Report to:

BUCHANS MINERALS CORPORATION

Preliminary Economic Assessment on the Lundberg and Engine House Deposits, Newfoundland, Canada

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TETRA TECH WARDROP

Report to:

PRELIMINARY ECONOMIC ASSESSMENT ON THE LUNDBERG AND ENGINE HOUSE DEPOSITS, NEWFOUNDLAND, CANADA

EFFECTIVE DATE: AUGUST 11, 2011

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GLOSSARY

UNITS OF MEASURE

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz

Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min

Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

ABBREVIATIONS AND ACRONYMS

Acid Rock Drainage	ARD
American Smelting and Refining Company	ASARCO
Anglo-Newfoundland Development	ANDC
Atomic Absorption	AA
Barite	$BaSO_4$
Barium	Ba
Billiton Resources Canada Inc.	Billiton
Buchans Minerals Corp.	BMC
Buchans Property	the Property
Buchans River Joint Venture	BRJV
Buchans River Ltd.	Buchans River
Canadian Environmental Assessment Act	CEEA

Canadian Environmental Assessment	CEA
Canadian Institute of Mining and Metallurgy	CIM
Concensus Economic Energy and Metal Forecast Group	EMCF
Contaminant of Concern	COC
Copper	Cu
Department of Fisheries and Oceans Canada	DFO
Electromagnetic	EM
Energy Reduction Credits	ERC
Environmental Baseline Study	EBS
Environmental Impact Assessment	EIA
Environmental Impact Statement	EIS
Fleet Production and Cost	FPC
G.T. Exploration Inc.	GT
General & Administrative	G&A
Global Positioning System	GPS
Gold	Au
Harmful Alteration, Disruption or Destruction	HADD
Health, Safety and the Environment	HSE
Impact Benefit Agreement	IBA
Induced Polarization	IP
Inductively Coupled Plasma Optical Emission Spectroscopy	ICPOES
Inductively Coupled Plasma/Mass Spectroscopy	ICP/MS
Internal Rate of Return	IRR
Inverse Distance Squared	ID^2
Lead	Pb
Lerchs-Grossman	LG
Life of Mine	LOM
Lock Cycle Test #1	LCT1
Lock Cycle Test #2	LCT2
Lock Cycle Test #3	LCT3
Major Project Management Office	MPMO
Mercator Geological Services Limited	Mercator
Metal Leeching	ML
Metal Mining Effluent Regulations	MMER
National Instrument 43-101	NI 43-101
National Topographic System	NTS
Nearest Neighbour	NN
Net Present Value	NPV
Net Smelter Return	NSR
Newfoundland and Labrador Hydro	NLH
Newfoundland and Labrador	NL
Preliminary Economic Assessment	PEA
Present Value	PV
Quality Assurance/Quality Control	QA/QC
Royal Roads Corporation	Royal Roads
SGS Lakefield	SGS

Silver	Ag
specific gravity	SG
Tailings Management Facility	TMF
Tailings Pond 1	TP1
Tailings Pond 2	TP2
Teck Resources Ltd.	Teck
Tetra Tech Wardrop	Tetra Tech
Uranium-Lead	U-Pb
Volcanogenic Massive Sulphide	VMS
X-ray Fluorescence	XRF
Zinc Equivalent Recovered	ZnEQR
Zinc Equivalent	ZnEQ
Zinc	Zn

1.0 SUMMARY

Buchans Minerals Corporation (BMC) is a Canadian exploration company focused on developing base metal properties and deposits in the Buchans mining camp in central Newfoundland.

This report is a National Instrument 43-101 (NI 43-101) Preliminary Economic Assessment (PEA) on the Lundberg and Engine House deposits, in central Newfoundland, Canada, prepared by Tetra Tech Wardrop (Tetra Tech).

The Lundberg and Engine House base metal deposits are situated at the former Lucky Strike mine, located approximately 90 km west of Grand Falls, Newfoundland and Labrador (NL) and 330 km northwest of St. John’s, the provincial capital of NL. The two deposits are located within the Buchans Property (the Property) comprised of 512 claims totalling 12,800 ha. Within the Property the Lundberg and Engine House base metal deposits are situated within Mineral Licence 10551M consisting of 215 mineral claims for a total area of 5,375 ha. The Property is located at latitude 48°49’ N and longitude 56°52’ W. The mineral rights to the Property are 100% held by Buchans River Ltd. (Buchans River), a wholly owned subsidiary of BMC.

This PEA is based on a NI 43-101 compliant resource estimate prepared by Mercator Geological Services Limited (Mercator), based in Halifax, Nova Scotia, on November 3, 2008.

1.1 DISCLOSURE

The PEA is based only on Inferred Mineral Resources and not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is therefore no certainty that the conclusions of the PEA will be realized.

1.2 GEOLOGY

Stockwork mineralization is typically associated with in situ ore and the best example is the Lundberg deposit. The Lundberg deposit sits stratigraphically below the historically mined Lucky Strike orebody and consists of quartz-barite-carbonate-sulphide veins and veinlets cutting strongly altered mafic volcanics with disseminated sulphide mineralization. The Lucky Strike orebody consisted of massive high-grade sulphides where American Smelting and Refining Company (ASARCO) mined 5.6 Mt of ore with head grades averaging 18.4% zinc (Zn%), 8.6% lead (Pb%), 1.6% copper

(Cu%), 112 g/t silver (Ag), and 1.7 g/t gold (Au) (calculated based on Thurlow and Swanson 1981, pages 122 to 128). The stockwork mineralization comes to surface on the eastern edge of the zone and forms an elongate, wedge shaped body that is 350 m deep on the western end. The highest concentration of sulphide mineralization appears to be proximal to the Lucky Strike orebody and more diffuse away from the historic workings. Mineralization typically occurs as fine to coarse grained euhedral pyrite as the dominant sulphide and occurs with varying amounts of chalcopyrite, sphalerite, galena and barite.

A second zone of stockwork mineralization is associated with the Engine House deposit immediately south of Lucky Strike and typically has a greater proportion of chalcopyrite. The Engine House deposit sits immediately south of the Lundberg deposit.

Drilling completed by BMC, then as Royal Roads Corporation (Royal Roads) and Buchans River included 53 drill holes drilled from surface totalling 8,058 m of drilling. Mercator compiled analytical data for zinc, lead, copper, silver, gold, and barite from this recent drilling in addition to historical assay results from previous drilling on the property. Results of this work resulted in an Inferred Mineral Resource Estimate that is considered compliant with disclosure requirements of National Instrument 43-101 (NI 43-101) as well as the Canadian Institute of Mining and Metallurgy (CIM) Standards for Mineral Resources and Mineral Reserves: Definitions and Guidelines. In addition to the resource estimate, a calculation outlining the percentage of the resource tonnage that lies within 100 m from surface was undertaken. These numbers do not constitute or suggest the amenity of economics of the resource contained therein, but do offer insight into the spatial distribution of minerals within the current block model.

This resource estimate updates an earlier released Inferred Resource having an effective date of September 15, 2008 (PR#17-08 Sept 17, 2008) and incorporates more complete historic precious metal assay data compiled from historic drilling and assays, resulting in a nominal increase in the precious metal contents. The results of the resource estimate are as follows:

Table 1.1 Lundberg Inferred Resource Estimate - Zn% Threshold – November 3, 2008

Zn% Threshold	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$)%	Percentage of Tonnage within 100 m of Surface
1.00	15,690,000	1.96	0.83	0.38	3.17	6.57	0.08	2.36	61.79%
1.50	9,300,000	2.46	1.03	0.43	3.92	8.26	0.10	2.84	66.40%
2.00	5,340,000	3.02	1.25	0.49	4.76	10.27	0.12	3.47	70.62%
2.50	3,170,000	3.56	1.46	0.53	5.55	12.28	0.14	4.65	72.83%

table continues…

Zn% Threshold	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	Barite ($BaSO_4$)%	Percentage of Tonnage within 100 m of Surface
3.00	1,880,000	4.13	1.66	0.57	6.36	14.32	0.14	6.20	75.68%
3.50	1,090,000	4.79	1.93	0.62	7.34	16.46	0.15	8.64	81.35%

Table 1.2 Engine House Inferred Resource Estimate - Zn% Threshold - November 3, 2008

Zn% Threshold	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	$BaSO_4$%	Percentage of Tonnage within 100 m of Surface
1.00	890,000	2.37	0.95	0.96	4.28	11.29	0.15	4.40	58.73%
1.50	600,000	2.89	1.10	1.05	5.04	12.17	0.16	4.87	60.56%
2.00	370,000	3.62	1.27	0.97	5.86	12.71	0.19	5.51	60.40%
2.50	240,000	4.35	1.41	0.94	6.70	12.34	0.22	5.56	52.04%
3.00	190,000	4.77	1.50	0.93	7.20	12.32	0.23	5.63	56.35%
3.50	140,000	5.28	1.56	0.91	7.75	12.33	0.23	5.60	56.28%

In completion of the Lundberg and Engine House resource estimates, Mercator also tabulated an Inferred Mineral Resource on the Lundberg and Engine House deposits based on a 1% combined base metal grade cut-off (Zn%+Pb%+Cu%). This tabulation is intended to compare the overall volume and grade of ASARCO's historic resource calculation with the modeling parameters used in the Lundberg resource estimate described above. The results of the Mercator 1% combined base metal resource are as shown in Table 1.3.

Table 1.3 Lundberg Inferred Resource Estimate – 1% Combined Base Metal (Zn%+Pb%+Cu%) Threshold - November 3, 2008

Threshold (Zn%+Pb%+Cu%)	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag g/t	Au g/t	$BaSO_4$%
1.00	20,700,000	1.68	0.72	0.38	2.78	5.92	0.07	2.11

These resource estimates reflect a three-dimensional deposit block model developed by Mercator using Surpac© Version 6.0.3 deposit modeling software. Analytical results for 178 diamond drill holes were used to calculate the resource estimate in this model, of which 42 drill holes are from recent Company drilling, and 136 drill holes are from validated historic data. The model utilized 1 m down-hole assay composites individually calculated for Zn%, Pb%, Cu%, g/t Ag, (%), Pb (%), Cu (%), Ag (g/t), Au (g/t), and $BaSO_4$ (%) assay values.

Model blocks measured 5 m x 5 m x 5 m with sub-blocking at 2.5 m x 2.5 m x 2.5 m within the Lundberg solid, and 2.5 m x 2.5 m x 2.5 m with sub-blocking at 1.25 m x 1.25 m x 1.25 m for the Engine House solid. The model was constrained by individual wireframed solids representing resource estimates for the Lundberg and Engine House deposits respectively. The wireframes were based on geological sections, that reflect a minimum included grade of approximately 1% combined base metal (Zn%+Pb%+Cu%) with dilution considerations limited to a maximum of 40% of the overall drill hole intersection. No high-grade capping factors were applied to high-grade samples. Historical underground development was reviewed and, where information was available, was modeled into 3D solids. All resource block model volumes that lay within the underground workings solids were removed from the resource estimate after the grade interpolation process was completed, and were not reported as part of the final volume and grade calculations of the resource estimate.

Metal grades were assigned to the block model using inverse distance squared (ID^2) interpolation methodology with blocks being peripherally constrained by wireframe solids. The Lundberg solid incorporated two interpolation domains which were defined as north and south of gridline 7930N. Two unique interpolation ellipses were determined for these two domains. Major and minor axis parameters were selected based on continuity and distribution of metal grade and reflect geological characteristics of the mineralized zones. The Engine House estimate was based within a single interpolation domain, and used an isotropic model with a 75 m range. Results of 1,577 separate laboratory determinations of specific gravity (SG) were used in the block model. The mean SG value from within the resource solids of 2.88 g/cm^3 was assigned to blocks occurring within the Lundberg and Engine House models.

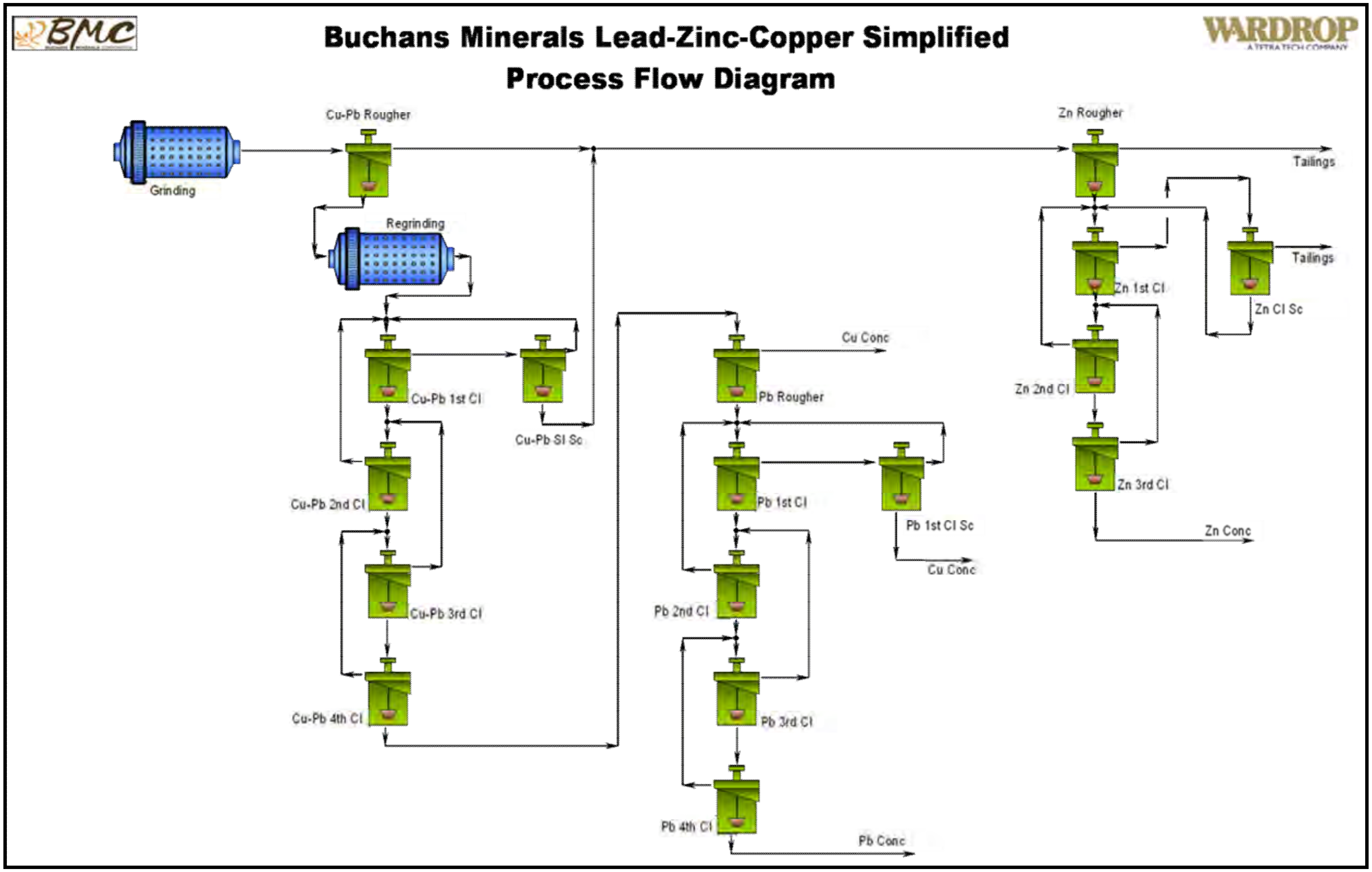
Potential to expand the deposit volume exists on the property and a Phase 1 drilling program of 2,500 m has been presented. The estimated Phase 1 budget for recommended exploratory core drilling peripheral to the Lundberg and Engine House deposits totals Cdn$630,000 and a Phase 2 program, contingent on positive Phase 1 results has a budget of Cdn$1,207,500 to upgrade Inferred Mineral Resources to the Indicated category and confirm historic drillhole data that exists underneath the Lucky Strike glory hole.

For the purposes of designing the open pit, Mercator had to modify their original NI 43-101 technical compliant resource model, (see section 16.2) such that the resource blocks were adjusted to be a consistent 5 m x 5 m x 5 m size throughout the resource model. The modified block model identified an Inferred Resource at a combined Zn-Pb-Cu cut-off of 1% of 22.21 million tonnes with average grades of 1.62% Zn, 0.69% Pb, 0.38% Cu, and 5.81g/t Ag.

1.3 Mineral Processing and Metallurgical Testing

The mineralization is a semi-massive sulphide of contained copper, lead and zinc sulphides, in association with a range of non-sulphide minerals. Metallurgical process development studies were conducted by SGS labs and were focused on base metals; copper-lead followed by a zinc flotation stage. The current test program was developed to determine the effectiveness of the proposed flowsheet shown in Figure 1.1 on two composite samples Met 2 and Met 4.

Figure 1.1 Buchans Lead-Zinc-Copper Simplified Process Flow Diagram



The head assays of the composite samples are shown in Table 1.4.

Table 1.4 Head Assays

Head Assays (Calculated)	Flotation Tests	Pb (Wt. %)	Cu (Wt. %)	Zn (Wt. %)	S (Wt. %)	Ag (g/t)
Met #2	LCT 1	1.91	0.42	3.76	12.7	9.22
Met #4	LCT 2	0.66	0.68	1.80	10.7	-
Met #4	LCT 3	0.64	0.65	1.75	10.0	8.8

The test work program undertaken at SGS labs was comprised of the following steps:

- Bond Work Index
- Heavy Liquid Separation
- Lock Cycle Test #1 (Met #2 sample)
- Lock Cycle Test #2 (Met #4 Sample)
- Lock Cycle Test #3 (Met #4 sample)

The metallurgical results of the three Lock Cycle Tests are presented below. All Lock Cycle Test showed varying degrees of stability even after 6 cycles.

Table 1.5 LCT 1 Results of Metallurgical Projection

Product	Weight		Assays (%) (g/t)						% Distribution					
	g	%	Pb	Cu	Zn	Au	Ag	S	Pb	Cu	Zn	Au	Ag	S
Pb 4th Cl Conc	592.5	3.00	59.1	1.92	11.6	0.28	135	20.1	93.1	13.7	9.3	7.9	44.1	4.8
Pb 1st Cl Sc Tail (Cu Conc)	132.3	0.67	2.79	22.0	9.33	0.52	47.0	35.0	1.0	35.0	1.7	3.3	3.4	1.8
Pb Ro Tail (Cu Conc)	154.8	0.78	2.60	14.0	25.5	1.21	77.0	32.8	1.1	26.1	5.3	9.1	6.5	2.0
Combined Cu Conc	287.0	1.45	2.69	17.7	18.0	0.89	63.2	33.8	2.0	61.1	7.0	12.3	10.0	3.9
Zn 3rd Cl Conc	1102.0	5.58	0.73	1.55	53.0	0.23	28.1	32.3	2.2	20.5	78.6	12.2	17.0	14.2
Zn 1st Cl Sc Tail	1799.5	9.11	0.13	0.06	0.34	0.17	7.3	12.4	0.6	1.3	0.8	14.3	7.2	8.9
Zn Ro Tail	15814.0	80.1	0.05	0.02	0.21	0.07	2.5	10.8	2.0	3.4	4.4	53.3	21.7	68.2
Zn Combined Tail	17613.5	89.2	0.06	0.02	0.22	0.08	3.0	11.0	2.7	4.7	5.2	67.6	28.9	77.1
Head (calculated)	19595.0	100.0	1.91	0.42	3.76	0.11	9.22	12.7	100.0	100.0	100.0	100.0	100.0	100.0

Table 1.6 LCT2 Results of Metallurgical Projection

Product	Weight		Assays (%)				% Distribution			
	g	%	Pb	Cu	Zn	S	Pb	Cu	Zn	S
Pb 4th Cl Conc	270.7	0.9	57.8	4.74	5.14	19.7	81.6	6.4	2.6	1.7
Pb 1st Cl Sc Tail (Cu Conc)	171.4	0.6	1.29	26.4	6.35	29.8	1.2	22.7	2.1	1.6
Pb Ro Tail (Cu Conc)	323.4	1.1	2.39	23.0	9.11	32.0	4.0	37.3	5.6	3.3
Combined Cu Conc	494.8	1.7	2.01	24.2	8.15	31.2	5.2	60.0	7.7	4.9
Zn 3rd Cl Conc	679.4	2.3	1.12	1.35	54.7	32.3	4.0	4.6	70.8	7.0
Zn 1st Cl Sc Tail	2150.2	7.3	0.26	2.09	0.29	17.8	2.9	22.6	1.2	12.2
Zn Ro Tail	25659.4	87.7	0.047	0.049	0.36	9.01	6.3	6.4	17.7	74.1
Zn Combined Tail	27809.6	95.1	0.064	0.21	0.36	9.69	9.2	29.0	18.9	86.3
Head (calculated)	29254.5	100.0	0.66	0.68	1.80	10.7	100.0	100.0	100.0	100.0

Table 1.7 LCT 3 Results of Metallurgical Projection

Product	Weight		Assays (%)					% Distribution				
	g	%	Pb	Cu	Zn	S	Ag	Pb	Cu	Zn	S	Ag
Pb 4th Cl Conc	214.4	0.7	73.9	1.03	4.55	16.1	359.5	78.7	1.1	1.8	1.1	27.8
Pb 1st Cl Sc Tail (Cu Conc)	276.3	0.9	2.28	26.5	5.20	33.6	45.2	3.1	35.4	2.6	2.9	4.5
Pb Ro Tail (Cu Conc)	491.5	1.6	1.04	22.7	6.23	34.0	33.7	2.5	53.9	5.5	5.3	6.0
Combined Cu Conc	767.8	2.4	1.48	24.1	5.86	33.8	37.8	5.6	89.3	8.1	8.2	10.5
Zn 3rd Cl Conc	690.4	2.2	1.33	1.96	53.0	32.6	33.0	4.6	6.6	66.2	7.1	8.2
Zn 1st Cl Sc Tail	2532.3	8.0	0.16	0.14	0.47	21.6	16.6	2.0	1.8	2.2	17.4	15.2
Zn Ro Tail	27442.0	86.7	0.045	0.041	0.37	7.82	3.87	6.1	5.4	18.5	68.1	38.3
Zn Combined Tail	29974.3	94.7	0.054	0.05	0.38	8.99	4.95	8.1	7.1	20.7	85.4	53.5
Head (calculated)	31646.9	100.0	0.64	0.65	1.75	10.0	8.8	100.0	100.0	100.0	100.0	100.0

One of the intentions of the test program is to derive a simple, operable, and reliable flowsheet with low circulating loads, and a simple reagent scheme to complement the process design. The estimated reagent consumption by category is outlined in Table 1.8.

Table 1.8 Reagent Consumption

Species	Consumption (g/t)
Collectors	
SIPX	42.5
3418A	22.5
Frothers	
U250/MIBC (50-50)	17.5
Modifiers	
Lime	1800
H_2SO_3	180
Activators and Depressants	
$CuSO_4$	250
NaCN	470
Other	
$ZnSO_4$	250
Activated Carbon	100

No deleterious components, in excessive amounts, that would incur smelter penalties have been identified.

1.4 MINING

The open pit design was optimized using the Lerchs-Grossman pit method, which was refined to a detail design with catch berms and in-pit ramps. Afterwards, mine development and production schedules were developed, mine equipment items were selected, the capital and operating costs were evaluated. Information pertaining to the 3D geological block model that was used in the optimization is provided in Section 16.2.

For this Study, Tetra Tech determined that the Lundberg/Engine House deposits were to be mined using conventional open pit mining methods. Both deposits are located close to surface. Section 16.3 outlines work completed by Tetra Tech in evaluating an underground mining option. Based on this analysis, the underground operation is not viable at this time.

The mine provides mill feed of resource at a rate of 5,000 t/d; beginning the first year of the mine life. The ultimate pit design for the selected base case pit contains 17.28 Mt of resource averaging a combined (arithmetic sum) metal grade of 2.72%

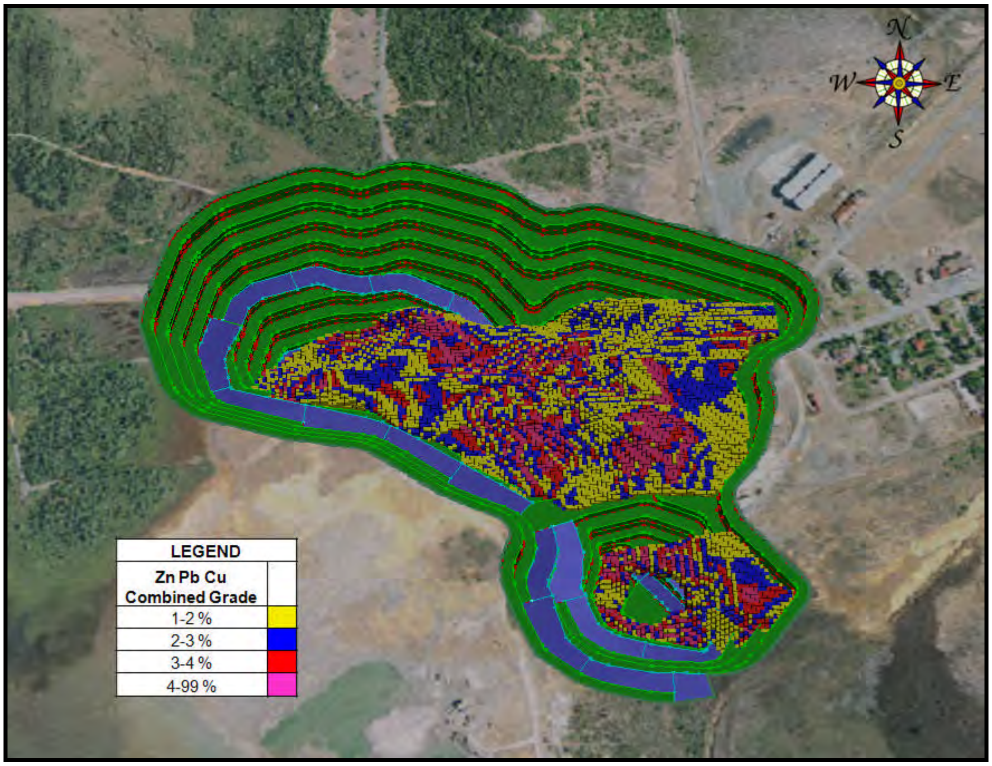
(Zn-Pb-Cu), using 1.0% cut-off grade. The overall stripping ratio is 3.06 t/t (waste/resource) and total of 52.93 Mt of waste material will be moved over the mine life of 10 years (Table 1.9).

The overall mining sequence was developed in three phases: one initial starter pit and two pushback phases. Due to this study being at PEA level, final pit designs for the starter pit and first pushback (phase II) were not completed. Whittle™ pit shells were used as interim pits and scheduling purposes. A detail design with catch berms and in-pit ramps was only refined for the final ultimate pit (Figure 1.2).

Table 1.9 Overall Ultimate Pit Production Schedule

Period (Years)	1	2	3	4	5	6	7	8	9	10	Total
Lundberg Waste (kt)	5,015	4,400	5,346	6,758	5,993	5,556	5,439	3,512	1,469	541	44,028
Engine House Waste (kt)	483	767	1,283	1,423	1,782	1,613	1,151	390	12		8,903
Total Waste (kt)	**5,497**	**5,167**	**6,628**	**8,180**	**7,776**	**7,169**	**6,590**	**3,902**	**1,481**	**541**	**52,931**
Lundberg Resource (kt)	1,669	1,694	1,732	1,643	1,702	1,694	1,540	1,536	1,714	1,529	16,452
Engine House Resource (kt)	81	56	18	107	48	56	210	214	36		827
Total Resource (kt)	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,529**	**17,279**
Zn (%)	2.51	1.83	2.25	1.16	1.51	1.48	1.33	1.45	1.36	1.34	1.63
Pb (%)	1.06	0.78	0.90	0.52	0.68	0.65	0.54	0.56	0.62	0.61	0.69
Cu (%)	0.57	0.48	0.47	0.48	0.40	0.28	0.36	0.35	0.34	0.24	0.40
Ba (%)	4.63	0.70	0.74	0.61	1.54	0.90	0.85	0.86	0.66	0.86	1.24
Ag (g/t)	10.14	5.21	8.51	4.14	6.01	4.55	5.09	5.59	4.99	5.23	5.96
Au (g/t)	0.10	0.07	0.10	0.04	0.06	0.04	0.04	0.08	0.11	0.05	0.07
Total Tonnage (kt)	**7,247**	**6,917**	**8,378**	**9,930**	**9,526**	**8,919**	**8,340**	**5,652**	**3,231**	**2,069**	**70,209**

Figure 1.2 3D Rendered Image of Ultimate Pit Design Including Block Model



Since the required geotechnical data is not available for determining the pit slope angle, Tetra Tech utilized an overall pit slope angle of 45°, based on conservative estimates from previous experience. As geotechnical data becomes available, pit slopes could potentially steepen and improve the mine plan and economic evaluation of study.

It is proposed that the operation will be carried out with an equipment fleet comprising a single, 251 mm diameter rotary blast hole drill rig for resource and waste, an 11 m^3 (bucket capacity) hydraulic face shovel with a fleet of 91 t haul trucks. These will be supplemented with support equipment of loader, grader, dozers, and a backhoe excavator, etc. The mine development will progress using 10m benches.

BMC has potential mining opportunities in close proximity to the Lundberg and Engine House deposit. The 100% BMC owned Daniels Pond deposit (located ~90km distant from Lundberg) currently has a NI43-101 compliant resource model and is a possible satellite mining operation which could provide additional ore feed to a potential Lundberg milling facility. BMC also has 100% ownership of the following massive sulphide exploration prospects in close proximity to Lundberg and Engine House; Buchans North, Clementine West and Little Sandy.

1.5 PROJECT INFRASTRUCTURE

1.5.1 *ELECTRICAL*

Electrical power is readily available in the area of the mine site. There is currently an 8.3 MVA transformer located 1 to 2 km outside of the town of Buchans. Currently 1 to 2 MVA is in use and there is deemed to be adequate capacity to serve the proposed mining operations. Other possibilities include using hydroelectric from nearby generation locations.

1.5.2 *WASTE ROCK*

The waste rock dump was sized to handle 22.3 Mm^3 of waste material over the 10 year mine life. This volume was calculated based on test data of the waste rock which indicated a unit density of 2.7 g/cm^3. A swell factor of 30% was used and is typical for the blasted rock.

1.5.3 *TAILINGS MANAGEMENT FACILITY*

The life of mine tailings storage capacity was based on the mine schedule of 5,000 t/d over a period of approximately 10 years. Based on experience from other similar base metal projects, an average tailings density of 1.6 g/cm^3 was used in the calculations. This equates to a total volume of 10.8 Mm^3 of tailings over the life of mine.

The existing historic tailings management facility (TMF) and downstream waterways have been impacted by mining and the area is considered a ‘brownfield’ site. This is one of the primary reasons for selecting an existing historic tailings pond as the location of future tailings deposition from the new open pit mine. In addition, there are no other green-field locations with favourable topography, storage capacity and proximity to the new mine location.

1.6 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

A mineral development project must be registered for environmental assessment through the Newfoundland and Labrador Department of Environment and Conservation under the provincial *Environmental Protection Act* and the *Environmental Assessment Regulations*. The project will be subject to the provincial environmental assessment process and may also be subject to an environmental assessment under the *Canadian Environmental Assessment Act* (CEAA) if an approval is required from a federal agency. Any requirement for a federal environmental assessment would be conducted in accordance with the Draft

Canada-Newfoundland and Labrador Agreement on Environmental Assessment Cooperation (2005).

No Environmental Baseline Studies (EBS) have been conducted specifically for the Project. EBS are necessary to understand the specific interactions between the project and the natural environment and to support the environmental assessment process. Baseline studies will need to focus on water and groundwater quality and quantity, fish and fish habitat, rare vegetation, and wildlife. In addition to the provincial environmental assessment process, other provincial permits will be required for operation of the facilities.

The project will involve at least one discharge of effluent that may originate from several sources on the project site, including: dewatering of the existing glory hole; open pit dewatering to remove groundwater seepage and precipitation; general site runoff including runoff from ore, waste rock, and overburden stockpiles; and, periodic releases of water from the tailings management area. A water treatment plant has been included in the Project design to ensure the quality of any project effluent can be managed to meet effluent criteria that will be applied to the project. Monitoring of any liquid discharge from the project to receiving waters will be required as part of any provincial environmental permit or approval. The basic monitoring requirements are those detailed in the *Metal Mining Effluent Regulations* (MMER), which require routine monitoring of deleterious substances. Neither the process water requirement for the mill or the water source has been determined at this time. When determined, water take from any natural surface water body will need to be licensed under the *Water Resources A*ct.

Waste rock will be stored in a valley area adjacent to the historic Tailings Pond 2. Some waste rock is expected to be acid generating, given this is a massive sulphide deposit, but the quantity of potentially acid generating waste rock has not yet been determined. The waste rock stockpile will incorporate a collection and containment system, ARD and metal leeching (ML) testing will be conducted in subsequent stages of project planning and design. Hydrogeological conditions in the vicinity of the open pit need to be described in order to estimate the potential for groundwater seepage into the pit, to design the necessary water diversion and water management works, and to assess how the project interactions with groundwater may affect nearby surface water bodies. Dewatering of the open pit and existing glory hole will be required over the course of the project and this also will be licensed under the *Water Resources A*ct.

Infrastructure from the existing mine site will be removed and disposed of as part of new project development, and existing buildings may contain asbestos. Old mine buildings that were insulated with asbestos may have been pushed into the glory hole, and in this situation these buildings will need to be removed and disposed of according to the *Asbestos Abatement Regulations,* the *Waste Material Disposal Act,* and the Asbestos Waste Disposal Directive.

A Development Plan and Rehabilitation and Closure Plan must be submitted prior to the commencement of project development, and typically should be developed at the same time as the environmental assessment for the project, in accordance with the guidelines to the provincial *Mining Act*. Financial assurance must be posted to secure the rehabilitation works, and needs to be sufficient to cover any continuing maintenance and monitoring that may be required.

The implementation of an effective Community and Aboriginal Engagement Program is fundamental to the successful environmental permitting of mining projects, and will ensure that all potentially affected persons, businesses, and communities have a full understanding of the Project and an opportunity to share information with respect to concerns regarding potential effects, and so the proponent has an opportunity to explain how these concerns are addressed in the Project design and operations. This program typically begins in the early stages of planning and continues through the life of the Project.

1.7 Capital and Operating Costs

The capital costs are calculated at Cdn$127 million (2011 base year). A 15% contingency has been applied to the direct capital costs. The indirect costs have been factored from the direct capital cost. The indirect costs have been calculated as 8% of the direct capital on an annual basis and the owner's cost has been calculated as 3% of the direct capital on an annual basis. Salvage value has been calculated as 10% of the direct capital costs for the following areas; open pit mining, processing, and non-process building.

The direct capital cost breakdown consists of the following major category descriptions:

- site development
- site utilities
- tailings management facility
- open pit mining
- processing facilities
- non-process buildings
- closure and reclamation costs.

The indirect capital costs breakdown consists of the following major category descriptions:

- owner's cost
- indirect cost

- contingency
- salvage
- direct and indirect capital costs.

The capital cost summary is presented below in Table 1.10.

Table 1.10 Capital Cost Summary

Item	Amount (Cdn$)
Direct Capital Costs	
Site Development	11,972,222
Utilities	8,940,883
Tailings Management Facilities	17,173,550
Open Pit Mining	22,675,003
Resource Processing	56,084,677
Non-process Buildings	3,760,000
Closure/Reclamation	6,600,000
Subtotal Direct Capital Costs	**127,206,336**
Indirect Capital Costs	
Indirect Costs	10,176,507
Owner's Costs	3,816,190
Contingency (15%)	19,080,950
Salvage	(8,251,968)
Subtotal Indirect Capital Costs	**24,821,679**
Total Capital Costs	**152,028,015**

1.8 Operating Costs

1.8.1 *Mining*

The total mine operating cost is estimated at Cdn$159,476,745 which equates to Cdn$2.27/t of material mined or Cdn$9.23/t of ore milled. The breakdown of the operating costs is shown in Table 1.11.

Table 1.11 Summary of Open Pit Operating Costs (x $1,000)

Item	Unit	Cost	%
Loading	Cdn$	10,128	6.35
Hauling	Cdn$	44,533	27.92
Drilling	Cdn$	6,882	4.32
Blasting	Cdn$	28,073	17.60
Roads & Dumps	Cdn$	20,871	13.09
Mine General	Cdn$	10,228	6.41

table continues…

Item	Unit	Cost	%
Miscellaneous	Cdn$	2,000	1.25
Ancillary Equipment	Cdn$	36,760	23.05
Total Mining Cost	**Cdn$**	**159,477**	**100**
Loading	Cdn$/t	0.14	6.35
Hauling	Cdn$/t	0.63	27.92
Drilling	Cdn$/t	0.10	4.32
Blasting	Cdn$/t	0.40	17.60
Roads & Dumps	Cdn$/t	0.30	13.09
Mine General	Cdn$/t	0.15	6.41
Miscellaneous	Cdn$/t	0.03	1.25
Ancillary Equipment	Cdn$/t	0.52	23.05
Unit Mining Cost	**Cdn$/t**	**2.27**	**100**

1.8.2 PROCESSING

Operating costs consist of labour, supplies and utilities. They are estimated to be Cdn$21.9 million per annum or Cdn$12.53 per tonne ore as shown in Table 1.12.

Table 1.12 Plant Operating Costs

Operating Cost Area	%	Total Annual Cost (Cdn$)	Cost (Cdn$/t)
Labour	20.37	4,465,250	2.55
Reagents and Consumables	51.729	11,337,834	6.48
Power and Utilities	27.91	6,117,035	3.50
Total Plant Operating Cost	100.00	21,920,119	12.53

1.8.3 GENERAL AND ADMINISTRATIVE COSTS

G & A costs represents cost incurred by the mine that is not directly associated with mine production or processing. Employee roles that are included in the G&A costs include management , financial, engineering ,geology support, purchasing, health, safety and the environment (HSE), and administration.

The G&A costs have been calculated to be $3,557,275 annually or $2.03/t of ore mined. The breakdown of the G&A costs is shown in Table 1.13.

Table 1.13 General and Administrative Operating Costs

Cost Centre	Costs/Year
Operational Costs	$2,578,775
Office Costs	$114,000
Legal/Professional Costs	$152,000
Mine Site Costs	$712,500
Total G&A Costs	$3,557,275
	$2.03/t

1.9 Economic Analysis

The financial analysis considered a total of 17.3 million tonnes of resource. The 17.3 million tonnes of resource are Inferred Resource from the Lundberg and Engine House deposits. The Net Smelter Return (NSR) values for the three concentrates are:

- Zn concentrate – US$845/dmt
- Cu concentrate – US$1699/dmt
- Pb concentrate – US$1574/dmt

Table 1.14 shows the Pre Tax Net Present Value (NPV) for the project at variable discount rates. As well the Internal Rate of Return (IRR) is shown as 43.94%.

Table 1.14 NPV and IRR

Item	Amount
Pre-tax & Pre-finance NPV @ 6%	$217,778,267
Pre-tax & Pre-finance NPV @ 8%	$186,377,690
Pre-tax & Pre-finance NPV @ 10%	$159,682,329
Pre-tax & Pre-finance NPV @ 12%	$136,873,842
Pre-tax & Pre-finance NPV @ 15%	$108,545,780
Pre-tax & Pre-finance NPV @ 20%	$73,093,623
Project IRR	43.94%

Table 1.15 details metal price used in the economic analysis for the Lundberg/Engine House PEA.

Table 1.15 Metal Prices

Metal	Metal Price	Units
Zinc	$1.22	US$/lb
Copper	$3.62	US$/lb
Lead	$1.10	US$/lb
Silver	$22.74	US$/oz

1.10 RECOMMENDATIONS AND OPPORTUNITIES

Recommended activities are described in section 26 for the following areas of study included in this report:

- Geology
- Mineral Processing
- Mining Operations
- Project Infrastructure
- Tailings Management Facility
- Environmental Planning and Assessment

A detailed list of required activities has been developed to bring the project to the completion of Prefeasibility Study level and Feasibility Study level. This list is summarized in Table 1.16.

Table 1.16 Prefeasibility and Feasibility Activities List

Prefeasibility Level Activity
Geology
PFS Permits and Licensing
PFS Geological Drilling
PFS Geological Assaying
PFS NI 43-101 Resource Model Update
Metallurgy
PFS Metallurgical Sample Prep
PFS Metallurgical Optimization Testing
Geotechnical
PFS Geodetic and Bathymetry Survey
PFS Site Field Investigation - Foundation and Tailings
PFS Tailings Characterization Test Work
PFS Borrow Studies

table continues…

Prefeasibility Level Activity
Environmental Studies
PFS Hydrogeology Study
PFS Surface Water Quality and Hydrology Study
PFS Community and Aboriginal Engagement
PFS Conduct Geochem (Acid Rock Drainage [ARD]) Test Work
Prefeasibility Report
PFS Study
PFS Financials and Report
Feasibility Level Activity
Geology
FS Permits and Licensing
FS Geological Drilling
FS Geological Assaying
FS NI 43-101 Resource Model Update
Metallurgy
FS Metallurgical Sample Prep
FS Metallurgical LCT Testing
FS Metallurgical Bulk Sample Variability Testing
Geotechnical
FS Site Investigation Foundation and Tailings
FS Geotechnical Core Holes and Analysis
FS Borrow Studies
FS Tailings Characterization Test Work
Environmental Studies
FS Surface Water Quality and Hydrology Study
FS Fish and Fish Habitat Assessment
FS Community and Aboriginal Engagement
FS Regulatory Submission
Feasibility Report
FS Study
FS Financials and Report

The cost for all activities for Prefeasibility and Feasibility Study level completion is estimated to be $3.6 million for Prefeasibility and $4.2 million for Feasibility. The cost summary is presented in Table 1.17.

Table 1.17 Prefeasibility and Feasibility Cost Breakdown

Activity	Cost ($Cdn)
Prefeasibility	
Geology	$2,250,000
Metallurgy	$207,500
Geotechnical	$115,000
Environmental Studies	$385,000
Prefeasibility Report	$675,000
Total for Prefeasibility	$3,632,500
Feasibility	
Geology	$1,127,500
Metallurgy	$307,500
Geotechnical	$500,000
Environmental Studies	$225,000
Feasibility Study Report	$2,000,000
Total for Feasibility	$4,160,000

2.0 INTRODUCTION

BMC is a junior mining company focused on developing its base metal properties and deposits in the Buchans mining camp in central Newfoundland.

This report is a NI 43-101 PEA on the Lundberg and Engine House deposits, in central Newfoundland, Canada, prepared by Tetra Tech.

The Lundberg and Engine House base metal deposits are situated at the former Lucky Strike mine, located approximately 90 km west of Grand Falls, NL and 330 km northwest of St. John's, the provincial capital of NL. The two deposits are located within the Property comprised of 512 claims totalling 12,800 ha. Within the Property the Lundberg and Engine House base metal deposits are situated within Mineral Licence 10551M consisting of 215 mineral claims for a total area of 5,375 ha. The Property is located at latitude 48°49' N and longitude 56°52' W. The mineral rights to the Property are 100% held by Buchans River, a wholly owned subsidiary of BMC and portions of the Property are subject to underlying net smelter royalties held by third parties.

The PEA is based only on Inferred Mineral Resources and are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is therefore no certainty that the conclusions of the PEA will be realized.

2.1 TERMS OF REFERENCE

This PEA is based on a NI 43-101 compliant resource estimate prepared by Mercator, based in Dartmouth, Nova Scotia, on November 3, 2008.

Tetra Tech has had no involvement on the Property prior to this PEA. Prior to this report, there have been two NI 43-101 compliant Technical Reports and resource estimates completed on the Property by Mercator.

The following PEA report conforms to the standards set out in NI 43-101, Standards and Disclosure for Mineral Projects and is in compliance with Form 43-101F1. The resource estimate in this report conforms to the CIM Mineral Resource and Mineral reserve definitions referred to in NI 43-101, Standards and Disclosure for Mineral Projects.

The designated Qualified Persons for this report are:

- Daniel Coley, P.Eng., Senior Metallurgist for Tetra Tech
- Daniel Gagnon, P.Eng., Senior Open Pit Engineer for Tetra Tech
- Mike McLaughlin, P.Eng., Project Manager for Tetra Tech
- Peter C. Webster, P.Geo., Senior Geologist for Mercator.
- Doug Ramsey, R.P. Bio., Manager – Environmental Assessment, Permitting, and Natural Resources for Tetra Tech.

Several site visits were conducted by both Tetra Tech and Mercator to the Lundberg and Engine House project site.

The initial site visit by Tetra Tech was completed by Mr. Gagnon between September 10 and 12, 2011. Mr. Gagnon was accompanied by Mr. Warren MacLeod, President and CEO for BMC; Mr. Paul Moore, Vice-President for BMC; and, Mr. Dennis MacLeod also of BMC.

A second site visit was conducted by Mr. Patrick MacKay between May 31 and June 3, 2011. Mr. MacKay was accompanied on site by Mr. W. McLeod and Mr. Moore of BMC.

In preparation for the resource estimate, Mr. Webster made several site visits to the Property between February 2007 and March 2008.

2.2 UNITS AND MEASURES

The unit and measures used in this report are metric and all units of cost are in Canadian dollars unless otherwise stated.

3.0 RELIANCE ON OTHER EXPERTS

Tetra Tech has relied upon others for information in this report. Tetra Tech has relied mainly on information, provided by BMC, found in the following independent NI 43-101 compliant technical report and resource estimate:

Webster, P.C. and Barr, P.J., 2008. Technical Report on the Mineral Resource Estimate for the Lundberg and Engine House Deposits, Buchans Area, Newfoundland, Canada; prepared for Royal Roads Corp. Effective Date: November 3rd, 2008. 3 November 2008. 2005. 90 pages.

Tetra Tech is relying on reports, opinions, and statements from experts who are not Qualified Persons for information concerning legal, environmental, political, or other issues and factors relevant to the technical report. Tetra Tech has not conducted an examination of land titles or mineral rights.

Information from third party sources is referenced in Section 21.0 – References. Tetra Tech used information from these sources under the assumption that the information is accurate.

The following excerpt has been taken from Webster and Barr (2008) with regards to the information for the resource estimate used in this PEA:

There has been no reliance on other experts in the preparation of this report.

This report was prepared by the author and Mercator staff for [Buchans], and the information and conclusions contained herein are based upon information available to Mercator at the time of report preparation. This includes data made available by both [BMC], and third party sources. Information contained in this report is believed reliable, but in part the report is based upon information not within Mercator's control. Mercator has no reason, however, to question the quality or validity of data used in this report. Comments and conclusions presented herein reflect Mercator's best judgment at the time of report preparation and are based upon information available at that time.

This report also expresses opinions regarding exploration and development potential for the project, and recommendations for further analysis. These opinions and recommendations are intended to serve as guidance for future development of the property, but should not be construed as a guarantee of success. Mercator is not a Qualified Person with respect to comments on environmental liability, validity of surface rights titles and other issues of land ownership in the province of NL.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Lundberg and Engine House base metal deposits are situated at the former Lucky Strike mine, located approximately 90 km west of Grand Falls, NL and 330 km northwest of St. John's, the provincial capital of NL. The two deposits are located within the Property comprised of 512 claims totalling 12,800 ha. Within the Property the Lundberg and Engine House base metal deposits are situated within Mineral Licence 10551M consisting of 215 mineral claims for a total area of 5,375 ha. The Property is located at latitude 48°49' N and longitude 56°52' W. The mineral rights to the Property are 100% held by Buchans River, a wholly owned subsidiary of BMC.

4.1 PROPERTY LOCATION

The Lundberg and Engine House deposits are located:

- within National Topographic System (NTS) 1:50,000 map sheet 12A15
- at approximately latitude 48°49' N and longitude 56°52' W, in central NL, eastern Canada
- at approximately 510,000 m E/5,407,900 m N (NAD83 Zone 21)
- approximately 330 km northwest of St. John's, the provincial capital of NL and approximately 90 km west of Grand Falls
- adjacent to the west of the town of Buchans
- bounded by Sandy Lake, 2 km to the north and by Red Indian Lake, 4 km to the south

The Property is situated as shown in Figure 4.1 and Figure 4.2.

Figure 4.1 Property Location Map

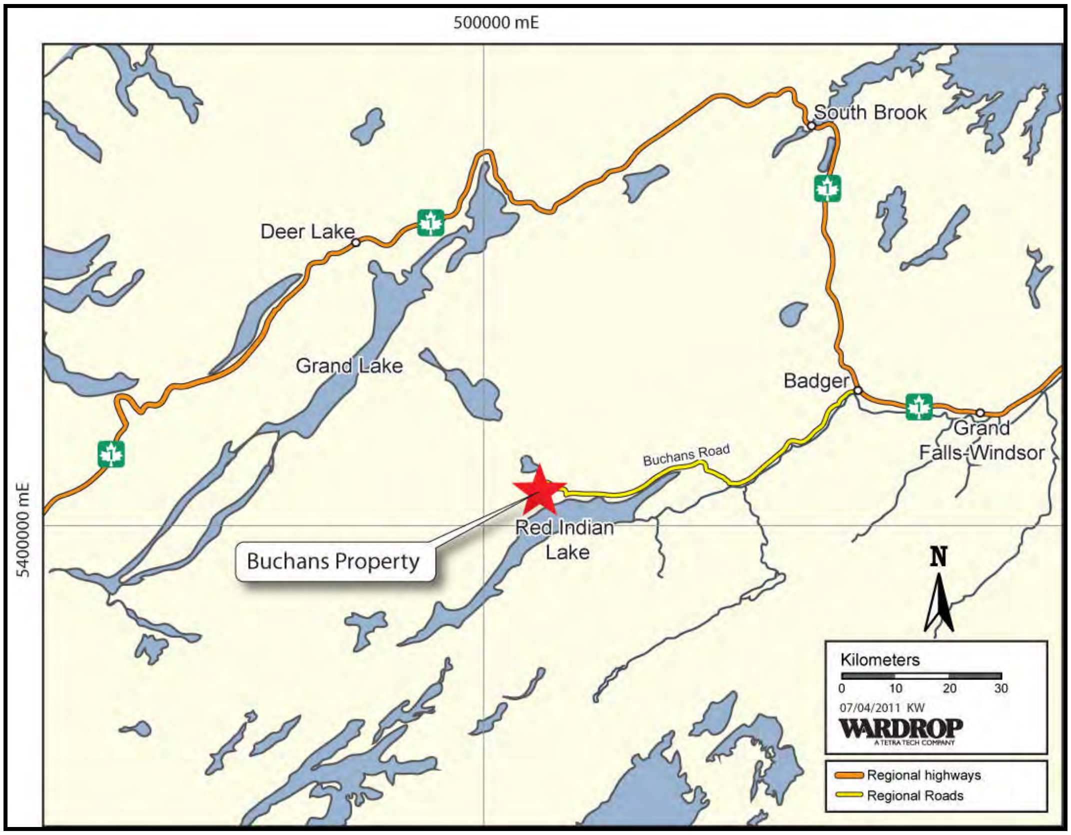
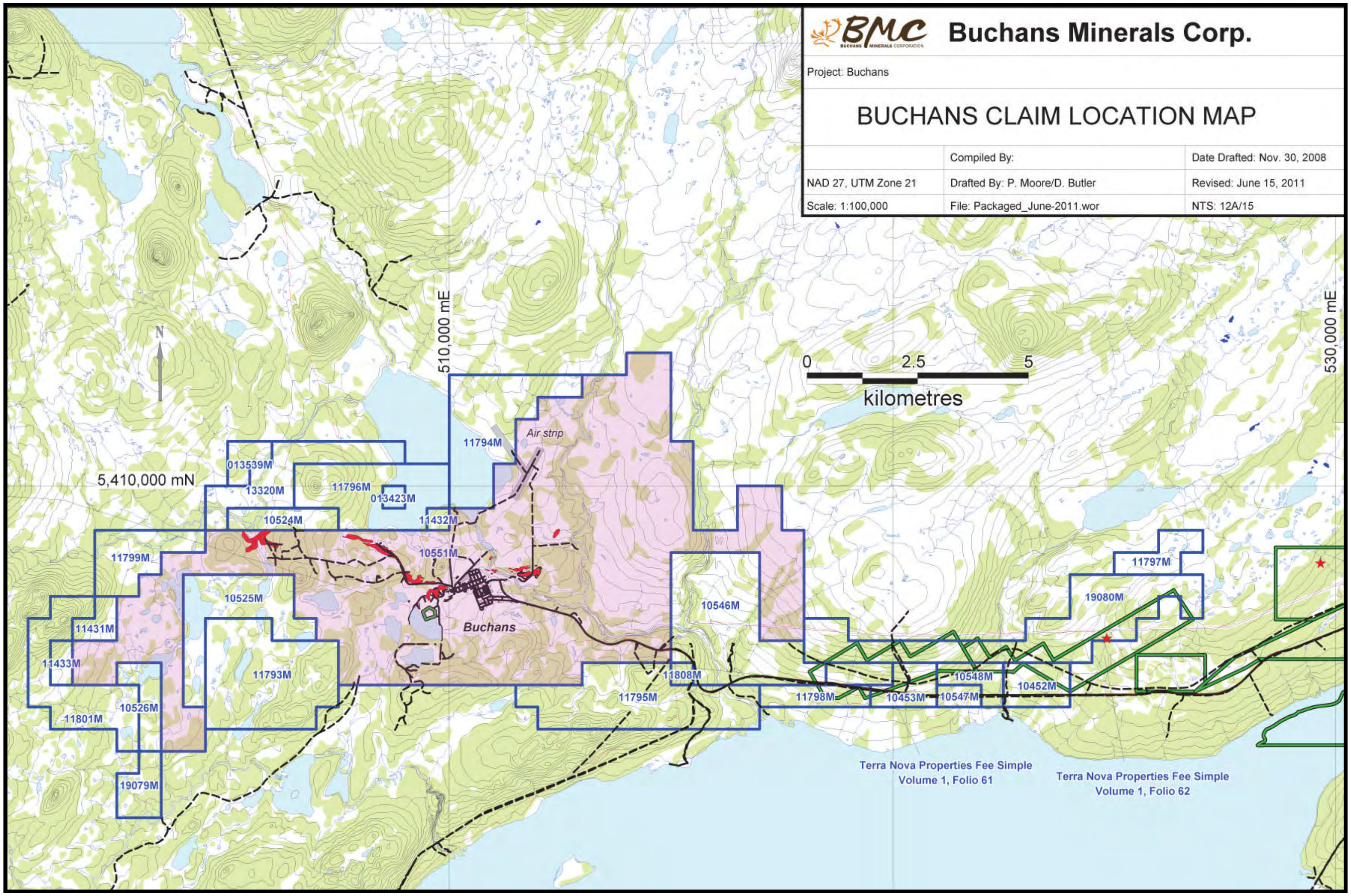


Figure 4.2 Claim Location Map



4.2 PROPERTY DESCRIPTION

The Lundberg and Engine House deposits are located within the Property comprised of 512 claims totalling 12,800 ha. Within the Property the Lundberg and Engine House base metal deposits are situated within Mineral Licence 10551M consisting of 215 mineral claims for a total area of 5,375 ha. . The mineral rights are 100% held by Buchans River, a wholly owned subsidiary of BMC. Table 4.1 summarizes the details of the Mineral Licence 10551M.

Table 4.1 Summary of Mineral Rights for the Property

Mineral Licence	Date Issued	Date of Expiry	Registered Owner	No. Of Claims	Area (ha)	NTS Mapsheet
10551M	1 Feb 1993	1 Feb 2013	Buchans River Ltd.	215	5,375	12A/15

Details on all mineral rights held by BMC in the Buchans areas may be found in Appendix A. Only the Mineral Licence 10551M is subject to this report. All other mineral rights are listed in the appendices for completeness. All claims are current and there are no outstanding issues with these claims.

4.3 ENCUMBRANCES AND AGREEMENTS

4.3.1 *WORK REQUIREMENTS*

Work requirements of the Newfoundland Government include an annual expenditure of $200 per claim in the first year, rising by $50 per claim until Year 5; then the cost is $600 per claim per year from Year 6 to Year 10, $900 per claim per year for Years 11 to 15, and $1,200 per claim per year for Years 16 to 20. The type of acceptable work for assessment purposes is defined in The Mineral Regulations 1983 of the Province of Newfoundland, and includes most conventional exploration survey methods (Webster and Barr, 2008)

4.3.2 *SURFACE RIGHTS*

As a licence holder, BMC has the exclusive right to explore for minerals within the boundaries of the mineral claim licences, but does not own the surface rights. However, the company has secured land access with surface right holders for the purpose of mineral exploration (Webster and Barr, 2008).

4.3.3 *OPTION AGREEMENTS*

Many of BMC's mineral rights in the Buchans area are held under option agreement through a number of previous licence holders. These properties are subject to NSR royalties, and these agreements are summarized in Table 4.2 (Webster and Barr, 2008).

Table 4.2 Summary of Option Agreements – Buchans Area

Current Licence	Licence at time of Agreement	Agreement	Company	Buchans River Obligations
010551M	4272, 4273	100% interest in property	CBM	3% NSR to CBM
	4875, 4317	100% interest in property	NME	2% NSR to NME
	4974M, 6973M, 4805, 4865, 4867-4869	100% transfer to Buchans River	GT	2% NSR to GT
	4823	100% interest to Buchans River	PD	2% NSR to PD
	4547, 4293, 4294, 4470, 4744, 4603	100% interest in property	NME	1% NSR to NME

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

The information found in this section is taken from Webster and Barr (2008).

5.1 ACCESSIBILITY

The Property is situated adjacent to the town of Buchans at the end of Route 370 (Buchans Highway). Route 370 is a 70 km long all-weather, well maintained, paved road which connects Buchans to the TransCanada Highway at the town of Badger. Badger is situated approximately 30 km west of Grand Falls by road.

The nearest airports are located at Gander or Deer Lake and access to the Property is along the TransCanada Highway and then Route 370. The distance from Deer Lake to the Property by road is approximately 250 km and from Gander, approximately 200 km. The distance by road to the Property from St. John's is approximately 530 km.

Much of the Property and Buchans area has been clear-cut in the past by Abitibi-Price (now AbitibiBowater) and this activity has led to the construction and refurbishment of a number of new and existing forestry roads in the area, permitting ready access to most of the Property.

5.2 CLIMATE

The climate of central Newfoundland is characterized as northern maritime, with relatively cool summers and winters with an overall annual average temperature of 3.5°C. The area receives an average annual precipitation of 873.3 mm of rain and 331 mm of snow, for a combined total average annual precipitation of 1,204.3 mm (data from Environment Canada, received at the Badger meteorological station).

5.3 LOCAL RESOURCES

The town of Buchans has a population of approximately 800 and some local resources available such as groceries, hardware and gas stations.

Field supplies, fuel and logistical support are available in Millertown or Buchans and contract geotechnical personnel including drill companies and analytical laboratories are available in either Grand Falls or Springdale.

5.4 INFRASTRUCTURE

The Property is connected by a paved, all-weather, road connecting the town of Buchans to the TransCanada Highway. The town of Buchans has electrical power, telephone and water services. There is cellular telephone coverage on the Property. The main power line from Grand Falls to Corner Brook passes through Buchans less than 2 km south of the proposed mine site.

Buchans currently has permission from the town of Buchans and the government of Newfoundland (surface rights) to conduct exploration activities within and adjacent to the town and surrounding areas.

Water is abundant on the Property.

The nearest major airports are at Gander and Deer Lake. There is a 4,000 ft gravel airstrip on the Property, 2 km north of the town of Buchans.

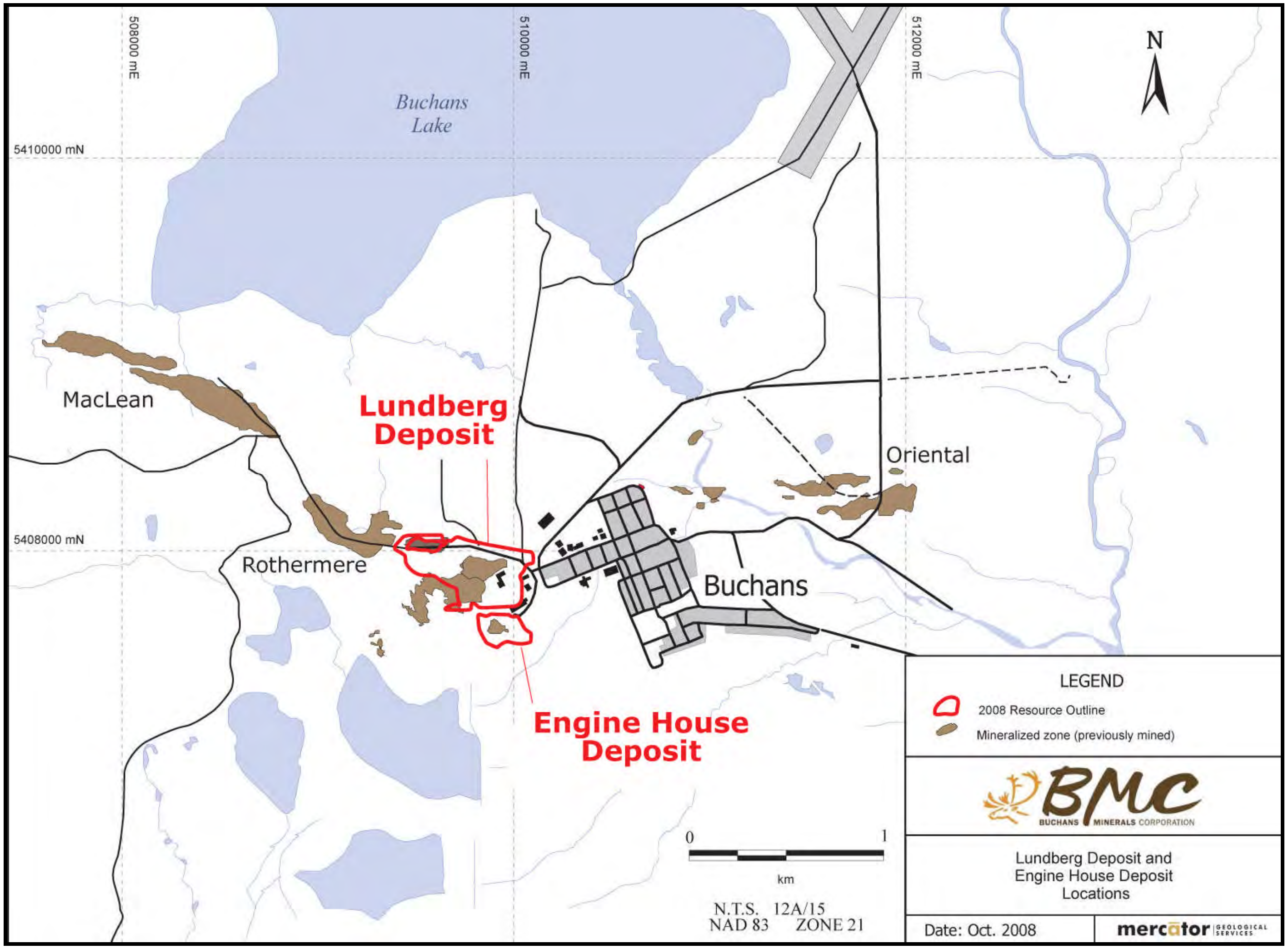
The closest deep-water ports are located in Botwood, 125 km northeast, and St. Georges, 160 km west, of the Property. The Botwood port was used as the loading terminus for the past-producing Buchans Mine, while St. Georges is currently used as the loading terminus for Teck Resources Ltd.'s (Teck) operating Duck Pond mine.

A core storage facility operated by the Newfoundland Government is available for use in Buchans. This facility is used by private exploration companies, and much of the core from historic drilling on the Buchans area properties and surrounding region is stored at this location. Viewing and re-sampling of core can be arranged under government supervision. Historic mine buildings and two large tailing ponds remain on the Property from past mining by ASARCO. The tailings ponds, however, are not permitted for use and are currently the responsibility of the provincial government.

5.5 PHYSIOGRAPHY

The Property is generally flat to gently rolling with elevation ranging from 155 m to 165 m at Red Indian Lake to approximately 130 m to 280 m inland. There are numerous small brooks which drain into Red Indian Lake with spruce and fir growing on the slopes. The northern portion of the Property is poorly drained and covered by areas of shallow bogs and extensive muskeg in the flat areas. The depth of till is approximately two metres with less than 5% outcrop exposure. To *the south of the Property, Red Indian Lake occupies a large northeast trending valley.*

Figure 5.1 Lundberg Deposit and Engine House Deposit Locations



6.0 HISTORY

The information found in this section is taken from Webster and Barr (2008).

The earliest report of lead-zinc mineralization in the Buchans area was in 1905 when Matty Mitchell a local Mi'kmaq Indian discovered a boulder with high-grade base metal mineralization just east of the current Buchans Mine site along the Buchans River shoreline. Anglo-Newfoundland Development Company (ANDC), who owned mineral rights in central Newfoundland including the Buchans area, formed a joint venture with ASARCO in 1926, which resulted in the Buchans Mine operating continuously from 1928 until the ore reserves were considered to be depleted in 1984 (Neary, 1981). In total, the Buchans orebodies have produced 16,196,876 tonnes of ore from the five known major orebodies. The average grades are reported to be 14.51% Zn, 7.65% Pb, 1.33% Cu, 126 g/t Ag, and 1.37 g/t Au (Thurlow, 1991).

The history of exploration is summarized as follows.

6.1 EARLY WORK (1911-1926)

In 1911, a report was commissioned by the New York firm Weed and Probert to assess the economic value of the Main (known) sulphide deposit, and to make recommendations for further exploration. In 1925, a geological examination was conducted simultaneously with an economic feasibility assessment by J.G. Baragwanath, a consulting engineer (Thurlow, 1991). It was stated that the area held great potential for finding further ore. In 1926, further exploration was conducted under the Swedish American Prospecting Company while the main orebody was being prepared for mining. Mr. Hans Lundberg perfected an exploration technique using an electrical survey called the equipotential line method and two anomalies were detected over the known deposits. Trenching and a small amount of diamond drilling were conducted over these areas. Dr W.H. Newhouse was commissioned to create a geological map of the area surrounding the mine in 1927. The expectation was that this map would help determine the source of the Pb-Zn boulders in Wileys River (Thurlow, 1991). Fieldwork, examination of the mine workings, and drill core logging was conducted over the next few years to facilitate the report. The result was a focus chiefly on the presence of quartz porphyry as an indicator of proximity of an orebody. Further exploration was recommended to the north of Wiley's River and in the Clementine Lake area. The manager of the mine changed these parameters in 1934, instead focusing on tracing mineralization, structural formations and hydrothermal alteration away from the known orebodies at Lucky Strike and Oriental. These tracings were conducted from surface and underground workings, *and resulted in the discovery of the south west extensions of the Lucky Strike deposit. Small scale spot drilling programs were also carried out in the region to the south and southwest of the Lucky Strike orebody.*

6.2 1928-1985 ASARCO

ASARCO commenced exploration of the Buchans Mine property in 1928. Between 1930 and 1984 extensive drilling programs produced more than 400 local surface and underground holes, leading to the discovery of most of the known mineralized zones and orebodies. Despite the scale of exploration, it was widely recognized that substantial gaps in the drill pattern could have overlooked additional targets. It was also recommended that shallow historic drill holes be extended; surface and underground holes during this period were drilled to varying depths, with many measuring only 200 m or less, and the deepest reaching about 1,100 m . All surface diamond drilling was vertical, with core sizes varying from 22 mm (EX core) to 47.6 mm (NQ core). Because much of the mineralization was previously defined as subeconomic, it was not sampled from drill core, and limited assay data is available from this period. The earlier ASARCO drilling program was closely spaced and concentrated primarily on the near surface equipotential anomalies outlined by Hans Lundberg resulting in the discovery of the Lucky Strike and Oriental orebodies. Later expansion of the program, by way of systematic outward extension of the drill sites, lead to the discovery of the Rothemere, Maclean and Maclean Extension orebodies in the mid 1940s (Swanson, 1981).

Gravity and Induced Polarization (IP) surveys were conducted primarily on the adjoining properties in the 1940s, and these yielded no new targets. In the 1960s, soil and till sampling southwest of the main Buchans property detected an anomalous trend believed to be derived from the Lucky Strike area, but at the time was not considered to be a significant enough indication to warrant further exploration.

6.3 1992-2001 GT Exploration Inc./Newminex/Buchans River Joint Venture

In 1992 GT Exploration Inc. (GT)/Newminex/Buchans River Joint Venture (BRJV) staked much of the former Buchans Mining Properties and through options and joint ventures assembled the current property holdings. Their exploration work began in 1997 with a re-logging program dedicated to re-interpreting the results of past drilling programs. The relogging was initiated by GT when it was realized that the effect of thrusting on the stratigraphy was not fully understood at the time of ASARCO's early diamond drilling. The majority of the surface drill holes in the Buchans Mine Property were relogged during the time period of 1997 to 2000. This program identified several potential targets not tested by the drilling, and *several drilled mineralized and altered zones were identified as new potential targets for further drilling. Finally, an updated geological database was compiled for use in conjunction with lithogeochemical and structural/stratigraphic studies.*

Billiton Resources Canada Inc. (Billiton), Buchans River Ltd., Newfoundland Mining & Exploration Ltd., and GT formed the BRJV in September of 1998. The agreement stipulated that Billiton would spend $3,500,000 on exploration, and earn 51% interest in all the claims held by other partners in the Buchans area. When the BRJV was terminated, in September 2001, Billiton did not retain or earn any interest in the property (Halpin, 2001).

The BRJV conducted an airborne electromagnetic (EM) survey in 1998, which indicated a faint anomaly in conductivity in the Clementine West region. The two holes drilled by ASARCO (2811 and 2813, which were flagged for the presence of possible ore horizon indicators) lie within the anomaly. To test this relationship hole BR-11-01 was drilled and promising results from its core lead to further IP, soil surveys, and more local diamond drilling.

A notable turning point in the exploration of this property occurred between 1997 and 2001 when litho-geochemical sampling allowed for the differentiation between footwall and hangingwall signatures in the ore horizon sequence. This development resulted in the modification of the previously interpreted stratigraphy, and allowed for prioritization of targets. Further, the new structural interpretation also suggested several new areas which may contain untested ore horizon sequences. Based on this work and the relogging of the ASARCO drilling, new targets were finalized and drilling began in 1999. Over the next two years significant new intersections of alteration and mineralization were discovered including Clementine West, the HAG Zone, Middle Branch, and the Airport Zone. Of particular interest was the area to the southeast of the Lucky Strike Mine known as the HAG Zone, which has a characteristic style of mineralization and many holes drilled intersected high-grade sulphide clasts hosted in shear zones, the source of which is still unknown. The presence of these zones appeared to confirm the idea of a new structural model, and further testing was recommended.

In 2000-2001 ERA-Maptec Ltd., a company based in Dublin, Ireland, conducted a structural reinterpretation of the Buchans mine site with on-site geologists working for Billiton. The study utilized the results of 3D modeling, with the goal of validating or developing the location of the ore horizon sequence on the existing structural model. The refining of the model saw the modification of the originally proposed anti-formal stack structure into a nappe-like structure overturned to the south (Millar, 2001). This new model indicates the possibility of significant regions which have been untested for ore horizons that may be present at exploitable depths on the overturned limb of the fold.

The structural reinterpretation of the region also entailed the re-examination of the relationships between the local major faults and their control on the area's mineralization. The study proposes that synvolcanic faults trending northeast are *major feeders for the in situ Buchans deposits. This idea implies that the intersections of these faults with the ore horizon are potentially valuable target areas.*

Recent developments in the structural and stratigraphic interpretations of the area lead drilling by the BRJV to be conducted in several stages. Six new holes were drilled on the Buchans Mine property between 1999 and 2000. Two of these holes intersected what is now known as HAG-type mineralization, or shear zone hosted massive sulphide clasts. The best results from these holes yielded a 40 cm clast assaying 14.4% Zn, 7.6% Pb, 0.4% Cu and 5.6 g/t Au, 253 g/t Ag. In the Middle Branch trend and the Airport Zone, the four holes in this area all encountered low grade mineralization.

The Newminex Buchans project initiated a litho-geochemical sampling program in 1997 which continued through until 2001. D. Wilton, L. Winter, and G. Jenner of Memorial University of Newfoundland conducted the work over various intervals. Eighty-three samples in total were collected and analyzed by X-ray Fluorescence (XRF) or Inductively Coupled Plasma/Mass Spectroscopy (ICP/MS) at the university. These studies indicated that hanging wall and footwall rocks from the Lucky Strike area can be differentiated from one another through major and trace element signatures. The transition between tholeiitic rocks and calc-alkaline marks the change from footwall to hangingwall, and also the location of the occurrence of the ore horizon sequence. A model of the chemical stratigraphy for the Buchans area has been composed based on major, trace, and rare earth element geochemistry. The purpose of this model was to potentially aid in identification of areas where unexplored ore horizon stratigraphy could be found under hanging wall rocks which have been previously misidentified as footwall (Jenner, 2001).

Billiton spent $2.4 million exploring the property before selecting targets as part of a detailed compilation and re-interpretation of the geology hosting the former Buchans mines. Billiton authored a report in May 2001 and presented a list of 126 high priority targets totalling 46,020 m of drilling. Billiton selected the targets based on their potential to host high-grade massive sulphide deposits similar in size and grade to the former Lucky Strike mine. When the BRJV was terminated, in September 2001, Billiton did not retain or earn any interest in the property (Halpin, 2001). Little or no new work has been completed on the property since that time.

6.4 2002-2008, Buchans River Ltd.

From 2002 to 2006, Buchans River completed some investigation into regional lithogeochemistry and a reinterpretation of the structural elements to the Buchans camp. The new interpretation posed new outlook into exploration strategies for the Buchans camp held under notion of a wide ranging co-relation between the host felsic volcanics of the historic high-grade Buchans orebodies, and the

> *potential that these units may be repeated within imbricated nappe structures not previously thought to exist.*
>
> *In December 2006 new management assumed control of the Buchans property. Numerous diamond drill programs were initiated included six drill holes (848 m) at the Little Sandy Property, four drill holes (1,160 m) on the Clementine West property, 53 drill holes (8,058 m) on the Lundberg and Engine House deposit, and eight drill holes for stratigraphic targeting within the proposed nappe structures for favourable Buchans River Formation rocks near the former Lucky Strike and Oriental mines (4,001 m).*
>
> *Buchans River also implemented a 3.6 x 5.1 km TITAN DCIP-MT deep crustal penetrating geophysical survey to the west of the town of Buchans.*
>
> *Mercator was contracted to complete a mineral resource estimate for the Lundberg deposit, and to initiate a property database for the BMC holdings.*

Since completion of the last Technical Report on the Property by Mercator in 2008, the Company has completed four additional diamond drilling campaigns totalling 8,559 metres of diamond drilling. The campaigns included programs designed to:

- test targets generated by Titan 24 geophysical surveys in the area west of the former Lucky Strike and MacLean orebodies (i.e., Clementine area)
- test the historic, undeveloped Old Buchans North high-grade massive sulphide prospect
- test the Clementine West stringer-stockwork sulphide prospect.

Additional drilling has been recommended for further testing of the Clementine West and Buchans North prospects.

The PEA is based only on Inferred Mineral Resources and are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is therefore no certainty that the conclusions of the PEA will be realized.

6.5 Royal Roads Corporation, 2006-2010

In December 2006, Royal Roads acquired a controlling interest in Buchans River and at that time new management took over operation of the company. In July 2008, Royal Roads purchased all remaining shares of Buchans River by way of a Plan of Arrangement giving Royal Roads an effective 100% interest in Buchans River and its property holdings, including the Buchans Project.

On July 2, 2010, Royal Roads changed their name to Buchans Minerals Corporation.

6.6 HISTORIC RESOURCE ESTIMATE 2007

The following is an excerpt from Webster and Barr (2008):

> *In September 2007, BMC,* then as Buchans River, *announced that it had located archived documents outlining an uncategorized resource estimate for a zone of stockwork type base metal mineralization peripheral to the former Lucky Strike mine. This historic uncategorized resource estimate reported approximately 13.1 st (11.9 million tonnes) with an average grade of 1.83% Zn, 0.67% Pb, 0.38% Cu, 0.16 oz/st Ag (5.5 g/t) and trace gold (ASARCO, 1974) (Buchans River Press Release PR #14-07). The 1974, ASARCO documents, plans and sections detail a mineralization zone referred to as the Lucky Strike Low Grade mineralization. Buchans River named this the Lundberg deposit, which represents the broad mineralized stockwork alteration halo underlying the Lucky Strike Glory hole which is suspected to have fed the higher grade in situ sulphide accumulation of the Lucky Strike ore. The ASARCO resource estimate is considered historic in nature and is not NI 43-101 compliant and therefore cannot be relied upon.*

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 GEOLOGICAL SETTING

7.1.1 REGIONAL GEOLOGY

The five main ore bodies historically mined at Buchans are thought to occur within a single felsic stratigraphic horizon within the Buchans Group, mainly the Buchans River Formation. The Buchans Group forms part of the Dunnage Zone, a Lower Paleozoic volcano-sedimentary terrain, which includes a complex assemblage of island-arc and back-arc basins formed by the integrated orogenic relationships that records the opening and subsequent closing of the Iapetus Ocean (Williams, 1978, 1979). The final closure of Iapetus, essentially fusing Laurentia and Gondwana, is delineated by the inferred Red Indian Line, which separates the respective Notre Dame subzone to the northwest and the Exploits subzone to the southeast (Williams 1988) (Figure 7.1).

The volcanics range in composition from basalt to rhyolite and tend to become increasingly felsic with height in the stratigraphy and may be laterally extensive and correlative with the Roberts Arm Group of Notre Dame Bay (Thurlow and Swanson, 1981). This variation from mafic to felsic volcanism is repeated several times within the Buchans Group and repetition was originally interpreted as repeating volcanic cycles (e.g. Thurlow et al., 1975). A new geological interpretation now considers the repetition to be largely caused by thrusting (Thurlow and Swanson, 1981).

The Buchans Group lies structurally above the ophiolitic Skidder Basalt in the southwest, and the Victoria Lake Group of Cambro-Ordovician origin to the southeast (Figure 7.2). The Feeder Granodiorite is an intrusive body that represents part of the subvolcanic magma chamber which fed the Buchans Group in some areas (Thurlow and Swanson, 1981). Geochemical evidence suggests the Feeder Granodiorite is the source of granitic boulders found within the breccia-conglomerate deposits within the transported ores at Buchans (Thurlow and Swanson, 1981).

Figure 7.1 Tectonostratigraphic Zones, Newfoundland

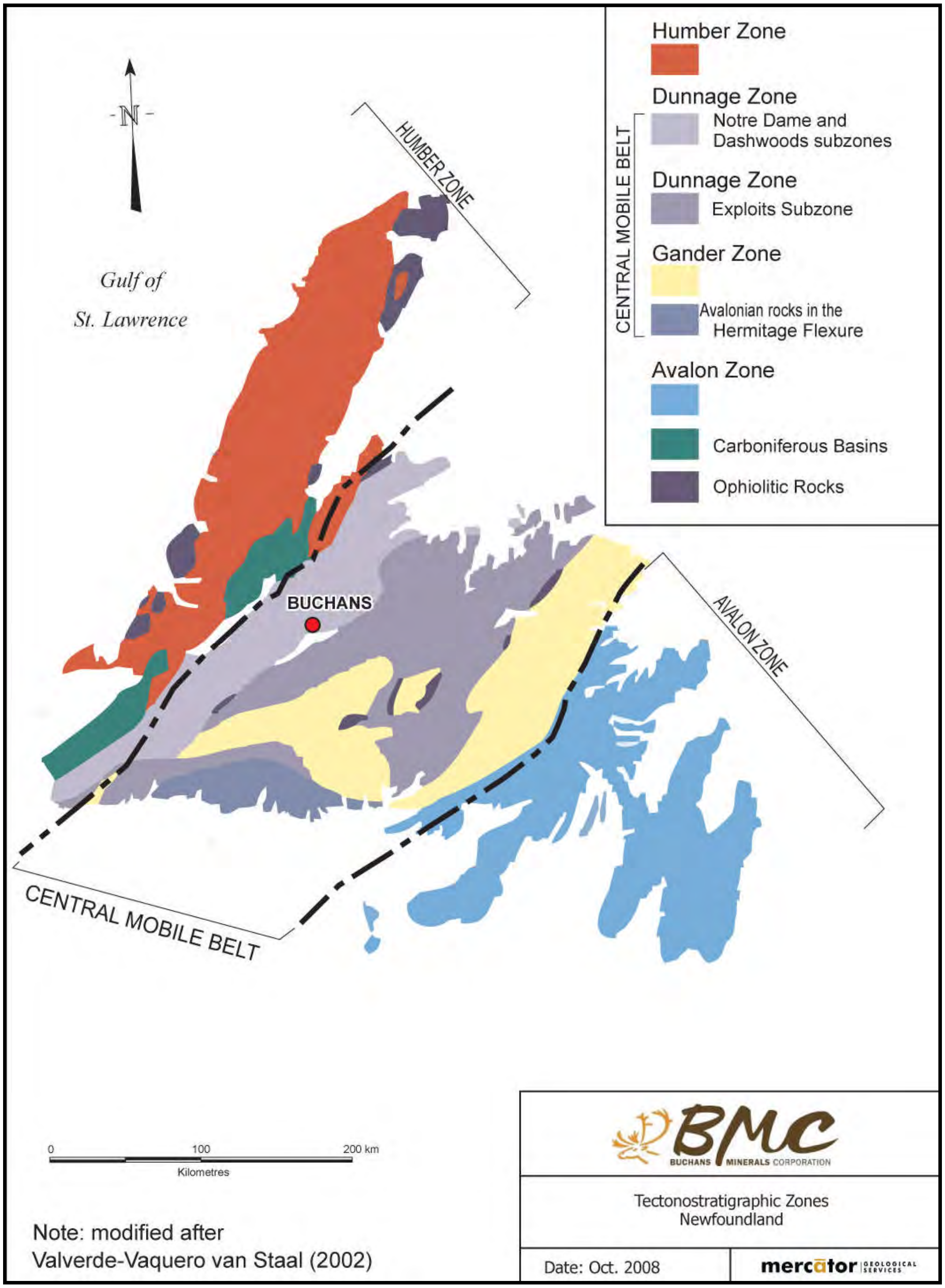
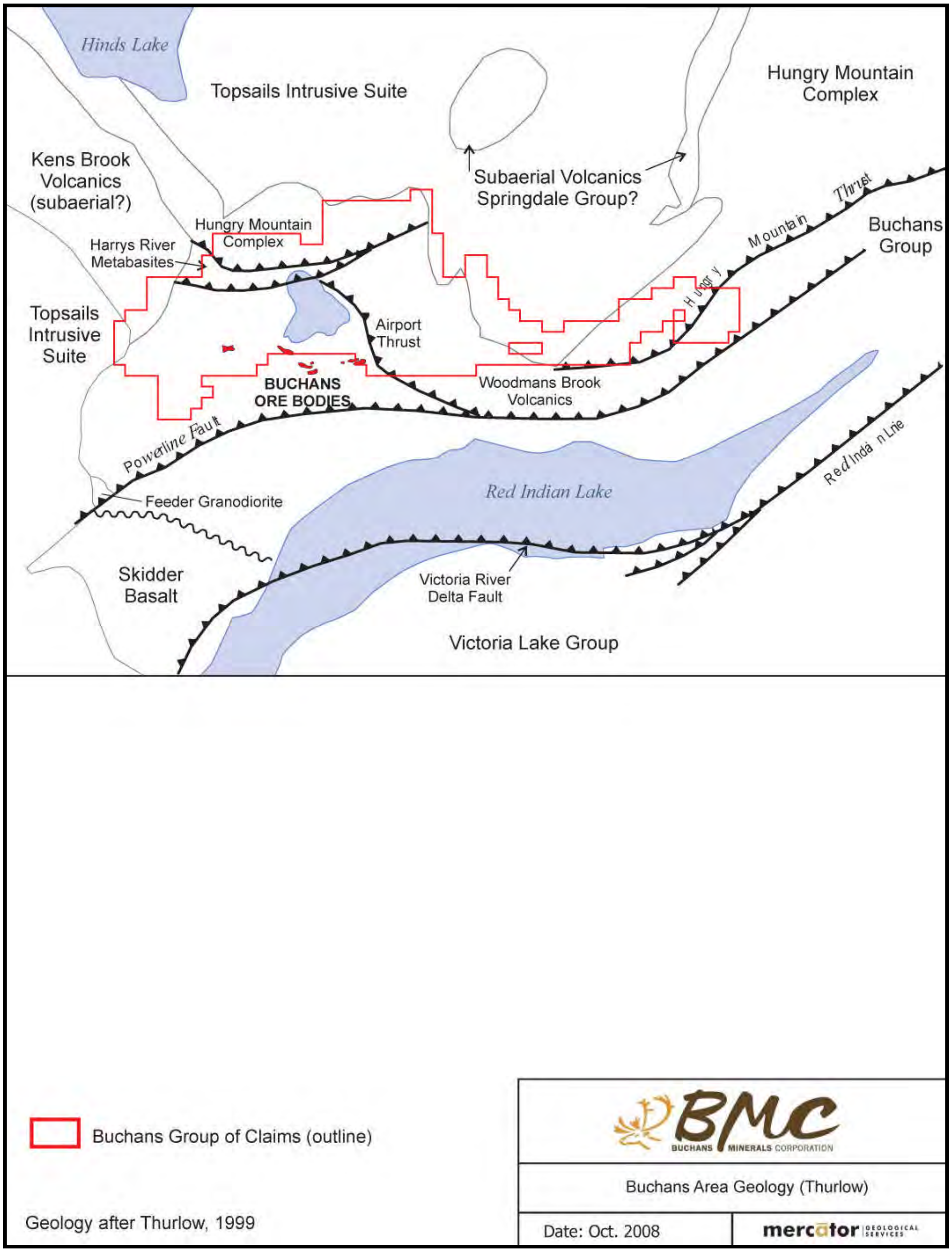


Figure 7.2 Buchans Area Geology (Thurlow)



Note: Claim boundaries shown in Figure 7.2 are valid as of Oct. 2008. For a current claim boundary map, see Figure 4.2.

Polydeformed intrusive rocks of the Cambro-Ordovician Hungry Mountain complex are thrust over the Buchans Group in the north and are intruded by the Devonian Topsails Granite in the northeast. In the northwest, Silurian subaerial volcanics unconformably overlie the Buchans Group and carboniferous red beds overlie the Buchans Group in the Red Indian Lake basin. The Kens Brook Volcanics are thought to overlie the Buchans Group but this relationship is not clearly understood (Thurlow and Swanson, 1987; Thurlow, 1999). The rocks in the Buchans area are metamorphosed to low grade prehnite-pumpellyite facies and a uranium-lead (U-Pb) zircon age of 473 ±2 Ma was obtained from the Buchans Group rhyolite (Dunning et al., 1987).

7.1.2 PROPERTY GEOLOGY

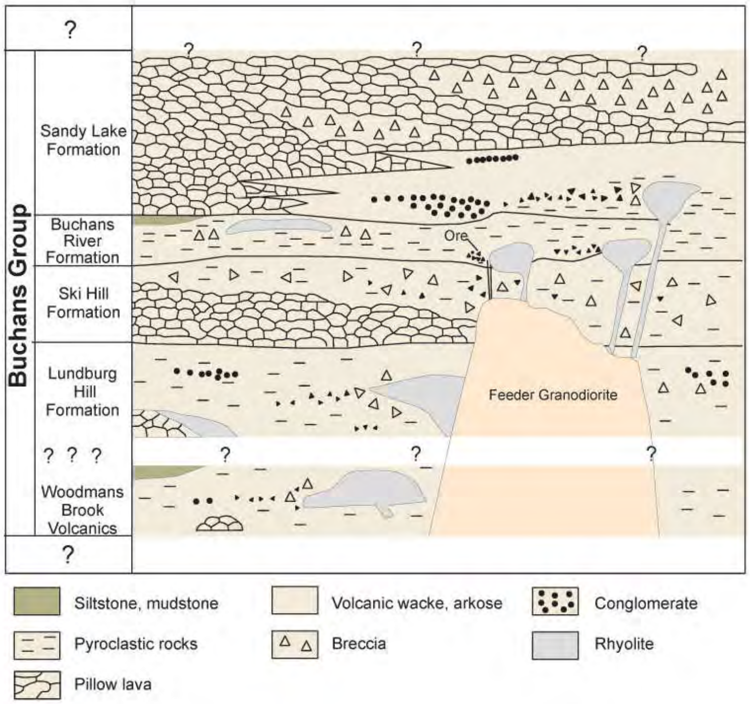
For the purpose of this report the property geology for the Buchans property has been described as it is considered to be relevant. The reader is referred to Webster et al. (2008b) for a complete geological description of the remaining Buchans area properties.

The five main ore bodies historically mined at Buchans are thought to occur within a single felsic stratigraphic horizon within the Buchan Group, however the recognition of this stratigraphy is regionally complex. Thurlow (1975) noted that the mafic to felsic volcanism was repeated several times within the Buchans Group and initially explained this as a cyclical reoccurrence. Subsequent studies following the mine closure resulted in the recognition of regional thrusting and the structural repetition of geology resulting in reinterpretation of the Buchans Group stratigraphy (Thurlow and Swanson, 1987). The stratigraphic re-interpretation of the Buchans Group is largely based on the relationship of fault bound mineralized blocks and has lead to the establishment of four sub-units within the Buchan Group (Figure 7.3).

The Buchans Group was broken down into four felsic and mafic formations, that include the Lundberg Hill, Ski Hill, Buchans River, and Sandy Lake Formations in addition to the Feeder Granodiorite and an unresolved unit named the Woodmans Brook Volcanics (Thurlow, 1999) (Figure 7.3).

The lowermost unit of the Buchans Group is the Lundberg Hill Formation, which is characterized by felsic pyroclastic rocks, coarse pyroclastic breccia, rhyolite, tuffaceous wacke, siltstone, and lesser basalt with minor chert and magnetic iron formations. The Lundberg Hill Formation has a maximum thickness which ranges from 200 to 1000 m (Dunning, Kean, Thurlow and Swinden 1987).

Figure 7.3 Stratigraphic Column Buchans Group



Stratigraphy of the Buchans Group

Formation	Maximum Thickness in Oriental Block	Maximum Thickness	Lithologies
Sandy Lake Formation	200m	2000m?	Basaltic pillow lava, pillow breccia intertonguing with coarse grained, redeposited clastic rocks of felsic volcanic derivation (arkosic conglomerate, arkose, wacke, siltstone). Local abundant tuff, breccia, polylithic pyroclastic breccia and tuffaceous sedimentary rocks.
Buchans River Formation	200m	400m?	Felsic tuff, rhyolite breccia, pyritic siltstone, wacke, polylithic breccia-conglomerate, granite boulder conglomerate, high-grade in situ and transported orebodies.
Ski Hill Formation	1000m	1000m?	Basaltic to andesite hyaloclastic breccia, pillow lava and massive flow with local polylithic breccia near the stratigraphic top. Minor felsic tuff.
Lundburg Hill Formation	200m minimum	1000m?	Felsic pyroclastic rocks, coarse pyroclastic breccia, rhyolite, tuffaceous wacke, siltstone, lesser basalt, minor chert and magnetic iron-formation

BMC BUCHANS MINERALS CORPORATION

Generalized Stratigraphy of the Buchans Group Newfoundland

Geology after Thurlow, 1999

Date: Oct. 2008

mercator GEOLOGICAL SERVICES

The Lundberg Hill Formation is conformably overlain by the Ski Hill Formation which is dominantly composed of dark green often amygdaloidal mafic pillow lavas, breccias and pyroclastic rocks (Dunning, Kean, Thurlow and Swinden 1987). The Ski Hill Formation is host to stockwork systems within the footwall to the in situ ore hosted within the overlying Buchans River Formation. The Buchans River Formation hosts the historically mined ore deposits at Lucky Strike, Old Buchans and Oriental, and is comprised of felsic tuff, rhyolite breccia, pyritic siltstone, wacke, polylithic breccia-conglomerate, granite boulder conglomerate, high-grade in situ and transported orebodies. This Formation ranges from 200-400 m in thickness (Dunning, Kean, Thurlow and Swinden 1987) in the mine area and more discrete amounts of the formation are found locally throughout the Buchans area (Dunning, Kean, Thurlow and Swinden 1987) (Figure 7.3).

The Buchans Group has been subjected to two major periods of deformation (Thurlow, 1981). The first was a Silurian episode of south-easterly-directed thrusting during which the Hungry Mountain Complex, which consists largely of pre-deformed granitoid rocks, was emplaced upon the Buchans Group. In addition, this period of thrusting may have caused repetition of units within the Buchans Group including the possible repetition of an originally continuous ore horizon sequence. The second deformational event consisted of a period of broad open folding during the Devonian, which imparted a weak, northeast-trending axial planar cleavage to all rock types. A major, northeast-trending syncline in the Buchans Group is related to this event (Thurlow, 1981).

In 2001, a new structural model was proposed to explain the repetition of geology within the Buchans Group. This re-interpretation suggested that instead of an imbricated thrust stack, the structural geology could be explained by recumbent nappe structures, over turned to the south (Millar, 2001). This model was thought to provide a clearer understanding of the inter-relationship between the various faults and the thrusts that controlled emplacement and deformation of the Buchans mineralization.

Lithogeochemical studies throughout the Buchans area also lead to a new interpretation of stratigraphy. The new hierarchy is based solely on geochemical parameters as opposed to the more subjective previous criteria of lithologies and textures (Jenner, 2001). These studies have identified two separate volcanic cycles and the felsic rocks which host the Buchans orebodies are interpreted to occur near the interface of Cycle 1 and Cycle 2. The implications of this work suggest that recumbent overturned folding may open up many untested zones where the ore horizon could be repeated (Jenner, 2001).

7.2 MINERALIZATION

The following excerpt is taken from Webster and Barr (2008).

Mineralization in the Buchans area is associated with the three main genetically related ore deposits types. The Lucky Strike and Orient #1 orebodies are the best known examples of the in situ type ore and represent the highest grade ore mined in the Buchans area and occur on the Property. The Lucky Strike orebody consisted of massive high-grade sulphides where ASARCO mined 5.6 Mt of ore with head grades averaging 18.4% Zn, 8.6% Pb, 1.6% Cu, 112 g/t Ag & 1.7 g/t Au (calculated based on Thurlow and Swanson, 1981, pp 122 to 128). Massive in situ ore occurs as several ore textures but the massive fine grained streaky ore seems to form within the bulk of the deposit occurring as aggregates of sphalerite, galena, barite and lesser chalcopyrite. Thurlow et al, (1975) reported trace amounts of enargite, native silver and argentite, ruby silver and gold tellurides, in addition to native silver and gold. Minor sulphides also include tetrahedrite-tenantite, chalcocite and bornite. Pyrite forms a relative minor part of the massive ore but is more common in association with stockwork ores (Thurlow and Swanson, 1981). The paragenetic sequence of mineral deposition is complex but includes resorption, fracturing and re-deposition. Pyrite appears to be the first mineral deposited and sphalerite, chalcopyrite and galena are thought be deposited at the same time but chalcopyrite is also seen as blebs, lamellae and veins (Strong 1981).

Transported ores occur as elongate-tabular accumulations of discrete high-grade fragments (Thurlow and Swanson, 1981). The deposits are transported by density flows that occur in paleotopographic lows, down slope from in situ ore bodies. MacLean, Rothermere, Clementine and Oriental #2 are examples of transported ore and together with the massive ore represent 98% of the ore mined in Buchans. The transported ore bodies occur as mechanically transported sulphide breccia lenses composed of sulphide bearing fragments derived from in situ ore (Thurlow and Swanson, 1981). They maintain grade and have been noted to travel distances of up to 2 km from source areas. Sulphide fragments range from angular to sub-rounded and display streaky textures with sphalerite galena, chalcopyrite and barite being the main minerals. Unlike the in situ ore they have no associated stockwork zone. All of these orebodies occur on [BMC's Mineral Licences].

Stockwork mineralization is typically associated with in-situ ore and the best example is the Lundberg deposit which was the subject of a drill program by Buchans River in late 2007 and early 2008 under management of BUV and the author. The Lundberg deposit is stratigraphically below the historically mined Lucky Strike orebody and consists of quartz-barite-carbonate-sulphide veins and veinlets cutting strongly altered mafic to intermediate volcanics with disseminated sulphide mineralization. The stockwork mineralization comes to surface on the eastern edge of the zone and forms an elongate, wedge shaped body that is 350 m deep on the western end. The highest concentration of sulphide mineralization appears to be within close proximity to the Lucky Strike orebody and more diffuse away from the historic workings. Unlike the in situ ores, fine to coarse grained euhedral pyrite is the dominate sulphide and occurs with varying amounts of chalcopyrite, sphalerite, galena and barite (Thurlow and Swanson, 1981).

A second zone of stockwork mineralization is associated with the Engine House deposit which lies almost immediately south of [the former Lucky Strike Mine] and typically has a greater proportion of chalcopyrite. The stockwork system of the Engine House deposit does not appear to connect directly to the Lundberg deposit, in their present configuration, as determined by historic drilling as well as drilling completed by BUV in 2008.

Mineralization is also found in association with high-grade clasts noted from drilling within the Buchans area and their source is not clearly understood (Thurlow and Swanson, 1981). Clasts range in size from grains and pebbles to 30 cm boulders of high-grade sulphide mineralization. The clasts contain galena, sphalerite, pyrite, chalcopyrite and gold and silver and are similar in metal grades to the in situ Buchans ores. They occur in polylithic conglomerates within the same stratigraphic horizon as the in situ ore but also at distances of up to 6.7 km away from any know in situ orebody (Thurlow and Swanson, 1981).

Base metal mineralization also occurs within undeveloped prospects at Clementine, Sandfill, Middle Branch and Little Sandy (Figure 4.1) that occur within BMC's other mineral licences.

8.0 DEPOSIT TYPES

The following is an excerpt from Webster and Barr (2008).

The Buchans area deposits and showings are generally volcanogenic massive sulphide (VMS) type ores and are primarily comprised of base-metal sulphides and barite, and show strong similarities to the Kuroko style deposits of Japan (Thurlow, 1981). The Buchans orebodies comprise three distinct, but genetically related deposit types and occur as in situ ore, mechanically transported ore, and stockwork ore (Thurlow and Swanson, 1981).

The zoned massive sulphides of the in situ deposits formed in close proximity to the volcanic discharge zone. They consist of thick lenses of high-grade ore and form the largest ore bodies in the Buchans area. The in situ ores are overlain by a capping of massive barite, which is characteristic of the Buchans deposits, and which may provide an important lithogeochemical exploration tool (Thurlow et al., 1975). The felsic volcanics also host lower grade, base metal enriched sulphide systems of hydrothermal alteration known as stockwork mineralization.

Stockwork mineralization consists of a network of sulphide veins and veinlets that cut strongly altered and sulphide impregnated hosts rocks. The largest known concentration of stockwork and disseminated mineralization underlies the Lucky Strike deposit and has been named the Lundberg deposit, and has been the subject of a diamond drilling program by BMC. The stockwork ore has a high proportion of pyrite to base metal sulphides than the in situ ores and consists of fine to coarse grained pyrite with lesser amounts of chalcopyrite, sphalerite, galena and barite.

Transported ore deposits are coarser grained and are interpreted to be debris flows from the in situ ores, which have been redeposited in paleochannels or downslope regions. The transported ores at Buchans are elongate-tabular accumulations of high-grade massive sulphide and lithic fragments that occur within paleo-topographic channels. Six of these channels, containing at least seven orebodies or sub-economic sulphide deposits have been recognized in the Buchans area (Walker and Barbour, 1981). The orebodies consist of discrete sulphide breccia lenses, which grade laterally into low-grade breccia conglomerate and granite conglomerate.

The breccia ores (i.e. occurrences of high-grade ore clasts) have been noted in historic drill holes and are wide spread within the Buchans area. Normally occurring as polylithic breccias they appear to be located in the same stratigraphic horizon as the major in situ and transported ores, although clasts have also been noted in areas up to 6.7 km from any known occurrence of in situ ore (Thurlow and

Swanson, 1981). They are thought to form as a result of disruption of in situ massive sulphides or transported as part of debris flows along paleo-topographic channels. The breccia ores consist of massive sulphide and lithic fragments in a matrix of finer grained material that is compositionally similar to the fragments. Clasts include various volcanic, sedimentary and plutonic lithologies, all of which are locally derived. Granitoid fragments show an anomalous composition and a higher degree of rounding then other fragments (Thurlow, 2001). Massive sulphides and barite occur both as clasts and matrix material.

The breccias display a wide variation in fragment type and in the development of sedimentary features (e.g. bedding, sorting, grading). All occur as channel fillings, having sharp footwall and hanging wall contacts and showing evidence of scouring and incorporation of fragments of underlying lithologies (Walker and Barbour, 1981).

The original ore deposit model for Buchans suggests that mineralization formed from a submarine exhalite caldera and that mineralization is bound by a structure to the south of Buchans (Thurlow, 1999). However, a new ore deposit model has been proposed and northeast-trending, synvolcanic normal faults have been suggested as the primary discharge zone for mineralizing fluids for the Buchans orebodies (Millar, 2001). It is thought that orebodies and alteration zones acted as loci for thrusting, with the result that all major orebodies are fault bound, and that massive sulphide clasts found along the faults are possibly derived from larger undiscovered bodies along or near the feeder structures. The new model suggests that in the Lucky Strike area the overall geometry is represented by recumbent folds or nappe structures. This interpretation suggests that untested ore zones may exist where the contact is re-interpreted to be overturned (Millar, 2001).

9.0 EXPLORATION

BMC has not conducted any exploration activities related to the Lundberg and Engine house deposits since 2008. See section 6.4 for 2008 exploration activity completed after the 2008 NI 43-101 technical report produced by Mercator.

The following excerpt of the most recent exploration is taken from Webster and Barr (2008).

9.1 EXPLORATION, 2006-2008

Relevant exploration in proximity to the Lundberg deposit has included 53 diamond drill holes (8,058 m) that were designed to specifically test the mineral grade and extent of the known stockwork mineralization. Other recent exploration in the Buchans area included four drill holes on the Clementine West prospect. Also completed was a Titan 24 MT-DCIP survey covering a 20 km^2 area immediately west of Lundberg deposit and extending to the northwest 5.1 km over the historically mined Buchan's orebodies, the Clementine prospect, and other prospective ground.

9.2 LUNDBERG DEPOSIT 2007-2010

The mineralized zone (at the Lundberg deposit) was described by Thurlow (1981) as a network of sulphide veins cutting strongly altered and sulphide impregnated host rocks occurring under the Lucky Strike deposit. He describes a wedge shaped zone of mineralization 360 m wide, extending 600 m down dip and having a thickness of up to 100 m. The zone subcrops within 1.5 m of surface at the east end of the Lucky Strike pit where it has returned sulphide rich mineralization from drill holes completed by ASARCO and BMC (then as Buchans River) (Figure 10.1). Highlights of the BMC drilling are summarized in Section 11 of this report.

In the fall of 2007, BMC contracted Mercator to complete a digital compilation of the historic diamond drilling and to propose a diamond drilling program that would test the mineralized zone and provide sufficient information for the completion of a NI 43-101 compliant resource estimate. Mercator outlined a 40 hole, 6,000 m diamond drill program to test the extent of the ASARCO outlined historic resource and adjacent areas. The program was completed in May 2008 having drilled a total of 53 drill holes (8,058 m). The results of these holes are discussed in Section 10.

10.0 DRILLING

The following excerpt is taken from Webster and Barr (2008).

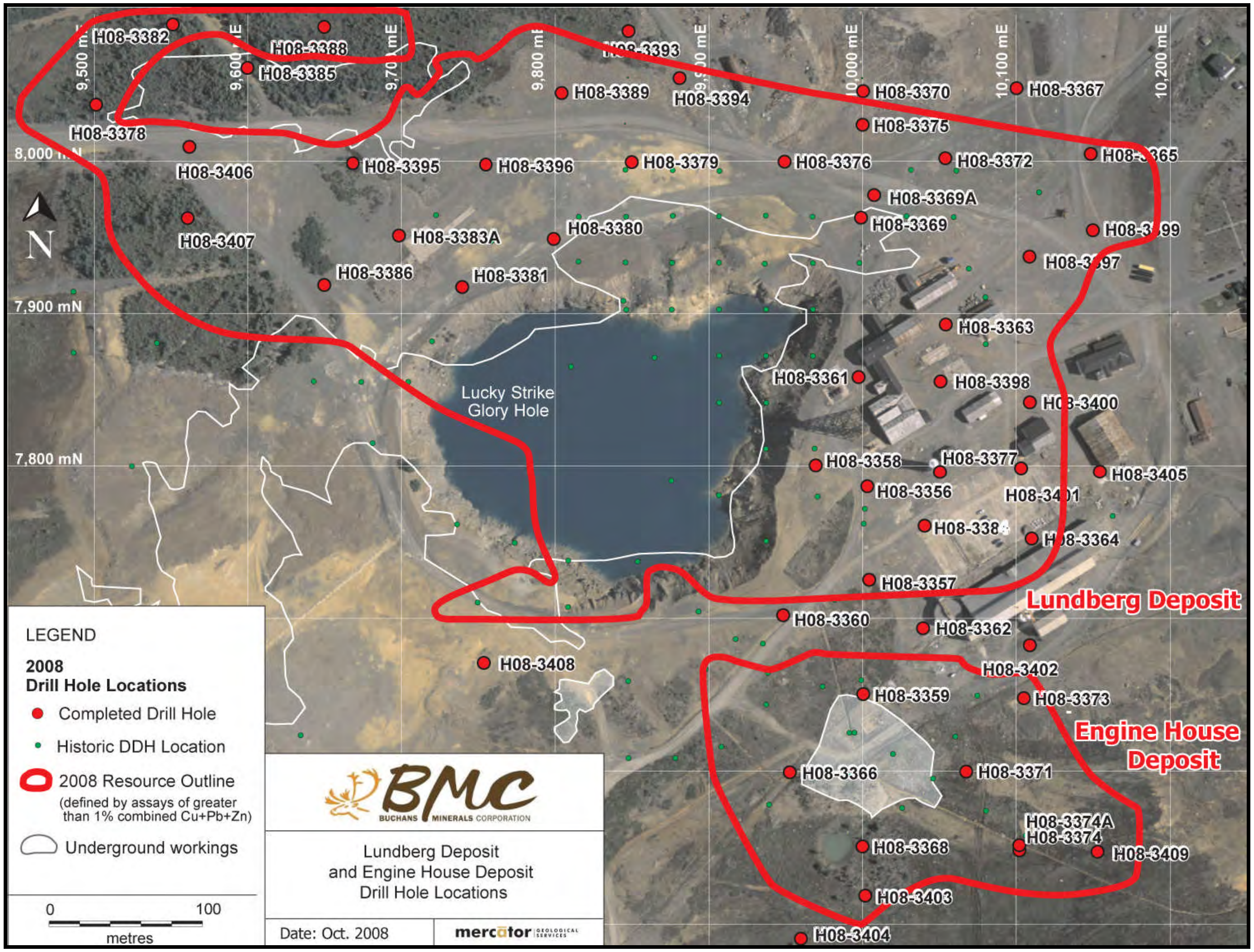
BMC (then as Royal Roads and Buchans River) completed eight diamond drilling programs since the fall of 2006. In the Buchans area, six holes totalling 850 m were completed at Little Sandy prospect, eight holes totalling 3,850 m were completed to test targets outlined by Billiton within the area of the historic Buchans area mines (Figure 4.2), and 43 holes totalling 6,853 m were completed within the Lundberg deposit, and ten holes totalling 1,205 m were completed within the Engine House deposit. In addition 88 drill holes totalling 17,078 m were completed on the Tulks North properties (Webster et al, 2008a). The programs were managed by Mercator under the supervision of BMC management and Mercator. All logging and sampling of drill core was performed by Mercator geologists in suitable onsite facilities in Buchans. The following describes the Lundberg deposit drilling program only.

10.1 LUNDBERG DEPOSIT

A discussion of the findings and results of 43 drill holes totalling 6,853 m are summarized as follows. The true widths of the mineralization have not been estimated from drillhole intersections due to the stockwork and disseminated nature of mineralization, however, a predominant sub-horizontal mineral trend exists in the stockwork system beneath the Lucky Strike glory hole (Figure 10.1). The Lundberg deposit is defined as a zone containing greater than 1% combined base metal grade (Zn% + Cu% + Pb%). This zone was previously interpreted by ASARCO in 1974 as a low grade alteration package.

The Lundberg deposit demonstrates a sub-horizontal zone of mineralization which appears to dip gently to the north and plunge to the northwest, and subcrops as an erosional surface in contact with overburden in the eastern portion of the zone. It is bound to the South by the Airport Thrust fault, to the north by the Ski Hill Thrust fault, and gradually wanes to the west where it is overlain by the transported ore deposits previously mined by Asarco. The deposit is underlain, by thrust contact of the Old Buchans Fault, with the younger Sandy Lake formation which is locally composed of hematitic amygdaloidal basalt and exhibits a weak magnetic character in comparison to the non-magnetic character of the Ski Hill Formation basalt. The stockwork zone does not locally penetrate the Sandy Lake Formation, indicating pre-tectonic mineral deposition.

Figure 10.1 Lundberg Deposit and Engine House Deposit Drill Hole Locations



The mineralization is dominantly composed of a quartz-carbonate-barite stockwork system hosted by brecciated and vesicular intermediate to mafic volcanics correlative with the Intermediate Footwall (Ski Hill Formation) described by Thurlow (1999). The quartz-carbonate-barite phase is accompanied by pyrite, chalcopyrite and fine-grained galena-sphalerite. Immediately to the east of the Lucky Strike Glory hole, an area is defined by the erosional surface of the stockwork sequence subcropping within 1.5 m of surface. Localized mottled semi-massive horizons of quartz, barite, carbonate and variable amounts of base-metals with up to 30% pyrite occurring as stringers, blebs and fracture fills, cut the quartz barite phase. Massive pyritic Zn-Pb-Cu sulphides were noted near the interpreted top of the stockwork zone immediately to the northwest of the glory hole but are not considered to be remnant Lucky Strike high-grade in situ massive sulphides as they are compositionally and texturally distinct. Alteration was observed to be most advanced in this area.

10.2 Engine House Deposit

The Engine House deposit has been defined as a separate mineralized body that sits immediately adjacent to the main Lundberg deposit and for the purposes of this resource estimate has been modeled as a separate zone (Figure 10.1). A total of ten holes totalling 1,205 m were completed within the Engine House deposit. Two intimately-related styles of base metal mineralization were identified on the periphery of the former Engine House orebody. The first style comprises a thin horizon of exhalative massive sulphides (likely corresponding to the historically mined Engine House orebody), capped by a red chert bed and then overlain by a felsic tuff of the Buchans River Formation. Stratigraphically below the narrow massive sulphide-zone is a stockwork system dominated by chalcopyrite-pyrite with lesser galena-sphalerite hosted within polylithic breccia-volcanics and an altered chloritic matrix, lithologically similar to that of the intermediate footwall underlying the Lucky Strike Glory hole. The stockwork veins contain notably less quartz-carbonate and slightly higher barite than the neighbouring Lundberg deposit to the north. The massive sulphide horizon was observed in drill holes within close proximity to the historically mined Engine House orebody.

10.3 Logistics

Springdale Forestry Resources of Springdale, NL, was contracted to provide core drilling services for the 2007 and 2008 Buchans River programs and supplied a diesel-powered skid-mounted Duralite 500 wire-line drilling unit equipped to recover NQ size rock core (4.76 cm in diameter). The company also provided all necessary support equipment, including an excavator and bulldozer for drill moves and site preparation work. Drilling was typically carried out on a 24 hour per day basis.

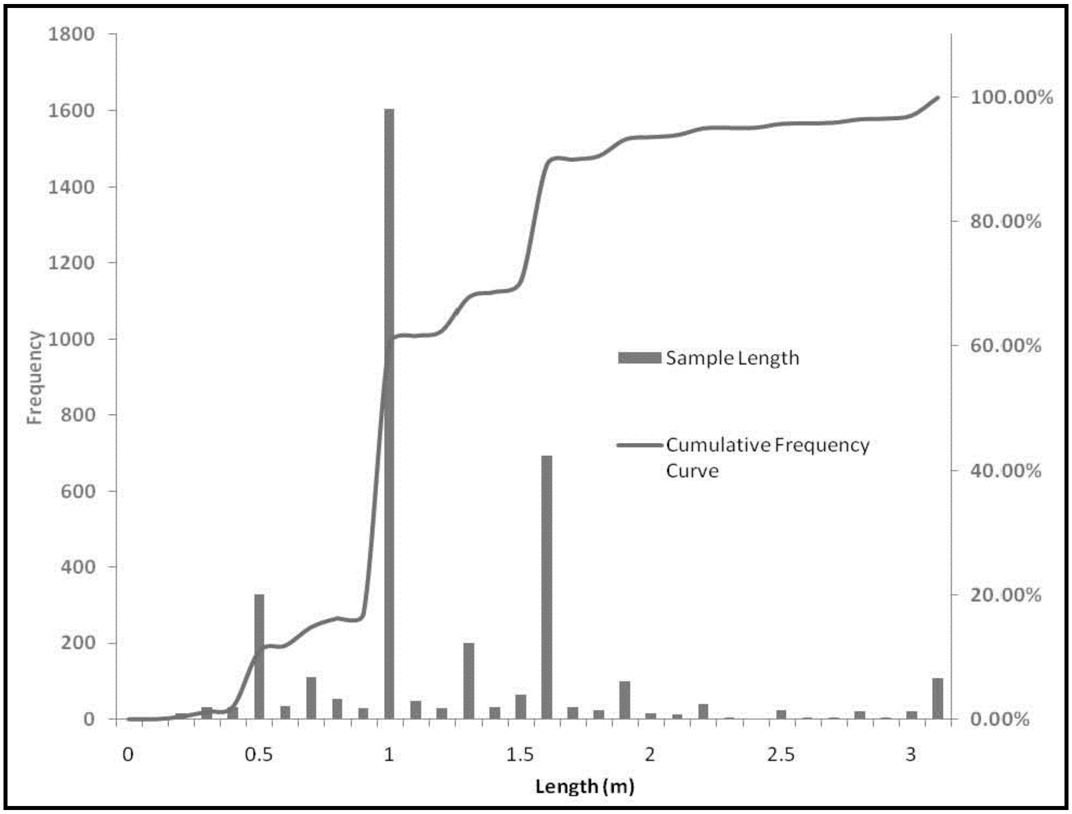
Mercator provided geological and technical staff for all drilling projects to facilitate day to day coring operations and logging functions, with project planning and *oversight provided through consultation with senior BMC and Mercator staff. All drilling, field and geological personnel were accommodated through local housing and restaurant facilities. Mercator provided field support including trucks for the entire field program, while BMC supplied accommodation and secure core logging facilities to Mercator staff.*

Drill collar locations for each drillhole were based on Universal Transverse Mercator Zone 21 (UTM NAD 83) as grid coordinates and were established initially in the field by handheld Global Positioning System (GPS) units, followed by differential GPS surveys upon completion of the drilling by Red Indian Surveys of Grand Falls, NL. Drill holes were typically tested for inclination and azimuth variation using Flex-it© down hole survey instruments and results were incorporated in the project database.

10.4 SAMPLING METHODS

Drill core was descriptively logged on site, aligned, marked for sampling by Mercator geologists in a secure logging facility in the town of Buchans. All drill core was cut in half, longitudinally, using a diamond saw blade by Mercator technical personnel. Samples consist of half NQ-size diamond core (47.6 mm diameter core), consistently taken from the right hand side of the core (looking down hole). The unsampled half of the core is preserved in core boxes for future reference. As part of quality assurance/quality control (QA/QC) protocol, samples of core are bagged, tagged, sealed and delivered directly to Eastern Analytical Limited laboratory in Springdale, NL by Mercator personnel respecting appropriate chain of custody measures. Sample numbers were recorded in both the core log and in sample record logs for the drill hole. In accordance with the QA/QC protocol set up by Mercator for this drill program, duplicate, blanks and standard samples were inserted in the number sequence at set intervals. The QA/QC program is discussed detail latter in this report. Base metal bearing samples are nominally 1 m to 1.5 m in length, except where specific geologic parameters require a smaller interval be sampled (Figure 10.2).

Figure 10.2 Sample Length Distribution



11.0 SAMPLE PREPARATION, ANALYSES & SECURITY

The information in this section is taken from Webster and Barr (2008).

Sample preparation was completed by Eastern Analytical with each sample crushed to approximately 75% of -10 mesh and split using a rifle splitter to approximately 300 g. Each sample split was pulverized using a ring mill to approximately 98% -150 mesh. In addition to regular samples, blank samples (one per 20 samples) and certified standards (one per 20 samples) were also submitted for sample preparation and assay. All coarse reject material was maintained for the duration of the drill program but has since been discarded. Representative pulp samples have been preserved by the company.

All assays were completed by Eastern Analytical of Springdale Newfoundland by the inductively coupled plasma method (ICP-11) for base metals (Cu, Pb, Zn) and to ore grade assay Cu, Pb and Zn if upper detection limits by ICP were exceeded for either element (upper detection limits; Cu 10,000 ppm, Pb 2,200 ppm, Zn 2,200 ppm). ICP analyses were completed using a 0.50 g sample digested in nitric and hydrochloric acid and analyzed by Inductively Coupled Plasma Optical Emission Spectroscopy (ICPOES). Base metal ore grade assays (Cu, Pb, Zn) were completed using a 0.20 g sample digested in nitric and hydrochloric acid and analyzed by the atomic absorption (AA) method. Silver assays were completed using a 1,000 mg sample digested in hydrochloric and nitric acid and analyzed by AA. Gold assays were completed by standard ½ assay ton fire assay using the AA method. All samples analyzed by the ore grade assay method were re-assayed as check assays by ALS Chemex of Vancouver, BC. Eastern also completed independent QA/QC protocols that include insertion of blanks and certified CanMet standards as part their routine analyses.

12.0 DATA VERIFICATION

The information in this section is taken from Webster and Barr (2008).

QA/QC protocols were implemented to monitor accuracy, reproducibility and precision through each stage of data collection. These stages include sampling, assaying and geological observation. During the 2007/2008 drilling programs Mercator, on behalf of BMC, carried out a comprehensive QA/QC program. Each step of this program lends itself to a different aspect of quality control and assurance. The insertion of blind standards allows observers to monitor the precision of assay results by comparison to known grades for the control sample. Blanks were used to detect and evaluate issues of cross contamination. The analysis of duplicate sample slits is used to determine the accuracy of reproducing results. In the final verification measure a split of selected sample pulps is sent to a second independent laboratory as means for checking assay of ore grade samples to ensure analytical validity in reported grades. The details are as follows.

12.1 CERTIFIED STANDARD PROGRAM

Canadian Resource Laboratories of Delta, BC provided two certified standards for use in the Lundberg drilling program. The standards were selected on a basis of mineral composition and grade range, including high-grade and low grade standards. The standards selected were CDN-HLHZ and CDN-FCM-4, the details of which can be found in Appendix C. The Standards were inserted blindly every even 20th sample number in chronologic order with core samples and marked accordingly in the sample record book. Results of each sample were checked against the laboratory supplied grade range tolerances. All certified standards used in drill programs within the Buchans area fall within acceptable limits and considered appropriate for VMS mineralization (Figure 12.1 and Figure 12.2).

Figure 12.1 HLHZ Standard Evaluation

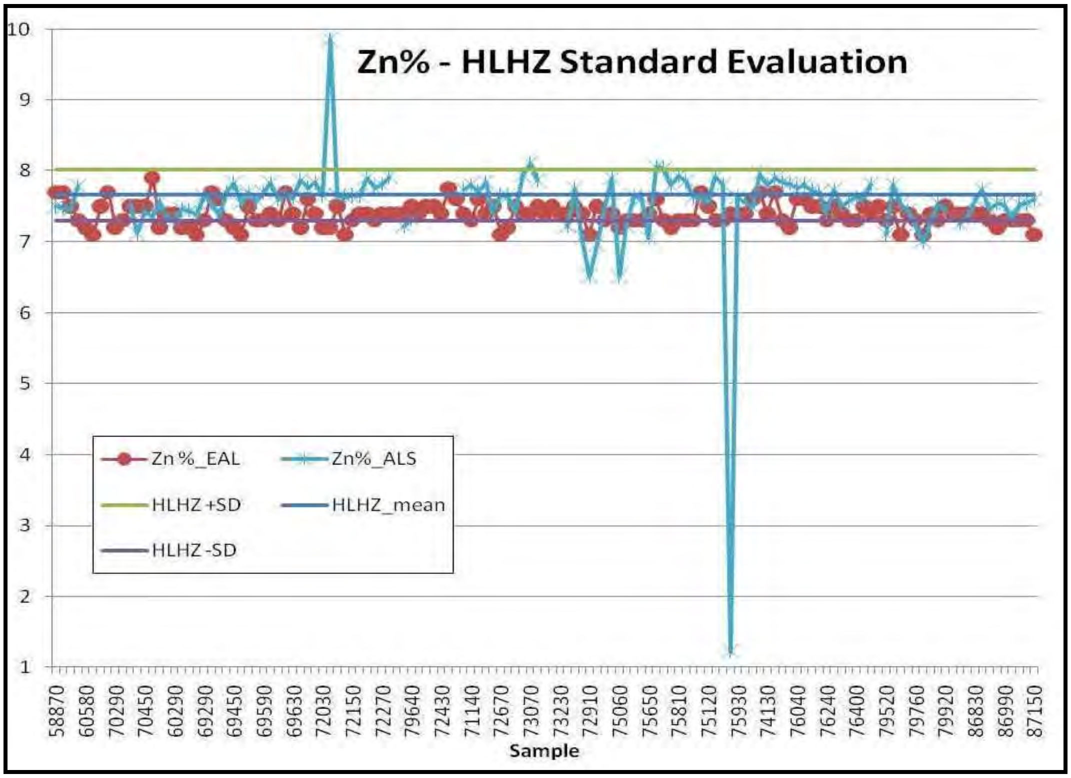
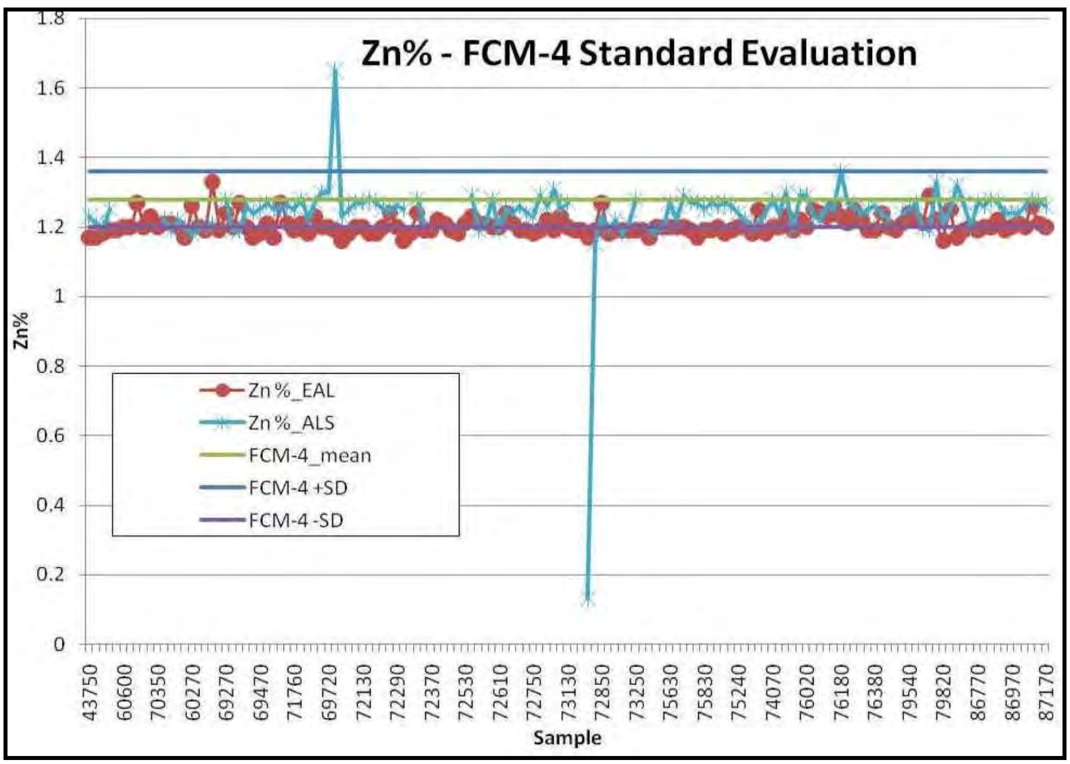


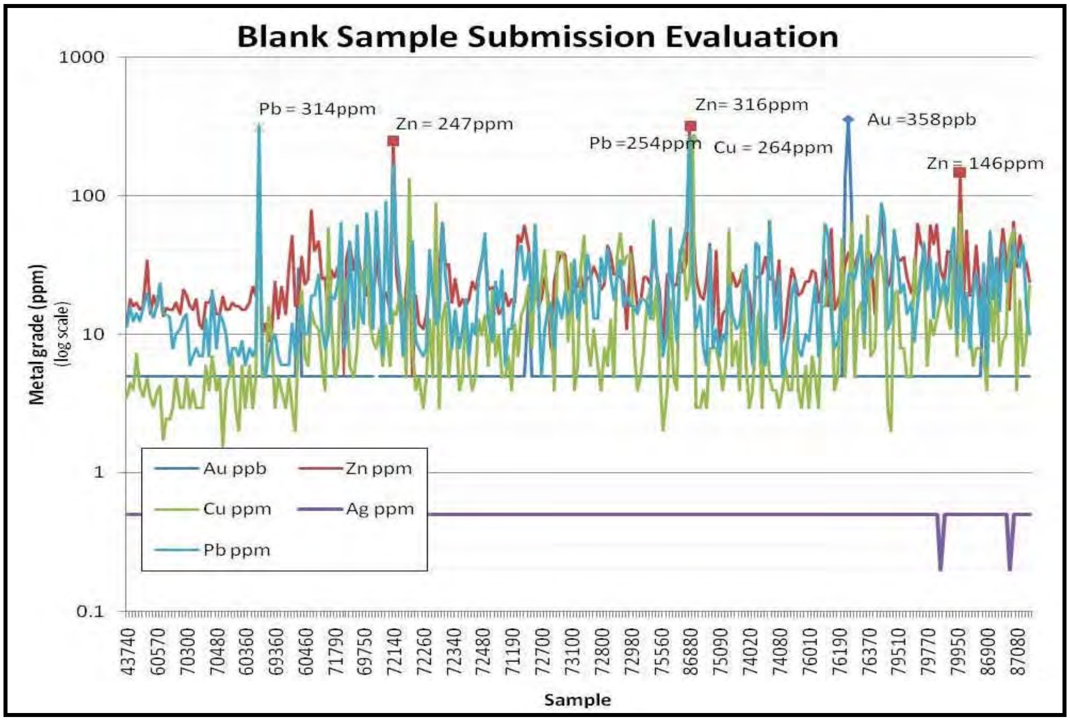
Figure 12.2 FCM-4 Standard Evaluation



12.2 BLIND BLANK SAMPLE PROGRAM

Blank samples were inserted in chronological succession at an interval of every odd 20th sample and noted in the sample record book as a blank. Blank sample material is comprised of benign sandstone collected by Mercator geologists from outcrops near the south shore of Red Indian Lake. Blanks samples were assayed by Eastern Analytical throughout the course of the drilling programs. Results for blank samples indicate a consistent background level and results fell with accepted limits (Figure 12.3).

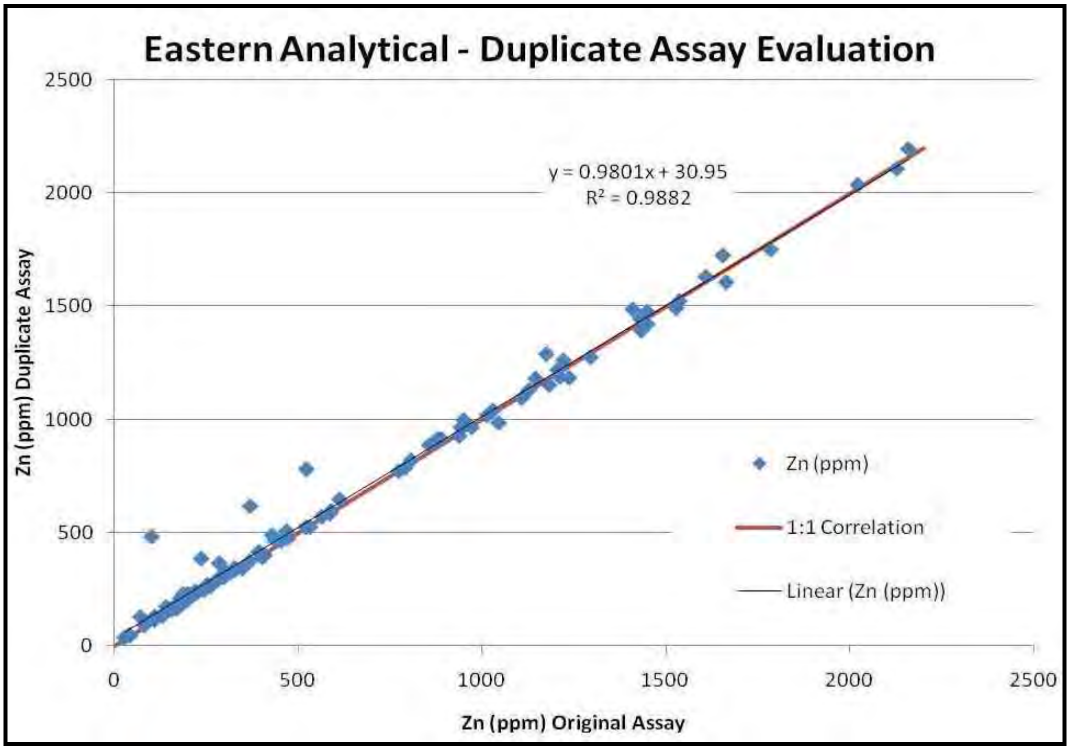
Figure 12.3 Blank Sample Evaluation



12.3 DUPLICATE SAMPLE PROGRAM

Duplicate samples were not collected as drill core samples. Duplicate sample pulps were prepared in the laboratory setting by Eastern Analytical for every 20th sample, and were used by Mercator and Eastern Analytical as in house quality control. Results for the duplicate pairs were reviewed with respect to lead and zinc values (ICP analysis only). Duplicates from Eastern Analytical show good reproducibility and check assays display 1:1 correlation which suggests the data is reliable (Figure 12.4).

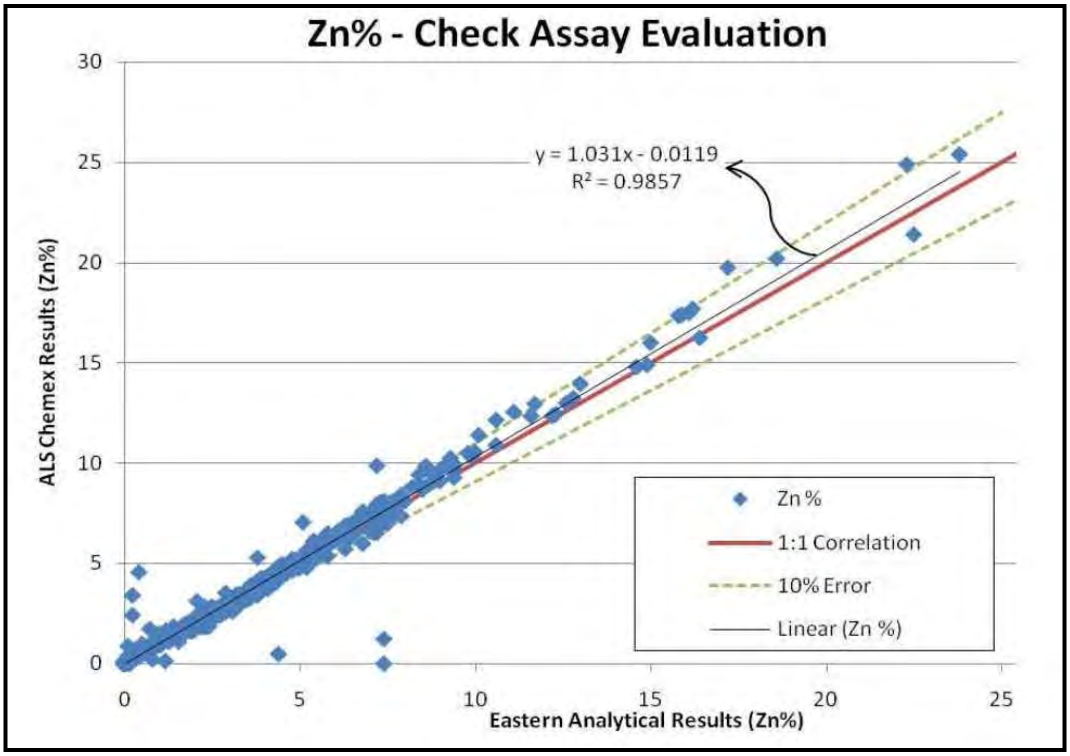
Figure 12.4 Duplicate Sample Evaluation



12.4 CHECK ASSAY PROGRAM

All blanks, standards and samples pulps contained within ore grade mineralized envelope (as defined by Mercator) were sent to ALS Chemex for check assay, respecting appropriate chain of custody measures. Comparison of analytical results between ALS Chemex and Eastern Analytical indicate strong 1:1 correlation. It is the opinion of this author that the data sets discussed in this report are acceptable (Figure 12.5).

Figure 12.5 Check Assay Evaluation



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The mineralization is a semi-massive sulphide of contained copper, lead and zinc sulphides, in association with a range of non-sulphide minerals. The non-sulphide minerals are dominated by quartz and chlorite with lesser amounts of barite.

The QEMSCAN, carried out by SGS Lakefield (SGS) revealed the following mineralization (Imeson, 2011):

> *The only copper bearing sulphide mineral identified was present as chalcopyrite. The only lead mineral was present as galena, and the zinc mineralization occurred exclusively as sphalerite. Iron was distributed in primary sulphides including chalcopyrite, sphalerite, galena and pyrite. There was no arsenopyrite present as quantified by the mineralogy. Practically no iron was contained in the non-sulphides.*

Metallurgical process development studies conducted on samples from Lundberg and Engine House by SGS focused on base metals; copper-lead followed by a zinc flotation stage. The current test program was developed to determine the effectiveness of the proposed flowsheet shown in Figure 13.7 on a composite sample, from the deposit Met #2 and Met #4. Met #2 (used in lock cycle test #1 (LCT 1)) was of a higher grade than Met #4 (used in lock cycle test #2 (LCT2) and lock cycle test #3 (LCT 3)). Since LCT 2 and LCT 3 were carried out using the same sample, both test were compared wherever there were perceived similarities.

13.1 ORE COMPOSITION

The ore composition is represented as calculated head assays for the respective test work outlined in Table 13.1. Met #2 represents that obtained for LCT1 and Met #4 for LCT 2 and LCT 3.

Table 13.1 Ore Composition

Head Assays (Calculated)	Flotation Tests	Pb (Wt. %)	Cu (Wt. %)	Zn (Wt. %)	S (Wt. %)	Ag (g/t)
Met #2	LCT 1	1.91	0.42	3.76	12.7	9.22
Met #4	LCT 2	0.66	0.68	1.80	10.7	-
Met #4	LCT 3	0.64	0.65	1.75	10.0	8.8

The test program aimed to provide a suitable foundation on which to base future metallurgical investigations:

- Preliminary estimate of basic grinding and processing of the mineralization to produce the optimal copper, lead and zinc concentrates, as well as determining the power and reagent cost to produce three marketable concentrates.
- Development of an appropriate laboratory scale flowsheet for treating a representative sample of Buchans mineralization. The intention is to derive a simple, operable, and reliable flowsheet with low circulating loads, and a simple reagent scheme to complement the process design.
  - The testwork, carried out by SGS was based on a sample that contained 0.64% lead, 0.65% copper, 1.75% zinc, 10.0% sulphur and 8.8% silver from which separate concentrates of 73.9% lead, 53% zinc, 24.1% copper and 359.5 g/t silver were achievable. Corresponding recoveries were 78.7%, 89.3% and 66.2% and 27.8% (silver in Pb 4th Cl Conc) respectively.
  - The estimated reagent consumption by category is outlined in Table 13.2.

Table 13.2 Reagent Consumption

Species	Consumption (g/t)
Collectors	
SIPX	42.5
3418A	22.5
Frothers	
U250/MIBC (50-50)	17.5
Modifiers	
Lime	1800
H_2SO_3	180
Activators and Depressants	
$CuSO_4$	250
NaCN	470
Other	
$ZnSO_4$	250
Activated Carbon	100

- This investigation also determined the process stages required to produce saleable grade copper, lead and zinc concentrates.
- Additionally, the deportment of deleterious and precious metal components was tracked through the proposed circuit (Table 13.3). This was required to determine the levels of minor elements that might influence smelter

acceptance of these products. No deleterious components, in excessive amounts, that would incur smelter penalties have been identified.

Table 13.3 Distribution of Deleterious and Metals throughout the Proposed Circuit

Sample ID	Cl (g/t)	F (%)	Hg (g/t)	Al (g/t)	As (g/t)	Ba (g/t)	Be (g/t)	Bi (g/t)	Ca (g/t)	Cd (g/t)	Co (g/t)	Cr (g/t)	Fe (g/t)
LCT 3 Pb 4th CL Conc F	20	<0.005	0.6	351	<30	181	<0.03	903	259	133	<20	6	28400
LCT 3 Pb 1st Cl Sc Tail F (Cu Conc)	74	<0.005	1.1	1510	<30	363	<0.03	206	1070	224	<20	89	284000
LCT 3 Pb Ro Tail F (Cu Conc)	93	0.006	1.0	2020	<30	584	<0.03	< 100	1570	289	<20	60	292000
LCT 3 Zn 3rd Cl Conc F	84	<0.005	4.7	1110	<30	493	<0.03	< 100	571	2050	<20	186	45800

Sample ID	K (g/t)	Li (g/t)	Mg (g/t)	Mn (g/t)	Mo (g/t)	Na (g/t)	Ni (g/t)	P (g/t)	Sb (g/t)	Se (g/t)	Sn (g/t)	Sr (g/t)	Ti (g/t)
LCT 3 Pb 4th CL Conc F	<200	< 30	412	34.9	342	34	<30	<200	<50	<30	<20	1.82	53
LCT 3 Pb 1st Cl Sc Tail F	<200	< 30	1440	99.7	302	75	<30	<200	<50	<30	<20	5.49	67
LCT 3 Pb Ro Tail F	<200	< 30	2270	191.0	129	107	<30	<200	<50	<30	<20	6.67	107
LCT 3 Zn 3rd Cl Conc F	<200	< 30	1160	109.0	182	63	<30	<200	<50	<30	<20	5.01	42

Sample ID	Tl (g/t)	U (g/t)	V (g/t)	Y (g/t)	In (%)	Sc (ppm)	Ru (ppm)		Pb (%)	Cu (%)	Zn (%)	S (%)	
LCT 3 Pb 4th CL Conc F	<30	<60	<2	0.3	<0.002	<5			73.9	1.0	4.5	16.1	
LCT 3 Pb 1st Cl Sc Tail F	<30	<60	7	0.8	0.005	<5			2.3	26.5	5.2	33.6	
LCT 3 Pb Ro Tail F	<30	<60	9	0.9	0.004	<5			1.0	22.7	6.2	34.0	
LCT 3 Zn 3rd Cl Conc F	<30	<60	5	0.4	0.003	<5			1.3	2.0	53.0	32.6	
									From LCT 3 Mass Balance				

13.2 TEST WORK PROGRAM

13.2.1 *BOND WORK INDEX*

Bond Work Index test results showed an index of 16.1 (metric) kW/t or 14.6 (imperial) kW/t

13.2.2 *HEAVY LIQUID SEPARATION*

SGS (Imeson, 2011) reported the following:

The heavy liquid separation tests carried out on Met 4 composite are summarised in the graphs below. The tests were carried for samples with S.G in the range 2.7 to 2.95. Results from this investigation show that less than 4% losses at 25% mass rejection was achieved when combining 2.75 sinks in the fines. There appears to be an optimum S.G. in the range 2.7 – 2.95 that may be worth pursuing.

Figure 13.1 HLS Balance 2.7-2.95 Weight Recovery (%)

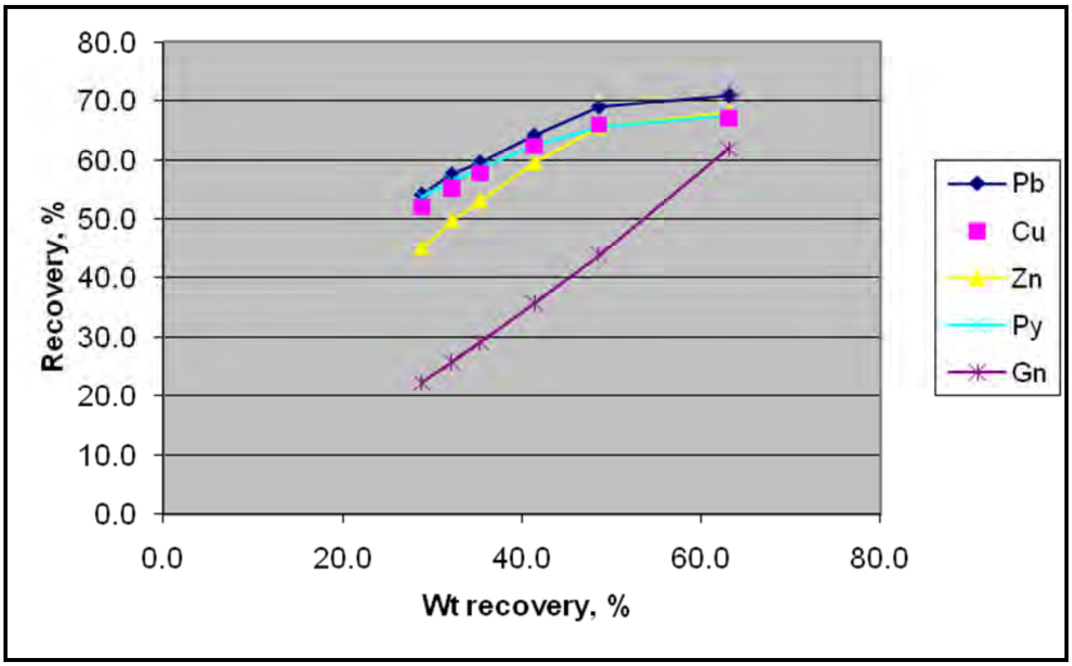
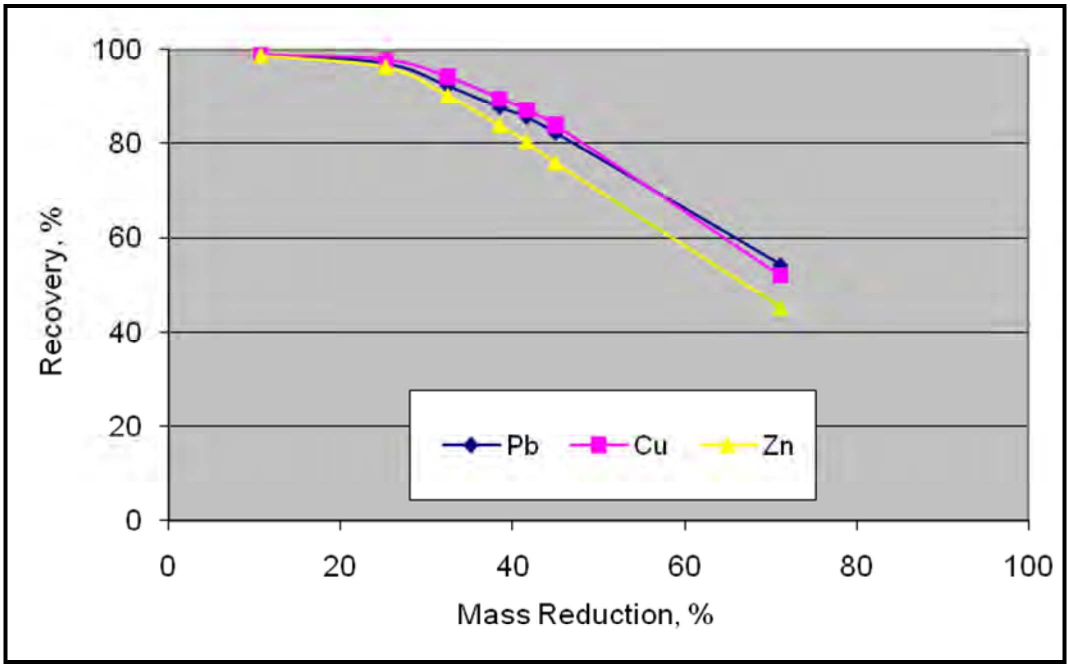


Figure 13.2 HLS Balance 2.7-2.95 Mass Reduction (%)



13.2.3 LOCK CYCLE TEST 1

LCT 1 was carried out to investigate the flotation response for sample Met #2. Concentrate grades for Pb Cu, Zn and silver were 59.1%, 17.7%, 53% and 135 g/t (Pb 4th cleaner conc.) .% respectively with corresponding recoveries of 93.1, 61.1, **78**.6% and 44.1%, respectively. Since the feed grade was high for Pb and Zn and a separate sample was used in subsequent tests, findings of this investigation could not be included in the test comparison.

Lead grade was marginally low since industry benchmark that informs payments, smelter returns, treatment costs and penalties is at 60%.

Table 13.4 LCT 1 Results of Metallurgical Projection - Cycles D-E (Product Output*)

Product	Weight		Assays (%) (g/t)						% Distribution					
	g	%	Pb	Cu	Zn	Au	Ag	S	Pb	Cu	Zn	Au	Ag	S
Pb 4th Cl Conc	592.5	3.00	59.1	1.92	11.6	0.28	135	20.1	93.1	13.7	9.3	7.9	44.1	4.8
Pb 1st Cl Sc Tail (Cu Conc)	132.3	0.67	2.79	22.0	9.33	0.52	47.0	35.0	1.0	35.0	1.7	3.3	3.4	1.8
Pb Ro Tail (Cu Conc)	154.8	0.78	2.60	14.0	25.5	1.21	77.0	32.8	1.1	26.1	5.3	9.1	6.5	2.0
Combined Cu Conc	287.0	1.45	2.69	17.7	18.0	0.89	63.2	33.8	2.0	61.1	7.0	12.3	10.0	3.9
Zn 3rd Cl Conc	1102.0	5.58	0.73	1.55	53.0	0.23	28.1	32.3	2.2	20.5	78.6	12.2	17.0	14.2
Zn 1st Cl Sc Tail	1799.5	9.11	0.13	0.06	0.34	0.17	7.3	12.4	0.6	1.3	0.8	14.3	7.2	8.9
Zn Ro Tail	15814.0	80.1	0.05	0.02	0.21	0.07	2.5	10.8	2.0	3.4	4.4	53.3	21.7	68.2
Zn Combined Tail	17613.5	89.2	0.06	0.02	0.22	0.08	3.0	11.0	2.7	4.7	5.2	67.6	28.9	77.1
Head (calculated)	19595.0	100.0	1.91	0.42	3.76	0.11	9.22	12.7	100.0	100.0	100.0	100.0	100.0	100.0

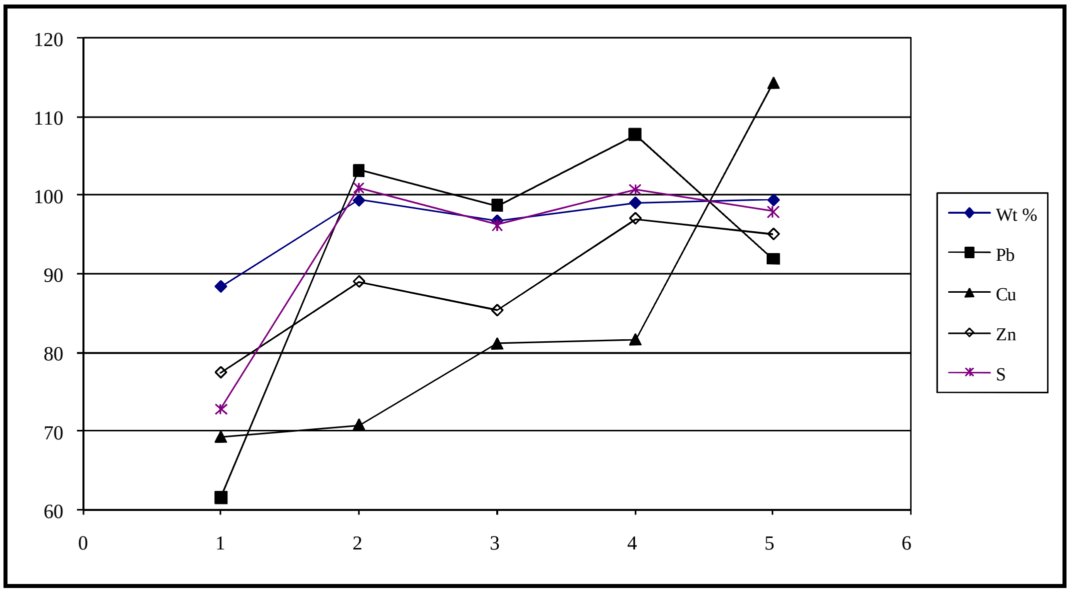
STABILITY

Satisfactory stability was not achieved in the test. Metal units out as a percentage of metal units in fluctuated between 92 and 108% for Pb and 81 and 114% for Cu. This, therefore, introduced uncertainty in the results. There was significant amount of copper still in the circulating products that had not come out after six cycles.

Table 13.5 LCT 1 Metal Stability

	Weight		Assays (%) (g/t)			
	g	%	Pb	Cu	Zn	S
Total In All Cycles	19748	100.0	1.91	0.43	3.92	12.76
Average In Per Cycle	3950	20.0				
	Weight		Units out as a % of Units in/Cycle			
Total Products Out Per Cycle	g	Wt %	Pb	Cu	Zn	S
Cycle A	3489	88.3	61.5	69.2	77.5	72.7
Cycle B	3924	99.3	103.1	70.8	88.9	100.9
Cycle C	3822	96.8	98.6	81.1	85.3	96.2
Cycle D	3912	99.0	107.7	81.6	96.9	100.7
Cycle E	3926	99.4	91.9	114.2	95.0	97.9
Average of C-E		99.2	99.8	97.9	95.9	99.3

Figure 13.3 LCT 1 Metallurgical Accounting of Metals



13.2.4 LOCK CYCLE TEST 2

LCT 2 was carried out with a separate sample to LCT 1, Met #4. Again the results were encouraging but with reduced lead recovery and lower than expected lead grade. The results of LCT 2 are outlined in Table 13.6.

The respective concentrate grades for Pb, Cu and Zn were 57.8, 24.2 and 54.7%, with corresponding metal recoveries of 81.6, 60 and 70.8%, respectively.

Table 13.6 Results of LCT 2

Product	Weight		Assays (%)				% Distribution			
	g	%	Pb	Cu	Zn	S	Pb	Cu	Zn	S
Pb 4th Cl Conc	270.7	0.9	57.8	4.74	5.14	19.7	81.6	6.4	2.6	1.7
Pb 1st Cl Sc Tail (Cu Conc)	171.4	0.6	1.29	26.4	6.35	29.8	1.2	22.7	2.1	1.6
Pb Ro Tail (Cu Conc)	323.4	1.1	2.39	23.0	9.11	32.0	4.0	37.3	5.6	3.3
Combined Cu Conc	494.8	1.7	2.01	24.2	8.15	31.2	5.2	60.0	7.7	4.9
Zn 3rd Cl Conc	679.4	2.3	1.12	1.35	54.7	32.3	4.0	4.6	70.8	7.0
Zn 1st Cl Sc Tail	2150.2	7.3	0.26	2.09	0.29	17.8	2.9	22.6	1.2	12.2
Zn Ro Tail	25659.4	87.7	0.047	0.049	0.36	9.01	6.3	6.4	17.7	74.1
Zn Combined Tail	27809.6	95.1	0.064	0.21	0.36	9.69	9.2	29.0	18.9	86.3
Head (calculated)	29254.5	100.0	0.66	0.68	1.80	10.7	100.0	100.0	100.0	100.0

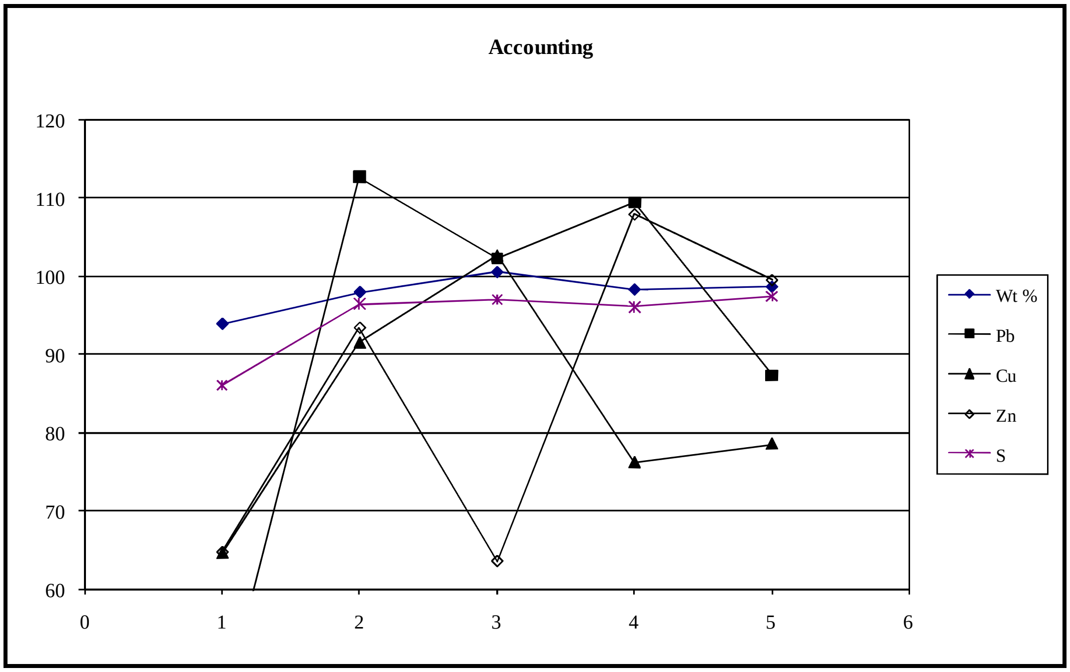
STABILITY

Lead and zinc demonstrated reasonable stability while copper stability was extremely poor.

Table 13.7 LCT 2 Metal Stability

	Weight		Assays (%)			
	g	%	Pb	Cu	Zn	S
Total In All Cycles	59173	100.0	0.65	0.67	1.78	10.6
Average In Per Cycle	9862	16.7				
Total Products Out Per Cycle	**Weight**		**Units out as a % of Units in/Cycle**			
	g	**Wt (%)**	**Pb**	**Cu**	**Zn**	**S**
Cycle A	9251	93.8	44.1	64.5	64.7	86.0
Cycle B	9653	97.9	112.6	91.5	93.4	96.4
Cycle C	9908	100.5	102.2	102.6	63.6	97.0
Cycle D	9688	98.2	109.4	76.1	107.8	96.0
Cycle E	9722	98.6	87.3	78.5	99.4	97.4
Cycle F	9845	99.8	100.3	83.6	96.3	101.1
Average of D-F		98.9	99.0	79.4	101.2	98.2

Figure 13.4 LCT2 Metallurgical Accounting of Metals



13.2.5 LOCK CYCLE TEST 3

Prior to the completion of LCT 3, SGS carried out a series of batch tests to re-examine the use of Fe depressants, the effect of dispersant, and primary grind size.

It was determined that the inclusion of H_2SO_3 in the primary grind together with the $ZnSO_4$/NaCN applied in LCT 2 enhanced Fe depression.

In addition, a finer primary grind appeared to provide an improved selective response against Zn flotation in the Cu-Pb rougher stage. Dispersant was found to be ineffective.

On the basis of these batch tests, LCT 3 was completed at a finer grind P80 of 52 µm versus 78 µm and the alternative depressant scheme

The results of LCT 3 are outlined in Table 13.8.

- The results of the LCT 3 were used to develop the determinations for this phase of the project. Concentrate grades of respective metals lead, copper, zinc and silver were 73.9%, 24.1% and 53% and 359.5 g with corresponding recoveries of 78.7, 89.3 , 66.2% and 27.8 (Pb 4th cleaner concentrate), respectively. Significant silver was reported to the lead concentrate-359.5 g/t, compared to the copper and zinc concentrates with 37.8 and 33 g/t respectively.

- On comparison with LCT 2, LCT 3 performed well giving slightly lower Pb recoveries at a lot higher Pb grade. The Cu recovery improved dramatically by 30% in LCT 3 at roughly the same grade as LCT 2. While this may be partly explained by the perceived increase in liberation due to the finer primary grind, it may be more related to the improved copper stability in the test. The Zn recovery was a little lower in LCT 3 at roughly the same grade as in LCT 2. LCT 3 was compared with LCT 2 as both were performed on the same sample (#4)

The results of this testing program are based on a single ore composite sample, Met#4. The results are therefore indicative. There results were also not optimized for reagent consumption and metal recovery versus concentrate grade. Un-optimized recovery for copper was encouraging at 89.3% for a marginally smelter acceptance grade of 24.1% (industry benchmark that inform payments, smelter returns, treatment costs and penalties is set at greater than 24% minimum).

- While the current results are preliminary, a concern is highlighted in the relatively low recovery of zinc (66.2%); hence future investigations should be geared to gaining a better understanding of this circuit.
- The optimization investigations of future drill hole and sampling campaign should present an opportunity to develop on the findings of this investigation.

Table 13.8 LCT 3 Metallurgical Projection - Cycles D-F (Product Output*)

Product	Weight		Assays (%)					% Distribution				
	g	%	Pb	Cu	Zn	S	Ag	Pb	Cu	Zn	S	Ag
Pb 4th Cl Conc	214.4	0.7	73.9	1.03	4.55	16.1	359.5	78.7	1.1	1.8	1.1	27.8
Pb 1st Cl Sc Tail (Cu Conc)	276.3	0.9	2.28	26.5	5.20	33.6	45.2	3.1	35.4	2.6	2.9	4.5
Pb Ro Tail (Cu Conc)	491.5	1.6	1.04	22.7	6.23	34.0	33.7	2.5	53.9	5.5	5.3	6.0
Combined Cu Conc	767.8	2.4	1.48	24.1	5.86	33.8	37.8	5.6	89.3	8.1	8.2	10.5
Zn 3rd Cl Conc	690.4	2.2	1.33	1.96	53.0	32.6	33.0	4.6	6.6	66.2	7.1	8.2
Zn 1st Cl Sc Tail	2532.3	8.0	0.16	0.14	0.47	21.6	16.6	2.0	1.8	2.2	17.4	15.2
Zn Ro Tail	27442.0	86.7	0.045	0.041	0.37	7.82	3.87	6.1	5.4	18.5	68.1	38.3
Zn Combined Tail	29974.3	94.7	0.054	0.05	0.38	8.99	4.95	8.1	7.1	20.7	85.4	53.5
Head (calculated)	31646.9	100.0	0.64	0.65	1.75	10.0	8.8	100.0	100.0	100.0	100.0	100.0

STABILITY

Test LCT-3 achieved better overall stability than the previous locked cycle tests. The certainty in the projected balance was therefore improved.

Figure 13.5 LCT 3 Metallurgical Accounting of Metals

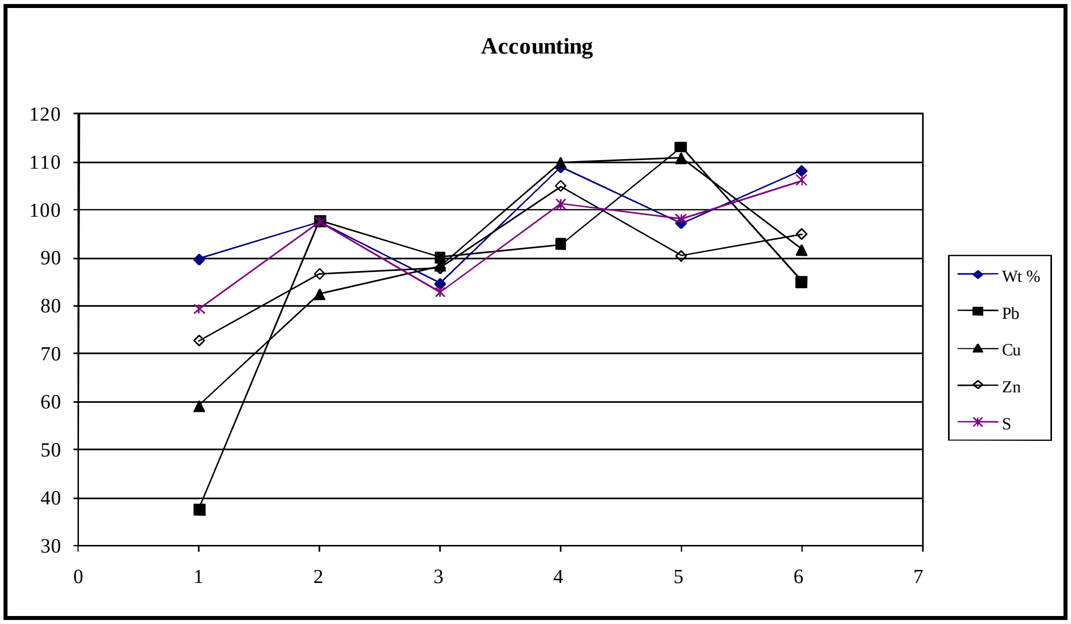


Table 13.9 LCT 3 Stability of Metals

	Weight		Assays,%			
	g	%	Pb	Cu	Zn	S
Total In All Cycles	60395	100.0	0.67	0.69	1.83	10.4
Average In Per Cycle	10066	16.7				
Total Products Out Per Cycle	**Weight**		**Units out as a % of Units in/Cycle**			
	g	**Wt (%)**	**Pb**	**Cu**	**Zn**	**S**
Cycle A	9032	89.7	37.5	59.2	72.8	79.4
Cycle B	9824	97.6	97.7	82.4	86.6	97.5
Cycle C	8524	84.7	90.1	88.4	87.8	82.8
Cycle D	10962	108.9	92.8	109.8	105.0	101.2
Cycle E	9796	97.3	113.1	110.8	90.4	98.2
Cycle F	10889	108.2	85.0	91.7	94.9	106.2
Average of D-F		104.8	97.0	104.1	96.7	101.9

13.2.6 INVESTIGATION OF FLOTATION RESPONSE OF BARITE

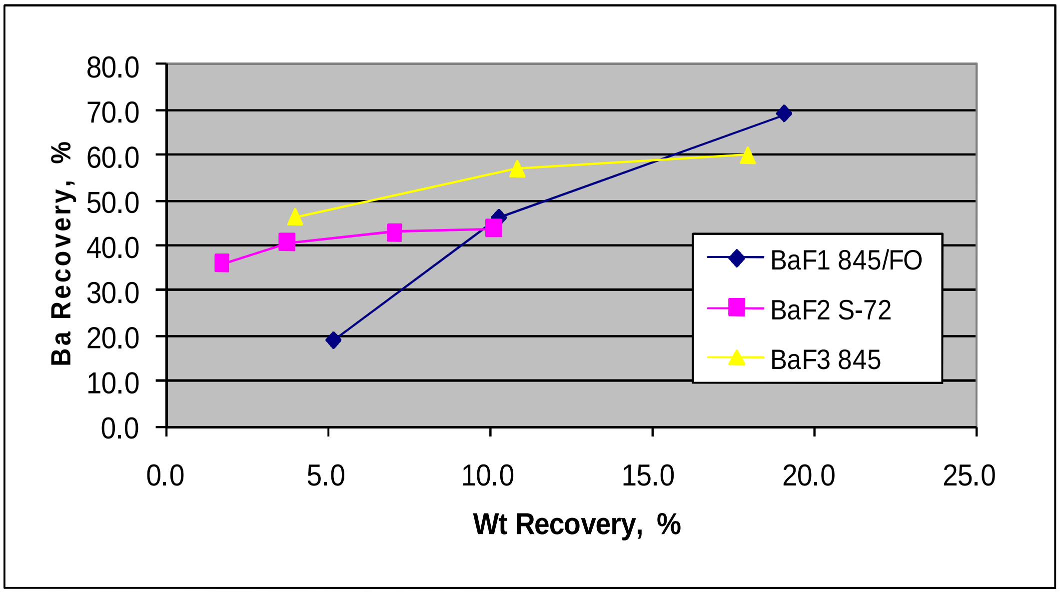
SGS conducted preliminary investigations to assess the potential of recovering barite from the Zn rougher tailings. The following parameters were investigated:

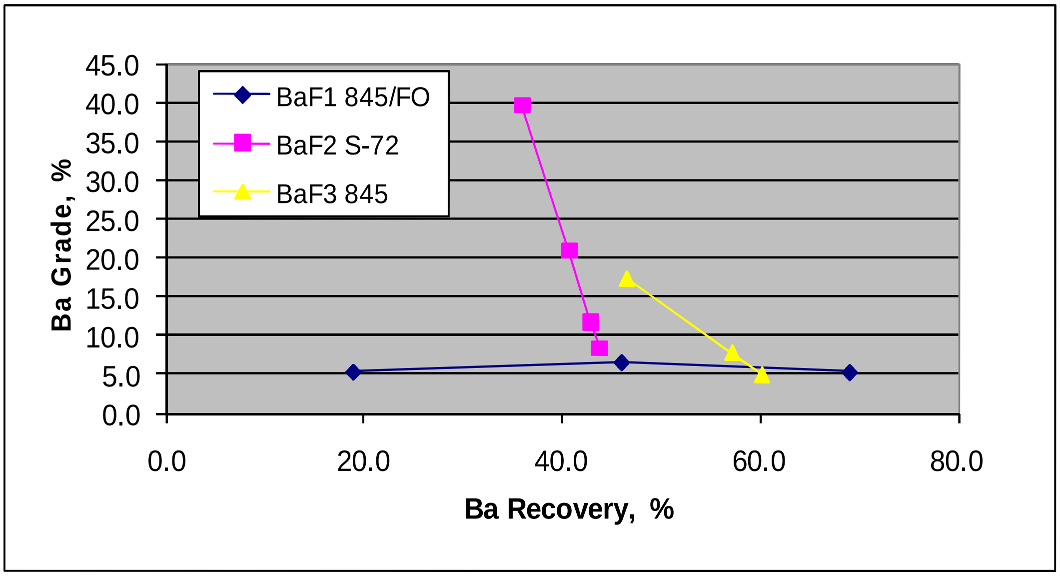
- A conceptual flowsheet included recovering residual sulphides (primarily pyrite) prior to recovery of the barite.
- A regime of lowering the pH with sulphuric acid to 6.5 together with additions of potassium amyl xanthate was capable of recovering approximately 85-90% of the pyrite in the Zn rougher tailings.

It was determined that desliming the pyrite rougher tailings was likely required prior to barite flotation. The desliming process lost about 10-40% of the barium into a comparable percentage of the mass. A dispersant (sodium silicate) together with several collectors (Aero 845 and S-72 manufactured by Clariant) were evaluated. In all cases, >95% of the residual barium after desliming was recovered into a rougher concentrate.

Further testing is required to properly develop the barite recovery flowsheet.

Figure 13.6 Evaluation of Dispersants and Collectors





13.3 PROCESS FLOWSHEET DESCRIPTION

The general process design is as per LCT 3 as conducted by SGS. A description of the overall flotation circuit follows. BMC is proposing a mill of nominal daily output of 5,000 t/d for the Lundberg and Engine House deposits.

The ore will be ground to 80% passing 52 µm. The ore will be pumped to the bulk copper/ lead flotation circuit. The rougher copper/lead concentrate will be reground before flowing to the copper /lead cleaning circuit for production of the bulk copper /lead concentrate. The final bulk copper/lead concentrate will be conditioned with depressants $ZnSO_4$ and NaCN. The slurry will be advanced to the Cu/Pb separation circuit. Copper concentrate will be depressed in the rougher and scavenger cells, then cyanide will be added to the scavenger tails (Pb cleaner feed junction box for Cu depression and the final lead concentrate will be produced. The Cu/Pb separation rougher, scavenger and Pb cleaner tails will comprise the copper concentrate.

The bulk copper/lead scavenger tails slurry will feed the zinc rougher-scavenger flotation circuit. The rougher concentrate will be treated in the second zinc cleaner circuit and the scavenger concentrate and second zinc cleaner tailings will feed the first cleaners.

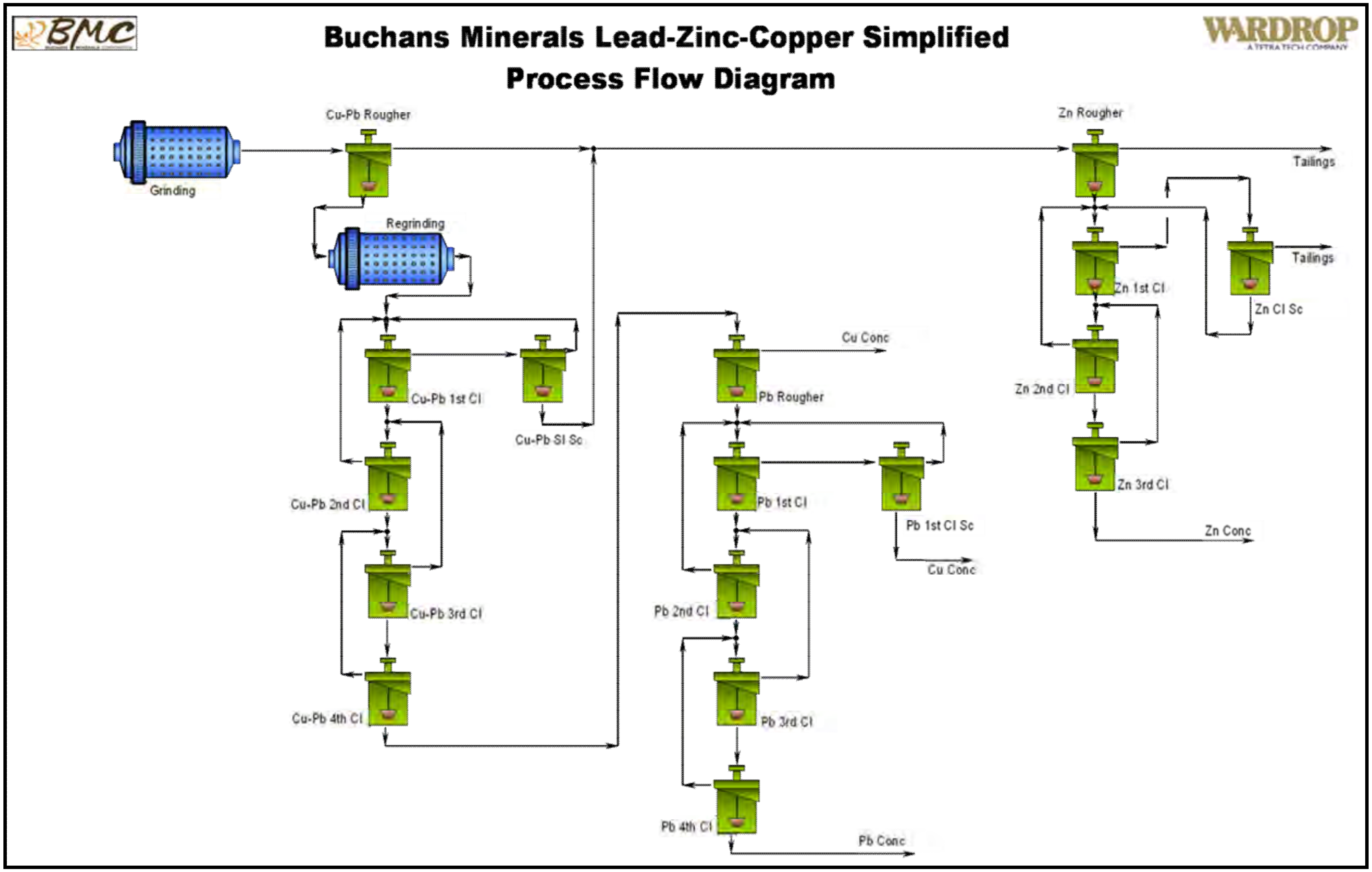
The cleaning of the zinc concentrate will be carried out in three stages. The first zinc cleaner concentrate and third zinc cleaner tails will feed the second zinc cleaners. Second zinc cleaner concentrate will feed the third zinc cleaner circuit to produce the final zinc concentrate.

Lime will be added to the zinc cleaning circuits to reject iron and allow the production of high grade zinc concentrate.

Included in this flowsheet is the consideration that all intermediate streams in all the circuits, utilizes conventional counter-current recirculation.

Additionally, for all circuits, the first cleaner scavenger tailings are rejected. This serves to reduce circulating loads beneficial to commercial operations.

Figure 13.7 Proposed Buchans Minerals Lead-Zinc-Copper Simplified Process Flow Diagram



13.4 PATH FORWARD

The following areas were identified for further investigation as we progress to future phases of the project:

- Environmental Analyses- The effluent from the final cycle of the lock cycle test will have to be analysed for mg/l CN, sulphide and acid base accounting to include multi-element analysis.
- Pyrite accumulation should be confirmed- this has implication for the tailings management facility and need for pyrite recovery.
- In order to confirm process ability, variability testing with additional samples and of significant difference in ore composition should be conducted.
- Optimization of main concentrates Cu, Zn and Pb for reagent consumption in relation to concentrate grade versus recovery.
- The upstream crushing and downstream operations of tailings management and concentrate treatment require further development in preparation for the receipt of the mill concentrates Cu, Zn, and Pb by the smelter.
- The next program should optimize the process variables including i) primary grind size, ii) depressant schemes, iii) regrind requirements
- The recovery of pyrite from the Zn tailings should be further investigated to assess the potential of generating a final non-acid generating tailings stream.
- Further testing to better develop a process for recovering barite should be completed if this is considered a priority for the project

14.0 MINERAL RESOURCE ESTIMATES

The following section is taken from Webster and Barr (2008).

14.1 GENERAL

The definition of mineral resource and associated mineral resource categories used in this report are those recognized under NI 43-101 and set out in the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines. Assumptions, metal threshold parameters, and deposit modeling methodologies associated with the current resource estimate are discussed below in report Sections 14.2 through 14.4.

Mercator and the authors were contracted to assist in the design and implementation of a diamond drilling program to test and verify base metal mineralization associated with the Lundberg and Engine House deposits, to confirm existing mineralization outlined by the ASARCO, and to improve the confidence in the distribution of mineral grade. These resource estimates have been completed from a database compiled from new and existing drill hole data and verified by Mercator. The database has been fully validated and formatted such that it is suitable for use in mineral resource estimation.

14.2 GEOLOGICAL INTERPRETATION USED IN RESOURCE ESTIMATION

In total, 53 diamond drill holes (comprising 8,057.82 m) were drilled vertically from surface over the Lundberg and Engine House deposits. Lithological descriptions were captured in digital format and displayed on geological cross-sections with corresponding assay results in conjunction with historical surface and underground diamond drilling results. Lithological and mineralogical correlations were determined based on these cross-sections. For the purpose of this resource estimate, mineralogical correlation was the defining parameter for the continuity and distribution of mineral grade used to confine the resource.

14.3 METHODOLOGY AND RESOURCE ESTIMATION

14.3.1 OVERVIEW OF CURRENT ESTIMATION PROCEDURE

These resource estimates reflect a three-dimensional deposit block model developed by Mercator using Surpac© Version 6.0.3 deposit modeling software.

Analytical results for 178 diamond drill holes were used to calculate the resource estimate in this model, of which 42 drill holes are from recent Company drilling, and 136 drill holes are from validated historic data. The model utilized 1 m down-hole assay composites individually calculated for Zn (%), Pb (%), Cu (%), Ag (g/t), Au (g/t), and $BaSO_4$ assay values. Model blocks measured 5 m x 5 m x 5 m with sub-blocking at 2.5 m x 2.5 m x 2.5 m within the Lundberg solid, and 2.5 m x 2.5 m x 2.5 m with sub-blocking at 1.25 m x 1.25 m x 1.25 m for the Engine House solid.

The model was constrained by individual wireframed solids representing resource estimates for the Lundberg and Engine House deposits respectively. The wireframes were outlined from geological sections and reflect a minimum included grade of approximately 1% combined base metal (Zn% + Pb% + Cu%). Internal dilution within individual drill holes was limited to a maximum of 40% of the overall drillhole intersection. No high-grade capping factors were applied to high-grade samples. The Lundberg resource solid occurs between the bedrock-overburden interface in the east and plunges to a maximum depth of approximately 350 m below surface in the west. The Engine House wireframe solid ranges in elevation from between 25 m to a depth of approximately 145 m below surface. Historical underground development was reviewed and where information was available, was modeled into three-dimensional solids for volume purposes. All underground workings volumes occurring within the resource block model volumes were removed from the resource estimate after the grade interpolation process was completed.

Metal grades were assigned to the block model using ID^2 interpolation methodology with blocks being peripherally constrained by wireframe solids.

The Lundberg solid incorporated two interpolation domains occurring north and south of gridline 7930N. Two unique interpolation ellipses were determined for these two domains. Major and minor axis parameters were selected based on continuity and distribution of metal grade and reflect geological characteristics of the mineralized zones. The domain south of line 7930N used an ellipse aligned at 305° azimuth (Az) with a dip and plunge of -7.5° and 7.5° respectively. The domain north of line 7930N used an ellipse aligned at 305° Az, with a dip and plunge of -25° and -25° respectively. Both ellipses have a major and semi-major axis range of 75 m and minor axis range of 37.5 m in order to preserve the relatively sub-horizontal character of mineralization.

The Engine House estimate incorporated four solids, each having independent interpolation domains, and used an isotropic model with a 75 m range. Three of the domains were modeled to reflect isolated stringer style mineralization in pyroclastic volcanics located below the main stockwork mineralization.

Results of 1,577 1 m composites of separate laboratory determinations of SG were used in the block model. The mean SG value from within the resource solids of 2.88 g/cm^3 was assigned to blocks occurring within the Lundberg and Engine House models.

The results of the mineral resource estimate can be seen as a schematic in Figure 14.1, where the grade cut-offs of 1% combined base metal (Zn% + Pb% + Cu%), 1% Zn, 1.5% Zn, 2% Zn, and 3% Zn can be seen as color contours of the extreme outline projected to surface. Figure 14.2 outlines the block model on an east-west vertical cross-sections through line 7,850N (A-A'), and Figure 14.3 outlines the block model in a north-south section through line 10,000E (B-B'). The Lucky Strike glory hole and underground workings can be seen as white lines.

A calculation of the percentage of the mineral resource lying within 100 m of surface was completed for the client. The results of this are discussed in section 17.3.7 of this report.

14.3.2 CAPPING OF HIGH-GRADE ASSAYS

No high-grade capping factors were applied to high-grade samples. Through analysis of metal grade distribution, it was concluded that high values lay within zones where geology and mineralogy support the presence of high-grade material. Figure 14.4 to Figure 14.9 demonstrate the log-frequency distribution and descriptive statistics of the one metre grade composites for base metal mineralization within the Lundberg and Engine House. Quality control measures outlined in Section 12.0 have confirmed the validity of assay data from recent company drilling.

14.3.3 COMPOSITING OF DRILL HOLES AND STATISTICS

All assay information from historic and current drilling was reviewed, validated and added to the Surpac© database by Mercator. All assays lying spatially outside of the interpreted 1% combined base metal cut-off wireframe were excluded from compositing methods. Individual composites for Zn%, Pb%, Cu%, Ag g/t, Au g/t, and Ba% were calculated over one metre intervals in a downhole direction, starting with the first sample within the resource wireframe and continuing to the last. These composites were used as point data in determining the block grades in the model. In areas where there was no sampling, or missing assay data, no composites were created. If there was information from historic drill logs that indicated mineralization occurred in these areas, the model was allowed to interpolate over these holes. If there was no information available to determine if mineralization was present the holes were deleted from the database. This applied to four drill holes. In total, 4,574 composites were used in the block model interpolation. Figure 14.4 through Figure 14.9 represent the histogram distribution and descriptive statistics for the one metre composite dataset for individual metals used in the calculation of this estimate.

Figure 14.1 Schematic of Resource Cut-offs Projected to Surface

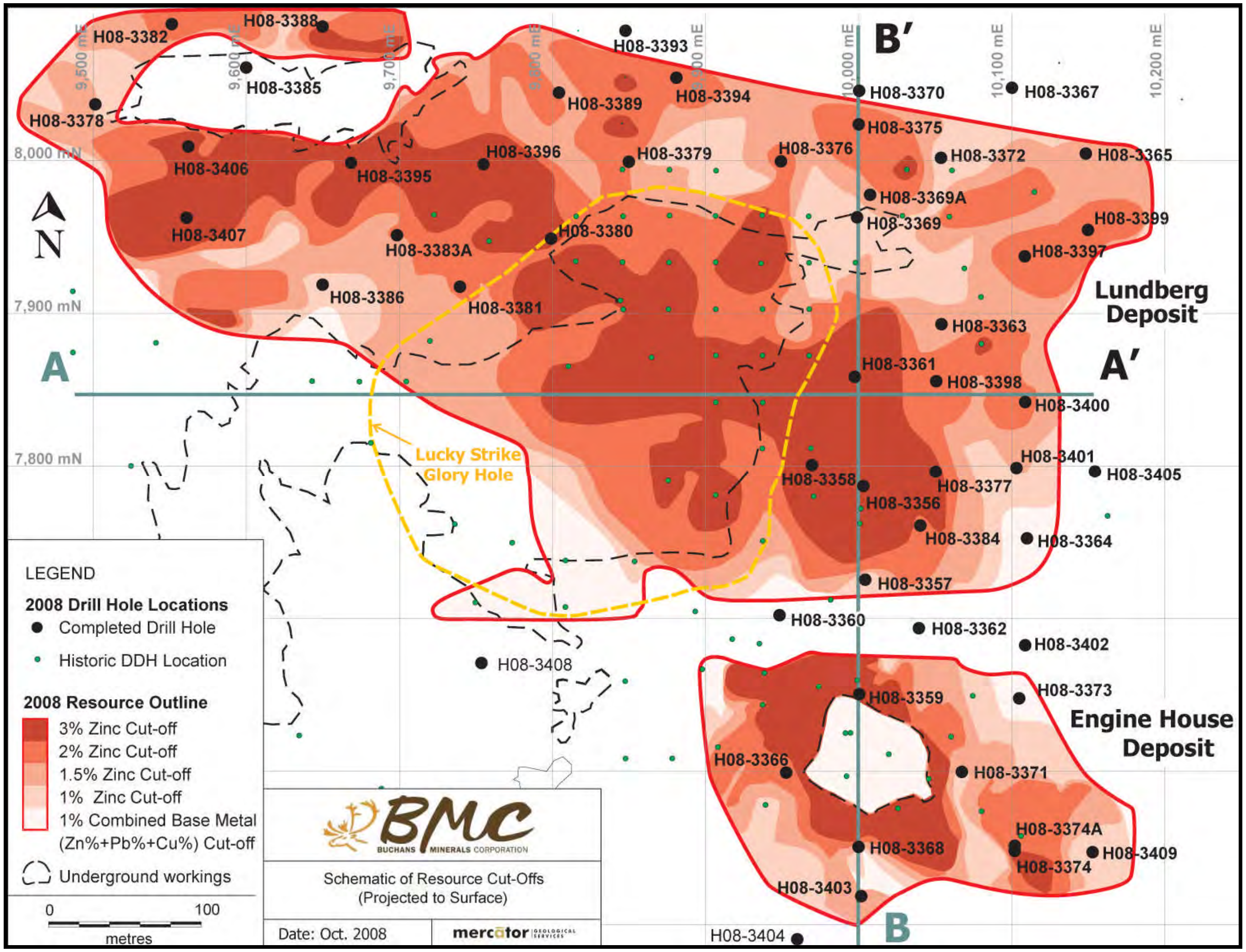


Figure 14.2 East-West Vertical Cross-sections Along line 7,850N

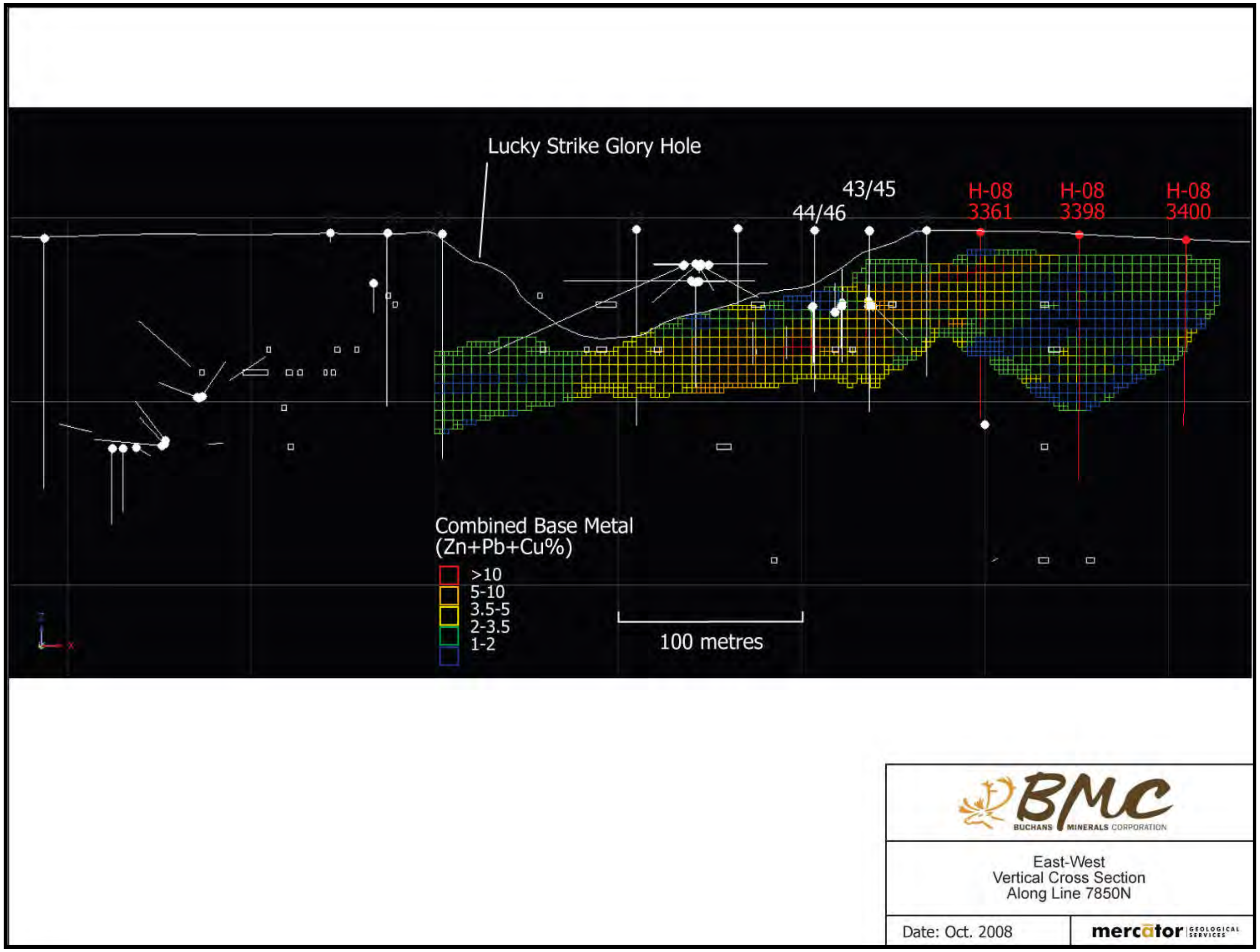
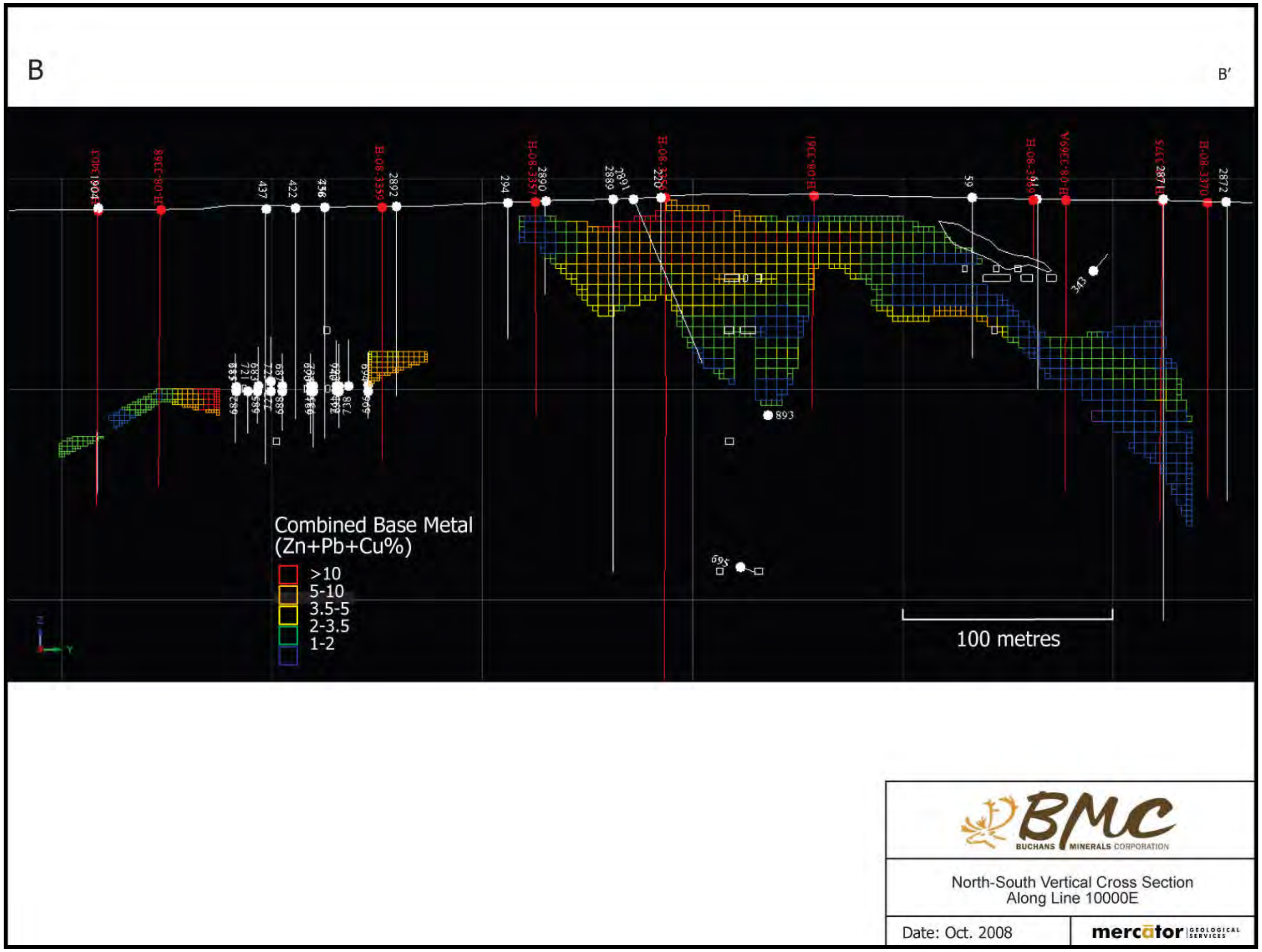


Figure 14.3 North-South Vertical Cross-sections Along line 10,000E



14.3.4 VARIOGRAPHY

An ID^2 interpolation method was used in the estimation of grade in the block modeling procedure, and as such no variography was performed in the calculation of the Lundberg or Engine House resource estimates.

14.3.5 SETUP OF 3-D BLOCK MODEL USED IN CURRENT RESOURCE

The block model extents were defined in local grid coordinates (metric) spanning from grid line 9,450E to 10190E and from line 7,500N to 9,000N. The local grid is oriented parallel to the UTM NAD83 (Zone 21) grid, and co-ordinates are obtained by subtracting 500,000 from the UTM easting and 5,400,000 from the UTM northing. Drillhole coordinates used in the resource model are included in Appendix B. The model extends from a maximum surface elevation of 300 m to -50 m (elevation relative to sea level datum, above sea level), with the nominal topographic surface around the Lundberg deposit being 195 masl. All resource solids respect the bedrock/overburden surface. Images of the block model can be seen in Figure 14.2 and Figure 14.3.

A standard block size for the model was established at 5 m x 5 m x 5 m with sub-blocking at 2.5 m x 2.5 m x 2.5 m within the Lundberg solid, and 2.5 m x 2.5 m x 2.5 m with sub-blocking at 1.25 m x 1.25 m x 1.25 m for the Engine House solid. A minimum sub-block size of 1.25 m x 1.25 m x 1.25 m was permitted to better constrain the model along geological, topographic and peripheral solid limits. Discretisation was 1 m x 1 m x 1 m and no block rotation was applied. The chosen block size locally smoothes the effects of intermittent stockwork and reduces overestimation of volume along mineral boundaries.

Resource estimation was completely constrained within a series of resource solids developed from systematic wireframing of the interpreted mineralized envelope limits. Vertical east-west and north-south cross-sections were used in creating the solids. The resource solids were developed using a minimum threshold of 1.00% combined base metal (Zn% + Cu% + Pb%) values over a minimum downhole length of 5 m. In areas with low assay values, 40% dilution of less than 1% combined material was included in the initial assessment of interpretation. For example, within any 5 m interval the model would accept 2 m of sub 1% combined material. In most cases, the Lundberg solid was constrained along the upper boundary of the Ski Hill Formation, under which the stockwork system was developed. In instances where mineralization was intersected by recent drilling in the Buchans River Formation, the grade would be included in the resource. The Engine House solids included both stockwork of the Ski Hill Formation and a narrow band of massive, presumably exhalative, sulphide contained in the lower Buchans River Formation.

Figure 14.4 Lundberg 1 m Zinc Composite Histogram

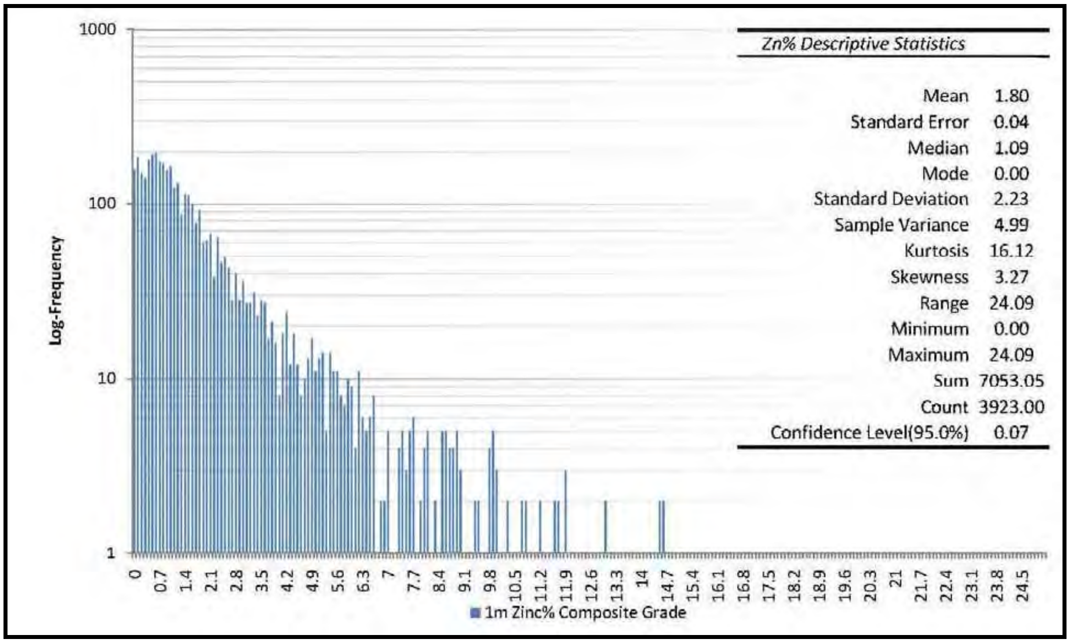


Figure 14.5 Lundberg, 1 m Lead Composite Histogram

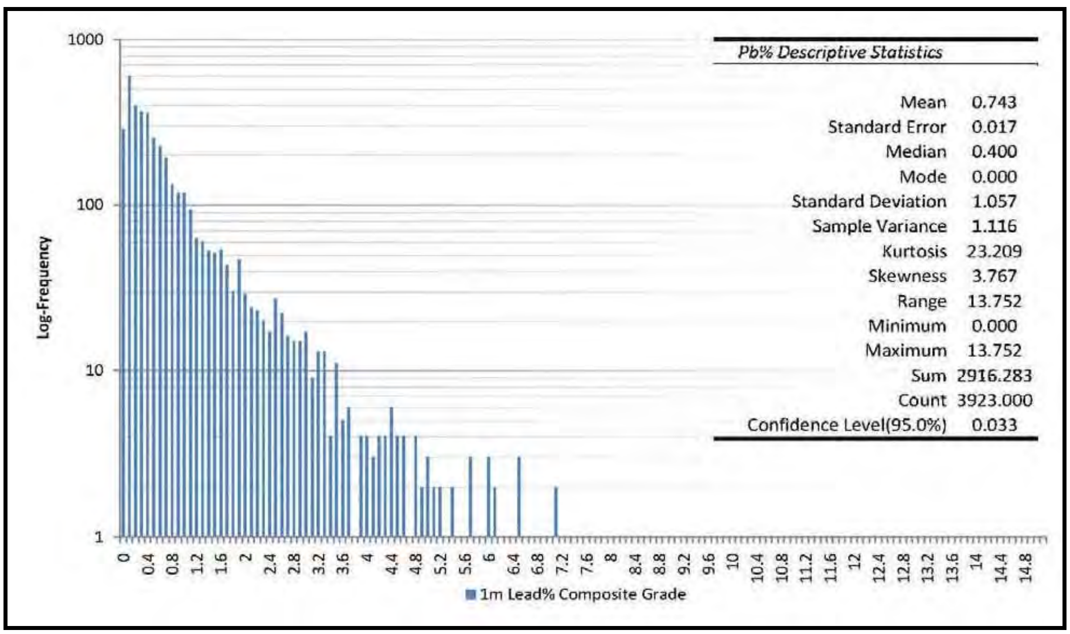


Figure 14.6 Lundberg, 1 m Copper Composite Histogram

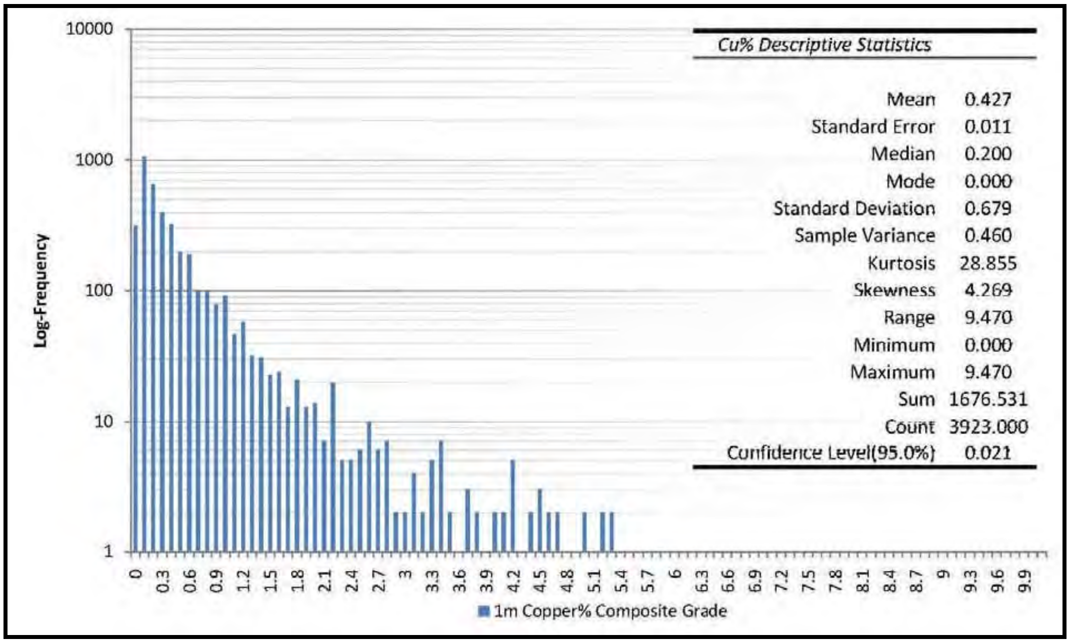


Figure 14.7 Engine House, 1 m Zinc Composite Histogram

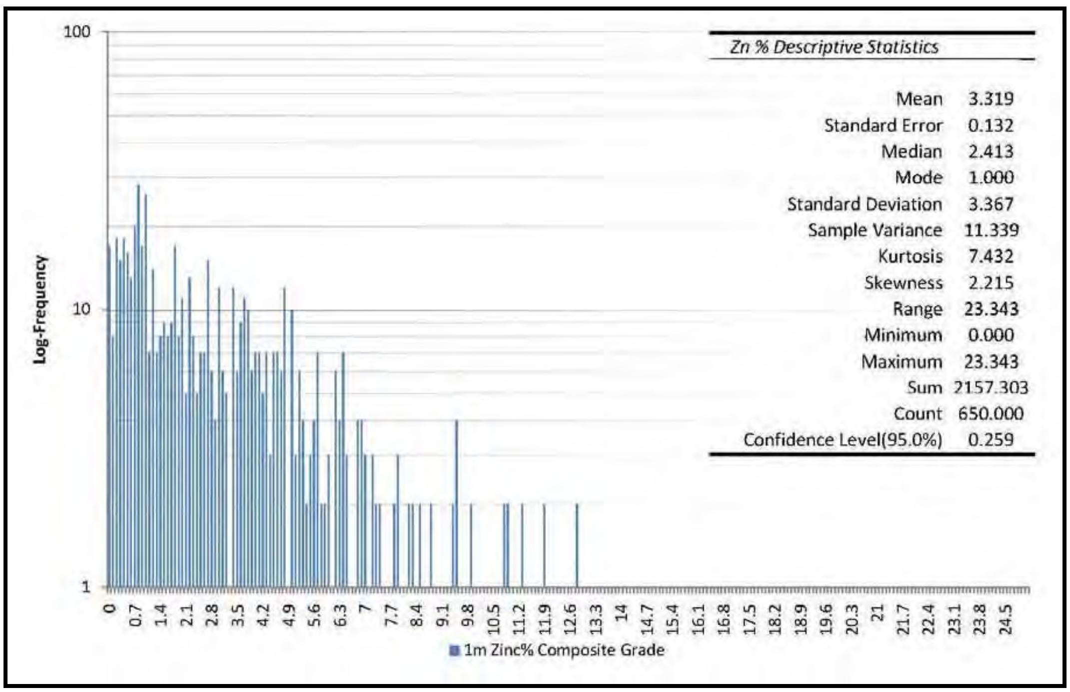


Figure 14.8 Engine House, 1 m Lead Composite Histogram

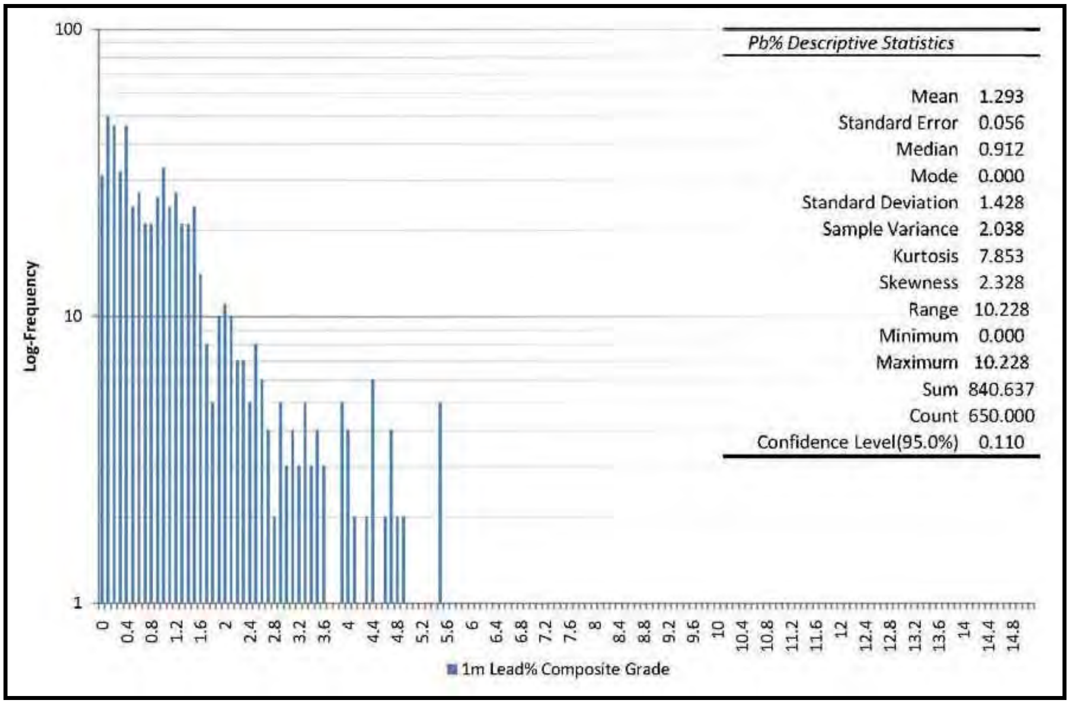
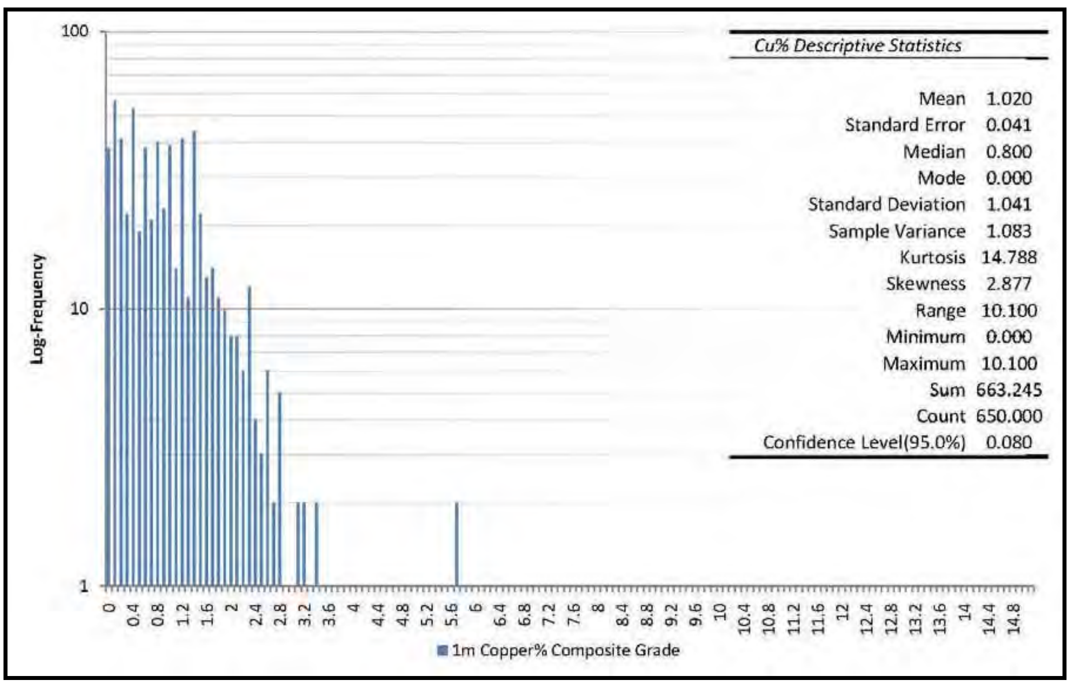


Figure 14.9 Engine House, 1 m Copper Composite Histogram



The limits of the resource extended 50 m horizontally from the last drill hole, or where the last drill hole was outside the wireframe, the midpoint between two drill holes was used. Any historic holes drilled from the surface or underground that lay within this boundary were included in the resource. The edges of the solid were interpreted and smoothed based on geology in areas of low density of drilling to avoid a jagged surface of the solid.

A three-dimensional model of the historic underground development was created from archived plan maps and sections, and volumes were developed based on the best information that was available. Where underground workings intersected model resource blocks, the associated volumes and grade of those blocks were removed from the final reported resource after the interpolation of the model was completed. Historic drillhole data that lay within the workings was used in the interpolation.

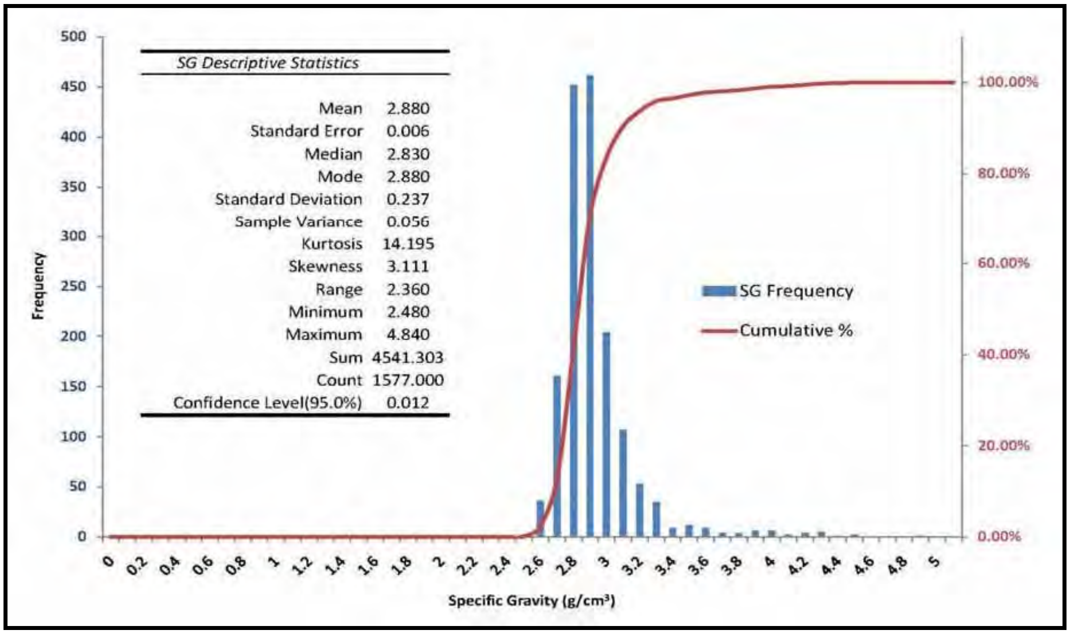
14.3.6 MATERIAL DENSITIES

Density information used in the resource estimate is based only on drill core data collected by Mercator staff during the current diamond drill program, as no historic data was available. All samples that lay within the base metal enriched mineralized zone, were analyzed using Ore Grade methods at Eastern Analytical and were then sent to ALS Chemex for check assay. In addition to quality assurance check assaying procedures, sample pulps were subjected to SG determinations using pycnometer methods (ALS method OA-GRA08b). In total 1,577 samples were analyzed, are relevant to and were used in this resource.

No information was obtained with reference to SG from historical drilling by ASARCO on this property. Sample data collected analyses by ALS Chemex as part of the current drilling is considered representative of the rock within the resource boundaries, and was incorporated into the database in order to determine a useable SG value. A detailed assessment of mineralogy and grade variability in relation to specific gravity is required to assign unique densities for volumes of rock within the Lundberg and Engine House deposits, due to their polymetallic nature.

The majority of mineralization occurs within a stockwork system hosted by mafic to intermediate volcanics with varying degrees of alteration. Preliminary analysis did not produce a useable proxy for density determination as related to mineral concentration or lithology. Figure 14.10 is a log normal histogram distribution of the rock density data obtained from current drilling that lies within the resource wireframe. The distribution of SG values range from 2.48 g/cm^3 to 4.84 g/cm^3. The mean value of 2.88 g/cm^3 was used ubiquitously as the specific gravity within these resource estimates.

Figure 14.10 1 m Specific Gravity Histogram



14.3.7 Interpolation Ellipse and Resource Estimate

ID^2 grade interpolation methodology was used to assign block model grades with blocks being fully constrained by limits of the resource solids. The search ellipse used for grade interpolation was developed on the basis of the deposit's geological model defined from interpolation of geological cross-sections. The Lundberg solid incorporated two interpolation domains defined north and south of gridline 7,930N. Two unique interpolation ellipses were determined for these two domains. Major and minor axis parameters were selected based on continuity and distribution of metal grade and reflect geological characteristics of the mineralized zones. The domain south of line 7930N used an ellipse aligned at 305°Az with a dip and plunge of -7.5° and 7.5° respectively. The domain north of line 7,930N used an ellipse aligned at 305°Az, with a dip and plunge of -25° and -25° respectively. Both ellipses have major and semi-major axes ranges of 75 m and minor axis range of 37.5 m in order to preserve the relatively sub-horizontal character of mineralization.

In the initial design of the model, passes of Nearest Neighbour (NN) and ID^2 grade interpolation using the search ellipse described above were completed in each bearing domain for zinc, lead, copper, gold, silver, and barite. These passes were followed by passes using a variety of major, semi-major and minor axis ranges. Optimum results occurred with a 75 m range interpolation ellipse provided the best fit of grade trends to the geological model and on this basis it was retained for the final estimation purposes. Use of smaller ellipses resulted in poor grade correlation in areas where reasonable geological certainty existed and larger ellipses extended higher grade values to some areas that did not have sufficient geological support.

The interpolation ellipse uses weighted average grades from up to 21 known data points (1 m composites) that occur closest to the block. A constraint of a maximum of seven composite samples per drill hole was applied when interpolating grade for each resource block. In effect, this forced the interpolation algorithm to search to adjacent drillholes for data points once a maximum of seven data points were reached from any one particular drill hole. This encouraged an influence of grade from surrounding drill holes in the interpolation method on any particular block, preserved the heterogeneity of grade along drill holes, and reduced the smoothing of variable grade of resource blocks along any drill hole. A maximum of 21 reporting composites from all surrounding drill holes was applied to limit excessive influence in areas of high density drilling.

Final constraints were applied to the block models in order to filter out blocks with a low degree of confidence. The volume and associated grade of blocks being interpolated from only one contributing composite, and blocks that had no contributing composites within 55 m were eliminated from the final estimate.

The final resource model was generated by running the interpolation ellipses within each domain. Estimation was performed using the ID^2 method as it provided a more satisfactory visual grade distribution and correlation than the nearest neighbour methodology. Block grade, block density and block volume parameters were combined to produce the final deposit tonnage and grade estimate. This estimate updates an earlier released Inferred Resource having an effective date of September 15, 2008 (PR#17-08 Sept 17, 2008) and incorporates more complete historic precious metal assay data compiled from historic drilling and assays, resulting in a nominal increase in the precious metal contents. Results of the resource estimation program are presented in Table 14.1 and Table 14.2 below and are compliant with the CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines as well as disclosure requirements of NI 43-101. Resources are classified in the Inferred and factors supporting such classification are discussed below in Section 14.3.8.

Upon request from the client, a calculation outlining the percentage of the resource tonnage that lies within 100 m from surface was undertaken. These numbers do not constitute or suggest the amenity of economics or mineability of the resource contained therein, but do offer insight into the spatial distribution of minerals within the current block model.

Table 14.1 Lundberg Inferred Resource Estimate - Zn % Threshold - Nov 3 2008

Zn% Threshold	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	$BaSO_4$%	Percentage of Tonnage Within 100 m of Surface
1.00	15,690,000	1.96	0.83	0.38	3.17	6.57	0.08	2.36	61.79%
1.50	9,300,000	2.46	1.03	0.43	3.92	8.26	0.10	2.84	66.40%
2.00	5,340,000	3.02	1.25	0.49	4.76	10.27	0.12	3.47	70.62%
2.50	3,170,000	3.56	1.46	0.53	5.55	12.28	0.14	4.65	72.83%
3.00	1,880,000	4.13	1.66	0.57	6.36	14.32	0.14	6.20	75.68%
3.50	1,090,000	4.79	1.93	0.62	7.34	16.46	0.15	8.64	81.35%

Table 14.2 Engine House Inferred Resource Estimate - Zn % Threshold - Nov 3 2008

Zn% Threshold	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	$BaSO_4$%	Percentage of Tonnage Within 100 m of Surface
1.00	890,000	2.37	0.95	0.96	4.28	11.29	0.15	4.40	58.73%
1.50	600,000	2.89	1.10	1.05	5.04	12.17	0.16	4.87	60.56%
2.00	370,000	3.62	1.27	0.97	5.86	12.71	0.19	5.51	60.40%
2.50	240,000	4.35	1.41	0.94	6.70	12.34	0.22	5.56	52.04%
3.00	190,000	4.77	1.50	0.93	7.20	12.32	0.23	5.63	56.35%
3.50	140,000	5.28	1.56	0.91	7.75	12.33	0.23	5.60	56.28%

14.3.8 Resource Category Definitions and Classifications

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

Inferred Mineral Resource

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

Only an Inferred Mineral Resource is reported in this document for extents of the block model estimation.

INDICATED MINERAL RESOURCE

An Indicated Mineral Resource is that part of a mineral resource for which quantity, grade or quality, densities, shape and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters to support mine planning and evaluation of the economic viability of the resource gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drill holes, that are spaced closely enough for geological and grade continuity to be reasonably assumed.

There is no Indicated Mineral Resource in this estimation.

MEASURED MINERAL RESOURCE

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

There is no Measured Mineral Resource in this estimation.

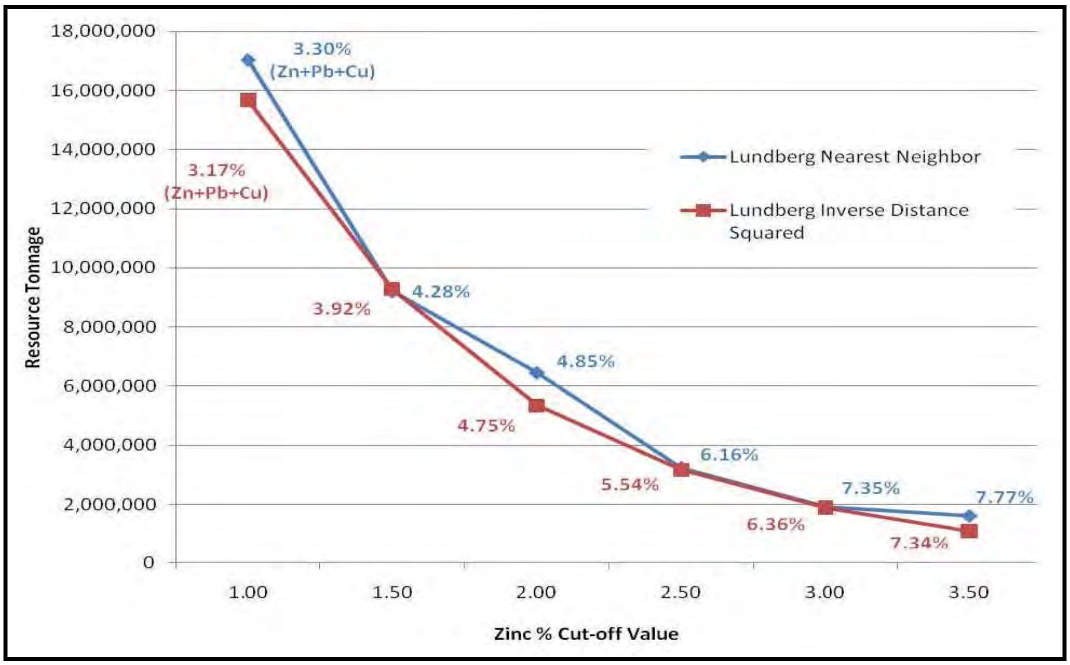
14.3.9 VALIDATION OF MODEL

Information presented within this resource calculation is considered to be accurate and reflects a level of confidence that falls within the definition of an Inferred Mineral Resource as defined by NI 43-101. Visual inspection of final grade distribution is considered reasonable with respect to that observed on geological cross-sections, and corresponds with mineral correlation defined by Mercator geologists.

To verify the mathematical derivation of the current mineral resource estimate, two separate modelling algorithms have been implemented on the data using the defined mineral resource solid and using identical block constraints. In the initial design of the model, an isotropic NN algorithm was used as a preliminary pass on *the resource block model to obtain an order of magnitude of resource volumes, and to gain insight in global grade trends. The model was composed of a single domain with an isotropic search ellipse and a range of 75 m. The final model utilized an ID^2 grade interpolation algorithms using the parameters described in Section 14.3.5. The final estimation was performed using the ID^2 method as it provided a more satisfactory visual grade distribution and correlation than the NN methodology.*

A comparison of the two methods can be seen in Figure 14.11. The smoothing effect of ID^2 modelling can be clearly seen between the two curves, as NN tends to segment various blocks of grade throughout the model and lacks the ability to blend grade trends between two known data points. A relict of this feature is noted at the 2 Zn% cut-off where the variance in the resource combined base metal value (Zn%+Pb%+Cu%) decreases when the volume variance increases. In general, the comparison of the two methods supports the use of the ID^2 model, and confirms the validity of the model's grade and volume distribution.

Figure 14.11 Block Model Validation, Nearest Neighbour vs. Inverse Distance Squared Interpolations



Mercator, in completion of the Lundberg and Engine House resource estimates, also tabulated mineral resource on the Lundberg and Engine House deposits based on a 1% combined base metal grade cut-off (Zn% +Pb% + Cu%). This tabulation is considered to be NI 43-101 compliant as it is based on the methods and data verification parameters outlined above. This tabulation is intended to compare the overall volume and grade of Asarco's historic resource calculation with the modeling parameters used in the Lundberg resource estimate described above. It should be noted that the overall footprint delineated by the Mercator *resource estimate is larger than that of ASARCO's determinations due to the additional drilling completed by BUV, and that the methods used to derive the totals are different than those used in the historic methods. The details of the Mercator 1% combined base metal resource are described in Table 14.3 and Table 14.4.*

Table 14.3 Lundberg Inferred Resource Estimate – 1% Zn + Pb + Cu Combined Threshold

Threshold (Zn%+Pb%+Cu%)	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	$BaSO_4$%
1.00	20,700,000	1.68	0.72	0.38	2.78	5.92	0.07	2.11

Table 14.4 Engine House Inferred Resource Estimate – 1% Zn + Pb + Cu Combined Threshold

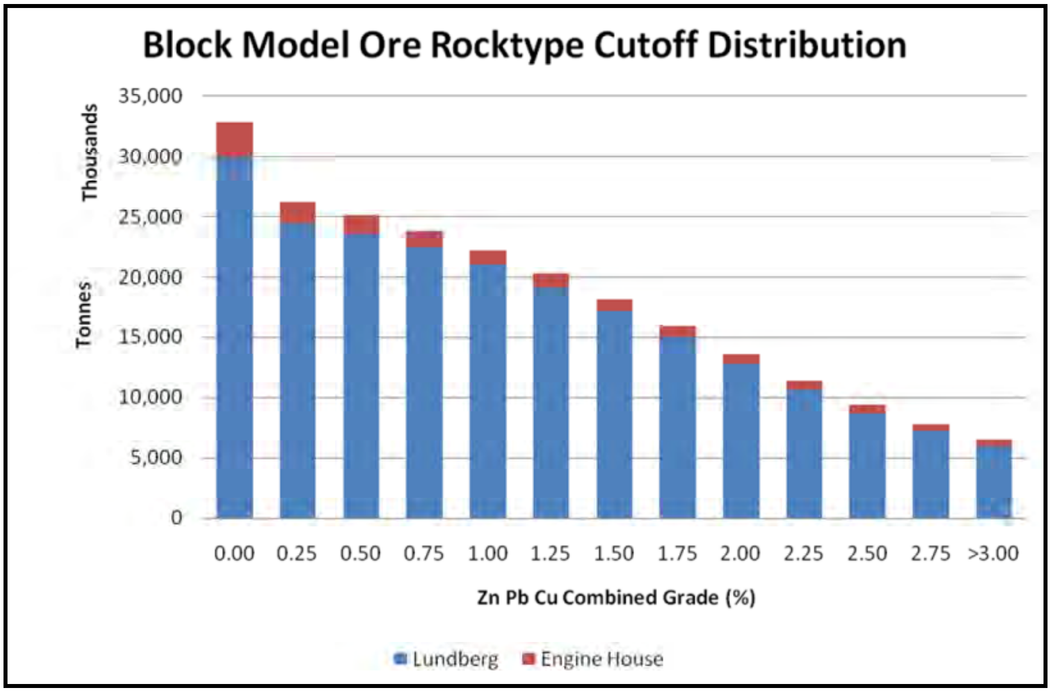
Threshold (Zn%+Pb%+Cu%)	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	$BaSO_4$%
1.00	1,120,000	2.04	0.85	0.82	3.71	9.79	0.12	3.74

For purposes of designing the open pit, Mercator Geological Services Ltd had to modify their original NI 43-101 technical compliant resource model, (see section 16.2) such that resource blocks were adjusted to be a consistent size throughout the resource model. The modified block model identified an Inferred Resource at a combined Zn-Pb-Cu cut-off of 1% of 22.21 million tonnes with average grades of 1.62% Zn, 0.69% Pb, 0.38% Cu, and 5.81g/t Ag (Table 14.5). See Figure 14.12 for the resource distribution for the re-blocked block model.

Table 14.5 Lundberg and Engine House Re-Blocked Inferred Resource Estimate – 1% Zn + Pb + Cu Combined Threshold

Threshold (Zn%+Pb%+Cu%)	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)
1.00	22,210,000	1.62	0.69	0.38	2.69	5.81

Figure 14.12 Block Model Ore Rocktype Cut-off Distribution



15.0 MINERAL RESERVE ESTIMATES

This section is not applicable to this report.

16.0 MINING METHODS

16.1 OVERVIEW

Tetra Tech completed an open pit design for BMC Lundberg and Engine House deposits, compliant with the requirements of NI 43-101 PEA. The open pit is located just west of the town of Buchans in central Newfoundland.

The open pit was designed using a two-stage approach. The first stage identified an optimum pit shell using the Lerchs-Grossman (LG) pit optimization method. In the second stage, phase mining and production schedules were developed, equipment selections were performed, and the capital and operating costs were estimated.

For this project, Tetra Tech determined that the mining operation will use a conventional open pit mining method (truck and shovel). The mine will provide mill feed of resource at a rate of 5,000 t/d, beginning the first year of the mine life.

The overall mining sequence was developed in three phases: a starter pit (Phase I) and two pushback phases (Phase II and Phase III). The mine development for the resource and the waste will progress using 10 m high benches.

The ultimate pit design for the selected base case pit contains 17.28 Mt of resource. The average grades over the life of mine will be 1.63% Zn, 0.69% Pb, 0.40% Cu, 5.96 g/t Ag, 0.07 g/t Au and 1.24% barium (Ba) for a combined base metal grade of 2.72% (Zn-Pb-Cu). The overall stripping ratio is 3.06 t/t (waste/resource). A total of 52.93 Mt of waste material will be moved over the mine life of 10 years.

It is proposed that the operation will be carried out with an equipment fleet comprising a single, 251 mm diameter rotary blast hole drill rig for resource and waste, an 11 m^3 (bucket capacity) hydraulic face shovel with a fleet of 91 t haul trucks. These will be supplemented with support equipment of loader, grader, dozers, and a backhoe excavator, etc.

16.2 GEOLOGY

The block models for Lundberg and Engine House were originally received by Tetra Tech from Mercator as text files exported from the Gemcom™ Surpac software program. The two block models were received with sub-blocks, which were not compatible with Gemcom™ GEMS. Mercator then re-blocked their models to 5 m x 5 m x 5 m blocks. Originally, the Lundberg block model used 5 m x 5 m x 5 m blocks with 2.5 m x 2.5 m x 2.5 m sub-blocks and the Engine House block model used 2.5 m 2.5 m x 2.5 m blocks with 1.25 m x 1.25 m x 1.25 m sub-blocks.

Following the re-blocking by Mercator, Tetra Tech converted both block models into a single Gemcom™ GEMS block model to encompass both the Lundberg and Engine House deposits and block models. The new, single Gemcom™ GEMS block model was utilized for open pit optimization and underground analysis.

16.3 Underground Analysis

Underground analysis was conducted to assess the viability of an underground mining operation to follow the completion of the open pit. The remaining resource extraneous to the open pit was depleted by old workings including development access ways, raises, and mined out stopes. Crown and boundary pillars were defined below and adjacent to the designed open pit and adjacent to the historic mine workings as a constraint for the potentially mineable underground resource design.

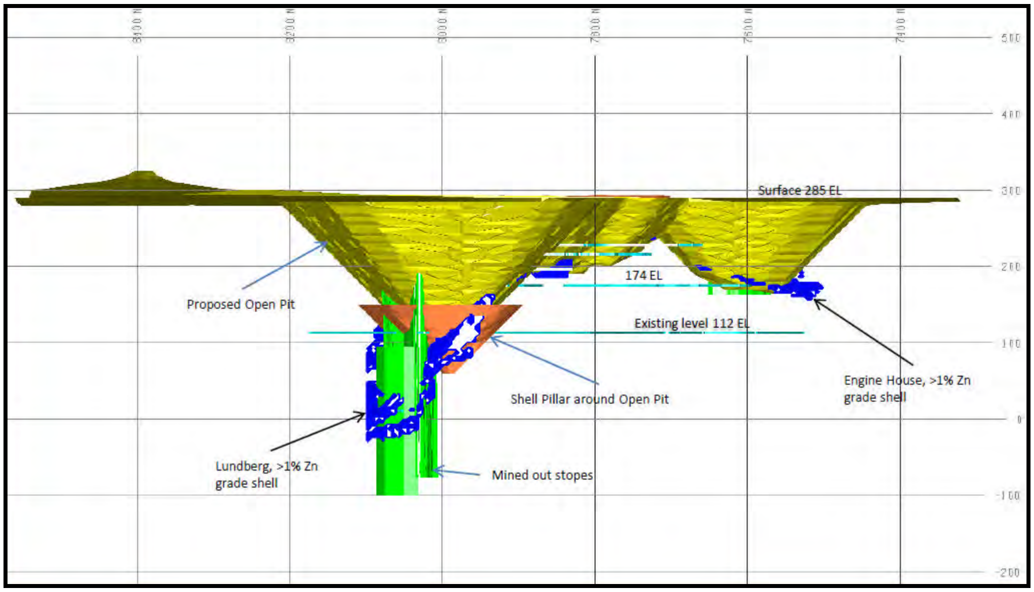
16.3.1 Criteria and Assumptions

The following criteria and assumptions have been used to evaluate the practicality of an underground design.

- Underground mining is to commence after the completion of open pit mining.
- Leave a 30 m stand-off pillar below the planned open pit floor
- Leave a 10 m shell pillar around the slopes of the open pit
- Leave a 10 m shell pillar around existing mined out stopes underground.

The resulting mineralization available for underground mine design is shown in dark blue in Figure 16.1. The figure also shows the outlines for the proposed open pit (gold), the shell pillars isolating the pit from underground workings (orange) and old mined out stopes (green). Cyan objects in the figure represent historic mine levels which have potential for use in the new mine design.

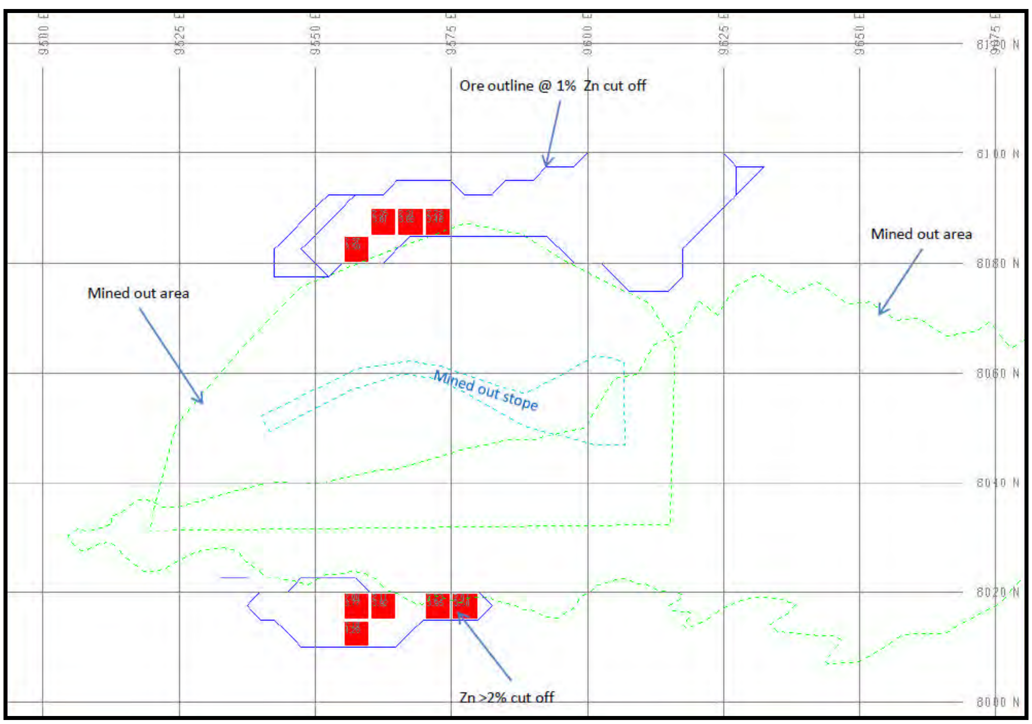
Figure 16.1 Section Looking North



Note: Showing pillars around open pit shell and available resource for underground mining.

Figure 16.2 shows the potential mineralization (resource at combined 1% cut-off) from a plan view perspective. The reader will observe that only sparse mineralization is available against the periphery of the pit boundary and that the mineralization is separated into two distinct and remote areas.

Figure 16.2 Plan at 35 Elevation (50 m below Pit Bottom)



16.3.2 CUT-OFF GRADE

The dark blue mineralization shown in Figure 16.1 was declared a mineable resource at a cut-off grade (5.58% Zn) determined through accumulation of the following costs.

- mining method
- processing costs
- general, administrative, and sales
- haulage.

The underground mining method chosen was "Cut and Fill" and was selected as the simplest approach to maximize recovery while minimizing dilution in as safe an environment as could be expected when surrounded by old workings. Other methods considered but discounted include Longhole and Sublevel Retreat).

A cut-off grade of 5.58% zinc equivalent recovered (ZnEQR) was used for estimating the underground mineable resource based on the allocation of tonnes of resource shown in the Table 16.1 below.

Table 16.1 Resource Estimate at 2% Zn Cut-off

	Resource Tonnes	Zn (%)
Open Pit Mine (Pit Shell)	5,112,691	3.03
Crown Pillar	260,014	2.84
Available for Underground	31,759	2.22
Total	**5,404,464**	**3.02**

Readers will observe that Table 16.1 shows only 32 thousand tonnes of mineralization at a grade of 2.22 % zinc equivalent (ZnEQ) available for consideration in an underground mining campaign. This is declared insufficient as an economic option to the Buchans operation.

For completeness the authors have considered a sensitivity analysis on the cut-off grade based on ±10% variance of the operating cost. The results are shown in Table 16.2. Even with the lower mining cut-off of 5.04% Tetra Tech declares the available resulting tonnes of mineralization uneconomic for an underground exercise at this time.

Table 16.2 Sensitivity Analysis

Scenario	Cost Summary (Cdn$) Mining	Processing	G&A	Smelt/Refining	Total	Required ZnEQ Cut-Off Grade (%)
-10% OPEX	43.90	12.26	3.33	47.24	106.74	6.11
Base Case	46.91	14.90	3.70	52.49	118.00	5.58
+10% OPEX	53.66	13.80	4.07	57.74	129.27	5.04

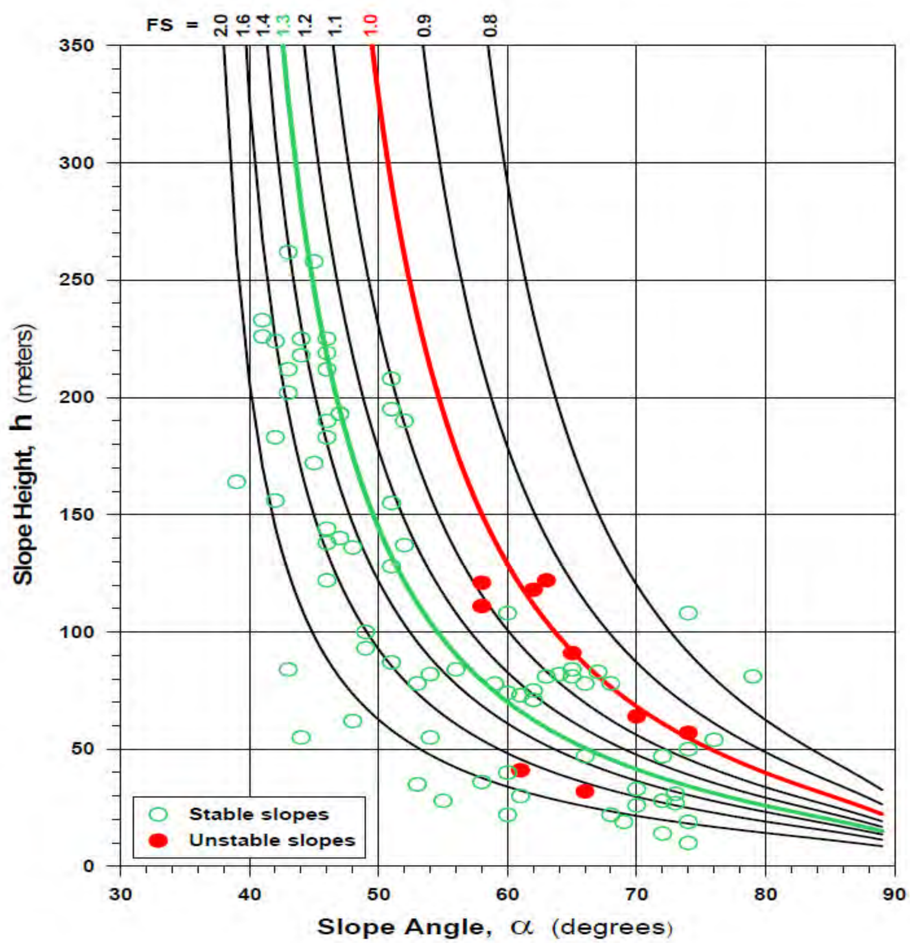
16.3.3 CONCLUSION

No appreciable tonnage was indicated above the required underground cut-off grade of 5.58% ZnEQR. Even with consideration of a reduced operating cost by 10%, the required cut-off grade of 5.04% ZnEQR cannot be achieved with the remaining resource available for underground mining. Therefore, based on this analysis using the economic conditions stated in the open pit design, the underground operation is not viable at this time.

16.4 OVERALL OPEN PIT SLOPE ANGLE

Since the required geotechnical data is not available for determining the pit slope angle, Tetra Tech utilized an overall pit slope angle of 45°, based on conservative estimates from previous experience. Table 16.3 indicates a 45° overall pit slope is a conservative stable slope to start evaluations. As geotechnical data becomes available, pit slopes could potentially steepen and improve the mine plan and economic evaluation of study.

Figure 16.3 Cases of Stable and Failed Rock Slopes Conditions



Note: Reference Paper - "A Risk Evaluation Approach for Pit Slope Design", figure provided by Hoek & Bray, 1974.

16.5 Pit Optimization

16.5.1 Pit Optimization Procedures

Pit optimization was performed to determine the optimum pit limits and evaluate the resource contained within the pit at the highest present value (PV). Tetra Tech used 3D LG algorithm of Gemcom Whittle™ 4.3 commercial software to perform the pit optimization for this project.

A 3D geological block model and other required economical and operational variables were used as input parameters of the LG algorithm. These variables include overall pit slope angle, mining cost, milling cost, metal prices, concentrate treatment costs and other parameters.

The LG algorithm progressively identifies economic blocks when resource mining and waste stripping, taking into account for a specified pit slope angle. The resulting pit outline identifies all the blocks that may be economically mined within the open pit shell.

Tetra Tech conducted the following design steps during the pit optimization process:

- combined both the Lundberg and Engine House block models
- conducted pit optimization
- evaluated preliminary production schedules based on best, worse and specified cases
- applied operation parameters to the production schedule
- selected an optimized (base case) pit shell that represents the highest PV of the specified case.

During the initial optimization, five separate pit optimizations were evaluated for various sensitivity scenarios. One for each of:

- complete full block model (no constraints)
- 1.0% Zn-Pb-Cu combined cut-off
- 1.0% Zn cut-off
- 1.5% Zn cut-off
- 2.0% Zn cut-off.

For each of the five optimized pit shells, schedules were considered and evaluated for the following resource tonnes per day productivity rates: 1800, 2000, 3000, 4000, 5000, 5500, and 6000.

The highest PV pit (pit shell 6) for the 5,000 t/d case was selected. The same optimized pit shell (pit shell 6) was then used for each of the other four separate optimization scenarios. During Optimization #1, the pit shells were kept constant in each case to evaluate the effects of the various productivity rates.

In reviewing results from the initial pit optimization set, decision was made to proceed with the 1.0% Zn-Pb-Cu combined, 5,000 t/d productivity rate option. This option contained the highest PV, while maintaining a mining life greater than 10 years.

Three additional Whittle™ optimizations sets were completed for this option.

OPTIMIZATION #1

- Initial Whittle™ input parameters.

OPTIMIZATION #2

- Updated concentrate grades for Pb and Cu.
- Updated transport land and ocean freight costs.
- Updated treatment charges for Zn and Cu.
- Addition of Stevedoring, representation, and miscellaneous charges.
- Addition of Ag and Cu concentrate refining charges.

OPTIMIZATION #3

- Updated metal prices provided by Buchans.

OPTIMIZATION #4

- Updated metal prices using Tetra Tech, November 2010 long-range forecast.

16.5.2 PIT OPTIMIZATION PARAMETERS

For the initial optimization, the required parameters were selected by Tetra Tech and Buchans in evaluating the most economic open pit profile. Although these parameters are not necessarily final, a reasonable degree of accuracy is required, since the analysis is an iterative process. The economic and operating parameters used in the final optimization (#4) are given in Table 16.3.

The Zn-Pb-Cu cut-off grade resource distribution is represented in Table 16.4 outlining 22.2 Mt (1% Zn-Pb-Cu combined) of potential resource available for the optimization.

Figure 16.4 Lundberg/Engine House Cut-off Grade Resource Distribution for Re-Blocked Block Model.

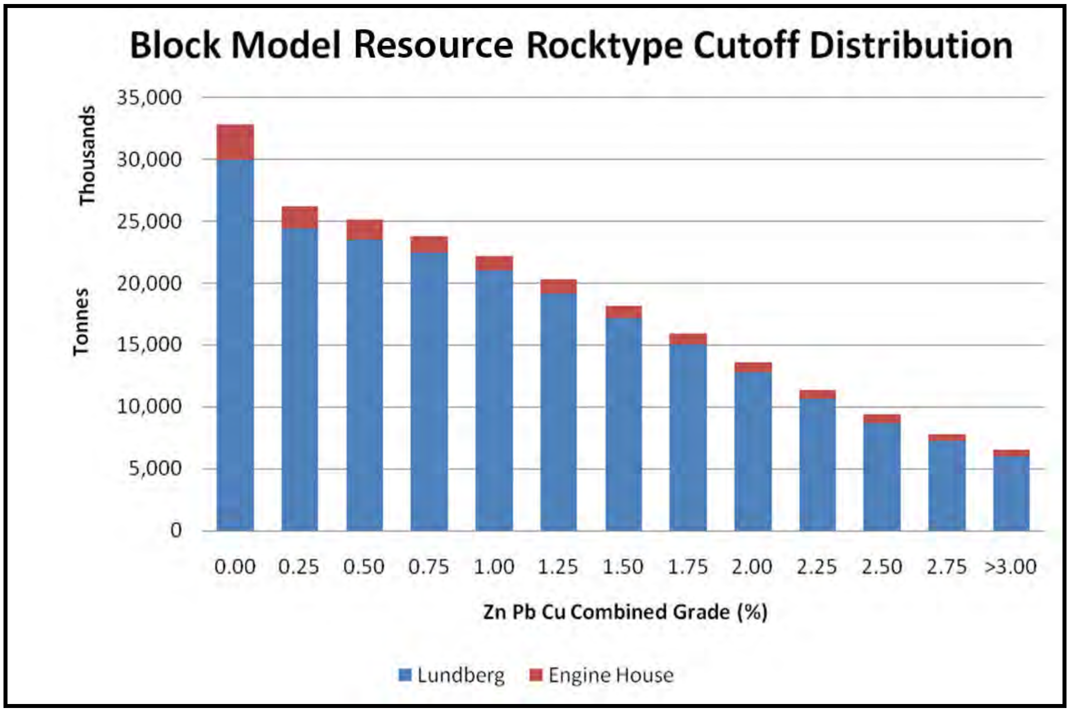


Table 16.3 Economic Parameters of Pit Optimization (Optimization #4)

Items		Units	Value	Comments
Exchange Rate		Cdn$:US$	1	
Discount Rate		%	6	
Metal Prices	Zinc	US$/lb	1.07	
	Lead	US$/lb	0.99	
	Copper	US$/lb	2.89	
	Silver	US$/lb	16.91	
	Gold	US$/lb	0.00	
Metal Recoveries	Zinc	%	72.3	85% mill x 85% smelter
	Lead	%	76.0	80% mill x 95% smelter
	Copper	%	77.2	80% mill x 96.5% smelter
	Silver	%	45.0	50% mill x 90% smelter
	Gold	%	0.0	
Concentrate Grade	Zinc	%	55	
	Lead	%	40	Same amount of concentrate is produced. Only 100% of 37% concentrate grade is payable.
	Copper	%	20	Same amount of concentrate is produced. Only 100% of 19% concentrate grade is payable.
	Silver	%		
	Gold	%		
Concentrate Costs	Transport Land (Zinc)	US$/wet tonne	$36.27	Based on quote $0.13/mile, 279 miles to Port George round trip, empty on return trip.
	Transport Land (Lead)	US$/wet tonne	$36.27	Based on quote $0.13/mile, 279 miles to Port George round trip, empty on return trip.
	Transport Land (Copper)	US$/wet tonne	$36.27	Based on quote $0.13/mile, 279 miles to Port George round trip, empty on return trip.
	Ocean Freight Charge	US$/wet tonne	$40.00	Europe or North America (Zinc)
	Ocean Freight Charge	US$/wet tonne	$40.00	Europe or North America (Lead)

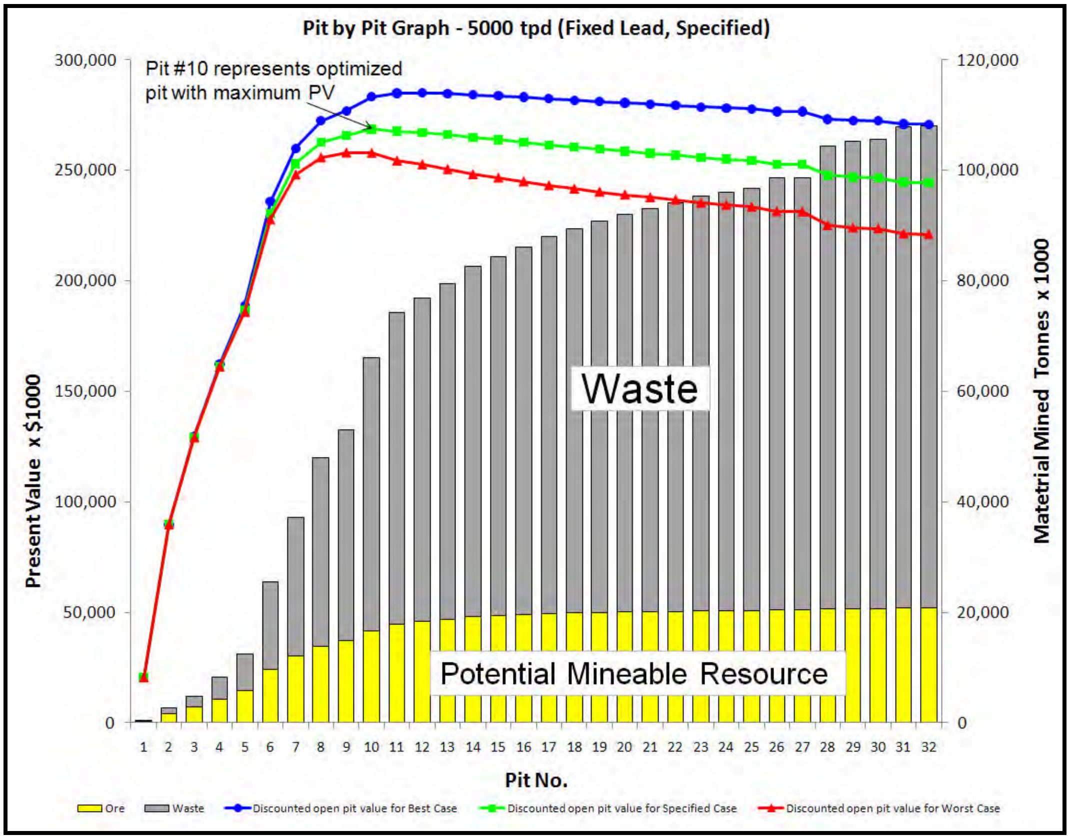
Items		Units	Value	Comments
	Ocean Freight Charge	US$/wet tonne	$115.00	Mainland China (Copper)
	Stevedoring Charge (Zinc)	US$/wet tonne	$15.00	
	Stevedoring Charge (Lead)	US$/wet tonne	$15.00	
	Stevedoring Charge (Copper)	US$/wet tonne	$15.00	
	Representation - Zinc	US$/wet tonne	$3.00	
	Representation - Lead	US$/wet tonne	$3.00	
	Representation - Copper	US$/wet tonne	$3.00	
	Miscellaneous - Zinc	US$/wet tonne	$10.00	
	Miscellaneous - Lead	US$/wet tonne	$10.00	
	Miscellaneous - Copper	US$/wet tonne	$5.00	
	Landing Freight	US$/lb FOB Mine		
Refining Charge	Lead Concentrate	US$/oz	$0.75	Ag refining charge
	Copper Concentrate	US$/tonne dry	$25.13	
Moisture Content	Zinc	%	8%	
	Lead	%	8%	
	Copper	%	8%	
Royalty Costs	Concentrate Agent Fee (Zinc)	US$/tonne dry	$25	Remove for 1,800 t/d case
	Concentrate Agent Fee (Lead)	US$/tonne dry	$25	Remove for 1,800 t/d case
	Concentrate Agent Fee (Copper)	US$/tonne dry	$25	Remove for 1,800 t/d case
	Third Party NSR	Zinc	3%	Percentage of total revenue less treatment & transport costs
	Third Party NSR	Lead	3%	Percentage of total revenue less treatment & transport costs
	Third Party NSR	Copper	3%	Percentage of total revenue less treatment & transport costs
Operating Costs	Mining Cost	US$/t mined	$1.80	
	Total resource production re-handled	%	6%	

Items		Units	Value	Comments
	Stockpile Re-handling for Resource	US$/resource t re-handled	$1.00	
	Processing	US$/t processed	$10.80-$15.50	$15.50 used for 1,800 t/d $10.80 used for 5,000 t/d
	G & A + Time Cost	US$/tonne resource	$2.00	
Mining Recovery		%	95%	
Mining Dilution		%	5%	

16.5.3 PIT OPTIMIZATION RESULTS

A series of nested pit shells were generated by varying the revenue factor. Table 16.4 demonstrates the relationship between the resource contained within the pit shell and the PV for each of the nested pit shells. Figure 16.5 shows the PV for each pit shell, in which the specified case represents the most practical operation scenario.

Figure 16.5 Nested Pits and Present Values



Note: Replicated from chart generated in the Whittle™ 4.3 commercial software.

The following features are identified:

- Pit shell #10 generates the highest PV at Cdn$268.7 million for the specified case, based on a 5,000 t/d operation over a mine life of approximately 10 years. The selected base case pit shell contains 16.6 Mt of resource with average grades of:
  - 1.68% Zn
  - 0.71% Pb
  - 0.42% Cu
  - 6.07 g/t Ag

- 0.07 g/t Au
- 1.26% Ba

Having a combined base metal grade of 2.81% (Zn-Pb-Cu).

- For pit shells larger than pit #10, although the resource is progressively increased, PV is gradually decreased. The decrease in cash flow results from resource grades gradually decreasing, stripping ratio gradually increasing, and the time value of money (discount rate).
- Pit #10 is an optimized pit shell or base case pit shell (Figure 16.5 and Figure 16.6).

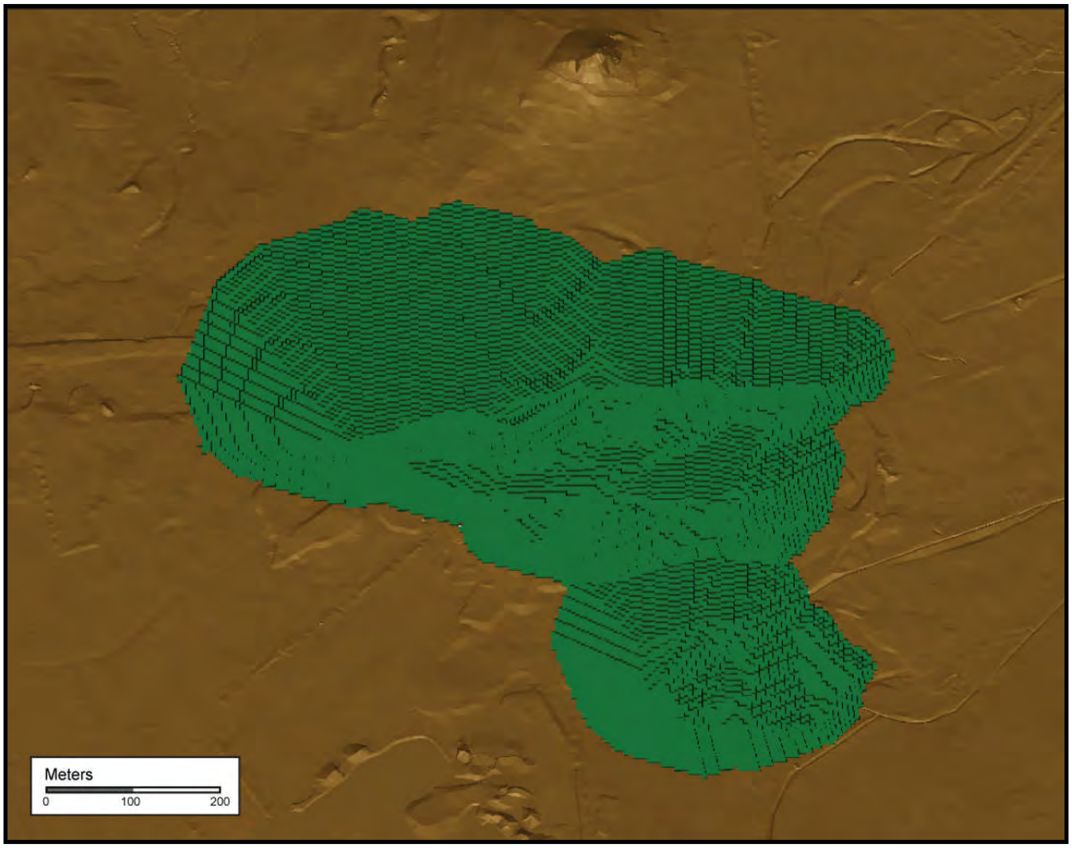
Table 16.4 Optimization Results of Nested Pits (Optimization #4)

Nested Pit No.	Open Pit Cash Flow: Best $ Discounted (x million)	Specified $ Discounted (x million)	Worst $ Discounted (x million)	Resource tonnes (x million)	Waste tonnes (x million)	Strip Ratio	Ag (g/t)	Au (g/t)	Ba%	Cu%	Pb%	Zn%	Zn-Pb-Cu%
1	20.6	20.6	20.6	0.3	0.1	0.80	20.44	0.15	12.92	0.60	2.12	5.39	8.11
2	89.8	89.8	89.8	1.7	0.9	0.84	11.83	0.11	4.38	0.72	1.39	3.47	5.58
3	129.5	129.1	129.1	2.9	1.9	0.80	9.47	0.10	2.85	0.64	1.16	2.84	4.64
4	162.4	161.2	161.2	4.3	4.0	1.04	9.17	0.10	2.33	0.56	1.03	2.51	4.11
5	189.1	186.9	185.9	5.8	6.6	1.23	8.53	0.09	1.91	0.55	0.92	2.22	3.69
6	235.9	230.6	227.8	9.6	15.8	1.76	6.94	0.07	1.62	0.48	0.82	1.93	3.22
7	260.0	253.0	248.1	12.1	25.0	2.13	6.57	0.08	1.42	0.46	0.76	1.80	3.02
8	272.4	262.7	255.8	13.9	34.1	2.49	6.32	0.08	1.35	0.45	0.74	1.74	2.92
9	277.0	265.9	258.0	14.9	38.1	2.57	6.17	0.08	1.29	0.43	0.72	1.70	2.85
10	**283.3**	**268.7**	**257.8**	**16.6**	**49.4**	**2.98**	**6.07**	**0.07**	**1.26**	**0.42**	**0.71**	**1.68**	**2.81**
11	284.9	267.7	254.4	17.9	56.4	3.16	5.90	0.07	1.24	0.40	0.70	1.65	2.75
12	285.1	267.2	252.7	18.3	58.5	3.20	5.84	0.07	1.23	0.40	0.69	1.64	2.73
13	284.9	266.2	250.5	18.7	60.8	3.25	5.79	0.07	1.23	0.39	0.69	1.62	2.71
14	284.3	264.8	248.2	19.2	63.6	3.32	5.80	0.07	1.23	0.39	0.68	1.61	2.68
15	283.8	263.9	246.7	19.4	65.0	3.35	5.78	0.07	1.22	0.39	0.68	1.61	2.67
16	283.2	262.8	245.0	19.6	66.6	3.40	5.77	0.07	1.22	0.39	0.68	1.60	2.66
17	282.5	261.6	243.2	19.7	68.3	3.46	5.76	0.07	1.21	0.38	0.68	1.60	2.65
18	281.8	260.5	241.7	19.9	69.7	3.51	5.76	0.07	1.21	0.38	0.67	1.59	2.65
19	281.2	259.6	240.3	19.9	70.8	3.55	5.75	0.07	1.21	0.38	0.67	1.59	2.65
20	280.6	258.6	239.0	20.0	72.0	3.59	5.74	0.07	1.21	0.38	0.67	1.59	2.64
21	280.0	257.7	237.8	20.1	73.0	3.63	5.74	0.07	1.21	0.38	0.67	1.59	2.64
22	279.5	256.9	236.7	20.2	73.9	3.66	5.73	0.07	1.21	0.38	0.67	1.59	2.64

table continues...

Nested Pit No.	Open Pit Cash Flow			Resource tonnes (x million)	Waste tonnes (x million)	Strip Ratio	Ag (g/t)	Au (g/t)	Ba%	Cu%	Pb%	Zn%	Zn-Pb-Cu%
	Best $ Discounted (x million)	Specified $ Discounted (x million)	Worst $ Discounted (x million)										
23	278.8	255.9	235.3	20.2	75.0	3.71	5.73	0.07	1.21	0.38	0.67	1.58	2.63
24	278.3	255.1	234.3	20.3	75.8	3.74	5.73	0.07	1.20	0.38	0.67	1.58	2.63
25	277.9	254.5	233.5	20.3	76.4	3.76	5.72	0.07	1.20	0.38	0.67	1.58	2.63
26	276.8	252.8	231.4	20.4	78.2	3.83	5.72	0.07	1.20	0.38	0.67	1.58	2.63
27	276.7	252.8	231.4	20.4	78.2	3.83	5.72	0.07	1.20	0.38	0.67	1.58	2.63
28	273.2	247.7	225.1	20.6	83.9	4.07	5.74	0.07	1.20	0.38	0.67	1.58	2.63
29	272.7	246.9	224.1	20.6	84.7	4.10	5.74	0.07	1.20	0.38	0.67	1.58	2.62
30	272.4	246.6	223.7	20.7	85.0	4.11	5.74	0.07	1.20	0.38	0.67	1.58	2.62
31	271.0	244.7	221.4	20.7	87.1	4.20	5.74	0.07	1.20	0.38	0.67	1.58	2.62
32	270.9	244.4	221.1	20.7	87.3	4.21	5.73	0.07	1.20	0.38	0.67	1.58	2.62

Figure 16.6 Base Case Pit Shell #10



Note:
Typical pit shell distance 845 m from west to east
Typical pit shell distance 760 m from south to north
Typical pit shell depth 200 m, from elevation 290 to 90 m
Overall slope angle is 45°

16.5.4 ULTIMATE PIT DESIGN PARAMETERS

Based on the optimization results and current geological model, Tetra Tech recommends that the detailed pit design should be based on pit shell #10 to maximize PV (see Figure 16.5 and Figure 16.6).

PIT DESIGN CRITERIA

Pit design criteria includes:

- 10 m benches
- 85° face angle
- double benching
- 15 m catch-bench

- overall pit slope of 50°, without inclusion of a ramp
- 26 m ramp width, double lane (based on 90 t haul trucks) (see Figure 16.7)
- 17.5 m ramp width, single lane (based on 90 t haul trucks).

Figure 16.7 Ramp Width Design – Concept

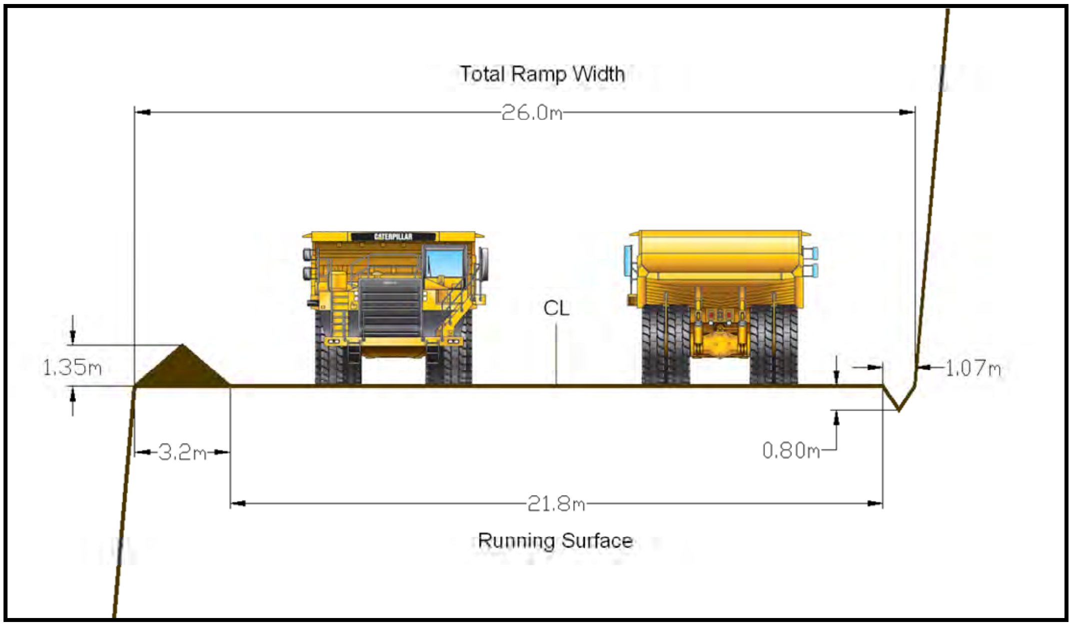


Table 16.5 Pit Design Ramp Width Calculation

Truck Parameters (CAT 777F)	
Operating Width	6.494 m
Double Lane (3.35) x Operating Width	3.35 m
Road Width	21.75 m
Berm	
Tire Overall Diameter	2.702 m
Height (1/2 of largest tire)	1.351 m
Slope (H:V)	0.85 m
Berm Width	3.18 m
Ditch	
Depth	0.8 m
Slope (H:V)	1.5 m
Ditch Width	1.07 m
Total Road Width	26.00 m

Truck Parameters (CAT 777F)	
Operating Width	6.494 m
Single Lane (2.00) x Operating Width	2.00 m
Road Width	12.99 m
Berm	
Tire Overall Diameter	2.702 m
Height (1/2 of largest tire)	1.351 m
Slope (H:V)	0.85 m
Berm Width	3.18 m
Ditch	
Depth	0.8 m
Slope (H:V)	1.5 m
Ditch Width	1.07 m
Total Road Width	17.23 m

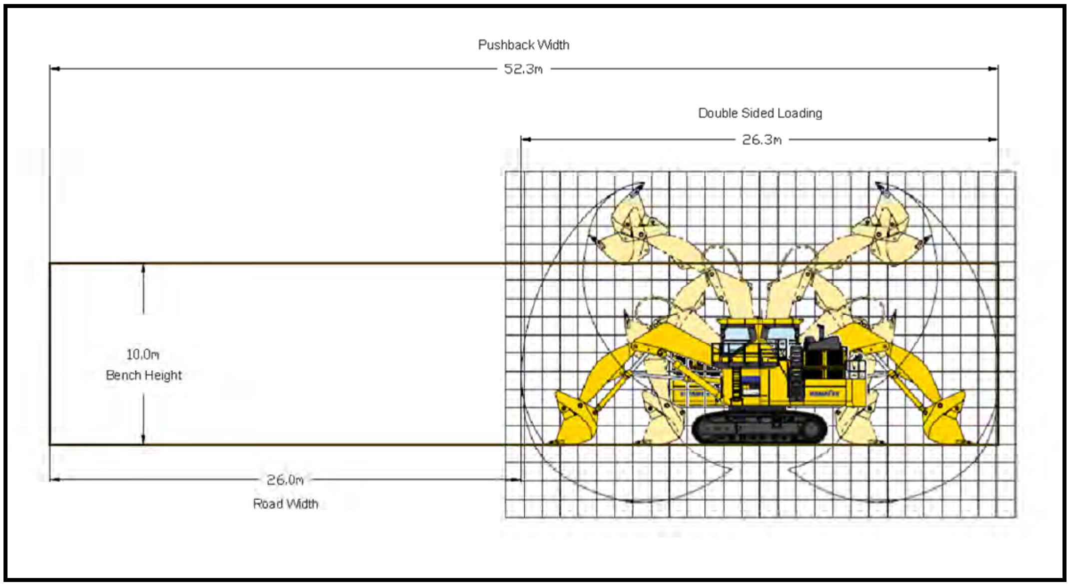
16.5.5 PUSHBACK WIDTH

An approximate pushback width was determined based on:

- A Komatsu PC2000-8 diesel hydraulic shovel, loading Caterpillar 777F haul trucks.
- A minimum double-side loading width of a diesel hydraulic shovel at 26.3 m for the Caterpillar 777F haul truck.
- A 26 m haul road width.
- The proposed minimum pushback width is the sum of the minimum double-side loading width at 26.3 m, and the haul road width at 26 m, for a total minimum width of 52.3 m.

Figure 16.8 outlines the minimum pushback width to be used in the design of the phase development.

Figure 16.8 Minimum Pushback Width



16.5.6 Resource Contained Within Pit Design

For purposes of designing the open pit, Mercator Geological Services Ltd. had to modify their original NI 43-101 technical compliant resource model, (see section 16.2 above). The modified block model identified an Inferred Resource at a combined Zn-Pb-Cu cutoff of 1% of 22.21 million tonnes with average grades of 1.62% Zn, 0.69% Pb, 0.38% Cu, and 5.81g/t Ag.

Table 16.6 shows the ultimate pit design contains 17.28 Mt of Inferred Resource and average grades.

Table 16.6 Ultimate Pit Design Results

Item	Tonnes (millions)	Zn (%)	Pb (%)	Cu (%)	Ba (%)	Ag (g/t)	Au (g/t)
Inferred Resource	17.28	1.63	0.69	0.40	1.24	5.96	0.07
Waste Rock	52.93						
Stripping Ratio	3.06						

Having a combined base metal grade of 2.72% (Zn-Pb-Cu). A mining resource recovery of 95% with an overall waste rock dilution of 5% was assumed.

16.6 Mine Development and Production Schedule

The mine development used a number of push-backs, or phases, designed to meet the following objectives:

- enable the mining of high grade resource as early as possible
- effectively reduce stripping ratio in the initial mining stage
- balance the stripping ratio over the period of the mine life
- maintain minimum mining width between two working phases.

16.6.1 Road Width

In-pit ramps are designed with an overall ramp width of 26 m with a maximum gradient of 10%. A 3.2 m wide and 1.35 m high safety berm and an internal 1.1 m wide water ditch will be provided for two lane traffic to accommodate 90 tonne Caterpillar 777F haul trucks, as shown in Figure 16.7 and Table 16.5.

16.6.2 Mine Development

In the case of the Lundberg/Engine House mine, three mineable phases have been identified to develop ultimate pit. Each phase or pushback is designed at least a minimum mining width of about 55 m to accommodate mining equipment that will operate on a working bench (Figure 16.8).

Phase I

Phase I is the first pit that would be designed from the initial economic pit shells generated by the Whittle™4.3 optimization run. Due to this study being at PEA level, a final pit design for the starter pit was not completed. Whittle™ pit shell #4 was used as the starter pit and scheduling purposes. The initial economic pit shells prioritize the high grade resource mining at the top-center portion of the resource body, and at the lowest amount of waste stripping. This will maximize cash flow and speed the capital recovery during the initial years. Phase I will mine 4.32 Mt of resource.

Phase II

Phase II geometry is expanded to West of the starter pit to mine the next economical blocks of the resource. Due to this study being at PEA level, a final pit design for the phase II pit was not completed. Whittle™ pit shell #6 was used as the phase II pit and scheduling purposes. Phase II contains 5.35 Mt of resource.

Phase III

Phase III expands to West and mines the remaining resource inside base case pit to achieve the final highwall. Phase III contains the remaining 7.61 Mt of resource.

Ultimate Pit

Overall, the ultimate pit contains 17.28 Mt of resource with an average stripping ratio 3.06 t/t (waste/resource). Details are outlined in Table 16.6.

Figure 16.9 and Figure 16.10 present general development sequence and relationship between mining phases and does not represent scaled drawings. To prioritize the high grade resource, all of Phase I and part of Phase II will be mined out in the first three years of the mine life. Phase III will not start to be mined until partially through the third year of mine life.

Figure 16.9 Sequence of Phase Development

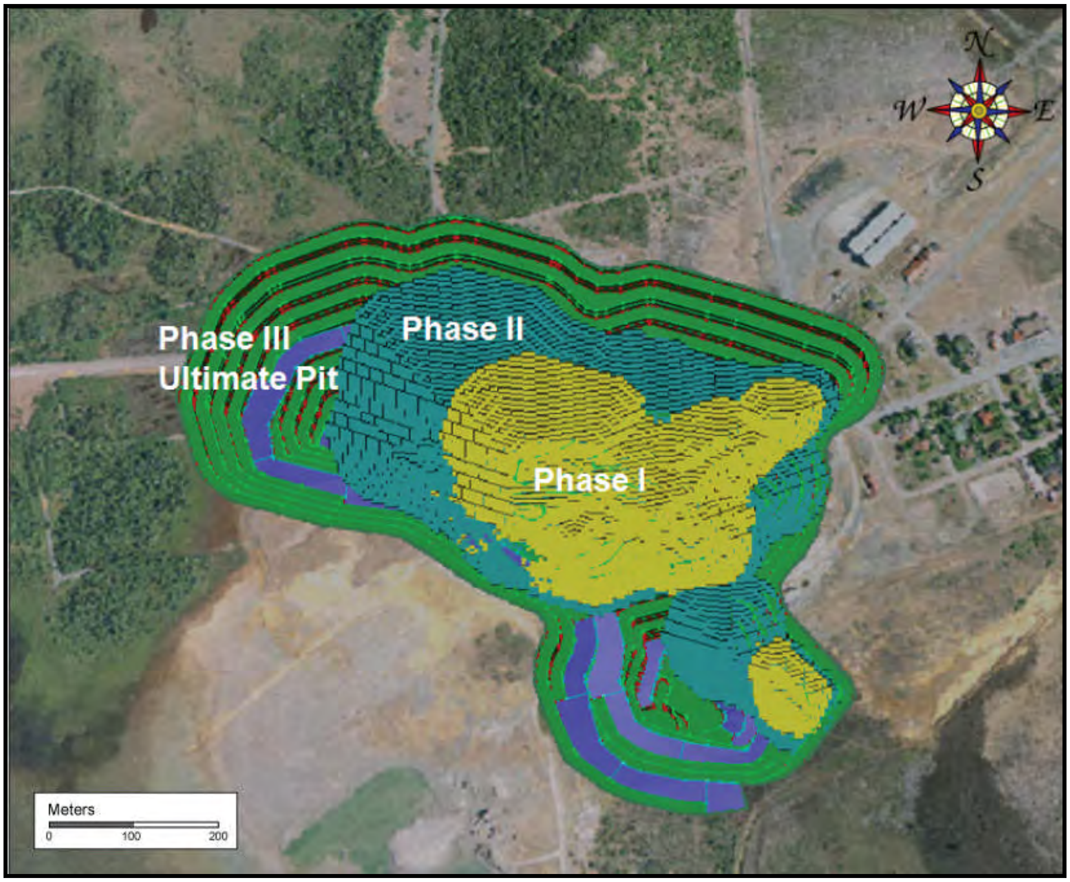


Table 16.7 Total Material Mined by Phase by Year

	1	2	3	4	5	6	7	8	9	10	Total
Phase 1 (Mt)	4.9	1.7	1.7								**8.3**
Phase 2 (Mt)	2.3	5.2	0.9	5.3	2.1	1.1	0.2				**17.1**
Phase 3 (Mt)			5.8	4.6	7.5	7.8	8.1	5.7	3.2	2.1	**44.8**

Figure 16.10 Sequence of Phase Production

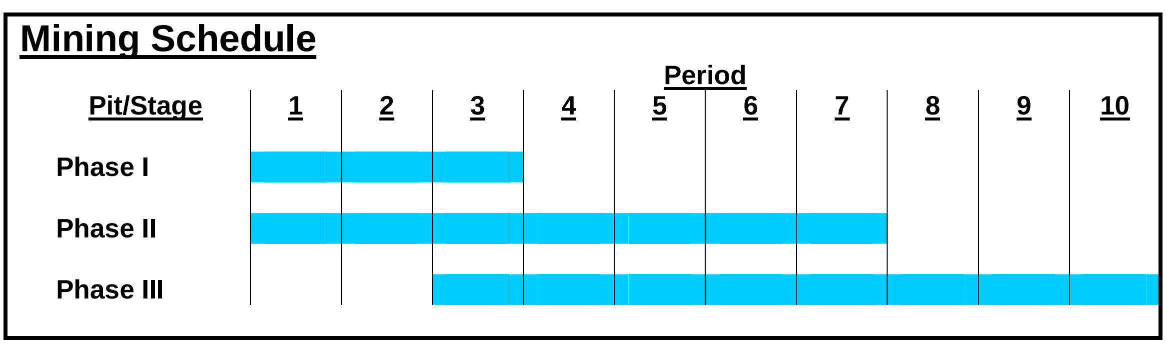


Figure 16.11 Waste and Resource Mined by Phase by Year

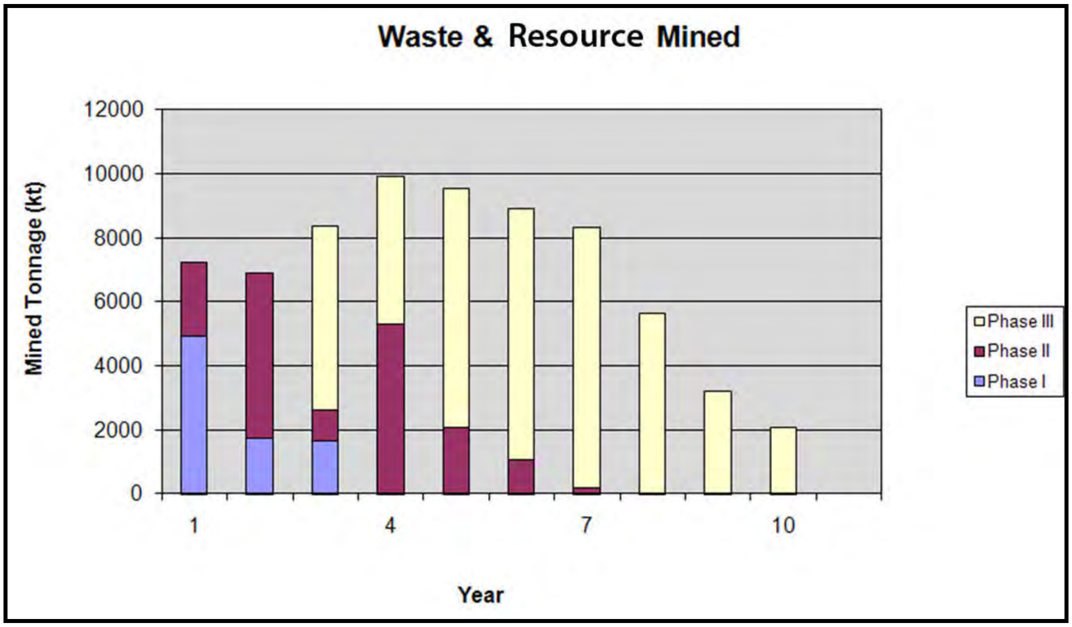
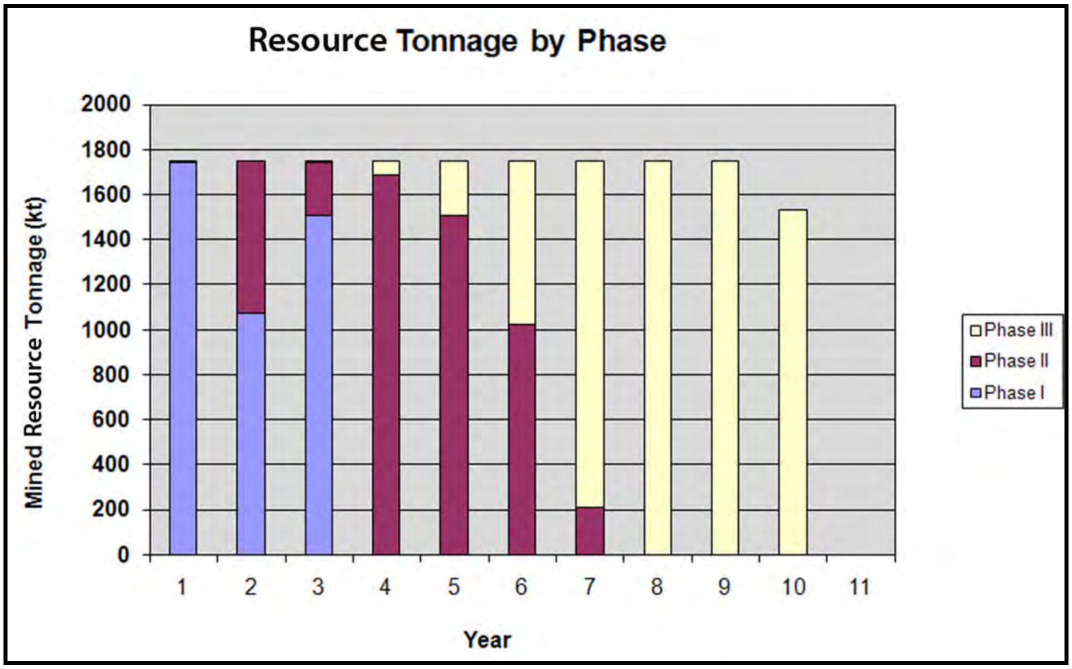


Table 16.8 and Figure 16.12 show the resource production schedule by phase.

Table 16.8 Mined Resource Tonnage by Phase

	1	2	3	4	5	6	7	8	9	10	Total
Phase 1 (kt)	1,740	1,072	1,507								**4,319**
Phase 2 (kt)	10	678	235	1,688	1,509	1,021	208				**5,349**
Phase 3 (kt)			8	62	241	729	1,542	1,750	1,750	1,529	**7,611**

Figure 16.12 Mined Resource Tonnage by Phase by Year



16.6.3 Production Schedule

A mill throughput of 5,000 t/d allows for an annual production of 1.75 Mt based on 350 days per year. Tetra Tech developed the production schedule having mine life of approximately 10 years in this study.

Buchans Lundberg and Engine House deposits will be mined simultaneously and will be capable to achieve full production during the first year of operation. The mill will be able to start up shortly after open pit operations commences. The resource will be placed into a stockpile on site before mill starts operation.

Figure 16.13 and Table 16.9 outlines the complete mining schedule period by period showing waste and resource tonnage mined by resource deposit along with associated resource grades. Resource grades are expressed in graphical format in Figure 16.14, Figure 16.15, and Figure 16.16.

Figure 16.13 Overall Ultimate Pit Production Schedule

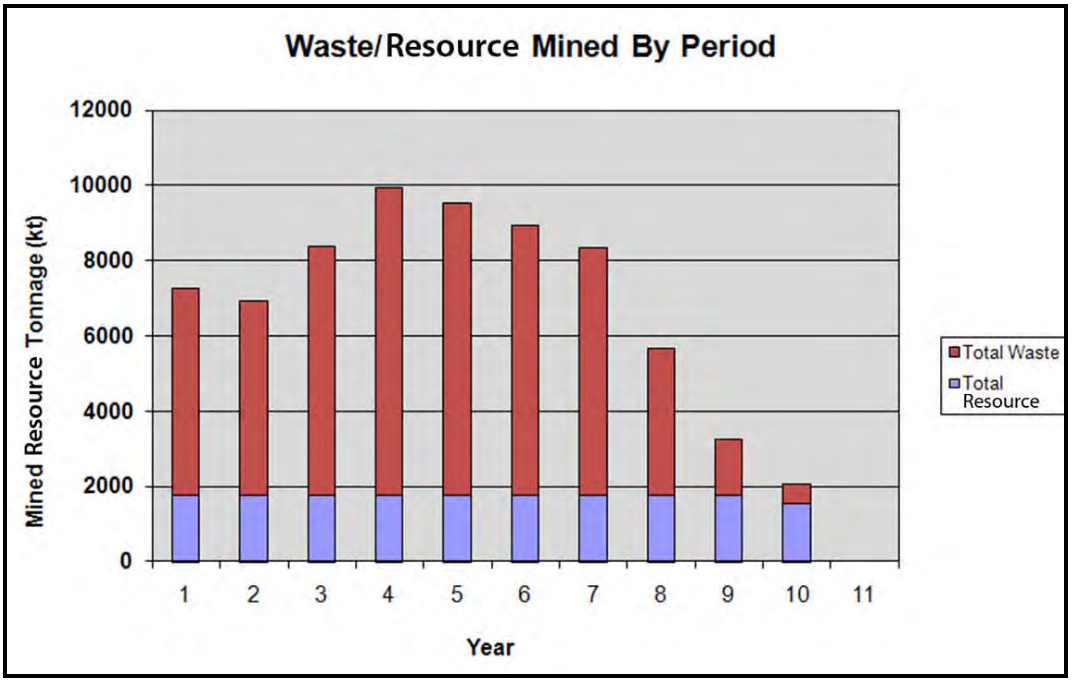


Table 16.9 Overall Ultimate Pit Production Schedule

Period (Years)	1	2	3	4	5	6	7	8	9	10	Total
Lundberg Waste (kt)	5,015	4,400	5,346	6,758	5,993	5,556	5,439	3,512	1,469	541	44,028
Engine House Waste (kt)	483	767	1,283	1,423	1,782	1,613	1,151	390	12		8,903
Total Waste (kt)	**5,497**	**5,167**	**6,628**	**8,180**	**7,776**	**7,169**	**6,590**	**3,902**	**1,481**	**541**	**52,931**
Lundberg Resource (kt)	1,669	1,694	1,732	1,643	1,702	1,694	1,540	1,536	1,714	1,529	16,452
Engine House Resource (kt)	81	56	18	107	48	56	210	214	36		827
Total Resource (kt)	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**1,529**	**17,279**
Zn (%)	2.51	1.83	2.25	1.16	1.51	1.48	1.33	1.45	1.36	1.34	1.63
Pb (%)	1.06	0.78	0.90	0.52	0.68	0.65	0.54	0.56	0.62	0.61	0.69
Cu (%)	0.57	0.48	0.47	0.48	0.40	0.28	0.36	0.35	0.34	0.24	0.40
Ba (%)	4.63	0.70	0.74	0.61	1.54	0.90	0.85	0.86	0.66	0.86	1.24
Ag (g/t)	10.14	5.21	8.51	4.14	6.01	4.55	5.09	5.59	4.99	5.23	5.96
Au (g/t)	0.10	0.07	0.10	0.04	0.06	0.04	0.04	0.08	0.11	0.05	0.07
Total Tonnage (kt)	**7,247**	**6,917**	**8,378**	**9,930**	**9,526**	**8,919**	**8,340**	**5,652**	**3,231**	**2,069**	**70,209**

Figure 16.14 Mined Resource Tonnage & Silver Grade by Period

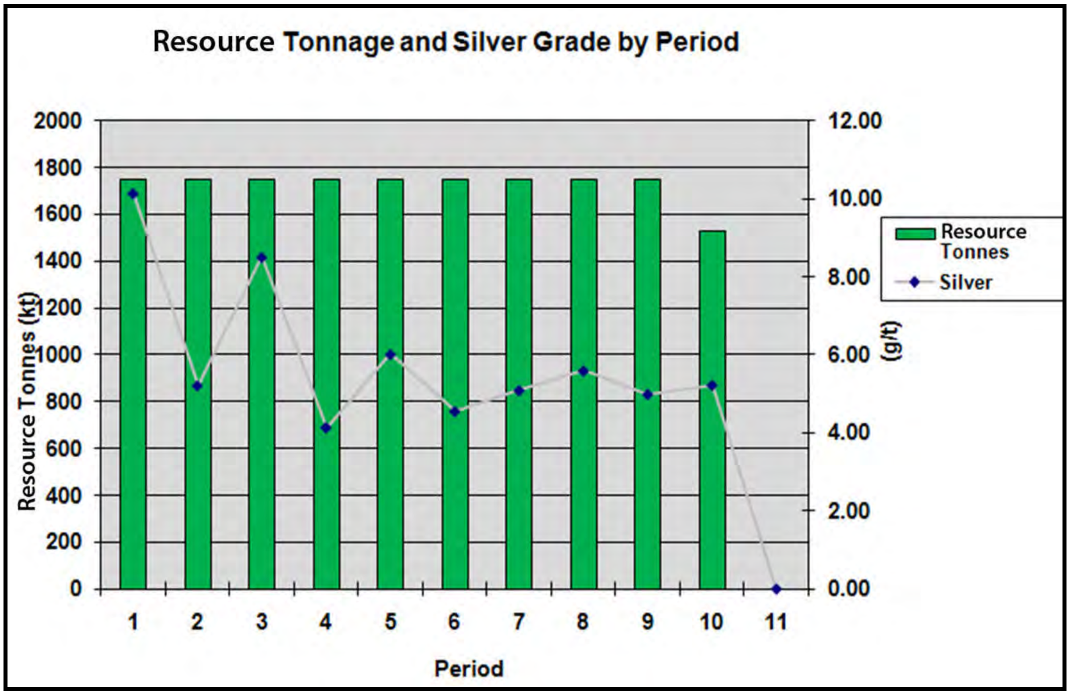


Figure 16.15 Mined Resource Tonnage & Gold Grade by Period

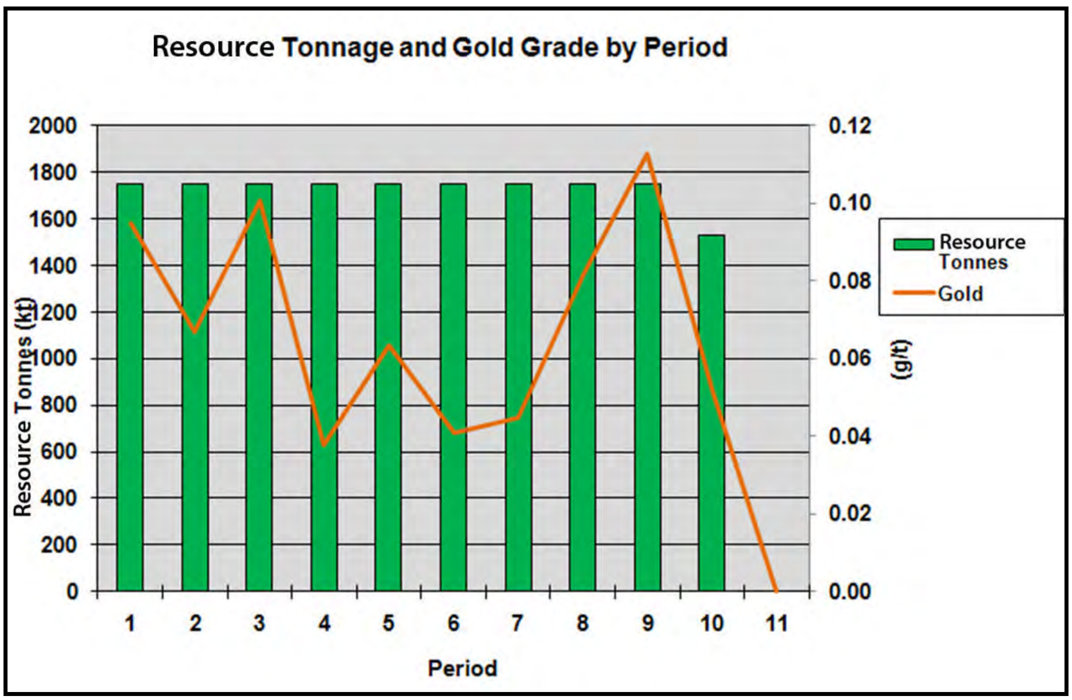
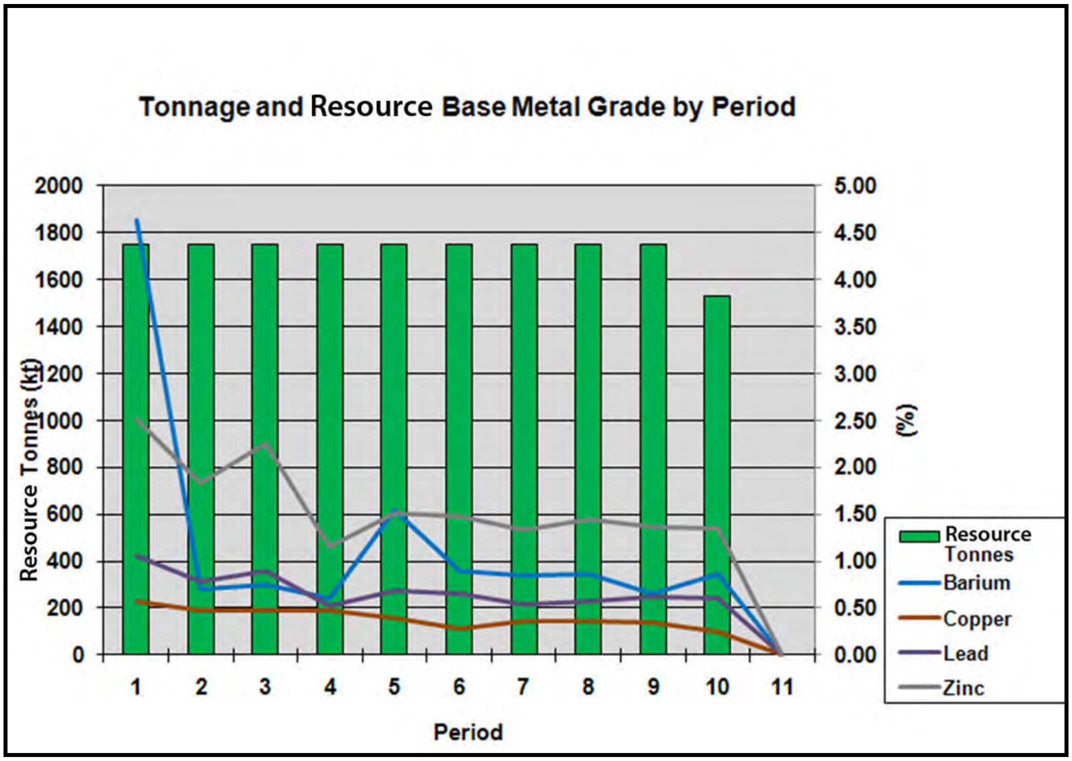


Figure 16.16 Mined Resource Tonnage & Resource Base Metal Grade by Period



Various graphical representations of the ultimate pit design are shown in Figure 16.17, Figure 16.18, and Figure 16.19 and general ultimate pit statistics are outlined in Table 16.10. Figures 16.18 and 16.19 are not scaled.

Figure 16.17 Ultimate Pit Design (Plan View) – note grid is UTM coordinates in meters.

Figure 16.18 3D Rendered Image of Ultimate Pit Design

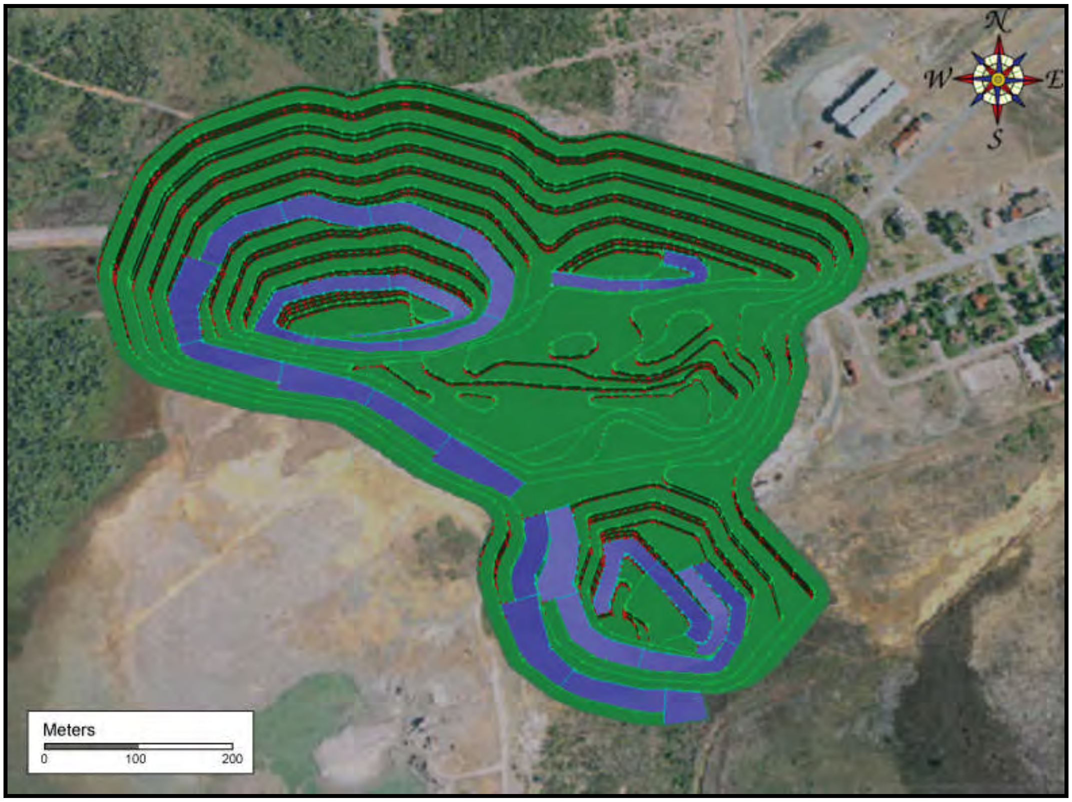


Figure 16.19 3D Rendered Image of Ultimate Pit Design Including Block Model

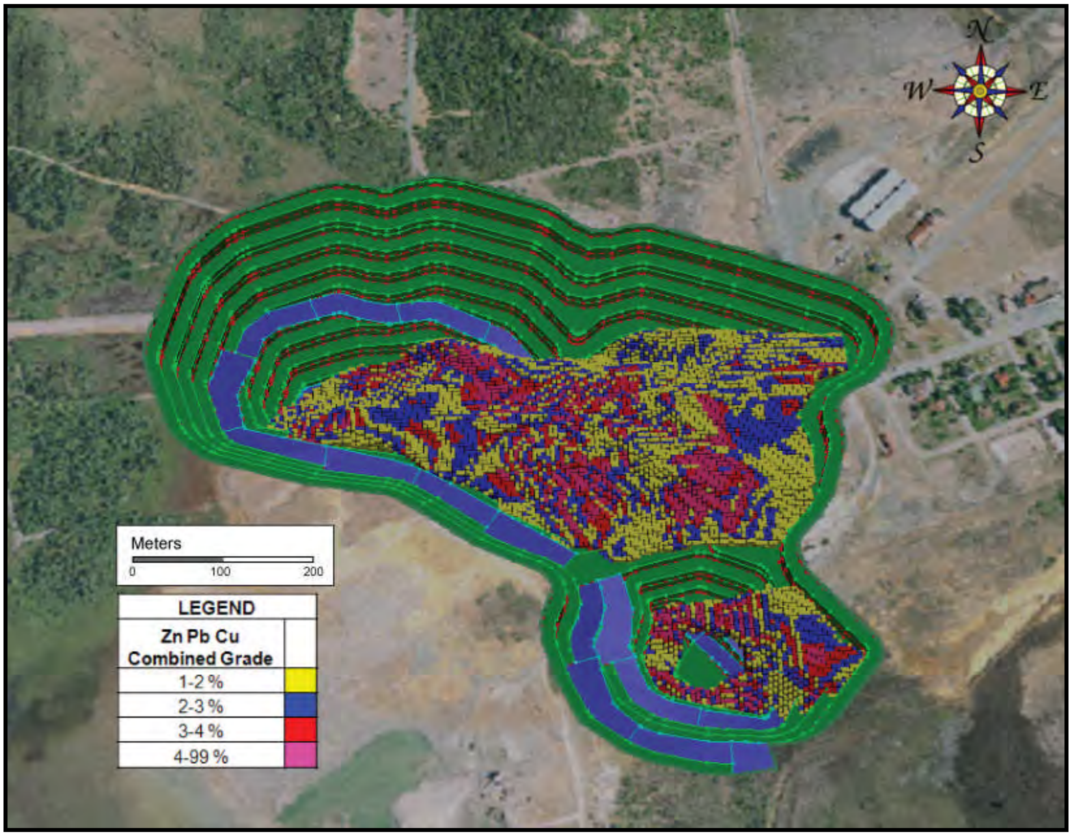


Table 16.10 General Pit Statistics

Item	Size
Pit Top Elevation	Approx. 295 m
Pit Bottom Elevation	90 m
Pit Depth	205 m
Volume of Pit	25.51 million m^3
Area of Pit Top	383,000 m^2
Perimeter at the Top of the Pit	2.70 km
Length from East to West	850 m
Length from North to South	760 m

16.6.4 PIT WATER HANDLING

The progressive development of the open pit will result in increasing water infiltration from precipitation and groundwater inflows. As the pit deepens and increases in footprint, it will be necessary to control water inflow through the construction of in-pit dewatering systems such as drainage ditches, sumps, pipelines and pumps.

16.7 Mine Equipment Selection

For this project, the mechanical availability of mining equipment for the 10 year mine life is presented in Table 16.11, Table 16.14, and Table 16.17.

An equipment fleet consists of a 251 mm blasthole drill, an 11 m^3 diesel hydraulic shovel, a 11.5 m^3 diesel hydraulic loader, and five 90-tonne haul trucks, supplemented by support equipment such as tracked dozers, graders, water truck and other minor support equipment.

16.7.1 Drilling and Blasting Parameters

Drilling Requirement

An 8 m x 8 m blasting pattern has been evaluated for waste and mineable resource. A 251 mm blasthole drill is selected as a primary drill. The preservation of rock mass integrity is to allow for the development of the steepest wall slope by applying careful blasting methods. A pre-shear and buffer blasting practice will be implemented adjacent to the final pit walls to minimize damage to the final pit walls due to blasting.

Table 16.11 gives the annual net operating hours available per drill unit. The average drilling rate is derived from drillhole parameters.

Table 16.11 Yearly Blasthole Drill Net Operating Hours Available per Unit

Drills		1	2	3	4	5	6	7	8	9	10
Total Time	hrs/unit	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400
Phys. Availability	%	85%	82%	79%	76%	73%	70%	70%	70%	70%	70%
Down Time	hrs/unit	1,260	1,512	1,764	2,016	2,268	2,520	2,520	2,520	2,520	2,520
Available Time	hrs/unit	7,140	6,888	6,636	6,384	6,132	5,880	5,880	5,880	5,880	5,880
Standby Time	hrs/unit	0	0	0	0	0	0	0	0	0	0
Gross Operating Hours	hrs/unit	7,140	6,888	6,636	6,384	6,132	5,880	5,880	5,880	5,880	5,880
Operating Delays	hrs/unit	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575
Net Operating Hours	hrs/unit	5,565	5,313	5,061	4,809	4,557	4,305	4305	4,305	4,305	4,305

Drilling Productivity

Table 16.12 and Table 16.13 show penetration and drilling rates for a set of drill hole parameters. A diesel-powered hydraulic percussion track drill might be used for secondary blasting of oversize material, sinking cut drilling, pre-shearing, etc. (not accounted for in OPEX/CAPEX).

Table 16.12 Penetration and Drilling Rates

	Units	Waste	Resource
Hole Depth	m	11.6	11.6
Penetration Rate	cm/min	65	65
Grade control sampling time	min	2.0	2.0
Move and Align Time	min	2.0	2.0
Total Time Per Hole	min	22.85	22.85
Holes Per Hour	holes	2.63	2.63
Average Drilling Rate	m/h	30.5	30.5

Table 16.13 Blasthole Drill Productivity and Blasting Parameters

Blast Hole Drill Productivity	Units	Waste	Resource
Hole Diameter	cm	25.1	25.1
Bench Height	m	10	10
Sub grade	m	1.6	1.6
Powder Factor	kg/t	0.23	0.28
Bank Density	t/m^3	2.712	2.880
Rock Mass per Hole	T	1,736	1,843
Spacing and Burden	m	8	8
Drilling Rate	m/h	30.5	30.5

Blasting Requirement

Overall explosive consumption was based on using a 70% ANFO and 30% emulsion mix product. Some blasting parameters may be seen in Table 16.13. Drillhole liners are to be used in wet holes where practical.

The preservation of rock mass integrity is to allow for the development of the steepest wall slope by applying careful blasting methods. A pre-shear and buffer blasting practice will be implemented adjacent to the final pit walls to minimize damage to the final pit walls due to blasting.

Table 16.13 identifies power factors of 0.23 kg/t and 0.28 kg/t (explosive/blasting material) are being used for resource and waste respectively.

The selected explosive supplier is to erect a plant and storage facility on site. Under the supervision of the drill/blast foreman, the supplier will be contracted to supply, deliver, and load explosives into the blastholes. The drill/blast foreman will also oversee the blasting crew who will prime, stem, and tie-in blastholes.

FINAL PIT WALL BLASTING

The preservation of rock mass integrity is to allow for the development of the steepest wall slope by applying careful blasting methods. A pre-shear and buffer blasting practice will be implemented adjacent to the final pit walls to minimize damage to the final pit walls due to blasting.

16.7.2 *MAJOR EQUIPMENT SELECTION*

The mining equipment was selected to match the mine production schedule, which is based on 350 days per year, with two crews working 12-hour shifts. Equipment selection, sizing, and fleet requirements were based on expected operating conditions, haulage profiles, production cycle times, mechanical availability, and overall utilization. To determine the number of units for each equipment type (drill, shovel, truck, etc.), annual operating hours were calculated and were compared to the available annual equipment hours (see Appendix E for details).

SHOVEL AND TRUCK REQUIREMENTS

In order to meet a production rate of 5000 t/d of resource, three 90 tonne trucks, one 11 m^3 shovel, and one 11.5 m^3 loader will initially be required. This will ramp up to four trucks in Year 3 and five in Year 4. The yearly equipment requirements are shown in Table 16.19.

Table 16.14 Truck, Shovel and Loader Requirements by Year

Equipment	1	2	3	4	5	6	7	8	9	10
Trucks										
Phys. Avail.	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Utilization	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%
Productivity (wmt/hr)	443	415	416	389	378	348	315	287	261	229
Number Required	3	3	4	5	5	5	5	4	3	2
Shovels										
Phys. Avail.	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Utilization	69%	69%	69%	69%	69%	69%	69%	69%	69%	69%
Productivity (wmt/hr)	1,744	1,744	1,744	1,744	1,744	1,744	1,744	1,744	1,744	1,744
Number Required	1	1	1	1	1	1	1	1	1	1
Loaders										
Phys. Avail.	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Utilization	69%	69%	69%	69%	69%	69%	69%	69%	69%	69%
Productivity (wmt/hr)	1,550	1,550	1,550	1,550	1,550	1,550	1,550	1,550	1,550	1,550
Number Required	1	1	1	1	1	1	1	1	1	1

Auxiliary Equipment Requirements

Support equipment such as dozer, grader, water, lube, and fuel trucks were matched with the major mining units. Emphasis has been placed on road construction and maintenance. Support equipment was included for the mechanical and electrical servicing of the mining fleet. The proposed mine equipment fleet is presented in Table 19.15.

Table 16.15 Proposed Mine Equipment Fleet

Equipment Fleet	Size	Model	Units
Haul Trucks	90.9 tonnes	HD785-7	5
Shovels	11 m^3	PC2000-8	1
Loader	1.5 m^3	WA900-3	1
Drills	251 mm	Sandvik D55SP	1
Track Dozers		CAT D9T	2
Graders	164 kW	CAT 16M	2
Water Truck	50 ton	CAT 773 EWT	1
Backhoe Excavator	1.5 m^3	CAT 329 DL	1
Utility Loader/Tire Handler			1
Boom Truck	National Crane 20 ton	600E2	1
Welding/Service Truck			1
Fuel/Lube Truck	13,000 L		1
Flatdeck/Float	20 ton		1
Crane	25 ton	Grove YB7725	1
Light Vehicles (Pick-up Trucks)	0.5 ton crew cab		15
Crew Buses	40 passenger		2
Portable Light Towers	6 kW		6
GEN-SET (Site Backup)	90 kW		2
Pumps	Submersible (90 kW)		2

Note: See Appendix E for details

Loading Fleet

The initial loading fleet is to consist of one 11 m^3 diesel hydraulic shovel and a 11.5 m^3 diesel hydraulic loader as back up. The shovel size has been matched with 90-tonne haul trucks to provide a swing cycle of 35 seconds and total truck load time of 2.8 minutes. The loader has been matched with 90-tonne trucks to enable loading in four passes for handling rock, overburden, and resource with a digging cycle of 40 seconds for each type of material.

Operating producing hours per loading unit is presented in Table 16.16.

Table 16.16 Yearly Loading Unit Producing Hours Available

Shovel PC 2000-8		1	2	3	4	5	6	7	8	9	10
Total Time	hrs/unit	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400
Standby Time	hrs/unit	0	0	0	0	0	0	0	0	0	0
Operating Delays	hrs/unit	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575
Available Hours	hrs/unit	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825
Phys. Availability	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Net Op. Hours	hrs/year	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801
Efficiency	%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Producing Hours	hrs/year	5,221	5,221	5,221	5,221	5,221	5,221	5,221	5,221	5,221	5,221
Utilization	%	69%	69%	69%	69%	69%	69%	69%	69%	69%	69%
CAT 992K Loader		**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Total Time	hrs/unit	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400
Standby Time	hrs/unit	0	0	0	0	0	0	0	0	0	0
Operating Delays	hrs/unit	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575
Available Hours	hrs/unit	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825
Phys. Availability	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Net Op. Hours	hrs/year	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801
Efficiency	%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Producing Hours	hrs/year	5,221	5,221	5,221	5,221	5,221	5,221	5,221	5,221	5,221	5,221
Utilization	%	69%	69%	69%	69%	69%	69%	69%	69%	69%	69%

Shovel/loader base productivities were calculated using the following sample parameters (see Table 16.17).

Table 16.17 Sample Shovel Productivity Calculation

		PC 2000-8 Shovel	992K Loader
Bucket Capacity (heaped)	m^3	11.0	11.47
Material Weight	dmt/bcm	2.712	2.712
Bulk Factor		1.3	1.3
Material Weight	dmt/lcm	2.086	2.086
Moisture	%	3%	3%
Material Weight	wmt/lcm	2.149	2.149
Fill Factor	%	95%	95%
Effective Bucket Capacity	m^3	10.46	10.90
Tonnes/Pass	wmt	22.47	23.41
Truck Size Capacity	wmt	90.9	90.9
Truck Fill Factor	%	90	90
Truck Size Capacity	wmt	81.8	81.8
Ave # Passes	Passes	4	4
Truck Spot Time	Sec	32	30
First Bucket Cycle Time	Sec	30	40
Subsequent Bucket Cycle Time	Sec	35	40
Load Time per Truck	Min	2.78	3.17
Maximum Productivity	trks/hr	21.6	18.9
	wmt/hr	1,744	1,550
Truck Availability to Shovel	%	90%	90%
Producing Hours	hrs/year	5,221	5,221
Annual Wet Tonnes	Mwmt /year	9.4	8.3
Base Productivity	wmt /NOH	1,617	1,437

Note: See Appendix F for details

The material weight in the sample calculation uses 2.712 t/bank m^3. The Base Productivity was used under normal ideal loading condition. Maximum productivities for both resource and waste materials of 1,744 and 1,550 wmt/hr for the shovel and loader respectively were reduced to 90% due to truck availability.

HAULAGE

General Hauling Conditions

The 90-tonne hauler was selected to match the 11 m^3 diesel hydraulic shovel and 11.5 m^3 front end loaders in determining the number of trucks required for each operating year.

Cycle times were based on measured haulage profiles from ultimate pit design based on material types. It was assumed that the waste dump would be within 1,000 m of the pit entrance and resource crushing facilities within 500 m of pit entrance.

Haul Truck Productivity

Truck productivities were based on expected operating conditions, haulage profiles, production cycle times. Cycle times were calculated using Caterpillar Inc's Fleet, Production and Cost (FPC) software. Each bench for each phase was assigned a specific cycle time according to its final destination. A table of all the cycle times are given in Appendix G. Haulage analysis assumptions are outlined in Table 16.18.

Table 16.18 Assumptions used for Cycle Time Calculations

	Value	Units
Max speed on dump and around shovel (300m)	20	km/h
Max speed in-pit (and ramps)	40	km/h
Max speed out of pit	60	km/h
Max speed on flat	50	km/h
Max speed on downhill for safety/account for corners	20	km/h
Loading time (CAT 992K)	3.2	min
Loading time (PC 2000-8)	2.78	min
Dumping time	0.5	min
Swell Factor	30	%
Waste Rock Density (Insitu)	2.712	t/m^3
Resource Density (Insitu)	2.880	t/m^3
Waste Rock Density (Loose)	2.086	t/m^3
Resource Density (Loose)	2.215	t/m^3
Truck used	CAT 777F (90t)	
Truck used (Komatsu)	HD 785-7 (90t)	
Resource Loader	CAT 992K	
Resource Loader (Komatsu)	WA900	
Waste Rock Shovel (Komatsu)	PC 2000-8	

Note: At top of pit, trucks slow to 10km/h each way to act as yielding at intersection (see Appendix G for details)

All ramps were assigned a grade of 10% in the pit and on the dumps. A maximum speed of 40 km/h was used in most conditions but was reduced to 30 km/h when on the main ramp in the pit for safety.

Net Productive Operating Time: The yearly breakdown for the net productive truck operating times is shown in Table 16.19.

Table 16.19 Yearly Truck Net Operation Hours Available per Unit

Trucks		1	2	3	4	5	6	7	8	9	10
Total Time	hrs/unit	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400
Standby Time	hrs/unit	0	0	0	0	0	0	0	0	0	0
Operating Delays	hrs/unit	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575	1,575
Available Hours	hrs/unit	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825	6,825
Phys. Availability	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Net Op. Hours	hrs/year	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801	5,801
Utilization	%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%	69.1%

17.0 RECOVERY METHODS

Please refer to Section 13.0 Mineral Processing and Metallurgical Testing.

18.0 PROJECT INFRASTRUCTURE

The proposed major infrastructure on site will include:

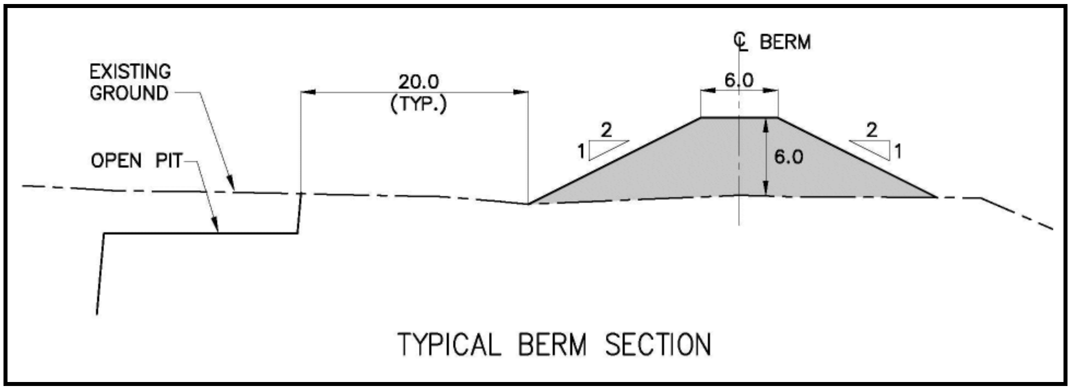
- processing plant and supporting operations buildings
- camp complex
- fuelling storage facilities
- stockpile
- waste dump
- access roads
- settling ponds.

The mill facilities will be located north of the open pit. This will allow for easy access to the ramp from the pit and will utilize the available hard ground for road and facility construction. In order to minimize noise and dust impacts on the town of Buchans from the mine operations, a 6 m high x 6 m wide berm has been included on the east side of the open pit. The berm length is 830 m. The proposed berm cross section is shown in Figure 18.1.

It is anticipated that some buildings within the existing town will require relocation in order to facilitate the mine operation and berm construction.

See Appendix H for detailed layouts of the mill facility and site layouts and a conceptual water balance flow sheet.

Figure 18.1 Berm Cross Section



18.1 CAMP COMPLEX

The camp complex is proposed to be located in the northwest corner of the site. The higher site elevation will eliminate on-site drainage and flooding concerns. The campsite will be graded to control surface water flow. A diversion ditch is planned around the north and west perimeter of the camp complex to capture natural runoff. The diversion ditch will discharge to the water shed located to the southwest of the camp complex.

The following buildings will be part of the camp complex:

- flotation mill and laboratory
- mine dry including shift change rooms
- mine office complex
- general maintenance workshop.
- fueling station
- Warehouse facility

A 23 m wide haul road is designed to exit the open pit on the south end and access the primary crusher located to the north of the pit. The haul road will traverse the western perimeter of the open pit. The haul road will extend to the south to access the proposed waste rock dump area. The total haul road distance will be approximately 2 km. The town of Buchans is on the east side of the pit and the location of the pit exit and haul road to the crushing facility is designed to minimize noise and dust impacts on the community.

The domestic water supply to the camp complex will be sourced from the town of Buchans. A fire water and pumping chamber will be located on the eastern side of the camp and will provide fire water to the site. Site sewer will be directed to the Buchans municipal sewer system. A plant site run off pond will be located in the camp complex and will be designed to capture water discharge from the processing plant and in-pit dewatering from the open pit. The discharge from the plant site run off pond will be directed to the water treatment plant.

18.2 MINED RESOURCE STOCKPILE

The stockpile is located in close proximity of the new mill location. It is proposed that the ROM stockpile capacity will be 75,000 t, which equates to 15 days of mine production.

18.3 UTILITIES

Electrical power is generated by Newfoundland and Labrador Power Generation and the power is sold to Newfoundland Power utility for transmission and distribution. Newfoundland and Labrador Hydro (NLH) is the primary generator of electricity in Newfoundland and Labrador. The company has an installed generating capacity of 1,637 MW. Over 80% of the energy generated in 2010 was clean, hydroelectric generation.

Newfoundland Power purchases approximately 90% of its electricity from NLH. Newfoundland Power maintains approximately 11,000 km of transmission and distribution lines.

There is currently an 8.3 MVA transformer located 1 to 2 km outside of the town of Buchans. Currently 1 to 2 MVA is in use and there is deemed to be adequate capacity to serve the proposed mining operations. Other possibilities include using hydroelectric from nearby generation locations.

Star Lake is a 15 MW hydroelectric facility currently operating on the island of Newfoundland and currently connected to a transmission station located near Buchans. The facility entered service in October of 1998 and is selling power to the provincial crown-utility, NLH. Star Lake is displacing energy from the facilities of NLH. The NLH resource being displaced is the 490 MW conventional oil-fired facility at Holyrood. Creating Energy Reduction Credits (ERCs) from renewable energy sources such as hydroelectric facilities encourages the continued operation and future development of these types of non-emitting sources of electricity, thereby enhancing long-term air quality and reducing environmental damage. There is a possibility that this mine project could be incented to use the hydroelectric generated power.

Available voltage at 12.5 kV would be accessible to the mine sight. Pole mounted transformers would be used for loads less than 300 kVA. Larger loads would be serviced by pad mounted transformers. Negotiations with the utility for sharing the costs of capital for electrical infrastructure to service the mine site is highly likely as the Buchans area is over serviced for their load profile, and there is an incentive to assist new mine development in the area.

18.4 WASTE ROCK DUMP

The waste rock dump location was selected based on its close proximity to the pit, distance from the town, as well as the bowl shaped topography of the valley. The selected site for the waste rock dump is a small valley located approximately 1 km south of the open pit, east of the existing Tailings Pond 2 (TP2).

Most of the region is underlain with dense glacial till and it had a relatively low permeability. Based on this assumption, it was determined that no synthetic base liner would be needed under the waste rock dump. Given the shape of the valley, all seepage water could be easily captured with a collection ditch and pond located on the north side. The water will be transferred to a water treatment plant prior to discharge.

The dump was sized to handle 22.3 Mm^3 of waste material over the 10 year mine life. This volume was calculated based on test data of the waste rock which indicated a unit density of 2.7 g/cm^3. A swell factor of 30% was used and is typical for the blasted rock. The dump will be constructed in 10 m lifts with 15 m wide benches for slope stability. The maximum height will be 40 m or elevation 300 m. At closure, the dump will be re-graded, covered with soil, and vegetated.

18.5 TAILINGS MANAGEMENT FACILITY DESIGN

18.5.1 DESIGN RATIONAL

The Buchans mine site is a historic mining area and was actively mined from 1928 to 1984. Over 16.2 million tonnes of ore were produced which resulted in about 5.7 million tonnes of copper, lead and zinc concentrates. Based on historical operations, approximately 10 million tonnes of tailings were produced over the life of mine and the majority was deposited in Tailings Pond 1 (TP1) and TP2.

The existing tailings management facility (TMF) and downstream waterways have been impacted by mining and the area is considered a ‘brownfield’ site. This is one of the primary reasons for selecting an existing historic tailings pond as the location of future tailings deposition from the new open pit mine. In addition, there are no other green-field locations with favorable topography, storage capacity and proximity to the new mine location.

In addition in the summer of 2010, the Newfoundland Government issued a contract for the improvement and stabilization of the containment dams and spillways around the existing TMF. The construction work was completed and the dams now meet the long term stability requirements recommended by the Canadian Dam Association (CDA, 2007). Currently Buchans has no ownership of the TMF and thus no liability.

TP1 has a large upstream watershed and placement of tailings in this pond would require treatment of an additional volume of water. There is also a potential for continuation of a barite tailings mine operation in TP1. TP2 has a very small watershed associated with it and has sufficient capacity for storage of all tailings from the new mine operation. As a result, TP2 was selected as the location for the new TMF.

18.5.2 DESIGN BASIS

The life of mine tailings storage capacity was based on the mine schedule of 5,000 t/d over a period of approximately 10 years. Based on experience from other similar base metal projects, an average tailings density of 1.6 g/cm^3 was used in the calculations. This equates to a total volume of 10.8 Mm^3 of tailings over the life of mine.

TP2 has remaining tailings storage capacity and since the containment dams have been reinforced and a new spillway constructed, there is no immediate need to reinforce or raise the dams prior to tailings deposition. Based on aerial photography and knowledge of site conditions, it has been estimated that there may be approximately 1.3 Mm^3 of available tailings storage capacity in TP2 which is roughly equivalent to one year of tailings production. Being able to utilize this available capacity would defer capital expenditures for the TMF for the first year of operations.

For the remainder of operations (Years 2 to 10), the existing tailings dams would be raised over the mine life to contain all of the milled tailings. For cost estimating purposes, it was assumed that the dams would be raised every two years to accommodate the milled tailings. Based on the current mine plan, the total volume of tailings produced over the 10 year mine life will result in the ultimate tailings dam having a crest elevation of 303.2 m and the total dam fill will be 1.72 Mm^3.

The tailings dams will be constructed predominantly of clean waste rock from the open pit. The final design will likely incorporate an internal low permeability core along with filter graded materials on the downstream side. For estimating purposes, it was assumed that the dam slopes will be constructed at 2.5 (V):1(H) utilizing the downstream method of construction. During later design stages, there may be some opportunities to further reduce construction costs (steeper side slopes, centerline construction, etc.).

In order to minimize long term operating costs, it is recommended that one permanent spillway be constructed in the first few years of operation along the southeast corner of TP2. This will eliminate the need to reconstruct a new spillway every two years with the associated dam raises.

It should be noted that if additional ore reserves are found during operations, the TMF capacity can be easily increased by 8.5 Mm^3 (equivalent to approximately 8 years of additional LOM based on a 5000 t/d mine operation) and remain within the same footprint.

18.5.3 OPERATIONS

The conventional tailings slurry will be spigotted along the slopes of the existing dams (west, north and east sides) in a rotating fashion to maximize storage capacity and to maintain a central pond area. A barge will be utilized to reclaim water from the TMF which will be pumped back to the mill and used as process water. Excess water from the TMF will be pumped to a central water treatment facility prior to discharge. All seepage water from the perimeter dams will be collected using toe ditches and pumped back into the TMF.

19.0 MARKET STUDIES AND CONTRACTS

This section is not applicable to this report.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

This section of the PEA identifies and examines environmental considerations associated with the project, including:

- environmental setting
- environmental assessment and permitting
- community and Aboriginal engagement
- mine closure and reclamation plan.

20.2 ENVIRONMENTAL SETTING

20.2.1 SITE HISTORY

The Project is located west of the town of Buchans in central Newfoundland and involves an open pit mine and associated infrastructure, including two existing tailings ponds, a mill, and waste rock storage. As outlined in Section 6.0, ASARCO operated the former Lucky Strike mine in this location from 1928 to 1984. Existing infrastructure located on the mine site includes historical underground workings; the Lucky Strike Glory hole; two tailings ponds; and previous operations buildings, some of which have been deposited in the Glory hole. The remaining operations buildings are thought to be constructed of asbestos siding and materials deposited in the open pit may contain asbestos. The Lucky Strike Glory hole and historical underground are filled with groundwater and surface water runoff. The water contained in the pit has not been treated.

In 2009, soil testing was conducted around the perimeter of the community of Buchans, and levels of heavy metals (lead) that exceeded human health guidelines were identified in some areas (DEC, 2011). In light of the financial difficulties of involved companies and some community interest, the Government of Newfoundland and Labrador took over remediation of the site. In 2010, the Provincial Government committed $9 million for projects to address the two main source areas that continued to contribute to the lead in soil levels in Buchans. These projects included capping of the Tailings Spill Area and remediation of the Mucky Ditch (completed by the Department of Environment and Conservation), and the rehabilitation of the tailings dams and capping of exposed tailings (completed by the Department of

Natural Resources) (DEC, 2011). The proposed BMC project will construct a new mine at this location using the existing TP2 tailings facility, which presents an opportunity to make use of pre-existing site disturbance and thereby minimize the requirement for additional land disturbance.

20.2.2 WATER RESOURCES

The Town of Buchans' drinking water source is Buchans Lake, which is located approximately 800 m north of the town. The province has designated the area surrounding Buchans Lake as a protected water supply area as of January 20, 2004 (Queen's Printer, 2006). With this designation, activities that may impair the quality of the water and/or unduly diminish the amount of water available in a public water supply area are prohibited under the provincial *Water Resources Act*. Mining infrastructure and operations will be located outside of the protected water supply area boundary; the edge of the protected water supply area is located approximately 600 m north of the centre of the Lucky Strike glory hole. Protection of water quality in Buchans Lake needs to be a consideration in the design and operation of all project components.

The lakes and rivers in this region support a variety of fish species, including Atlantic salmon, brook trout, arctic char, rainbow smelt, American eel, and three-spine and nine-spine sticklebacks (PAA, 2000).

20.2.3 PROJECT LANDSCAPE

The Project is located on the border of the Buchans Plateau/Topsails subregion (within the Long Range Barrens ecoregion) and the Red Indian Lake subregion (within the Central Newfoundland Forest ecoregion) in Newfoundland (PAA, 2000). The northern portion of the Red Indian Lake subregion is characterized by dense forests, bogs (domed bogs are the most common type), and rolling hills. The Buchans Plateau/Topsails subregion is characterized by low, dense thickets of tuckamore, with small patches of stunted forest that is dominated by black spruce. Annual precipitation is typically 1100 to 1500 mm. The project area is generally flat to gently rolling with elevation ranging from 155 m to 165 m at Red Indian Lake to approximately 130 m to 280 m inland. Numerous small brooks draining into Red Indian Lake are present, with spruce and fir growing on the slopes. The northern portion of the property is characterized by poor drainage, shallow bogs, and extensive muskeg in the flat areas. To the south of the property Red Indian Lake occupies a large northeast trending valley.

Wildlife species that occur within this subregion are typical of boreal, forested, and bog ecosystems. The Gaff Topsails and Buchans caribou herds can be found in these sub-regions, and a small population of the provincially and federally threatened Newfoundland marten occurs around Red Indian Lake.

20.2.4 ENVIRONMENTAL BASELINE STUDIES

No EBS's have been conducted specifically for the Project. An EBS is necessary to understand the specific interactions between the project and the natural environment and to design the project to avoid or minimize potential adverse effects. The EBS also would support the preparation of a registration document for the project and an Environmental Impact Statement in the event that it is required by the province (detailed below). An EBS is typically conducted over a minimum of 12 continuous months to provide coverage of all four seasons. Studies may continue beyond this12-month period as may be justified by the occurrence of abnormal seasonal conditions. In cases where the EBS may focus on specific information gaps the study period may be shorter than 12 months. The EBS scope is typically developed in consultation with the local and regional resource management and regulatory agencies in order to ensure agency concerns can be addressed with the study results. The initial EBS report is typically completed within 14 to 16 months of the start of the field program and the Environmental Impact Assessment (EIA) is typically based upon this initial EBS report.

Baseline studies will need to focus on water and groundwater quality and quantity, and on fish and fish habitat. Terrestrial environmental components that may also have to be included in the study include migratory birds and raptors, rare vegetation, and wildlife (particularly Newfoundland marten populations if the project interacts directly with suitable habitat in the vicinity of Red Indian Lake).

20.3 ENVIRONMENTAL ASSESSMENT AND PERMITTING

The project will be subject to the provincial environmental assessment process and may also be subject to an environmental assessment under the Canadian Environmental Assessment Act (CEAA) if an approval is required from a federal agency. These processes are summarized below.

20.3.1 PROVINCIAL PROCESS

ENVIRONMENTAL ASSESSMENT

Under the provincial *Environmental Protection Act* and the *Environmental Assessment Regulations*, a mineral development project must be registered for environmental assessment through the Newfoundland and Labrador Department of Environment and Conservation. The registration document outlines the proposed project and its potential bio-physical and socio-economic effects. In the registration document, the proponent must demonstrate how best management practices and technologies will be used to minimize harmful effects.

For the Buchans project, the registration document should include potential effects to the local landscape, including water quality and quantity, fish and fish habitat, wildlife, and vegetation; this includes federal and provincial Species at Risk. Consideration of these potential effects will require site-specific information. Water quality and water management options will be an important component of the registration document. The following should be included in the registration document: practices to manage proposed tailings management facilities; waste rock and potential acid rock drainage issues; water requirements and water source for the mill; and protection of the community water supply. Socio-economic considerations should also be included; for example, community and Aboriginal engagement plans, protection of community health, and an outline of economic opportunities for local communities and First Nations.

Within 45 days of receiving a registration, the Minister will issue a decision on the proposed project. All decisions are announced in the Environmental Assessment Bulletin. There are three possible decisions (DEC, 2009):

1. The undertaking may be released and the proponent may proceed as indicated in the registration. An Environmental Preview Report may be required to provide additional information not contained in the registration so that the Minister may determine if an Environmental Impact Statement (EIS) is required.
2. An EIS may be required if significant negative environmental effects are indicated or where there is significant public concern. An EIS requires comprehensive environmental review of a project.
3. The undertaking may be rejected if an unacceptable environmental effect is indicated, the project is not in the public interest, and/or the proposal is inconsistent with existing law or government policy.

If an EIS is required, during its preparation the proponent must also meet with interested members of the public in the local area to provide information on the proposed undertaking, and to record and respond to any concerns regarding the environmental effects of the project.

PERMITTING

Proponents should follow the *Environmental Guidelines for Construction and Mineral Exploration Companies* (DNR, 2011) provided by the Newfoundland and Labrador Department of Natural Resources. The *Guidebook to Exploration, Development and Mining in Newfoundland and Labrador* (GNL, 2010) also provides useful guidance on the regulatory process.

Water Quality Management

The project will involve at least one discharge of effluent that may originate from several sources on the project site, including: dewatering of the existing glory hole; open pit dewatering to remove groundwater seepage and precipitation; general site runoff including runoff from ore, waste rock, and overburden stockpiles; and, periodic releases of water from the tailings management area. A water treatment plant has been included in the project design to ensure the quality of any project effluent can be managed to meet effluent criteria that will be applied to the project.

The control and management of water resources in Newfoundland and Labrador is legislated by the *Water Resources Act,* although related development activities cannot be permitted or undertaken without first obtaining authorization from the Province under the *Environmental Protection Act*.

Surface Water

Licences under the *Water Resources Act* will be required prior to release of any effluent. Effluents discharged to surface water from mining activities must, at minimum, comply with Sections 3, 19.1, and 20 of the MMER (Table 20.1). Site specific effluent quality criteria may be imposed as a condition of any approval in the event that compliance with the MMER does not provide adequate protection of receiving water quality. Effluent treatment is expected to be required to meet effluent quality limits for TSS, ammonia, and potentially for management of metal concentrations. Specific treatment requirements will be developed in subsequent project planning phases.

Monitoring of any liquid discharge from the project to receiving waters will be required as part of any provincial environmental permit or approval. The basic monitoring requirements are those detailed in the MMER, which require routine monitoring of deleterious substances (Table 20.1) and effluent volume. Periodic effluent characterization also is required, which includes the deleterious substances and analyses of alkalinity, hardness, aluminum, cadmium, iron, mercury, molybdenum, ammonia, nitrate, major anion and cation species, and project-specific contaminants of concern (COC). Possible COCs for the project may include uranium, antimony, barium, molybdenum, thallium, selenium, and silver. The MMER also require periodic receiving water quality monitoring, and environmental effects monitoring.

Table 20.1 Metal Mining Effluent Regulations, SOR/2002-222 – Authorized Limits of Deleterious Substances

Item	Deleterious Substances	Maximum Authorized Monthly Mean Concentrations	Maximum Authorized Concentration in a Composite Sample	Maximum Authorized Concentration in a Grab Sample
1	Arsenic	0.50 mg/L	0.75 mg/L	1.00 mg/L
2	Copper	0.30 mg/L	0.45 mg/L	0.60 mg/L
3	Cyanide	1.00 mg/L	1.50 mg/L	2.00 mg/L
4	Lead	0.20 mg/L	0.30 mg/L	0.40 mg/L
5	Nickel	0.50 mg/L	0.75 mg/L	1.00 mg/L
6	Zinc	0.50 mg/L	0.75 mg/L	1.00 mg/L
7	Total Suspended Solids	15.00 mg/L	22.50 mg/L	30.00 mg/L
8	Radium 226	0.37 Bq/L	0.74 Bq/L	1.11 Bq/L

Note: All concentrations are total values.
Cyanide only required for mines using cyanide in the metallurgical process.
Current version as posted between Apr 3, 2009 and Apr 15, 2009. SOR/2006-239, s. 25.
Source: Department of Justice 2011

Neither the process water requirement for the mill or the water source has been determined at this time. When determined, water take from any natural surface water body will need to licensed under the *Water Resources A*ct.

Waste rock will be stored in a valley area adjacent to Tailings Pond 2. Some waste rock is expected to be acid generating, given this is a massive sulphide deposit, but the quantity of potentially acid generating waste rock has not yet been determined. The waste rock stockpile will incorporate a collection and containment system, ARD and ML testing will be conducted in subsequent stages of project planning and design.

Groundwater

Hydrogeological conditions in the vicinity of the open pit need to be described in order to estimate the potential for groundwater seepage into the pit, to design the necessary water diversion and water management works, and to assess how the project interactions with groundwater may affect nearby surface water bodies. The natural groundwater may contain metal concentrations above MMER effluent quality limits; for example copper, lead, zinc, and arsenic due to the nature of the deposit.

Dewatering of the open pit and existing glory hole will be required over the course of the project and this also will be licensed under the *Water Resources A*ct

Asbestos Management and Disposal

Infrastructure from the existing mine site will be removed and disposed of as part of new project development. Existing buildings may contain asbestos, and old mine buildings that were insulated with asbestos may have been pushed into the glory hole; these buildings will need to be removed and disposed of according to the provincial regulations detailed below. The handling management, and disposal of asbestos containing materials on the site must be conducted in accordance with the *Asbestos Abatement Regulations* administered by the Occupational Health and Safety Branch of the Department of Environment. These regulations require notification of the Branch in advance of the work with an outline of the proposed Project and location, the type and quantity of ACM, the number of workers that maybe exposed to ACM while completing the proposed work, anticipated duration of the work, and the protective measures to be taken.

All asbestos containing materials must be disposed of in accordance with the *Waste Material Disposal Act* and the Asbestos Waste Disposal Directive, which require that asbestos only be disposed of in an approved waste disposal site as described under the *Consolidated Newfoundland and Labrador Regulation* and/or in accordance with provisions of this policy. Waste disposal areas are listed under the *Regulation*. The nearest waste disposal area to the Project is 192 km away at Fortune Harbour. All other waste disposal areas are >200 km from the Project area. Transporters of asbestos waste must hold a valid Certificate of Approval under provisions of the *Waste Material Disposal Act*.

Other Permits

Mining Lease

A mining lease must be obtained under the provincial *Mineral Act* for exclusive rights to develop, extract, remove, deal with, sell, mortgage, or otherwise dispose of all the unalienated materials, or those specified in the lease, in, on or under the land described in the lease (GNL, 2010). Surface rights that include the entire footprint of the mine and related infrastructure must also be obtained under the *Mineral Act.*

Mill License

A mill license is required for operation of a mill in conjunction with a mining operation, as per Section 5 of the *Mining Act*. Mill licenses are issued by the Department of Natural Resources to the holder of a mining lease (GNL, 2010), and a mill may not be operated without first obtaining a mill license.

Fuel Storage and Handling

Fuel storage and handling in Newfoundland and Labrador is regulated by *The Storage and Handling of Gasoline & Associated Products Regulations*, and a Certificate of Approval for a fuel storage system must be obtained from the Department of Government Services and Lands. Registration is required for all underground and above ground storage facilities for the storage and handling of fuel and associated products.

Explosives

Explosives must be stored at least 22.86 m from a road and 30.48 m from an occupied building. Explosives in excess of 68.04 kg can be kept only on premises which have been licensed under *The Explosives Act* (Canada). All transportation of explosives must conform to *The Fire Commissioners Act* and *The Explosives Act* (Canada). Permits related to explosives are often held by the explosives supplier in circumstances where the onsite storage facilities are owned and operated by the supplier.

20.3.2 FEDERAL PROCESS

ENVIRONMENTAL ASSESSMENT

Any requirement for a federal environmental assessment would be conducted in accordance with the Draft Canada-Newfoundland and Labrador Agreement on Environmental Assessment Cooperation (2005). The Provincial government and

CEA Agency will advise proponents at the earliest opportunity about the potential for a cooperative environmental assessment of a proposed project.

Canadian Environmental Assessment Act

The Project registration document will be circulated to the Canadian Environmental Assessment (CEA) Agency and to federal authorities such as Environment Canada, Health Canada, Fisheries and Oceans Canada, Natural Resources Canada and Transport Canada. The federal agencies will determine if a federal environmental assessment is necessary. A federal environmental assessment is typically triggered when a federal authority determines it must provide a license, permit or an approval that enables a project to be carried out (e.g., authorization under the federal *Fisheries Act*).

If a federal agency determines that it must issue a permit or approval for the project, the federal family would then determine the level of environmental assessment to be applied to the project. The level of environmental assessment that is necessary for a mining operation in the presence of a CEAA trigger is determined by a number of factors which are outlined in the *Comprehensive Study List Regulations* under CEAA. The basic level of assessment is the screening level. The next level is the comprehensive study, which is typically applied to larger and more complex projects.

A metal mine with a planned production rate of 3,000 t/d or greater is subject to a comprehensive study. As such, the proposed mine, with a planned production rate of 5,000 t/d, would undergo a comprehensive study in the event that a federal approval is required. This assessment would capture all aspects of the Project, including the mine and adjacent surface facilities, any new roads, the mill, tailings management areas, and waste rock management areas. Federal permits or approvals cannot be issued until the federal agencies complete and sign off on a comprehensive study report.

The proposed project is considered a natural resource development which triggers involvement of the Major Project Management Office (MPMO) to provide overarching project management for a federal environmental assessment if required. The MPMO is administered by Natural Resources Canada whose role is to provide guidance to project proponents and other stakeholders, coordinate project agreements and timelines between federal departments and agencies, and to track and monitor the progression of major resource projects through the federal regulatory review process.

FISHERIES ACT

Fisheries and Oceans Canada (DFO) is responsible for protecting fish and fish habitat in Canada. Under section 35(1) of the federal *Fisheries Act*, works that result in the harmful alteration, disruption or destruction (HADD) of fish habitat must be authorized in advance by DFO, (DFO 2002).

If a DFO Authorization is required, it can take anywhere from one month to several years to obtain an Authorization, depending on the type of approval required, the complexity of the project, and any associated field studies. Other Project activities (e.g., construction of crossing structures [culverts] through fish habitat, any work in or about a fish-bearing watercourse that may disturb, alter or destroy fish habitat] will require an Authorization under the Fisheries Act if they result in a HADD.

Habitat compensation is an option for achieving no net loss when residual impacts of projects on habitat productive capacity are deemed harmful after relocation, redesign or mitigation options have been implemented. Habitat compensation involves replacing the lost habitat with newly created habitat or improving the productive capacity of some other natural habitat. Depending on the nature and scope of the compensatory works, habitat compensation may require (but is not limited to) five years of post-construction monitoring (DFO 2002).

20.4 Community and Aboriginal Engagement

The implementation of an effective community and Aboriginal engagement program is fundamental to the successful environmental permitting of mining projects. The purpose of this program is to ensure that all potentially affected persons, businesses, and communities have a full understanding of the project and an opportunity to share information with respect to concerns regarding potential effects, and so the proponent has an opportunity to explain how these concerns are addressed in the project design and operations. This program typically begins in the early stages of project planning and continues through the life of the project.

The community engagement phase of the project will ideally be initiated as early as possible and requires very careful thought and planning. Evidence of community engagement is required throughout the provincial and federal environmental assessment processes. If mining plans are likely to change as the project progresses, it is important to keep the community well informed.

Consultation with Aboriginal groups should also be initiated as early as possible. For the Lundberg and Engine House deposits, the Federation of Newfoundland Indians should be contacted to determine which Mi'kmaq communities or organizations should be included in the public engagement program. The Sple'tk (Exploits) First Nation currently has members in the Buchans community.

In addition to a continuing public engagement program, it may be necessary to negotiate an impact/benefit agreement (IBA) with potentially affected stakeholder groups in order to, in part, address potential adverse effects of the project on traditional resource users. These agreements can take many forms and no single formula is applicable to all situations. However, the agreements typically lay out various forms of economic stimulation or benefit specifically designed and intended to benefit specific affected stakeholder groups.

20.5 Mine Closure and Rehabilitation Plan

A Development Plan and Rehabilitation and Closure Plan (GNL, 2010) must be submitted prior to the commencement of project development, and typically should be developed at the same time as the environmental assessment for the project. The rehabilitation and closure plan addresses the closure of all components of the project under provincial mine closure regulations, and the plan must be included with the application for a mineral lease. Mine closure plans are developed in accordance with the Guidelines to the *Mining Act*. Financial assurance must be posted to secure the rehabilitation works, and needs to be sufficient to cover any continuing maintenance and monitoring that may be required.

Throughout the term of the mining lease, the proponent is required to take all reasonable steps to progressively rehabilitate the mine site, whether or not the closure plan has commenced. A description of progressive rehabilitation work plans must be submitted for each year of the mining lease term (DNL, 2010).

It is recognized that a practical reclamation strategy may change from the inception of a mining project due to unforeseen changes in the mining plan, site conditions or improving technology. A proponent is required to revise and resubmit the reclamation strategy in the event that a project deviates from the documented development plan.

Closure and reclamation of the project site will involve the removal/demolition of all surface structures and buildings. Concrete foundations will be demolished to grade and covered with nonARD-generating crushed waste rock. All disturbed areas will be graded to restore the natural drainage patterns to the extent possible and scarified and seeded to promote re-vegetation. Waste rock stockpiles will be contoured to promote drainage and maintain stable side slopes, covered with 30 cm of overburden and seeded to promote re-vegetation. The tailings management area will be closed out with a dry, water shedding cover. All seeding will be with grass species appropriate to the local ecology. Run-off and seepage quality will be monitored following completion of the closure works to verify the performance of the works in maintaining acceptable water quality.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

21.1.1 SUMMARY

Table 21.1 presents a detailed summary of the direct and indirect capital costs for the project. These costs are calculated at Cdn$127 million (2011 base year). A 15% contingency has been applied to the direct capital costs. The indirect costs have been factored from the direct capital cost. The indirect costs have been calculated as 8% of the direct capital on an annual basis and the owner's cost has been calculated as 3% of the direct capital on an annual basis. Salvage value has been calculated as 10% of the direct capital costs for the following areas; open pit mining, processing, and non-process building. Detailed capital costs are located in Appendix I.

Table 21.1 Direct and Indirect Capital Costs

Item	Amount (Cdn$)
Direct Capital Costs	
Site Development	11,972,222
Utilities	8,940,883
Tailings Management Facilities	17,173,550
Open Pit Mining	22,675,003
Resource Processing	56,084,677
Non-process Buildings	3,760,000
Closure/Reclamation	6,600,000
Subtotal Direct Capital Costs	**127,206,336**
Indirect Capital Costs	
Indirect Costs	10,176,507
Owner's Costs	3,816,190
Contingency (15%)	19,080,950
Salvage	(8,251,968)
Subtotal Indirect Capital Costs	**24,821,679**
Total Capital Costs	**152,028,015**

21.1.2 DIRECT CAPITAL

The direct capital cost breakdown consists of the following major category descriptions:

- site development
- site utilities
- tailings management facility
- open pit mining
- processing facilities
- non-process buildings
- closure and reclamation costs.

The direct capital costs are further defined by areas, which comprise the major components of each category. The package definition is the level of detail which capital costs were determined. In some instances costs were determined based on historical data from projects of similar size and in other instances engineering first principles were used to determine equipment and labour requirements, and thus capital costs. Sustaining capital has been considered for open pit mining and processing and is included in the direct capital costs totals. Sustaining capital has been calculated based on the expected replacement required to sustain the operation for major equipment.

21.1.3 INDIRECT CAPITAL

The indirect capital costs breakdown consists of the following major category descriptions:

- owner's cost
- indirect cost
- contingency
- salvage.

Owner's costs were estimated at 3% of the total direct capital cost. This was allocated in the same year as the direct capital cost. The cost values were determined based on historical data from projects as well as engineering estimates. For this study, spare parts and first fill costs have not been considered.

21.1.4 MINING

The total mine capital costs for the mining portion of this project are estimated to be Cdn$22.7 million consisting of Cdn$17.4 million in initial capital and Cdn$5.3 million in sustaining capital over a period of 10 years. The mining capital cost breakdown is outlined in Table 21.2 and

Table 21.2 Initial Mine Equipment Capital Cost

Equipment Fleet	Size	Model	Qty	Unit Cost (Cdn$)	Total (Cdn$)
Haul Trucks	90.9 t	HD785-7	3	1,245,000	3,735,000
Shovels	11 m^3	PC2000-8	1	2,320,000	2,320,000
Loader	1.5 m^3	WA900-3	1	1,400,000	1,400,000
Drills	251 mm	Sandvik D55SP	1	1,950,690	1,950,690
Track Dozers		CAT D9T	2	1,333,185	2,666,370
Graders	164 kW	CAT 16M	2	1,155,000	2,310,000
Water Truck	50 ton	CAT 773 EWT	1	315,000	315,000
Backhoe Excavator	1.5 m^3	CAT 329 DL	1	283,500	283,500
Utility Loader/Tire Handler			1	315,000	315,000
Boom Truck	National Crane 20 ton	600E2	1	231,000	231,000
Welding/Service Truck			1	100,800	100,800
Fuel/Lube Truck	13,000 L		1	132,300	132,300
Flatdeck/Float	20 ton		1	145,050	145,050
Crane	25 ton	Grove YB7725	1	325,500	325,500
Light Vehicles (Pick-up Trucks)	0.5 ton crew cab		15	42,000	630,000
Crew Buses	40 passenger		2	105,000	210,000
Portable Light Towers	6 kW		6	15,750	94,500
GEN-SET (Site Backup)	90 kW		2	24,127	48,254
Pumps	Submersible (90 kW)		2	73,500	147,000
Total Open Pit Equipment					**17,362,964**

Pricing for major production equipment was obtained from vendors (Komatsu Canada, Sandvik, and Caterpillar) and Tetra Tech's in-house cost database.

A sustaining CAPEX for additions, replacements, and re-builds of mining equipment has been estimated to match the annual production schedule tonnages and unit operating hours of the mining fleet. The costs are as follows:

- Cdn$1.245 million during Years 3 and 4
- Cdn$0.569 million during Year 5
- Cdn$1.917 million during Year 6
- Cdn$0.084 million during Years 5 to 10.

Table 21.3 Sustaining Mine Equipment Capital Cost

Equipment Fleet	Size	Model	Qty	Unit Cost (Cdn$)	Total (Cdn$)
Haul Trucks	90.9 t	HD785-7	2	1,245,000	2,290,000
Shovels	11 m^3	PC2000-8	0		
Loader	1.5 m^3	WA900-3	0		
Drills	251 mm	Sandvik D55SP	0		
Track Dozers		CAT D9T	1	1,333,185	1,333,185
Graders	164 kW	CAT 16M	0		
Water Truck	50 ton	CAT 773 EWT	0		
Backhoe Excavator	1.5 m^3	CAT 329 DL	0		
Utility Loader/Tire Handler			0		
Boom Truck	National Crane 20 ton	600E2	0		
Welding/Service Truck			1	100,800	100,800
Fuel/Lube Truck	13,000 L		1	132,300	132,300
Flatdeck/Float	20 ton		0		
Crane	25 ton	Grove YB7725	0		
Light Vehicles (Pick-up Trucks)	0.5 ton crew cab		18	42,000	756,000
Crew Buses	40 passenger		2	105,000	210,000
Portable Light Towers	6 kW		6	15,750	94,500
GEN-SET (Site Backup)	90 kW		2	24,127	48,254
Pumps	Submersible (90 kW)		2	73,500	147,000
Total Open Pit Equipment					**5,312,039**

21.1.5 PROCESSING

Plant capital costs for processing 5,000 t/d ore from both the Lundberg and Engine House deposits are estimated to be Cdn$56.1 million (Table 21.4). Costs for the equipment are based on a typical equipment list considering the proposed flowsheet. Costs for concrete and electrical are calculated based on comparison with other similar Tetra Tech projects, and reference to Tetra Tech's database for an equivalent production rate of 5,000 t/d.

Table 21.4 Processing Capital Costs

No.	Area	2011 Cdn$
1.	Mill Building/Equipment	29,036,991
2.	Installation Labour	12,119,940
3.	Concrete	1,631,080
4.	Piping	5,474,280
5.	Structural Steel	1,737,340
6.	Instrumentation/Electrical	3,543,953

table continues…

No.	Area	2011 Cdn$
7.	Coatings/Sealants/Insulation	941,094
8.	Operating Capital	1,600,000
Total Processing Capital Cost		**56,084,677**

21.2 Operating Costs

21.2.1 *Mining*

The total mine operating cost is estimated at Cdn$159,476,745 which equates to Cdn$2.27/t of material mined or Cdn$9.23/t of ore milled. The breakdown of the operating costs is shown in Table 21.5 and Table 21.6.

The mine operating costs includes mine operations supervision and light duty vehicle operating costs outlined in Table 21.2 and Table 21.3.

The unit operation cost is estimated to be Cdn$2.27. Manpower salaries, diesel price, and explosive price are obtained from vendors and the in-house Tetra Tech cost database. Table 21.7 presents a summary of the open pit operating costs for a nominal mill throughput 5,000 t/d, with an average strip ratio of 3.06 t/t (waste/ore) during the 10 years of mine life.

Table 21.5 Mine Operating Cost By Expense (x $1,000) by Period

Item	1	2	3	4	5	6	7	8	9	10	Total
Fuel	$3,456	$3,479	$3,854	$4,390	$4,335	$4,331	$4,265	$3,408	$2,368	$1,913	**$35,799**
Lube	$147	$147	$160	$177	$175	$175	$165	$136	$100	$82	**$1,465**
Tires	$1,923	$1,977	$2,349	$2,922	$2,883	$2,924	$3,008	$2,248	$1,476	$1,145	**$22,854**
Equipment Maintenance	$1,530	$1,535	$1,710	$1,952	$1,924	$1,915	$1,906	$1,505	$1,009	$807	**$15,793**
Drilling	$65	$62	$76	$90	$86	$81	$75	$51	$29	$18	**$634**
Explosives	$2,736	$2,649	$3,036	$3,448	$3,341	$3,180	$3,026	$2,313	$1,670	$1,259	**$26,658**
Dewatering	$25	$25	$25	$25	$25	$25	$25	$25	$25	$25	**$250**
Manpower	$5,699	$5,774	$5,849	$6,074	$6,074	$6,074	$6,149	$5,262	$3,842	$3,225	**$54,024**
Miscellaneous	$200	$200	$200	$200	$200	$200	$200	$200	$200	$200	**$2,000**
Total	**$15,781**	**$15,848**	**$17,259**	**$19,279**	**$19,044**	**$18,905**	**$18,819**	**$15,147**	**$10,719**	**$8,676**	**$159,477**
$/tonne Mined	**$2.18**	**$2.29**	**$2.06**	**$1.94**	**$2.00**	**$2.12**	**$2.26**	**$2.68**	**$3.32**	**$4.19**	**$2.27**
$/tonne Ore	**$9.02**	**$9.06**	**$9.86**	**$11.02**	**$10.88**	**$10.80**	**$10.75**	**$8.66**	**$6.13**	**$5.68**	**$9.23**

Table 21.6 Mine Operating Cost By Function (x $1,000) by Period

Item	1	2	3	4	5	6	7	8	9	10	Total
Loading	$1,046	$1,022	$1,165	$1,316	$1,277	$1,218	$1,161	$899	$581	$443	**$10,128**
Hauling	$3,661	$3,776	$4,525	$5,694	$5,622	$5,728	$5,928	$4,461	$2,898	$2,240	**$44,533**
Drilling	$700	$685	$776	$872	$847	$809	$773	$606	$456	$357	**$6,882**
Blasting	$2,879	$2,792	$3,180	$3,592	$3,484	$3,323	$3,169	$2,456	$1,813	$1,384	**$28,073**
Roads & Dumps	$2,356	$2,356	$2,356	$2,356	$2,356	$2,356	$2,356	$1,895	$1,325	$1,158	**$20,871**
Mine General	$1,159	$1,159	$1,159	$1,159	$1,159	$1,159	$1,031	$1,031	$648	$566	**$10,228**
Miscellaneous	$200	$200	$200	$200	$200	$200	$200	$200	$200	$200	**$2,000**
Ancillary Equipment	$3,780	$3,857	$3,899	$4,089	$4,098	$4,112	$4,200	$3,599	$2,798	$2,327	**$36,760**
Total	**$15,781**	**$15,848**	**$17,259**	**$19,279**	**$19,044**	**$18,905**	**$18,819**	**$15,147**	**$10,719**	**$8,676**	**$159,477**
$/tonne Mined	**$2.18**	**$2.29**	**$2.06**	**$1.94**	**$2.00**	**$2.12**	**$2.26**	**$2.68**	**$3.32**	**$4.19**	**$2.27**
$/tonne Ore	**$9.02**	**$9.06**	**$9.86**	**$11.02**	**$10.88**	**$10.80**	**$10.75**	**$8.66**	**$6.13**	**$5.68**	**$9.23**

Table 21.7 Summary of Open Pit Operating Costs (x $1,000)

Item	Unit	Cost	%
Loading	Cdn$	10,128	6.35
Hauling	Cdn$	44,533	27.92
Drilling	Cdn$	6,882	4.32
Blasting	Cdn$	28,073	17.60
Roads & Dumps	Cdn$	20,871	13.09
Mine General	Cdn$	10,228	6.41
Miscellaneous	Cdn$	2,000	1.25
Ancillary Equipment	Cdn$	36,760	23.05
Total Mining Cost	**Cdn$**	**159,477**	**100**
Loading	Cdn$/t	0.14	6.35
Hauling	Cdn$/t	0.63	27.92
Drilling	Cdn$/t	0.10	4.32
Blasting	Cdn$/t	0.40	17.60
Roads & Dumps	Cdn$/t	0.30	13.09
Mine General	Cdn$/t	0.15	6.41
Miscellaneous	Cdn$/t	0.03	1.25
Ancillary Equipment	Cdn$/t	0.52	23.05
Unit Mining Cost	**Cdn$/t**	**2.27**	**100**

Open Pit Operations Manpower

The mine will be operated for 24 hours per day, 350 days per year to incorporate 15 public holidays. Total available working time would be 8,400 hours per year (no scheduled operations downtime).

Since the town of Buchans is not a remote site, labourers will be working a schedule of four days on and four days off; mine staff will be working five days on and two days off. Table 21.8 lists the proposed manpower for the open pit operation and maintenance during peak operations (Year 4).

The labour details (Table 21.8) are prepared and estimated by Tetra Tech for the purposes of this study, based on the required personnel for a 5,000 t/d open pit mining operation. Rates from other similar Tetra Tech projects in the area were considered for the purposes of the PEA mining plan. An additional 35% (30% + 5%) is considered to be salaries for health benefits, pension, overtime, training, etc.

Table 21.8 Operations Manpower Example (Year 4)

Position	No.	Annual Cost per Person	Total Annual Cost (Cdn$)	Cost (Cdn$/t mined)	Cost (Cdn$/t ore)
Mine Operations					
Mine General Foreman	0	100,000	0	0.00	0.00
Senior Mine Foreman	2	95,000	190,000	0.02	0.11
Mine Foreman	4	80,000	320,000	0.03	0.18
Drill & Blast Foreman	1	85,000	85,000	0.01	0.05
Drill Operator	3	60,000	180,000	0.02	0.10
Drill Helper	0	50,000	0	0.00	0.00
Blaster	1	60,000	60,000	0.01	0.03
Blaster Helper	1	45,000	45,000	0.00	0.03
Shovel Operator	3	60,000	180,000	0.02	0.10
Loader Operator	1	60,000	60,000	0.01	0.03
Haul-Truck Operator	14	55,000	770,000	0.08	0.44
Dozer Operator	5	55,000	275,000	0.03	0.16
Grader Operator	4	55,000	220,000	0.02	0.13
Backhoe Excavator Operator	1	55,000	55,000	0.01	0.03
Water Truck Operator	1	55,000	55,000	0.01	0.03
Fuel/Lube Truck Operator	2	55,000	110,000	0.01	0.06
Shift Labourer	5	45,000	225,000	0.02	0.13
Subtotal	**48**		**2,830,000**	**0.28**	**1.62**
Mine Maintenance					
Maintenance Forman	4	80,000	320,000	0.03	0.18
Maintenance Planner	2	65,000	130,000	0.01	0.07
Heavy Duty Mechanic	8	70,000	560,000	0.06	0.32
Welder	4	70,000	280,000	0.03	0.16
Washbay	2	40,000	80,000	0.01	0.05
Light Duty Mechanic	2	55,000	110,000	0.01	0.06
Electrician	2	70,000	140,000	0.01	0.08
Subtotal	**24**		**1,620,000**	**0.16**	**0.93**
Benefits: Health Pension: 30%			1,335,000.0	0.13	0.76
Addition: Overtime, Training: 5%			289,250.00	0.03	0.17
Total Labour	**72**		**6,074,250**	**0.61**	**3.47**

Note: All technical services, Mine Manager, and administration staff are not covered under the outlined mine operating costs.

21.2.2 PROCESSING

Operating costs consist of labour, supplies and power. They are estimated to be Cdn$21.9 million per annum or Cdn$12.53 per tonne ore (Table 21.9).

Labour costs are based on required manpower and rates to operate the plant. The suggested scheme employs 62 people; 48 for the plant operations and 14 for mill maintenance. The total annual cost is Cdn$4.5 million and the cost per tonne ore is Cdn$2.55.

The total annual costs for the supplies are Cdn$11.3 million and the cost per tonne ore is Cdn$6.48 in total and includes the required reagent types and consumptions, as well as grinding steel, mill liners, and maintenance supplies, etc.

The power cost is based on a Cdn$0.0752/kWh rate and is supported by the Newfoundland Power schedule of rates which outlines a monthly charge, demand charge, and a two tiered energy consumption charge. For the plant, this is determined to be Cdn$6.1 million per annum or Cdn$3.50 per tonne ore with a high range estimate total of 8,460 kW installed.

Table 21.9 Plant Operating Costs

Operating Cost Area	%	Total Annual Cost (Cdn$)	Cost (Cdn$/t)
Labour	20.37	4,465,250	2.55
Reagents and Consumables	51.729	11,337,834	6.48
Power and Utilities	27.91	6,117,035	3.50
Total Plant Operating Cost	100.00	21,920,119	12.53

REAGENTS

The reagent and supply types and consumptions are based on the results from the recently completed lock cycle test carried out by SGS (Table 21.10).

Table 21.10 Reagents and Supplies

Reagents and Supplies	Consumption (t/a)	Consumption (kg/t ore)	Total Annual Cost (Cdn$)	Cost (Cdn$/t)
Collectors	113	0.06	543,621	0.31
Frothers	31	0.02	170,543	0.10
Activators and Depressants	1,261	0.72	4,424,7660	2.53
pH Regulators and Modifiers	3,465	2.16	1,144,631	0.65
Other Reagents	613	0.60	1,451,363	0.83
Total Reagents	5483	3.56	7,734,923	4.42
Grinding Steel and Mill Liners	409	1.17	388,161	0.22
Maintenance Supplies	105	0.06	519,750	1.54
Sundry Items			2,695,000	1.54
Total Consumables	**514**	**1.23**	**3.602,911**	**2.06**
Total Reagents and Supplies	**5,996**		**11,337,834**	**6.48**

LABOUR

The labour details are prepared and estimated by Tetra Tech for the purpose of this study based on the required personnel for a 5,000 t/d plant considering the proposed process flowsheet and rates from other similar Tetra Tech projects in the area. An additional 30% (25%+5%) is considered to be the salaries for health benefits, pension, overtime, and training, etc.

Table 21.11 Process Plant Labour

Position	Workers/day	Annual Cost per Person (Cdn$)	Total Annual Cost (Cdn$)	Cost (Cdn$/t)	Annual Cost plus Benefits
Hourly Personnel Requirements					
Control Room Operator	4	70,000	280,000	0.16	91,000
Crusher Operator	2	60,000	120,000	0.07	78,000
Grinding Operator	4	60,000	240,000	0.14	78,000
Flotation Operator	4	60,000	240,000	0.14	78,000
Reagents Operator	1	60,000	60,000	0.03	78,000
Dewatering and Loadout Operator	2	60,000	120,000	0.07	78,000
Filter Operator	4	60,000	240,000	0.14	78,000
Dryer Operator	4	60,000	240,000	0.14	78,000
Assayer	2	60,000	120,000	0.07	78,000
Sampler	2	55,000	110,000	0.06	71,500
Labourer/Helper	8	50,000	400,000	0.23	65,000
Mechanic	7	65,000	455,000	0.26	84,500
Electrician/Instrumentation	4	65,000	260,000	0.15	84,500
Subtotal	**48**		**2,885,000**	**1.65**	
Salaried Personnel Requirements					
Mill Superintendent	1	110,000	110,000	0.06	143,000
General Foreman	0	85,000	0	0.00	110,500
Maintenance Foreman	1	75,000	75,000	0.04	97,500
Plant Foreman	4	75,000	300,000	0.17	97,500
Senior Metallurgist	0	0	0	0.00	0
Metallurgist	2	80,000	160,000	0.09	104,000
Process Technician	2	70,000	140,000	0.08	91,000
Instrument Technician	2	70,000	140,000	0.08	91,000
Process Foreman	2	75,000	150,000	0.09	97,500
Subtotal	**14**		**485,000**	**0.28**	

POWER

The power cost of Cdn$0.0752/kWh is used for the purposes of the Project. The installed kW are based on the requirements for a 5,000 t/d plant producing three concentrates. The data was collected from other similar Tetra Tech projects. The costs for diesel fuel of Cdn$1/L and fuel oil of Cdn$1/L are fixed for the purposes of this Project.

Table 21.12 Power Costs

Power Area	Energy Charge (Cdn$/kWh or L)	Installed kW/t/a	Total Annual Cost (Cdn$)	Cost (Cdn$/t)
Electricity	0.07521	8460.2	5,115,785	2.92
Diesel Fuel	1	10,050	10,050	0.01
Fuel Oil	1	991,200	991,200	0.57
Total Power			**6,117,035**	**3.50**

GENERAL AND ADMINISTRATIVE COSTS (G&A)

G&A costs represents cost incurred by the mine that is not directly associated with mine production or processing. Employee roles that are included in the G&A costs include management ,financial, engineering ,geology support, purchasing, health, safety and the environment (HSE), and administration. The G&A costs have been calculated to be $3,557,275 annually or $2.03/t of ore mined. The breakdown of the G&A costs is shown in Table 21.13.

Table 21.13 G&A Costs

Cost Centre	Description	Per Year
Operational Costs	Salaries	$1,964,442
	Overhead (30% of salaries)	$589,333
	Equipment Rental	$25,000
	Subtotal	$2,578,775
Office Costs	Computer Supplies & Software	$24,000
	Office and Warehouse Supplies	$20,000
	Postage, Courier & Light Freight	$10,000
	Communications/Telefax	$60,000
	Subtotal	$114,000
Legal/Professional Costs	Insurance	$110,000
	Permits & Licences	$4,000
	Bank Charges	$12,000
	Professional Fees - Accounting	$10,000
	Professional Fees - Legal	$6,000
	Professional Fees - General	$10,000
	Subtotal	$152,000

Mine Site Costs	Recruitment	$5,250
	Security	$15,750
	Safety, Clothing and Training	$52,500
	First Aid	$10,500
	Public Relations/Community Support	$10,500
	Environmental	$10,500
	Travel & Accommodation - Business	$31,500
	Power	$72,000
	Freight	$504,000
	Subtotal	$712,500
Total General & Administrative Costs		**$3,557,275**
$/t		**$2.03**

22.0 ECONOMIC ANALYSIS

22.1 METAL PRICING

Currently, Tetra Tech's metal prices are set quarterly. The prices are based on the Consensus Economic Energy and Metal Forecast Group (EMCF) of London. This group provides quarterly long term forecast (5-10 years) for a variety of metals based on a selection of analysts and the EMCF averages the 20 or 30 projections in a single average (consensus) forecast. Tetra Tech considers these forecasts to be independent, transparent, consistent and generally aligned with the timelines for potential construction and operation of mine opportunities considered in most economic studies.

To set the metal prices, Tetra Tech uses the average of three quarterly reports. The reason for averaging three periods is to avoid any single outlier forecast, and to smooth any large fluctuation between quarterly forecasted prices.

Table 22.1 below details metal price used in the economic analysis for the Lundberg/Engine House PEA.

Table 22.1 Metal Prices

Metal	Metal Price	Units
Zinc	$1.22	US$/lb
Copper	$3.62	US$/lb
Lead	$1.10	US$/lb
Silver	$22.74	US$/oz

22.2 FINANCIAL ANALYSIS

The financial analysis considered a total of 17.3 million tonnes of resource. This represents the Inferred Resource from the Lundberg and Engine House deposits, as the geology of the two deposits are defined in Section 14 of this report. The financial analysis is based on the open pit mine design and schedule as defined in Section 16. Revenue contribution is calculated from NSR for three concentrates. Silver contributes to the NSR value for the lead concentrate and the copper concentrate. The NSR values for these concentrates are:

- Zn concentrate – US$845/dmt
- Cu concentrate – US$1699/dmt
- Pb concentrate – US$1574/dmt

A detailed report of the financial analysis and cash flow by year is provided in Appendix I.

Using the NSR values above and an exchange rate of $1.012 Cdn$:US$, and an operating cost of Cdn$26.00 per tonne of ore, the pre-tax IRR for the project has been calculated at 43.94%.

Overall, the total capital and operating expenditures will be Cdn$152 million and Cdn$411 million, respectively. A 10% salvage value has been assumed against the estimated capital costs for the processing plant and equipment, mining equipment and non process buildings, the salvage value is estimated to be Cdn$8.2 million and is recognized in the last year of the life-of-mine (LOM) analysis. Additionally, the total revenue before taxes from concentrate sales will be Cdn$915 million during a LOM of 10 years.

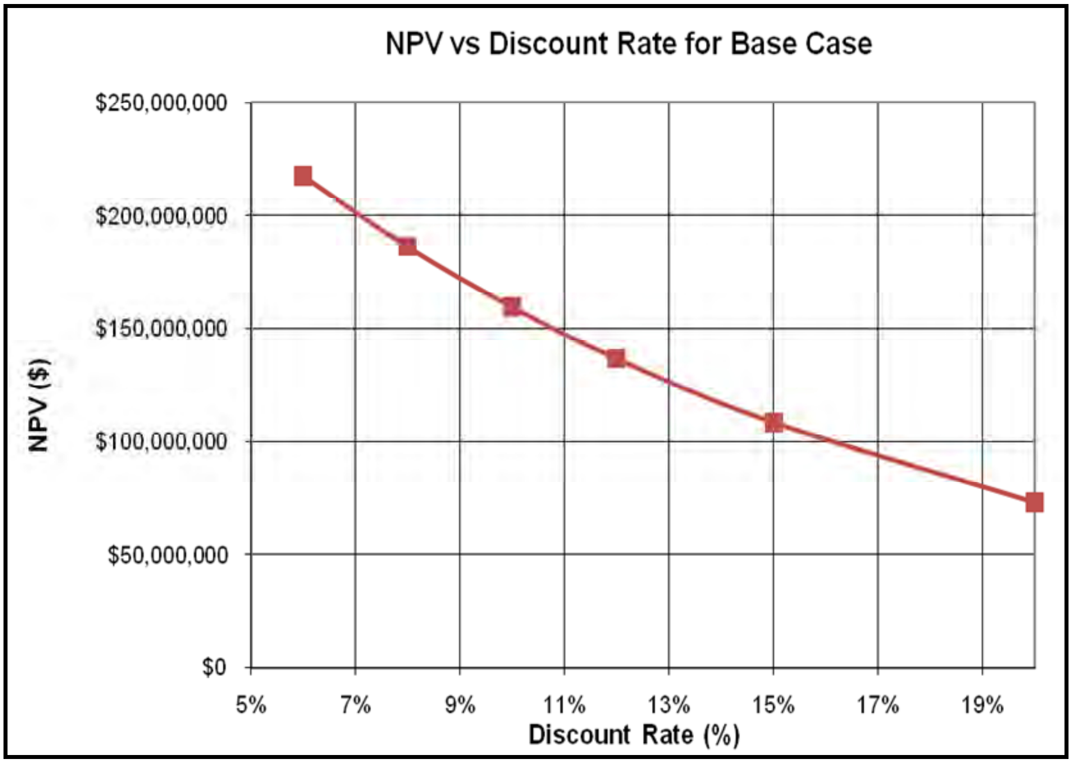
Table 22.2 illustrates the pre-tax NPV for the project at variable discount rates. As well the IRR is shown as 43.94%.

Table 22.2 Pre-tax Net Present Value and Internal Rate of Return

Item	Amount
Pre-tax & Pre-finance NPV @ 6%	$217,778,267
Pre-tax & Pre-finance NPV @ 8%	$186,377,690
Pre-tax & Pre-finance NPV @ 10%	$159,682,329
Pre-tax & Pre-finance NPV @ 12%	$136,873,842
Pre-tax & Pre-finance NPV @ 15%	$108,545,780
Pre-tax & Pre-finance NPV @ 20%	$73,093,623
Project IRR	43.94%

Figure 22.1 shows the NPV for the various discount rates for the base case scenario, as defined by the above metal prices, NSR and currency exchange.

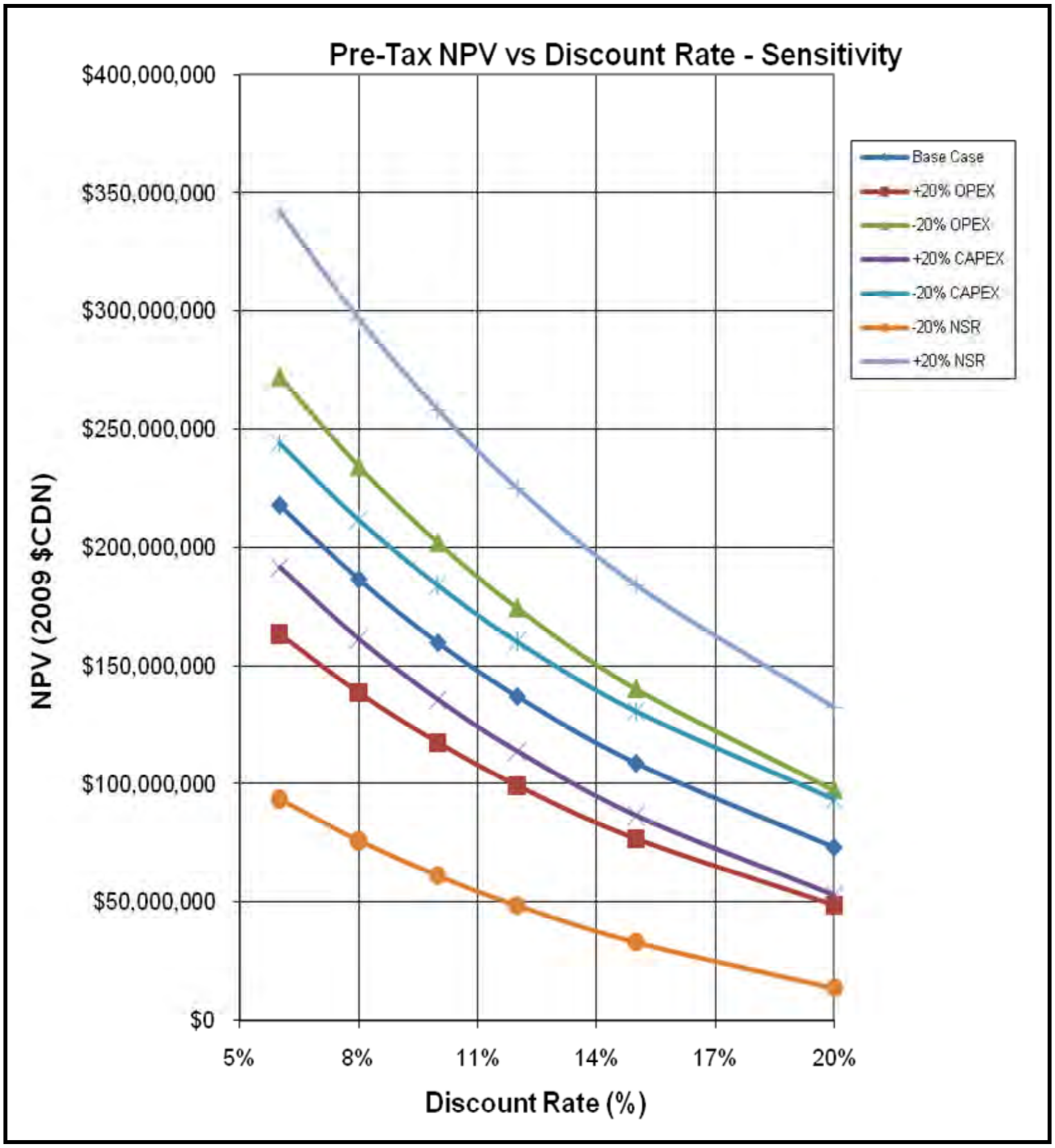
Figure 22.1 NPV vs. Discount Rate for Base Case



22.3 Sensitivity Analysis

Several parameters were varied by ±20% to examine the sensitivity of the net present value of the project as the discount rate changes. The capital and operating costs, as well as the NSR, were individually increased and reduced by 20% off the base case, and the pre-tax results plotted as shown in Figure 22.2.

Figure 22.2 Capital and Operating Cost, and NSR Sensitivity



The analysis shows that the project is most sensitive to variations in the NSR. Tetra Tech ran several sensitivity analyses to determine the effect on key financial statistics if the following basic parameters change:

- the operating costs increased and decreased by 20%
- the capital costs increased and decreased by 20%
- the NSR increased and decreased by 20%
- the NSR for zinc only increased and decreased by 20%.

For ease of reference, the results are summarized in Table 22.3 and Table 22.4. In addition, the IRR Sensitivity is shown in Figure 22.3.

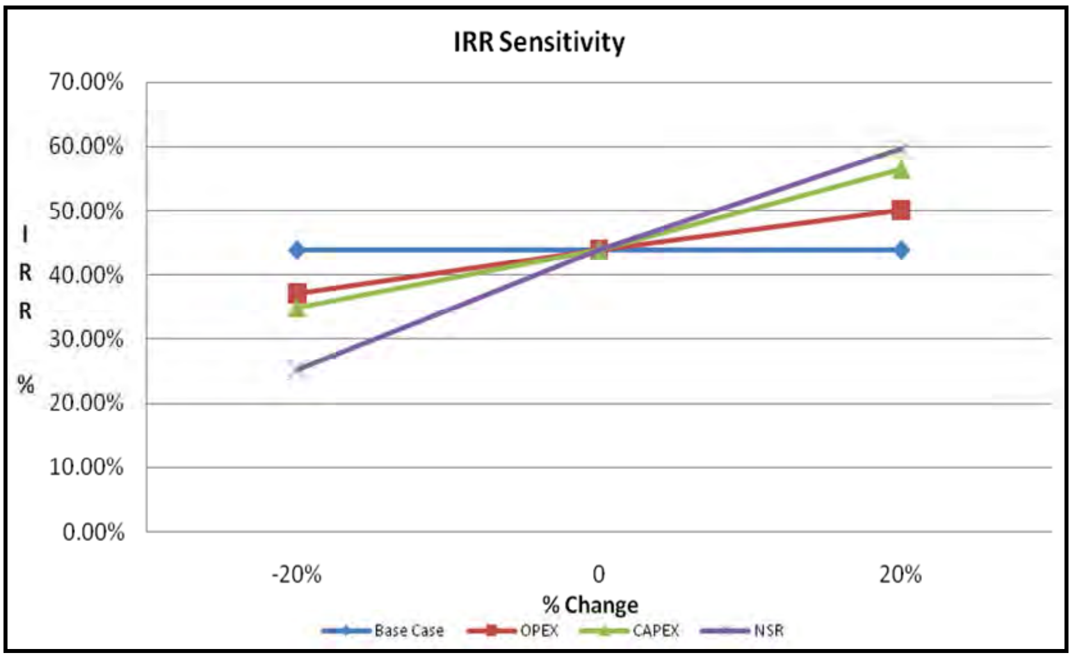
Table 22.3 Operating and Capital Cost Sensitivity

	Discount Rate	NPV	NPV Difference	IRR
Base Case				
	6%	$217,778,267	-	43.94%
	8%	$186,377,690	-	
	10%	$159,682,329	-	
	12%	$136,873,842	-	
	15%	$108,545,780	-	
	20%	$73,093,623	-	
Operating Cost				
Increase 20%	6%	$163,446,326	($54,331,941)	37.14%
	8%	$138,546,319	($47,831,371)	
	10%	$117,369,256	($42,313,073)	
	12%	$99,270,313	($37,603,529)	
	15%	$76,787,032	($31,758,748)	
	20%	$48,653,018	($24,440,604)	
Decrease 20%	6%	$272,110,207	$54,331,941	50.10%
	8%	$234,209,062	$47,831,371	
	10%	$201,995,401	$42,313,073	
	12%	$174,477,371	$37,603,529	
	15%	$140,304,527	$31,758,748	
	20%	$97,534,227	$24,440,604	
Capital Cost				
Increase 20%	6%	$191,461,612	($26,316,655)	34.94%
	8%	$161,169,054	($25,208,636)	
	10%	$135,483,458	($24,198,871)	
	12%	$113,599,529	($23,274,312)	
	15%	$86,521,837	($22,023,943)	
	20%	$52,858,471	($20,235,152)	
Decrease 20%	6%	$244,094,921	$26,316,655	56.54%
	8%	$211,586,327	$25,208,636	
	10%	$183,881,200	$24,198,871	
	12%	$160,148,154	$23,274,312	
	15%	$130,569,723	$22,023,943	
	20%	$93,328,775	$20,235,152	

Table 22.4 NSR Sensitivity

	Discount Rate	NPV	NPV Difference	IRR
Decrease 20%	6%	$93,574,018	($124,204,249)	25.17%
	8%	$76,062,145	($110,315,546)	
	10%	$61,233,919	($98,448,409)	
	12%	$48,621,232	($88,252,610)	
	15%	$33,053,933	($75,491,847)	
	20%	$13,799,142	($59,294,481)	
Increase 20%	6%	$341,982,515	$124,204,249	59.70%
	8%	$296,693,236	$110,315,546	
	10%	$258,130,738	$98,448,409	
	12%	$225,126,451	$88,252,610	
	15%	$184,037,627	$75,491,847	
	20%	$132,388,104	$59,294,481	

Figure 22.3 IRR Sensitivity



Based on the sensitivity analysis results, it is clear that the project is most sensitive to variation of NSR value, much less sensitive on capital costs and least sensitive on operating cost. Moving forward, an attempt should be made to refine and improve the smelter terms, through discussions with Doe Run, Xstrata and any local smelters, to decrease the shipping costs to a minimum and improve the smelter term conditions. This could lead to improved project financials.

23.0 ADJACENT PROPERTIES

There are no adjacent properties, as defined by NI 43-101, that are pertinent to this report.

24.0 OTHER RELEVANT DATA AND INFORMATION

The following section is taken from Webster and Barr (2008).

A large amount of historical data relevant to the Buchans area properties exists but is not included in this report. This information is available in government files or in various other publications prepared by previous and present owners, external consultants, contractors and both government and academic researchers. In addition, a number of the properties discussed in this report are under option to third party exploration companies who may have completed exploration work required under the terms of the option agrees.

Due to the historical nature and multiple sources of much of the data being used in this resource, a number of corrections, assumptions, and adjustments have been made in order to merge the data into a useable database. The following is a list of such manipulations:

- *Historical drill hole collar elevations have been converted to metric elevations relative to sea level, and have had 8 m added to the values used historically in order to match them with modern survey datum surveyed by Red Indian Surveys of Grand Falls, NL.*
- *The Lucky Strike Glory hole was modeled by Eagle mapping from stereographic triangulation of historic aerial photography. The elevation of the Glory hole and surrounding surface DEM elevation has been supplied in UTM (NAD83) co-ordinates. The elevation datum for these data has been increased by 3 m to match them with the modern survey datum surveyed by Red Indian Surveys of Grand Falls, NL.*
- *Where historical drill logs listed "Tr" as an assay value, it was assumed that this was a trace amount above detection, and was given a numerical value of 0.001 in the database in their respective unit of measure.*
- *Where historical drill logs listed "NIL" as an assay value, it was assumed that this was an amount below detection, and was given a numerical value of 0.0001 or 0 (zero) in the database in their respective unit of measure.*
- *In some cases, drill hole location in association with lithological descriptions did not correspond with the reported assay values, and therefore it was decided that these holes be discarded from the database. These holes include 48, 49, 50 and 384.*
- *In some cases, no assay information could be located for drill holes and these holes were left in the database as they would have no impact on the*

grade interpolation methods used in the mineral resource calculation. In cases where lithological descriptions were found for these holes, this information was used in correlating geological boundaries.

- *Underground workings have been reproduced to be as representative as the source information would allow. Assimilation of historic 2D plan view sections, 2D cross-sections, and known drill hole elevations has provided the basis for the model generation. In most cases, 2D plan view maps were the only source for the lateral extent of the development, with stoping information being extrapolated from rare cross-sectional views. It is acknowledged that the depiction of the underground workings is not precise, but where uncertainties exist, a liberal approach was taken to ensure that the removed volumes were not underestimated.*

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 GEOLOGY

BMC holds 100% mineral interest in the Buchans claims group which is host to numerous base metal prospects and to the Lundberg and Engine House deposits. Exploration within this claim group has targeted polymetallic (Zn-Pb-Cu-Ag-Au-Ba) VMS style mineral environments.

The Lundberg and Engine House deposits exist in the footwall volcanics as a base metal sulphide enriched stockwork system that is in conformable contact with the overlying Buchans River Formation. The Lundberg deposit underlies the previously mined Lucky Strike glory hole, and the Engine House deposit surrounds the previously mined underground Engine House orebody. The mineralized stockwork system which comprises the Lundberg deposit was previously delineated by ASARCO in 1974 as a low-grade mineral resource, which was never subsequently mined. Review and validation of historic exploration data, incorporation of results from previous core drilling and twinning of selected holes showed that historic data were of acceptable quality for resource estimation purposes. Recent drilling by BMC was completed under the supervision of Mercator and BMC, and delineated the mineralized the base metal stockwork zone. On this basis, a fully constrained 3D block model for the deposit was developed using Surpac© 6.02 deposit modeling software. Model blocks measured 5 m x 5 m x 5 m with sub-blocking at 2.5 m x 2.5 m x 2.5 m within the Lundberg solid, and 2.5 m x 2.5 m x 2.5 m with sub-blocking at 1.25 m x 1.25 m x 1.25 m for the Engine House solid.

Grade interpolation was accomplished using ID^2 methodology. This estimate updates an earlier released Inferred Resource having an effective date of September 15, 2008 (PR#17-08 Sept 17, 2008) and incorporates more complete historic precious metal assay data compiled from historic drilling and assays, resulting in a nominal increase in the precious metal contents. The model was fully constrained within a deposit solid based on a minimum metre 1% zinc cut-off resulting in the definition of an Inferred Mineral Resource that is compliant with NI 43-101 and CIM reporting standards. Table 25.1 and Table 25.2 summarize the Inferred Mineral Resource Estimate.

Table 25.1 Lundberg Inferred Resource Estimate - Zn % Threshold - Nov 3 2008

Zn% Threshold	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	BaSO4%	Percentage of Tonnage within 100 m of Surface
1.00	15,690,000	1.96	0.83	0.38	3.17	6.57	0.08	2.36	61.79%
1.50	9,300,000	2.46	1.03	0.43	3.92	8.26	0.10	2.84	66.40%
2.00	5,340,000	3.02	1.25	0.49	4.76	10.27	0.12	3.47	70.62%
2.50	3,170,000	3.56	1.46	0.53	5.55	12.28	0.14	4.65	72.83%
3.00	1,880,000	4.13	1.66	0.57	6.36	14.32	0.14	6.20	75.68%
3.50	1,090,000	4.79	1.93	0.62	7.34	16.46	0.15	8.64	81.35%

Table 25.2 Engine House Inferred Resource Estimate - Zn % Threshold - Nov 3 2008

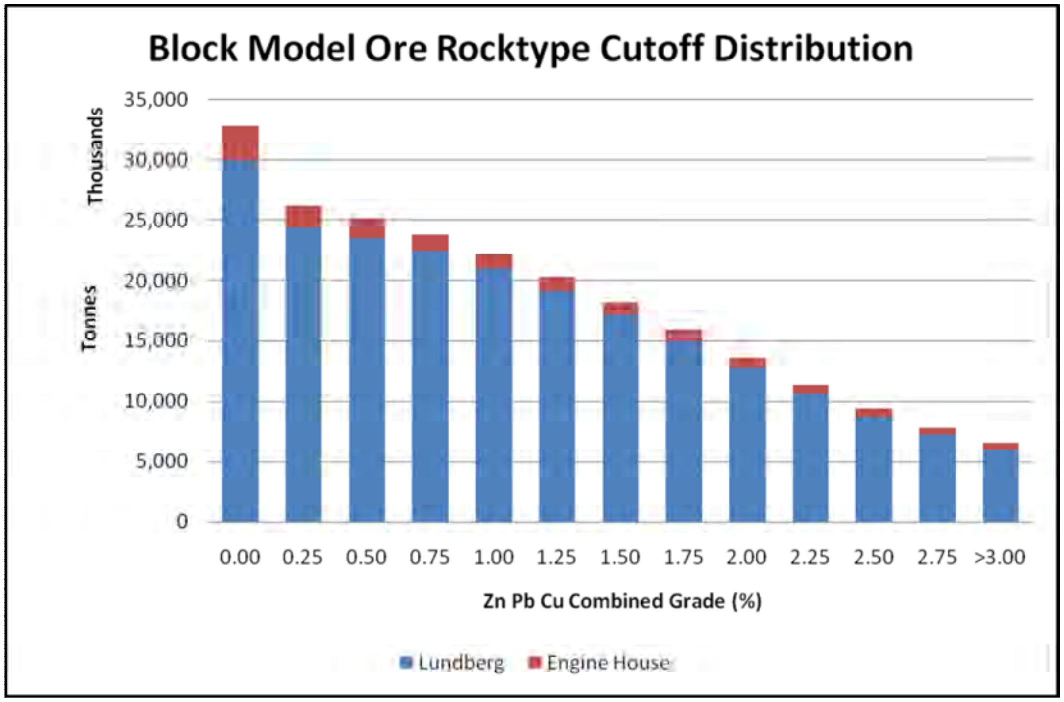
Zn% Threshold	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)	Au (g/t)	BaSO4%	Percentage of Tonnage within 100 m of Surface
1.00	890,000	2.37	0.95	0.96	4.28	11.29	0.15	4.40	58.73%
1.50	600,000	2.89	1.10	1.05	5.04	12.17	0.16	4.87	60.56%
2.00	370,000	3.62	1.27	0.97	5.86	12.71	0.19	5.51	60.40%
2.50	240,000	4.35	1.41	0.94	6.70	12.34	0.22	5.56	52.04%
3.00	190,000	4.77	1.50	0.93	7.20	12.32	0.23	5.63	56.35%
3.50	140,000	5.28	1.56	0.91	7.75	12.33	0.23	5.60	56.28%

For purposes of designing the open pit, Mercator Geological Services Ltd had to modify their original NI 43-101 technical compliant resource model, (see section 16.2) such that resource blocks were adjusted to be a 5 m x 5 m x 5 m consistent size throughout the resource model. The modified block model identified an Inferred Resource at a combined Zn-Pb-Cu cut-off of 1% of 22.21 million tonnes with average grades of 1.62% Zn, 0.69% Pb, 0.38% Cu, and 5.81 g/t Ag. See Table 25.3 and Figure 25.1 for the resource distribution for the re-blocked block model.

Table 25.3 Lundberg and Engine House Re-Blocked Inferred Resource Estimate – 1% Zn + Pb + Cu Combined Threshold

Threshold (Zn%+Pb%+Cu%)	Tonnes Rounded	Zn%	Pb%	Cu%	Combined Zn+Pb+Cu%	Ag (g/t)
1.00	22,210,000	1.62	0.69	0.38	2.69	5.81

Figure 25.1 Block Model Ore Rocktype Cut-off Distribution



25.2 Mineral Processing and Metallurgical Testing

Based on the metallurgical and processing work performed by SGS on Met#4 – LCT 2 and LCT 3, these are the following major conclusions:

- The preliminary data from the metallurgical test program suggest concentrate grades as follows: copper concentrate – 24.1%, lead concentrate – 73.9%, zinc concentrate – 53%. Silver was identified in significant amounts in the lead concentrate with 359.5 g/t of silver.
- Metallurgical testing has also indicated that the Buchans deposit could yield total recoveries of 89.3% for zinc, 66.2% for copper, 78.7% for lead, and 27.8% (Pb 4th cleaner concentrate) for silver.
- The mineralogical studies conducted for the Buchans Lunberg and Engine House ore samples show that Iron was distributed in the primary sulphides including chalcopyrite, sphalerite, galena and pyrite. Practically no iron was identified in any of the non-sulphides. No deleterious components, in excessive amounts, were identified. The indication from this investigation suggests that no smelter penalties can be expected.
- The mill feed is determined to be 5,000 t/d based on the maximum capacity of the mining production from the open pit.
- Three concentrate products will be produced out of the mill:
  - copper concentrate containing payable copper and silver

- zinc concentrate contains payable zinc and silver
- lead concentrate containing payable lead and silver.

The plant capital costs are estimated to be Cdn$56.1 million. The operating costs are estimated to be Cdn$21.9 million per annum or Cdn$12.53 per tonne ore.

25.3 Mining Operations

Since the required geotechnical data was not available for determining the pit slope angle, Tetra Tech utilized an overall pit slope angle of 45°, based on conservative estimates from previous experience.

The base case pit contains17.28 Mt of resource (potential ore). The average grades over the life of mine will be 1.63% Zn, 0.69% Pb, 0.40% Cu, 5.96 g/t Ag, 0.07 g/t Au and 1.24% Ba for a combined base metal grade of 2.72% (Zn-Pb-Cu). The overall stripping ratio is 3.06 t/t (waste/ore), which has considered 5% of mining dilution and 95% of mining recovery. A total of 52.93 Mt of waste material will be moved over the mine life of 10 years.

It is proposed that the operation will be carried out with an equipment fleet comprising a single, 251 mm diameter rotary blast hole drill rig for mineable resource (ore) and waste, an 11 m^3 (bucket capacity) hydraulic face shovel with a fleet of 91 t haul trucks. These will be supplemented with support equipment of loader, grader, dozers, and a backhoe excavator, etc.

Due to the ore being close to surface, little waste is mined to enable access to ore and the full 5,000 tpd production is achievable the first year of mining activity.

As this is a PEA Study, with the primary focus of providing an economic and financial justification for any further investment. The aim of the pit optimization and pit design is to provide an input resource for inclusion into the financial model.

25.4 Economic Analysis

The preliminary assessment is favourable with a pre-tax IRR of 43.9% and a NPV of Cdn$186.4 million at an 8% discount rate. The IRR will drop following tax considerations. To achieve this return it will require a capital investment of Cdn$152 million.

Payback will take approximately 1.4 years after start of mine production.

This study is the first in a series of development studies for the project that assesses the viability of developing the Lundberg and Engine House deposits and mill construction. The project considers utilizing the existing infrastructure. This study quantifies the project's cost parameters and identifies the additional exploration and detailed engineering work required to ultimately define the optimal scale of the operation for a completed Feasibility Study.

Future studies on mill and TMF sites will look at further definition of the resources based on the exploration potential and finding ways to reduce capital, mine and process operating costs.

Additional investigation is required to refine and improve the smelter terms, through discussions with local smelters. The aim is to better define and/or decrease the shipping costs and negotiate favourable smelter terms.

26.0 RECOMMENDATIONS

26.1 OVERVIEW

A detailed review was undertaken to estimate the schedule and cost required to progress the Lundberg and Engine House deposits to completion of the feasibility stage. An estimate of schedule is included to bring the project through detailed engineering, procurement, construction and commissioning.

The detailed schedule to bring the project to commissioning stage completion is shown in Figure 26.1. An estimated cost breakdown for anticipated field and study work to bring the project to the completion of the feasibility stage is presented in Table 26.1.

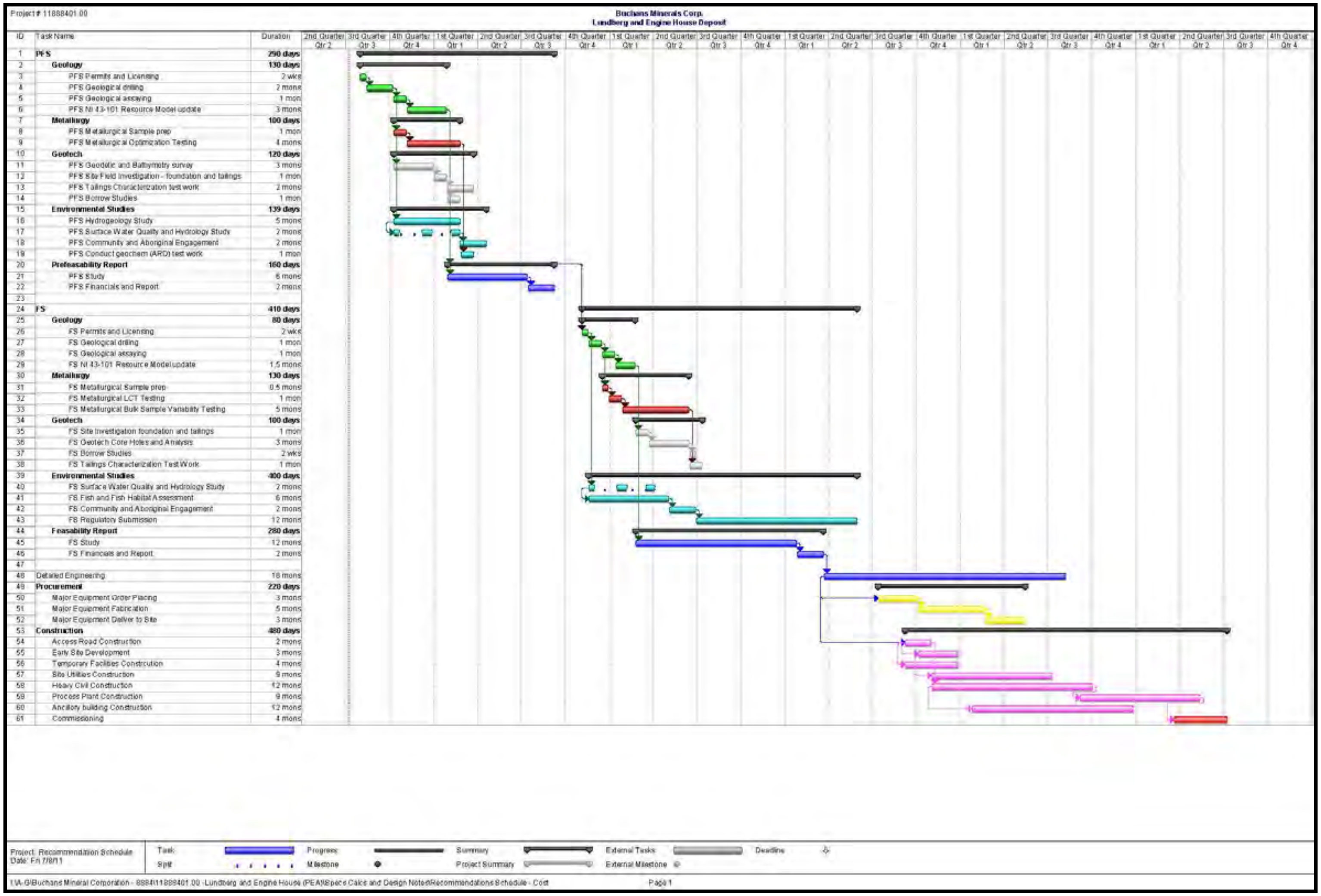
Table 26.1 Recommended Activity Cost for Prefeasibility and Feasibility Studies

Activity	Cost ($Cdn)
Prefeasibility	
Geology	
PFS Permits and Licensing	$5,000
PFS Geological Drilling and Assaying	$2,125,000
PFS Geological Management	$50,000
PFS NI 43-101 Resource Model Update	$70,000
Subtotal	**$2,250,000**
Metallurgy	
PFS Metallurgical Sample Preparation	$7,500
PFS Metallurgical Optimization Testing	$200,000
Subtotal	**$207,500**
Geotechnical	
PFS Geodetic and Bathymetry Survey	$25,000
PFS Site Field Investigation - Foundation and Tailings	$40,000
PFS Tailings Characterization Test Work	$25,000
PFS Borrow Studies	$25,000
Subtotal	**$115,000**
Environmental Studies	
PFS Hydrogeology Study	$300,000
PFS Surface Water Quality and Hydrology Study	$30,000
PFS Community and Aboriginal Engagement	$35,000

table continues...

Activity	Cost ($Cdn)
PFS Conduct Geochem (ARD) test work	$20,000
Subtotal	**$385,000**
Prefeasibility Report (Study, Financials and Report)	$675,000
Total for Prefeasibility	**$3,632,500**
Feasibility	
Geology	
FS Permits and Licensing	$5,000
FS Geological drilling and assaying	$1,062,500
PFS Geological management	$25,000
FS NI 43-101 Resource Model update	$35,000
Subtotal	**$1,127,500**
Metallurgy	
FS Metallurgical Sample prep	$7,500
FS Metallurgical LCT Testing	$100,000
FS Metallurgical Bulk Sample Variability Testing	$200,000
Subtotal	**$307,500**
Geotechnical	
FS Site Investigation foundation and tailings	$75,000
FS Borrow Studies	$25,000
FS Tailings Characterization Test Work	$25,000
FS Mine geotech core holes (approx 6 x 300m)	$315,000
FS Mine excavation stability study	$60,000
Subtotal	**$500,000**
Environmental Studies	
FS Surface Water Quality and Hydrology Study	$30,000
FS Fish and Fish Habitat Assessment	$60,000
FS Community and Aboriginal Engagement	$35,000
FS Regulatory Submission (excludes full EIS)	$100,000
Subtotal	**$225,000**
FS Report (Study, Financials and Report)	$2,000,000
Total for Feasibility	**$4,160,000**

Figure 26.1 Detailed Schedule



26.2 GEOLOGY

Based on the results of the Inferred Mineral Resource Estimate completed for this report the author provides the following recommendations with respect to future exploration on the Lundberg and Engine House deposits.

26.2.1 PHASE 1

1. Complete relogging of historic drill holes in areas where potential exists to expand the current resource.
2. Complete geological compilation, and planning for a 2,500 m diamond drilling program.
3. Complete an additional 2,500 m of diamond drilling on the Lundberg and Engine House deposits to confirm mineralization and extend the limits of the current resource estimate.

The following list of 11 vertical drill hole locations are recommended.

Table 26.2 Recommended Phase 1 Drill Holes

Hole ID	Easting	Northing	Depth (m)	Comment
PROP08-01	10200	8000	150	Near Surface Extension East of Lundberg
PROP08-02	10200	7950	125	Near Surface Extension East of Lundberg
PROP08-03	10150	7875	100	Near Surface Extension East of Lundberg
PROP08-04	9750	8090	225	Infill to N of Lundberg, area of incomplete drilling
PROP08-05	9750	7960	200	Delineate high-grade trend, deepen historical drilling
PROP08-06	9700	7975	225	Delineate high-grade trend, deepen historical drilling
PROP08-07	9600	7960	225	East of H-08-3407, within current resource
PROP08-08	9500	7960	225	West of H-08-3407, extend current resource
PROP08-09	9600	7900	150	South-East of H-08-3407, extend current resource
PROP08-10	9525	7900	175	South-West of H-08-3407, extend current resource
PROP08-11	9900	7600	200	West of H-08-3366, and confirm results of 2895
Contingency			500	
Total			**2500**	

Table 26.3 Estimated Phase 1 Budget

Survey	Units		Costs
Geology (field, relogging, etc.) includes assistant and expenses	120	days	$65,000
Geology (reports, planning, compilation)	50	days	$65,000
Drilling (includes assays and geologist, etc.)	2500	m	$500,000
Total			**$630,000**

26.2.2 PHASE 2

1. Complete 5,000 m of additional diamond drilling necessary to upgrade the current Inferred Mineral Resource to and Indicated category, based on positive results of Phase 1 drilling.

2. Plan to confirm resource delineated underneath the Lucky Strike Glory hole, as only historic drilling information exists for this area.

Based on the recommendation presented above the following Phase 1 and Phase 2 budgets are proposed.

PROPOSED PHASE 2 BUDGET

Phase 2 exploration is contingent on positive results of the Phase 1 programs.

Table 26.4 Estimated Phase 2 Budget

Survey	Units		Costs
Geology (field, reclogging, etc.) includes assistant and expenses	240	days	$130,000
Geology (reports, planning, compilation)	60	days	$77,500
Drilling (includes assays and geologist, etc.)	5000	m	$1,000,000
Total			**$1,207,500**

26.3 MINERAL PROCESSING AND METALLURGICAL TESTING

The results of this stage of investigation were premised on the investigation of two samples Met4 and Met2. LCT 3, an improvement on LCT 2, was used for the metallurgical and processing projections. The inference therefore is that the conclusions are limited by the resource investigation. The following is therefore recommended:

- The effluent from the final cycle of the lock cycle test will have to be analysed for mg/l CN, sulphide and acid base accounting to include multi-element analysis.
- Pyrite accumulation should be confirmed- this has implication for the tailings management facility and need for pyrite recovery.
- In order to confirm process ability, variability testing with additional samples and of significant difference in ore composition should be conducted.
- Optimization of main concentrates Cu, Zn and Pb for reagent consumption in relation to concentrate grade versus recovery.

- The upstream crushing and downstream operations of tailings management and concentrate treatment require further development in preparation for the receipt of the mill concentrates Cu, Zn, and Pb by the smelter.
- The next program should optimize the process variables including i) primary grind size, ii) depressant schemes, iii) regrind requirements
- The recovery of pyrite from the Zn tailings should be further investigated to assess the potential of generating a final non-acid generating tailings stream.
- Further testing to better develop a process for recovering barite should be completed if this is considered a priority for the project

26.4 MINING OPERATIONS

26.4.1 GEOTECHNICAL STUDY

A comprehensive geotechnical study is required for the next level of study, since the pit slope angle was based on conservative estimates from previous experience.

A comprehensive hydrological study is required for the next level of study.

26.4.2 OPEN PIT MINING

The ultimate pit design for the selected base case pit contains 17.28 Mt of resource (potential ore). The average grades over the LOM will be 1.63% Zn, 0.69% Pb, 0.40% Cu, 5.96 g/t Ag, 0.07 g/t Au and 1.24% Ba for a combined base metal grade of 2.72% (Zn-Pb-Cu). This is sufficient for approximately 10 years of mine life. All ore is classified as Inferred Resource, and should be drilled further to bring it into the Indicated and Measured Resource for the next level of study.

With an updated geological model, a re-optimization of the pit and pit design should be completed.

Details on the wall slopes by rock types needs to be determined for the typical orientations.

The stability of the waste dump needs to be assessed for the next level of study.

The Lundberg Engine House Open Pit project is in close proximity to the Town of Buchans. In order for the project to proceed, consultation with the town regarding the effect of pit blasting will have to be undertaken. Blasting controls are used to reduce vibration, noise, and fly rock potential. A blasting study is required to determine the areas of concern that are applicable. The blasting study will in turn have an effect on size/type of drill to be used and blasting methods. The impact of the blasting study may have an effect on mine operating costs.

Potential trade-off studies should be investigated in a future study to determine if better economic results exist for contractor mining versus owner mining.

26.4.3 MINING OPPORTUNITIES

The following mining opportunities exist in terms of further enhancement of the Lundberg and Engine House project:

- Completing a conceptual mining plan and metallurgical tests on the 100% owned Daniels Pond deposit (located ~90km distant from Lundberg) with a view to evaluating it as a possible satellite mining operation and providing additional feed to the Lundberg milling facility. The Daniels Pond NI 43-101 Indicated Resource of 929,000 tonnes and Inferred Resource of 332,000 tonnes (at a 2% Zn cut-off) may be viewed at the Company's website at http://www.buchansminerals.com.
- Exploring for additional high grade massive sulphide resources at the 100% owned Buchans North, Clementine, Clementine West and Little Sandy prospects located in close proximity to Lundberg.

26.5 PROJECT INFRASTRUCTURE

- Engage in discussion with the town of Buchans regarding the impact of open pit operations on the community.
- Negotiations with the power utility and power distribution companies regarding load requirements and capital funding programs available.
- Detailed inventory and investigation of the scope of work required to relocate or move buildings due to proximity to the proposed open pit mine blast zone.

26.6 TAILINGS MANAGEMENT FACILITY

- A Geodetic and Bathymetry survey of the existing tailings dams and basin to identify surface topography for tailings dams and storage capacity within the existing tailings basins.
- Thorough review of the site geotechnical conditions, as well as tailings dam geotechnical and hydrology design criteria. An additional geotechnical site investigation might be warranted depending on the results of the review.
- Laboratory tests on representative tailings samples to identify tailings geotechnical and geochemical properties.
- PFS level design for the selected expansion option will be required including
  - Engineering stability, seepage and settlement analyses for the selected expansion option as necessary.

- Thorough review and assessment of tailings basin water balance and site-wide water management strategy.
- Tailings deposition plan.
- Design of water control structures (e.g. overflow spillways/emergency spillways, ditches, collection ponds, etc.).
- Preparation of a PFS report including results of detailed analyses, modelling, test work and design drawings.
- Identify borrow sources for tailings dams construction.
- Plan and schedule construction for the selected expansion option and develop detailed quantity take-offs and construction costs.

26.7 Environmental Planning and Assessment

- Hydrogeology – given the importance of Buchans Lake as a drinking water source for the community and its proximity to the project site, a hydrogeology study will be necessary to examine the potential for a groundwater connection between the pit and the lake. This study also will provide better information for estimation of seepage rate (and therefore dewatering requirements) into the pit and of seepage water quality. There are opportunities to use new exploration or geotechnical boreholes to collect the hydrogeological data, with piezometers installed in the boreholes to provide permanent monitoring wells.
- Initiate/continue community and Aboriginal engagement to inform potentially affected stakeholders of project plans and next steps and to provide opportunities to discuss issues and opportunities. The Federation of Newfoundland Indians should be consulted to identify potential affected aboriginal communities and stakeholders to be included in the consultation program in addition to the previously identified town of Buchans and Sple'tk (Exploits) First Nation.
- Plan and initiate EBS to support the preparation of a registration document. Baseline studies will need to focus on surface water quantity and quality and fish and fish habitat in, and downstream from, areas of potential water withdrawals and effluent discharges. Groundwater quality and quantity around the open pit, tailings management area, and waste rock stockpile also needs to be documented to support assessment of potential effects on down-gradient ground and surface water resources and support estimates of groundwater seepage to the pit. The potential for interaction between the project and terrestrial resources needs to be further evaluated and, where there is the potential for interaction outside the already disturbed area initiate specific terrestrial baseline studies as well. Terrestrial environmental components that may have to be considered include migratory birds and raptors, rare vegetation, and wildlife (particularly Newfoundland marten populations if the project interacts with suitable habitat).

- Initiate a geochemical testing program to determine ARD and ML potential for waste rock, ore and tailings to be generated by the project to support estimation of runoff quality during operations and pit water quality at closure. Collect and analyze water samples from the open pit to assess treatment requirements. Sampling needs to cover a range of depths from surface to bottom.
- Integrate environmental planning into overall project planning to avoid potential adverse environmental effects, where possible, through planning rather than by mitigation. This should include the consideration of progressive reclamation and closure requirements in the design of all major project components, including the tailings management area, waste rock stockpiles, and the open pit.

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ENVIRONMENTAL

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MINERAL PROCESSING

Imeson, D. 2011 Email June 19 [Dan.Imeson@sgs.com] – Buchans Lunberg and Engine House Mineralization Description.

Stewart, K. 2011 Email February 23 (Lakefield) [Kevin.Stewart@sgs.com] – Sample #4 Tests.xls – LCT1 and LCT2.

Stewart, K. 2011 Email April 12 (Lakefield) [Kevin.Stewart@sgs.com] – Barite flotation test results.

Stewart, K. 2011 Email April 25 (Lakefield) [Kevin.Stewart@sgs.com] – LCT3 Concs. Impurities.

Stewart, K. 2011 Email March 23(Lakefield) [Kevin.Stewart@sgs.com] – LCT3.

MacLeod, D. 2011 Email February 18 [dsmacleod@shaw.ca] – Heavy Liquid Separation spread sheet and meeting minutes.

MacLeod, D. 2011 Email March 9 [dsmacleod@shaw.ca] – Testing on the Lundberg Sample #4 – Ore comparison, Disperson (Calgon) versus collector (3418A) investigation, and tests –F26 –F32.

MacLeod, D. 2011 Email March 23 [dsmacleod@shaw.ca] – LCT summary.

MacLeod, D. 2011 Email March 31 [dsmacleod@shaw.ca] – LCT3 Ag.

28.0 CERTIFICATES OF QUALIFIED PERSONS

MIKE MCLAUGHLIN, P.ENG.

I, Mike McLaughlin, P.Eng., of Barrie, Ontario, do hereby certify:

- I am a Project Manager with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Lundberg and Engine House Deposits, Newfoundland, Canada, dated August 11, 2011 (the "Technical Report").
- I am a graduate of McMaster University, B.Eng. in Mechanical Engineering, 1990. I am a member in good standing of the Association of Professional Engineers Ontario (License #10084932). My relevant experience includes +14 years of engineering experience, successfully managing projects involving front end mining resource models and economic evaluation studies. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 1, 2, 3, 15, 17, 18, 19, 21, 22, 25.4, 26.1, 26.2, 26.5, 26.6 and 27 of the Technical Report.
- I am independent of Buchans Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the technical report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 11th day of August, 2011 at Toronto, Ontario.

"Original document signed and sealed by Mike McLaughlin, P.Eng.

Mike McLaughlin, P.Eng.
Project Manager
Wardrop Engineering Inc.

DOUG RAMSEY, R.P. BIO (BC)

I, Doug Ramsey, R.P. Bio (BC), of Vancouver, BC, do hereby certify:

- I am a Manager – Environmental Assessment, Permitting, and Natural Resources with Tetra Tech, with a business address at 800 – 555 West Hastings St., Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Lundberg and Engine House Deposits, Newfoundland, Canada, dated August 11, 2011 (the "Technical Report").
- I am a graduate of the University of Manitoba, Winnipeg, Manitoba, (B.Sc. (Hons), Zoology, 1979, and M.Sc. Zoology, 1985). I am a member in good standing of the College of Applied Biology, British Columbia, as a Registered Professional Biologist (#1581). My relevant experience is 29+ years experience as an environmental consultant working in environmental permitting and 23 years of experience in the environmental permitting monitoring and closure of mining projects. My mining permitting and planning experience includes NB, NL,PQ,ON, MB, SK, NWT, YK, and BC and includes coal, gold, base metal, rare earth elements, and potash. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 20, and 26.7 of the Technical Report.
- I am independent of Buchans Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the technical report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 11th day of August, 2011 at Toronto, Ontario.

"Original document signed and sealed by Doug Ramsey, R.P. Bio. (BC)"

Doug Ramsey, R.P. Bio. (BC)
Manager – Environmental Assessment, Permitting, and Natural Resources
Tetra Tech

DANIEL COLEY, MBA, P.ENG.

I, Daniel Coley, MBA, P.Eng., of Toronto, Ontario, do hereby certify:

- I am a Senior Process Engineer with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Lundberg and Engine House Deposits, Newfoundland, Canada, dated August 11, 2011 (the "Technical Report").
- I am a graduate of the University of the West Indies (B.Sc. Chemical Engineering, 1991) and Nova South Eastern University (MBA, 1991). I am a member in good standing of the Professional Engineers of Ontario (License #100132643) and the Association of Professional Engineers and Geoscientists of Saskatchewan (License #16618). My relevant experience is [20 years experience in mining minerals and chemical processes covering design and engineering integration for processes to include base metals, rare earth elements, alumina, potash, boron, graphite, sulphuric acid, potash, water purification and management, reagent systems design. EPCM experience covers scoping studies prefeasibility studies, process design criteria development, feasibility studies, detailed design, construction management, commissioning and start-up, operator training and process system design validation. Experience also extends to project costing and economic analyses.]. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 13, 25.2, and 26.3 of the Technical Report.
- I am independent of Buchans Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the technical report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 11th day of August, 2011 at Toronto, Ontario.

"Original document signed and sealed by Daniel Coley, MBA, P.Eng."

Daniel Coley, MBA, P.Eng.
Senior Process Engineer
Wardrop Engineering Inc.

DANIEL GAGNON, P.ENG.

I, Daniel Gagnon, P.Eng., of Sudbury, Ontario, do hereby certify:

- I am a Senior Open Pit Mining Engineer with Wardrop Engineering Inc. with a business address at 101-957 Cambrian Heights, Sudbury ON, P3C 5M6.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Lundberg and Engine House Deposits, Newfoundland, Canada, dated August 11, 2011 (the "Technical Report").
- I am a graduate of Laurentian University (B.Sc. Mining Engineering, 1993). I am a member in good standing of the Association of Professional Engineers Ontario (License #100101928). My relevant experience includes 18 years of open pit mining experience with a strong mine planning background and considerable operational experience in base metals, precious metals, and coal. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was September 10th, 2011 for 3 days.
- I am responsible for Sections 16, 25.3, and 26.4 of the Technical Report.
- I am independent of Buchans Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the technical report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 11th day of August, 2011 at Toronto, Ontario.

"Original document signed and sealed by Daniel Gagnon, P.Eng."

Daniel Gagnon, P.Eng.
Senior Open Pit Mining Engineer
Wardrop Engineering Inc.

PETER C. WEBSTER, P.GEO.

I, Peter C. Webster*, P. Geo.* do hereby certify that:

I currently reside in Dartmouth, Nova Scotia and I am currently employed as President and Senior Manager with Mercator Geological Services Limited, 65 Queen Street, Dartmouth, Nova Scotia, Canada, B2Y 1G4.

- I graduated with a Bachelors Degree in Geology from Dalhousie University in 1981. In addition, I obtained a Certificate in Environmental Management (C.E.M.) from the Technical University of Nova Scotia in 1996.
- I am a registered member in good standing of the Association of Professional Geoscientists of Nova Scotia, registration number 047. I am a member in good standing of the Association of Professional Engineers and Geoscientists of Newfoundland and Labrador, member number 03337.
- I have worked as a geologist in Canada and internationally for over 27 years since my graduation from university in 1981. I have a wide variety of commodity experience including, gold, VMS, base metals, nickel, and industrial minerals. I have completed numerous NI43-101 compliant Technical Reports and Resource Estimates. I have extensive relevant work experience and authored numerous reports on similar style mineral deposits in Newfoundland.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 24, 25.1, and 26.2 of the Technical Report.
- To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.
- I am independent of Buchans Minerals Corp. applying all of the tests in Section 1.5 of National Instrument 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and believe that this Technical Report has been prepared in compliance with that instrument and form.

Signed and dated this 11th day of August, 2011 at Dartmouth, Nova Scotia

"Original document signed and sealed by Peter Webster, P.Geo."

Peter C. Webster, P.Geo.
President
Mercator Geological Services Limited