BACTECH ENVIROMENTAL CORPORATION

NI43-101 PRELIMINARY ECONOMIC ASSESSMENT STUDY FOR THE BACTECH SNOW LAKE RECLAMATION PROJECT SNOW LAKE, MANITOBA, CANADA

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Project Report #: 1204

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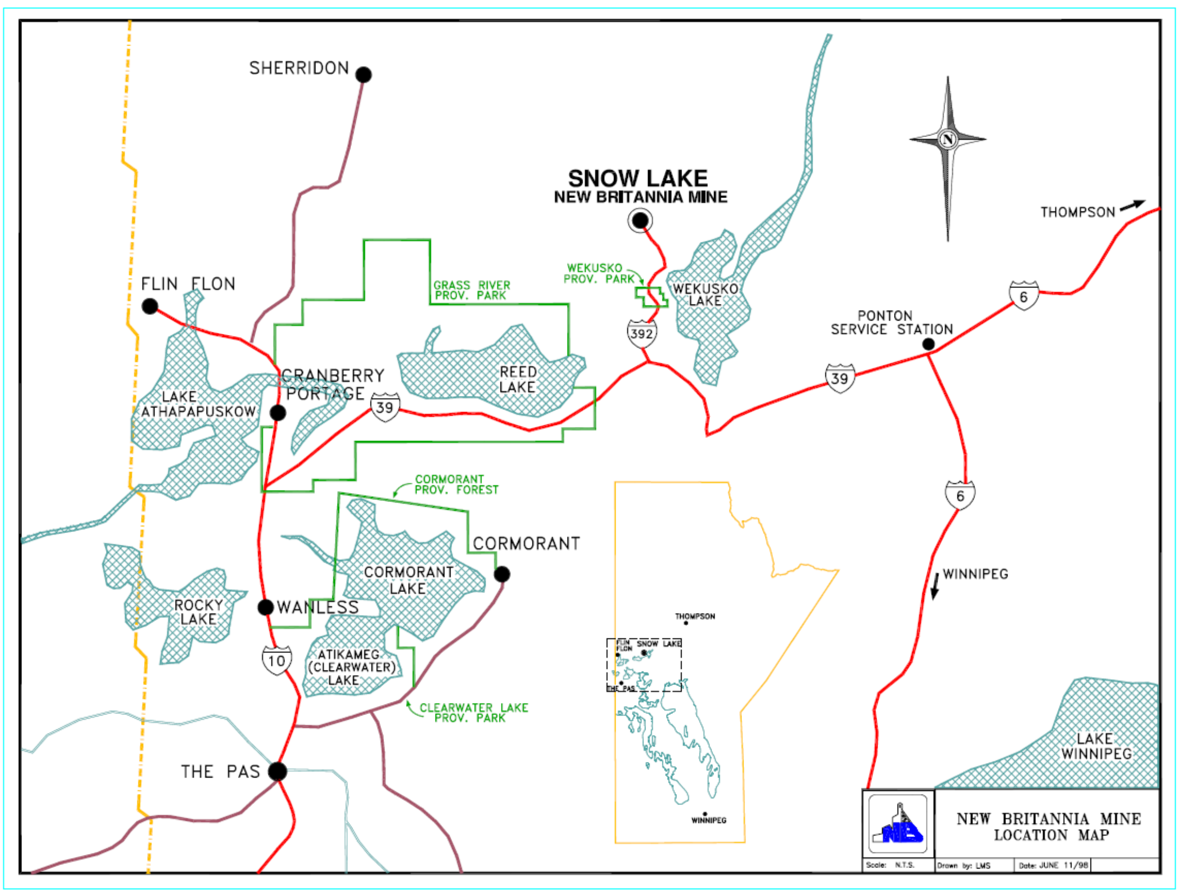
1.0 SUMMARY

BacTech Environmental Corporation (BacTech) was granted the rights to remediate a gold-bearing arsenical stockpile on April 25, 2011 by Manitoba Innovation, Energy and Mines. BacTech will use their licensed bioleaching technology to stabilize the arsenic contained therein, and will retain the recovered gold as payment. This Preliminary Economic Assessment (PEA) report is prepared to the standards of NI 43-101.

The gold residue stockpile ("GRS") occupies an area of approximately 180 m by 105 m within the community of Snow Lake, Manitoba and is shown in Figure 1.1. The town of Snow Lake is situated approximately 685 km north of Winnipeg, Manitoba. The GRS is on the Snow Lake Mine site (formerly the New Britannia Mine and the Nor-Acme Mine) now owned by QMX Gold Corporation, (QMX, formerly Alexis Minerals Corporation). In 2004 the ownership of the residue was transferred to the Province of Manitoba. QMX currently holds the surface rights and the mineral rights underlying the GRS.

Figure 1.1
Location of the Snow Lake Property, Manitoba



In May 2011, BacTech carried out a 33-hole drill program on the stockpile at a 20 m grid spacing. One half of the recovered core has been shipped to Inspectorate Exploration &

Mining Services Ltd., Metallurgical Division ("Inspectorate") in Vancouver for bioleach test work. Samples were split off from the remaining half and sent to the Saskatchewan Research Council in Saskatoon ("SRC") for analysis for gold and silver, the results of which were used to estimate a Mineral Resource for the property.

A Measured Mineral Resource of 265,000 tonnes grading 9.7 grams per tonne ("g/t") gold and 2.17 g/t silver was estimated for the deposit. In addition, an Indicated Mineral Resource of 9,300 tonnes grading 9.2 g/t gold and 2.15 g/t silver and an Inferred Mineral Resource of 28,000 tonnes grading 7.0 g/t gold and 2.4 g/t silver was estimated. This Mineral Resource Estimate was disclosed in a NI 43-101 Technical Report entitled "Gold Residue Stockpile, Snow Lake, Manitoba, Mineral Resource Estimation" by N Ralph Newson P.Geo., dated July 20, 2011.

1.1 PROPERTY DESCRIPTION AND LOCATION

The subject property (GRS) consists of refractory mill tailings with a high residual gold content which were impounded in a special enclosure separate from other tailings, with the hope that technological advances would eventually permit the residual gold to be recovered. The area of the GRS is about 180 m by 105 m.

The property is in west-central Manitoba at 54°53'16" north latitude and 100°01'20" west longitude within the community of Snow Lake, on the Snow Lake Mine site.

An agreement to have access over the Snow Lake Mine property of QMX is required to exploit the property.

BacTech has been given the right by the Manitoba Government, the owner of the tailings, to remediate the tailings by converting the arsenic in the stockpile to an inert and therefore environmentally safe form and to retain any gold recovered during the process. The stockpiled tailings are situated on a mining lease controlled by QMX and any subsequent deal must be accompanied by QMX's permission to access the site.

1.2 ROYALTIES

A royalty of CAD$5/oz gold is payable to the Town of Snow Lake in respect of production from the GRS. In addition, a 2% net smelter return royalty (NSR) is due to the province of Manitoba once initial capital has been repaid. Both of these royalties are provided for in the cash flow model.

1.3 ENVIRONMENTAL ISSUES

Due to its high arsenic content, the property is presently an environmental liability to its owner, the Manitoba Government. This liability is not transferred to BacTech. All environmental liability is the responsibility of the Province of Manitoba.

1.4 PERMITS

None of the environmental or permitting issues identified in Micon's review of the Snow Lake Project would appear to be insurmountable or a fatal flaw for the project. High Arsenic content is the major environmental issue currently being dealt with at the site and will need to be addressed before permits are issued to reopen the facility.

All required permits were in place during active mining operations of the New Britannia Mine under TVX Gold, and subsequently under Kinross Gold and many of the required permits are in place today for the Project's current care and maintenance status. Most permits will need to be updated, modified, amended or replaced prior to the mine resuming operations to reflect any changes anticipated.

Permits required include those under the Manitoba Mine Closure regulation 67/99, the Contaminated Sites Remediation Act C205, The Workplace Safety and Health Act W210, the Federal Fisheries Act, The Metal Mining Effluent Regulations, The Dangerous Goods Handling and Transportation Act, and regulations of Manitoba Conservation. To date none of the permits have been obtained.

The authors are not aware of any other significant factors and risks that may affect access, or the right or ability to perform work on the property.

1.5 LOCAL RESOURCES AND INFRASTRUCTURE

There is an available local source of experienced and qualified labour, chemical and materials suppliers, mining and drilling contractors, and mining industry service companies.

Electrical power to the property is via existing 64 kV power lines and a substation on the property. Process water is available from a well located approximately 3.2 km east of the plant site.

1.6 PROPERTY HISTORY AND HISTORIC RESOURCE ESTIMATES

The GRS consists of a cyanide-treated, refractory arsenopyrite gold concentrate generated from 1949 to 1958 by the Nor-Acme Mine, in Snow Lake, Manitoba. The material was piped into a waste rock impoundment, measuring 185 m long and 105 m wide, constructed north of the original mill. The principle commodity extracted at Nor-Acme Mine was gold, which existed as free gold and refractory "invisible" gold bound in arsenopyrite grains. (Salzsauler et al. 2005).

The site was operated from 1949 to 1958 as the Nor-Acme Mine under the supervision of Howe Sound Exploration Company, and its subsidiary, the Britannia Mining and Smelting Company Limited. Following closure, further exploration activities were conducted by various companies from 1959 to 1987. The Nor-Acme mine reopened as the New Britannia Mine ("NBM") in 1995 as a joint venture between High River Gold Mines Limited and TVX

Gold Limited (now Kinross Gold Corporation), and closed in late 2004. Upon the re-opening of NBM, the stockpile was declared to be an orphaned site and to be the responsibility of the Crown. Surface drainage from the pile seeped to the north, towards Snow Lake, which is the main recreational and drinking water source for the Town of Snow Lake. NBM financed the emplacement of a multilayer cap of waste rock and silt on the pile in the late 1990s as an attempt to reduce surface runoff (DNE Knight Piesold, 1999; Salzsauler et al., 2005). QMX currently holds the mineral and surface rights underlying the property.

In 2008, the Province retained AECOM Canada Ltd. to provide project management services and to oversee the detailed design and implementation of remedial solutions that would address environmental and human health concerns associated with the site.

In April, 2010, BacTech approached the provincial government with a proposal to test the concentrate, at no cost to the government, using bioleaching to stabilize the arsenic and keeping the contained gold for BacTech's account.

1.7 GEOLOGY AND MINERALIZATION

The GRS is located in the Paleoproterozoic Flin Flon - Snow Lake volcano-sedimentary greenstone belt, which is part the Churchill province of the southern Reindeer Terrane of the Trans Hudson Orogen (Richardson and Ostry, 1996; Fulton, 1999). The rocks of the belt are well documented because of the mineralization potential for copper, zinc and gold.

The Nor-Acme mine process resulted in total gold recovery by flotation and cyanidation of approximately 83%. (Convey et al, 1957). The filtered cake of refractory processed material contained up to 10 grams of gold per tonne. Refractory gold was not recoverable by conventional techniques at that time.

The refractory "cake" was impounded in a waste rock berm for possible future re-treatment. The base of the GRS is sitting partly on bedrock and partly on clay.

The GRS consists of high sulphide mine waste containing around 60% sulphides including up to 55% arsenopyrite with minor pyrite and pyrrhotite.

1.8 EXPLORATION AND ADJACENT PROPERTIES

In April, 2010, BacTech obtained samples of the GRS from Dr. Barbara Sherriff of the University of Manitoba. The samples were from four drill cores taken from the GRS in March, 2002 as part of a study to investigate the mobility and migration of arsenic in the GRS (Salzsauler et al., 2005). The drill holes were approximately evenly spaced along the long axis of the GRS.

Half of the core from each drill hole was taken to generate four composite samples. The samples were then transferred to the mineral processing laboratory in the Mining Engineering Department of Queen's University, Kingston, Ontario, for sample preparation and preliminary analysis under the supervision of Dr. Wan Tai Yen. These samples were then sent to Inspectorate's IPL Analytical Division in Vancouver for further analytical work to investigate the possibility of commercial production of the GRS using BacTech's BACOX bioleaching process. Inspectorate's IPL Analytical Division in Vancouver is an ISO 9001 accredited laboratory.

Results of this work were described by BacTech as "encouraging" in a press release dated February 15, 2010. This led management to take the next step of determining a Mineral Resource for the deposit.

In September, 2010 BacTech presented the positive results of its test to the Government of Manitoba and has followed up with a proposal to conduct a larger metallurgical/bioleach study. The remediation and processing was awarded to BacTech by the Province of Manitoba, and BacTech was granted permission to test the stockpile in April, 2011 by the Government of Manitoba.

Thirty-three vertical holes were drilled into the GRS by BacTech in May, 2011. Since the gold residue was deposited as a slurry, the primary stratification is horizontal. The sample intervals therefore represent the true thicknesses of any layering that may be present. The drill used a vibrating drill stem designed for the recovery of unconsolidated material.

1.9 MINERAL PROCESSING AND METALLURGICAL TESTING

Snow Lake Project testwork was conducted using BACOX technology to treat the Gold Residue Stockpile ("GRS") material, which is a highly refractory sulphide material with a high arsenic content.

The key findings of the metallurgical testing programs were very positive in providing BacTech with an evaluation into the refractory nature of the GRS material and its suitability to bio-oxidation processing. Diagnostic leaching confirmed that the majority of the gold present in the GRS material was refractory and high oxidation levels will be required to maximize extraction by cyanidation. A maximum gold recovery of 96.5% was obtained after hot nitric acid digestion followed by cyanidation. This extraction compares to a gold extraction of only 9.4% for direct carbon-in leach (CIL) cyanidation.

1.10 MINING AND RECLAMATION METHODS

The reclamation of the deposit material can be performed with minimal supervision and by conventional earthmoving equipment with the use of local contractor. The low variability in grade enables the reclamation to be performed at any location close to the processing facilities or access road. This allows for a potential increase in production rate with multiple working faces with an increase in the equipment fleet.

The challenges will be working during winter or wetter months and the stability of the excavated front during operation when there is a "highwall". Provision of covering blankets or alternative temporary shelter will be required at the working face, temporary storage or stockpile areas to prevent freezing of the material. The material can be excavated in benches to increase stability or reduce the occurrence of "highwall" during mining of thick deposit areas. Provision will have to be accounted for portable heating systems and dewatering pumps at the face.

An increase the sampling frequency for Inferred Resource material is recommended so that it can be developed into a higher geological classification and possibly improve the overall resource tonnage.

1.11 PROCESS OVERVIEW

The Gold Residue Stockpile (GRS) is currently capped with coarse angular mine waste, which will be removed to expose the residue material in sections minimizing the exposure to the various weather elements. The reclaimed GRS material will be stored in a weather proof structure to prevent exposure of the material to wind, rain, and snow.

The Process Facility equipment will consist mainly of two grinding mills, tanks, agitators, piping, pumps, and electrical components, all of which will make-up the following 5 circuits;

1. Material Preparation Circuit (Area 10).
2. Bioleach Circuit (Area 20).
3. Residue Liquor Separation Circuit (Area 30).
4. Neutralization Circuit (Area 40).
5. Reagent Circuit (Area 50).

Material will be fed from the bioleach facility stockpile into feed hopper at the Material Preparation Circuit at an approximate rate of 100-125 t/d by a front end loader.

1.12 PROJECT INFRASTRUCTURE

The base case total power requirement is 612 kWh/t which results in a maximum demand of approximately 3.3 MW for this 107 t/d operation. Power will be supplied by connecting to the national high voltage electricity grid. Electrical power to the property is currently supplied by 64 kV power lines and a substation on the property.

Make-up water which satisfy project requirements are available directly from Snow Lake following approval from the Manitoba Government.

The site will accommodate the construction of a process plant, with possible auxiliary out-building on the site to house the technical department offering a total available space of 615 m^2, laboratory, a maintenance shop for equipment, a heated warehouse, a gatehouse, and a covered concentrate storage facility.

Water tanks for industrial use will be installed, including one for fire suppression water. The fire suppression water will be distributed to the protected area through an underground water pipe network.

A fuel tank with an adjacent fuelling station will primarily supply mining equipment needs.

Sewage services for the facility will be tied into the municipal services for treatment at the town's sewage treatment plant.

Communication facilities will be comprised of a redundant fibre communication backbone system which will link and manage the data transmission of the distributed control system (DCS), third party programmable logic controllers (PLCs), motor controls, fire detection system, and computers around the mine site. It is expected that the site will be connected to the local phone system network (MTS) which is complemented by a 2G cellular service; a satellite internet service will be established as well. In the case that this would not be possible, then a Voice over Internet Protocol (VoIP) telephone system would be required. In any case, a transceiver or cellular radio tower will be installed in order to optimize the utilization of cellular telephones, as the current coverage in the area is not adequate.

At this stage of project development, there is a high degree of flexibility in the siting of the process plant and other surface infrastructure. Micon has not, therefore, made specific recommendations as to location of these works pending discussion with affected landowners. Progress is on-going between the town of Snow Lake and BacTech to acquire teh crown land located on Cedar Avenue.

1.13 CAPITAL AND OPERATING COSTS

1.13.1 Capital Costs

Capital expenditures and capitalized development costs for the base case total CAD$21.4M. All costs associated with loading and transporting the GRS material to the plant are covered by the proposed contractor's operating costs.

The capital estimate for the processing plant totalled CAD$10.0M. Ongoing maintenance is covered by operating costs, so there is no sustaining capital forecast.

The indirect capital cost estimate is CAD$11.4M. Indirect costs include EPCM costs, estimated to be 11% of the direct capital cost estimate for the process plant. A provision equivalent to 20% of direct costs is also made for the temporary hire of construction equipment (e.g., mobile cranes) and general site costs.

Owner's costs include first fills of reagents and consumables in the plant, construction insurance, commissioning, recruitment/training costs and owner's site costs (including site management and supervision).

Environmental bonding costs are estimated to be $1.25 million, reflected in the cash flow as an up-front cost incurred prior to production start-up.

A total contingency of $2.82 million equates to approximately 15% of the initial capital, including indirect costs.

1.13.2 Operating Costs

Operating costs for transport of the GRS material assume a contract rate for loading and hauling of $3.70/t. Provision is also made for feeding GRS material into the processing plant, using the project owner's own equipment, for a total operating cost of $7.91/t of GRS feed.

Processing costs of $158.78/t include a total of $57.42/t for labour, $75.08/t in reagents, spares and consumables and $26.28/t in electrical power. The latter assumes a power cost of $0.041/kWh, including maximum demand charges.

General and Administration costs include site management, mobile equipment, office running costs, environmental management and insurance costs.

Estimated cash operating costs over the life of the project are summarized in Table 1.1.

Table 1.1
Summary of Life-of-Mine Operating Costs

Area	Life-of-mine Cost ($ 000)	Unit Cost $/t ore treated
Contract Load & Haul	1,119	3.70
Plant Feed (front-end loader)	1,274	4.21
Sub-total Plant Feed	**2,393**	**7.91**
Labour - Metallurgy	2,037	6.74
- Laboratory	972	3.22
- Production	10,739	35.52
- Maintenance	3,610	11.94
Maintenance	4,114	13.61
Reagents	18,583	61.47
Power	7,945	26.28
Sub-total Processing	**47,999**	**158.78**
Labour	1,740	5.76
Mobile Equipment Operation	124	0.41
G&A (other)	2,700	8.93
Sub-total General and Administrative	**4,564**	**15.10**
Total Operating Costs	**1,021,431**	**181.79**

1.14 ECONOMIC ANALYSIS

The objective of the study was to determine the viability of the proposed process plant to rehabilitate the GRS repository. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

Given that the mineral resource estimate for the GRS carries less uncertainty than is seen in many mining projects, Micon considers a beta value of 1.0 (i.e., equal to the average for the equity market as a whole) to be appropriate for the base case. Sensitivity of the project is tested using a range from 0.7 (typical for some gold producers) to 1.3 (for the mining industry in general). Thus, CAPM gives an estimated cost of equity for the Project of between 5% and 9%. Micon has taken a figure of 7% (i.e., in the middle of this range) as its base case, and provides the results at alternative rates of discount for comparative purposes.

At the end of July, 2012, the three-year trailing averages for each metal were US$1,396/oz gold and US$26.67/oz silver, and these metal prices were selected for the base case. These prices were applied consistently throughout the operating period. Silver contributes approximately 0.4% of the projected total revenue for the base case, so the impact of changing the silver price forecast is minimal.

A royalty of $CAD5/oz gold is payable to the Town of Snow Lake in respect of production from the GRS. In addition, 2% of NSR value is due to the Province of Manitoba once initial capital been repaid. Both these royalties are provided for in the cash flow model.

Sales revenue derived from the product shipped is based on a yield (i.e., gold recovery by the purchaser) of 88.6%, and payability factor of 99%. Transport and treatment charges of $25/wet metric tonne and $125/dry metric tonne, respectively, are applied to concentrate material shipped.

Direct operating costs average $181.79/t milled over the LOM period, comprised of $7.91/t GRS reclamation, $158.78/t processing, and $15.10/t general and administrative costs.

Pre-production capital expenditures are estimated to total US$21.4 million, including $10.0 million for plant feed and processing, $4.2 million indirect costs, $4.4 million in owner's costs and contingencies totalling $2.8 million.

Working capital has been estimated to include 15 days product inventory, and 15 days receivables from despatch of concentrate. Stores provision is for 60 days of consumables and spares inventory, less 30 days accounts payable. An average of $1.2 million of working capital is required over the LOM period.

1.14.1 Base Case Cash Flow

The LOM base case project cash flow is presented in Table 1.2.

Table 1.2
Life-of-Mine Cash Flow Summary

	CAD 000	CAD/t	US$/oz Au
Gross Revenue (Gold)	114,411	378.47	1,396.00
Operating Costs			
Mining costs	2,393	7.91	29.19
Processing costs	47,999	158.78	585.67
General & Administrative costs	4,564	15.10	55.69
Direct operating cost	54,956	181.79	670.55
TC/RC (Gold)	24,272	80.29	296.16
less NSR (By-products)	(386)	(1.28)	(4.71)
Cash operating cost	78,842	260.81	961.99
Royalty	879	2.91	10.73
Total cash cost	79,721	263.71	972.72
Net Operating Margin	34,691	114.76	423.28
Capital Expenditure	21,356	70.64	260.57
Pre-tax Cash Flow	13,335	44.11	162.71
Taxation	3,600	11.91	43.93
Net Cash Flow After Tax	9,734	32.20	118.78

On the pre-tax, undiscounted cash flow, payback occurs at 4.9 years.

Payback on the undiscounted cash flow occurs at the end of Year 5, leaving approximately 3 years of the LOM period remaining. On a discounted basis, payback occurs at 7.2 years, less than one year before the end of the life of mine period.

The base case evaluates to an IRR of 11.1% before taxes and 8.9% after tax. At a discount rate of 7.0%, the net present value (NPV_7) of the cash flow is $3.9 million before tax and $1.6 million after tax.

1.15 RECOMMENDATION

Micon concludes that, with a continuation of the trend towards higher gold prices over the past five years, this study demonstrates the potential viability of the project as proposed, and that further development is recommended.

The proposed budget for further project development is presented below:

Process Optimizsation Testwork	$75,000	Q4 2012
EAP-Golder Study and Application Process	$65,000	Q4 2012
Geotechnical Report	$25,000	Q4 2012
Process Optimisation Testwork – Phase 2	$80,000	Q2 2013
Closure Plan	$65,000	Q2 2013
Front End Engineering Design (FEED)	$300,000	Q2 2013

2.0 INTRODUCTION

At the request of BacTech Environmental Corporation (BacTech), Micon International Limited (Micon) has prepared a Preliminary Economic Assessment (PEA) of the Snow Lake Project, which is located in Snow Lake Manitoba, Canada. This Technical Report, prepared in accordance with the reporting standards and definitions required under Canadian National Instrument (NI) 43-101, summarizes the results of that study. This PEA is based upon an estimate of the Snow Lake Gold Residue Stockpile (GRS) deposit mineral resources originally prepared by Newson in July, 2011. The GRS is a stockpile of arsenopyrite-rich tailings remaining at the Snow Lake Mine site (formerly the New Britannia Mine) now owned by QMX Gold Corporation, (QMX, formerly Alexis Minerals Corporation). In 2004 ownership of the GRS was transferred to the Province of Manitoba.

The scope of the proposed project includes the reclamation of a GRS deposit to extract the gold-bearing material that comprises the resource by treating that material using a bacterial leach process to produce a stable arsenic compound in the tailings and recover the contained gold and silver. At the proposed scale of operation of 107 t/d, the project has a projected life of about 7 years.

The independent Technical Report for the GRS mineral resource estimate is dated July 20, 2011 and was filed on SEDAR on December 9, 2011 (Newson, 2011). Since the completion of Newson's mineral resource estimate, no further exploration work has been done.

BacTech acquired the Snow Lake GRS project on April 25, 2011 and retained Micon to prepare a PEA report.

The geological setting of the property, mineralization style and occurrences, and exploration history were described in reports that were prepared by Newson (2011). The relevant sections of this report are reproduced herein.

For the PEA, author B. Damjanovic visited the property on January 16-18, 2012, and R. Newson visited the property on May 11-13, 2011. In addition to an inspection of the surface deposit, the author (Damjanović) visited the surface of the potential project site and potential tailings storage sites in the district, and core storage/logging facility and offices in Snow Lake, Manitoba.

Messrs. Newson, Foo, Damjanović, and Jacobs are all Qualified Persons (QP) as defined in NI 43-101.

The Mineral Resource Estimate is based on exploration results and interpretation current as of July 20, 2011. The PEA has an effective date of August 27, 2012.

The conclusions and recommendations in this report reflect the authors' best judgment in light of the information available to them at the time of writing. The authors and Micon International Limited (Micon) reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to them subsequent to the date of this report. Use of this report acknowledges acceptance of the foregoing conditions.

This report is intended to be used by BacTech Enviromental Corporation Inc. (BacTech) subject to the terms and conditions of its agreement with Micon. That agreement permits BacTech to file this report as a National Instrument 43-101 Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report, by any third party, is at that party's sole risk.

Table 2.1
List of Abbreviations

Abbreviation	Unit or Term
'	minutes of longitude or latitude
~	approximately
%	percent
<	less than
>	greater than
º	degrees of longitude, latitude, compass bearing or gradient
ºC	degrees Celsius
2D	two-dimensional
3D	three-dimensional
µm	microns, micrometres
Ag	silver
As	arsenic
Au	gold
CDN$	Canadian dollar(s)
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre(s)
Cu	Copper
d	Day
DETI	Department of Enterprise, Trade and Investment
E	east
et al.	and others
EM	electromagnetic, usually in reference to and EM geophysical survey
EPCM	Engineering, Procurement and Construction Management
ES	Environmental Statement
FA	fire assay
ft	foot, feet
g/t	grams per tonne
g/t Au	grams per tonne of gold
GPS	global positioning system
GRS	gold residue stockpile
h, h/d	hour(s), hours/day
ha	hectare(s)
HMS	Heavy Media Separation
HP	Horsepower
HQ	H-sized core, Longyear Q-series drilling system
ICP	inductively coupled plasma
ICP-AES	inductively coupled plasma-atomic emission spectrometry

Abbreviation	Unit or Term
ID	Inverse distance grade interpolation
in	inch(es)
IP	induced polarization (geophysical survey)
kg	kilogram(s)
km, km^2	kilometre(s), square kilometre(s)
kW, kWh, kWh/t	Kilowatt, kilowatt hours, kilowatt hours per tonne
L	litre(s)
lb	pound(s)
LOM	Life of Mine
m	metre(s)
m^3	cubic metre(s)
m/s	metres per second
M	million(s)
Ma	million years
masl	metres above sea level
mg	milligram
mm	millimetre(s)
mL	millilitre(s)
Mt	million tonnes
Mt/y	million tonnes per year
MW	Megawatt
N	north
n.a.	not applicable, not available
Na	sodium
NaCN	Sodium cyanide
NI 43-101	Canadian National Instrument 43-101
NPV, NPV_n	Net Present Value, Net Present $Value_{(annual\ discount\ rate)}$
NQ	N-sized core, Longyear Q-series drilling system
NSR	Net smelter return (royalty)
OK	ordinary kriging grade interpolation
oz	troy ounce(s)
oz/ton	troy ounces per short ton
PAG	Potentially Acid Generating (waste rock)
Pb	lead
PEA	Preliminary Economic Assessment
pH	concentration of hydrogen ion (-log_{10} of)
ppb	parts per billion
ppm	parts per million, equal to grams per tonne (g/t)
QA/QC	quality assurance/quality control
QEMSCAN	Quantitative Evaluation of Minerals by SCANning electron microscopy
QP	qualified person
RC	reverse circulation
ROM	Run of Mine
RQD	rock quality designation (data)
s	second
S	south
Sb	antimony
SD	standard deviation
SG	specific gravity
SI	International System of Units
t	tonne(s) (metric)

Abbreviation	Unit or Term
t/h	tonnes per hour
t/d	tonnes per day
t/m^3	tonnes per cubic metre
t/y	tonnes per year
ton, T	short ton (2,000 lbs)
UG	Underground
UK	United Kingdom of Great Britain and Northern Ireland
US	United States of America
US$	United States dollar(s)
US$/oz	United States dollars per ounce
US$/t	United States dollars per tonne
VLF-EM	very low frequency - electromagnetic geophysical surveys
W	west or Watt
wt %	percent by weight
y	year
yd, yd^3	yard, cubic yard
Zn	zinc

3.0 RELIANCE ON OTHER EXPERTS

Micon has reviewed and analyzed exploration data provided by BacTech, and its consultants, and has drawn its own conclusions therefrom, augmented by its direct field examination. Newson has carried out independent exploration work, and drilled holes to carry out a program of sampling and assaying on this project.

Large scale mining of precious metals has taken place in the immediate area, and there is confirmation that a GRS deposit high in arsenic and gold grades was produced as a by-product during the operations of the Nor-Acme Mine.

While exercising all reasonable diligence in checking, confirming and testing it, Micon has relied upon the data presented by BacTech, in formulating its opinion.

The various agreements under which BacTech holds title to the GRS deposit for this project have not been thoroughly investigated nor confirmed by Micon and Micon offers no opinion as to the validity of the mineral title claimed.

The description of the property is presented here for general information purposes only, as required by NI 43-101. Micon is not qualified to provide professional opinion on issues related to mining and exploration title or land tenure, royalties, permitting and legal and environmental matters. Accordingly, the authors have relied upon the representations of the issuer, BacTech, for this report, and have not verified the information presented therein.

Micon has relied on the expertise of Golder in its reporting of social, environmental and permitting issues.

The conclusions and recommendations in this report reflect the authors' best judgment in light of the information available at the time of writing. Micon reserves the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to it subsequent to the date of this report. Use of this report acknowledges acceptance of the foregoing conditions.

Those portions of the report that relate to the location, property description, infrastructure, history, deposit types, exploration, drilling, sampling and assaying (Sections 4 to 11) are taken, at least in part, from previous Technical Reports prepared by others as well as updated information provided by BacTech.

4.0 PROPERTY DESCRIPTION AND LOCATION

The subject property contains an asset, the GRS, which consists of refractory mill tailings with a high residual gold content which were impounded in a special enclosure separate from other tailings with the hope that technological advances would eventually permit the residual gold to be recovered. The area of the GRS is about 180 m by 105 m. The GRS is located on the Snow Lake Mine property, on a mining lease which belongs to QMX Gold Corporation, ("QMX", formerly Alexis Minerals Corporation).

The property is in west-central Manitoba at 54°53'16" north latitude and 100°01'20" west longitude within the community of Snow Lake, on the Snow Lake Mine site (see Figures 4.1 and 4.2).

An agreement to have access over the Snow Lake Mine property is required to exploit the asset. BacTech controls no mining claims or leases at the project site.

BacTech has entered into an agreement with the Manitoba Government, the owner of the tailings, to remediate the ARS material by converting the arsenic in the stockpile to an inert, and therefore environmentally safe, form, and to keep any gold they can recover in payment for doing this. BacTech is currently in discussios with QMX to implement an Indemnification Agreement for access to the ARS..

Due to its high arsenic content, the GRS is presently an environmental liability to its owner, the Manitoba Government. This liability is not transferred to BacTech. All environmental liability is the responsibility of the Province of Manitoba.

Permits required include those under the Manitoba Mine Closure regulation 67/99, the Contaminated Sites Remediation Act C205, The Workplace Safety and Health Act W210, the Federal Fisheries Act, the Metal Mining Effluent Regulations, The Dangerous Goods Handling and Transportation Act, and regulations of Manitoba Conservation. To date none of the permits have been obtained.

The authors are not aware of any other significant factors and risks that may affect access, or the right or ability to perform work on the property.

Figure 4.1
Location Map Showing the Study Site Location in Manitoba

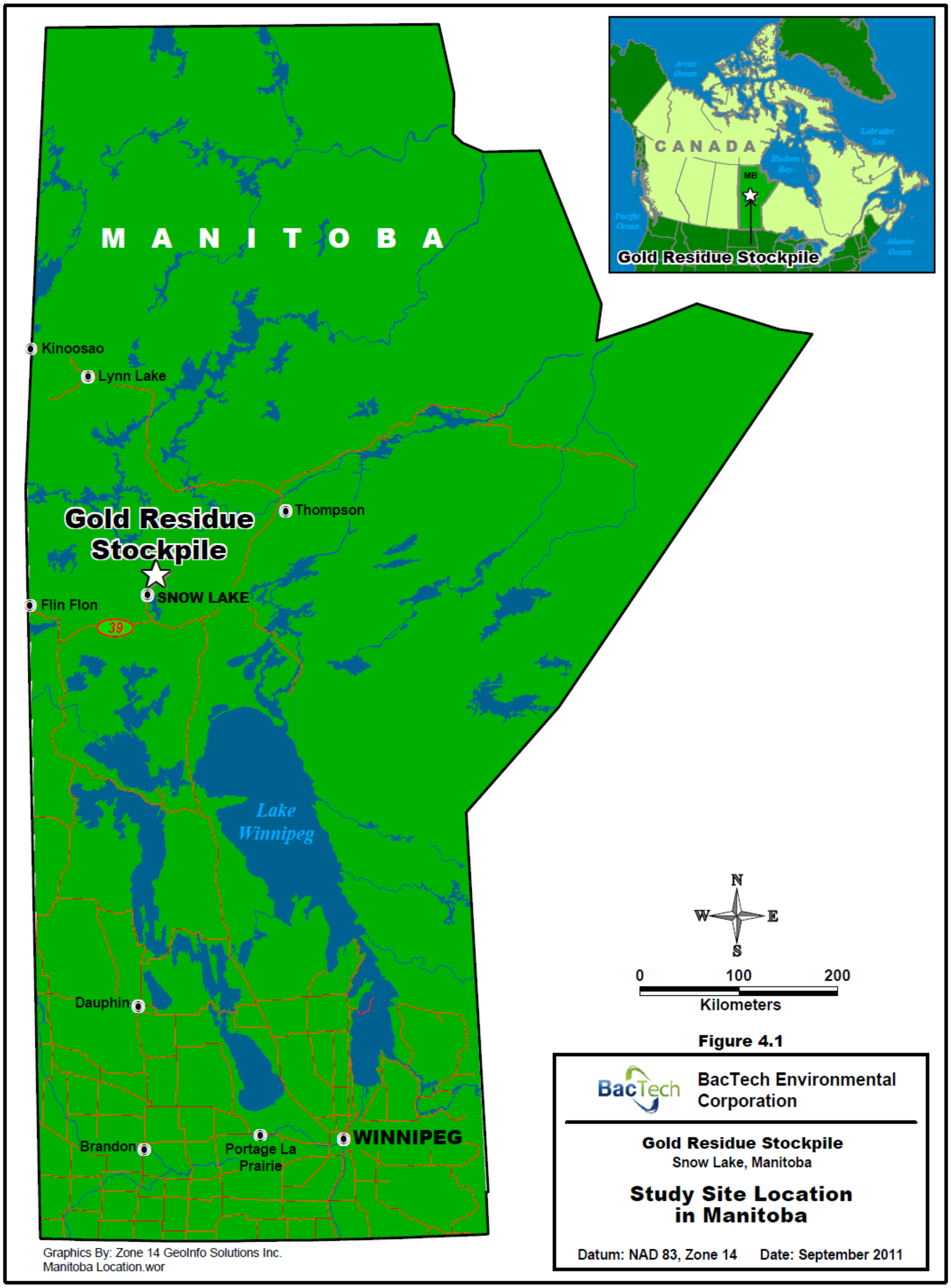
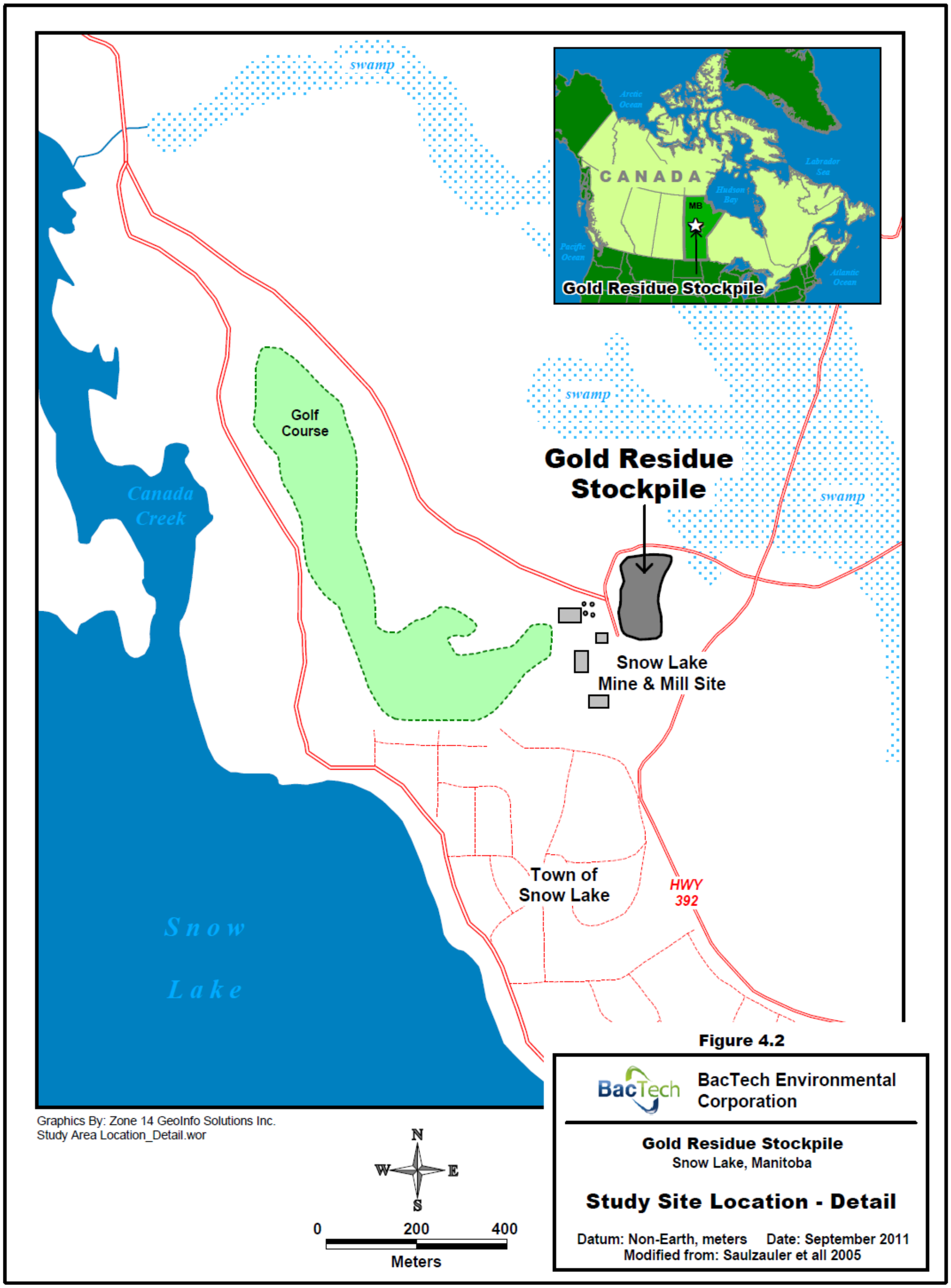


Figure 4.2
Project Site Detail



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The topography of the region is typical of the Canadian Shield, with low rounded outcrops surrounded by swamps and boreal forest, and generally low topographic relief. The GRS sits on fairly flat, level, prepared ground in the Snow Lake Mine site.

The centre of the GRS is about 350 m from a paved street of the Town of Snow Lake. From that street it is accessed over the Snow Lake Mine property of QMX. The entire periphery of the GRS is accessible by car or light truck, and the top is accessible by light truck. The streets of the Town of Snow Lake connect to paved Provincial highway number 392, which is part of the provincial road system.

The Town of Snow Lake is situated approximately 685 km north of Winnipeg, Manitoba’s provincial capital. Snow Lake lies at the geographical centre of Manitoba's three northern cities of Flin Flon (about 180 km), The Pas (about 200 km) and Thompson (about 240 km), and is easily accessible by paved highways.

Snow Lake has an airstrip, but scheduled service is infrequent. The authors flew to Flin Flon and drove from there to carry out the site visit.

The climate in Snow Lake is classified as “continental subarctic” with short warm summers, and long cold winters. Historical extreme temperatures recorded at nearby weather stations are as low as -46ºC in the winter and up to +40ºC in the short summer (Sikamen Resources, 1988). The average date for late spring frost in the region was reported to be May 31 in the late 1980’s, and the average date of the first fall frost was September 20. In general, the average frost-free period was 119 days (Sikamen Resources, 1988). Annual precipitation varies from 300 to 500 mm. With the infrastructure present, any operation that might be established on the property should be able to continue year round.

Subject to an agreement with QMX, surface access for mining operations is adequate. The property and the likely locations for a processing plant are all on the provincial power grid. Water is abundant in the area, but specific arrangements for a supply have not been made yet.

With a rich history in mining, the town of Snow Lake is home to a vibrant business community catering to the many needs of mineral related projects. Mining has been the mainstay of the community. Currently the Chisel North Mine and Concentrator are operating and HudBay Minerals is in the construction stages of developing a new zinc gold mine at Lalor. QMX is in the process of reopening the Snow Lake Mine. Experienced mining personnel should be readily available in the area.

Snow Lake’s estimated population is 915 according to the town’s website, www.snowlake.com.

Once the GRS has been processed, the neutralised precipitate will be pumped to a storage impoundment area approximately 1.4 km northeast of teh bioleach facility. The storage impoundment will be clay and HDPE lined to prevent any seepage into the environment. There will be 2 HDPE pipelines used to transport the neutralized precipitate to the impoundment area (1 pipeline will be in operation and 1 pipeline will be on stand-by). A third pipeline will be required to transfer reclaimed water from the Ferric Arsenate impoundment back to the facility to be utilised in the process.

6.0 HISTORY

The GRS consists of cyanide-treated, refractory arsenopyrite gold concentrate generated from 1949 to 1958 by the Nor-Acme Mine, in Snow Lake, Manitoba. The material was piped into a waste rock impoundment, measuring 185 m long and 105 m wide, constructed north of the original mill. The principle commodity extracted at Nor-Acme Mine was gold, which existed as free gold and refractory “invisible” gold bound in arsenopyrite grains. (Salzsauler et al. 2005)

The site was operated from 1949 to 1958 as the Nor-Acme Mine under the supervision of Howe Sound Exploration Company, and its subsidiary, the Britannia Mining and Smelting Company Limited. Following closure, further exploration activities were conducted by various companies from 1959 to 1987. The Nor-Acme mine reopened as the New Britannia Mine (NBM) in 1995 as a joint venture between High River Gold Mines Limited and TVX Gold Limited (now Kinross Gold Corporation), and closed in late 2004. Upon the re-opening of NBM, the stockpile was declared to be an orphaned site, and to be the responsibility of the Crown. Surface drainage from the pile seeped generally to the north, towards Snow Lake, which is the main recreational and drinking water source for the Town of Snow Lake. NBM financed the emplacement of a multilayer cap of waste rock and silt on the pile in the late 1990’s as an attempt to reduce surface runoff (DNE Knight Piesold, 1999; Salzsauler et al., 2005). QMX currently holds the mineral and surface rights underlying the property.

In 2008, the Province retained AECOM Canada Ltd. to provide project management services and to oversee the detailed design and implementation of remedial solutions that would address environmental and human health concerns associated with the site.

In April, 2010 BacTech approached the provincial government with a proposal to test the concentrate, at no cost to the government, using bioleaching for neutralizing the arsenic and keeping the contained gold for BacTech’s account.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The GRS is located in the Paleoproterozoic Flin Flon - Snow Lake volcano-sedimentary greenstone belt, which is part of the Churchill province of the southern Reindeer Terrane of the Trans Hudson Orogen (Richardson and Ostry, 1996; Fulton, 1999). The rocks of the belt are well documented because of the mineralization potential for copper, zinc and gold.

The Nor-Acme mine process resulted in total gold recovery by flotation and cyanidation of 83%. (Convey et al, 1957). The filtered cake of refractory, processed material contained up to 10 grams of gold per tonne. Refractory gold was not recoverable by conventional techniques at that time.

The refractory "cake" was impounded in a waste rock berm for possible future re-treatment. The base of the GRS is sitting partly on bedrock and partly on clay.

The GRS consists of high sulphide mine waste containing about 60% sulphides comprising up to 55% arsenopyrite with minor pyrite and pyrrhotite.

8.0 DEPOSIT TYPES

The authors are not aware of a systematic classification for tailings. The type of deposit might be described as "refractory" or "arsenical gold-bearing refractory sulphide".

9.0 EXPLORATION

In April, 2010 BacTech obtained samples of the GRS from Dr. Barbara Sherriff of the University of Manitoba. The samples were from four drill cores taken from the GRS in March, 2002 as part of a study to investigate the mobility and migration of arsenic in the GRS (Salzsauler et al., 2005). The drill holes were approximately evenly spaced along the long axis of the GRS.

Half of the core from each drill hole was taken to generate four composite samples. The samples were then transferred to the mineral processing laboratory in the Mining Engineering Department of Queen's University, Kingston, Ontario, for sample preparation and preliminary analysis under the supervision of Dr. Wan Tai Yen. These samples were then sent to Inspectorate's IPL Analytical Division in Vancouver for further analytical work to investigate the possibility of commercial production of the GRS using BacTech's BACOX bioleaching process. Inspectorate's IPL Analytical Division in Vancouver is an ISO 9001 accredited laboratory.

Results of this work were described by BacTech as "encouraging" in a press release dated February 15, 2010. This led management to take the next step of determining a Mineral Resource for the deposit.

In September, 2010 BacTech presented the positive results of its test to the Government of Manitoba and has followed up with a proposal to conduct a larger metallurgical/bioleach study. The remediation and processing was awarded to BacTech by the Province of Manitoba, and BacTech was granted permission to test the stockpile in April, 2011 by the Government of Manitoba.

10.0 DRILLING

Thirty-three vertical holes were drilled into the GRS by BacTech in May, 2011 (see Figures 14.1 and 14.2). Since the gold residue was deposited as a slurry, the primary stratification is horizontal. The sample intervals therefore represent the true thicknesses of any layering that may be present. The drill used a vibrating drill stem designed for the recovery of unconsolidated material. The inside diameter of the core tube was 3.5" (8.9 cm). The recovered core was placed in plastic bags with a diameter greater than the diameter of the core.

The tailings are not only unconsolidated, but have a significant water content, and the cores deform plastically after being recovered. The plastic material slumped in the bags, and a 10-foot (3.05 m) run of core usually ended up occupying less than 10 feet (3.05 m) in the bag.

The oxidized material at the top of gold residue did not deform plastically. The iron oxide appeared to have grown into a crust that made the oxidized material act somewhat like a solid, especially with respect to the plastic material below it. The effect of this was that the oxidized material stuck in the bottom of the core tube and prevented the plastic tailings below from entering the core tube, so that few samples of the unoxidized material in the upper ten feet (3.05 m) of tailings were recovered. Since there appears to be no grade bias as a function of depth, and since the grade is shown to have a low variability (see discussion under Section 14.0), this is not believed to cause a significant uncertainty in the overall grade of the deposit.

The driller could "feel" the difference when going from the broken rock cap into the oxidized material, and from that to the unoxidized material. He noted those depths, and they were entered in the drill logs. The end of each ten-foot run, and the bottom of the hole are also hard numbers. The clay, where present at the bottom of the holes, is more competent than the unoxidized tailings, so the top of the clay (picked by the geologist in charge) defined the bottom of the unoxidized material for those holes that intersected clay. Those holes that did not end in clay ended at the bedrock surface, the depth of which is accurately known. Thus, although the unoxidized tailings deformed in their bags, the total interval occupied by them is known with sufficient accuracy for the purposes of this report.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The long bags containing the core were laid out on tables, the bags were slit open, and the total length of the core was measured. The hole depths at each end of each 10-foot run were converted to metres, and the core was marked at intervals representing 50 cm of hole depth. In most cases the length of core was less than 3.05 m, so the physical intervals were shortened in the ratio of the shortening of the run from which the samples were taken.

The core was split in two halves longitudinally with a knife or spatula. Half of the core was taken for metallurgical testing, and part of the other half was sampled for the purpose of estimating the concentration of gold and silver in the tailings to use in the resource estimation. The remaining part has been retained for later examination and/or re-sampling as required.

Each sample taken for resource estimation represented a 50 cm interval of the residue marked as described above. Each sample was put in a plastic sample bag, and numbered in sequence by a four-digit number from a set of sample books provided by the author (Newson) for this purpose. The sample books contained tags in two parts, one part of which was put in the sample bag with the sample, and the other part of which was retained for a record. On the retained parts were recorded the name of the company, hole number and interval in the hole, and the geology of the material sampled. These data were also recorded on a sample log, for a redundancy of information. The sample tag in the bag contained no information other than the sample number, and, by itself, could not be used by unauthorized persons to learn anything about the samples. The sample number was also written twice on the outside of each bag with a black marker.

Each metallurgical sample consisted of three intervals of 50 cm, and was identified by the three corresponding resource sample numbers. Each sample was put in a small plastic pail, and the samples were put on a pallet and sent to the metallurgical consultants.

The resource samples were sent to the Saskatchewan Research Council (“SRC”) laboratories in Saskatoon. Only about half of the samples were sent in order to have a quicker turn-around. Alternate samples plus duplicate samples were selected.

SRC's quality management system operates in accordance with ISO/IEC 17025:2005 (CAN-P-4E), General Requirements for the Competence of Mineral Testing and Calibration Laboratories. It is also compliant with CAN-P-1579, Guidelines for Mineral Analysis Testing Laboratories. SRC's management system and selected methods are accredited by the Standards Council of Canada (Scope of accreditation # 537). All electronic information and results sent out by SRC are protected by password, and are backed-up daily. Access to the laboratory buildings is restricted by an electronic security system, and the buildings are patrolled by security guards 24 hours per day.

The samples were sent in two batches. The first batch consisted of samples taken by the author (Newson) or taken by others under his direct supervision. The author stored these samples in the trunk of a rented car to which only he had a key. They were taken by him to the bus terminal in Snow Lake and shipped to the SRC laboratory. The rest of the samples were taken by or under the supervision of a director of BacTech.

Gold was determined by fire-assay. Since the material sampled did not require crushing, the first step was to split out a sub-sample using a riffler. The sub-sample was pulverized using a puck and ring-grinding mill. The mill was cleaned between each sample using steel wool and compressed air, or silica sand. The pulp was transferred to a labeled snap-top vial. An aliquot of the pulp was mixed with SRC's standard fire assay flux in a clay crucible, a silver in-quart was added, and the mixture fused. The fusion melt was poured into a form and cooled. The lead bead was recovered and cupelled until only the gold bead remained. The bead was then parted in a test tube with a solution heated in boiling water until the silver is dissolved. The solution containing the silver was decanted, leaving the gold in the test tube. Aqua regia was added to the gold in the test tube and heated until the gold dissolved. The sample was then diluted to volume and analyzed by ICP-OES (Inductively Coupled Plasma-Optical Emission Spectroscopy). Eight replicate samples were also added to the fire assay stream and reported.

Silver and a suite of other metals were determined by direct analysis of the pulp by ICP-OES (SRC's Method ICP3). An aliquot of pulp was dissolved in a mixture of concentrated hydrochloric acid and nitric acid in a bath of boiling water then topped up with de-ionized water. The sample solution was introduced into an argon plasma which is at a temperature of 8,000°C. At this temperature all elements become thermally excited and emit light at their characteristic wavelengths. The light produced was passed through a diffraction grating which separates it into its constituent wavelengths. The intensity of the various wavelengths is a function of the concentration of the element, and by comparing the intensity spectrum of a sample to known standards the concentration of the element in the sample was determined. Samples from a standard were added at fourteen points in the sample stream, and the same eight replicate samples as above were analyzed for silver.

12.0 DATA VERIFICATION

There are no data from previous work that can be verified by the author by, for example, re-analyzing old samples or re-sampling material previously collected. The entire program discussed herein was designed to verify the quantity of tailings available for exploitation and the concentration of gold and silver contained therein.

During the current program, duplicate samples were taken at more or less regular intervals and sent to the laboratory along with the regular samples as a check of the laboratory by the author (Newson). SRC also analyzed replicate samples for gold and silver, repeatedly reanalyzed a standard for silver, and reported them as noted in Section 11.0. The purpose in analyzing replicate samples was to check the accuracy of their work and the purpose of re-analyzing the same standard repeatedly was to check the precision of their work. The purpose in sending duplicate samples was to verify the accuracy of the SRC laboratory.

Thirteen duplicate samples were taken by the author (Newson) as part of the sampling program. The average gold grade of the duplicate samples differed from that of the primary samples by 2.6%. The averages of the silver analyses were identical. The duplicate samples represented the same intervals as the original samples, but were different splits from the core, so a difference of 2.6% in thirteen samples is acceptably close agreement.

Eight replicate samples were re-assayed for gold by SRC. These replicate samples came from the same pulp as the original, so they should exhibit less variability than the duplicates taken by the author, and they do. At 1.86%, this is a quite acceptable result. The 14 repeat analyses of the silver standard averaged 69.5 grams per tonne (g/t), with a standard deviation of 0.86, which is acceptable precision on the part of the laboratory. The accepted mean for the silver standard is 69.7 g/t, and SRC considers a range of 67.5 to 71.9 g/t to be acceptable. No individual analysis fell outside this range.

The average grade of the samples under the complete control of the author was somewhat higher than the average grade of the samples submitted by BacTech (10.2 g/t gold vs. 9.6 g/t gold).

As part of the data verification process the author (Newson) kept logs of drill holes separate from those kept by BacTech personnel during the time spent on site. There were no significant discrepancies between the author’s logs and the logs provided by BacTech.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Snow Lake Project testwork was conducted during Stage I (April to September, 2010) and Stage II (June, 2011 to March 2012) metallurgical testing programs using BACOX technology to treat the Gold Residue Stockpile ("GRS") material, which is a highly refractory sulphide material with a high arsenic content.

The key findings of the metallurgical testing programs were very positive in providing BacTech with an evaluation into the refractory nature of the GRS material and its suitability to bio-oxidation processing. Diagnostic leaching confirmed that the majority of the gold present in the GRS material was refractory and high oxidation levels will be required to maximize extraction by cyanidation. A maximum gold recovery of 96.5% was obtained after hot nitric acid digestion followed by cyanidation. This extraction compares to a gold extraction of only 9.4% for direct carbon-in leach (CIL) cyanidation.

A total of four bioleach tests with 5%, 10% and 15% pulp solids concentrations were undertaken by BacTech. The best results were achieved in the bioleach test with 10% pulp solids concentration. The BACOX process oxidized 95% of the sulphides rendering 88.6% of the gold contained in the sulphides cyanide leachable. BacTech believes that it is extremely likely that with further work, this gold extraction could be improved to approach the value of 96.5%, which was achieved in the diagnostic leach work. The bioleach test with 5% pulp solids concentration would probably have achieved better results if it had been allowed to continue to a higher oxidation level. Nevertheless, in that test 74% sulphide oxidation gave a 66.8% recovery of gold by cyanidation.

A number of issues have been identified which could be successfully addressed using appropriate design for a commercial plant. First of all, water washing confirmed that the GRS material contains a significant amount of thiocyanate, the majority of which can be extracted with water. Secondly, the arsenopyrite rich GRS material is characterized by a low Fe/As ratio. An innovative way to manage the situation is to take advantage of significant arsenic re-precipitation behavior during bio-oxidation. As dissolved arsenic starts to re-precipitate under suitable conditions, the iron:arsenic molar ratio in the bioleach solution changes as well.

In the bioleach test with 10% pulp solids concentration, the iron:arsenic molar ratio increased from 1.0:1 to 3.5:1. A rather intense hydrochloric acid wash on the bio-oxidized residue that contained the re-precipitated arsenic suggested that the Fe(III)/As(V) precipitate would be very stable. A large quantity of As(III) would be generated by the oxidative dissolution of arsenopyrite. Due to iron deficiency, the arsenite in the bioleach liquor may take time to be converted to pentavalent arsenate, which is more conducive for higher rates of bacterial activity. Importantly, the tests showed that the material could be successfully bio-oxidized to a level of 94.9% at a pulp density of 10% in a batch time of 50 days which equates to a residence time of approximately 6 days for a continuous commercial plant.

As As(III) increases in the bioleach liquor, it exacerbates the negative effects of thiocyanate on the bacterial activities, meaning that washing of feed material to reduce thiocyanate to acceptable levels should be an integral part of the material preparation process. It may also be the case that the amount of arsenite in the bioleach liquor affects the neutralization process and the characteristics of the neutralized residue. This needs consideration with respect to waste disposal.

The arsenic level in the leachates generated by the TCLP testing on the neutralized residues exceeded the US EPA arsenic limit even in the case where Fe/As molar ratio has been adjusted to 5.5.

Solution change and oxidation of the final leach solutions prior to neutralization or addition of other oxidants such as peroxide should be considered in the process flowsheet. Another alternative yet to be confirmed by testwork is to increase the Fe/As ratio to a level no less than 3 in the bioleaching stage in order to oxidize dissolved As(III) to As(V) in time. This would benefit the overall leaching process and prepare the leach solution for effective neutralization.

In conclusion, the key findings of the metallurgical testing programs support the applicability of BACOX technology to the Snow Lake GRS remediation project.

14.0 MINERAL RESOURCE ESTIMATES

14.1 MEASURED MINERAL RESOURCE ESTIMATES

14.1.1 Grade Estimations, Unoxidized Residue

The variability in gold content of the samples of unoxidized concentrate is low. The grade standard deviation of all samples, unweighted, is 2.73, in a distribution with an unweighted mean of 9.79 g/t gold. As a further check on the variability of the assays, the standard deviation of the average grades of individual holes (weighted by the intercept length of each assay) was calculated to be 1.36.

In a further test of variability of the gold assays, the statistics for the individual samples were recalculated excluding the two highest assays (25.7 g/t and 19.5 g/t) and the two lowest (1.25 g/t and 1.92 g/t) of the 202 samples used in the resource estimation. The standard deviation of the reduced sample is 2.262, a reduction of 17%, which is a large change for the elimination of so few values, and also indicates a strong central tendency in the distribution of assay values. (The mean was reduced by 0.47%.)

Another observation is of interest in the discussion of the variability of the gold assays. The mean of all of the assays, without weighting of any kind, is 9.79 g/t gold. The mean of the mean assay values for all of the holes, which values are weighted by the interval length of the assays, is 9.71 g/t gold. Adding a further weighting of the tonnes estimated for each hole yields a mean value of 9.76 g/t gold. This is a difference of 0.33% between a simple, unweighted, arithmetic mean of the values of all samples and a mean weighted both by interval length of the samples and by the tonnage indicated by each hole. The grade of this deposit is thus almost independent of any weighting.

Using the statistical parameters in Table 14.1, the gold grade of the unoxidized concentrate is estimated to be 9.76 g/t. The confidence interval of the grade, at the 95% confidence level, is 9.28g/t to 10.24 g/t. At the 99% confidence level, it is 9.12 g/t to 10.41 g/t.

Given the above, it is the opinion of the author (Newson) that no significant gain in accuracy of the mean, or increase in confidence, would be achieved by assaying the rest of the samples.

The silver concentration in the unoxidized concentrate, weighted by core length of the samples to get the average content for each hole, then using that value to calculate an average for the deposit weighted by the tonnage indicated by each hole, is 2.17 g/t silver.

Table 14.1
Snow Lake Mineral Resource Data

Hole Number	Surface Area of Influence (m^2)	Total Residue Thickness (Metres)	Thickness Less Oxidized Layer (m)	Volume Without Oxidized Layer (m^3)	Volume Reduction For Berm (m^3)	Net Volume (m^3)	SG	Tonnes	Gold Grade (ppb)	Gold Tonnes X Grade	Silver Grade (ppm)	Silver Tonnes X Grade	Oxide Thickness (m)	Oxide Volume (m3)	Oxide Tonnes (assumes SG = 4)
14	212	9.8	8.9	1,887	534	1,353	4	5,411	11,226	60,746,131	2.71	14,664	0.9	191	763
15	360	9.1	8.5	3,060	683	2,377	4	9,508	10,124	96,258,992	2.16	20,537	0.6	216	864
16	475	9.5	8	3,800	1,231	2,569	4	10,276	11,509	118,266,484	2.39	24,560	1.0	475	1,900
17	516	7.5	6.7	3,457	995	2,462	4	9,849	9,064	89,269,523	2.01	19,796	0.8	413	1,651
18	450	6.3	5.5	2,475	914	1,561	4	6,244	8,250	51,513,000	2.10	13,112	0.8	360	1,440
19	371	4.3	4	1,484	366	1,118	4	4,472	10,550	47,179,600	1.99	8,899	0.3	111	445
20	361	1.3	1.2	433	92	341	4	1,365	9,300	12,692,640	1.70	2,320	0.3	108	433
22	361	5.3	4.9	1,769	708	1,061	4	4,244	8,225	34,903,610	2.15	9,124	0.4	144	578
23	437	6.1	5.8	2,535	641	1,894	4	7,574	8,336	63,140,198	2.47	18,709	0.3	131	524
24	375	5.4	4.7	1,763	436	1,327	4	5,306	8,100	42,978,600	1.63	8,649	0.7	263	1,050
25	347	8	7.5	2,603	972	1,631	4	6,522	8,839	57,647,958	2.22	14,479	0.5	174	694
27	421	8	7.2	3,031	105	2,926	4	11,705	9,082	106,302,994	2.23	26,102	0.8	337	1,347
28	299	6.3	5.7	1,704	590	1,114	4	4,457	8,542	38,073,402	2.03	9,048	0.6	179	718
29	352	4.3	4	1,408	361	1,047	4	4,188	10,457	43,793,916	1.80	7,538	0.3	106	422
30	316	5.5	5	1,580	353	1,227	4	4,908	8,198	40,235,784	2.28	11,190	0.5	158	632
31	318	6.9	6.3	2,003	361	1,642	4	6,570	9,399	61,747,670	2.09	13,730	0.6	191	763
32	345	7.9	7.3	2,519	472	2,047	4	8,186	8,745	71,586,570	1.70	13,916	0.9	311	1,242
33	408	6.4	5.8	2,366	721	1,645	4	6,582	8,855	58,280,068	1.98	13,032	0.6	245	979
35	414	4.7	4.3	1,780	0	1,780	4	7,121	8,776	62,492,141	2.00	14,242	0.4	166	662
36	458	6.6	5.7	2,611		2,611	4	10,442	9,337	97,500,689	2.35	24,540	0.9	412	1,649
42	323	10.4	8.6	2,778		2,778	4	11,111	10,103	112,256,454	1.97	21,889	1.8	581	2,326
44	357	5.8	4.5	1,607		1,607	4	6,426	10,407	66,875,382	1.74	11,181	1.3	464	1,856
45	375	4.7	4.4	1,650		1,650	4	6,600	7,634	50,384,400	1.83	12,078	0.3	113	450
46	372	7.2	6.6	2,455		2,455	4	9,821	9,474	93,042,259	2.39	23,472	0.6	223	893
47	415	7.1	6.1	2,530		2,530	4	10,120	7,838	79,320,560	2.26	22,871	1.0	415	1,660
48	464	7	6.4	2,970		2,970	4	11,878	8,729	103,686,554	2.81	33,378	0.6	278	1,114
49	452	5.5	5.5	2,486		2,486	4	9,944	8,643	85,945,992	2.40	23,866	0.0	0	0
50	419	5.8	5.8	2,430		2,430	4	9,721	9,356	90,947,805	1.66	16,137	0.0	0	0
51	389	7.1	6.5	2,529		2,529	4	10,114	12,566	127,092,524	2.28	23,060	0.5	195	778
52	394	10	9.7	3,822		3,822	4	15,287	11,485	175,573,492	2.59	39,594	0.3	118	473
53	318	10.8	10.8	3,434		3,434	4	13,738	12,446	170,978,170	2.14	29,398	0.0	0	0
54	326	10.1	10.1	3,293	793	2,500	4	9,998	11,851	118,491,038	1.76	17,597	0.0	0	0
55	313	9.7	9.7	3,036	1,809	1,227	4	4,908	10,864	53,324,858	2.41	11,829	0.0	0	0
										2,582,529,458	SD=0.30	574,538			
					Measured tonnes:			264,596	grading:	9.76	g/t gold		Inferred tonnes		28,307
										2.17	g/t silver				

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14.1.2 Tonnage Estimation, Unoxidized Residue

The deposit has a quite regular shape, (Figures 14.1 and 14.2), and it was deemed by the author (Newson) to be appropriate to use a simple block method of estimating the volume. By this method the deposit is divided into blocks, each of which is centered on a drill hole. The measured dimensions of each block define its volume. The tonnage is determined by multiplying the volume by the specific gravity of the material. The overall average grade of the deposit is determined by estimating the mean grade of each hole, assigning those values to the blocks surrounding the corresponding holes, then calculating the mean grade of the entire deposit weighted by the tonnages of the blocks.

Figure 14.1
Map Showing Measured Resource Thickness and the Location of Drill Holes

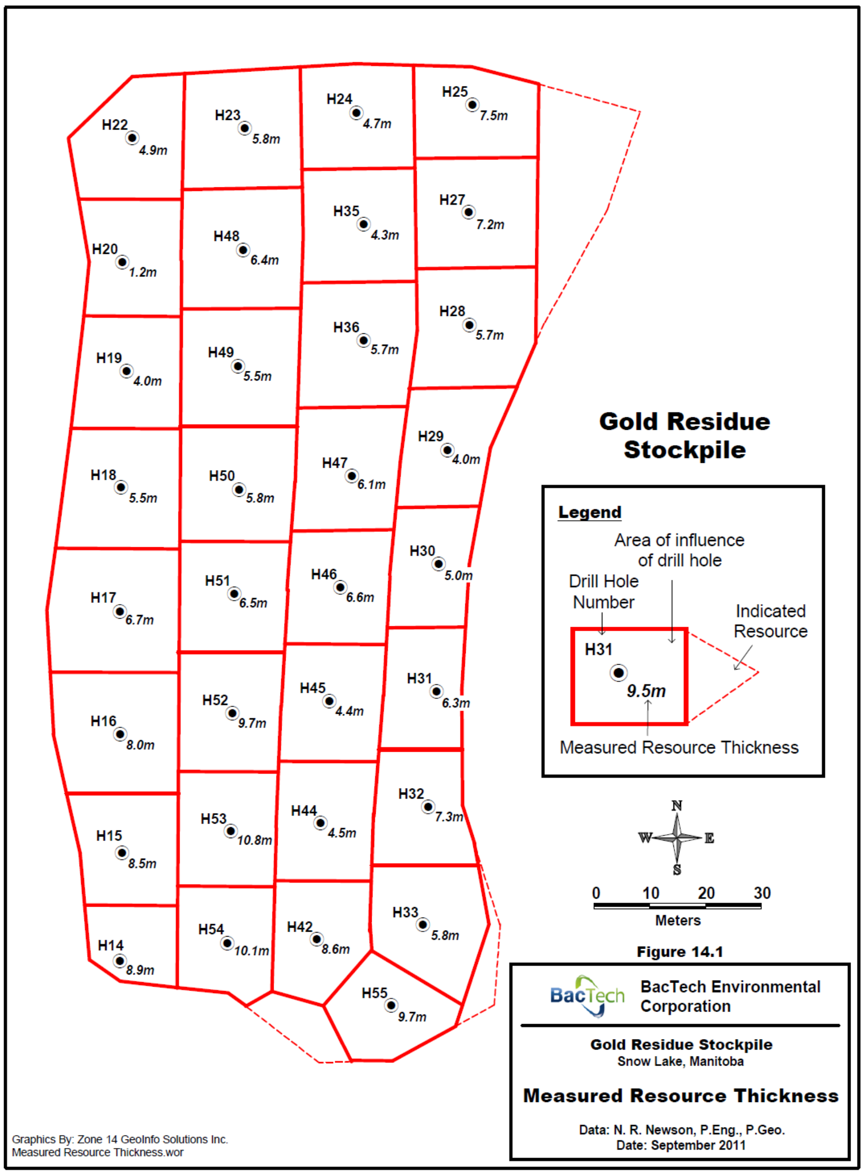
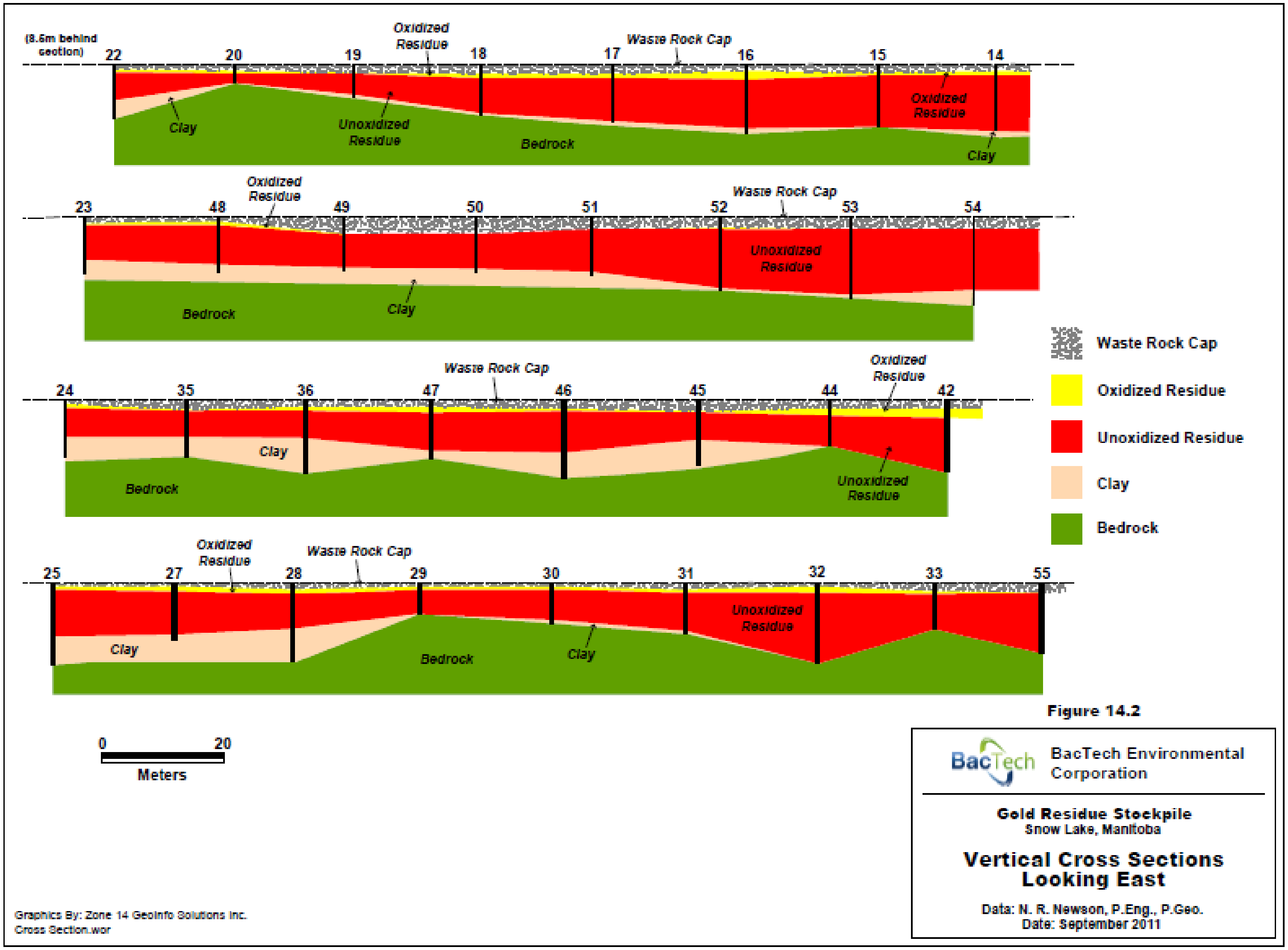


Figure 14.2
Vertical Cross Sections Looking East



Blocks were constructed using plan view, which shows the two greater dimensions. A hole spacing of 20 m had been chosen for the drill holes, and, in light of the relative insensitivity of the grade to weighting by the area of influence of the drill hole as discussed above, this spacing appears to have been adequate. Thus, all material between holes is considered to be Measured.

On the outer edge of the pile, the Measured area of influence of a hole was extended to a maximum of 12 m, which the author (Newson) believes to be appropriate. This leaves some parts of the pile out of the Measured category, and the author believes those parts to be Indicated (see Figure 14.1).

One uncertainty arises from the fact that the pile was covered with a cap of waste rock in an attempt to prevent the arsenic in the pile from escaping into the environment. The thickness of the cap at each drill collar was easily determined by the driller, but the thickness was not determined at the edge of the pile and on the outside slope of the pile. To compensate for this, the edge of the concentrate was estimated to be a distance inward from the physical edge of the pile equal to the thickness of the cap at the nearest hole. The outer edge of the deposit as shown in Figure 14.1 is the estimated edge of the concentrate, not the physical edge of the pile.

Simply multiplying the surface area by the vertical thickness of tailings intersected in the hole to calculate the volume of the block works well for the blocks in the interior of the pile, but a significant uncertainty exists for almost all of those blocks which are on the edge of the pile. That uncertainty results from not knowing the internal shape of the basin into which the concentrate was placed. The Request for Proposal issued by the Manitoba government with respect to this project referred to the pile as a “large waste rock impoundment...” An oblique air photo dated circa 1950 shows a berm of material which may be waste rock, although the size of the rock fragments is not determinable on the photo (seen in Salzsauler et al, 2005). However, the photo clearly shows the outer slope standing at what must be the critical angle of repose for whatever material it is, and one would normally assume that it has the same slope on the inside. It is not possible to measure the angle on the photo, but it is likely to be between 35° and 45°, measured from the horizontal.

A contradiction arises when projecting either of these angles from the edge of the pile inward. The contradiction is that most of the holes near the edge of the pile should have intersected the waste rock of the berm, but only one, hole 30, shows even a hint of such material. (Hole 47, an interior hole, also shows angular rock fragments, but that must be from another source.) In particular, hole 14, less than 5 m from the edge of the residue, should have intersected the berm over much of its 11.2-m length, but it did not intersect the berm at all.

For the purposes of the tonnage estimations presented in this report, the bottom of the concentrate was assumed to slope upwards from its lowest point in the nearest hole to the estimated edge shown on Figure 14.1. Since no hole, except possibly number 30, intersected any material that could be from a rock berm, this in a conservative approach, and there may be more tonnes of residue than are included in the estimates.

The mass of the material in the block is estimated by using a specific gravity determination by the metallurgical consultants. Their estimation was 4.00, based on 20 determinations. Note that this is based on a dry weight, not the in-situ weight of the material, which includes water. The analyses also were done on a dry basis, so both components of the resource estimates are on the same dry basis.

The author (Newson) believes that 265,000 dry tonnes of the deposit, grading 9.76 g/t gold and 2.17 g/t silver, qualify as a Measured Mineral Resource. This is based on his belief that the quantity, grade, density, shape and physical characteristics of the deposit are so well established that they can be used with sufficient confidence to support production planning and to evaluate the economic viability of the deposit.

Note that no recovery factor or dilution factor is included in this estimate.

14.2 INDICATED MINERAL RESOURCE ESTIMATES

14.2.1 Tonnage and Grade Estimations, Distal Unoxidized Residue

The distal material is defined as that material more than 12 m from a drill hole. All such material is at the edge of the pile. Three areas of Indicated Resource are shown on Figure 14.1, i.e. in the northeastern corner of the pile, in the southeastern corner, and in the south-central area. The author (Newson) believes that this material belongs in the Indicated Mineral Resource category because the quantity, grade, density, shape and physical characteristic can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters to support mine planning and the economic viability of the deposit. The relative insensitivity of the grade to weighting by tonnage demonstrated above gives a high level of confidence in the continuity of grade in these areas, and, since most of this Resource is above the level of the surrounding area, the shape of it can be seen and therefore inferred with a high level of confidence.

To estimate this category the nearest blocks were extended to the edge of the pile and the volume of the extension was measured, less the assumed berm. The grade of the Indicated material was assumed to be the grade of the hole. By this method, the northeastern area is indicated to contain 7,600 tonnes with a grade weighted by tonnage of 8.95 g/t gold, and 2.2 g/t silver. The southeastern area is estimated to contain 460 tonnes grading 8.86 g/t gold and 1.98 g/t silver. The south-central area is estimated to contain 1,200 tonnes grading 11.0 g/t gold and 2.09 g/t silver. The grade assigned to the south-central area is the average grade of the three nearest holes.

The total Indicated Mineral Resource is 9,260 tonnes grading 9.2 g/t gold and 2.15 g/t silver.

14.3 INFERRED MINERAL RESOURCE

14.3.1 Tonnage and Grade Estimations, Oxidized Residue

Core recovery of the oxidized crust on the top layer of concentrate was poor, and samples of it are under-represented in the data. Eight samples of oxidized material were assayed, and the mean gold content is 7.00 g/t, with a standard deviation of 1.03.

Notwithstanding the poor recovery, the thickness is known from the driller's report with a degree of confidence that puts it in the Inferred category. A mass of 28,000 tonnes is estimated for the oxidized zone. The best estimate of its grade is the mean grade of the samples analysed, i.e. 7.0 g/t gold and 2.4 g/t silver.

This material is an Inferred Mineral Resource because the quality and grade can be reasonably assumed, but with a lower level of confidence than is the case for the Indicated and Measured Mineral Resources. Sampling is limited, and the density has not been measured. However, since the oxidized material is derived from the underlying unoxidized material, its continuity can be reasonably inferred. In any case, since it overlies the Measured Resource, it would have to be handled at least once if the property is ever mined, regardless of whether it is taken for mill feed or waste, so it will have to be part of any feasibility study. Such a study would have to determine if the incremental cost of processing it is worthwhile at the lower gold grade indicated.

14.4 MINERAL RESOURCE SUMMARY

The mineral resources estimated for the GRS are set out in Table 14.2.

Table 14.2
GRS Mineral Resources Summary

Category	Tonnes	Gold (g/t)	Ounces	Silver (g/t))	Ounces
Measured Resource	265,000	9.76	82,643	2.17	18,488
Indicated Resource	9,300	9.2	2,750	2.15	642
Inferred Resource	28,000	7.0	6,300	2.4	2,160

Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Other than the uncertainties noted above, there are some risk factors that might affect the mineral resource estimates. The high arsenic content of the GRS may well attract more attention from environmental regulators than would be the case if no arsenic were present. However, since one of the intended results of the processing of the material in the GRS is that the arsenic will be put into a more stable compound that it presently is, it would seem unlikely that an environmental permit would be refused on the basis of the arsenic content alone.

The fact that the GRS is presently owned by the Government may raise the possibility of political risk to the project. However, the issuing of a request for proposals to remediate the site, and to assume the Government's responsibility for the remediation implies that the Government would like to see the project succeed. The political risk for this project should thus be no greater than for any other project in Manitoba.

The location of the GRS, on the property of another mining company, within about 100 m of some of their buildings, represents an inconvenience, since arrangements will have to be made to have access to the site in a manner that will not disrupt the operations of the other company. This may or may not result in increased costs compared with what the costs would have been if the GRS were on Crown land.

Changes in economic parameters, i.e. metal prices, taxation, labour costs, energy costs are risks that affect all similar projects, and have the potential affect this one. Because of the short life of the project there is likely to be more stability in these parameters than would be the case in a longer-lived project.

15.0 MINERAL RESERVE ESTIMATES

As there has been no prefeasibility or feasibility study completed on the recovery of gold from the GRS there are no mineral reserves.

16.0 MINING METHODS

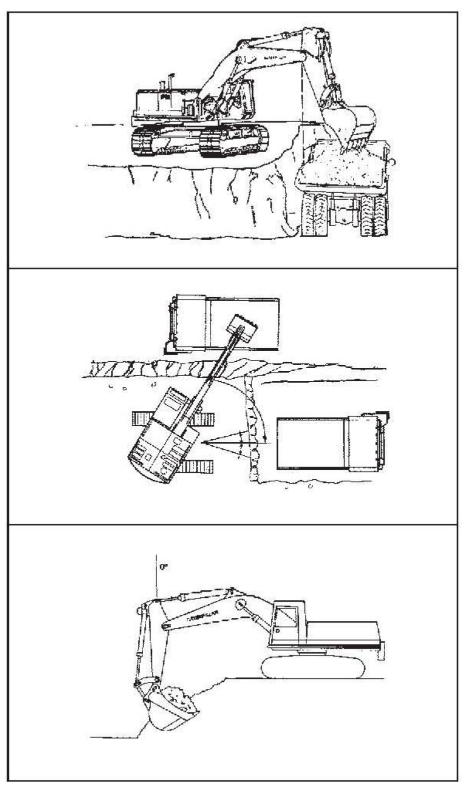
16.1 INTRODUCTION

The Snow Lake tailings deposit will be reclaimed with convention earth moving heavy machineries such as the hydraulic excavator or Frontend Loaders (FEL). The reclamation of the mineralized material does not require any drilling and blasting activities subjected by conventional mining operations. Excavated material can be transported to the processing plant with either articulated or rigid frame trucks.

The reclamation process will commence from the closest location to access road or processing facilities working towards the remaining in-situ material. The project does not require intensive selectivity or grade control because grade variability of the deposit material is reported to be low. This enables the material to either be removed by layer or through trenching.

The relative insensitivity of the grade change in the deposited material permits the material to be excavated from any locations, either by layer or trenching or with variable faces if an increase capacity is required. The figure below presents a typical schematic of a hydraulic excavator digging and loading a truck on a reclamation face (Figure 16.1).

Figure 16.1
Schematic of Hydraulic Excavator Loading a Truck (Source: Caterpillar, 2007)



Due to the physical condition of the deposited material, the reclamation process can be performed by local contractors equipped with heavy earth moving equipment.

16.2 DILUTION AND RECOVERY

The use of earth moving equipment such as the hydraulic excavator or FEL enables high selectivity during material excavation with an experienced operator. This will minimize the amount of dilution or material loses (i.e. clay liner at the bottom or wasted impoundment surrounding the outer perimeter of the deposited material) to be excavated during reclamation.

At this stage, no losses (100% recovery) or dilution (0%) are anticipated for the recovery of the deposited material. This is because during the exploration, the results of the holes near the edge of the pile did not intercept waste rock with exception of one hole. Un-mineralized external material (dilution) located on the contact or capping material can be easily identified visually by the operator and can be temporary stored or backfilled into the excavated section beside the excavator. A schematic of the site layout is presented in Figure 16.2.

16.3 PRODUCTION RATE AND MINING EQUIPMENT

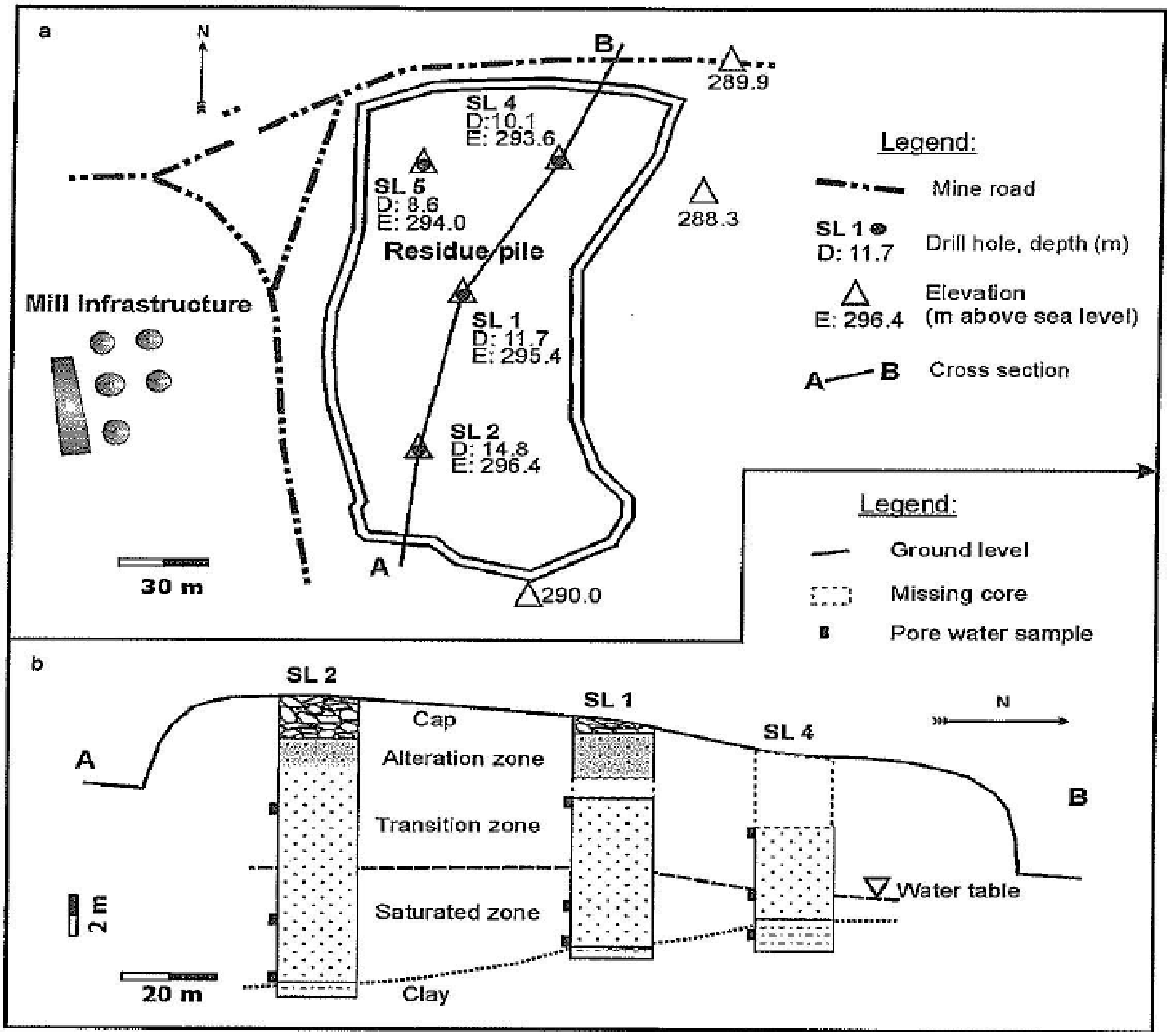
The overall site reclamation process will be operating 7 days a week at 24 hours per day for the duration of 8 months per year (243 days).

The equipment fleet will consist of one 2.3-3.0 m^3 bucket capacity excavating equipment (hydraulic excavator or FEL) with one 28 t payload truck. This equipment will be able to transport approximately 450 t of material in an 8 hours shift based on a preliminary estimate of the production cycle time.

Based on this estimate, there will be excess capacity so that the excavation operation can be concentrated during the day shift. An operation of 243 days has the capacity of producing 109,350 t of material compared to the proposed annual production rate of 46,650 t of wet tonnes.

The reclamation will only be operating 8 months per year and will be shut down during the winter months. Any forecasted operation into the winter months will require the working area to be covered with an insulating material or thick geotextile to prevent freezing. It is recommended to cover the stockpiled material exposed to the weather and heat if necessary to avoid freezing or excess exposure to moisture during spring or rainy days.

Figure 16.2
Schematic Plan View of the Site



(a) The elevation and location of drill holes in the AGRS.
(b) A schematic cross-section (AB) from the south (SL2) to north (SL4), not to scale (Salzauler, 2004; Salzauler, Sidenko and Sherriff, 2005)

16.4 CONCLUSION

The reclamation of the deposit material can be performed with minimal supervision and by conventional earthmoving equipment with the use of local contractor. The low variability in grade enables the reclamation to be performed at any location close to the processing facilities or access road. This allows for a potential increase in production rate with multiple working faces with an increase in the equipment fleet. The material has an SG of 4.0 and this will have to be accounted for during the equipment selection by the contractor and the operator.

The challenges will be working during winter or wetter months and the stability of the excavated front during operation when there is a “highwall”. Provision of covering blankets or alternative temporary shelter will be required at the working face, temporary storage or stockpile areas to prevent freezing of the material. The material can be excavated in benches to increase stability or reduce the occurrence of “highwall” during mining of thick deposit areas. Provision will have to be accounted for portable heating systems and dewatering pumps at the face.

An increase the sampling frequency for Inferred Resource material is recommended so that it can be developed into a higher geological classification and possibly improve the overall resource tonnage.

17.0 RECOVERY METHODS

17.1 PROCESS OVERVIEW

The proposed development consists of three (3) main components:

1. Gold Residue Stockpile (GRS).
2. Process Facility.
3. Storage Impoundment Area.

17.2 GRS

The GRS is currently capped with a coarse angular mine waste which will be removed to expose the residue material in sections minimizing the exposure to the various weather elements. The GRS material contains approximately 16% moisture which was determined during a drilling campaign in 2010. The moisture content will alleviate the need for dust control, However, dust control measures will be implemented if and when the need arises. The GRS will be excavated at a rate of approximately 135-150 t/d over an eight month period to accumulate a stockpile which will accommodate a 12 month operation at the Process Facility. The material will be transported utilizing an end-dump dump truck from the GRS to the process facility approximately 600 m, (2,000 ft.) on a non-designated private roadway. The GRS material will be stored in a weather proof structure to prevent exposure of the material to wind, rain, and snow. A water management plan will be implemented at both the GRS and the storage structure to collect and utilize all water accumulated by snow melt or rain to mitigate any risk of contamination to the surrounding environment.

17.3 PROCESS FACILITY

The Process Facility will be contained in a building approximately 30 m long by 20 m wide, (600 m^2). The process equipment will consist mainly of two grinding mills, tanks, agitators, piping, pumps, and electrical components, all of which will make-up the following 5 circuits:

1. Material Preparation Circuit (Area 10).
2. Bioleach Circuit (Area 20).
3. Residue Liquor Separation Circuit (Area 30).
4. Neutralization Circuit (Area 40).
5. Reagent Circuit (Area 50).

Material will be fed from the bioleach facility stockpile into the feed hopper at the Material Preparation Circuit at an approximate rate of 100-125 t/d by a front end loader. A grizzly screen will remove debris before entering the repulp tank where wash water will be added to give a 40% wt. pulp. The pulped material will feed a polishing regrind process which will operate in closed circuit with two cyclones and a polishing mill, with the cyclone overflow being fed to a hi-rate thickener. The cyclone underflow will be returned to the regrind circuit after further grinding in the polishing mill.

The purpose of the regrind polishing is to freshen the surfaces of the particles and reduce the size of the coarse fraction present to achieve a particle size P_{80} 75µm. This process will also result in the removal of any residual cyanide species present from the previous operation which could otherwise interfere with the bacterial oxidation process. The thickener overflow will feed the mill process water pond for re-use in the circuits. Thickened underflow from the regrind circuit will be pumped to an agitated feed storage tank at the head of the Bioleach Circuit. The storage tank is designed for a 30 hour buffer storage capacity to allow for downtime maintenance of the regrind circuit.

The thickened pulp from the storage tank will be pumped to the primary bacterial oxidation reactors with in-line water dilution being added using automatic control to give a feed operating density of 10% solids. The diluted pulp will be fed to a splitter header box where the pulp will be divided equally between three primary reactors operating in parallel.

The bacterial oxidation circuit consists of six reactors of identical size with three reactors operating in parallel as primary reactors with the combined product from these reactors feeding a train of three reactors operated in series. Pulp will flow between the different reactor stages by gravity, using a system of open launders. The total residence time for the process will be 5 - 6 days with a minimum oxidation level of 95% pyrite being achieved and total oxidation of arsenopyrite. Acid addition will be made to the primary reactors to satisfy the acid balance requirements for arsenopyrite oxidation. A nutrient mix consisting of milligram levels of nitrogen phosphorous and potassium will be fed to the primary bacterial oxidation reactors.

The reactors will be agitated with fixed speed drives and hydrofoil type impellors for three phase mixing, and sparged with air through a ring main delivered from two low pressure blowers. The process is exothermic and heat is removed by cooling tube bundles located in the reactors which also act as baffles to ensure good mixing and solids suspension. The temperature of the reactors will be maintained at 40°C by regulating the flow of cooling water to each of the reactors.

The oxidized pulp will be pumped from the last bioleach reactor to the residue thickener at the head of the Residue Liquor Separation Circuit. The overflow from the thickener reports to the Neutralization Circuit and the oxidized underflow solids will be washed and filtered using a belt filter. The filter cake consisting of the oxidized residue containing gold and silver values will be shipped off-site for precious metals extraction.

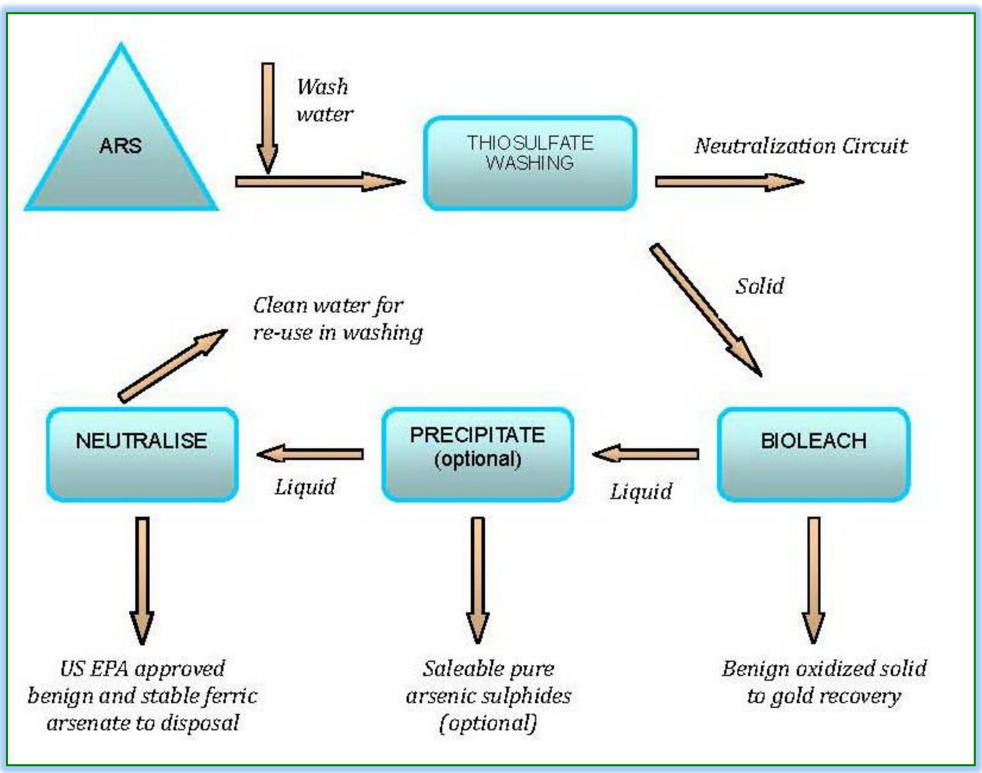
In the Neutralization Circuit the bioleach liquor from the residue thickener overflow containing soluble iron, arsenic and acid, together with the filtrate from residue filtration operation will be neutralized with limestone to produce a stable ferric arsenate precipitate for disposal. The neutralization section consists of four agitated leach tanks reactors, 75 m^3 in volume each, that operate in series. The design residence time for liquor neutralization is six hours and the pH of the liquor will be increased to pH 6.5 using limestone slurry delivered through a ring main.

The neutralization process results in the formation of ferric arsenate in a matrix of gypsum and ferric hydroxide. The neutralized pulp feeds the neutralized precipitate thickener where a thickened underflow of precipitate will be produced, which will be pumped to the storage impoundment area. Water will be reclaimed from the storage impoundment area after settlement of the precipitate and the pore water is exposed. The return water together with the clean neutral overflow water from the neutralization thickener will report to the process water pond where it will be re-used as dilution water for bacterial oxidation process.

17.4 FERRIC ARSENATE STORAGE IMPOUNDMENT

The underflow slurry from the Precipitate Thickener (neutralized ferric arsenate) will be pumped from the process facility to the impoundment area through 2 High Density Polyethylene pipelines. One pipeline will be in operation, with the second pipeline in stand-by mode. A third pipeline will be required to return reclaimed water from the impoundment back to the Mill Process Water Pond for consumption in the process. The most suitable location for the Storage Impoundment is a brown field location adjacent to the existing QMX tailings pond approximately 1.4 km northeast of the facility. The impoundment area will be a clay and HDPE lined pit. Engineered details are currently in progress to determine the appropriate size and materials of construction.

Figure 17.1
Snow Lake Project Simplified Metallurgical Flowsheet



The preliminary process design criteria used for the PEA was developed from the metallurgical testing results and is summarised in Table 17.1.

Table 17.1
Process Design Criteria

	Units (metric)	Value
Feed Composition:		
Gold	g/t	9.68
Silver	g/t	2.17
Iron	%	25.18
Arsenic	%	21.86
Reclamation:		
Moisture content of stockpile	%	16.00
Annual production rate	wet tpa	46,650
Annual production rate	dry tpa	39,186
Day reclamation rate on annual basis	dry t/d	107
Operating time	mo/yr	8
Operating time	d/wk	7
Reclamation operating days	days/yr	243
Operating time	wks/yr	35
Operating time	h/d	24
Operating time	h/yr	5,840
Reclamation rate required	dry t/h	6.7
	dry t/d	161
	wet t/h	8.0
	wet t/d	192
Stockpile size required for winter feeding of process plant	dry t	11,611
	wet t	13,823
	wet vol m^3	5,115
Stock Pile foot print at 10 m height	m^2	511
Regrind Wash Facility With Thickener:		
Wash regrind plant operating time	wks/yr	49.40
Operating time	d/wk	7
Operating time	h/d	24
Operating time	h/y	8,299
Reclamation rate from plant stockpile to feed mill	dry solids t/h	4.72
	t/h water	0.90
Water SG		1.00
Reclamation rate from plant stockpile to feed mill	wet t/h	5.62
SCN content of moisture	mg/l	10.00
Iron content of moisture	mg/l	5.00
Arsenic content of moisture	mg/l	20.00
Pulp density in initial repulp (same as cyclone overflow density)	%	40%
Water addition to repulp tank (to balance cyclone overflow loss)	t/h water	6.18
Water make-up SG		1.00
	m^3/h	6.18

	Units (metric)	Value
Total Volume of repulp tankage	m^3	5
Regrind mill recirculating load	%	350%
Water in overflow	t/h	7.08
Total feed into mill including recirculating load	dry solids t/h	21.25
	water t/h	12.59
	%solids	62.79%
Feed rate to high rate thickener	t/h solids	4.72
	t/h water	7.08
	m^3/h	8.26
Pulp flow underflow	t/h	9.08
	m^3/h	5.54
Water in overflow	t/h	2.72
Feed Buffer Storage For Bioleaching:		
Pulp flow rate to feed storage tank	m^3/h	5.54
Bioleaching:		
Bioleach operating time	wks/yr	49.40
Bioleach Inflow:		
Total liquid entering bioleach	t/d	1,018
	m^3/d	1,018
Total solids entering bioleach	t/d	113
	m^3/d	28
Total pulp flow entering bioleach	t/d	1,131
	m^3/d	1,046
	m^3/h	43.60
Bioleach Reaction Kinetics:		
Total pyrite oxidation	%	95
Total arsenopyrite oxidation	%	100
Total pyrrhotite oxidation to completion	%	100
Final arsenic liquor grade	g/L	21.22
Final iron liquor grade	g/L	23.03
Final copper liquor grade	g/L	0.04
Total ionic grade major elements	g/L	44.29
Final liquor SG		1.050
Final pulp density	%	3.63
SG of leached solids		2.7
Evaporative loss in bioleach circuit	%	10
Bioleach Reactor Volumes:		
Air hold-up volume in reactors	%	4
Total reactor working volume required	m^3	5,877
No. of reactors		6
Working volume of each reactor	m^3	979
Dimensions of each reactor		
Reactor free board allowance	m	1.0

	Units (metric)	Value
Reactor diameter (assume 1:1 h:d for working volume ratio)	m	10.8
Reactor height	m	11.8
Aeration Requirement:		
Total oxygen required from air	kg/Mol/hr	56.91
	$N/m^3/hr$	1,275
Air required @ 100% utilisation	$N/m^3/hr$	6,070
Oxygen utilisation	%	35
Blower air required (dry basis)	$N/m^3/hr$	17,344
Assume worst case 600F wet bulb (0.023mol H_2O/mol dry air)	$N/m^3/hr$	23,958
Safety allowance (pipe losses; oxidation chalcopyrite; etc)	%	5
Allowance for elevation of project	%	0
Actual blower air required	$N/m^3/hr$	25,156
Total blower power rating	Kw/h	1,258
Input no of operating blowers (none standby)		2
Individual blower air delivery (normal LP 108 kPaG rating)	$N/m^3/hr$	12,578
Individual blower power rating	kW/h	629
Agitator Power		
Agitation Requirement:		
Total agitator power required	kW/h	1,293
No of agitators	-	6
Individual agitator power primary (4 required)	kW/h	230
Individual agitator power secondary/tertiary (1 required)	kW/h	162
Individual agitator power quartenary (1 required)	kW/h	103
Check totals - only rough agreement required	kW/h	1184
Oxidised Bioleach Pulp Thicken And Wash Filter:		
BioLeached pulp flow feed to high rate thickener	m^3h	39.05
Thickener density underflow density	%	55
Solids Mass Flow in underflow	t/h	2.36
Solids Volume in underflow	m^3h	0.87
Liquor Mass Flow in underflow	t/h	1.93
	m^3h	1.84
Original bioLiquor mass flow in overflow reporting to neutralisation	t/h	38.16
	m^3h	36.34
Wash water ratio to solids for belt filter		3.00
Wash water used	m^3h	7.08
	t/h	7.08
Total liquor to neutralisation from thickener wash filtration circuit	m^3h	43.16
	t/h	45.24
Filtered washed solids density	%	90
Wash efficiency	%	95
Solids in cake	t/h	2.36
	t/d	56.61
Water in cake	t/h	0.26
	m^3h	0.26
Spent wash water	m^3h	8.74

	Units (metric)	Value
	t/h	8.74
Bioleach Liquor Neutralisation:		
Bioleach Liquor overflow from Thickener	t/h	38.16
	m^3h	36.34
Bioleach liquor from filter discharge	t/h	8.74
	m^3h	8.74
Neutralisation reactor volume required	m^3	301
Number of neutralisation reactors	-	4
Volume per reactor	m^3	75
Reactor free board allowance	m	0.5
Reactor diameter (assume 1:1 h:d for working volume ratio)	m	4.6
Reactor height	m	5.1
Number of days inventory of limestone for storage	d	4
limestone stockpile required	t	285
Lump size of delivered limestone	mm	12.5
Assumed bulk density	t/m^3	1.36
Stock pile foot print at 10 m height	m2	21
Ferric Arsenate Precipitate Pulp Thickener:		
Precipitate flow feed to high rate thickener	m^3h	50.16
	m^3h	13.94
Clean water as thickener overflow recycle to water pond	m^3h	33.22
	t/h	33.22
Ferric Arsenate Gypsum Product Production Rate:		
Pumping rate to tailings pond	m^3h	16.94
	t/h	21.44
	t/d	515
Assumed retained moisture after settlement	%	50
	t/d	180.11
Water to be returned from tailings pond	t/d	154.38
	t/h	6.43
Wet quantity of what will become final settled sludge	t/h+ water crys + moisture	15.01
	t/d	360.22
	t/h + water cryst	7.21
	t/d	173
	t of pptate / t of feed	3.19

18.0 PROJECT INFRASTRUCTURE

18.1 POWER SUPPLY

The base case total power requirement is 612 kWh/t which results in a maximum demand of approximately 3.3 MW for this 107 t/d operation. Power will be supplied by connecting to Manitoba Hydro’s high voltage electricity grid directly adjacent to the proposed process facility.

Electrical power to the property is currently supplied by 64 kV power lines and a substation on the property.

18.2 WATER SUPPLY

Micon considers that several options for make-up water to satisfy project requirements are available. Process water will be drawn directly from the Snow Lake. Potable water is available directly from the municipal supply of the town of Snow Lake.

18.3 BUILDINGS

The site will accommodate the construction of a process plant, with possible auxiliary out buildings on the site to house the technical department offering a total available space of 615 m^2, laboratory, a maintenance shop for equipment, a heated warehouse, a gatehouse, and a covered concentrate storage facility.

18.4 ANCILLARY FACILITIES

Water tanks for industrial use will be installed, including one for fire suppression water. The fire suppression water will be distributed to the protected area through an underground water pipe network.

A fuel tank with an adjacent fuelling station will primarily supply mining equipment needs.

Sewage services for the facility will be tied into the municipal services for treatment at the towns collected from the various facilities and pumped into a sewage treatment plant.

Communication facilities will be comprised of a redundant fibre communication backbone system which will link and manage the data transmission of the distributed control system (DCS), third party programmable logic controllers (PLCs), motor controls, fire detection system, and computers around the project site. It is expected that the site will likely be connected to the local phone system network.

For the duration of the construction work, portable temporary office and facility trailers will be used.

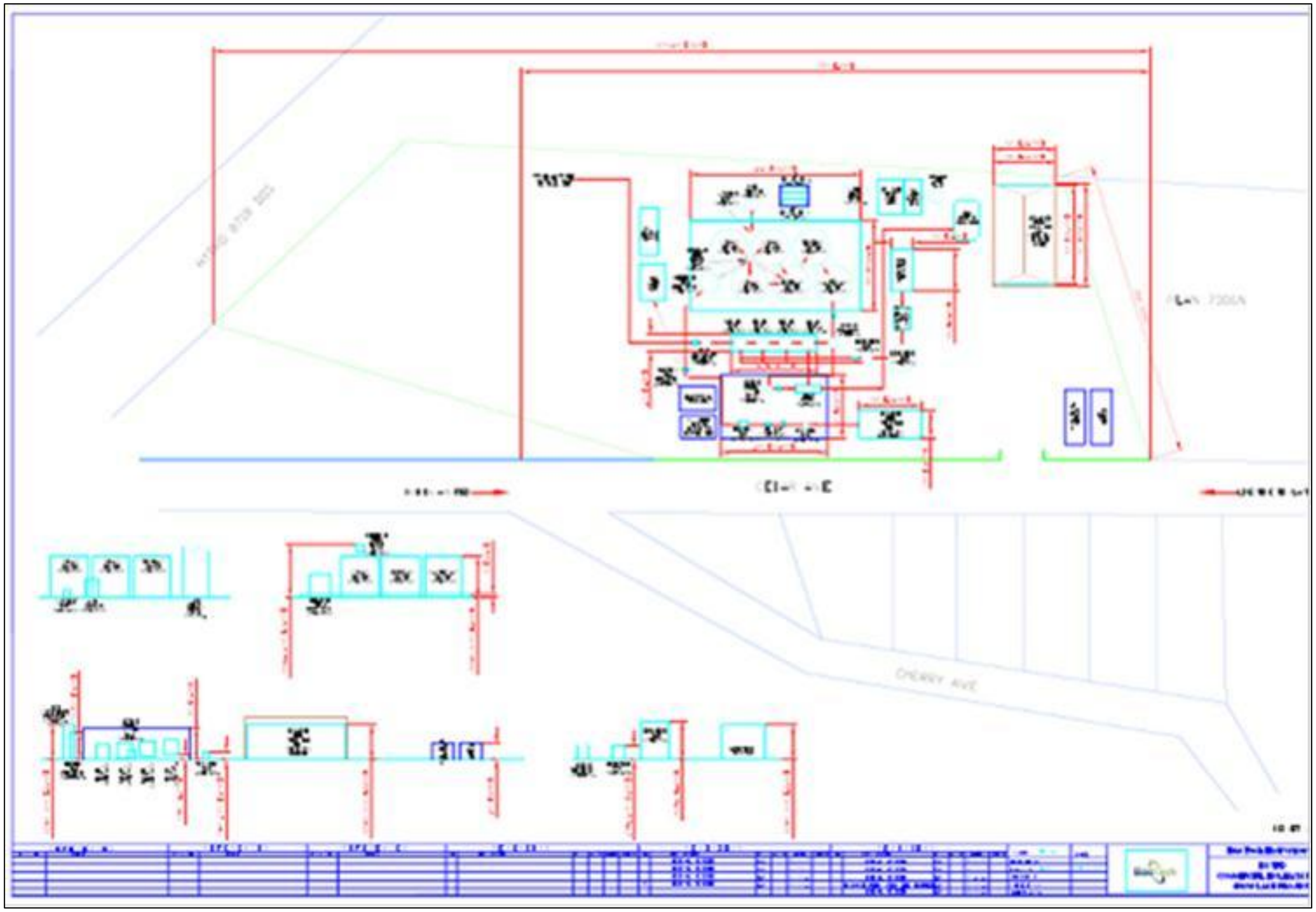
18.5 TAILINGS STORAGE FACILITY

The underflow slurry from the Precipitate Thickener (ferric arsenate) will be pumped from the process facility to the impoundment area through one operating and one stand-by High Density Polyethylene pipeline. A third pipeline will be required to return reclaimed water from the impoundment back to the Mill Process Water Pond for consumption in the process. The most suitable location for the Storage Impoundment is a brown field location adjacent to the existing QMX tailings pond approximately 1.4 km northeast of the facility. The impoundment area will be a clay lined pit. Engineered details are currently in progress to determine the appropriate size and materials of construction as required.

18.6 SCHEMATIC SITE LAYOUT

Micon has not made specific recommendations as to location of these works pending discussion with affected landowners. Accordingly, the layout shown in Figure 18.1 is schematic only. Progress is on-going between the town of Snow Lake and BacTech to acquire a suitable site in the town.

Figure 18.1
Snow Lake Project Plant Site Layout



19.0 MARKET STUDIES AND CONTRACTS

In evaluating the project, Micon made assumptions about the terms on which products from the project might be sold which, based on its experience on similar projects elsewhere, it believes to be reasonable. Micon is not aware of any project-specific contract or off-take terms having been negotiated for sales from the BacTech project.

Significant contracts that will be required prior to entering production will be for:

- Supply of electrical power to the site.
- Supply of fuels, cement, limestone, grinding media and other reagents and consumables for its mining and processing activities.

Micon has used publicly available information to derive its estimate of the cost of power required by the project from the relevant utilities, and has used knowledge and experience gained on other projects to derive other significant input unit costs. Micon is not aware of any project-specific contract or terms having been finalised for supplies to the Snow Lake project.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The GRS is situated on the site where the previous Nor-Acme Mine operated from 1949 to 1958 in Snow Lake, Manitoba. Today, this 250,000 tonnes stockpile has been identified as the largest source of contaminant loading at the site. Groundwater in a monitoring well south of the GRS reportedly has concentrations of >20 mg/L As (Salzsauler, Sidenko and Sherriff, 2005), which is 40 times greater than the Canadian Environmental Quality Guidelines and Summary (CCME) Effluent Regulations for arsenic release from active mine sites in Canada of 0.5 mg/L.

There are three issues which give rise to environmental concern and the urgent need for remediation:

1. The presence of considerable amounts of arsenic which could possibly leach into surrounding water sheds causing potential hazards to life forms both currently and in the future;

2. The presence of cyanide compounds giving rise to similar concerns as arsenic. Distilled water leaching tests by BacTech confirmed that the GRS material contains a significant amount of thiocyanate that is partially extractable with water and

3. The presence of considerable amounts of sulphide minerals in the material can lead to acid mine drainage pollution.

BacTech's strategy is to eliminate the root cause of arsenic mobility by fully oxidizing the sulphides contained in the GRS. This would turn the GRS material into a benign and environmentally safe product that has very low water solubility, is insusceptible to atmospheric and bacterial oxidation and is suitable for long term storage in an engineered disposal site. Bioleaching has the unique ability to address several tailings-related environmental issues at once, while generating revenue that can effectively fund the remediation process.

Table 20.1
List Of Major Permits Required

Organisation	Permit	Timeline
Town of Snow Lake	Potable Water service	Q4 2012
MB Conservation	Environmental Act License (EAP approval)	Q1 2013
MB Water Stewardship	Water Rights License	Q1 2013
MB Mines Branch	Closure plan	Q2 2013
Town of Snow Lake	Building & Plumbing permit	Q2 2013
Manitoba Infrastructure and Transportation	Road access	Q2 2013
Manitoba Hydro	Electrical permit	Q2 2013

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

Capital expenditures and capitalized development costs for the base case are summarized as initial and sustaining costs in Table 21.1. The estimates are expressed in third quarter 2012 Canadian dollars, without escalation, unless otherwise noted. The expected accuracy of the estimates is ± 30%.

Table 21.1
Capital Cost Summary

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Processing Plant and Equipment	9,996	-
Indirect Costs	4,195	-
Owner's Costs	4,346	-
Contingency	2,818	-
Total	**21,356**	**-**

All costs associated with loading and transporting the GRS material to the plant are covered by the proposed contractor's operating costs.

21.1.1 Processing Plant & Equipment

The capital estimate for the processing plant breaks down as shown in Table 21.2. Ongoing maintenance is covered by operating costs, so there is no sustaining capital forecast.

Table 21.2
Process Plant Capital Estimate

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Mobile Equipment	100	-
Regrinding Circuit	939	-
Dewatering	1,250	-
Common Systems	3,285	-
BACOX Area	4,422	-
Total	**9,996**	**-**

21.1.2 Indirect Capital, Owner's Cost and Contingency

The indirect capital cost estimate is shown in Table 21.3.

Table 21.3
Indirect Capital Cost Estimate

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Construction Equipment	1,522	-
EPCM	1,102	-
Internal costs (BacTech)	586	-
Incidental Expenses	550	-
Sub-total Indirect Costs	**4,195**	**-**
First Fills, other start-up costs	2,154	-
Insurance	150	-
Recruitment & Training	96	-
Owners Site costs	691	-
Rehabilitation and Closure Bonding	1,255	-
Sub-total Owner's Costs	**4,346**	**-**
Contingency	2,818	**-**
Total	**11,359**	**-**

Indirect costs include EPCM costs, estimated to be 11% of the direct capital cost estimate for the process plant. A provision equivalent to 20% of direct costs is also made for the temporary hire of construction equipment (e.g., mobile cranes) and general site costs.

Owner's costs include first fills of reagents and consumables in the plant, construction insurance, commissioning, recruitment/training costs and owner's site costs (including site management and supervision).

Environmental bonding costs are estimated to be $1.25 million, reflected in the cash flow as an up-front cost incurred prior to production start-up.

A total contingency of $2.82 million equates to approximately 15% of the initial capital, including indirect costs.

21.2 OPERATING COSTS

21.2.1 GRS Feed Operating Costs

Operating costs for transport of the GRS material assume a contract rate for loading and hauling of $3.70/t. Provision is also made for feeding GRS material into the processing plant, using the project owner's own equipment, for a total operating cost of $7.91/t of GRS feed.

21.2.2 Processing Operating Costs

Processing costs of $158.78/t include a total of $57.42/t for labour, $75.08/t in reagents, spares and consumables and $26.28/t in electrical power. The latter assumes a power cost of $0.041/kWh, including maximum demand charges.

21.2.3 General and Administration Costs

General and Administration costs include site management, mobile equipment, office running costs, environmental management and insurance costs.

Estimated cash operating costs over the life of the project are summarized in Table 21.4.

Table 21.4
Summary of Life-of-Mine Operating Costs

Area	Life-of-mine Cost ($ 000)	Unit Cost $/t ore treated
Contract Load & Haul	1,119	3.70
Plant Feed (front-end loader)	1,274	4.21
Sub-total Plant Feed	**2,393**	**7.91**
Labour - Metallurgy	2,037	6.74
- Laboratory	972	3.22
- Production	10,739	35.52
- Maintenance	3,610	11.94
Maintenance	4,114	13.61
Reagents	18,583	61.47
Power	7,945	26.28
Sub-total Processing	**47,999**	**158.78**
Labour	1,740	5.76
Mobile Equipment Operation	124	0.41
G&A (other)	2,700	8.93
Sub-total General and Administrative	**4,564**	**15.10**
Total Operating Costs	**1,021,431**	**181.79**

22.0 ECONOMIC ANALYSIS

22.1 BASIS OF EVALUATION

Micon has prepared its assessment of the Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to determine the viability of the proposed process plant to rehabilitate the GRS repository. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

22.2 MACRO-ECONOMIC ASSUMPTIONS

22.2.1 Exchange Rate and Inflation

Unless otherwise stated, all results are expressed in Canadian dollars. Cost estimates and other inputs to the cash flow model for the project have been prepared using constant, third quarter 2012 money terms, i.e., without provision for escalation or inflation. Using a trailing 36-month average to July, 2012, an exchange rates of CDN$1.02/US$ is applied in the base case.

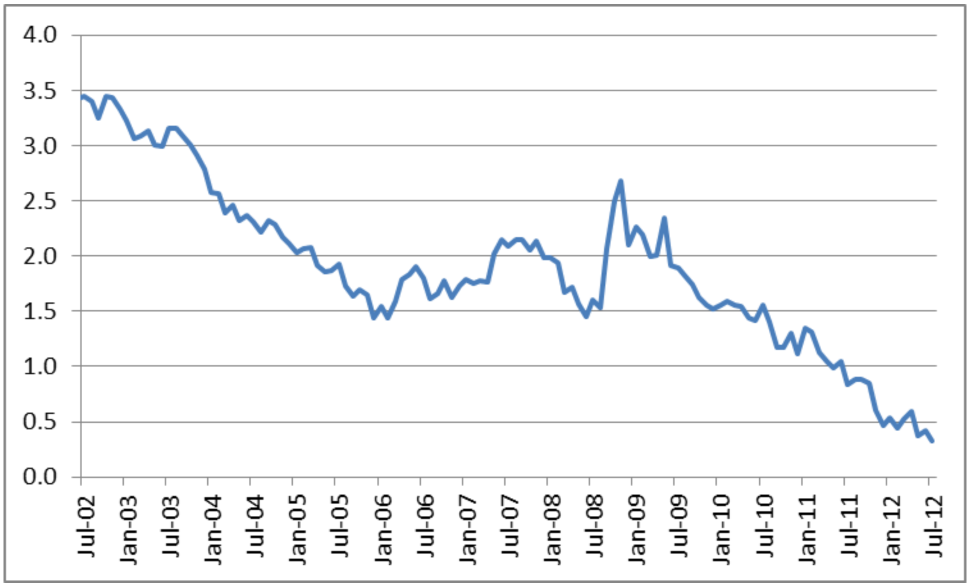
22.2.2 Weighted Average Cost of Capital

In order to find the NPV of the cash flows forecast for the project, an appropriate discount factor must be applied which represents the weighted average cost of capital (WACC) imposed on the project by the capital markets. The cash flow projections used for the valuation have been prepared on an all-equity basis. This being the case, WACC is equal to the market cost of equity, and can be determined using the Capital Asset Pricing Model (CAPM):

where $E(R_i)$ is the expected return, or the cost of equity. R_f is the risk-free rate (usually taken to be the real rate on long-term government bonds), $E(R_m)$-R_f is the market premium for equity (commonly estimated to be around 5%), and beta (β) is the volatility of the returns for the relevant sector of the market compared to the market as a whole.

Figure 22.1 illustrates the real return on Canadian long bonds computed by the Bank of Canada, taken as a proxy for the risk-free interest rate. Recently, this has dropped from around 2.0% to less than 0.5%. Nevertheless, it is generally accepted that using a long-term average rate will give a more reliable estimate of the cost of equity. Micon has therefore used a value of 2.0% for the base case risk free rate, close to the real rate of return averaged over 10-years.

Figure 22.1
Real Return on Canadian Long Bonds
(Source: Bank of Canada)



Given that the mineral resource estimate for the GRS carries less uncertainty than is seen in many mining projects, Micon considers a beta value of 1.0 (i.e., equal to the average for the equity market as a whole) to be appropriate for the base case. Sensitivity of the project is tested using a range from 0.7 (typical for some gold producers) to 1.3 (for the mining industry in general). Thus, CAPM gives an estimated cost of equity for the Project of between 5% and 9%, as shown in Table 22.1. Micon has taken a figure of 7% (i.e., in the middle of this range) as its base case, and provides the results at alternative rates of discount for comparative purposes.

Table 22.1
Estimated Cost of Equity

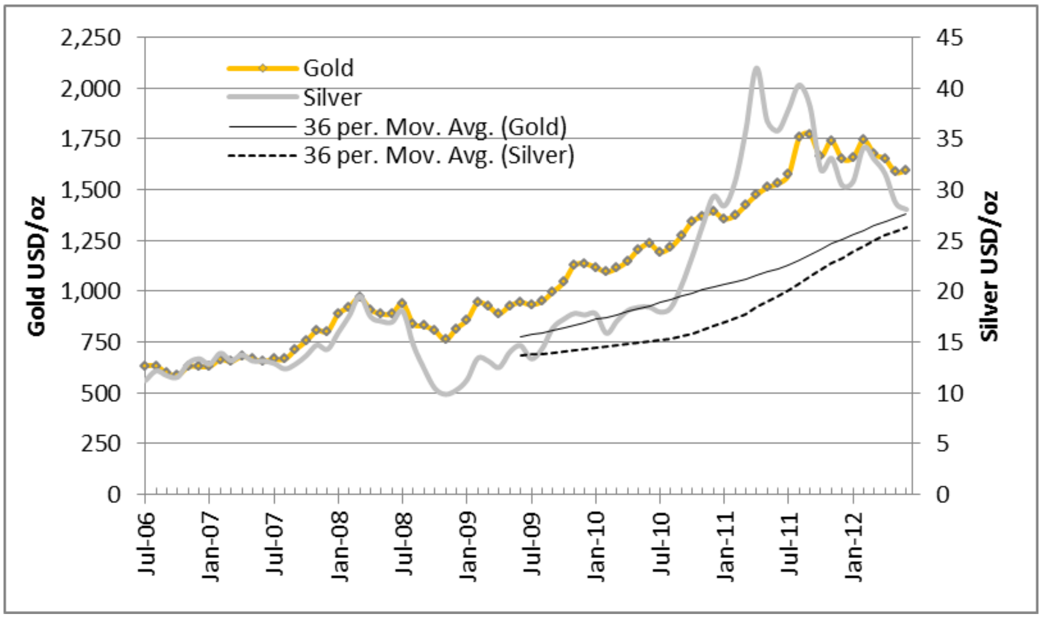
Range	Lower	Middle	Upper
Risk Free Rate (%)	1.5	2.0	2.5
Market Premium for equity (%)	5.0	5.0	5.0
Beta	0.7	1.0	1.3
Cost of equity (%)	**5.0**	**7.0**	**9.0**

22.2.3 Expected Metal Prices

Figure 22.2 shows the monthly average gold and silver prices over the past six years, together with the 3-year trailing averages. At the end of July, 2012, the three-year trailing averages for each metal were US$1,396/oz gold and US$26.67/oz silver, and these metal prices were selected for the base case. These prices were applied consistently throughout the operating period.

Silver contributes approximately 0.4% of the projected total revenue for the base case, so the impact of changing the silver price forecast is minimal.

Figure 22.2
Monthly Average Gold and Silver Prices since July 2006
(source: Kitco.com)



For comparison, Micon also evaluated the sensitivity of the project to using recent (1 month), and 1, 2, 3, 5 and 10-year price averages. The prices used in each of these cases are shown in Table 22.2. As part of its sensitivity analysis, Micon also tested a range of prices 30% above and below base case values.

Table 22.2
Metal Price Averages

Item	Units	1-month Jul-2012	1-year average	2-year average	3-year avg. Base Case	5-year average	10-year average
Gold	US$/oz	1,594	1,674	1,538	1,396	1,182	825
Silver	US$/oz	27.43	32.29	31.38	26.67	21.68	14.84

22.2.4 Taxation Regime

Manitoba and Canadian federal income tax payable on the project has been forecast using rates, of 12% and 15% respectively. Micon expects that the Snow Lake Project will bedeemed to not be a mine, in which case the Manitoba mining tax is not applicable.

22.2.5 Royalty

A royalty of $5/oz gold is payable to the Town of Snow Lake in respect of production from the GRS. In addition, 2% of NSR value is due to the Province of Manitoba once initial capital been repaid. Both these royalties are provided for in the cash flow model.

22.2.6 Selling Expenses

Sales revenue derived from the product shipped is based on a yield (i.e., gold recovery by the purchaser) of 88.6%, and payability factor of 99%. Transport and treatment charges of $25/wet metric tonne and $125/dry metric tonne, respectively, are applied to concentrate material shipped.

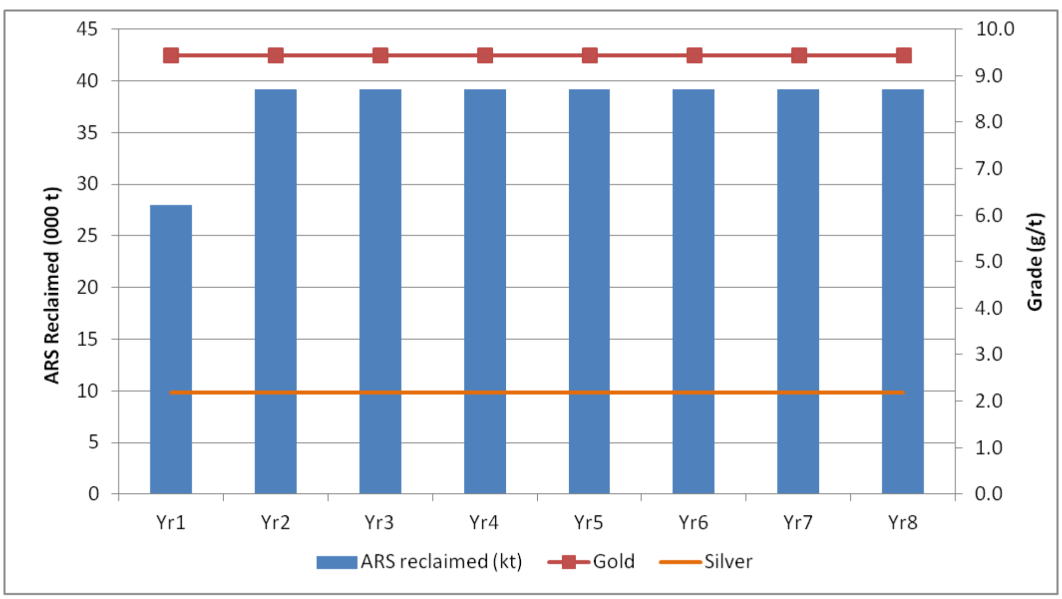
22.3 TECHNICAL ASSUMPTIONS

The technical parameters, production forecasts and estimates described elsewhere in this report are reflected in the base case cash flow model. These inputs to the model are summarised below. The measures used in the study are metric except where, by convention, gold and silver content, production and sales are stated in troy ounces.

22.3.1 GRS Production Schedule

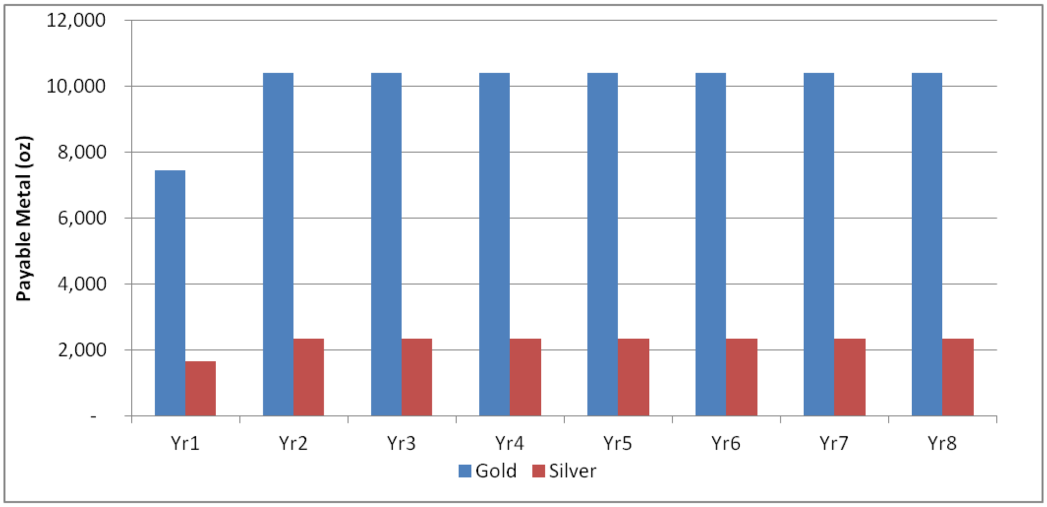
Figure 22.3 shows the annual tonnage of GRS material reclaimed, as well as the gold and silver grades of that material. Any material reclaimed is assumed treated in the same year, notwithstanding temporary stockpiling that may take place at the new process plant site.

Figure 22.3
Annual GRS Reclamation Schedule



As a consequence of steady tonnage, grade and recovery from process feed, annual production of gold and silver remain steady over the LOM period (Figure 22.4).

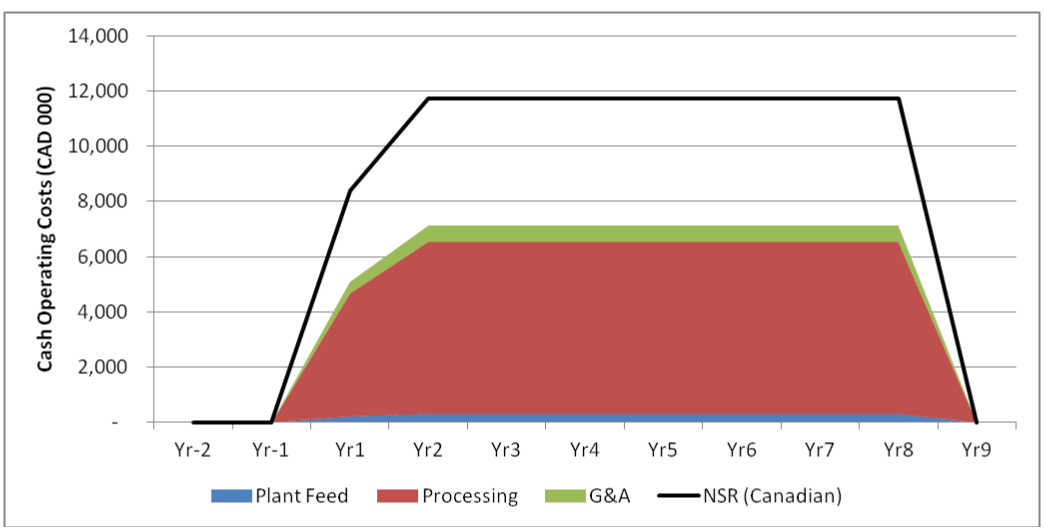
Figure 22.4
Annual Production Schedule



22.3.2 Operating Costs

Direct operating costs average $181.79/t milled over the LOM period, comprised of $7.91/t GRS reclamation, $158.78/t processing, and $15.10/t general and administrative costs. Figure 22.5 shows these expenditures over the LOM period, compared to the net sales revenue, showing positive operating margin maintained over the LOM period.

Figure 22.5
Direct Operating Costs



22.3.3 Capital Costs

Pre-production capital expenditures are estimated to total US$21.4 million, including $10.0 million for plant feed and processing, $4.2 million indirect costs, $4.4 million in owner's costs and contingencies totalling $2.8 million.

Working capital has been estimated to include 15 days product inventory, and 15 days receivables from despatch of concentrate. Stores provision is for 60 days of consumables and spares inventory, less 30 days accounts payable. An average of $1.2 million of working capital is required over the LOM period.

22.3.4 Base Case Cash Flow

The LOM base case project cash flow is presented in Table 22.3 and Figure 22.6.

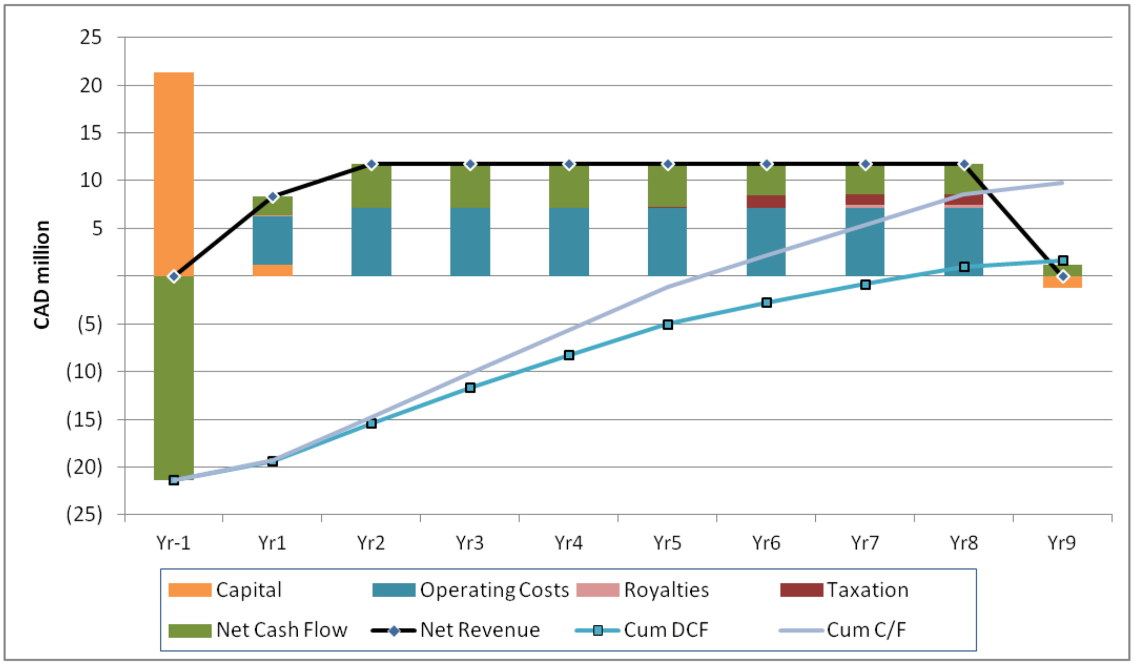
Table 22.3
Life-of-Mine Cash Flow Summary

	CAD 000	CAD/t	US$/oz Au
Gross Revenue (Gold)	114,411	378.47	1,396.00
Operating Costs			
Mining costs	2,393	7.91	29.19
Processing costs	47,999	158.78	585.67
General & Administrative costs	4,564	15.10	55.69
Direct operating cost	54,956	181.79	670.55
TC/RC (Gold)	24,272	80.29	296.16
less NSR (By-products)	(386)	(1.28)	(4.71)
Cash operating cost	78,842	260.81	961.99
Royalty	879	2.91	10.73
Total cash cost	79,721	263.71	972.72
Net Operating Margin	34,691	114.76	423.28
Capital Expenditure	21,356	70.64	260.57
Pre-tax Cash Flow	13,335	44.11	162.71
Taxation	3,600	11.91	43.93
Net Cash Flow After Tax	9,734	32.20	118.78

On the pre-tax, undiscounted cash flow, payback occurs at 4.9 years.

Payback on the undiscounted cash flow occurs at the end of Year 5, leaving approximately 3 years of the LOM period remaining. On a discounted basis, payback occurs at 7.2 years, less than one year before the end of the life of mine period.

Figure 22.6
Life-of-Mine Cash Flows



Annual cash flows are presented in Table 22.4.

22.3.5 Base Case Evaluation

The base case evaluates to an IRR of 11.1% before taxes and 8.9% after tax. At a discount rate of 7.0%, the net present value (NPV_7) of the cash flow is \$3.9 million before tax and \$1.6 million after tax.

Table 22.5 presents the results in Canadian dollars at comparative annual discount rates of 5%, 7% and 9%.

Table 22.4
Base Case Life of Mine Annual Cash Flow

Production Forecast			LOM TOTAL	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9
Processing Plant Production	**tonnes 000**		**302.3**	**-**	**28.0**	**39.2**	**39.2**	**39.2**	**39.2**	**39.2**	**39.2**	**39.2**	**-**
Gold (g/t)	average	g/t	9.435	-	9.435	9.435	9.435	9.435	9.435	9.435	9.435	9.435	-
Silver (g/t)	average	g/t	2.191	-	2.191	2.191	2.191	2.191	2.191	2.191	2.191	2.191	-
Gold	contained	kg	2,852	-	264	370	370	370	370	370	370	370	-
Silver	contained	kg	662	-	61	86	86	86	86	86	86	86	-
Concentrate shipped													
Concentrate	dry basis	t 000	151.15		14.00	19.59	19.59	19.59	19.59	19.59	19.59	19.59	-
	wet basis	t 000	160.80		14.89	20.84	20.84	20.84	20.84	20.84	20.84	20.84	-
Gold	average	g/t	18.85	-	18.85	18.85	18.85	18.85	18.85	18.85	18.85	18.85	-
Silver	average	g/t	4.38	-	4.38	4.38	4.38	4.38	4.38	4.38	4.38	4.38	-
Gold	Recov/Payable	%	87.7%		87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	
Silver	Recov/Payable	%	79.7%		79.7%	79.7%	79.7%	79.7%	79.7%	79.7%	79.7%	79.7%	
Gold		oz	80,350	-	7,442	10,415	10,415	10,415	10,415	10,415	10,415	10,415	-
Silver		oz	18,025	-	1,669	2,337	2,337	2,337	2,337	2,337	2,337	2,337	-

Cash Flow Forecast (CAD 000)	CAD/t treated	USD/oz gold	TOTAL CAD 000	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9
Gross Revenue	**378.47**	**1,396.00**	**114,411**	**-**	**10,596**	**14,831**	**14,831**	**14,831**	**14,831**	**14,831**	**14,831**	**14,831**	**-**
Operating Costs													
Mining	7.91	29.19	2,393	-	222	310	310	310	310	310	310	310	-
Processing	158.78	585.67	47,999	-	4,446	6,222	6,222	6,222	6,222	6,222	6,222	6,222	-
G&A	15.10	55.69	4,564	-	423	592	592	592	592	592	592	592	-
S/T Direct operating cost	181.79	670.55	54,956	-	5,090	7,124	7,124	7,124	7,124	7,124	7,124	7,124	-
TC/RC (Gold)	80.29	296.16	24,272	-	2,248	3,146	3,146	3,146	3,146	3,146	3,146	3,146	-
less NSR (By-products)	(1.28)	(4.71)	(386)	-	(36)	(50)	(50)	(50)	(50)	(50)	(50)	(50)	-
S/T Cash operating cost	260.81	961.99	78,842	-	7,302	10,220	10,220	10,220	10,220	10,220	10,220	10,220	-
Royalty	2.91	10.73	879	-	38	53	53	53	53	53	288	288	-
Total cash cost	**263.71**	**972.72**	**79,721**	-	7,340	10,273	10,273	10,273	10,273	10,273	10,508	10,508	-
Operating Margin	114.76	423.28	34,691	-	3,256	4,558	4,558	4,558	4,558	4,558	4,323	4,323	-
Capital Costs	**70.64**	**260.57**	**21,356**	**21,356**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Loading equipment capital	0.33	1.22	100	100	-	-	-	-	-	-	-	-	-
Processing Capital	32.74	120.75	9,896	9,896	-	-	-	-	-	-	-	-	-
Indirect Capital	28.25	104.21	8,541	8,541	-	-	-	-	-	-	-	-	-
Contingency	9.32	34.39	2,818	2,818	-	-	-	-	-	-	-	-	-
Change in Working Cap	(0.00)	(0.00)	(0)	-	1,192	-	-	-	-	-	-	-	(1,192)
Pre-tax c/flow	**44.11**	**162.71**	**13,335**	**(21,356)**	**2,065**	**4,558**	**4,558**	**4,558**	**4,558**	**4,558**	**4,323**	**4,323**	**1,192**
Tax payable	11.91	43.93	3,600	-	-	-	-	-	35	1,231	1,167	1,167	-
C/flow after tax	**32.20**	**118.78**	**9,734**	**(21,356)**	**2,065**	**4,558**	**4,558**	**4,558**	**4,522**	**3,327**	**3,156**	**3,156**	**1,192**
Cumulative C/Flow				(21,356)	(19,291)	(14,733)	(10,176)	(5,618)	(1,096)	2,231	5,387	8,543	9,734
Discounted C/Flow (7%)	5.44	20.05	1,644	(21,356)	1,930	3,981	3,720	3,477	3,224	2,217	1,965	1,837	648
Cumulative DCF				(21,356)	(19,426)	(15,445)	(11,725)	(8,248)	(5,024)	(2,807)	(841)	995	1,644
Max funding rqmt to positive cashflow			(22,547)	(21,356)	**(22,547)**	(19,291)	(14,733)	(10,176)	(5,653)	(2,326)	-	-	-

Table 22.5
Base Case Cash Flow Evaluation

CAD 000	IRR	Undiscounted	Discounted		
	%		5%	7% (base)	9%
Gross Revenue (Gold)		114,411	91,821	84,601	78,201
Operating Costs					
Mining costs		2,393	1,920	1,769	1,635
Processing costs		47,999	38,522	35,493	32,808
General & Administrative costs		4,564	3,663	3,375	3,119
Direct operating cost		54,956	44,105	40,637	37,563
TC/RC (Gold)		24,272	19,480	17,948	16,590
less NSR (By-products)		(386)	(310)	(286)	(264)
Cash operating cost		78,842	63,275	58,299	53,889
Royalty		879	655	586	526
Total cash cost		79,721	63,929	58,885	54,415
Net Operating Margin		34,691	27,892	25,716	23,786
Capital Expenditure		21,356	21,722	21,821	21,900
Pre-tax Cash Flow	**11.1**	**13,335**	**6,170**	**3,895**	**1,886**
Taxation		3,600	2,566	2,251	1,981
Net Cash Flow After Tax	**8.9**	**9,734**	**3,604**	**1,644**	**(96)**

The PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the PEA will be realized.

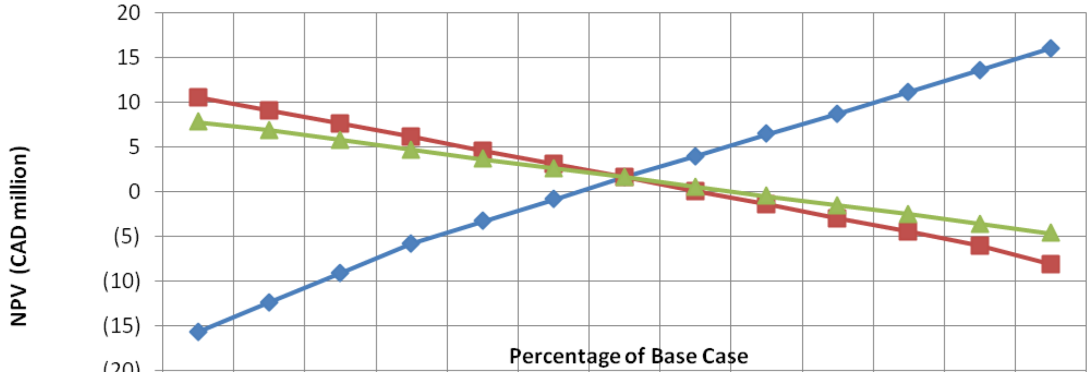
22.4 SENSITIVITY STUDY

22.4.1 Capital, Operating Costs and Revenue Sensitivity

The sensitivity of project returns to changes in all revenue factors (including grades, recoveries, prices and exchange rate assumptions) together with capital and operating costs was tested over a range of 30% above and below base case values. The results show that the project is most sensitive to revenue factors, with an adverse change of less than 5% reducing after-tax NPV_7 from $1.6 million to zero. The impact of changing operating costs is somewhat less, with an adverse change of more than 5% required in order to reduce NPV_7 to zero. The project is least sensitive to capital costs, with an increase of approximately 7% in capital reducing NPV_7 to zero.

Figure 22.7 shows the results of changes in each factor separately.

Figure 22.7
Sensitivity Diagram

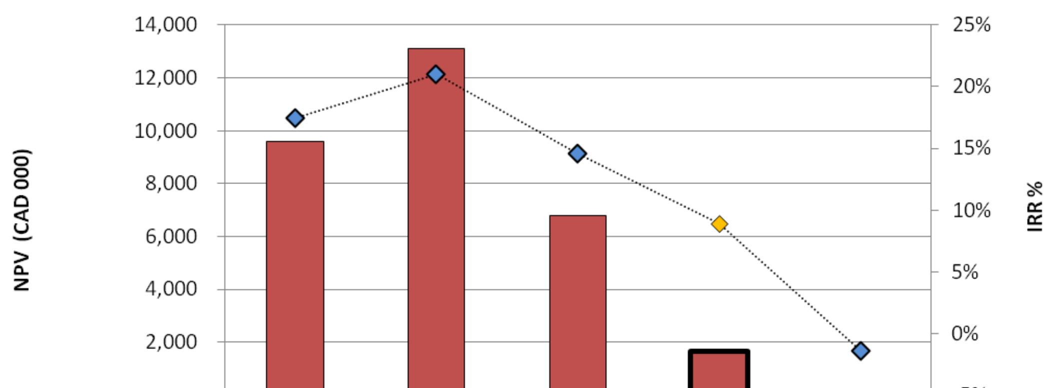


	70	75	80	85	90	95	100	105	110	115	120	125	130
Product Price	(15.7)	(12.4)	(9.1)	(5.8)	(3.3)	(0.9)	1.6	4.0	6.5	8.8	11.2	13.6	16.1
Opcosts	10.6	9.1	7.7	6.2	4.6	3.1	1.6	0.1	(1.4)	(3.0)	(4.4)	(6.1)	(8.1)
Capex	7.8	6.9	5.8	4.7	3.7	2.6	1.6	0.6	(0.5)	(1.6)	(2.5)	(3.6)	(4.7)

22.4.2 Metal Price Sensitivity

The sensitivity of the project to variation in gold price was tested using 1 month, and, 1, 2, 3 and 5-year trailing averages applied over the life-of-mine period. Figure 22.8 shows the impact of metal prices on after-tax NPV and IRR, while Table 22.6 shows these results pre- and post-tax.

Figure 22.8
Sensitivity to Metal Prices



	Jul-12	1-yr trailing	2-yr trailing	3-yr trailing	5-yr trailing
NPV	9,595	13,116	6,797	1,644	(7,078)
IRR	17.5%	21.0%	14.6%	8.9%	-1.4%

Table 22.6
Sensitivity to Metal Prices

	CAD 000	Jul-12 average	1-yr trailing	2-yr trailing	3-yr trailing	5-yr trailing
Average Gold Price (US$/oz)		1,594.00	1,674.00	1,538.00	1,396.00	1,182.00
Total cash cost (US$/oz)		989.03	989.62	990.54	**972.72**	946.70
Pre-Tax	Undiscounted	27,740	34,184	22,632	**13,335**	(1,504)
	NPV_5	17,710	22,865	13,620	**6,170**	(5,740)
	NPV_7	14,522	19,266	10,756	**3,895**	(7,078)
	NPV_9	11,703	16,084	8,224	**1,886**	(8,257)
	IRR (%)	21.2	25.3	17.8	**11.1**	-1.4
	Undisc. P/B	3.5	3.1	3.9	**4.9**	7.7
After-Tax	Undiscounted	20,250	24,954	16,522	**9,734**	(1,504)
	NPV_5	12,177	15,984	9,155	**3,604**	(5,740)
	NPV_7	9,595	13,116	6,797	**1,644**	(7,078)
	NPV_9	7,304	10,572	4,705	**(96)**	(8,257)
	IRR (%)	17.5	21.0	14.6	**8.9**	-1.4
	Undisc. P/B	3.6	3.2	4.0	**5.0**	7.7
	Discounted P/B	4.6	4.0	5.3	**7.2**	7.7

These results demonstrate that the project is potentially profitable at recent metal prices, although it remains economically marginal when using longer-term average prices.

22.5 CONCLUSION

Micon concludes that, with a continuation of the trend towards higher gold prices over the past five years, this study demonstrates the potential viability of the project as proposed, and that further development is warranted.

23.0 ADJACENT PROPERTIES

BacTech does not own any mineral property related to this GRS deposit. Rather it has purchased the right to reclaim this surface deposit. Therefore there are no related adjacent properties relevant in this case.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional relevant data available at this time.

25.0 INTERPRETATION AND CONCLUSIONS

This PEA Study is based on the proposed reclamation and processing of the Snow Lake GRS deposit and its measured, indicated and inferred mineral resources previously defined by N. R. Newson in an estimate reported in July 2011.

Based on its economic evaluation of the base case and sensitivity studies, Micon concludes that this PEA demonstrates the viability of the BacTech project as proposed, and that further development is warranted. The mineral resource estimate on which the PEA is based is on surface and fixed in size.

25.1 MINERAL RESOURCES

The PEA is based on the proposed mining and processing of the combined measured, indicated and inferred mineral resources as defined in Section 14. Mineral resources that are not mineral reserves do not have demonstrated economic viability. CIM (2010) defines a mineral reserve as the economically mineable part of a Measured or Indicated mineral resource demonstrated by at least a preliminary feasibility study. No mineral reserves have been estimated for the Snow Lake GRS deposit.

The mineral resource estimate is compliant with the current CIM standards and definitions as required under NI 43-101 and is, therefore, reportable as a mineral resource by BacTech. However, the reader should be cautioned that mineral resources are not mineral reserves and do not have demonstrated economic viability.

The stated mineral resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, unless stated in this report. There are no known mining, metallurgical, infrastructure or other factors that materially affect this mineral resource estimate.

The mineral resource reported above has been estimated. There are no indications of exploration potential for the currently defined deposit.

25.2 MINERAL PROCESSING

The PEA considers one possible process flowsheet for the processing of GRS deposit to produce a product amenable to cyanidation process for the extraction of gold. Nevertheless, Micon considers additional testwork, including optimizing grind, reagent strengths and retention times are required, and additional pilot studies are required before flowsheet development and design specifications can be finalized.

Total gold recoveries, based on existing metallurgical test work, are expected to be approximately 88.6% going to the concentrator (with 99.0% payable).

25.3 INFRASTRUCTURE AND CAPITAL COSTS

Infrastructure required for the project has been identified and is provided for in the evaluation. A site-specific layout has not been developed, though, pending discussion with town of Snow Lake over the siting of these works.

Mineral resources for the Snow Lake GRS deposit comprise measured and indicated resources of 274,300 t grading 9.7 g/t Au, and 2.17 g/t Ag, and an inferred resource of 28,000 t grading 7.0 g/t Au, and 2.4 g/t Ag.

A mine plan has been developed using the combined measured, indicated and inferred resources.

The PEA Study is based on the following:

- The Snow Lake GRS mineralization will be extracted using standard back hoe reclamation methods.
- Nominal throughput rate of 39,200 t/y ore.
- The life of the operating mine is approximately 7.7 years.
- BACOX bio-leaching technology will be used to produce a single gold and silver bearing product amenable to gold and silver recovery by cyanidation.
- Estimated life-of-mine payable gold recovery of 87.7% and silver recovery of 79.7% upon toll-processing and precious metals recovery.
- Production of an 18.5 g/t Au and 4.3 g/t Ag product.
- All major facilities (including the mill) will be located in close proximity to the Snow Lake GRS deposit.
- All neutralised precipitate will be safely contained and stored in close proximity to the processing facility.
- Access to site will be via an all-season road maintained by the town of Snow Lake.
- Electrical power will be provided by the town of Snow Lake grid available at the mine site property boundary.

The results of the study are summarized in Table 25.1. All dollars are Canadian dollars.

Table 25.1
Summary of the PEA Study Base Case Results

Item	Unit	Value
Total life-of-mine ARS reclaimed	kt	302.3
Average Au grade	g/t	9.435
Average Ag grade	g/t	2.191
Average Au recovery/payability factor	%	87.62
Average Ag recovery/payability factor	%	84.66
Annual payable Au production (average)	oz (000's)	10.4
Annual payable Ag production (average)	oz (000's)	2.3
Life of the mine	Years	7.7
Average cash operating cost	$/t milled	181.79
Average base case Au price	US$/oz	1,396.00
Average base case Ag price	US$/oz	26.67
Total Cash Cost	US$/oz	972.72
LOM net revenue (NSR)	$000	90,256
LOM Royalties	$000	879
LOM operating cost	$000	54,956
Pre-production capital cost	$000	21,356
Sustaining capital	$000	nil
Project cash flow before tax	$000	13,335
Pre-tax NPV @ 5.0 % discount rate	$000	6,170
Pre-tax NPV @ 7.0 % discount rate	$000	3,895
Pre-tax NPV @ 9.0 % discount rate	$000	1,886
Pre-tax IRR	%	11.1
Project cash flow after tax	$000	9,734
After-tax NPV @ 5.0 % discount rate	$000	3,604
After-tax NPV @ 7.0% discount rate	$000	1,644
After-tax NPV @ 9.0 % discount rate	$000	(96)
After-tax IRR	%	8.9

The PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the PEA will be realized.

Sensitivity analyses indicate that the project returns are most sensitive to revenue factors, with a 5% adverse change resulting in a negative NPV_7. The project is slightly less sensitive to capital and operating costs, with an adverse change of around 7% required to produce a negative NPV_7.

25.4 RISKS AND OPPORTUNITIES

Micon has assigned a level of confidence to individual key parameters as high, medium or low with a corresponding risk assessment as low, medium or high, as summarized in Table 25.2.

Table 25.2
Snow Lake Project, Risk Assessment

Subject or Technical Area	Confidence Level	Risk Level
Mineral and Surface Rights	High	Low
Geology	Medium to High	Low
Resources	Medium to High	Low to Medium
Geotechnical	Low to Medium	Low
Mining/Reclamation	Medium	Low
Metallurgical Testing	Medium	Medium
Plant Design	Medium	Medium
Utilities and Services	Medium to High	Medium
Surface Infrastructure	Medium to High	Low
Logistics (Climate, Access and Roads)	Medium to High	Low to Medium
Environmental	Medium	Medium to High
Capital Costs	Medium	Medium to High
Operating Costs	Medium	Medium to High
Economic Assessment	Medium	Medium to High
Socio/Governmental Consultations	Medium to High	Medium to High
Overall	**Medium**	**Low to Medium**

Overall the project is considered to be of medium risk. Work is continuing in several areas, including environmental and infrastructure components.

Opportunities exist in several areas:

- Infrastructure development synergies with other stakeholders.
- Infrastructure synergies with development of other projects in the area, including nearby abandoned tailings deposits.
- Potential infrastructure and service synergies with other companies operating in the region.
- Local employment, training and development.
- Metallurgical testwork to clarify reagent consumption rates.

26.0 RECOMMENDATIONS

It is recommended that BacTech continues to develop the project. During the Pre-Feasibility Study the following areas of work should be considered:

- Geotechnical evaluation, particularly for process plant and building construction.
- Continue planned stakeholder engagement.
- Continue with preparation of environmental and social impact studies to meet provincial, federal and international standards.
- Conduct additional mineralogical studies.

The proposed budget for further project development is presented below:

Process Optimizsation Testwork	$75,000	Q4 2012
EAP-Golder Study and Application Process	$65,000	Q4 2012
Geotechnical Report	$25,000	Q4 2012
Process Optimisation Testwork – Phase 2	$80,000	Q2 2013
Closure Plan	$65,000	Q2 2013
Front End Engineering Design (FEED)	$300,000	Q2 2013

27.0 DATE AND SIGNATURE PAGE

MICON INTERNATIONAL LIMITED

"Christopher Jacobs" {signed and sealed}

Christopher Jacobs, CEng MIMMM
Mineral Economist

Effective Date: 27 August, 2012
Report Date: 11 October, 2012

"Bogdan Damjanović" {signed and sealed}

Bogdan Damjanović, P.Eng.
Senior Metallurgist

Effective Date: 27 August, 2012
Report Date: 11 October, 2012

"N. Ralph Newson" {Signed and sealed}

N. Ralph Newson, M.Sc., P.Eng., P.Geo.

Effective Date: 27 August, 2012
Report Date: 11 October, 2012

"Barnard Foo" {signed and sealed}

Barnard Foo, M.Eng., P.Eng., MBA.
Senior Mining Engineer

Effective Date: 27 August, 2012
Report Date: 11 October, 2012

28.0 REFERENCES

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29.0 CERTIFICATES

CERTIFICATE OF AUTHOR

Christopher Jacobs

As co-author of this report entitled "NI 43-101 Preliminary Economic Assessment Study for the Bactech Snow Lake Reclamation Project, Snow Lake, Manitoba, Canada", with an effective date of 27 August, 2012 (the "Technical Report"), I, Christopher Jacobs, do hereby certify that:

1. I am employed as a mineral economist by, and carried out this assignment for:
   Micon International Limited, Suite 900 – 390 Bay Street, Toronto, ON, M5H 2Y2
   tel. (416) 362-5135 email: cjacobs@micon-international.com
2. I hold the following academic qualifications:
   B.Sc. (Hons) Geochemistry, University of Reading, 1980;
   M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.
3. I am a Chartered Engineer registered with the Engineering Council of the U.K. (registration number 369178);

   Also, I am a professional member in good standing of: The Institute of Materials, Minerals and Mining; and The Canadian Institute of Mining, Metallurgy and Petroleum (Member);
4. I have worked in the minerals industry for 30 years; my work experience includes 10 years as an exploration and mining geologist on gold, platinum, copper/nickel and chromite deposits; 10 years as a technical/operations manager in both open pit and underground mines; 3 years as strategic (mine) planning manager and the remainder as an independent consultant when I have worked on a variety of precious and base metal deposits;
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101;
6. I have not visited the property that is the subject of the Technical Report;
7. I am responsible for the preparation of Section 22, and the portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
8. I am independent of BacTech Environmental Corporation, as defined in Section 1.5 of NI 43-101;
9. I have had no previous involvement with the property;
10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 11th day of October, 2012

"Christopher Jacobs" {signed and sealed}

Christopher Jacobs, CEng MIMMM

micon INTERNATIONAL LIMITED | mineral industry consultants

CERTIFICATE OF AUTHOR

Bogdan Damjanović

As co-author of this report entitled "A Preliminary Economic Assessment of the Main Zone, J&L Deposit Revelstoke B.C., Canada", with an effective date of October 11, 2012 (the "Technical Report"), I, Bogdan Damjanović, do hereby certify that:

1. I am employed as a metallurgist by, and carried out this assignment for:
   Micon International Limited, Suite 900 – 390 Bay Street, Toronto, ON, M5H 2Y2
   tel. (416) 362-5135 email: bdamjanovic@micon-international.com
2. I hold the following academic qualifications:
   B.A.Sc., Geological and Mineral Engineering, University of Toronto, 1992
3. I am a Professional Engineer registered with the Professional Engineers of Ontario. (registration number 90420456);

   Also, I am a professional member in good standing of: The Canadian Institute of Mining, Metallurgy and Petroleum (Member);
4. I have worked in the minerals industry for 20 years; my work experience includes 8 years as a metallurgist on gold, copper/nickel and lead/zinc/gold deposits; and the remainder as an independent consultant when I have worked on a variety of precious and base metal deposits;
5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101;
6. I visited the Snow Lake property on January 16-18, 2012;
7. I am responsible for the preparation of Section 13, 17, 18, 19, 20, 23, 24 and the portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
8. I am independent of BacTech Environmental Inc., as defined in Section 1.5 of NI 43-101;
9. I have had no previous involvement with the property;
10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;
11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 11th day of October, 2012

"Bogdan Damjanović" {signed and sealed}

Bogdan Damjanović, P.Eng.

CERTIFICATE OF AUTHOR

Norman Ralph Newson

I, Norman Ralph Newson, of 205- 3 Apple Street, Brockville, Ontario, do hereby certify as follows:

1. That his certificate applies to a report titled *BacTech Environmental Corporation, NI43-1 01 Preliminary Economic Assessment Study for the Bactech Snow Lake Reclamation Project, Snow Lake, Manitoba, Canada,* dated effective August 27, 2012.

2. That I am responsible for sections 4.0 to 12.0 inclusive and sections 14.0 and 15.0 of the report identified above, and that these sections have an effective date of July 20, 2011.

3. That I am a graduate geologist, with B.Sc. and M.Sc. degrees from Queen's University at Kingston, Ontario, received in 1964 and 1970 respectively. I have practised my profession continuously since receiving my undergraduate degree, except for the time spent on course and thesis work for my graduate degree.

4. That my qualifications to write a report of this nature derive not only from my academic qualifications, but from increasingly responsible positions in the mining industry, including senior management. I have used the resource estimation methods described herein to estimate reserves in producing mines. My consulting career includes giving advice on metal content of a tailings pond.

5. That I am a Member of the Association of Professional Engineers & Geoscientists of Saskatchewan (with Permission to Consult), a Licensee of the Association of Professional Engineers & Geoscientists of New Brunswick, and a Member of the Association of Professional Engineers & Geoscientists of Manitoba.

6. That I visited the property from May 11 to 13 inclusive, 2011.

7. That I am independent of BacTech Environmental Corporation as described in section 1.5 of the Instrument.

8. That I have had no prior involvement with the subject property of this report.

9. That I believe I am a “qualified person” as defined in the Instrument. I have read the Instrument, and believe that the report has been prepared in compliance with it and with Form 43-101 F1.

10. That, as of the effective date of this report, to the best of my knowledge, information and belief, this report contains all scientific and technical information that is required to be disclosed to make the report not misleading.

The effective date of the parts of this report for which I am responsible is July 20, 2011. Signed at Brockville, Ontario, October 10, 2012.

“N. Ralph Newson” {Signed and sealed}

N. Ralph Newson, M.Sc., P.Eng., P.Geo.

CERTIFICATE OF AUTHOR

Barnard Y. Foo

As co-author of this report entitled "NI 43-101 Preliminary Economic Assessment Study for the Bactech Snow Lake Reclamation Project, Snow Lake, Manitoba, Canada", with an effective date of 27 August, 2012 (the "Technical Report"), I, Barnard Foo, do hereby certify that:

12. I am employed as a senior mining engineer by, and carried out this assignment for:
Micon International Limited,
Suite 205 – 700 West Pender, Vancouver, BC, V6C 1G8
Tel. (604) 647-6463 Email: bfoo@micon-international.com

13. I hold the following academic qualifications:
    - Laurentian University, B.Eng., Mining Engineering 1998
    - University of British Columbia, M. Eng., Rock Mechanics 2007
    - University of Northern British Columbia, Executive MBA 2010

14. I am a registered Professional Engineers of Ontario (registration number 100052925);

15. I have worked as a mining engineer in the minerals industry for 14 years;

16. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 4 years as an mining engineer in cassiterite, base and precious metal deposits, 5 years in underground and open pit geotechnical engineering and 5 years with in mine design and mining project evaluations for the mineral industry;

17. I have not visited the property that is the subject of the Technical Report;

18. I am responsible for the review of Sections 16 and the portions of Sections 1.10, and 26.0 summarized therefrom, of the Technical Report.

19. I am independent of BacTech Environmental Corporation, as defined in Section 1.5 of NI 43-101;

20. I have had no previous involvement with the property;

21. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;

22. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 11th day of October, 2012

"Barnard Foo" {signed and sealed}

Barnard Foo, M.Eng., P.Eng., MBA.
Senior Mining Engineer