diameter HDPE that will be insulated and heat traced. The tailings pipeline will be 500 meters long.

The gravity rejects pipeline transfers the gravity rejects slurry (approximately 60% solids) from the mill to the WRSA. The gravity rejects pipeline will be 10” HDPE that is insulated and heat traced and will be 1,800 meters long.

In the WRSA, the water (precipitation and gravity rejects water) will be transferred from within the waste rock pile to the water management reservoirs for monitoring and release to Frontier Lake. A portable diesel pump and 8” aluminum piping will be used to transfer the water as required.

If required, a diesel fuel transfer pipeline from the barge loading facility to the bulk fuel storage would be approximately 500 meters long and will be constructed of 3” diameter Schedule 40 carbon steel pipe.

All four pipeline systems will be located above ground due to the natural terrain. This will allow ease of visual inspection. Pile and service utilidors are not planned for any of the pipelines.

18.7 MINE OFFICE AND DRY FACILITY

The mine office and dry facility at site will be pre-manufactured modular buildings and will be located adjacent to the mine service complex. The office facilities will be located on the second floor above the dry facility. This will minimize the overall footprint and provide a structure that is more energy efficient.

The dry facility will accommodate 250 lockers on each of the clean and dirty sides. The dry will include men's and women's separate washroom and shower facilities and also a laundry facility for washing work clothing.

The first aid center (25 m^2) will have storage cabinets for typical first aid equipment and supplies and an area for treating two (2) patients. The first aid center will include two beds, a toilet, and sink. It will be located on the first level with double (1,830 mm) wide exterior doors and steps.

Operations office space close to the deployment point (dry area) will be approximately 60 m^2. Administration & technical office space will be approximately 200 m^2 on the second floor and will have separate washroom facilities.

The facility's heating requirements will be by fuel oil and all waste water will be collected in an above ground insulated and heat traced septic tank and hauled to the TMF. Electrical power will come to the facility by overhead distribution.

18.8 PROJECT RESIDENCE

The project residence will be located in Uranium City on existing Brigus Goldfields property. The existing building will be demolished and the site prepared for the modular camp facility. The facilities will be constructed using pre-manufactured modular buildings and will include the following: three 49 person dorms, a 5 unit kitchen/dining complex, a 4 unit recreation complex, and a 3 unit office complex

The facility will have a capacity of up to 150 and the site can accommodate one additional dorm if required during the construction period. The site will have a large parking lot behind the facility and a small lot off the street front. The camp facility will tie into town services.

18.9 POWER

Electricity for the project will be provided by SaskPower. An existing transmission line is currently in service to Uranium City. An abandoned line from Uranium City to Goldfields exists that will require refurbishment by the local utility. The abandoned line includes high voltage towers and conductors that can be reused with minor upgrades. The existing transmission network is 115 kV. On-site, a substation will be installed to drop site voltage to 15 kV for distribution.

The substation will be an outdoor air insulated unit arranged in a single bus scheme comprised of a transformer (10/12.5 MVA - 115/13.8 kV), protection system, metering system, grounding system, and switchgear room. The substation equipment will comply with appropriate codes, standards and SaskPower requirements.

The power distribution system at the mine site will be a combination of overhead lines and cable trays. Underground cabling is not a feasible option due to the mine site terrain being mainly bedrock. The overhead lines will include the 15 kV main power and 600 V emergency power to each facility. Step down transformers will be installed to suit the loads for each facility. The emergency power will be supplied by an on-site diesel generator. Emergency power will be designed to operate the key electrical loads to prevent damage to the operation in the event of a power outage. This will include essential pumps, agitators, heat trace, and facility heating.

18.10 BULK FUEL STORAGE

A bulk fuel storage facility will be installed with a total capacity of three million liters of diesel fuel. Fuel will be stored in dual walled Envirotanks. Total capacity is determined based on four months supply. Fuel shipments would be received in the winter via ice road and in the summer via barge service. The four months storage bridges the gap between the ice road and barge services. Only diesel fuel will be stored on site and all equipment will be specified to operate on diesel fuel including facility heating equipment. For equipment that is not available for use with diesel fuel, gasoline will be supplied through current storage and distribution located in Uranium City.

18.11 ROADS

18.11.1 OFF-SITE ROAD

The Goldfields project, located 13 kilometers southeast of Uranium City, is accessible by an existing 25 kilometer all-season road. A portion of the road is a portage point for the winter ice road from Lake Athabasca to Beaverlodge Lake. The existing road will be upgraded with a target average travel speed of 60 km/hr. Typical upgrades may include but are not limited to:

- clearing the right of way
- road widening
- improving the line of site
- improving the road’s crown, shoulders, grades, drainage, intersections, and turn outs
- upgrading horizontal and vertical alignments

18.11.2 ON-SITE HAUL ROADS

The haul road will be constructed with waste rock generated during the construction period. The road will be 25 meters in width with safety berms as required, will have a maximum gradient of 10%, and will accommodate two-way traffic. The road will have a typical ditch and culvert system to allow the natural flow of water runoff. The haul road distance from the Box pit to the mill crusher is approximately 1,100 m and also approximately 1,100m to the WRSA. The haul road distance from Athona will be approximately 2,400 m to the center of the WRSA and approximately 2,500 m to the mill crusher.

18.11.3 ON-SITE LIGHT VEHICLE ROAD

The light vehicle roads run throughout the site and will connect the off-site road to all the site areas. Light vehicle roads will be designed and constructed taking into consideration the volume and weight of expected traffic. Total length of light vehicle roads is 3,500 meters.

18.12 BARGE LOADING FACILITY

The mine site will include a facility for the loading and off-loading of barges. The facility will consist of a removable barge dock plus a transition ramp to the shoreline. The barge facility will be available during the open water season, which is typically June – October depending on weather conditions. The dock will be removed from the water when Lake Athabasca freezes over and barge transport is no longer feasible. The barge loading facility will be located near the bulk fuel storage and the site laydown area since the majority of material will be stored in these two locations and the shoreline is appropriate for the intended purpose.

18.13 LANDFILL

A project specific landfill will be established approximately four kilometers from the mine site. The landfill will be designed to accept all municipal solid waste and approved industrial and construction waste. The area is a low sloped valley and is surrounded by bedrock. Existing overburden is sand and gravel to bedrock. The area will be fenced to control litter and to control wildlife. Piezometers will be installed to monitor ground water and surface runoff from the landfill. Brigus will operate this landfill until the mine closes and reclamation is complete.

18.14 EXPLOSIVES AND CAP MAGAZINES

Pre-manufactured explosive and cap magazines with capacity to store a four month supply of packaged explosives will be installed west of the mine so that the facilities are compliant with the Quantity-Distance criteria of the Explosives Regulatory Division of Natural Resources Canada, Minerals and Metals Sector, and Explosives Safety & Security Branch.

The packaged explosives magazines will be type 4 units measuring 3.65 m x 12.2 m with a per unit capacity of 37,200 kg. Six of these pre-manufactured explosive magazines are required. Berms will be constructed around the magazines. The separation distance (D2) is 83 m between magazines, 275 m (D4) from very lightly traveled roads (from 20 to 500 vehicles per day), and 760 m (D7) from inhabited buildings.

The single detonator magazine will be a type 4 unit measuring 3.05 m x 6.09 m with a capacity for 720 cases of detonators. It will be located adjacent to the magazine site road before the explosive magazines.

Access to the explosives & detonator magazines road will be controlled with a lockable gate.

18.15 BULK EXPLOSIVES STORAGE

The bulk explosives plant will be located between the WRSA and Athona. The location of the plant satisfies the Quantity-Distance criteria of the Explosives Regulatory Division of Natural Resources Canada, Minerals and Metals Sector, and the Explosives Safety & Security Branch.

The fenced plant area will be 60 m x 35 m and will include the following:

- an area for storage of 600 tonnes of explosives grade ammonium nitrate (AN) in shipping containers
- an AN unloading facility consisting of a hopper and auger to transfer the AN prills to an overhead silo
- a pre-engineered insulated metal 9.1 m x 15.2 m bulk explosives truck garage and wash bay
- a portable workshop storage container
- a modular office

18.16 COMMUNICATION NETWORK

Telephone service in Uranium City is currently provided through a satellite based land line system (Sasktel). No cellular networks are present in the area. Internet service is provided by consumer based satellite systems.

Telephone service for the project residence and office in Uranium City would be provided by Sasktel through existing networks. Telephone service for the mine site would be supplied through voice over internet protocol (VoIP) service.

The internet service requirements for the project residence and office in Uranium City and the mine site will be supplied through a very small aperture terminal (VSAT)satellite based system. The VoIP service would connect through the VSAT.

On-site communication would be provided through the citizen band (CB) radio frequencies.

18.17 ORGANICS AND TOPSOIL STOCKPILE

To prepare for mine reclamation at the end of operation, some of the organics and topsoil that is removed during construction will be stockpiled for future use. The stockpile will be located on the west end of the property for easy access. Trees, organics, and topsoil will be collected and stored.

18.18 PRELIMINARY CONSTRUCTION SCHEDULE

The preliminary construction schedule was developed based on a three year construction duration. The assumption was that detailed design would be completed in parallel with the year 1 construction. The construction activities planned for year 1 are the preparation activities that do not require substantial engineering prior to commencement. The schedule is dependent on the approval of construction applications by the Saskatchewan Ministry of Environment. An approved EIS exists.

Based on the preliminary schedule, operations would commence in the second half of the year 3 construction period. Total construction duration is estimated at 26 months.

YEAR 1 CONSTRUCTION

The onset of site construction would be scheduled to commence in May of the first year of construction. The initial construction season would be utilized to prepare the mine site for the installation of the site infrastructure.

This would incorporate the following tasks:

- Clearing and grubbing
- Off site roads, on-site temporary roads and light vehicle roads
- Site grading
- Dewatering Vic Lake
- Barge loading facility installation
- Upgrade Lodge Bay causeway
- Prepare landfill
- Residence Facility
- Bulk Fuel Storage

YEAR 2 CONSTRUCTION

Year 2 would encompass the majority of the infrastructure construction. During this construction period the processing facilities, environmental containments, and general site infrastructure would be constructed.

This would include the following major tasks:

- Construction contractor mobilization
- Concrete foundations
- Mill superstructure
- Mine service complex
- Mill equipment
- Pipelines
- Electrical
- Bulk explosives storage
- Dykes and berms

YEAR 3 CONSTRUCTION

During the final construction season, the construction of the milling facilities would be completed in the first quarter. The construction tasks estimated for the first quarter of year 3 include the following:

- Complete mine services facilities
- Complete mechanical installation
- Complete the electrical installation
- Install the crushing area equipment
- Build the cell #1 dyke containment area
- Haul road

The second quarter construction on the site would include:

- Geo-synthetic installation
- Cell #2 dyke,
- Commissioning
- Demobilization of contactors

19 MARKET STUDIES AND CONTRACTS

19.1 MARKETS

Markets for the gold produced by the Goldfields operation are readily available. These 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 2011 was US$1,129/oz.

19.2 CONTRACTS

Brigus currently has contracts with consulting and service companies to complete the pre-feasibility engineering work, environmental work, process test work, exploration drilling, and pit wall slope studies. Currently there are no contracts material to the issuer that are required for property development, including construction, mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements.

19.2.1 DETAILED ENGINEERING

When Brigus advances the Goldfields Project to the detailed engineering phase, an engineering firm will be contracted to undertake the detailed engineering. This will include design of the mining, electrical, civil, environmental, process, open pit, and structural elements for the entire mine project. Brigus and the engineering company will work together to design the operation, to enable cost effective capital, operating, and decommissioning costs.

19.2.2 PROCUREMENT

Procurement of all services, mine equipment, and mill equipment will be completed once the engineering reaches a level to support purchase and contract decisions. Procurement can be handled by the engineering firm as part of an EPCM agreement or by an independent entity that specializes in project procurement.

19.2.3 CONSTRUCTION MANAGEMENT

Construction management will be required once the decision to construct the mine has been approved. The construction management services can be provided by the engineering firm as part of an EPCM agreement or by an independent entity that specializes in construction management.

19.2.4 PROCESS TESTWORK

Brigus has contracted with SGS Canada Inc. to conduct process testwork to confirm the process design, determine recovery rates, and characterize waste products. This work is on-going to be completed in the last half of 2011.

19.2.5 PIT WALL SLOPE STUDIES

Brigus has contracted with Klohn Crippen Berger to complete a geotechnical assessment of Box and Athona to determine maximum allowable pit wall angles for the pit design. This work is on-going to be completed in the last half of 2011.

19.2.6 CIVIL AND GEOMEMBRANE CONSTRUCTION

The civil work required for the Goldfields Project includes: clearing and grubbing, demolition, road upgrade, and site preparation. Where practical, local contractors will be utilized to conduct the work and to reduce mobilization costs for the heavy equipment.

The geomembrane systems that will be supplied and installed on the berms and dykes will be contracted out to a company specializing in geomembrane systems.

19.2.7 SITE FACILITY CONSTRUCTION

Where practical, a general contractor will be retained to provide construction of the site facilities. They will be responsible for all aspects of the construction including subcontractors. Contractors with experience working in Saskatchewan's north and fly-in operations would be preferred. This work has not been put out to tender and no contract has been signed for site facility construction.

19.2.8 LOGISTICS

Access to Uranium City and Goldfields is limited to air, barge, and winter ice road. The barge season is limited to June to October and is weather dependent. The winter ice road is typically available for six weeks in February and March. Air service is available year round to the Uranium City airport.

Air services will provide all transportation of non-local personnel during construction, operation, and decommissioning. Currently there are two air service providers that provide regular flights to northern Saskatchewan. This has not been put out to tender and no contract has been signed.

The access road to Stony Rapids (Highway 905) from Points North is very rough and trucking is therefore limited. Only logistics companies that have the experienced personnel and appropriate equipment to handle the rough roadway may be considered to provide trucking services for the project. The main contractors that provide trucking to Stony Rapids have been contacted in the prefeasibility phase for price quotes on trucking to Stony Rapids but this has not been put out to tender and no contract has been signed.

Currently there is only one barging service operating on Lake Athabasca that is operated out of Stony Rapids. The barge is approximately 30 ft x 120 ft with a capacity of 200 tonnes. No contract has been signed for barging services.

19.2.9 PROJECT RESIDENCE AND MINE OFFICE AND DRY FACILITY

During mine site construction and mine operation, construction crews and mine personnel will be housed in a project residence in Uranium City provided by a camp services company. The service company will also provide all meals and house-keeping to the camp facility.

Mine offices, dry facilities, and house-keeping services for the mine site buildings located at Goldfields will be provided and maintained by the same services company.

Several service companies have been contacted for budget quotes but this has not been put out to tender and no contract has been signed for this service.

19.2.10 BULK EXPLOSIVES FACILITY AND PIT BLASTING

During open pit mining, bulk explosives supply will be contracted out to a qualified and experienced contractor. The chosen contractor will provide all equipment, facilities for mixing and delivering the explosives to the blast holes. Loading and initiating of the blasts will be done by the mining company.

Companies providing these services have been in contact for budget quotes during the prefeasibility phase but this has not been put out to tender and no contract has been signed for this service.

19.2.11 POWER

During the operation of the Box in the 1930s and 40s, electrical power was provided to Goldfields by SaskPower. During the mine operation, electrical power will be provided by SaskPower on the existing 115 kV line.

Currently, SaskPower is preparing a quote for re-establishing this power line, but no contract has been signed with SaskPower.

19.2.12 DIESEL FUEL PROVIDER

Brigus will require bulk fuel on site to construct, operate and decommission the mine. A bulk fuel supplier will be contracted to deliver bulk fuel shipments to site on a regular basis. No contract has been signed for this service.

19.2.13 SITE SECURITY

Site security will be provided by a security company, which will have a security officer at site 24 hours a day, 7 days a week.

This has not been put out to tender and no contract has been signed for site security.

19.2.14 PHYSICIAN CONSULTANT AND NURSE

Due to the remote location of Uranium City and the low population, currently there are no permanent medical services. Brigus will either provide medical services on site on a contract basis or will work with the Athabasca Health Authority to develop a plan to provide appropriate medical services in the community. A budget quote was obtained from a medical service provider during the prefeasibility phase. There are a number of organizations that provide contract nursing services to northern mine sites. No contract or agreement has been negotiated for health services for the project.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 SUMMARY

An Environmental Base Line Study, a Rare Plant Study and a hydrogeological study were completed for the Goldfields.

The climate of the study area is characterized by short, cool summers and extremely cold winters. The average temperatures are typically below freezing from November through April. Total mean annual precipitation over the latest 30 year period at Uranium City was approximately 360 mm. Wind measurements taken at the Uranium City meteorological station show that prevailing winds were from the east, with an average wind speed of approximately 11 km/h.

Water quality data were collected from each water body in the Project Study Area (PSA). The pH levels in the PSA were near neutral or slightly alkaline, with all values meeting the Canadian Environmental Quality Guidelines(CEQG). Water chemistry data showed that the majority of the study water bodies were characterized as oligotrophic (low nutrient concentrations), consistent with other lakes found throughout northern Saskatchewan. Concentrations of aluminum, cadmium, copper, iron, and uranium exceeded the SSWQO and CEQG in some water bodies within the PSA. Deadman Lake, historically an unimpacted lake, showed concentrations of aluminum, cadmium and iron above these guidelines indicating a naturally high occurrence of these elements in the environment.

The concentration of metals in the sediments in the Small Lake's study area were generally low, with the mean sediment concentrations of copper, lead, zinc, vanadium, and arsenic exceeding either the Interim Sediment Quality Guidelines the Lowest Effects Level or the Probable Effects Level guidelines.

Sedges were collected from three lakes and four bays on Lake Athabasca in the Goldfields PSA for chemical analyses of whole plants. The mean concentrations of aluminum, cadmium, copper, nickel, selenium, and titanium, were notably higher in Small Lake compared to both Frontier Lake and Deadman Lake. The concentrations of analytes in the sampled bays of Lake Athabasca were highly variable.

A fish community survey was completed in August 2010 for Deadman Lake and Goldfields Bay of Lake Athabasca. Flesh samples were collected from water bodies in the PSA for chemical analyses. Mean metal, trace element, and radionuclide concentrations/activities in fish flesh were generally near or below analytical test detection limits.

A fish habitat survey was also conducted in August 2010 for Deadman Lake and Goldfields Bay of Lake Athabasca. There were a variety of habitat types identified in both water bodies that potentially provide spawning, rearing, feeding, and overwintering habitat for fish. In general, many areas were identified as marginal or moderate spawning habitat and four habitat sections were each rated as highly suitable spawning habitat for northern pike, walleye, lake whitefish, and lake trout.

Rare plant surveys were conducted in July and September, 2010. Two hundred and ninety-nine plant species, including 31 rare species, were observed. None of the 31 rare species observed in the study area are protected under the Species at Risk Act (SARA) or the Wildlife Act.

Incidental wildlife recorded during the 2010 rare plant surveys included two federally protected species. A common nighthawk nest and numerous northern leopard frogs were observed. The common nighthawk is listed as threatened under SARA Schedule 1 and the Committee on the Status of Endangered Wildlife in Canada .The western boreal/prairie population of northern leopard frogs are listed as a species of special concern under SARA Schedule 1 and COSEWIC.

Brigus is committed to a strategic impact reduction and mitigation approach that extends through all project phases, from design to decommissioning. All aspects of Brigus' strategic approach conform to (meet or exceed) the requirements of the approved EIS and the associated Ministerial Approval. It is anticipated that most impacts to rare plants, native vegetation, and rare wildlife species will be temporary.

Application of Brigus's company strategy for impact reduction and mitigation is anticipated to reduce the extent and duration of impacts to rare plants, native vegetation, and wildlife species at risk that occur in the Goldfields PSA. The impact on surface waters is also expected to be temporary. There are currently no environmental issues known to the proponent that could materially impact the ability to extract the mineral resources or mineral reserves.

20.2 SITE WASTE MANAGEMENT

The WRSA will be used to store all waste rock that is not needed for site construction. Gravity rejects from the mill process will also be stored in the WRSA together with the waste rock. The WRSA layout uses the natural terrain supplemented by four relatively small engineered containment structures to provide natural containment of the waste rock and gravity rejects. The natural containment also facilitates an effective water management system. All waste rock will be stored in natural valleys that drain towards Frontier Lake and Neiman Bay. Each valley makes up one cell of the WRSA.

Each cell includes an impermeable control structure at the natural low point of the basin. This structure will contain the water within the WRSA until it can be transferred to the engineered water management reservoirs. Water will be transferred by a portable diesel powered pump. All pumps and pipes will be contained within the outline of the WRSA to eliminate the possibility of an accidental discharge to the environment.

The water management system will consist of a settling reservoir and a monitoring reservoir located within the WRSA. The settling reservoir will receive all water from the WRSA. Water from the settling reservoir will decant into the monitoring reservoir. Vertical baffles within the reservoirs will control the settling of suspended solids. Based on operational data the water management system will be operated as a continuous discharge facility with upstream predictive monitoring and a downstream physical point of control before Frontier Lake.

Water management in the WRSA will provide effective control over the release of water to the environment. The drainage basin that reports to Neiman Bay, Lake Athabasca will be controlled with an impermeable structure that will contain all surface runoff from the WRSA. This water will be transferred to the water management system for monitoring and release to Frontier Lake. All discharge to Frontier Lake will meet MMER requirements.

Vic Lake was historically used as a tailings disposal facility and is considered very poor aquatic or wild life habitat. It is proposed to use the existing tailings disposal area as the TMF for the currently proposed operation. Tailings from the mill process will be deposited sub-aqueously by a pipeline with a moveable discharge point into the deepest areas of the lake. During the winter months, tailings will be deposited through the ice to the bottom of the lake. A 3 m water cover will be maintained over the tailings to prevent potential acid generation.

At the south end of Vic Lake, an engineered control structure will be constructed that will control out flow from the lake into Box Bay, Lake Athabasca. This dyke will be constructed using waste rock as the structure, lined with an impermeable liner system and will have a top elevation of 221 m, giving a 1 m free-board above the maximum lake level.

At the north end of the lake, overflow into Frontier Lake will be controlled by an outlet structure that will allow water above the 220 m elevation to flow through. Prior to mill operation, Vic Lake will be pumped down to an elevation of 215 m. This will still allow minimum tailings coverage of 3 m of water and will allow adequate retention time to ensure natural destruction of cyanide. The lake's elevation will not reach 220 m until approximately Year 3 of milling. During this time, water quality will be monitored, allowing Brigus to determine treatment alternatives as previously mentioned in Section 18. In the event cyanide concentrations exceed MMER requirements, cyanide destruction contingency measures including accelerated evaporation, recycling to mill for further treatment, or the installation of a remote cyanide destruction plant at the discharge to Frontier Lake will be considered.

Water released to the environment from process activities, impoundment areas, and waste management facilities will be subject to a comprehensive monitoring and control program in accordance with MMER and Saskatchewan Ministry of Environment (MOE) requirements. Currently there are active piezometers around potential outflows of Vic Lake and also at the base of the valleys at the proposed WRSA, where water flow and samples can be measured and taken for testing during construction and mine operation.

The proponent is also committed to complying with other environmental monitoring required by regulation, project permitting, or government approvals.

An offsite sanitary landfill will be established in accordance with regulatory requirements and expectations.

20.3 PERMITTING

The permitting and licensing requirements for the Goldfield Project are expected to be similar to those for other non-uranium mines operating in Northern Saskatchewan. A staged approach commensurate with the level of progress on the project will be adopted for licensing and permitting applications.

Permits that are required for construction of the mine, operation of the mine and establishing suitable access from Uranium City will be completed as required.

20.4 ENVIRONMENTAL, PERMITTING AND SOCIAL FACTORS

The environmental assessment studies have been completed and the EIS for the Project received Ministerial Approval on May 28, 2007. The project will house its workforce in Uranium City, SK located approximately 25 km from the site and plans to erect a project residence with the associated support facilities within the town site. The project has the potential to provide much needed employment and economic stimuli for local contractors and businesses in the area.

A Surface Lease Agreement between Brigus and the Province of Saskatchewan is currently being negotiated. The agreement addresses issues including land tenure, environmental protection measures, occupational health and safety provisions, and socioeconomic benefits for neighboring communities and Residents of Saskatchewan's North.

Prior to starting construction, permits will be required from Saskatchewan MOE.

20.5 MINE CLOSURE

Mine Closure and decommissioning will include contouring general site areas and abandoned areas to blend with background landscape, minimize soil erosion, and encourage vegetation re-growth. Re-vegetation activities will be carried out using techniques conventional and proven effective for northern sites. All water bodies, natural drainage areas and shorelines that may have been disturbed will be rehabilitated to the pre-disturbed condition whenever possible.

All old mining facilities will be dismantled and removed. The debris will be cleaned up and the vent raises from the old mine will be plugged. The old tailings at the Vic Lake northern shore will be decommissioned with the newly placed tailings from the proposed project.

The road surfaces will be broken and contoured to match those of the surrounding landscape. Former road bed within the surface lease will then be re-vegetated.

At project closure, parts of the WRSA will be reclaimed and re-vegetated. The expected goal for the waste rock deposit is not 100% top soil and vegetative cover; rather, a preferred approach will be to re-contour the deposit surface in a manner that creates microenvironments which – following ecological succession – will provide a greater diversity of plant and animal habitats. Currently, large surface outcroppings of bedrock on site do not support the accumulation of top soil or vegetation, other than lichen.

The pits will be secured to prevent any unauthorized entry and warning signs will be posted. To prevent unauthorized access via the haul ramp, the roadway will be trenched immediately prior to descending into the pit. Upon completion of the mining, the pits will be left open to allow natural flooding. In the long term it will create a body of water with an areas up to 30 ha. In areas of steep wall, where it would not be possible to climb out of the water, the pit rim will be contoured to minimize risks of entrapment and drowning. The pits will be monitored with regards to Acid Rock Drainage and nitrate concentration in the water.

Pipelines will be dismantled, decontaminated, and buried in the WRSA. All on site power lines, electrical substation, and diesel generators will be removed from site. All waterworks, freshwater intakes, culverts, docks and any other installations in water bodies will be removed, unless otherwise approved by an appropriate regulatory agency.

At the cessation of milling operations, monitoring of TMF water quality, water balance, and water height will continue annually. By the fifth year after cessation of active tailings discharge, the water in the TMF is likely to meet water quality objectives for discharge into Frontier Lake. Water quality will continue to be monitored to evaluate the effectiveness of natural degradation until the water quality objectives are being met.

The causeway crossing Lodge Bay will be removed, effectively limiting unauthorized access during summer months, and provide for long term unobstructed fish passage from inner Lodge Bay to Lake Athabasca. Signage will be posted surrounding the mine site to prevent access from the lake side, or by snowmobile traffic in winter.

The Goldfields site will be returned to the institutional control of the Province of Saskatchewan after all required decommissioning, reclamation, and monitoring has been completed and approval for abandonment has been received from the appropriate regulatory agencies.

An irrevocable letter of credit will be registered with the Saskatchewan MOE to cover the estimated cost of closure, decommissioning and abandonment.

20.6 COMMUNITY RELATIONS

The proponent will continue facilitating, hosting, and documenting consultations in neighboring communities that are close to the Goldfields PSA. The project has met the Saskatchewan government’s interim guidelines for “Duty to Consult” as part of the EIS approval process. Additional community consultations will be held as the project progresses.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

21.1.1 INTRODUCTION

March Consulting prepared a pre-feasibility level capital cost estimate for the Goldfields Project. The estimate includes costs associated with the construction of the Box mine and mill infrastructure. It was assumed that when Athona is ready for operation it would use the site infrastructure and equipment developed for Box.

21.1.2 BASIS OF ESTIMATE

The main components of the Basis of Estimate (BOE) are listed in Table 21-1.

Table 21-1: Main Components of the BOE

Item	BOE
Geographical location	Actual
Maps and surveys	Maps available, survey at 1 m contours
Geotechnical testwork	Not available
Process definitions	
Process selection	Provided by Brigus
Design criteria	Pre-feasibility stage
Flowsheets/plant capacity	Provided by Brigus
Bench-scale tests	Various
Pilot plant test	Various
Mass balance	Provided by EHA Engineering
Mill equipment list	Provided by EHA Engineering
Process facilities design	General arrangement drawings by March Consulting
Infrastructure definition	Provided by March Consulting
Capital cost estimating methodology	Provided by March Consulting
Direct Costs	
Productivity	Productivity factor of 1.2
Site work	Quantities determined from site layouts
	Fill material created from local blasting
Clearing and grubbing	Areas determined from site layout

Item	BOE
Site grading	Load, haul, place, compact to specifications
	Minimize blasting requirements by following natural terrain
Demolition of Government Building	Based on quotations
Dykes and berms	Load, haul, place, compact to specifications
	WRSA berms #1 & 2 included in cost estimate
	Vic Lake will be used as TMF
Roads	Load, haul, place, compact to specifications
	Minimize blasting requirements by following natural terrain
Geo-synthetics	Quantities based on site layout
	Costs based on supplier quotes
Site power	Refurbishment of existing SaskPower 115 kV transmission line
	On-site 15 kV substation
	600 V emergency diesel generator
Site pipelines	Lengths determined from site layout
	Surface run, no utilidors, insulated and heat traced as required
Barge loading facility	Allowance for floating dock and shore improvements
Mine and site equipment	Based on budgetary quotations from suppliers
Foundations	Concrete volumes based on building footprints
Project residence	Modular construction
	Costs based on quotes from suppliers for supply and install
Mine office and dry facility	Modular construction
	Costs based on quotes from suppliers for supply and install
Building packages	Preliminary sizing from CAD model
	Costs are based on supplier quotes factored for building size
Equipment support and interior structure	Cost based on estimated weight of steel from models
Building Services	Costs are factored based on building volume
Architectural	4% of total building cost including: foundation, building package, interior structural steel, and building services
HVAC	Costs based on supplier quotes and historical estimates
Mechanical equipment	Major equipment costs based on supplier quotes
	Pumps, tanks, agitators, and conveyors are estimated based on historical information
Chutes, launders and ducts	Costs are based on a 4% factor of the equipment costs
Installed mechanical equipment	Mechanical equipment plus chutes, launders, and ducts
Piping	17% of installed mechanical equipment

Item	BOE
Electrical	17% of installed mechanical equipment
Instrumentation & controls	9% of installed mechanical equipment
Total installed mechanical equipment	Sum of installed mechanical equipment, piping, electrical, and instrumentation and controls
Indirect Costs	
EPCM	11% of total direct costs excluding mine and site equipment
Construction temporary facilities	1% of total direct costs
Fuel & other consumables	2% of total direct costs
Flying, room & board	9% of total direct costs
Freight allowance	3.7% of total direct costs plus 20% contingency to cover unaccounted loads
Contingency	10% for supplier quotes, 15% for independently reviewed, 20% for estimates and allowances
Owner's costs	
Miscellaneous owner costs	0.6% of total directs including risk insurance, fees, permits, legal expenses, public relations, and government liaison
Staff labour and expenses	Allowance of 3 months for salaried employees
First fills	Allowance of 30 days of consumables
Site equipment spares	1.3% of site equipment direct costs
	Mine equipment spares are on consignment from suppliers
Mill mechanical equipment spares	2.7% of total installed mechanical costs
Commissioning & 3rd party reviews	1.2% of total installed mechanical costs
Currency exchange rates	$ CAD = 0.96 USD
	$ CAD = 1.07 AUD
	$ CAD = 0.70 EURO
Project costs	Based on 2010 – 4th quarter $ CAD
Estimate accuracy	AACE Class 4, +30%/-20%

21.1.3 MINE

21.1.3.1 PIT PIONEERING

The ore body at Box is exposed at surface. Minimal preproduction and pit pioneering is scheduled to occur in the six months prior to production. HG ore mined during this period will be stockpiled adjacent to the crusher, LG ore will be used for stope backfill and waste rock will be used for haul road and WRSA berm construction.

Costs associated with this phase have been included under the civil costs in the site infrastructure section.

21.1.3.2 MOBILE EQUIPMENT

The scope of the mobile equipment capital cost estimate includes all equipment necessary for the mining operations including drilling, excavating, hauling, grading, employee transportation, mine services, mill services, site services and emergency response.

MINE MOBILE EQUIPMENT

The mine mobile equipment includes the equipment required to perform the drilling, blasting, loading, hauling, haul road maintenance, and stockpiling of ore and waste from the pit. The mine mobile equipment includes: hydraulic shovels, haul trucks, wheel loader, wheel dozer, track dozers, blasthole drills, and motor graders. For more detailed information about the mining equipment requirements refer to Section 16.5.

SITE MOBILE EQUIPMENT

The site mobile equipment includes the equipment required to provide mobile service requirements for the mining equipment, site maintenance, environmental monitoring, crew transport, emergency response, and mill service. The mobile service equipment includes the fuel truck, fuel and lube truck, mechanics service truck, tire handler, and heavy duty low boy trailer and off highway tractor. The site maintenance equipment includes a motor grader, snow plow/sanding truck, water truck, and wheel loader. The equipment required for environmental monitoring includes boats and all terrain vehicles. Crew transport from Uranium City to the mine site will utilize buses. Emergency response will be handled by an on-site ambulance and fire truck. The mobile equipment required for the mill includes a wheel loader, telehandler, warehouse forklift, container handler, and emergency generator. For more detailed information about the site mobile equipment requirements refer to Section 16.5.

COST ESTIMATE

The mobile equipment cost estimate was completed based on supplier quotes. The total capital cost for all equipment was determined as presented in Table 21-2. Lease options for major equipment were investigated to determine the impact on capital and operating costs. Since lease payments would commence during the construction period and extend into operations, an average of six months of payments for all major equipment and 12 months for the project residence was included in the capital cost estimate. The remaining lease costs were included in the operating costs. Quoted lease terms for mining equipment ranged from three to four years with zero down payment. The lease term for the project residence was quoted at eight years.

Total capital cost for the mobile equipment is estimated at $46,498,000 compared to a cost for leasing the mining equipment of $11,741,000. The difference between the purchase cost and lease cost of $34,757,000 was applied to the operating costs including interest associated with the lease.

Table 21-2: Mobile Equipment Cost

Description	Model/Size	Qty	Unit Cost $000s	Purchase Cost ($000s) (Including 7% SSA[a])	Lease Option ($000s) (Including 7% SSA[a])
Mine Equipment					
Hydraulic Shovel	HITACHI EX1900	2	$3,353	$7,174	$983[b]
Off Highway Haul Trucks	CAT 777F	9	$1,799	$17,326	$2,430[b]
Wheel Loader	CAT 992K	2	$2,142	$4,584	$643[b]
Motor Grader	CAT 16M	1	$785	$840	$118[b]
Wheel Dozer	CAT 844H	1	$1,773	$1,898	$225[b]
Track Dozer	CAT D10T	2	$1,459	$3,122	$438[b]
Blasthole Drill DTH Crawler	Sandvik QXR 1320 DTH	2	$1,234	$2,640	$819[b]
Single Pass Blasthole Drill	Sandvik D55SP	1	$1,594	$1,706	$529[b]
Site Equipment					
Motor Grader	CAT 14M	1	$550	$588	$83[b]
Tire Manipulator	CWS	1	$706	$755	$106[b]
Telehandler	CAT TL1055	1	$139	$148	$21[b]
Integrated Tool Carrier	CAT IT62H	1	$346	$370	$52[b]
Forklift (warehouse)	CAT EP6000	1	$58	$62	$9[b]
Fuel & Lube Truck		1	$685	$733	$733
Fuel Truck		1	$250	$268	$268
Mechanic's Service Truck		1	$309	$331	$331
Water Truck	5,000 gal	1	$250	$268	$268
Rough Terrain Crane	65 - 75 t	1	$465	$497	$497
Fire Truck		1	$303	$324	$324
Container Handle	Hyster Yardmaster II	1	$686	$734	$734
Shipping Containers	20' Standard	45	$4	$182	$182
Vacuum Truck		1	$250	$268	$268
Sanding/Snow Plow Truck		1	$256	$274	$274
Ambulance		1	$99	$106	$106
Bus - Employee Transport	55 passenger	2	$110	$235	$235
Pickup Trucks	4WD, ¾ ton, crew	8	$47	$405	$405
Quad runners	500 cc, 4 x 4	2	$8	$18	$18
Boats	16 ft aluminum	2	$7	$14	$14
Light Plant		8	$12	$100	$100
Western Star Tractor	4900 SA	1	$195	$209	$209
Lowboy Trailer	Aspen 90-100 ton	1	$215	$230	$230
HDPE Pipe Fusing Machine	1" - 4"	1	$5	$5	$5
HDPE Pipe Fusing Machine	4" - 12"	1	$40	$43	$43
Fusion Machine Generator	7 kW	1	$38	$41	$41
Mobile Equipment Capital Costs				$46,498	$11,741

Note:
a: Selected Statistical Adjustmentis applied to account for changes in item definition or scope between time of estimation and next level of estimate or procurement of item.
b: Lease quotes provided by suppliers for inclusion in the cost estimate.

21.1.4 MILL PROCESS PLANT

The capital cost estimate for the mill is divided into sections correlating to the specific areas of the mill processes. The costs for each process area of the mill are detailed in Table 21-3.

Table 21-3: Mill Process Plant Capital Cost by Area

Area	**Extended Total Capital ($000s)**
Crushing	$3,674
Grinding	$18,142
Flotation	$7,823
Rejects Thickening	$1,902
Concentrate Thickening	$1,319
Leach Plant	$1,522
Leach Filtration	$2,625
Clarification	$252
Merrill Crowe Circuit	$870
Gold Refining	$621
Water Distribution	$4,026
Barren Distribution	$377
Reagents	$954
Services/Utilities	$731
Total Mill Facility Direct Cost	$**44,838**

21.1.5 SITE INFRASTRUCTURE

The site infrastructure costs were calculated from quotations from contractors and suppliers for labour, materials, and equipment. Table 21-4 shows the costs associated with site infrastructure.

Table 21-4: Site Infrastructure Capital Cost Summary

Area	Description	Extended Total Capital ($000s)
Site Work	Clearing and Grubbing	$1,849
	Site Grading	$1,764
	Dykes/Berms	$3,775
	Roads	$4,611
	Geosynthetics	$395
Site Power	Sub Station and Transformer Pads	$6,610
Site Pipelines	Raw Water	$194
	Tailings	$75
	Gravity Rejects	$570
	Diesel Fuel	$67
	WRSA Water Management	$347
Bulk Fuel Storage	Tanks and Equipment	$2,887
Barge Loading Facility	Floating Dock and Loading Facility Shore Improvements	$396
Bulk Explosives Plant	Building Package (Contractor Supplied)	$0
	Foundation	$119
Camp, Office and Dry Facilities	Camp, Office and Dry Facilities	$6,957
Explosive Cap and Magazines	Explosive Cap and Magazines	$333
Mine Service Complex	Foundation	$2,000
	Prefabricated Steel Building Package	$1,514
	Equipment Support and Interior Structure	$903
	Architectural	$191
	HVAC	$463
	Building Services (Electrical and Fire Protection)	$354
Mill Building	Foundations	$4,800
	Building Packages	$2,374
	Equipment Support and Interior Structure	$3,054
	Architectural	$438
	HVAC	$646
	Building Services	$720
Rejects Thickener Area	Foundations	$74
Crushing Area	Foundations	$854
	Total Site Infrastructure Cost	**$49,334**

21.1.6 INDIRECT AND OWNERS COSTS

Indirect costs were calculated as a percentage of the total directs as indicated in Table 21-5.

Table 21-5: Indirect and Owners Cost

Description		Factored Percentage	Total Capital ($000s)
Construction Indirects	EPCM (total directs less mining and site equipment)	11%	$10,355
	Temporary Construction Facilities	1%	$1,419
	Fuel and Other Consumables	2%	$2,838
	Fights, Room and Board	9%	$12,767
Freight	Freight Allowance	3.7%	$5,249
Owners Cost	Miscellaneous Owner's Costs (includes indented below in factor)	0.6%	$852
	All risk insurance		
	Fees, permits and legal expense		
	Public relations		
	Liaison with government agencies and regulatory bodies		
	Staff labor and expense	Allowance	$1,200
	First Fills (30 days of consumables)	Allowance	$1,500
	Site Equipment Spares (excludes spares on consignment from supplier)	1.3%	$108
	Mill Mechanical Equipment Spares	2.7%	$931
	Commissioning and 3rd Party Reviews (Installed Mech. Only)	1.2%	$549
	Total Indirect Cost	**27%**	**$37,768**

21.1.7 FIRST FILLS CRITERIA

The first fills were calculated as the consumables and fuel required for the first thirty days of operation. The first fill consumables are included in Table 21-6.

Table 21-6: First Fills Cost Summary

Item	Cost ($/t milled)	Cost ($ for 30 days)
Reagents and Consumables	$3.80	$570,000
Operating Supplies	$0.31	$47,000
Mill Maintenance	$0.51	$78,000
Office Materials and Supplies	$0.05	$8,000
Fuel	$5.04	$757,000
Ground Engaging Tools	$0.23	$34,000
Totals	**$9.94**	**$1,494,000**

Notes:

- The fuel requirements for HVAC have not been included. It is assumed that the first 30 days of operation will occur during the summer months.
- The mobile equipment tires and tracks costs were not included. Supplier quotes included the costs associated with these items.
- The maintenance supplies for the mobile equipment were not included.
- The mobile equipment spares would be on consignment from suppliers and is not included.

21.1.8 CONTINGENCY

Total project contingency was determined based on the confidence level of the costs associated with all parts of the capital cost estimate. Table 21-7 summarizes the contingency for each area. Total project contingency of 14% is a weighted average of all individual contingencies. Each project area was assigned a contingency level that was representative of the confidence level of the quotes and estimates. The following contingency guidelines were used for assigning contingency. The contingencies do not include allowances for scope changes, extraordinary events such as strikes or natural disasters, or escalation or exchange rate fluctuations.

- 10 % - direct costs are based on quotes for equipment with stable price history
- 15 % - direct costs are based on estimates and allowances and have undergone an independent review
- 20 % - direct costs are based on estimates and allowances

Table 21-7: Overall Project Contingency Breakdown

Description	Factored Percentage of Total Directs	Total Capital ($000s)
Contingency		
Site Work	20%	$2,479
Site Power	15%	$992
Site Pipelines	20%	$251
Bulk Fuel	15%	$433
Barge Loading Facility	20%	$80
Mining Equipment	10%	$619
Site Equipment	10%	$678
Buildings		
Foundations	15%	$1,177
Building Packages	10%	$638
Equipment Support and Interior Structure	20%	$792
Architectural	20%	$126
Building Services	20%	$215
HVAC	15%	$167
Total Installed Mechanical	15%	$4,745
Total Installed Piping	15%	$744
Total Installed Electrical	15%	$807
Total Installed Instrumentation	15%	$428
Freight	20%	$1,050
Indirects	10%	$2,738
Overall Contingency	**14%**	**$19,159**

21.1.9 SUMMARY

Table 21-8 summarizes the total capital cost for the project. The capital cost was estimated at $159,235,000. This cost includes leasing a portion of the mobile equipment plus the project residence in Uranium City. The remainder of the lease payments is accounted for during the operations period.

Table 21-8: Capital Cost Summary

	Description	Total Capital ($000s)
Directs	Infrastructure	$44,535
	Mine	$12,956
	Mill	$44,838
	Subtotal	$102,329
Indirects	Construction Indirects	$27,379
	Freight Indirects	$5,249
	Owners Costs	$5,119
	Subtotal	$37,747
Contingency	Contingency	$19,159
	Subtotal	$19,159
	Total Capital Cost	**$159,235**

21.2 OPERATING COST ESTIMATE

21.2.1 INTRODUCTION

The operating costs were determined based on the project details developed for Box. For the economic analysis operating costs for Athona and the recovery of ore stockpiles were derived from the Box costs using factors based on haul distance cycle times and mineable ore and waste volumes. A summary of Athona and ore stockpile recovery operating costs is included in Table 21-26.

21.2.2 BASIS OF ESTIMATE

21.2.2.1 GENERAL

The operating costs were separated into the following categories: mining, milling, and general & administrative (G&A) costs. March Consulting utilized data from Brigus' Black Fox operation, equipment suppliers, and historical resources to determine the operating costs for Box. The consumption rates for process consumables and materials were estimated based on previous test work.

21.2.2.2 SHIFT SCHEDULE

The mine and process plant's operating schedule was as follows:

- Two shifts per day
- Twelve hours per shift
- Two weeks in/two weeks out (fly-in)

The two week in/two week out schedule is preferred for the fly in operations as it provides employees with adequate time off between shifts and provides the optimum flight cost structure.

21.2.3 MINING

21.2.3.1 BASIS OF ESTIMATE

The mine operating cost estimate incorporated the cost of operations based on the labor, materials, consumables and equipment required.

The mining operating cost estimate took into consideration the following:

- Equipment utilization
- Mechanical availability
- Maintenance parts and supplies (i.e. fuel, tires and tracks, ground engaging tools, explosives, etc.)
- Labor – mining, maintenance, and engineering

The operating costs were accounted for on an equipment and manpower basis, not on a task basis. The life of the mine average was used to determine the quantity of vehicles required for operations.

21.2.3.2 LABOR COSTS

The mine labor cost including a 30% burdern were estimated as shown in Table 21-9.

Table 21-9: Box Labor Costs

Area	Number of Employees	Annual Cost ($000s)	Operating Cost ($/t mined)
Mine Open Pit	68	$5,376	$0.53
Open Pit Mine Maintenance	26	$2,435	$0.24
Open Pit Engineering	6	$710	$0.07
Geology and Grade Control	10	$1,015	$0.10
Total Mine Labour Costs	**110**	**$9,536**	**$0.94**

21.2.3.3 EQUIPMENT OPERATING COSTS

Table 21-10 summarizes the estimated mine equipment operating costs.

Table 21-10: Estimated Mine Equipment Operating Costs

Mine and Auxiliary Equipment	Qty	Annual Cost ($000s)	Fleet Maint Op Cost ($/t mined)
Hitachi EX1900-6 - Hydraulic Shovel	2	$1,958	$0.193
Cat 777F - Haul Truck	10	$7,407	$0.730
Cat 992K - Wheel Loader	1	$345	$0.034
Cat 16M - Motor Grader	1	$233	$0.023
Cat 844H - Wheel Dozer	1	$588	$0.058
Cat D10T - Track Dozer	2	$700	$0.069
Sandvik QXR 1320 - DTH Blast hole Drill	2	$1,116	$0.110
Sandvik D55SP - Single Pass DTH Blast hole Drill	1	$640	$0.063
Blasting	LOT	$3,044	$0.300
Fuel & Lube Truck - Western Star 4900 SA	1	$122	$0.012
Water Truck - Western Star 4900 SA	1	$20	$0.002
Tractor - Western Star 4900 SA	1	$20	$0.002
F750 - Mechanics Service Truck	1	$101	$0.010
Tire Manipulator	1	$20	$0.002
RT Crane 65T	1	$91	$0.009
Lowboy (90 – 100 t)	1	$10	$0.001
Container Handler	1	$71	$0.007
Light Plants	8	$355	$0.035
Total Operating Mine Equipment Cost		**$16,841**	**$1.66**

21.2.4 MILLING

21.2.4.1 BASIS OF ESTIMATE

The milling operating costs were based on a daily mill feedrate of 5,000 tonnes per day and an annual mill feed of 1,825,000 tonnes per year.

21.2.4.2 LABOR COSTS

The mill labor costs including 30% burden were estimated as shown in Table 21-11.

Table 21-11: Mill Labor Costs

Area	Number of Employees	Annual Cost ($000s)	Operating Cost ($/t milled)
Mill Operations	33	$2,829	$1.55
Mill Maintenance	18	$1,588	$0.87
Total Mill Labour Costs	**51**	**$4,417**	**$2.42**

21.2.4.3 MOBILE EQUIPMENT COSTS

The estimated operating costs for the mill mobile equipment are shown in Table 21-12.

Table 21-12: Mill Mobile Equipment Operating Costs Summary

Mill Mobile Equipment	Qty	Annual Cost ($000s)	Fleet Maint Op Cost ($/t milled)
Wheel Loader - CAT 992K	1	$1,715	$0.94
Telehandler - CAT TL1055	1	$73	$0.04
Stand-by Generator -1500 KV	1	$237	$0.13
Total Mill Mobile Equipment Operating Cost		**$2,026**	**$1.11**

21.2.4.4 CONSUMABLE COSTS

The consumable costs included reagents, grinding media, crusher liners, grinding mill liners, etc. The reagent unit consumptions were estimated based on previous test work. Unit costs were provided by budgetary quotations. Tables 21-13 and 21-14 provide the summary of the reagents, consumables and supplies for operation. The total reagents cost including freight was $3.80/tonne milled and the total mill supplies cost including freight was $0.36/tonne milled.

Table 21-13: Reagents and Consumables Operating Costs

Item	Consumption (kg/t milled)	Annual Cost ($000s)	Delivered Cost ($/t milled)
PAX	0.060	$341	$0.187
R208	0.020	$165	$0.090
MIBC	0.030	$217	$0.119
Cyanide	0.045	$227	$0.124
Lime to CN	0.0096	$12	$0.006
Lime to CN destruction	0.120	$139	$0.076
Flocculant	0.015	$153	$0.084
CuSO4	0.003	$21	$0.011
Metabisulphite	0.160	$413	$0.226
Descalant	0.001	$6	$0.003
Precoat	0.250	$948	$0.519
3” to 5" balls	0.386	$1,463	$0.802
1.5" balls	0.691	$2,733	$1.497
Zinc	0.016	$33	$0.018
Pb nitrate	0.002	$3	$0.002
Flux	0.016	$61	$0.034
Reagents and Consumables Cost		**$6,935**	**$3.80**

Table 21-14: Mill Operating Supplies Costs

Item	Consumption	Annual Cost ($000s)	Delivered Cost ($/t milled)
SAG Mill Liners	one liner replacement/yr	$319	$0.175
Ball Mill Liners	one liner replacement/yr	$100	$0.055
Crusher liners	one liner replacement/yr	$55	$0.030
Miscellaneous		$183	$0.100
Operating Supplies Cost		**$657**	**$0.36**

21.2.4.5 PROCESS EQUIPMENT MAINTENANCE COSTS

Process equipment maintenance costs were factored based on 3% of the total installed mechanical equipments costs as shown in Table 21-15.

Table 21-15: Estimated Process Equipment Maintenance Costs

Area	Annual Cost ($000s)	Maintenance Cost ($/t milled)
HVAC - Mill	$18	$0.010
HVAC - Maintenance Shop	$11	$0.006
Crushing	$84	$0.046
Grinding	$349	$0.191
Leach & Flotation	$164	$0.090
Rejects Thickener	$40	$0.022
Concentrate Thickener	$28	$0.015
Leach Plant	$33	$0.018
Leach Filters	$55	$0.030
Clarification	$5	$0.003
Merrill Crowe	$18	$0.010
Gold Refining	$15	$0.008
Water Distribution	$86	$0.047
Leach Barren Distribution	$7	$0.004
Reagents	$20	$0.011
Reagents Utilities	$16	$0.009
Total Process Equipment Maintenance Cost	**$949**	**$0.52**

21.2.4.6 POWER COSTS

Power would be supplied by SaskPower through the provincial grid. The power requirements and estimated costs are summarized in Table 21-16.

Table 21-16: Power Operating Costs

Item	Units	Value
Total Installed Power	kW	8,920
Average Running Load	kW	6,240
Energy Consumption	kWh/tonne milled	29.96
Energy Charge per tonne milled	$ /tonne milled	$1.39
Basic Monthly Charge	$	$6,440
Total Monthly Demand Charge	$	$38,240
Annual Estimated Electricity Cost		$3,079,000
	Total Estimated Electricity Cost	**$1.69/t milled**

21.2.4.7 HVAC

The mill building was designed to have 41 m^3/h ventilation rate while maintaining the temperature at 15°C. An air-to-air heat exchanger having an operating efficiency of 50% will be installed in the air make-up units to recover heat from the exhaust stream.

The mill facility and mine service complex will be heated with a hot water fuel oil fired boiler at 80% boiler efficiency. The mine office and dry facility will utilize forced air fuel oil furnaces. Make-up heat is not required for process water. Table 21-17 summarizes the site heating costs.

Table 21-17: Summary of the Mine HVAC Operating Costs

Facility	Annual Cost ($000s)	Operating Cost $/tonne milled
Mill Building	$967	$0.53
Mine Service Complex	$456	$0.25
Office and Dry Facility	$37	$0.02
Total HVAC Operating Cost	**$1,460**	**$0.80**

21.2.5 GENERAL & ADMINISTRATIVE

21.2.5.1 BASIS OF ESTIMATE

The general and administrative (G&A) costs included the following:

- G&A labor costs
- G&A equipment costs
- Office and administration costs
- Flight costs
- Camp and catering costs
- Refining costs

21.2.5.2 LABOR COSTS

Table 21-18 summarizes the G&A labor costs.

Table 21-18: G&A Labor Costs

Area	Number of Employees	Annual Cost ($000s)	Operating Cost ($/t milled)
Site General and Administrative Staff	12	$1,241	$0.68
Site Services	6	$438	$0.24
Contractor Services	48	$547	$0.30[a,b]
Total G&A Labor Costs	**66**	**$2,226**	**$1.22**

Notes:

a: The camp staffing costs (40 personnel) were included in the camp and catering rates.

b: Bulk explosives labor costs (2 personnel) were included in the blasting rates.

21.2.5.3 OFFICE COSTS

Table 21-19 presents the office expenditures based on historical data provided by Brigus.

Table 21-19: Office Operating Costs

Facility	Annual Cost ($000s)	Operating Cost ($/t milled)
Other Personnel Costs	$125	$0.068
Materials & Supplies	$100	$0.055
External Services	$100	$0.055
Computer Hardware & Software	$40	$0.022
Dues & Memberships	$25	$0.014
Fees & Permits	$15	$0.008
Rents	$25	$0.014
Taxes & Licenses	$150	$0.082
Telephone	$50	$0.027
Donations	$20	$0.011
Public Relations	$20	$0.011
Subscriptions	$1	$0.001
Insurance	$750	$0.411
Other Expenses	$20	$0.011
Total Office Operating Cost	**$1,441**	**$0.79**

21.2.5.4 EMPLOYEE TRANSPORTATION COSTS

The flight costs were based on the number of required flights to transport personnel to site. Table 21-20 summarizes the total site labor requirements.

Table 21-20: Mine Manpower Loading

Facility	Number of Personnel
Mining	
Mining Open Pit	64
Blaster	4
OP Mine Maintenance	26
Subtotal	94
Mine Engineering	
Engineering	6
Geology & Grade Control	10
Subtotal	16
Site Services & Blasting	
Site Services	6
Subtotal	6
Mill	
Mill Operations	33
Mill Maintenance	18
Subtotal	51
G&A	
G&A	12
Subtotal	12
Contract Services	
Contract Services	48
Subtotal	48
Total Site Employees	**227**

To account for the two week in/two week out schedule, transportation for each shift requires half the number of total employees. Required flight capacity of 120 employees per shift was used to allow for the employees that will be required to be on-site on a weekly schedule. The flight costs were calculated assuming that 40% of the employees will originate from northern Saskatchewan with the remaining from southern Saskatchewan. This yields a total weekly capacity requirement of 26 seats from northern Saskatchewan and 40 seats from southern Saskatchewan including a 10% contingency. A minimum of two flights per week from both northern and southern locations are required to accommodate employees on a weekly schedule. Northern flights will originate in Stony Rapids and southern flights will originate in Saskatoon.

Table 21-21 shows the cost breakdown of the required annual flights. Costs were provided through supplier quotation.

Table 21-21: Flight Costs

Flight Information	Flights per week	Value
Southern Flights:		
Saab 340 Annual Cost[a]	1	$781,000
Beech 1900 Annual Cost	1	$511,000
Subtotal		$1,292,000
Northern Flights		
Twin Otter Annual Cost	2	$242,000
Subtotal		$242,000
Total Annual Cost		$1,534,000
Total Flight Cost per Tonne Milled		$0.84

Note:
a: The flight costs are based on the assumption that the Uranium City Airport runway can be recertified to its original runway length to accommodate the larger planes with full capacity. Currently, the runway length is reduced to minimize the maintenance cost. With the original runway length the Saab 340 can carry its full capacity of 30 passengers. Without the recertification the carrying capacity of the flight would be reduced by 50%.

21.2.5.5 PROJECT RESIDENCE COSTS

The project residence costs were based on entering into a service contract to provide catering and maintenance costs for the operation of the camp. The following costs as shown in Table 21-22 were provided by supplier quotation.

Table 21-22: Residence Costs

Residence Costs	Value
Capacity	
Total Employees per Shift	120
Annual Costs	
Catering Costs	$2,443,000
Janitorial Costs	$371,000
Catering Equipment Costs	$14,000
Administration Costs	$36,000
Total Annual Cost	$2,864,000
Total Residence Cost per Tonne Milled	**$1.57**

21.2.5.6 REFINING COSTS

The refining costs were based on historical costs from Brigus based on an existing contract with Johnson Matthey Limited for the Black Fox Operation. Transportation costs were provided by supplier quote from Brinks Global Services. Total refining costs are presented in Table 21-23.

Table 21-23: Off-site Refining Costs

Refining Costs	Value
Variables	
Total Annual Ore (t/year)	1,825,000
Average Grade (g/t)	1.39[a]
Recovery Rate	91%
Refinery Payable	99.91%
Costs	
Refining ($/oz)	$0.55
Refining ($/Tonne Milled)	$0.02
Gold Freight ($/Tonne Milled)	$0.12
Total Annual Cost	$255,000
Total Refining Costs ($/t milled)	**$0.14**

Note:
a: Operating cost was determined based on a preliminary average grade. Average grade has been revised based on updated production schedule.

21.2.6 ADDITIONAL CAPITAL

21.2.6.1 SUSTAINING CAPITAL

Sustaining capital expenditures are capital expenditures resulting from improvements to and major renewals of existing assets.

The sustaining capital was evaluated based on the following:

- Mill on-going capital expenditures excluding regular equipment maintenance.
  - The annual mill capital expenditure were calculated as 2% of the installed mechanical costs.
- Mobile Equipment
  - Mid life and full life equipment rebuilds as required.
  - Additional mining equipment (i.e. haul trucks) as required to meet production demand due to increased cycle times.
- Dykes and Geosynthetics

- Based on volumes and budget quotations for the completion of berms for cells 3 & 4 of the WRSA and the monitoring pond between Vic Lake and Frontier Lake when required.

The sustaining capital was estimated as shown in Table 21-24. The total sustaining capital costs were calculated and then averaged over the LOM starting in year 2 of operation. The average annual cost of $3,692,000 was incorporated into the economics analysis.

21.2.6.2 ATHONA CAPITAL

The capital costs associated with establishing Athona are also included in Table 21-24. The costs include the preparation of the EIS, clearing and grubbing, and the construction of the haul road to the WRSA and crusher.

Table 21-24: Estimated Sustaining and Athona Capital Expenditures for the LOM

Year	2	3	4	5	6	7	8	9	10	11	12	13	Total
Susataining Capital													
Mill													
Mill Upgrades	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	**$1,200**
Mine													
Equip Rebuilds	$621	$7,072	$1,711	$1,868	$12,210	$2,218	$1,950	$8,088	$874				**$36,613**
Equip Additions			$1,799			$4248			$713				**$2,940**
Infrastructure													
Dyke #3 & 4 Fill	$3,300												**$2,700**
Dyke #3&4 Geosyn	$167												**$141**
Annual Total	**$4,188**	**$7,172**	**$3,610**	**$1,968**	**$12,310**	**$2,746**	**$2,050**	**$8,188**	**$1,687**	**$100**	**$100**	**$100**	**$44,219**
Annual Average	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**$3,692**	**-**
Athona Capital													
EIS		$800											**$800**
Clearing & Grubbing				$160									**$160**
Pit Pioneering						$1,300							**$1,300**
Haul Road Const					$2,067								**$2,067**
Annual Total		**$800**		**$160**	**$2,067**	**$1,300**							**$4,327**

21.2.6.3 MOBILE EQUIPMENT REBUILD CRITERIA

The mobile equipment rebuild criteria were developed from CAT product literature. The mobile equipment rebuild criteria used in determining the sustaining capital are presented in Table 21-25. For non-CAT equipment, the same mid life and full life rebuild criteria were used. It was estimated that the majority of all heavy equipment would require two mid-life rebuilds and one full life rebuild. Additional equipment replacement, with the exception of light vehicles, is not anticipated for the life of the mine.

Table 21-25: Equipment Rebuild Criteria

Equipment Hours	Rebuild Type	Estimated Cost
12,500 – 15,000	Mid Life Rebuild	35% of Purchase Price
25,000 – 30,000	Full Life Rebuild	60% of Purchase Price

21.2.6.4 CAPITALIZED DEVELOPMENT WASTE

The Box pit has been designed in four production phases. In order to prepare for subsequent production phases each phase has a component of waste rock that can be attributed to future expansion that can be capitalized. Figure 16-4 shows the relative proportions of production waste to development waste. For the economic analysis the costs associated with handling the development waste were subtracted from the operating expenses and included as a capital cost. Costs were based on the Box mining costs of $2.60/t mined.

A portion of the G&A costs that are attributable to the mining activity for the development waste can also be capitalized. This was calculated based on the mining manpower required for the development activities each year. It was determined that 68% of the non G&A labour was for mining. The ratio of development waste to total mine production on an annual basis was used to determine what portion of the mining G&A could be capitalized.

Table 21-26 shows the total quantity of development waste is 34 million tonnes accounting for a total capitalized cost of $102 million.

Table 21-26: Capitalized development waste

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Development Waste (t)	**3,163,279**	**9,130,960**	**7,553,860**	**4,473,685**	**7,762,114**	**1,722,797**	**419,609**	**34,226,304**
Capitalized Waste Cost ($000)	$8,225	$23,740	$19,640	$11,632	$20,181	$4,479	$1,091	**$88,988**
Capitalized G&A Costs ($000)	$823	$3,015	$2,665	$1,711	$3,124	$1,368	$436	**$13,143**
Total Capitalized Cost ($000)	**$9,047**	**$26,755**	**$22,305**	**$13,342**	**$23,306**	**$5,848**	**$1,528**	**$102,131**

21.2.7 SUMMARY

Table 21-27 provides a summary of the operating costs for the LOM costs for the Goldfields Project. Operating costs were calculated for Box, Athona, and stockpile recovery. The Athona and stockpile recovery costs were determined for use in the economic analysis. Mining costs per tonne milled were based on a strip ratio of 4.56 for Box and 1.10 for Athona.

The total operating cost for Box was $30.17/t milled. This included a mining cost for ore and waste of $2.60/t mined, a milling cost of $10.70/t milled, and a G&A cost of $4.99/t milled.

The total operating cost for Athona was $19.55/t milled. This included a mining cost for ore and waste of $1.97/t mined, a milling cost of $10.70/t milled, and a G&A cost of $4.70/t milled. The lower total operating cost for Athona, even though the haul distance is higher, is due to the low strip ratio and reduced manpower for mining.

The total operating cost for the recovery of the LG stockpile was $15.37/t milled. This included a transport cost for ore of $0.73/t mined, a milling cost of $10.70/t milled, and a G&A cost of $3.94/t milled. The lower G&A cost is due to the reduction of mining personnel and the associated reduction in camp and flight costs.

Process cost variations between the processing of Box, Athona, and LG stockpile ore were not considered in this analysis. Further process test work is required to determine if any process changes are required. For the purpose of the study the process costs were assumed to be constant.

Table 21-27: Operating Cost Summary

Description	BOX			ATHONA			LG STOCKPILE RECOVERY		
	$/t milled	$/t mined	Annual Costs ($000s)	$/t milled	$/t mined	Annual Costs ($000s)	$/t milled	$/t mined	Annual Costs ($000s)
Mine									
Mine Labour	$ 5.22	$0.94	$ 9,533	$ 1.36	$ 0.65	$ 2,482	$ 0.12	$ 0.12	$ 213
Mine Equipment	$ 9.25	$1.66	$ 16,882	$ 2.79	$ 1.33	$ 5,088	$ 0.61	$ 0.61	$ 1,117
Subtotal	**$ 14.47**	**$2.60**	**$ 26,415**	**$ 4.15**	**$1.97**	**$ 7,570**	**$ 0.73**	**$ 0.73**	**$ 1,330**
Mill									
Mill Labour	$ 2.42	-	$ 4,425	$ 2.42	-	$ 4,425	$ 2.42	-	$ 4,425
Mill Equipment	$ 1.11	-	$ 2,024	$ 1.11	-	$ 2,024	$ 1.11	-	$ 2,024
Mill Consumables	$ 7.17	-	$ 13,085	$ 7.17	-	$ 13,085	$ 7.17	-	$ 13,085
Subtotal	**$ 10.70**	**-**	**$ 19,533**	**$ 10.70**	**-**	**$19,533**	**$ 10.70**	**-**	**$19,533**
G&A									
Labour	$ 1.22	-	$ 2,227	$ 1.22	-	$ 2,227	$ 1.22	-	$ 2,227
Equipment	$ 0.43	-	$ 786	$ 0.43	-	$ 786	$ 0.43	-	$ 786
G&A Costs	$ 3.20	-	$ 5,839	$ 2.91	-	$ 5,317	$ 2.21	-	$ 4,031
Refining Costs	$ 0.14	-	$ 255	$ 0.14	-	$ 255	$ 0.08	-	$ 146
Subtotal	**$ 4.99**	**-**	**$ 9,106**	**$ 4.70**	**-**	**$ 8,584**	**$ 3.94**	**-**	**$ 7,189**
Total	**$ 30.17**	**-**	**$ 55,055**	**$ 19.55**	**-**	**$ 35,688**	**$ 15.37**	**-**	**$ 28,053**

22 ECONOMIC ANALYSIS

22.1 BASIS OF ANALYSIS

The economic analysis was conducted using a base case discounted cash flow model. The IRR, NPV, and payback period were determined on a before tax basis for Box, Athona, and the overall project.

For the purposes of the economic analysis the costs required for the preparation of the Athona EIS and establishment of haul roads and pit pioneering were estimated. Minimal capital was included in the economic analysis to establish Athona since existing infrastructure and process equipment would be used for the Athona operations.

22.2 ECONOMIC MODEL PARAMETERS

The cost and production information that was developed during the study was used to perform the economic analysis. The mine production schedule in Table 16-2 provided the quantities of ore and waste rock and average grades. The analysis included Box, Athona, and LG stockpile recovery.

The capital cost summary is included in Table 21-8. The total capital cost was $159,235,000.

Operating costs were determined for Box, Athona, and LG stockpile recovery. The operating cost summary is located in Table 21-27.

The economic analysis included a lease payment schedule for the equipment and project residence. The sustaining capital and estimated capital expenditures for Athona were also included as summarized in Table 21-24. Table 22-1 summarizes other criteria used in the economic analysis.

Table 22-1: Economic Analysis Criteria

Revenue Parameters	
Time frame validity	2010, fourth quarter
Gold price	$1250 per troy ounce
Gold recovery	
Box	91%
Athona	89%
Royalties	2% - Franco Nevada
Environmental bond	Carrying cost of $300,000 over LOM
Closure costs	Estimated at $2,500,000 at end of mine life
Taxes	Excluded
Sustaining Capital	Cost were averaged over entire mine life

22.3 SUMMARY

An economic analysis was conducted to determine the NPV, IRR, payback period, and cash cost per ounce for Goldfields. For the economic analysis, an average gold price of $1,250/troy oz was used. The economic indicators are presented in Table 22-2. The NPV at a 5% discount rate was $144,308,000 with an IRR of 19.6%. The cash cost per ounce was $601.

Table 22-2: Economic Analysis Results

Variable	Location	Values
NPV @ 5%	**Project**	**$144,308,000**
	Box	$80,110,000
	Athona	$64,197,000
IRR	**Project**	**19.6%**
	Box	15.5%
	Athona	151.1%
Cash Cost ($/oz)	**Project**	**$601**
	Box	$605
	Athona	$585
Total Cost ($/oz)	**Project**	**$940**

22.4 SENSITIVITY ANALYSIS

A sensitivity analysis was completed for the project economics to determine which variable had the greatest impact on the project economics. It was determined that of the four variables that were anlysed, process recovery was the most sensitive followed closely by gold price. Operating cost and capital cost were found to be less sensitive. Figure 22-1 illustrates the relative sensitivities of the various parameters.

The sensitivity chart has the following characteristics:

- Steepest slope indicates highest sensitivity
- Variables were analysed based on their effect on project IRR
- Positive slope indicates IRR increases as the variable increases (i.e gold price and process recovery)
- Negative slope indicates IRR decreases as the variable increases (i.e. capital and operating costs)

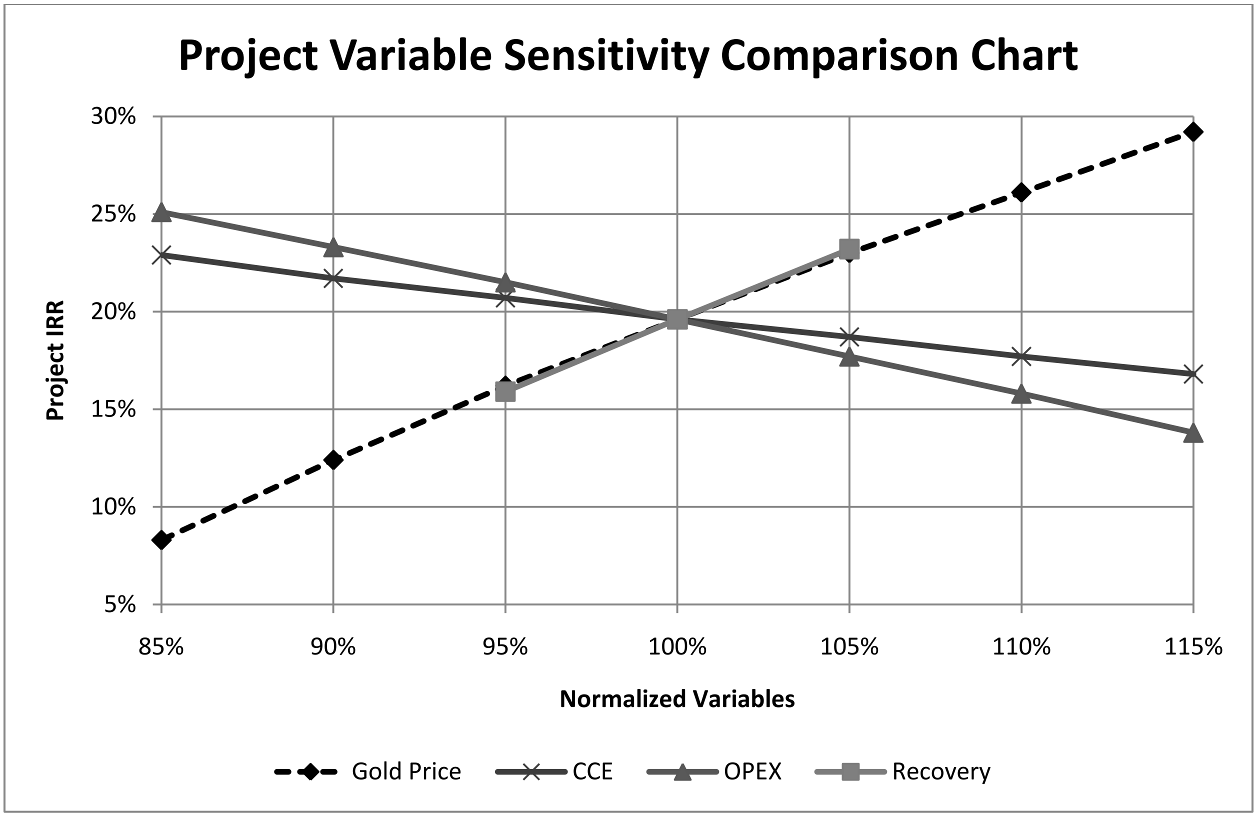

Figure 22-1: Sensitivity Analysis Results

Figure 22-2 illustrates the impact on project IRR with variability in the gold prices from $1,063 – 2,000/oz. Project IRR varied from a low of 8% to a high of 53%. A gold price of $1,250/oz was used as the baseline.

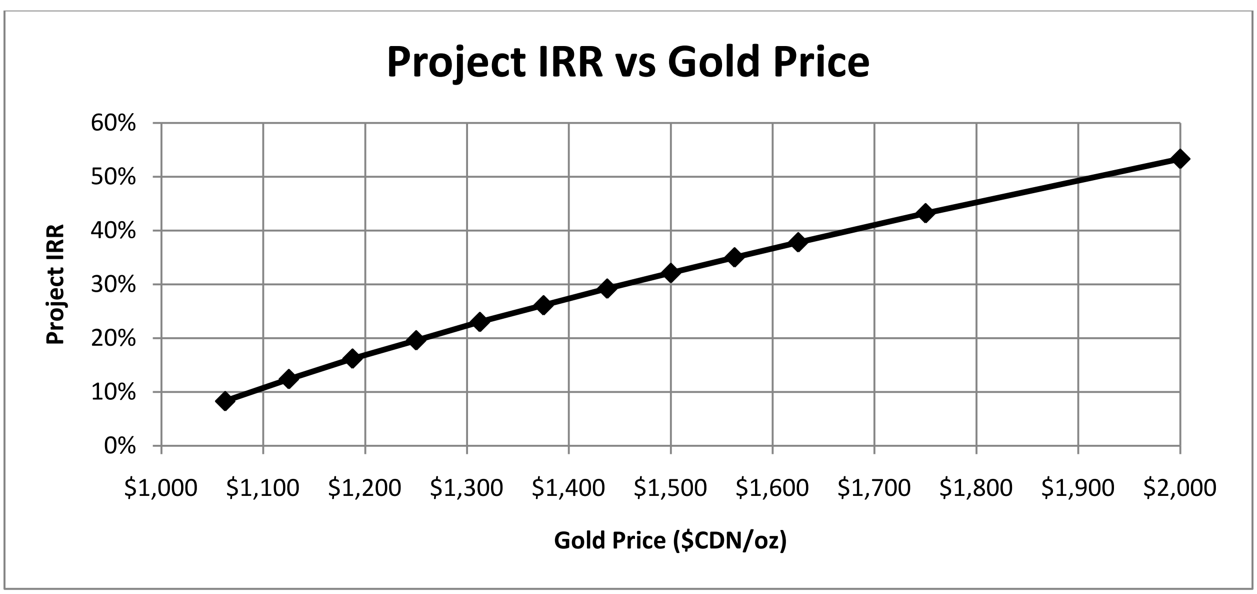

Figure 22-2: Project IRR vs. Gold Price

Figure 22-3 illustrates the effect of gold prices from $1,063 – 2,000/oz on the project NPV at a 5% discount rate. Project NPV ranged from a low of $27,900,000 to a high of $611,200,000. The gold price of $1,250/oz was used as the baseline.

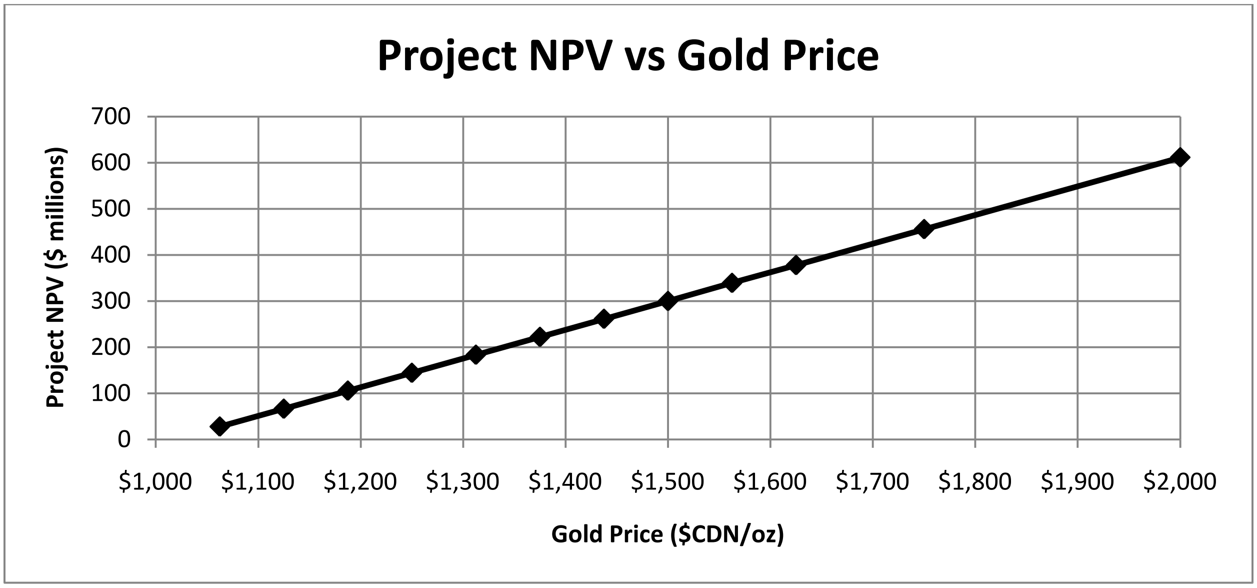

Figure 22-3: Project NPV vs. Gold Price

Figure 22-4 shows the project sensitivity to process recovery from 86 – 96% for Box. The IRR range was from 16% to 23%.The Box recovery at 91% and Athona recovery at 89% was used as the baseline.

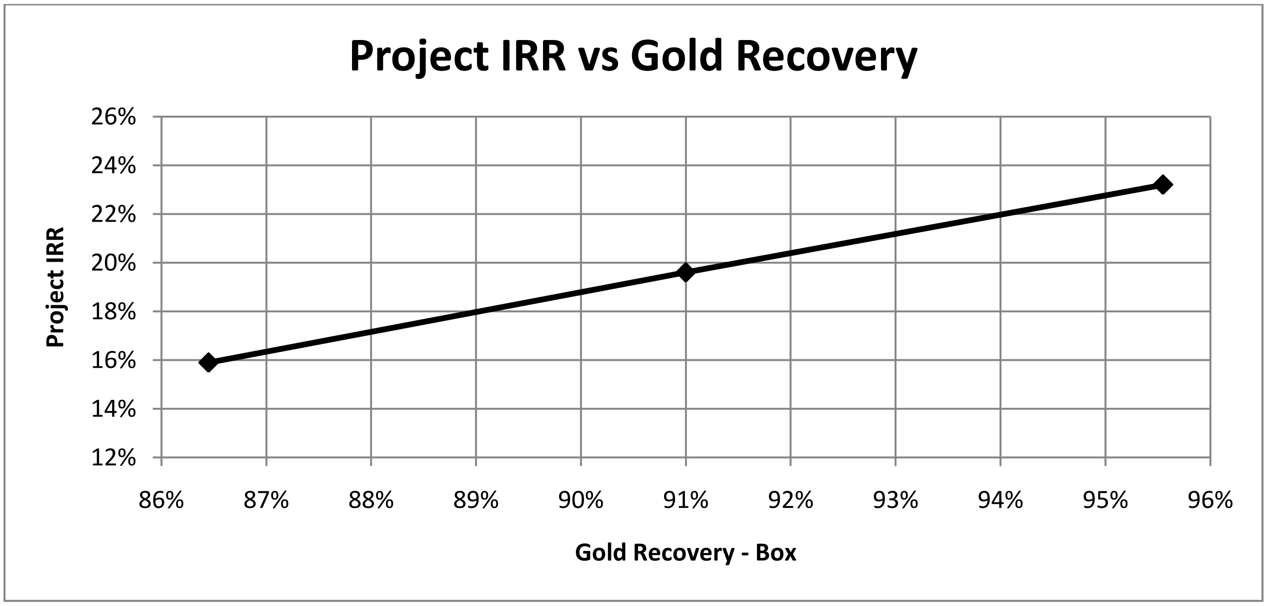

Figure 22-4: Project IRR vs. Gold Recovery

Figure 22-5 shows the effect that total capital cost has on project economics. Total capital of $160,000,000 for Box was used as the base case. Capital costs from $136,000,000 to $184,000,000 were analysed, the IRR ranged from a high of 23% to a low of 17%.

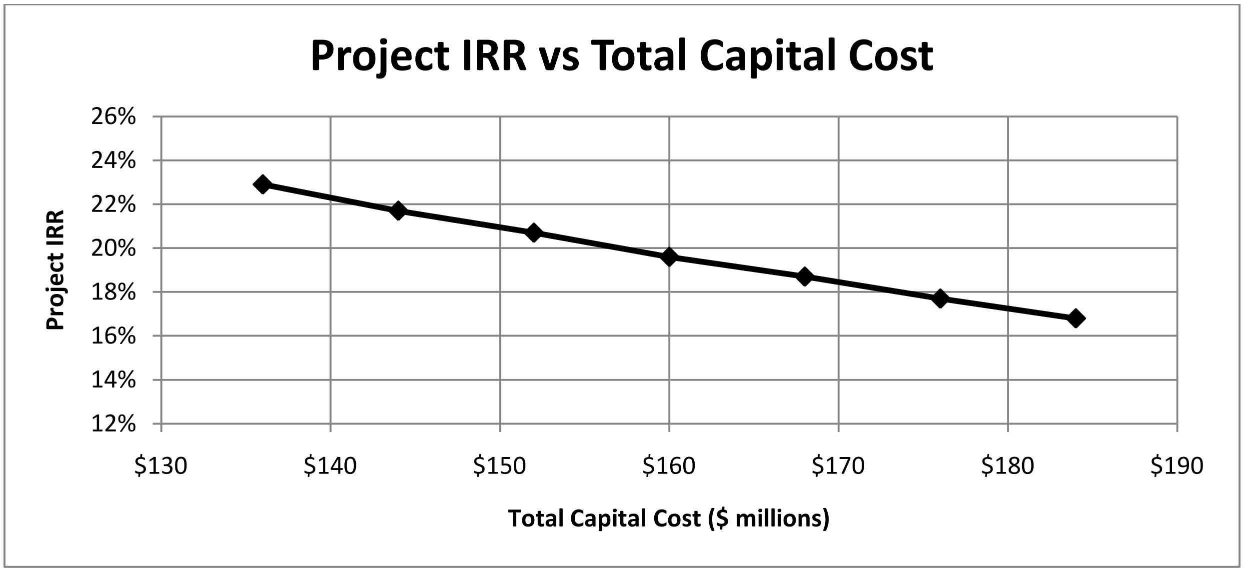

Figure 22-5: Project IRR vs. Total Capital Cost – Lease Option

Figure 22-6 demonstrates the project sensitivity to the total operating cost. For the baseline, the operating costs for Box at $30/t milled were used. Operating costs were varied from $26 – 35/t. The project IRR varied from a high of 25% to a low of 14%.

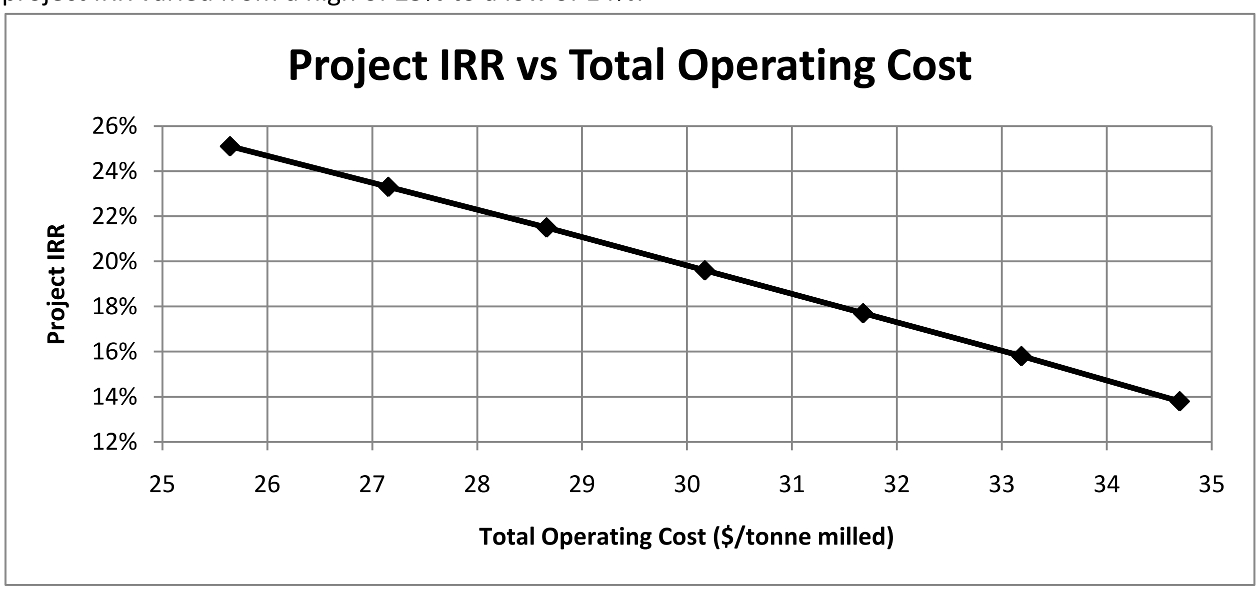

Figure 22-6: Project IRR vs. Total Operating Cost

Figure 22-7 demonstrates the effect on project NPV of varying the discount rate. A baseline rate of 5% was used for the analysis. The discount rate was varied from 2 – 10% while project NPV varied from a high of $208,000,000 to a low of $72,000,000 compared to the baseline NPV at 5% of $144,000,000.

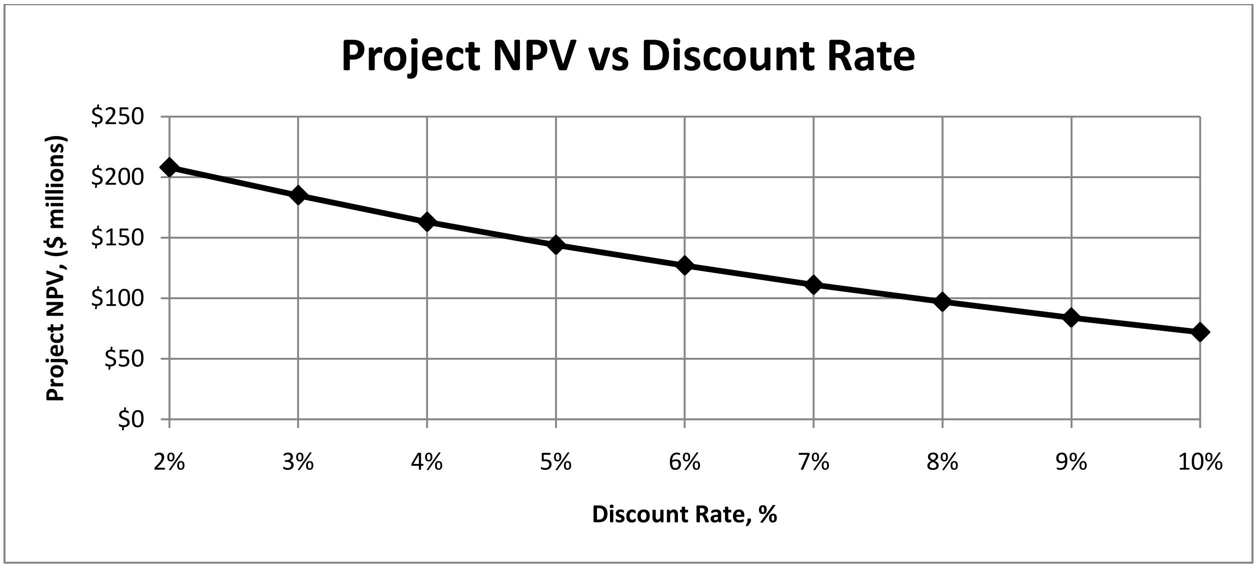

Figure 22-7: Overall Project NPV vs. Discount Rate

22.5 OPERATING PERIOD CASHFLOW

Table 22-3 summarizes the operating period cashflow. Payback period was calculated at five years from the start of operations. A cashflow shift of 25% has been incorporated into the economic analysis. The cashflow shift represents the potential delay between the production of gold and the realization of income from the gold sales. The total annual cashflow is presented after the cashflow shift has been applied.

Table 22-3: Operating Period Cashflow

	Operating Period ($000s)												
Year of Operation	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**
Net Revenue													
Box	$73,156	$130,047	$127,433	$126,779	$121,551	$128,740	$87,742	$0	$12,767	$33,328	$33,328	$15,670	$0
Athona	$0	$0	$0	$0	$0	$0	$32,483	$102,901	$62,694	$0	$0	$17,261	$24,257
Total	**$73,156**	**$130,047**	**$127,433**	**$126,779**	**$121,551**	**$128,740**	**$120,225**	**$102,901**	**$75,461**	**$33,328**	**$33,328**	**$32,941**	**$24,257**
Costs													
Operating Costs													
Box	$42,001	$63,405	$64,717	$68,838	$46,714	$44,357	$30,420	$0	$10,745	$28,050	$28,050	$13,196	$0
Athona	$0	$0	$0	$0	$0	$0	$15,189	$39,395	$24,163	$0	$0	$14,854	$20,874
Total	**$42,001**	**$63,405**	**$64,717**	**$68,838**	**$46,714**	**$44,357**	**$45,609**	**$39,395**	**$34,908**	**$28,050**	**$28,050**	**$28,050**	**$20,874**
Capitalized Development Waste	$9,047	$26,755	$22,305	$13,342	$23,306	$5,848	$1,528	$0	$0	$0	$0	$0	$0
Athona Capital Expenditures	$0	$0	$800	$0	$160	$2067	$1,300	$0	$0	$0	$0	$0	$0
Sustaining Capital	$0	$3692	$3692	$3692	$3692	$3692	$3692	$3692	$3692	$3692	$3692	$3692	$3692
Environmental Bond	$25	$25	$25	$25	$25	$25	$25	$25	$25	$25	$25	$25	$0
Reclamation/ Closure	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$2,500
Total Annual Cashflow	**-$12,687**	**$27,298**	**$36,875**	**$42,076**	**$43,431**	**$70,366**	**$70,513**	**$62,567**	**$42,575**	**$10,380**	**$1,562**	**$1,271**	**($1,586)**

23 ADJACENT PROPERTIES

The Goldfields Claim Group lies in the historical uranium mining camp of Eldorado, Goldfields and Beaverlodge Lake. There are currently no operating mines in the Uranium City area. Most of the closed and abandoned mines were primarily uranium mines such as the Eldorado, Gunnar, Lorado, and Beaverlodge uranium mines.

The Goldfields Claim Group encompasses the historic Box, Athona, and Frontier gold mines and the Golden Pond gold occurrence. All these deposits lie within a three kilometre radius of the Box.

There are no known significant gold occurrences adjacent to the Goldfields Claim Group; however, the claim group is surrounded by several mineral claims, held by companies and individuals, mainly for uranium exploration. The mineral claims adjacent to the Goldfields claim group are held by Canalaska Uranium Ltd. (Canalaska) and Red Rock Energy Inc. (Red Rock) to the south; JNR Resources Inc. and Red Rock to the west; Red Rock and Mr. Rod Dubnick to the north and northeast, and; Canalaska to the east.

The following information has not been verified by any of the qualified persons for this report, is offered for information purposes only and is not necessarily indicative of the mineralization on the Goldfields. This information was extracted from the technical report prepared in 2009 for the Goldfields entitled "TECHNICAL REPORT PURSUANT TO NATIONAL INSTRUMENT 43-101 OF THE CANADIAN SECURITIES ADMINISTRATORS" prepared for Linear by Bikerman Engineering & Technology Associates, Inc.

23.1 FRONTIER MINE

23.1.1 LOCATION AND MINERALIZATION

The Frontier Mine deposit and the Golden Pond occurrence are the closest to the proposed Box mill complex and are more advanced than the other gold mineralization occurrences. Both of these areas indicate the potential for substantial gold mineralization and may warrant future exploration activities. The Frontier Mine deposit was explored by a horizontal adit approximately 100.0 meters long, which was driven northwest from the end of a small lake, and 185.6 meters (609 feet) of drifting and 104.0 meters (341 feet) of cross-cuts, prior to 1938. Currently, the Frontier mineralization appears to be of higher grade narrow vein gold style deposit with gold grades ranging from 8.16 g/t (0.238 oz/st) to 28.18 g/t (0.822 oz/st) while the recent diamond drilling at the Golden Pond occurrence indicates that it may have the potential of open pit mining with gold grades of 5.07 g/t (0.148 oz/st) over 15.0 meters (49.2 feet) and 16.53 g/t (0.482 oz/st) over 13.6 meters (44.6 feet) (Jensen, 2003).

23.1.2 GEOLOGY

The geological setting of the Frontier Mine deposit consists of alternating sequence of amphibolites and quartzites to arkosic quartzites subjected to auriferous quartz veining in two prominent directions of N010°E and N315°E. The mineralization occurs in a sill-like body of fine grained pink granite 6 to 18 m thick which is emplaced into the quartzite below an amphibolite sill. The granite, quartzite and amphibolite dip 30° southeast. The quartzites have undergone silicification and varying degrees of hematitization. Beavan (1938) noted that the oxidation extends at least 180 m below the surface. The main auriferous quartz veining zone is exposed by surface trenching which is located near the quartzite amphibolite contact. The quartzite zone containing the quartz veining is truncated by the north-northwest Triangle Lake fault (west) and extends along strike to a N315°E fault at the eastern end of Frontier Lake.

The underground workings indicate a lower and upper quartzite-arkosic zones which have been silicified and hematitized and striking in northeasterly direction. The eastern portion of the workings and diamond drilling indicated the existence of two higher grade gold quartz vein zones striking N010°E and N315°E, while the western portion contains a N008°E higher grade gold quartz vein zone.

The location of Frontier Lake and its orientation may indicate a zone of weakness which was susceptible to erosion and a fault zone parallel to the lithological units.

23.2 'BEARCAT' GOLD SHOWING (CODY PROPERTY)

23.2.1 LOCATION

This property is located approximately 1.6 km northwest of Mackintosh Bay and 0.4 km southeast of Wabba Lake.

23.2.2 HISTORY AND DEVELOPMENT

The Bearcat group of claims was staked in 1934 by E. Cody, O. Knutson and J.G. Paulsen. The property was optioned to Ventures Limited in 1935. Considerable trenching was done 1935-36. No further work has been reported other than exploration for uranium.

23.2.3 GEOLOGY

East-southeast (ESE) trending, south-dipping quartzites and calcareous sediments have been invaded by a stock-like body of red to grey granite. The granite is cut by northeast-trending quartz veins and stringers, some of which carry pyrite and minor galena. It is reported that no samples carry more than 0.02 ounce per ton gold.

23.2.4 REFERENCES

The following references were used for this section of the report:

- Alcock (1936a);
- Beck (1959), p. 50-51;
- SEM Assessment File 74N080028 (location of trenching); and
- SMDI 74-N-8-NW Au-4

23.3 MURMAC SHOWING

23.3.1 LOCATION

This property is located at the north end of Cornwall and Goldfields Bay.

23.3.2 HISTORY AND DEVELOPMENT

The Murmac group of 15 claims was staked by J.E. Day and associates in 1934 and later transferred to Murmac Lake Athabasca Mines Limited. Apparently several gold discoveries were made during 1934-35. Cominco drilled six (6) holes (480 m total) on two (2) of the showings in 1936. There is no record of the results.

23.3.3 GEOLOGY

The area is underlain by Murmac Bay supracrustals invaded by granitic rocks. Gold mineralization is reported from at least three (3) parts of the claim group. Just west of the north end of Cornwall Bay gold occurs in a quartz vein in grey granite. At the boundary of Murmac Fraction No. 6 and the Cominco property (i.e. approximately 0.8 km northeast of the end of Frontier Lake) gold also occurs in a quartz vein cutting grey granite. Approximately 75 m north of the latter location a series of parallel quartz stringers carry gold.

23.3.4 REMARKS

Also in the same general vicinity, approximately 0.8 km west-northwest of the north end of Cornwall Bay, Beavan (1938) reports gold mineralization on the Rita and Maud groups of claims of Goldfields Mining Company Limited. Here an irregular lens of granite dips gently south and is overlain by an amphibolite sill. Fracturing is localized in the granite below the amphibolite. Narrow, discontinuous quartz-filled fractures trend southwesterly as well as easterly. Pyrite, sphalerite, galena and gold occur in the quartz veins.

23.3.5 REFERENCES

The following references were used for this section of the report:

- Alcock (1936a);
- Beavan (1938);
- Beck (1959), p. 51;
- SEM Assessment File 74N080101; and
- SMDI 74-N-8-NW Au-5

23.4 NORTHWEST MINERALS SHOWINGS (HAZEL SHOWINGS)

23.4.1 LOCATION

This property is located at Caldwell Bay area.

23.4.2 HISTORY AND DEVELOPMENT

Prospectors C.W. Shearing and R. Alloway discovered gold in the vicinity of Caldwell Bay in the fall of 1934. Northwest Minerals Limited staked the Hazel group of 12 claims over the showings, as well as several other adjacent groups of claims, in September to October of 1934. Several gold-bearing zones were discovered and trenched in 1935, especially on Hazel claims 1 and 2. Two drill holes were emplaced on the `Centre zone' on Hazel claim 2 late that year. Further drilling and shaft sinking were recommended for 1936, but there is no record of this work being completed. Gold mineralization was also discovered on the Juca and Max groups. There is no record of development work on these claims. No further work is reported until 1953 when Loranda Uranium Mines Limited carried out some work for uranium in the area. Nothing was done on the gold showings. In 1973, Calgary International Energy Limited did further work in the area. Limited exploration was done in the vicinity of the gold showings which resulted in the discovery of a further gold occurrence.

23.4.3 GEOLOGY

Gold mineralization on the Hazel 1 and 2 claims occurs within the southwest part of the large Mackintosh Bay granite body, close to the contact with surrounding quartzites. In a private report to Northwest Minerals Limited, C.W. McKee describes the mineralization as follows: "A mineralized fractured and sheared zone occurs in intrusive porphyritic granitic rock, varying in width from 100 ft. to 800 ft. and some 2500 ft. long. It is estimated that 35 percent of the zone is exposed rock. The exposed area is mineralized with pyrite throughout, with occasional chalcopyrite, pyrrhotite and galena, and cut by numerous cross and parallel quartz veins and stringers. Fine flakes, clusters and coarse kernels of gold have been located at several points in the zone over a length of 1000 ft. and over a width of 240 ft.

Substantial gold values have also been obtained from the host rock from various pits throughout the zone."

Within this broad area several mineralized zones have been identified by Northwest Minerals Limited. Calgary International Energy sampled a mineralized zone west of the above zones. A sample from this zone (termed the "upper vein" by the company) returned 0.24 ounce per/ton gold over a 0.6 m width.

23.4.4 REFERENCES

The following references were used for this portion of the report:

- Alcock (1936a);
- Beck (1959), p. 51-52;
- SEM Assessment Files 74N080094, 74N080103; and
- SMDI 74-N-8-NW Au-1

23.5 YAH SHOWINGS

23.5.1 LOCATION

This property is located east and southeast of the east arm of Fish Hook Bay.

23.5.2 HISTORY AND DEVELOPMENT

Athabasca-Beaverlodge Gold Mines Limited staked the Yah group of ten claims in 1934. A considerable amount of surface work was carried out prior to 1936. There is no record of any further work on the gold showings since that time.

23.5.3 GEOLOGY

Mineralization on the Yah group occurs within the Mackintosh Bay granite, close to the western contact with the adjacent Murmac Bay supracrustals. The granite contains pyrite and numerous quartz stringers. The latter also carry pyrite and minor galena. Visible gold is present in three zones. No record is available of the grades present

23.5.4 REFERENCES

The following references were used for this portion of the report:

- Alcock (1936a);
- Beck (1959),p. 52; and
- SMDI 74-N-8-NW Au-7

23.6 NEELY LAKE (BOREALIS SYNDICATE)

23.6.1 LOCATION

This property is located approximately 16 km northeast of Uranium City, at the southwest end of Neely Lake.

23.6.2 HISTORY AND DEVELOPMENT

Gold was discovered at Neely Lake in 1935 (?). The Borealis Syndicate carried out extensive trenching and sunk a small prospect shaft 1935-36. Exploration for uranium was undertaken at later dates but there is no record of any further work on the gold showings.

23.6.3 GEOLOGY

The rocks of the area around Neely Lake form part of the 'Black Bay Straight Belt' (Macdonald, 1983). They are mostly quartzofeldspathic gneisses and granitic gneisses as well as subordinate amphibolites. At Neely Lake, 'Tazin Group' grey quartzites and surrounding granitic gneisses are invaded by small masses of fresh red granite. A granite dyke on the northwest shore of the lake is cut by numerous quartz veins and stringers that occur up to 20 cm wide. Pyrite occurs within both the quartz veins and fractured granite. Gold content of the mineralization is unknown.

23.6.4 REFERENCES

The following references were used for this portion of the report:

- Alcock (1936a);
- Beck (1959), p. 52;
- Christie (1952); and
- SMDI 74-N-9-NW Au-1

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relelvant data or information required for this report.

25 INTERPRETATION AND CONCLUSIONS

The Goldfields Project has progressed to the point of defined resources and reserves. The capital and operating costs have been determined and are found to provide acceptable project economics. At $1,250/oz gold price the NPV at 5% discount rate was $144 million with an IRR of 19.6%. Cash cost was estimated at $601/oz and total cost was $940/oz. The capital cost was determined to be $160 million with an estimated construction duration of three years. Operations are scheduled to commence in the second half of the third year of construction. An EIS for the Goldfields project was approved on May 28, 2008.

26 RECOMMENDATONS

Based on the results of the pre-feasibility study the following opportunities were identified for the project:

- Continue exploration drilling in relevant areas of both deposits to enhance the resource estimate
- Conduct project specific process test work and optimize process recovery
- Complete the geotechnical assessment and update the ore reserve models to reflect the potential revised pit design
- Advance the project planning and design to minimize potential execution risks

Estimated cost for these recommendations was $1.8 million.

27 REFERENCES

Beck, L.S.: 1959: Mineral Occurrences in the Precambrian of Northern Saskatchewan (Excluding Radioactive Minerals); Saskatchewan Department of Mineral Resources, Report No.36, 134p.

AMEC, 2004a. DRAFT – Site Visit to Box and Athona – Goldfields Project, Canada, Memo Submitted from Kevin Francis to Rod Ogilvie, John Dixon, and Bob Kasner. October 7, 2004.

AMEC, 2004b. Goldfields Project – Geologic Data and Resource Model Review, 2004 AMEC report.

AMEC, 2004c. Data Review of Goldfields Project, Greater Lenora Resources, Memo from Susan Lomas to Kevin Francis. October 18, 2004

AMEC, 2004d. GLR Resource Estimate Status – DRAFT, Memo from Kevin Francis to Will Brandsema. 23 October 2004.

AMEC, 2005e, "February 2005 Box Mine Project," February 2005 report for GLR, Mike Lechner (RMI) and Kevin Francis, AMEC.

AMEC, 2005f, Box Mine Project: Resource Estimate for Greater Lenora Resources. April 2005. 51 pages.

Carmicheal, S., 1990. Report on the 1989 Reverse Circulation Drilling Program, Goldfields Project. Beaverlodge Area, Saskatchewan for RJK Mineral Corp.

Jensen, K.A., 2003. Technical Report on the Goldfields Property for GLR Resources Inc., in the Beaverlodge Lake Area, NTS Map Sheets 74N-06, 74N-07, 74N-08, 74N-09, and 74N-10, Northern Mining District, Saskatchewan, Canada. 12 December 2003. 167 pages.

Wardrop, 2007. Technical Report on the Athona Deposit, SK. May 17, 2007. 77 pages.

Quantec, 2010. Titan-24 DC/IP Survey Geophysical Report. Goldfields Project (near Uranium City, SK) on behalf of Linear Gold Corp., Toronto, ON. 9 April, 2010. 180 pages.

Swanson, C., 1938. Notes on the Geology of the Box Mine. Goldfields, Saskatchewan. July 1938. 19 pages.

Bikerman, Box Mine-Goldfields Project, Uranium City, SK, Canada. September, 2009. 303 pages

UMA Engineering Ltd. Report Greater Lenora Resources Inc., Environmental Impact Statement, September, 2007;

Box Starter Pit Mine, Goldfields Project, Northern Saskatchewan as submitted in January 2007

H.A. Simons, September 1995. Greater Lenora Resources Corporation Goldfields Project, Box and Athona Deposits Geological Reserve Estimate

Proposed Grinding Circuit Design for The Greater Lenora Concentrator of Greater Lenora Resources Corporation Based on Pilot Plant Tests at Lakefield Research, Revised August 1995

Richard C. Swider, Consulting Engineers Limited, Revised October 2, 1995, Greater Lenora Resources Corporation, Goldfields Project, Review of Metallurgical Testing

28 ILLUSTRATIONS

Illustrations are included throughout the report in their applicable sections.

29 GLOSSARY

29.1 MINERAL RESOURCES AND RESERVES

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.

29.1.1 MINERAL RESOURCES

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 drill holes.

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 drill holes 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 drill holes that are spaced closely enough to confirm both geological and grade continuity.

29.1.2 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.

29.2 GLOSSARY

29.2.1 DEFINITIONS OF TERMS

Adit – Is an entrance to an underground mine which is horizontal or nearly horizontal by which the mine can be entered, drained of water and ventilated

Flocculant – A chemical for producing flocculation of suspended particles.

Kriging – Is a group of geostatistical techniques to interpolate the value of a random field (elevation of the landscape as a function of the geographic location) at an unobserved location from observations of its value at nearby locations.

Leachate – Is a solution formed by leaching, any liquid containing soluble material removed from a solid mixture through which the liquid has passed.

Lithological –Of a rock unit is a description of its physical characteristics visible at outcrop, in hard or core samples with low magnification. Such as texture, colour, grain size or composition.

Mineralization – The hydrothermal deposition of economically important metals in the formation of ore bodies.

Oligotrophic – Low nutrient concentrations in lakes

Piezometer – Is a small diameter observation well used to measure the hydraulic head of groundwater in aquifers. It may also be a standpipe, tube, or manometer used to measure the pressure of a fluid at a specific location.

Process water – Any untreated water used in the process could be considered to be process water.

Raw water – Is water taken from the environment that has not been purified or treated.

Stopes - Excavation in the form of steps made by the mining of ore from steeply inclined or vertical veins.

Talus slopes - A steep, concave slope consisting of an accumulation of talus, also known as a debris slope.

Vuggy – Is a small cavity in a rock or vein, often lined with crystals.

Wallrock – Is the rock that constitutes the wall of an area undergoing activity, as in the rock along the neck of a volcano.

29.2.2 ABBREVIATIONS AND ACRONYMS

Athona Mine	Athona
Athona Mine Granite	AMG
Athona West Granite	AWG
Azimuth	Az
Basis of Estimate	BOE
Box Mine	Box
Box Mine Granite	BMG
Brigus Gold Corp.	Brigus
Canadian Environmental Quality Guidelines	CEQG
Canalaska Uranium Ltd	Canalaska
Consolidated Mining and Smelting Company of Canada Limited	Cominco
Cut off grade	COG
Dan Mackie Associates	DMA
Department of Fisheries and Oceans	DFO
EHA Engineering Ltd.	EHA
Electromagnetic	EM
Environmental Impact Statement	EIS
Foot Wall	FW
General & Administrative	G&A
Global Positioning System	GPS
GLR Resources Inc.	GLR
Gold	Au
Gold and Platinum Group Elements	Au-PGE
Goldfields Property	Goldfields
Hanging Wall	HW
Heating Ventilation Air Conditioning	HVAC
High Grade	HG
Induced Polarization	IP
Interim Sediment Quality Guidelines	ISQG
Internal Rate of Return	IRR
Inverse Distance Squared	ID^2
Life of Mine	LOM

Linear Gold Corp. .... Linear
Low Grade .... LG
March Consulting Associates Inc. .... March Consulting
Marginal cut off grade .... MCOG
Metal Mining Effluent Regulations .... MMER
Methylisobutyl carbine .... MIBC
National Instrument 43-101 .... NI 43-101
National Topographic System .... NTS
Nearest Neighbour .... NN
Net Present Value .... NPV
Net Smelter Return .... NSR
Ordinary kriging .... OK
Environmental Protection Agency .... EPA
Pre-feasibility Study .... PFS
Process Flow Diagram .... PFD
Professional Geologist .... P. Geo.
Project Study Area .... PSA
Quality Assurance/Quality Control .... QA/QC
Quantec Geoscience Ltd. .... Quantec
Red Rock Energy Inc. .... Red Rock
Reverse Circulation .... RC
Royal Canadian Mounted Police .... RCMP
Saskatchewan .... SK
Saskatchewan Environmental Regulations Management .... SERM
Saskatchewan Ministry of Environment .... MOE
Saskatchewan Surface Water Quality Objectives .... SSWQO
Species at Risk Association .... SARA
Tailings Management Facility .... TMF
Universal Transverse Mercator .... UTM
Very Small Aperture Terminal .... VSAT
Voice over Internet Protocol .... VoIP
Wardrop, a Tetra Tech Company .... Wardrop
Waste Rock Storage Area .... WRSA

29.2.3 UNITS OF MEASURE

Above sea level .... ASL
Acre .... ac
Ampere .... A
Annum (year) .... a
Centimetre .... cm
Cubic Centimetre cc
Cubic metre .... m^3
Cubic meter per hour .... m^3/h
Cubic yard .... yd^3
Dead weight tonnes .... DWT
Degree .... °

Degrees Celsius .... °C
Dollar (American) .... US$
Foot .... ft
Gallon .... gal
Gram .... g
Grams per ounce .... g/oz
Grams per tonne .... g/t
Greater than .... >
Hectare (10,000 m^2) .... ha
Inch .... "
Kilo (thousand) .... k
Kilogram .... kg
Kilograms per cubic metre .... kg/m^3
Kilograms per hour .... kg/h
Kilograms per square metre .... kg/m^2
Kilometre .... km
Kilometres per hour .... km/h
Kilovolt .... kV
Kilovolt-ampere .... kVA
Kilowatt .... kW
Kilowatt hour .... kWh
Kilowatt hours per tonne (metric ton) .... kWh/t
Less than .... <
Litre L
Litres per minute .... L/m
Megavolt-ampere .... MVA
Metre .... m
Metric ton (tonne) .... t
Microns .... µm
Millimetre .... mm
Million .... M
Million tonnes .... Mt
Minute (plane angle) .... '
Ounce .... oz
Ounce .... oz/st
Overflow .... o/f
Oversized material .... o/s
Parts per billion .... ppb
Percent .... %
Second (plane angle) .... "
Second (time) .... s
Specific gravity .... SG
Short ton per day .... st/d
Tonne (1,000 kg) .... t
Tonnes per day .... t/d
Tonnes per hour .... t/h
Tonnes per year .... t/y

Underflow .... u/f
Undersized material .... u/s
Volt V
Weight/weight .... w/w
Wheel Drive .... WD

APPENDIX A

CERTIFICATES OF AUTHORS

DMA
Dan Mackie & Associates

CERTIFICATE of AUTHOR

I, Dan Mackie, P.Eng. do hereby certify that:

1. I am President of:

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2. I graduated with a degree in mechanical engineering from McGill University in 1961.

3. I am a member of the Professional Engineers of Ontario.

4. I have worked as a mechanical engineer for a total of 50 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 sections 13 and 17 of the technical report titled NI 43-101 Technical Report, Pre-feasibility Study, Brigus Gold Corporation, Goldfields Project, Saskatchewan, Canada and dated October 6th, 2011 (the "Technical Report") relating to the Goldfields property.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is development of a feasibility study for GLR Resources.

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.

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Signature of Qualified Person

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I, Al Hayden, P.Eng. do hereby certify that:

1. I am President of:

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2. I graduated from the University of British Columbia, Vancouver, B. C. in 1967 with a Bachelor of Applied Science in Metallurgical Engineering.

3. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum and a Professional Engineer and Designated Consulting Engineer registered with Professional Engineers Ontario.

4. I have worked as a metallurgical engineer for a total of 44 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 sections 13 and 17 of the technical report titled NI 43-101 Technical Report, Pre-feasibility Study, Brigus Gold Corporation, Goldfields Project, Saskatchewan, Canada and dated October 6th, 2011 (the "Technical Report") relating to the Goldfields property.

7. I have had no 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.

Dated this 6th Day of October, 2011.

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I, Clifford Lusby, *P. Eng.* do hereby certify that:

1. I am Principal Mine Engineer Associate of:

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2. I graduated with a degree in Bachelor of Engineering in mine engineering with distinction from the Technical University of Nova Scotia in 1979.

3. I am a member of the Association of Professional Engineers & Geoscientists of Saskatchewan - Member #5643 and the Association of Professional Engineers & Geoscientists of British Columbia - Member #31378.

4. I have worked as a mine engineer for a total of 32 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 sections 15 and 16 of the technical report titled NI 43-101 Technical Report, Pre-feasibility Study, Brigus Gold Corporation, Goldfields Project, Saskatchewan, Canada and dated October 6th, 2011 (the "Technical Report") relating to the Goldfields property. I visited the Goldfields property on March 18th, 2010 and September 10th to 13th, 2010 for five days.

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.

Dated this 6th Day of October, 2011.

"Original document signed and Sealed by Cliff Lusby, P.Eng."

Signature of Qualified Person

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I, Tim Maunula, P.Geo. do hereby certify that:

1. I am Chief Geostatistician of:

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2. I graduated with a H.B.Sc. degree in Geology from Lakehead University in 1979. In addition, I have obtained a Citation in Geostatistics from the University of Alberta in 2004.

3. I am a member of the Association of Professional Geoscientists of Ontario (Registration Number 1115).

4. I have worked as a Geologist for a total of 32 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 sections 6.1 - 6.8, 7 - 12, 14, and 27 of the technical report titled NI 43-101 Technical Report, Pre-feasibility Study, Brigus Gold Corporation, Goldfields Project, Saskatchewan, Canada and dated October 6th, 2011 (the "Technical Report") relating to the Goldfields property. I visited the Goldfields - Athona property on August 14 to 18, 2006.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is completion of a NI 43-101 compliant technical report for GLR Resources Inc. on the Athona property in 2006.

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.

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Dated this 6th Day of October, 2011.

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Signature of Qualified Person

Tim Maunula

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2. I graduated with a degree in B.Sc. Geology, Specialization from Concordia University in 1989.

3. I am a member in good standing of the Association of Geoscientists of Ontario (Registration No. 1592) and the Association of Professional Engineers and Geoscientists of Saskatchewan (Registration No. 10665).

4. I have worked as a geologist for a total of 21 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 sections 6.1 - 6.8, 7 - 12, 14 and 27 of the technical report titled NI 43-101 Technical Report, Pre-feasibility Study, Brigus Gold Corporation, Goldfields Project, Saskatchewan, Canada and dated October 6th, 2011 (the "Technical Report") relating to the Goldfields property. I visited the Goldfields property on May 11 and 12, 2011.

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.

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2. I graduated with a degree in Bachelor of Engineering in Agricultural and Bioresource engineering from the University of Saskatchewan in 1996.

3. I am a member of the Association of Professional Engineers & Geoscientists of Saskatchewan - Member #9583.

4. I have worked as an engineer for a total of 15 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 sections 1 – 5, 6.9 – 6.10, and 18 - 28 of the technical report titled NI 43-101 Technical Report, Pre-feasibility Study, Brigus Gold Corporation, Goldfields Project, Saskatchewan, Canada and dated October 6th, 2011 (the "Technical Report") relating to the Goldfields property. I visited the Goldfields property on March 18th, 2010 and September 10th to 13th, 2010.

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.

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