VOTORANTIM METALS CANADA INC.
Suite 1330, 4 King St. West, Toronto, Ontario, M5H 1B6, Canada

650 - 555 West 12th Ave.,
Vancouver, B.C. V5Z 3X7 Canada

TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT (PEA) OF THE MURRAY BROOK PROJECT NEW BRUNSWICK, CANADA

Latitude 47° 31'30'' North
Longitude 66° 26'00'' West

For

VOTORANTIM METALS CANADA INC.
AND
EL NINO VENTURES INC.

By
P&E Mining Consultants Inc.
Suite 202- 2 County Court Blvd
Brampton, Ontario,
L6W 3W8

NI-43-101F1
TECHNICAL REPORT No. 270

Mr. Eugene Puritch, P.Eng.
Mr. Gerald Harron, P.Geo.
Mr. Alfred Hayden, P.Eng.
Mr. Yungang Wu, P.Geo.
Mr. Ken Kuchling, P.Eng
Mr. David Orava, P.Eng.
Ms. Tracy Armstrong, P.Geo.
Mr. Kirk Rodgers, P.Eng.

Effective Date: June 4, 2013
Signing Date: July 19, 2013

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1.0 SUMMARY

This report was prepared to provide a National Instrument 43-101 ("NI 43-101") compliant Technical Report, Resource Estimate and Preliminary Economic Assessment ("PEA") of the Copper- Lead-Zinc mineralization contained in the Murray Brook deposit, located approximately 60 kilometres to the west of the town of Bathurst, New Brunswick. The Murray Brook Project ("the Property" or "the Project") is subject to a Joint Venture ("MBJV") arrangement between Votorantim Metals Canada Inc. (the project operator) and El Nino Ventures Inc. who hold a 65% and 35% interest respectively.

This report was prepared by P&E Mining Consultants Inc. ("P&E") at the request of Mr. Rodney Thomas, General Manager at Votorantim Metals Canada Inc.

The Property is located approximately 60 km west of Bathurst in the Parish of Balmoral, Restigouche County, New Brunswick, Canada and consists of surveyed Mineral Lease # 252, which covers approximately 505 ha. A 5 km gravel access road extends southward from Highway 180 to the mine site. Bathurst in the east provides access to rail and ocean shipping facilities. Several communities in the region offer commercial goods, social, educational and financial amenities, as well as a pool of skilled labour.

Physiographically the property is located in the Miramichi Highlands, characterized by rounded glacially scoured hills. Land use in the area is mainly for tourism, forestry and mining.

1.1 MINERAL RESOURCES

The PEA incorporates P&E's NI 43-101 resource estimate for sulphide and oxide mineral resources at a C$21/t Net Smelter Return ("NSR") cut-off that is summarized in Table 1.1. The drilling database of the Murray Brook Project contains 10,045 samples, all of which were analyzed for copper ("Cu")%, lead ("Pb")%, zinc ("Zn")%, gold ("Au") g/t and silver ("Ag") g/t. A total of 7,964 assays from 141 drill holes have been utilized for the resource estimate.

Grade capping was investigated on the one metre composite values within the constraining domains to ensure that the possible influence of erratic high values did not bias the database. Based on the log-normal histogram performance, Cu was capped at 6%, Pb at 10% and Ag at 250g/t while no capping was applied for Zn and Au. The capped average grade decreased less than 1% from the average grade of the composites.

The Murray Brook resource block model was constructed using Gemcom modeling software. The block model is oriented with X axis at 110° azimuth with 3m x 3m x 3m blocks. Inverse Distance Squared ($1/d^2$) grade interpolation was utilized for the Cu, Pb and Zn grade interpolation while Inverse Distance Cubed ($1/d^3$) was used for the Au and Ag grade interpolation, both with the capped composites. The average block-model mineralized bulk density was calculated to be 4.08 tonnes per cubic metre.

The resource classification was determined with Zn interpolation due to Zn generating the highest proportionate NSR value in the block model. Based on the semi-variogram performance and density of the drilling data, the Measured Resource category was justified for blocks interpolated by the first pass using at least seven composites from a minimum of four drill holes within a spacing of 25m along strike, 40m down dip and 15m on the across dip direction.

Indicated Resources were classified to the blocks interpolated with the second pass; while Inferred Resources were categorized for all remaining unclassified blocks. The classifications of some blocks have been manually adjusted to represent the resource classification more reasonably.

The open pit NSR cut-off was based on three year trailing average metal prices as of Jan 31, 2013, including Cu at US$3.68/lb, Pb at US$1.00/lb, Zn at USS$0.95/lb, Au at US$1500.00/oz, Ag at US$29.00/oz, and a $US/$C Exchange Rate of 1:1. Taking into consideration processing costs, concentrate recoveries, smelter payables, treatment charges, humidity factors, and General and Administration ("G&A"), the NSR value of the mineralized blocks were calculated using the following formula:

NSR = (Cu% x 38.54 + Pb% x 9.13 + Zn% x 15.81 + Au x 0.0 + Ag x 0.44) - 11.43

The basis of this construction of this formula is provided in Section 14.10.

TABLE 1.1
MURRAY BROOK IN-PIT MINERAL RESOURCE ESTIMATE AT C$21/T NSR CUT-OFF(1-3)

Zone	Category	Tonnes	Cu %	Cu M lb	Pb %	Pb M lb	Zn %	Zn M lb	Au g/t	Au K oz	Ag g/t	Ag M oz
Oxide	Measured	981,000	0.90	19.5	0.89	19.2	2.73	59.0	0.33	10.5	39.8	1.3
	Indicated	302,000	1.02	6.8	0.69	4.6	2.05	13.7	0.54	5.3	33.9	0.3
	M+I	1,283,000	0.93	26.3	0.84	23.8	2.57	72.7	0.38	15.8	38.4	1.6
	Inferred	4,000	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.4	0.0
Sulphide	Measured	11,306,000	0.40	100.7	1.04	258.3	2.97	741.2	0.50	182.7	42.5	15.4
	Indicated	6,578,000	0.57	82.9	0.91	131.6	2.32	336.8	0.74	155.5	40.3	8.5
	M+I	17,884,000	0.47	183.6	0.99	389.9	2.73	1,078.1	0.59	338.2	41.7	23.9
	Inferred	284,000	1.57	9.8	0.50	3.1	1.36	8.5	0.47	4.3	28.7	0.3

(1) Mineral Resources which are not mineral reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

(2) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.

(3) The Mineral Resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.

(4) "M" means millions; "K" means thousands.

The NSR cut-off sensitivities to the In-Pit resource estimate are tabulated in Table 1.2.

TABLE 1.2
OPEN PIT SENSITIVITY TO RESOURCE ESTIMATE OF THE MURRAY BROOK PROJECT

Zone	Category	Cut-Off NSR $/t	Tonnage tonnes	Cu %	Cu M lb	Pb %	Pb M lb	Zn %	Zn M lb	Au g/t	Au K oz	Ag g/t	Ag M oz
Oxide	Measured	100	358,681	1.20	9.5	1.53	12.1	4.76	37.6	0.31	3.5	64.43	0.7
		50	714,514	1.02	16.0	1.09	17.2	3.39	53.4	0.33	7.7	47.66	1.1
		45	770,240	0.99	16.8	1.04	17.7	3.23	54.8	0.34	8.4	45.85	1.1
		40	828,244	0.97	17.7	1.00	18.2	3.07	56.1	0.34	9.0	44.08	1.2
		35	883,714	0.94	18.4	0.96	18.6	2.94	57.3	0.34	9.6	42.55	1.2
		30	929,200	0.92	18.9	0.92	18.9	2.84	58.1	0.34	10.1	41.28	1.2
		25	959,643	0.91	19.3	0.90	19.1	2.77	58.7	0.34	10.3	40.40	1.2
		21	980,755	0.90	19.5	0.89	19.2	2.73	59.0	0.33	10.5	39.78	1.3
		15	1,002,207	0.89	19.6	0.87	19.3	2.68	59.2	0.33	10.7	39.13	1.3
		10	1,010,970	0.88	19.7	0.87	19.3	2.66	59.3	0.33	10.7	38.86	1.3
	Indicated	100	79,901	2.30	4.1	0.77	1.4	2.75	4.8	0.40	1.0	44.77	0.1
		50	219,083	1.25	6.1	0.77	3.7	2.35	11.4	0.54	3.8	38.85	0.3
		45	238,103	1.19	6.3	0.75	4.0	2.28	12.0	0.54	4.1	37.64	0.3
		40	253,796	1.15	6.4	0.74	4.1	2.22	12.4	0.54	4.4	36.60	0.3
		35	272,203	1.10	6.6	0.72	4.3	2.16	13.0	0.54	4.7	35.65	0.3
		30	287,446	1.06	6.7	0.71	4.5	2.11	13.4	0.55	5.1	34.82	0.3
		25	296,972	1.03	6.8	0.70	4.6	2.07	13.6	0.54	5.2	34.18	0.3
		21	301,728	1.02	6.8	0.69	4.6	2.05	13.7	0.54	5.3	33.86	0.3
		15	305,211	1.01	6.8	0.69	4.6	2.04	13.7	0.54	5.3	33.60	0.3
		10	306,717	1.01	6.8	0.69	4.6	2.03	13.7	0.54	5.4	33.46	0.3
	Inferred	100	3,706	3.90	0.3	0.18	0.0	0.61	0.0	0.45	0.1	26.40	0.0
		50	4,100	3.73	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.45	0.0
		45	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		40	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		35	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		30	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		25	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		21	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		15	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		10	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
Sulphide	Measured	100	2,645,129	0.41	24.0	2.12	123.3	6.26	364.9	0.58	48.9	79.59	6.8
		50	7,433,505	0.42	68.3	1.34	219.8	3.87	633.8	0.54	129.9	53.55	12.8
		45	8,150,214	0.42	74.9	1.27	229.0	3.67	659.5	0.54	140.5	51.18	13.4
		40	8,898,154	0.42	81.6	1.21	237.7	3.48	683.5	0.53	151.1	48.91	14.0
		35	9,651,043	0.41	88.2	1.15	245.3	3.31	704.4	0.52	161.5	46.78	14.5
		30	10,362,032	0.41	94.0	1.10	251.4	3.16	721.8	0.51	171.0	44.88	15.0
		25	10,941,894	0.41	98.4	1.06	255.8	3.04	734.2	0.51	178.3	43.39	15.3
		21	11,306,015	0.40	100.7	1.04	258.3	2.97	741.2	0.50	182.7	42.47	15.4
		15	11,696,931	0.40	102.9	1.01	260.5	2.90	747.7	0.50	186.9	41.48	15.6
		10	11,901,885	0.40	103.9	1.00	261.5	2.86	750.4	0.49	188.8	40.94	15.7
	Indicated	100	1,059,076	0.86	20.0	1.83	42.7	5.03	117.4	0.88	30.0	72.47	2.5
		50	4,540,579	0.63	63.5	1.12	111.8	2.84	283.9	0.84	123.0	48.68	7.1
		45	4,983,256	0.62	68.1	1.07	117.5	2.72	298.5	0.82	131.2	46.77	7.5
		40	5,420,390	0.61	72.4	1.02	122.3	2.61	311.4	0.80	138.7	44.94	7.8
		35	5,815,306	0.59	76.3	0.98	126.1	2.51	321.6	0.78	144.9	43.36	8.1
		30	6,135,008	0.59	79.2	0.95	128.7	2.43	328.8	0.76	149.4	42.08	8.3
		25	6,398,280	0.58	81.5	0.93	130.5	2.37	333.8	0.74	153.1	41.05	8.4
		21	6,578,261	0.57	82.9	0.91	131.6	2.32	336.8	0.74	155.5	40.33	8.5
		15	6,766,188	0.56	84.2	0.89	132.6	2.28	339.4	0.72	157.4	39.54	8.6
		10	6,855,652	0.56	84.7	0.88	133.0	2.25	340.5	0.72	158.3	39.14	8.6
	Inferred	100	95,998	2.73	5.8	0.63	1.3	1.75	3.7	0.49	1.5	37.30	0.1
		50	207,120	1.93	8.8	0.56	2.5	1.52	6.9	0.53	3.5	32.53	0.2
		45	223,903	1.84	9.1	0.54	2.7	1.48	7.3	0.52	3.7	31.70	0.2
		40	239,206	1.77	9.3	0.53	2.8	1.45	7.7	0.51	3.9	30.95	0.2
		35	254,458	1.70	9.5	0.52	2.9	1.43	8.0	0.50	4.1	30.18	0.2
		30	268,595	1.63	9.7	0.51	3.0	1.40	8.3	0.49	4.2	29.49	0.3
		25	278,112	1.59	9.8	0.50	3.1	1.38	8.5	0.48	4.3	29.04	0.3
		21	284,487	1.57	9.8	0.50	3.1	1.36	8.5	0.47	4.3	28.74	0.3
		15	292,234	1.53	9.9	0.49	3.2	1.34	8.6	0.46	4.4	28.36	0.3
		10	294,655	1.52	9.9	0.49	3.2	1.33	8.6	0.46	4.4	28.24	0.3

1.1.1 Potentially Economic Portion of the Mineral Resources

A potentially mineable portion of these Mineral Resources was determined as a basis for a Preliminary Economic Assessment of the property. The envisaged open pit mining methods are estimated to experience mining dilution in the order of 11.7% at a diluting grade of 0.12% Cu, 0.07% Pb, 0.2% Zn and 4.56 g/t Ag. Mineralization losses during extraction mining are estimated to be 3%.

This Potentially Mineable Portion of the Mineral Resources ("the Deposit") contains Inferred Mineral Resources. This material has not been sufficiently drilled to confidently demonstrate economic viability. In addition, the work undertaken on the Murray Brook Project to date is considered to be at conceptual levels of study only. As such, and according to the NI 43-101 Disclosure Guidelines, it is not possible to declare a mineral reserve of any kind.

A conceptual mining and processing plan has been developed to assess the potential of economically extracting metals from the Property.

1.2 CONCEPTUAL MINING AND PROCESSING PLAN

The Murray Brook Project open pit would be a conventional mining operation, producing approximately 6,000 tonnes per day of mill feed. Mining would be carried out by drilling and blasting of mineralization and waste rock, followed by truck loading and haulage operations. The open pit is scheduled to produce approximately 18.9 million tonnes of mill feed over a 10 year mine life from a single open pit. In addition, varying amounts of waste rock will be produced, with a life-of-mine average stripping ratio of 4.32:1.

The estimated capital expenditures for the mining operation have been estimated at $335 million over the mine life. These costs are a combination of pre-stripping operations, mine equipment purchase costs, surface facility construction, environmental costs and sustaining capital. In addition, allowances for engineering, procurement and construction management ("EPCM"), contractor's overhead costs and contingency have been included.

Whereas the yearly mine operating cost will vary depending on mining depth, it has been estimated that the average cost over the mine life will be about $2.30 per rock tonne mined. This corresponds to a life of mine average cost of $12.24 per mill feed tonne milled, taking into account the average stripping ratio at the mine.

Scoping level mineralogical and metallurgical test work investigating the potential recovery of copper, lead and zinc concentrates have been carried out.

Capital costs for the processing operations are based on an average daily throughput of approximately 6,000 tonnes per day and the general flowsheet is described in Section 17. The total capital cost of the processing operation, including equipment, direct and indirect costs and contingency is $167 million.

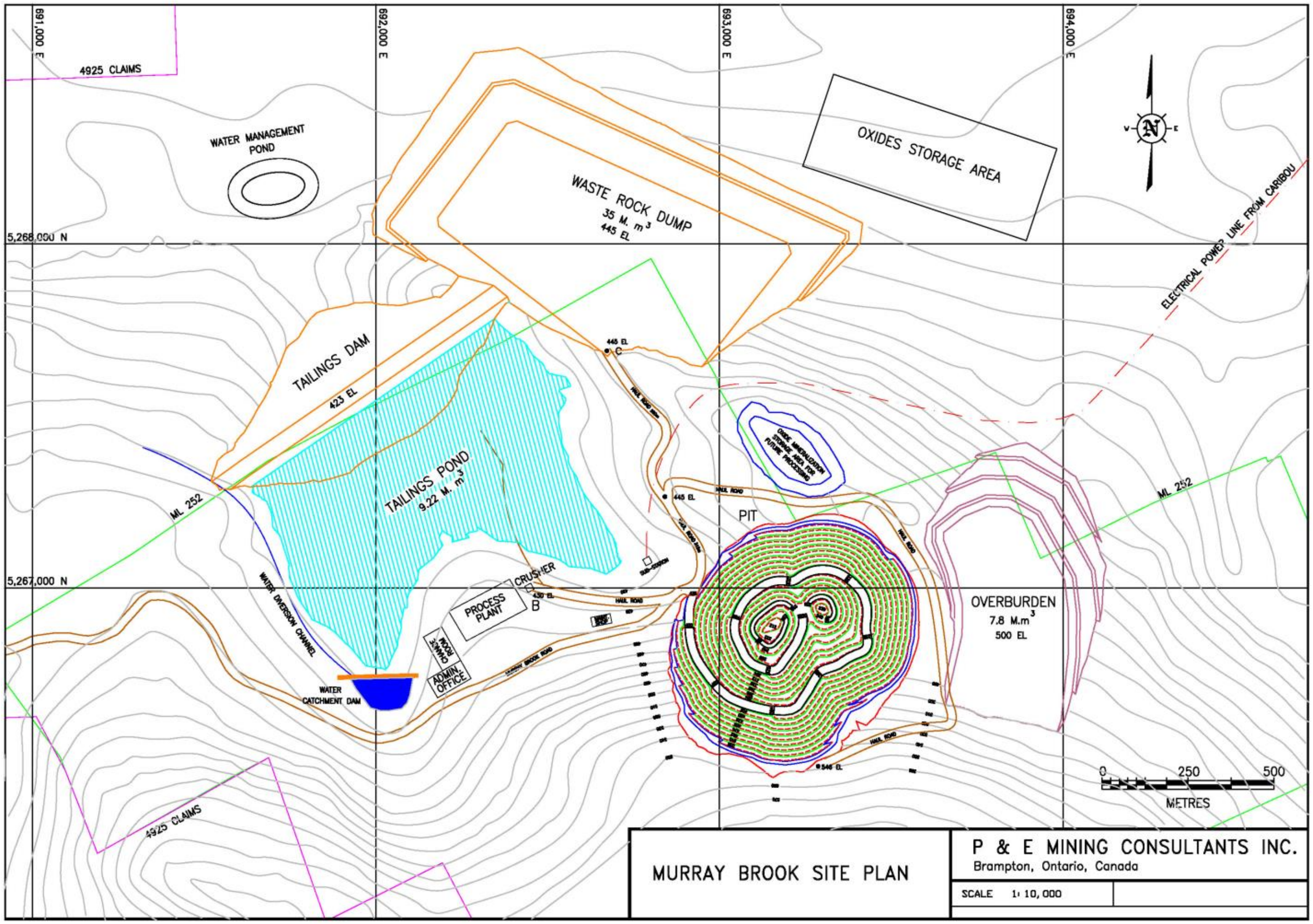
The total average processing cost is estimated to be in the order of $14.25 per tonne milled.

The Murray Brook Project will have access to the substantial infrastructure, services and skilled labour in the area. There will be minimal access infrastructure cost requirements due to its location near Hwy 180 and about 60 km from the historic mining town of Bathurst. The regional labour force includes experienced equipment operators, mine workers and material and equipment suppliers.

The mine plan for the Murray Brook Project has an approximate ten year mill production life with a total mill feed to a single semi-autogenous grinding mill on the Project of approximately 2.0 Mt per year.

A site plan with proposed locations of the site infrastructure is provided in Figure 1.1.

Figure 1.1 Site Plan



Site infrastructure capital costs expended in the preproduction period (excluding those costs related to the processing plant) have been estimated at $35 million. This includes a tailings management facility located approximately north of the processing plant, a water treatment plant, buildings, electrical delivery and distribution facilities, preparation of a waste rock storage facility, establishment of a site water supply and other necessary facilities. In addition, approximately $61 million have been included over the mine life, primarily for environmental monitoring and site reclamation. Allowances for EPCM, contractor overhead costs and contingency have been included in these capital costs.

G&A costs for the mine operating life have been estimated at $2.5 million per year or $1.32/tonne processed.

1.3 ENVIRONMENTAL IMPACT AND REHABILITATION

Approximately 1.2 Mt of gossan material was mined at the Murray Brook site between 1989 and 1992. This was treated for gold and silver recovery using an agglomeration vat-leaching cyanidation process. The site was reclaimed in 2000. The reclamation plan involved the dewatering of the historic pit and the relocation of 600 kt of treated gossan and ~50 kt of sulphide mineralization to the pit and compacted till capping. Under the reclamation plan, the pit groundwater preferentially flows around the low permeability gossan placed in the pit. Historic gossan stockpiled outside of the pit was encapsulated between a bottom liner and till top cover. The results of post-reclamation environmental monitoring indicate that the reclamation measures have reduced but not eliminated contaminated groundwater migration downstream. The receiving water courses are known as Gossan Creek and Copper Creek.

MBJV’s environmental consultants have conducted additional groundwater and surface water quality sampling, environmental effects monitoring and sampling in downstream receiving waters, and met with provincial regulators to gain an understanding of New Brunswick’s objectives for the mitigation of historic impacts to water quality in Gossan and Copper Creeks where the water quality in parts of the streams is understood be insufficient to support a fish population. The historic mine site operator had put $0.5M in rehabilitation security in place for an Approval to Operate for the reclaimed Murray Brook historic mine site which was to expire in October 2014. During the preparation of the present PEA, MBJV was in negotiations with provincial regulators to obtain an Approval to Operate for the historic reclaimed site. The present PEA has assumed that MBJV would increase the rehabilitation security for the reclaimed site from the historic $0.5M level to $2M. Votorantim has since reported that it has received an Approval to Operate (ATO) No. I-8297 valid to March 31, 2018 for the operation of the Reclaimed Murray Brook Mine Site identified as a “source” including the historic tailings (such as the processed gossan) pile, the reclaimed open pit, and the groundwater monitoring well. ATO I-8297 does not include approval to remove bulk material from the in-filled open pit or disturb the tailings pile (such as the historic capped gossan material). Votorantim also reports that it has replaced the historic $0.5M security with a $0.5M cash security with the understanding that its cash security will be returned when Votorantim provides a $2M rehabilitation bond in the form of an irrevocable letter of credit to the Government of New Brunswick.

The Project as currently envisaged would make use of best management practices and engineered controls to eliminate or mitigate potential environmental impacts and would be designed to take reclamation requirements into consideration. The proposed project includes controls to inhibit sulphide oxidation and impacts to downstream water quality. As examples, the tailings would be disposed underwater to inhibit sulphide oxidation and acidic drainage, and effluent would be actively treated prior to release over the operating life of the mine and during the site reclamation phase.

It is envisaged that the reclamation plans for the proposed Project and the historic treated gossan would be combined into an integrated reclamation plan with the key objective of reducing or eliminating active long-term care and maintenance requirements where possible. Key elements of the conceptual closure plan include the following:

- The historic treated gossan would be relocated to an engineered containment area commencing at the proposed pre-production phase. Till and non-treated/non-processed gossan excavated from the proposed open pit would be stockpiled separately;
- It has been assumed that the acid generating / metal leaching waste rock excavated from the proposed open pit would be segregated and separately stockpiled and then relocated back to the pit and kept submerged. Sulphide tailings would be kept saturated within the engineered tailings management storage area. Non-acid generating / non-metal leaching waste rock would be separately stockpiled with physically stable final slopes and re-vegetated;
- The processing plant, site infrastructure and mine and support equipment would be salvaged or otherwise demolished. Waste materials would be disposed of in accordance with regulatory requirements. Disturbed land areas would be reclaimed and left in a safe and stable condition;
- The performance of the reclamation plan would be assessed using a 3 – 5 year long post-reclamation monitoring program.

The PEA includes the following environmental cost allowances:

- $6.5M in additional closure security allowances comprised of a $3M reclamation security, a $1.5M historic reclamation security top-up amount; and a $2M security cost allowance for the effluent treatment plant. These costs are additional to the historic $0.5M security amount;
- $23M for initial environmental costs. The estimated cost is intended to cover the costs of additional acid rock drainage and metal leaching testing; additional groundwater plume (e.g. from the historic treated gossan to the downstream receiving creek) investigations; integrated reclamation plan development; treated gossan relocation and storage; initial tailings management storage area and effluent treatment plant construction; and waste rock storage pad development;
- $24M for environmental costs that would be incurred over the operating life of the Project for tailings management storage area expansion, effluent recycling and treatment, and progressive reclamation. Environmental monitoring costs over the operational life of the Project are included in the project G&A costs;
- A provisional cost of $22M is included for the final reclamation works and post-reclamation monitoring and care and maintenance costs.

P&E reviewed the conceptual integrated rehabilitation plan for the Project with an environmental consultant familiar with the Project site who has been involved in the reclamation of other mine sites in the region of the Project. Key aspects of the Project that will require further study and may have the potential to affect the projected economic outcome of the Project are the actual quantities of acid generating/metal leaching materials and the mine waste and water management plans. P&E has recommended that appropriate studies be carried out in these areas.

Votorantim has obtained an Approval to Operate (No. I-8297) for the reclaimed Murray Brook mine site. Under ATO I-8297, should Votorantim not register a project under the Environmental Impact Assessment Regulation by December 31, 2016, a mitigation plan would need to be developed for regulatory review and the approved mitigation plan implemented. The present PEA assumes that MBJV would register the Project under the Environmental Impact Assessment Regulation, undertake the mine approvals and environmental impact assessment processes, including public input and consultation, obtain the necessary approvals and permitting for the Project, and operate and reclaim the site based on an integrated reclamation plan for the Project including the historic Murray Brook mine site.

As part of MBJV's efforts and focus to advance the Project to the present PEA level, MBJV has interfaced with regulatory authorities and some people in the vicinity of the project. MBJV reports that it has received generally positive support for the Project to date and is working towards the commencement of a wider scope community communication program. MBJV understands the importance of obtaining input and maintaining good relationships with local communities, First Nations and others.

1.4 FINANCIAL EVALUATION

The Murray Brook Project's financial results are summarized in Table 1.3 and indicate an after-tax net present value ("NPV") of $96.4 million at a 5% discount rate, an internal rate of return ("IRR") of 11.4% and a 5.4 year payback. The initial capital expenditure would be $260.8 million with a life-of-mine capital cost of $334.8 million. All currency values are expressed in Canadian dollars unless otherwise noted.

TABLE 1.3
FINANCIAL RESULTS SUMMARY

NPV (0%)	$229	millions
NPV (5%)	$96	millions
NPV (7%)	$60	millions
IRR	11.4%	
Payback	5.4	years
Total Life-of-Mine Capital	**$335**	**millions**

The financial results are based on April 30, 2013 three year trailing average metal prices of $US 3.70/lb copper, $US 1.00/lb lead, $US 0.94/lb zinc, $US 1,540/oz gold, $US 30.09/oz silver. The assumed exchange rate is $US:$C = 1:1.

1.5 CONCLUSIONS AND RECOMMENDATIONS

The Base Case of this PEA shows that the Project has economic potential for producing copper, lead and zinc concentrates.

Note: This PEA is preliminary in nature and its mineable tonnage includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves and there is no certainty that the preliminary assessment will be realized. Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

P&E recommends that MBJV advance the project with extended and advanced technical studies particularly in metallurgical, geotechnical and environmental matters with the intention to advance the project to a Pre-Feasibility Study level.

Specifically, it is recommended that MBJV take the following actions:

- Continue metallurgical test work on a larger scale to confirm and improve the process design with the goal of improving metal recoveries;
- Assess the possibility of re-processing the reclaimed gossan, while taking metallurgical, environmental protection and legal aspects into consideration;
- Characterize the acid generation / acid consuming potential and characteristics of the mine materials likely to be produced by the Project;
- Develop a preliminary mine waste plan and site water management plan at the next technical assessment stage of the Project;
- Carry out preliminary geotechnical investigations in the area of the proposed open pit;
- Carry out a preliminary hydrogeological investigation and modelling study for the Project;
- Review the envisaged Project with regulatory authorities including possible environmental and social impact assessment study requirements and related public consultation aspects, time lines, etc. and consider proactively commencing studies that are likely to be required or that may require an extended time whilst also recalling that environmental assessment supporting studies requirements are established as part of the environmental impact assessment process;
- Investigate and negotiate preliminary commercial parameters of key project components such as power supply, fuel and grinding media and key reagents;
- The flotation behaviour of partially altered feed material in the mill is inferior to primary material and future test work should assess the volume and characteristics of the potential for partially altered feed material at Murray Brook.

P&E also recommends that other exploration targets in the area continue to be identified and investigated to provide supplemental mill feed in the future.

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 TERMS OF REFERENCE

At the request of Votorantim Metals Canada Inc. (“VMC”), P&E Mining Consultants Inc. (“P&E”) have been retained to prepare a National Instrument 43-101 (NI 43-101) compliant technical report for the Murray Brook Project, New Brunswick, Canada. The purpose of the Technical Report is to update the mineral resource estimate and complete a Preliminary Economic Assessment for this copper-lead-zinc-silver-gold-bearing massive sulphide deposit. At the time of writing, the Murray Brook Project is held by a Joint Venture of 65% VMC and 35% El Nino Ventures Inc. (MBJV)

VMC is a subsidiary of Votorantim Metais, a company that is part of the Votorantim Group, which was founded in Brazil in 1918 and now operates in over twenty countries. Votorantim Metais is the largest electrolytic producer in Latin America and one of the world’s leaders in production of zinc and aluminum.

The address of VMC is:
Votorantim Metals Canada Inc.
Suite 1330, 4 King St. W. Toronto,
Ontario, Canada M5H 1B6.
The company is a non-issuer for purposes of security laws in Canada.

This report is considered current as of June 4, 2013.

Mr. Eugene Puritch, P.Eng. of P&E, a Qualified Person (“QP”) under the terms of NI 43-101 conducted a site visit to the Property on March 18, 2013. An independent verification of the MBJV sampling program was conducted previously by Mr. Gerald Harron, P.Eng., who visited on October 15 and 16, 2012.

In addition to the site visit, P&E carried out a study of relevant parts of the available literature on documented results concerning the project, and held discussions with technical personnel from the company regarding pertinent aspects of the project. The reader is referred to these data sources that are outlined in the References section of this report for further details on the project.

The purpose of the current report is to provide an independent Technical Report and Resource Estimate and Preliminary Economic Assessment of the base metal and precious metal mineralization present at the Murray Brook Project in conformance with the standards required by NI 43-101 and Form 43-101F. The estimate of Mineral Resources contained in this report conforms to the CIM Mineral Resource and Mineral Reserve definitions referred to in National Instrument (NI) 43-101 Standards of Disclosure for Mineral Projects.

2.2 SOURCES OF INFORMATION

This report is based in part on internal company technical reports and maps, published government technical reports, published scientific papers, company letters and memoranda, and public information listed in Section 27.0 “References” at the conclusion of this report. Several sections from reports authored by other consultants have been directly quoted or summarized in this report and indicated as such in the appropriate sections. P&E held discussions with technical personnel from the company regarding pertinent aspects of the project. P&E has not conducted detailed land status evaluations, and has relied on previous qualified reports, public documents and statements by VMC management regarding the Property status and legal title to the Project.

The present Technical Report is prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101) and in compliance with Form NI 43-101F1 of the Ontario Securities Commission (OSC) and the Canadian Securities Administrators (CSA). The Resource Estimate is prepared in compliance with the CIM Definitions and Standards on Mineral Resources and Mineral Reserves that are in force as of the effective date of this report.

2.3 UNITS AND CURRENCY

Unless otherwise stated all units used in this report are metric. Gold assay values (Au) are reported in grams per tonne of metal ("g/t Au") unless ounces per tonne ("oz/T Au") are specifically stated. The C$ is used throughout this report unless the US$ is specifically stated. At the time of this report the rate of exchange between the US$ and the C$ is 1 US$ = 1.00 C$.

The following list shows the meaning of the abbreviations for technical terms used throughout the text of this report.

Abbreviation	**Meaning**
$C	Canadian dollar
Ag	silver
ANFO	ammonium nitrate/fuel oil
As	Arsenic
AOI	Area of Interest
Au	gold
BOJV	Bathurst Option Joint Venture
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre
CSA	Canadian Securities Administrators
Cu	copper
DDH	diamond drill hole
DMT	Dry Metric Tonne
ELN	El Nino Ventures Inc.
EPCM	Engineering, Procurement and Construction Management
G&A	General and Administration
g/t or gm/t	grams per tonne
GAHA	G.A. Harron & Associates Inc.
g or gm	gram
ha	hectare(s)
HLEM	Horizontal-Loop Electromagnetic (system)
IP/RES	Induced Polarization/Resistivity survey
K	thousands

kg	kilogram
kg/t	kilograms per tonne
km	kilometre(s)
kV	kilovolts
Kwh or kWh	kilowatt hour
$1/d^2$	inverse distance squared method
lbs/ton	pounds per short ton
LCT	Locked Cycle Test
LOM	Life-of-mine
m	metre(s)
M	millions
m3	cubic metres
Ma	millions of years
masl	meters above sea level
MBJV	Murray Brook Joint Venture
mm	millimetre
Mt	Millions of tonnes
NB	Province of New Brunswick
NI 43-101	National Instrument 43-101
NN	Nearest Neighbour method
NSR	Net Smelter Return
nsv	no significant values
OSC	Ontario Securities Commission
oz	troy ounce
oz/T Au	troy ounces of gold per metric tonne
P&E	P&E Mining Consultants Inc.
Pb	lead
PEA	Preliminary Economic Assessment
ppb	part per billion
ppm	part per million
Sb	Antimony
t	tonne
ton	Short ton (2,000 lbs.)
t/m3	tonnes per cubic meter
TMF	Tailings Management Facility
TSL	TSL Laboratories Inc.
US$	United States dollars
VMC	Votorantim Metals Canada Inc.
WMT	Wet Metric Tonne
XZN	Xstrata Canada Corporation-Xstrata Zinc Canada Division
Zn	Zinc

3.0 RELIANCE ON OTHER EXPERTS

P&E has also relied upon information received from Neil Seldon of Neil S. Seldon & Associates Ltd., with respect to the marketability of potential concentrate production from the Murray Brook project. Neil S. Seldon & Associates Ltd provides marketing consulting advice with respect to various metals, minerals, ores, concentrates and other resource products, with particular emphasis on copper, zinc, lead and precious metal concentrates.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The property is located approximately 60 km west of Bathurst in the Parish of Balmoral, Restigouche County, Province of New Brunswick, Canada (Figure 4.1).

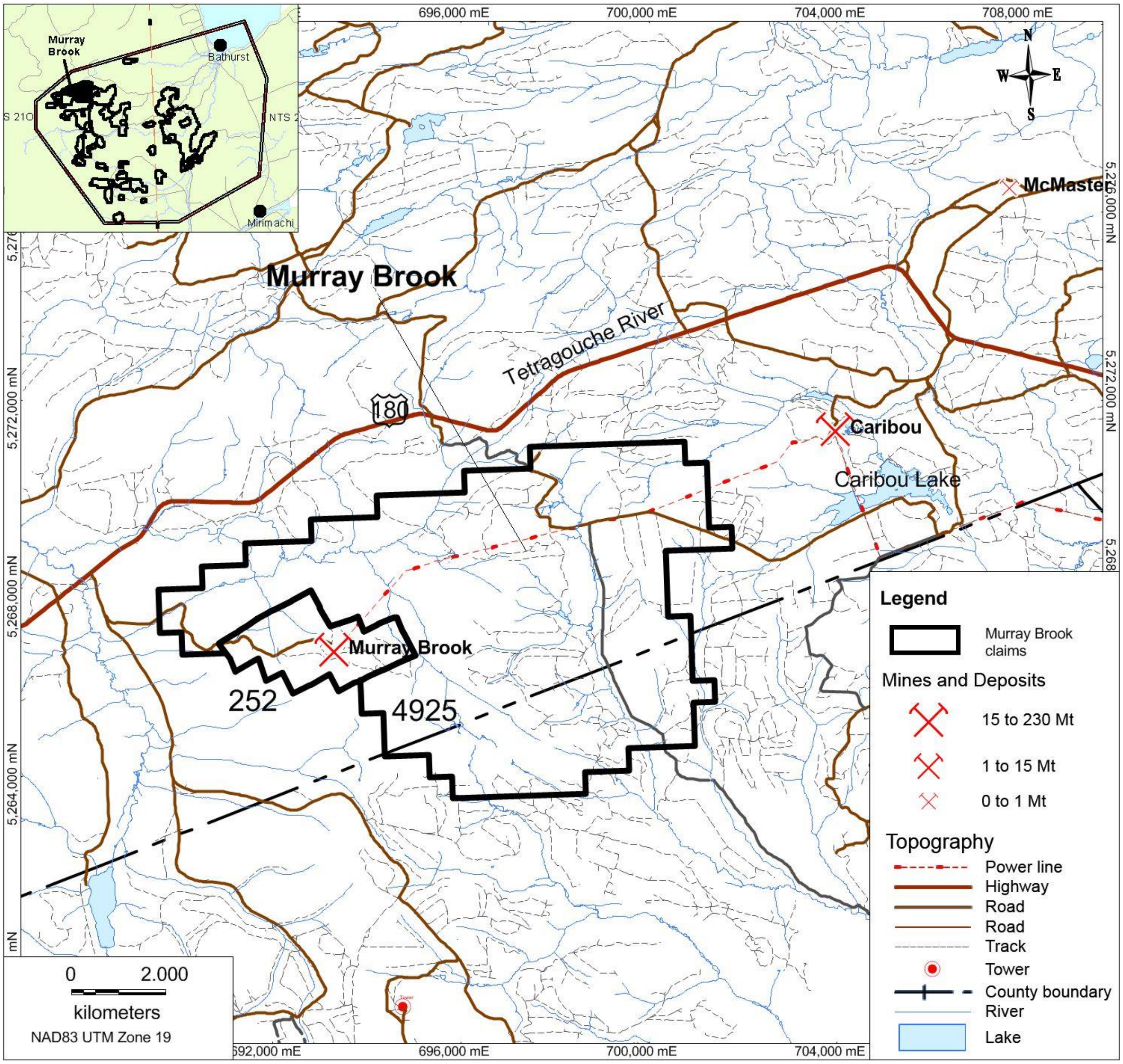
Figure 4.1 General Location Map, Murray Brook Property

4.2 PROPERTY DESCRIPTION AND TENURE

The property consists of surveyed Mineral Lease # 252, which covers approximately 482 ha and is illustrated in Figure 4.2. This lease was recorded on October 17, 1989 by Murray Brook

Resources Inc. The initial term is for 20 years with three automatic twenty year renewals. The current expiry date is October 16, 2029 and the rental fees are current. The annual fee is C$3,030.

Figure 4.2 Property Map, Murray Brook Property



The Murray Brook property is situated within the Area of Interest ("AOI") as defined by the Bathurst Option and Joint Venture ("BOJV") Agreement made as of March 24th, 2010 between Xstrata Canada Corporation-Xstrata Zinc Canada Division ("XZN"), ELN and VMC, as subsequently amended by an Amending Agreement dated September 30th, 2010 ("the BOJV Amending Agreement"). The BOJV Amending agreement provides provisions, with respect to the rights of the other parties of the BOJV Agreement, which the acquiring party must adhere to when acquiring any property or mineral interest, within, or partly within, the BOJV AOI.

VMC optioned the Murray Brook property consisting of Mineral Lease 252 and claim number 4925, Murray Brook east (aka Camel Back claims) from Murray Brook Resources Inc. and Murray Brook Minerals Inc. ("Owners"), respectively on November 1, 2010.

The terms of the option and joint venture agreement between VMCI and the Owners stipulated that VMCI could earn a 50% interest in the Murray Brook property by making a series of option payments totalling $300,000 and total work expenditures of $2,250,000 on or before October 31, 2013. A further option exercisable within 60 days of obtaining the 50% interest permitted VMCI to acquire an additional 20% interest in the Murray Brook Property by incurring additional work expenditures within an additional two-year period.

Pursuant to the BOJV Amending Agreement VMC offered to both XZN and ELN the right to acquire 50% (25% each) of its right to earn 70% in the Murray Brook Property. XZN declined participation and ELN subsequently agreed to participate including with respect to the 25% share which had been declined by XZN. On January 3rd, 2011 VCMI and ELN entered into a Participation Agreement whereby ELN could obtain the right to acquire 50% of the interest in the Murray Brook Property acquired by VMC.

On April 1st, 2012 VMC, ELN and the Owners executed an Acknowledgement of Earned Interest and Joint Venture Formation as VMC had expended in excess of $2,250,000 and had made the requisite $300,000 in option payments to earn an interest in the Murray Brook Property. Subsequently, on April 5th, 2012 VMC provided notice to the Owners of its intention to earn an additional 20% by way of incurring additional work expenditures of $2,250,000 within a two-year period.

On August 27th, 2012 VMC provided notice to ELN that VMC had received the final earn in amount payable by ELN to earn 50% of the additional 20% interest acquired by VMC in the Murray Brook Property under the terms of the Participation Agreement dated January 3rd, 2011 between VMC and ELN and pursuant to the Murray Brook Project Joint Venture which was formed on April 1st, 2012 the parties interest in the project were VMC 35%, ELN 35% and the Owners 30%.

On August 28th, 2013 VMC entered into a purchase agreement between VMC and the Owners to purchase the Owner's remaining interest in the Murray Brook Property. The purchase agreement provides for a series of staged payments over five years totalling $6,000,000. The first payment of $1,000,000 was due upon execution and has been paid. The second payment of $1,000,000 is due upon completion of a pre-feasibility study or December 31, 2013, whichever is earlier. The third payment of $1,000,000 is due upon the completion of a feasibility study or December 31, 2015, whichever is earlier. The fourth and final payment of $3,000,000 is due upon the earlier circumstance of either the project reaching commercial production or December 31, 2017. The Owners retain a 0.25% NSR payable from the first anniversary of commercial production to the end of the life of the mine. In addition VMC agreed to pay and satisfy the Owners proportionate and outstanding joint venture expenditures of $216,388.11 for the period ending on October 5th, 2012, which was the closing date for the aforementioned Purchase Agreement.

Pursuant to the Amending Agreement of September 30th, 2010 ELN was provided with notice by VMC of the terms of the Purchase Agreement. ELN subsequently indicated that it would acquire 50% of the Acquired Interest by the Purchase Agreement in a letter dated September 26th, 2012 but did not pay to exercise the option within the option period stipulated by the Amending Agreement of September 30th, 2010 and consequently at the time of writing the Joint Venture (MBJV) remains at VMC 65%: ELN 35%.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The property is located approximately 60 km west of the town of Bathurst, New Brunswick. At kilometre 60 (measured from Bathurst), a 5 km gravel road extends southward from highway 180 to the mine site. Highway 180 continues westward to Sainte Quentin, New Brunswick. Bathurst to the east, provides access to rail and ocean shipping facilities.

Several communities in the region offer commercial goods, social, educational and financial amenities, as well a pool of skilled labour.

The climate of the area is a typical boreal forest ecosystem, with forests of coniferous trees and mixed hardwood trees. Climatic conditions are influenced by the Atlantic Ocean, which includes winter months with 1-2 m of snow cover and sub-zero temperatures. Summer conditions are typically moist and warm with rain showers and temperatures in the 20s Celsius extending from May through September.

Physiographically, the property is located in the Miramichi Highlands, characterized by rounded, glacially scoured hills. Topographic maps show a broad plateau in the east at approximately 630 m with deeply incised water courses reaching down to about 490 m in the western portion of the area. Drainage in the area is eastward towards the Atlantic Ocean. Land use in the area is mainly for tourism, forestry and mining.

6.0 HISTORY

The Murray Brook claim group was staked originally by Kennco Explorations in 1955 to cover seven airborne electromagnetic anomalies. Ground follow-up of the anomalies, however, proved that the electromagnetic responses were caused by graphitic sedimentary rocks rather than sulphide mineralization. In 1956 an "intermediate lava" float assaying 1.35% Cu was discovered in the western half of the claim group (Perusse, 1957), which lead to further exploration. Ground geophysical surveys missed the Murray Brook Deposit because there was no airborne survey immediately over the deposit. Field determinations of heavy metal contents of active steam and bank sediments pinpointed an anomaly source at the head of a small creek called Gossan Creek.

Subsequent trenching outlined an area of gossan measuring 760 m by 120 m. Packsack drilling failed to intersect fresh sulphides below the gossan. A HLEM survey was carried out to determine if any part of the gossan was underlain by massive sulphides. Results indicated that massive sulphide lenses were present.

In 1956 a drill hole intersected 89 m of massive sulphides under a cover of 16 m of gossan. By 1958, Kennco had sufficient drilling to estimate a "reserve of 21.5 million tonnes of 2.81% combined Pb-Zn (Rennick, 1992), Perusse (1958) estimated a historical resource of 23.6 million ton of mineralization averaging 0.44% Cu, 0.86% Pb, 1.95% Zn and 31.2 g/t Ag.

The QP has not done sufficient work to classify the two historical estimates as current estimates. The Company is not treating the historical estimates as a current estimate. A current estimate of Inferred, Indicated and Measured Resources using CIM categories is presented in Section 14 of this technical report.

In 1970, the property was optioned to Cominco who drilled three holes which did not increase the tonnage.

In 1973 the property was optioned to Gowganda Silver Mines Limited. In 1974 Canex Placer Explorations Ltd. gained control of the deposit through exploration expenditures. An extensive drilling program was carried out to obtain material for metallurgical testing.

The property reverted to Kennco Explorations in 1979.

In 1985 Northumberland Mines Ltd. optioned the property primarily for the precious metals content of the gossan. Thirty-six drill holes and related metallurgical tests systematically tested the gossan. In 1986 a vat leaching process was approved by the Department of Natural Resources and Energy for gold and silver production.

In 1988 Northumberland Mines and the Murray Brook deposit were acquired by NovaGold Resources Ltd., and the vat leaching operation commenced commercial production in 1989. In 1992 mining activities related to the vat leaching of the gossan zone were discontinued and the pit and property reclaimed in 1996.

Starting in 1998 the primary sulphide historical "resources" was partitioned into four units (1) a Primary Copper Zone, (2) Secondary Copper Zone, (3) Zinc Zone, all hosted within (4) a Sulphide Envelope.

A summary of the NovaGold (1998) historical resource estimations follows in Table 6.1.

TABLE 6.1 NOVAGOLD HISTORICAL RESOURCE ESTIMATES							
	Cut-Off	**Millions tonnes**	**Cu %**	**Pb %**	**Zn %**	**Au (g/t)**	**Ag (g/t)**
Primary Cu Zone	2%	0.75	2.81	0.35	0.81	0.41	29.9
Secondary Cu Zone	2%	0.35	3.28	0.26	0.72	0.07	54.2
Zinc Zone	5%	1.61	0.22	2.39	6.13	0.85	79.1
Gossan Zone (remaining)	1 g/t Au	0.39	0.12	1.63	0.04	1.51	46.5
Total Sulphide Envelope	**n.a.**	**20.2**	**0.29**	**0.57**	**1.32**	**0.32**	**25.2**

Data from Derosier 2008

The QP has not done sufficient work to classify any of these historical estimates as current estimates. The Company is not treating any of these historical estimates as a current estimate. A current estimate of Inferred, Indicated and Measured Resources using CIM categories is presented in Section 14 of this technical report.

In 2007 Murray Brook re-sampled 645.65 m of NovaGold core for Cu, Pb, Zn, Au and Ag. The assays indicated comparable Cu and Pb values slightly elevated Zn values and a 10% decrease in Ag values compared to those previously reported. This indicates that the historical NovaGold resource estimations made in 1998 are credible.

In January 2008, GEOSTAT Systems International Inc. completed a study of open pit exploitation of the copper mineralization (Desrosiers, C., 2008). It was noted that Cu grade decreases with depth, and Pb, Zn and Au values increase with depth. The estimated Mineral Resources using the Inverse Squared Distance Interpolation method indicated a resource of 2,087,000 tonnes averaging 2.04% Cu, 0.44% Pb, 1.10% Zn, 0.26 g/t Au and 45.54 g/t Ag using a 1% cut-off for Cu only.

The QP has not done sufficient work to classify the historical estimate as a current estimate. The Company is not treating the historical estimate as a current estimate. A current estimate of Inferred, Indicated and Measured Resources using CIM Categories is presented in Section 14 of this technical report.

In 2008 Murray Brook Minerals Inc. carried out a 42 line-km Magnetic Survey and a 20.8 line-km Induced Polarization/Resistivity survey (“IP/RES”). The magnetic survey delineated the volcanic rocks of the Boucher Brook Formation. The IP/RES survey delineated a 900 m long conductive anomaly with a positive chargeability and a low resistivity response. The response is comparable with the response of massive sulphides below the open pit, and appears to be a southwest extension of the known massive sulphide zone. The IP/RES response is validated by the presence of a gravimetric anomaly and a soil geochemical copper anomaly.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The Murray Brook area is located in the Bathurst Mining Camp (BMC) in northern New Brunswick. The BMC is an Ordovician back-arc complex of polydeformed sedimentary, felsic volcanic and mafic volcanic rocks formed in separate sub-basins within the back-arc basin, and which have been juxtaposed by five periods of folding and thrusting, and collectively referred to as the Bathurst Supergroup. The sedimentary and volcanic rocks have been intruded by gabbro, diabase and quartz porphyritic rocks of Ordovician age.

The BMC hosts at least 46 volcanogenic massive sulphide deposits with a total sulphide resource of over 500 million tonnes (McCutcheon and Walker, 2009). The camp also hosts the world famous Brunswick No. 12 mine, which to the end of 2007 has produced 123,600,000 tonnes grading 3.5% Pb, 8.8% Zn, 0.36% Cu and 103g/t Ag. The mine is being de-commissioned at the time of writing.

The massive sulphide deposits of the Bathurst Camp occupy more than one stratigraphic position; 32 are in the Tetagouche Group and 13 occur in the possibly coeval California Lake Group. Within the Tetagouche Group, massive sulphide deposits are largely concentrated in the first volcanic cycle, represented by crystal tuffs of the Nepisiguit Falls Formation. Most are hosted by chloritic mudstones at or near the top of this formation ("Brunswick Horizon") and are associated with oxide facies iron formation. The Murray Brook deposit is hosted by sedimentary rocks of the Mount Brittain Formation in the California Lake Group (which is probably coeval with the Tetagouche Group) and has no associated iron formation.

7.1 REGIONAL GEOLOGY AND MINERALIZATION

The Bathurst Mining Camp is underlain by Cambro-Ordovician age rocks of the Bathurst Supergroup.

The New Brunswick Ministry of Energy and Mines' Bedrock Lexicon (DNR) states that the Bathurst Supergroup "comprises felsic to mafic volcanic and sedimentary rocks of the Fournier, California Lake, Tetagouche and Sheephouse Brook groups, in order of highest to lowest structural level (Figure 7.1). The various groups were juxtaposed by thrusting and internally imbricated into thrust nappes during their successive incorporation into the Brunswick subduction complex.

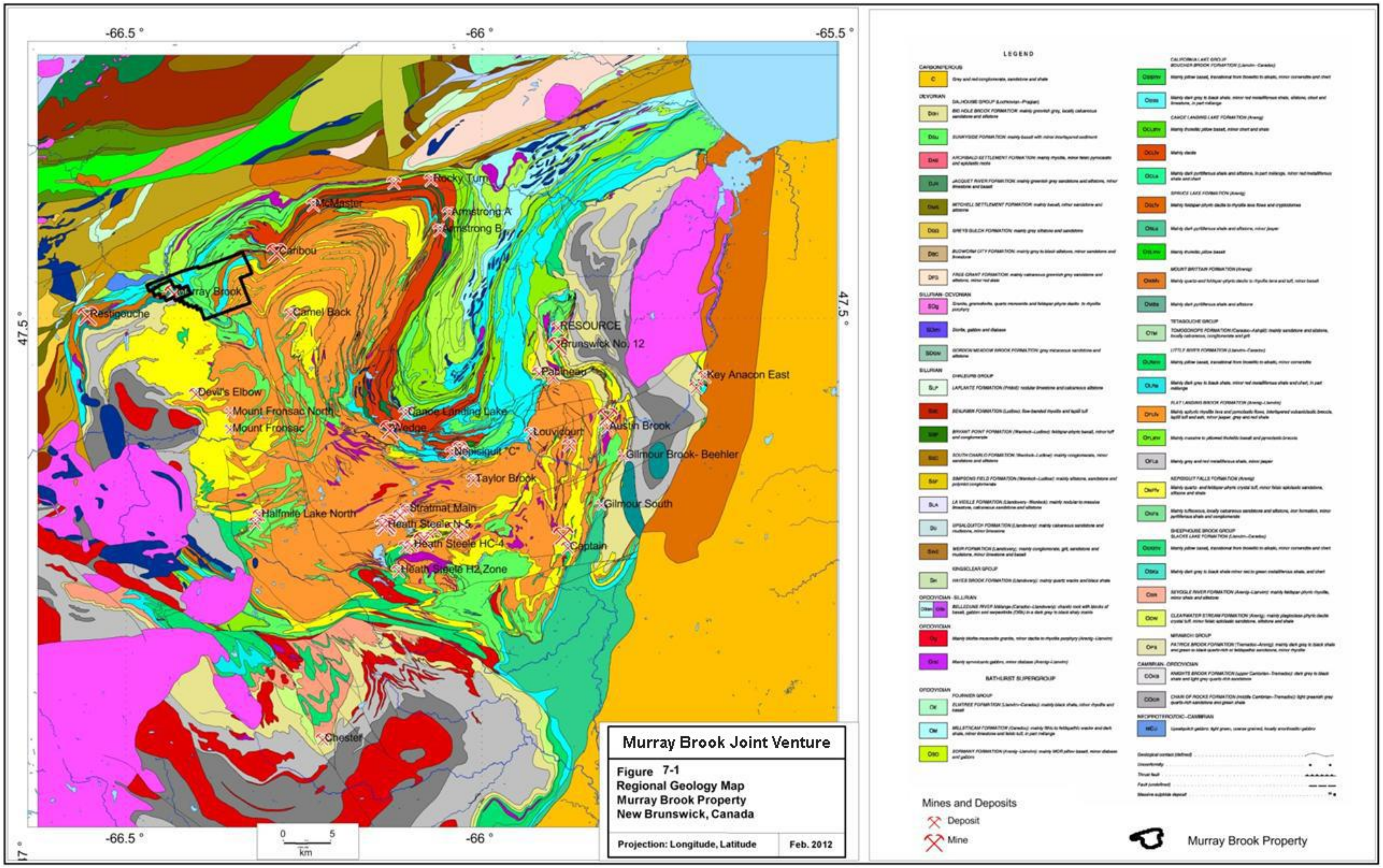
The Bathurst Supergroup encompasses all Ordovician volcanic and sedimentary rocks overlying the Miramichi Group in the Bathurst Mining Camp, a roughly circular area 70 km in diameter in the northern Miramichi Highlands, as well as Ordovician rocks of the Elmtree Inlier, an elliptical area measuring about 25 x 15 km on the shore of Chaleur Bay.

Rocks of the Bathurst Supergroup lie conformably to disconformably on the Miramichi Group, and are unconformably overlain by, or in fault contact with, Silurian rocks of the Chaleurs Group to the north and west, and the Silurian Kingsclear Group and Carboniferous Mabou and Pictou groups to the east.

Cambro-Ordovician aged rocks in the Bathurst Camp have undergone five episodes of regional deformation. Two structural domains are recognized: (1) a flat belt in the south and west parts of the Camp characterized by recumbent or overturned F2 folds; and (2) a steep belt in the north and east in which F2 folds are upright. Thrusting related to closure of the Iapetus back arc basin (van Staal, 1987) and regional faulting has also affected the present distribution of major stratigraphic units.

The Murray Brook Mineral Lease 252 and adjoining claim 4925, overlie a structurally compressed area that juxtaposes formations and members of the Miramichi, Tetagouche, California Lake and Fournier Groups.

Figure 7.1 Regional Geology Map, Murray Brook Property



Source: *van Staal et al., 2003.*

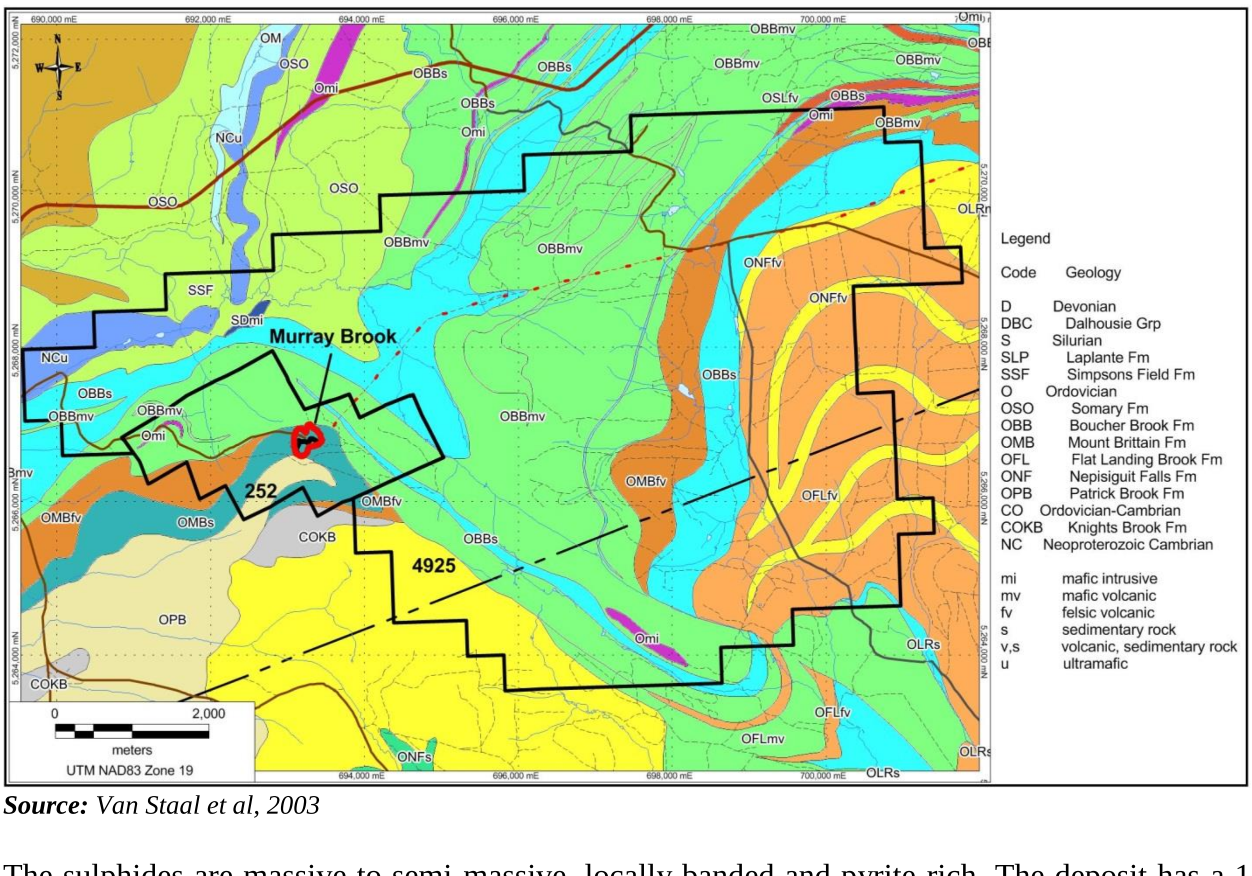
7.2 PROPERTY GEOLOGY AND MINERALIZATION

The geology of the Murray Brook area was mapped by Gower and van Staal (1997) and incorporated into a regional geological map of the Bathurst Mining Camp shown in Figure 7.1 (van Staal et al, 2003). Figure 7.2 shows a detail of that map.

The Murray Brook deposit is hosted by sedimentary rocks of the Charlotte Brook Member in the lower part of the Mount Brittain Formation. The upper felsic volcanic member of the Mount Brittain Formation is host to the Restigouche deposit, some 10 km to the west. The Mount Brittain Formation is believed to be equivalent of the Spruce Lake Formation which hosts the Caribou Mine, ten kilometres to the east.

The Murray Brook deposit dips moderately to the west, plunges gently to the north and appears to pinch-out at depth and to the east. The geometry of the deposit was probably lens-shaped, but the up-dip portion of the body has been eroded and pre-Pleistocene weathering has produced a gossan.
While the deposit is a single body of massive sulphide, drilling performed by Votorantim has indicated that it comprises two connected thick lenses or lobes, the western lense being richer in zinc and lead, and the eastern lense richer in copper.

Figure 7.2 Local Geology, Murray Brook Property



Source: *Van Staal et al, 2003*

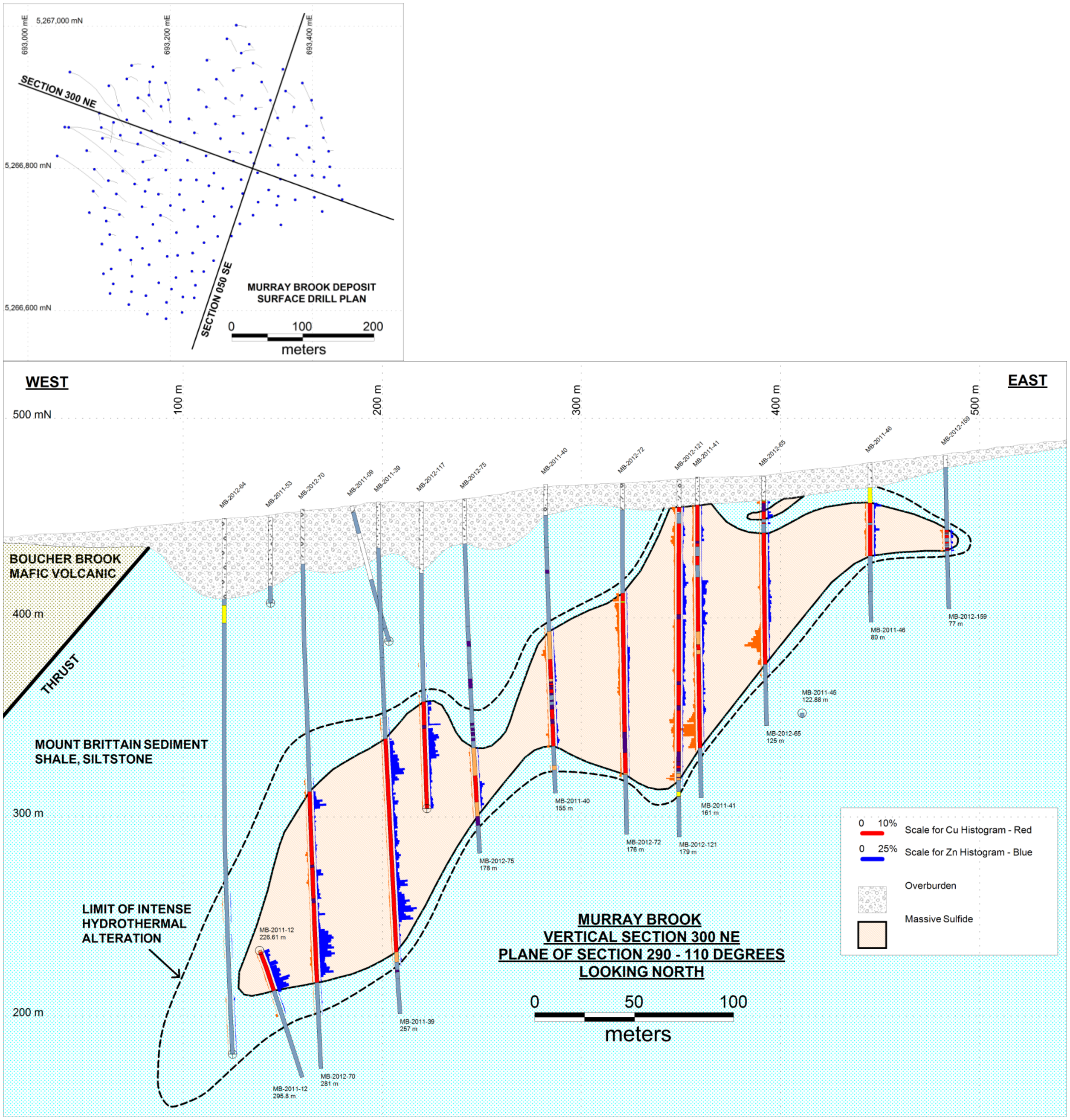
The sulphides are massive to semi-massive, locally banded and pyrite-rich. The deposit has a 1 to 3 m wide halo composed of chloritized sedimentary rocks containing disseminated pyrite. The hanging wall is moderately chloritic and is locally intensely deformed. The footwall consists of fine grained felsic tuff, and tuffaceous sediments with moderate to strong chlorite and sericite alteration.

Sulphides are mainly fine grained, massive, vaguely laminated pyrite with disseminated and banded sphalerite, chalcopyrite and galena, with minor tetrahedrite, covellite, marcasite and arsenopyrite.

The gossan zone capping the sulphide zones has been more or less completely mined out, and is not further discussed in this report.

Metal zoning indicated by drill hole assays allows division of the sulphides into copper, pyrite, lead-zinc zones (Figure 7.3).

Figure 7.3 Local Geology Vertical Section, Murray Brook Property



8.0 DEPOSIT TYPE

The Murray Brook sulphide mineralization is classified as a volcanogenic massive sulphide deposit (P., W., C., Shanks III, et al) 2009. This type of deposit is characterized by massive to semi-massive iron sulphide minerals including variable amounts of base metals and precious metals. This type of deposit is well studied and documented. Genetically the deposits are coeval with felsic volcanic centres, and are generally lens-like, parallel to the stratigraphy, with a discordant hydrothermal "pipe" at the stratigraphic base of the sulphide accumulation.

As is well illustrated by the discovery of the Murray Brook deposit, heavy mineral and soil geochemical surveys are an effective search tools. Geophysical surveys including magnetic, electromagnetic, induced polarization/ resistivity, and gravity surveys are also effective exploration tools for concealed deposits.

9.0 EXPLORATION

In 2010 and 2011, VMC carried out a Fugro Airborne Gravity Gradiometry survey and a Fugro HeliTEM electromagnetic survey over their various properties and areas of interest, including the Murray Brook deposit. The ground geophysical portion of the exploration consists of a gravimetric survey, which detected the sulphide mass at depth.

The results of these surveys have not been reviewed in detail by the authors.

10.0 DRILLING

VMC's drilling at Murray Brook commenced in 2010 with the drilling of four 'due diligence' holes totalling 595.2 m (hole MB-10-14 was abandoned at 39 m). These holes were consistent with historical results with significant intersections of zinc, copper, lead, gold and silver were reported. VMC duly finalized its Agreement with the Owners.

In 2011 63 vertical drill holes totalling 10,499.4 m were drilled. The results were announced in ELN news releases (August 30, 2011, November 28, 2011, January 16, 2012 and January 23, 2012). The drill hole specifications for the 2010-2011 drilling are presented in Table 10.1. The composite assay results are presented in Table 10.2.

TABLE 10.1
DIAMOND DRILL SPECIFICATIONS PHASE I AND PHASE II(1)(2)

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Inclination (°)	Length (m)
MB-10-15	693051.7	5266858	449.3162	90	-60	276
MB-10-16	693213.0	5266763	469.3306	360	-90	164.2
MB-10-17	693301.8	5266765	470.1422	360	-90	116
MB-2011-01	693246.1	5266753	469.4524	360	-90	125
MB-2011-02	693186.2	5266734	469.7448	110	-75	197
MB-2011-03	693328.6	5266854	471.6884	360	-90	206
MB-2011-04	693300.8	5266903	467.0972	360	-90	233.6
MB-2011-05	693283.6	5266824	468.2612	360	-90	177
MB-2011-06	693416.3	5266788	479.2408	360	-90	107
MB-2011-07	693395.2	5266829	477.5667	360	-90	167
MB-2011-08	693351.8	5266761	471.3537	360	-90	92
MB-2011-09	693158.6	5266850	455.8107	120	-75	179.1
MB-2011-10	693197.5	5266897	456.4858	360	-90	245
MB-2011-11	693193.2	5266921	451.0143	360	-90	277
MB-2011-12	693059.1	5266935	443.6026	110	-70	295.8
MB-2011-13	693210.4	5266678	470.0887	360	-90	153
MB-2011-14	693057.4	5266858	449.6076	110	-70	305
MB-2011-15	693266.7	5266705	470.6467	360	-90	98
MB-2011-16	693041.1	5266817	457.2328	110	-75	272
MB-2011-17	693160.9	5266688	473.44	360	-90	182
MB-2011-18	693170.3	5266665	474.5566	360	-90	131
MB-2011-19	693146.6	5266649	478.8235	360	-90	100
MB-2011-20	693195.8	5266676	471.1625	360	-90	152
MB-2011-21	693167.9	5266595	482.5402	360	-90	63.5
MB-2011-22	693208.2	5266647	472.6848	360	-90	119
MB-2011-23	693208.6	5266710	469.7802	360	-90	155
MB-2011-24	693247.6	5266696	470.1636	360	-90	98
MB-2011-25	693261.0	5266670	471.1106	360	-90	74
MB-2011-26	693153.2	5266704	473.1269	360	-90	173
MB-2011-27	693238.6	5266718	469.581	360	-90	86
MB-2011-28	693261.8	5266733	470.0341	360	-90	72
MB-2011-29	693286.3	5266746	470.3547	360	-90	86
MB-2011-30	693236.7	5266784	470.6409	360	-90	128
MB-2011-31	693160.2	5266746	479.836	360	-90	218
MB-2011-33	693169.6	5266784	475.8033	360	-90	243
MB-2011-34	693113.1	5266783	477.1142	360	-90	245
MB-2011-37	693161.5	5266809	470.4897	360	-90	251
MB-2011-38	693261.4	5266794	469.1051	360	-90	152.5
MB-2011-39	693173.7	5266853	457.96	360	-90	257

TABLE 10.1 DIAMOND DRILL SPECIFICATIONS PHASE I AND PHASE II(1)(2)						
Hole ID	**Easting (m)**	**Northing (m)**	**Elevation (m)**	**Azimuth (°)**	**Inclination (°)**	**Length (m)**
MB-2011-40	693250.3	5266818	466.8232	360	-90	155
MB-2011-41	693322.8	5266794	470.605	360	-90	161
MB-2011-42	693298	5266722	470.7863	360	-90	75
MB-2011-43	693340.3	5266832	473.05	360	-90	170
MB-2011-44	693351.9	5266805	472.7825	360	-90	143
MB-2011-45	693375.7	5266790	474.9985	360	-90	125
MB-2011-46	693402.5	5266760	477.7486	360	-90	80
MB-2011-47	693413.6	5266740	479.7626	360	-90	75
MB-2011-48	693193.6	5266798	468.892	360	-90	182
MB-2011-49	693183.5	5266764	473.796	360	-90	227
MB-2011-50	693150	5266778	476.79	360	-90	251
MB-2011-51	693133.2	5266797	472.067	360	-90	245
MB-2011-52	693123	5266824	464.54	360	-90	257
MB-2011-53	693120.2	5266865	450.281	200	-85	233
MB-2011-54	693200.2	5266866	455.924	360	-90	251
MB-2011-55	693263.4	5266863	463.544	360	-90	191
MB-2011-56	693349.9	5266749	471.227	360	-90	75
MB-2011-57	693162.7	5266881	451.73	360	-90	266
MB-2011-58	693129.3	5266685	474.855	360	-90	140
MB-2011-59	693183.9	5266639	472.838	360	-90	116
MB-2011-60	693230.5	5266656	469.118	360	-90	141
MB-2011-61	693128.5	5266735	484.121	360	-90	209
MB-2011-62	693124.3	5266764	477.198	360	-90	233
MB-2011-63	693170.1	5266903	452.595	360	-90	272

(1) Coordinates are in UTM NAD83 Zone 19;
(2) Drill holes MB-2011-32, MB-2011-35 and MB-2011-36 were abandoned.

TABLE 10.2 SIGNIFICANT PHASE I AND PHASE II DRILL INTERCEPTS								
Hole ID	**From (m)**	**To (m)**	**Width (m)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Au (g/t)**	**Ag (g/t)**
MB-2010-15	163.8	167.7	3.9	0.10	2.00	9.16	1.6	77.00
and	201.0	218.0	17.0	0.10	4.24	7.51	0.96	84.0
and	232.0	235.0	3.0	0.13	2.65	6.25	1.02	58.0
MB-2010-16	44.0	71.0	27.0	0.47	3.39	9.56	0.21	122.0
and	79.0	91.0	12.0	0.08	2.22	6.89	0.10	89.0
MB-2010-17	15.0	95.0	80.0	0.33	0.63	1.28	0.65	26.32
MB-2011-01	53.0	77.8	24.8	0.11	1.22	3.34	0.20	41.00
MB-2011-02	42.5	164.3	121.8	0.26	1.07	3.32	0.26	34.61
MB-2011-03	92.0	172.6	80.6	0.83	0.98	1.89	0.80	47.06
MB-2011-04	141.0	217.0	76.0	0.53	1.21	2.35	1.31	45.28
MB-2011-05	73.5	155.0	81.5	1.12	0.44	1.02	0.46	20.97
MB-2011-06	47.4	77.7	30.3	0.26	0.99	2.13	0.90	46.31
MB-2011-07	112.5	138.1	25.6	0.34	1.05	2.28	0.92	46.52
MB-2011-08	11.0	64.2	53.2	0.30	0.80	1.74	1.04	39.66
MB-2011-09	101.0	176.1	75.1	0.13	1.43	3.84	0.36	59.37
MB-2011-10	179.5	216.0	36.5	0.25	1.90	4.65	1.00	69.86
MB-2011-11	187.6	201.8	14.2	0.20	1.52	3.73	0.79	52.94
MB-2011-13	27.0	126.0	99.3	0.22	1.16	3.38	0.62	40.59
MB-2011-14	160.5	225.0	64.5	0.23	0.78	3.87	0.66	35.5
MB-2011-15	29.0	35.3	6.3	0.16	1.21	3.84	0.11	8.38
MB-2011-17	24.1	126.65	102.6	0.65	0.47	1.84	0.20	23.65

Table 10.2 Significant Phase I and Phase II Drill Intercepts								
Hole ID	**From (m)**	**To (m)**	**Width (m)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Au (g/t)**	**Ag (g/t)**
MB-2011-18	47.0	107.0	60.0	1.01	0.04	0.19	0.20	11.95
MB-2011-19	23.0	77.0	54.0	0.40	0.43	1.14	0.86	22.29
MB-2011-20	15.0	125.0	110.0	0.32	0.71	2.41	0.25	27.34
MB-2011-21	19.65	31.6	12.0	0.90	0.04	0.15	0.15	10.72
MB-2011-22	17.6	95.2	77.6	0.29	0.81	2.42	0.44	32.96
MB-2011-23	31.5	107.0	75.5	0.38	0.68	2.16	0.30	24.69
MB-2011-24	38.0	55.9	17.9	0.08	0.43	0.68	0.03	8.56
MB-2011-25	nsv							
MB-2011-26	29.0	142.7	113.7	0.31	0.26	1.19	0.26	18.94
MB-2011-27	38.0	69.5	31.5	0.51	0.20	0.63	0.04	7.82
MB-2011-28	38.0	42.5	4.5	0.34	0.20	0.63	0.04	7.82
MB-2011-29	21.0	57.3	36.3	0.19	0.92	1.90	0.80	33.39
MB-2011-30	44.0	103.0	59.0	0.14	1.55	4.58	0.51	68.15
MB-2011-31	53.0	193.3	140.3	0.32	1.03	3.73	0.27	43.24
MB-2011-33	59.0	215.1	156.1	0.23	0.85	2.64	0.41	29.94
MB-2011-34	129.6	212.0	82.4	0.13	1.19	5.05	0.3	44.03
MB-2011-37	88.0	234.4	146.4	0.16	1.33	3.83	0.45	49.2
MB-2011-38	46.10	111.6	65.54	0.59	0.40	0.84	0.78	21.5
MB-2011-39	118.9	222.0	103.1	0.11	1.81	5.45	0.51	65.7
MB-2011-40	83.0	131.0	48.0	0.33	0.41	0.93	0.69	21.2
MB-2011-41	14.0	136.0	122.0	0.89	0.73	1.58	0 .94	37.8
MB-2011-42	15.0	18.0	3.0	0.21	0.32	2.65	0.73	22.8
MB-2011-43	76.0	143.8	67.8	0.41	0.60	0.97	0.71	36.1
MB-2011-44	36.0	60.0	24.0	0.53	0.59	1.00	0.61	32.9
and	65.8	110.7	44.9	0.66	0.79	1.49	0.90	40.4
MB 2011-45	20.2	33.6	13.4	0.31	1.14	2.73	0.97	47.0
and	41.0	95.0	54.0	0.42	0.74	1.78	0.63	35.1
MB-2011-46	20.15	46.80	26.6	0.39	0.63	1.79	0.26	33.9
MB-2011-47	nsv							
MB-2011-48	60.5	161.0	100.5	0.16	1.71	4.65	0.36	56.5
MB-2011-49	35.0	181.0	146.0	0.59	1.40	3.85	0.63	56.1
MB-2011-50	55.0	223.1	168.1	0.28	1.12	3.62	0.38	41.6
MB-2011-51	83.7	220.8	137.1	0.35	0.73	2.23	0.53	28.5
MB-2011-52	134.5	145.0	10.5	0.19	0.06	0.53	0.07	3.8
and	159.5	231.7	72.2	0.26	2.33	5.61	0.71	77.7
MB-2011-53	170.0	204.8	34.8	0.47	0.20	0.59	0.13	13.2
MB-2011-54	156.2	201.0	44.8	0.17	1.55	4.26	0.70	60.7
MB-2011-55	102.0	149.2	47.2	0.99	0.39	0.79	0.37	16.0
and	153.2	155.7	2.47	0.67	0.05	0.09	0.09	5.3
MB-2011-56	15.2	43.0	27.8	0.22	0.35	1.08	0.29	17.0
MB-2011-57	143.3	231.0	87.7	0.14	2.77	7.23	0.61	103.3
MB-2011-58	23.0	72.0	49.0	0.45	0.31	2.02	0.38	23.2
and	98.1	105.0	6.9	1.09	0.05	0.17	0.13	6.9
MB-2011-59	24.50	88.0	63.5	0.47	0.26	1.03	0.21	19.5
MB-2011-60	21.5	54.0	32.5	0.89	0.09	0.44	0.08	6.2
MB-2011-61	80.2	178.0	98.8	.3	0.22	0.80	1.15	15.8
MB-2011-62	118.9	201.0	82.1	0.15	0.98	3.17	0.31	39.8
MB-2011-63	168.4	240.0	71.6	0.18	1.89	4.98	0.91	79.9

Three objectives of the Phase I and Phase II drilling program were realized: (1) infill drilling to close large (100 m) gaps in the historical drill coverage; (2) step-out drilling to define the margins of the deposit, and (3) due diligence drilling (595.2 m) to confirm results from historical drill programs. The results of the 2011 drilling provided additional data for use in estimating Indicated and Inferred Resources (see section 17.0). An analysis of the 2011 drill program indicated that approximately 18,000 m of additional infill and definition drilling was warranted in a 2012 drill program.

The objective of the 2012 drilling was to upgrade the Inferred and Indicated Resources to Measured Resources, define additional near-surface resources along the northwest margin of the deposit, as well as completing preliminary metallurgical testing on selected portions of the deposit. The drill program commenced in February 2012 and consisted of 99 vertical drill holes totalling approximately 18,264 m. The drill hole specifications are presented in Table 10.3. In the period 2010 to 2012, 166 drill holes have been drilled for a total of 29,718 m.

Analysis of the drilling results identified two distinct north-trending massive sulphide zones with different mineralogical characteristics and thicknesses. The western zone appears to be thicker and richer in Zn, Pb and Ag, whereas the eastern zone is thinner and richer in Cu-Au mineralization.

The location of ELN and historical and current drill collars are shown in Figure 10.1. Figure 10.2 illustrates the dimensions of the deposit. The massive sulphide portion of the deposit measures approximately 320 m north-south by approximately 300 m east-west with a thickness of 120 to 150 m as two north-south lobes. Typical cross sections, normal and perpendicular to the axes of deposit are presented in Figure 10.3, Figure 10.5 and Figure 10.5.

Figure 10.1 Location of Current and Historical Hole Collars

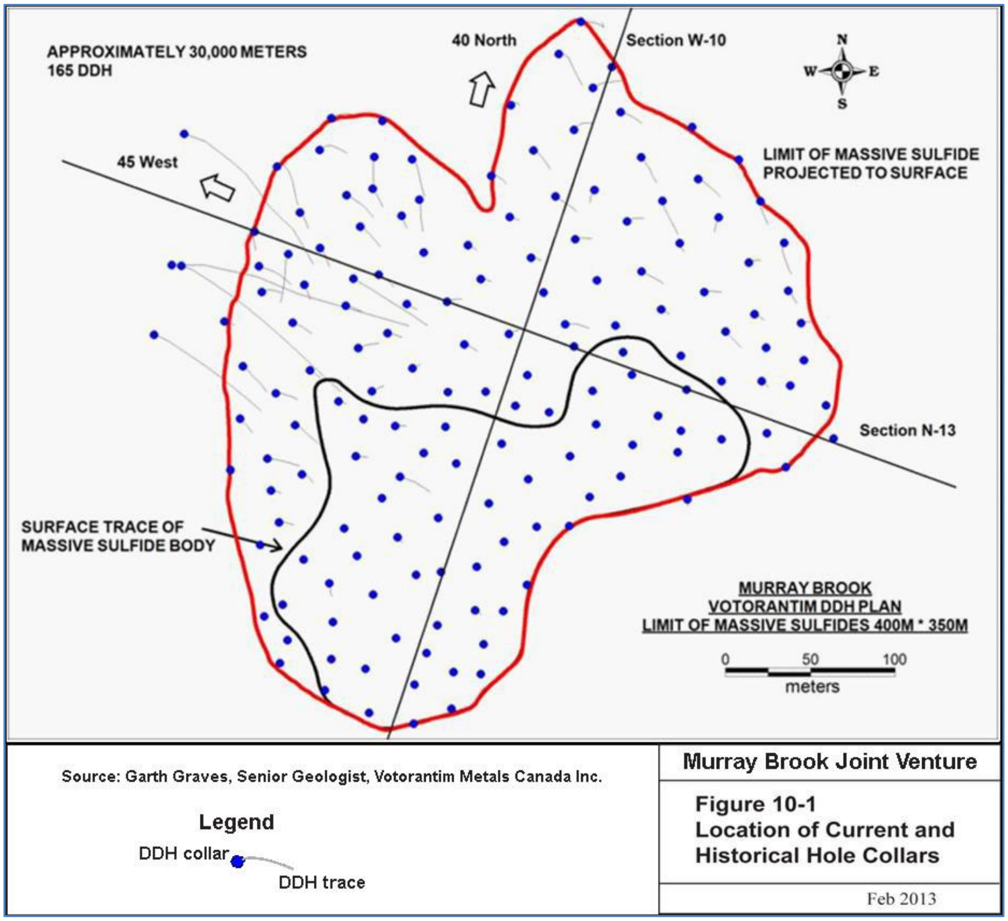


Figure 10.2 Deposit Thickness, 10 m Contours

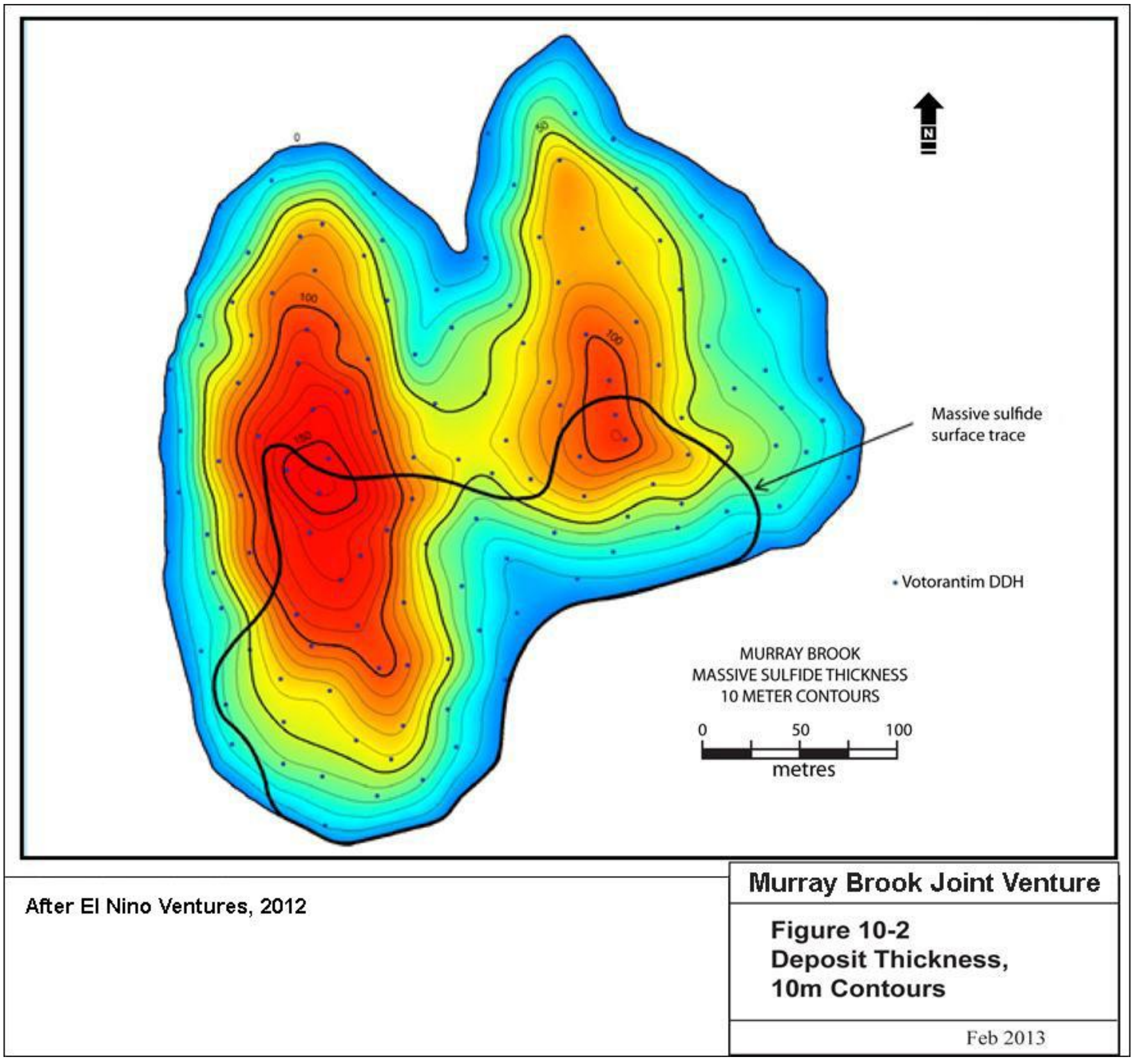


Figure 10.3 Orientation of Sections 300NE and 050SE, Murray Brook Property

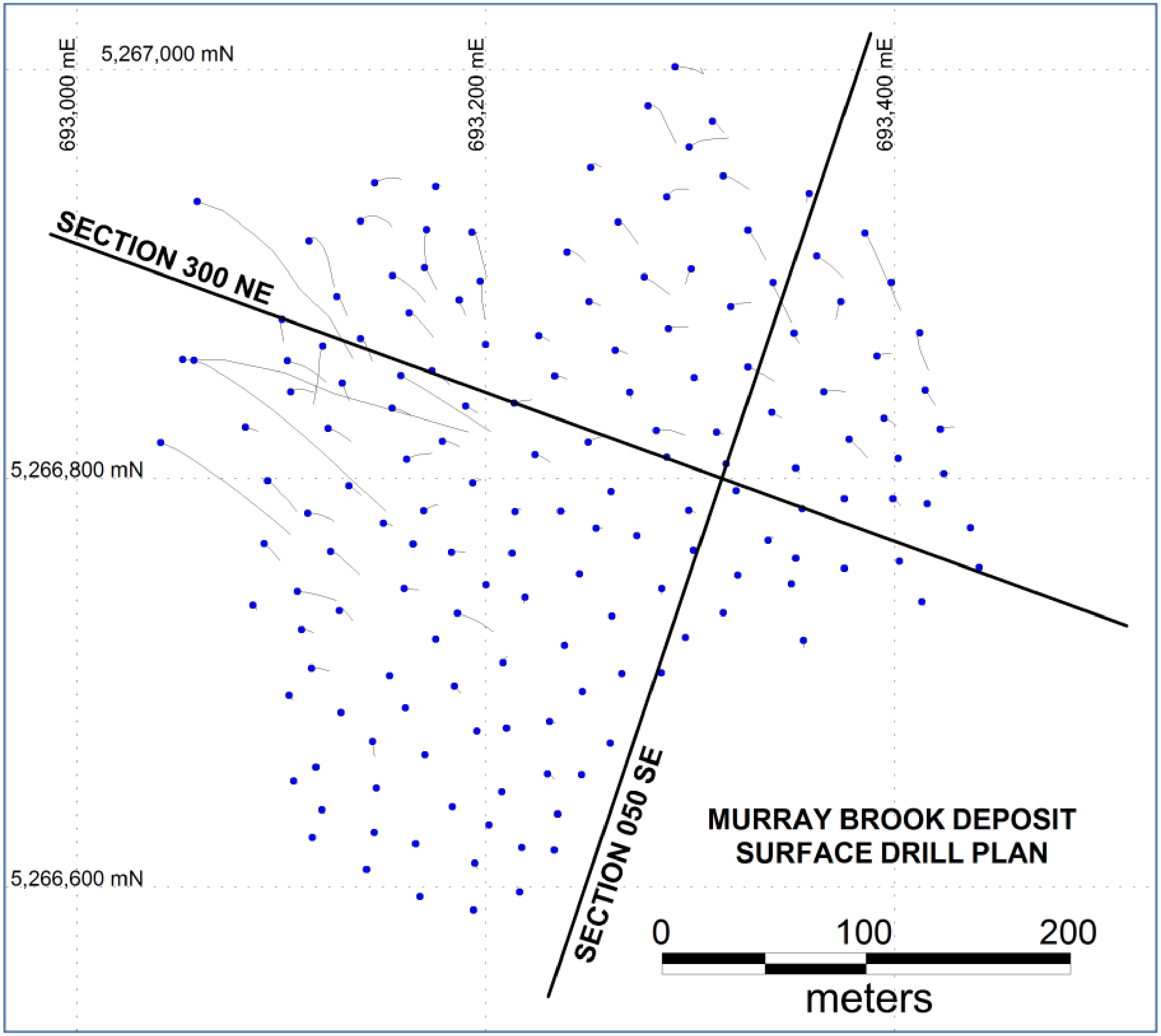


Figure 10.4 Typical Vertical Cross Section 300NE, Murray Brook Property

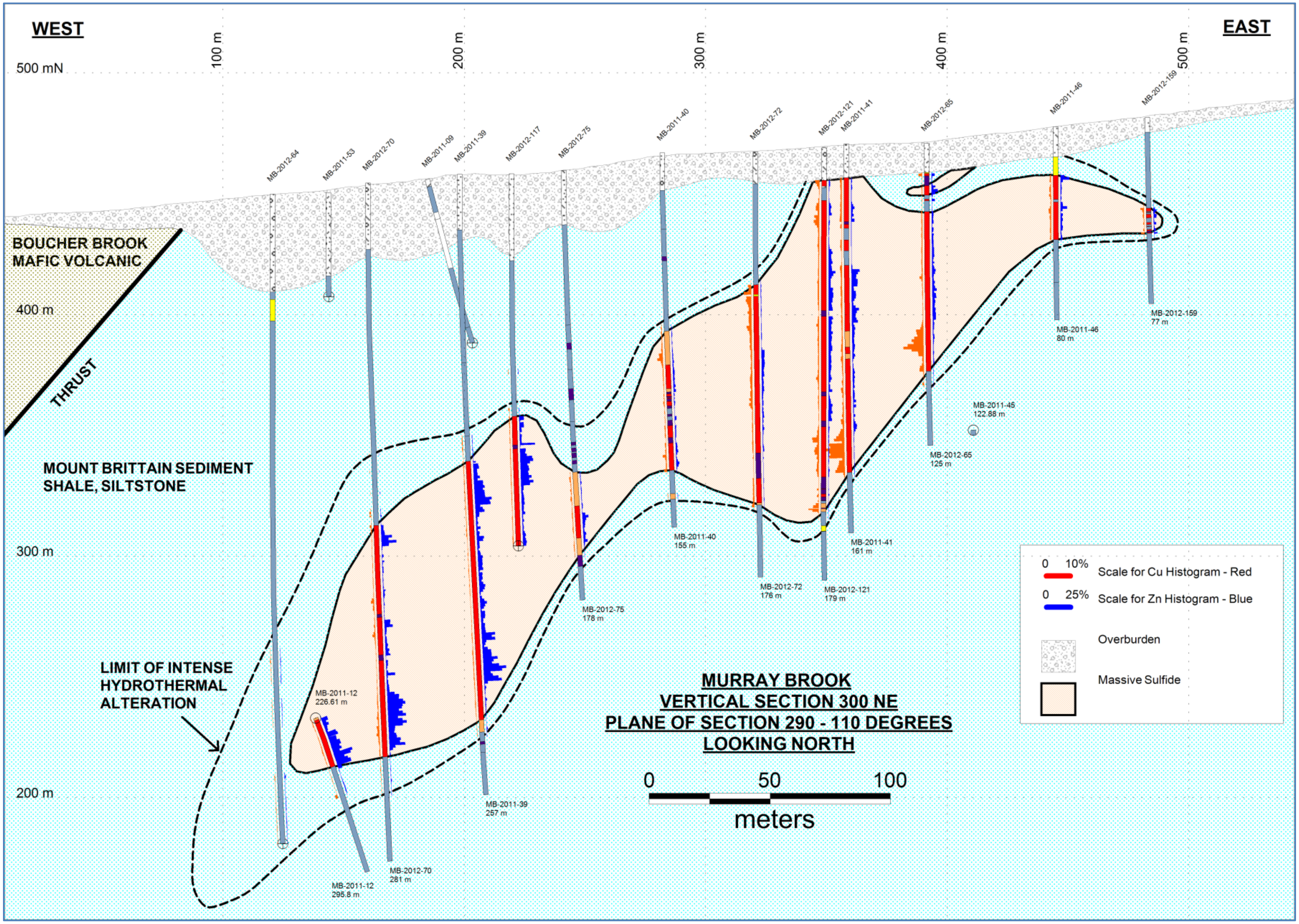
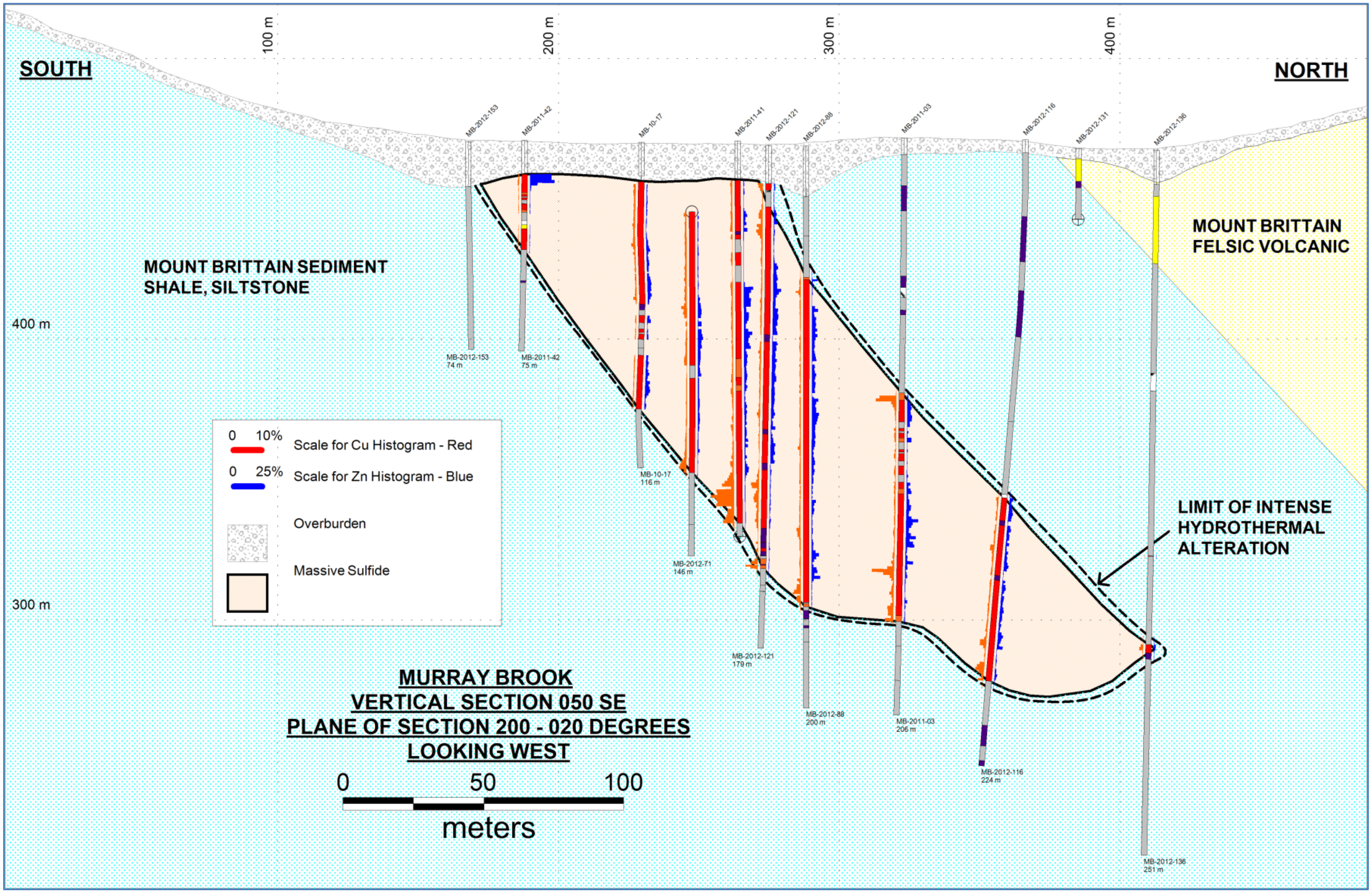


Figure 10.5 Typical Vertical Cross Section 050SE, Murray Brook Property



The objective of the 2012 Phase III diamond drilling was to upgrade the Inferred and Indicated Resources to Measured Resources, define additional near-surface resources along the Northwest margin of the deposit. In addition, three HQ size diamond drill holes MB-2012-121, MB-2012-124 and MB-2012-132 were drilled to yield an approximately three tonne sample for metallurgical testing.

Table 10.3 presents the diamond drill specifications of the Phase III program, which commenced in February 2012 and ended June 17, 2012. A total of 99 NQ size vertical holes, totalling 18,624 m were completed in this program designed to infill gaps in the drill data, and to better define the shape and size of the mineralized zone (ELN News Release August 14, 2012). Table 10.4 provides the Significant Phase III Drill Intercepts for 2012.

TABLE 10.3
DIAMOND DRILL SPECIFICATIONS FOR PHASE III DRILL PROGRAM(1)(2)

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Inclination (°)	Length (m)
MB-2012-64	693100.5	5266878	449.818	360	-90	275
MB-2012-65	693355.0	5266785	470.778	360	-90	125
MB-2012-66	693338.2	5266770	469.328	360	-90	104
MB-2012-67	693323.5	5266753	470.01	360	-90	98
MB-2012-68	693127.3	5266889	451.97	360	-90	275
MB-2012-69	693316.3	5266734	469.117	360	-90	74
MB-2012-70	693139.0	5266868	454.403	360	-90	281
MB-2012-71	693299.6	5266784	468.799	360	-90	146
MB-2012-72	693288.7	5266810	467.267	360	-90	176
MB-2012-73	693224.2	5266812	462.774	360	-90	176
MB-2012-74	693274.1	5266772	467.535	360	-90	125
MB-2012-75	693213.9	5266837	459.587	360	-90	178
MB-2012-76	693254.2	5266776	466.78	360	-90	125
MB-2012-77	693187.2	5266887	455.011	360	-90	269
MB-2012-78	693375.7	5266756	473.294	360	-90	77
MB-2012-79	693145.7	5266945	443.493	360	-90	300
MB-2012-80	693378.0	5266819	474.53	360	-90	161
MB-2012-81	693399.5	5266790	476.298	360	-90	120
MB-2012-82	693138.9	5266926	444.516	360	-90	302
MB-2012-83	693401.9	5266810	476.315	360	-90	152
MB-2012-84	693422.7	5266824	478.104	360	-90	152
MB-2012-85	693113.7	5266916	444.473	360	-90	350
MB-2012-86	693424.3	5266802	478.31	360	-90	137
MB-2012-87	693437.3	5266776	479.839	360	-90	99
MB-2012-88	693313.1	5266823	468.776	360	-90	200
MB-2012-89	693292.8	5267001	462.233	360	-90	308
MB-2012-90	693154.6	5266899	448.847	360	-90	275
MB-2012-91	693302.2	5266849	467.692	360	-90	215
MB-2012-92	693175.8	5266943	446.921	360	-90	299
MB-2012-93	693270.7	5266842	465.704	360	-90	188
MB-2012-94	693289.5	5266873	463.791	360	-90	230
MB-2012-95	693233.8	5266850	460.274	360	-90	200
MB-2012-96	693316.4	5266948	465.507	360	-90	259
MB-2012-97	693320.0	5266884	468.445	360	-90	242
MB-2012-98	693226.2	5266870	457.844	360	-90	203
MB-2012-99	693299.8	5266962	463.992	360	-90	269
MB-2012-100	693250.8	5266886	459.087	360	-90	192
MB-2012-101	693351.0	5266871	471.609	360	-90	202
MB-2012-102	693164.6	5266768	474.175	360	-90	242

TABLE 10.3
DIAMOND DRILL SPECIFICATIONS FOR PHASE III DRILL PROGRAM(1)(2)

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Inclination (°)	Length (m)
MB-2012-103	693391.7	5266860	475.133	360	-90	191
MB-2012-104	693288.8	5266938	464.286	0	-90	287
MB-2012-106	693365.5	5266842	473.916	360	-90	200
MB-2012-107	693214.4	5266784	473.139	360	-90	179
MB-2012-108	693251.6	5266952	458.636	360	-90	302
MB-2012-109	693277.8	5266899	462.611	360	-90	233
MB-2012-110	693154.3	5266834	461.565	360	-90	275
MB-2012-111	693264.9	5266926	461.335	360	-90	251
MB-2012-112	693328.4	5266922	466.237	360	-90	251
MB-2012-113	693219.4	5266742	468.493	360	-90	152
MB-2012-114	693200.2	5266748	469.994	360	-90	206
MB-2012-115	693201.7	5266631	473.951	360	-90	107
MB-2012-116	693340.8	5266896	470.951	360	-90	224
MB-2012-117	693190.3	5266835	458.227	360	-90	245
MB-2012-118	693231.5	5266681	468.931	360	-90	123
MB-2012-119	693239.9	5266911	454.801	360	-90	260
MB-2012-120	693235.4	5266636	469.169	360	-90	74
MB-2012-121	693317.8	5266807	468.983	360	-90	179
MB-2012-122	693217.8	5266619	473.357	360	-90	101
MB-2012-123	693194.8	5266612	477.089	360	-90	86
MB-2012-124	693175.6	5266721	467.394	360	-90	188
MB-2012-125	693165.8	5266621	477.044	360	-90	101
MB-2012-126	693184.9	5266698	468.488	360	-90	176
MB-2012-127	693145.6	5266627	478.604	360	-90	92
MB-2012-128	693119.9	5266638	482.329	360	-90	101
MB-2012-129	693144.6	5266671	473.284	360	-90	152
MB-2012-130	693117	5266659	481.793	360	-90	125
MB-2012-131	693362.1	5266909	467.886	360	-90	227
MB-2012-132	693178.9	5266818	466.111	360	-90	227
MB-2012-133	693103.0	5266858	452.171	360	-90	251
MB-2012-134	693373.9	5266886	471.516	360	-90	224
MB-2012-135	693093.5	5266799	472.149	360	-90	258
MB-2012-136	693358.5	5266939	467.312	360	-90	251
MB-2012-137	693107.9	5266745	484.628	360	-90	224
MB-2012-138	693129.9	5266847	455.717	360	-90	269
MB-2012-139	693110.1	5266726	486.42	360	-90	200
MB-2012-140	693114.9	5266707	483.924	360	-90	185
MB-2012-141	693104.7	5266842	458.368	360	-90	245
MB-2012-142	693103.8	5266694	485.232	360	-90	152
MB-2012-143	693082.6	5266825	460.873	360	-90	224
MB-2012-144	693412.6	5266871	475.406	360	-90	200
MB-2012-145	693398.7	5266896	472.164	360	-90	200
MB-2012-146	693086.2	5266738	488.972	360	-90	176
MB-2012-147	693385.8	5266920	470.419	360	-90	200
MB-2012-148	693091.9	5266768	479.47	360	-90	230
MB-2012-149	693415.1	5266843	477.212	360	-90	167
MB-2012-150	693141.8	5266609	483.862	360	-90	74
MB-2012-151	693279.7	5266982	460.746	360	-90	290
MB-2012-152	693247.1	5266655	468.811	360	-90	74
MB-2012-153	693286.0	5266705	470.395	360	-90	74
MB-2012-154	693233.8	5266618	472.955	360	-90	75
MB-2012-155	693216.6	5266598	474.174	360	-90	74
MB-2012-156	693194.2	5266589	477.111	360	-90	74

Table 10.3 Diamond Drill Specifications for Phase III Drill Program(1)(2)						
Hole ID	**Easting (m)**	**Northing (m)**	**Elevation (m)**	**Azimuth (°)**	**Inclination (°)**	**Length (m)**
MB-2012-157	693115.3	5266624	483.817	360	-90	74
MB-2012-158	693106.2	5266652	484.115	360	-90	101
MB-2012-159	693441.7	5266757	481.436	360	-90	77
MB-2012-160	693355.6	5266721	477.454	360	-90	74
MB-2012-161	693171.1	5266922	449.351	360	-90	302
MB-2012-162	693311.0	5266975	463.92	360	-90	287

(1) Coordinates are in UTM NAD83 Zone 19;
(2) Drill hole MB-2012-105 was abandoned.

Table 10.4 Significant Phase III Drill Intercepts (2012)								
Hole ID	**From (m)**	**To (m)**	**Width (m)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Au (g/t)**	**Ag (g/t)**
MB-2012-65	17.5	23.9	6.4	0.09	0.89	2.75	0.56	31.4
and	43.0	47.0	4.0	0.29	1.10	2.05	1.01	47.4
and	51.0	54.0	3.0	0.28	1.75	2.92	1.10	62.7
and	57.0	73.0	16.0	0.11	1.09	2.79	0.50	41.8
MB-2012-66	32.0	48.5	16.5	0.37	1.50	3.03	1.41	65.2
and	51.0	56.0	11.0	0.17	1.69	3.40	1.22	65.4
MB-2012-67	15.0	41.0	26.0	0.32	1.41	3.35	1.07	62.0
and	77.65	83.0	5.35	0.11	1.26	2.92	0.60	61.8
MB-2012-68	207.22	216.0	8.78	0.06	2.24	7.29	0.16	62.5
and	220.0	236.0	16.0	0.10	2.34	6.83	0.64	85.3
MB-2012-69	17.0	19.15	2.15	0.31	1.03	2.13	1.63	53.5
MB-2012-70	141.75	150.40	8.65	0.16	3.67	6.43	0.60	85.5
and	181.0	184.0	3.0	0.22	2.82	6.60	0.31	69.4
and	191.0	195.75	4.75	0.05	0.86	3.08	0.22	24.2
and	201.0	211.0	10.0	0.09	1.59	4.07	0.61	54.7
and	211.0	235.0	24.0	0.49	4.55	11.58	1.53	147.8
MB-2012-71	56.00	61.25	5.25	1.18	1.29	2.59	0.79	50.3
and	65.3	76.0	10.7	0.41	1.57	2.80	0.97	56.1
MB-2012-72	82.0	93.0	11.0	0.43	1.24	2.59	0.92	51.2
MB-2012-74	55.0	62.0	7.0	0.96	1.42	2.52	0.82	58.0
MB-2012-75	142.0	154.0	12.0	0.17	0.93	2.58	0.64	36.0
MB-2012-76	32.1	80.80	48.7	0.49	0.63	1.53	0.49	22.1
MB-2012-77	166.0	226.0	60.0	0.25	1.61	4.43	1.24	69.8
MB-2012-78	12.0	28.0	16.0	0.44	0.83	1.89	1.27	38.1
MB-2012-80	70.40	76.2	5.8	0.42	1.16	2.75	1.13	60.5
and	90.6	120.35	29.75	0.32	0.98	2.02	1.14	47.3
MB 2012-81	38.6	42.3	3.7	0.27	1.90	4029	1.06	71.5
and	48.35	85.10	36.75	0.50	1.16	2.90	0.71	56.6
MB-2012-82	249.15	265.0	15.85	0.13	1.96	5.02	0.82	99.9
MB-2012-83	67.65	76.0	8.35	0.46	1.11	2.46	1.09	53.2
and	90.95	93.6	2.65	0.37	1.14	2.48	1.60	49.4
and	97.70	116.45	18.75	0.41	0.82	2.01	0.87	42.1
MB-2012-84	100.45	103.3	2.85	0.66	1.35	2.85	1.35	56.9
MB-2012-86	82.0	105.2	23.2	0.28	1.15	2.46	1.19	51.0
MB-2012-87	64.4	67.25	2.85	0.27	1.42	3.41	0.49	90.1
MB-2012-88	47.5	162.45	114.95	0.39	0.96	2.06	0.74	38.2
MB-2012-89	227.95	229.90	1.95	0.90	0.94	1.91	1.15	45.8
MB-2012-90	151.16	198.28	47.12	0.12	2.97	7.94	0.68	114.6
and	205.64	240.0	34.36	0.34	3.41	7.02	2.0	1.07

**TABLE 10.4
SIGNIFICANT PHASE III DRILL INTERCEPTS (2012)**

Hole ID	From (m)	To (m)	Width (m)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
MB-2012-91	73.65	184.65	111.0	0.68	0.73	1.53	0.72	35.71
MB-2012-92	208.06	210.60	2.54	0.10	1.16	2.86	0.51	39.29
MB-2012-93	86.6	141.05	54.45	0.96	0.47	0.84	0.48	20.44
MB-2012-94	118.0	196.1	78.1	0.52	0.86	1.66	1.04	36.64
MB-2012-95	153.45	156.4	2.95	0.09	0.65	2.27	0.34	19.45
MB-2012-96	189.9	228.3	38.4	0.52	1.03	2.12	1.14	48.82
MB-2012-97	126.0	201.0	75.0	0.46	1.13	2.66	1.05	56.84
MB-2012-98	167.0	174.05	7.05	0.29	1.55	4.66	1.45	57.38
MB-2012-99	195.9	240.5	44.6	0.93	0.90	1.92	1.73	44.95
MB-2012-101	118.0	171.72	53.72	0.49	1.07	2.36	0.95	50.36
MB-2012-102	68.0	118.2	50.2	0.83	1.05	4.15	0.32	42.5
incl	83.0	104.0	21.0	1.07	1.56	6.12	0.27	60.8
and	122.5	174.00	51.5	0.09	0.89	3.39	0.20	34.7
MB-2012-103	133.1	150.3	17.2	0.79	1.06	2.24	1.38	47.8
and	159.0	162.25	3.25	0.32	1.25	2.11	1.31	30.20
MB-2012-104	165.0	182.0	17.0	1.37	0.65	1.32	0.58	33.7
and	183.0	241.0	58.0	0.44	1.06	1.89	1.38	49.3
and	246.51	254.57	8.06	2.52	0.10	0.18	0.38	17.3
MB-2012-106	93.95	104.45	10.5	0.33	1.57	3.86	1.30	74.2
and	129.0	149.5	20.5	0.32	1.23	2.99	1.11	54.5
MB-2012-107	54.80	112.0	57.2	0.15	1.82	5.89	0.32	79.9
Incl	62.0	93.0	31.0	0.18	2.58	9.23	0.34	108.7
MB-2012-109	128.0	202.95	74.95	1.29	0.27	0.67	0.67	22.3
MB-2012-110	108.0	233.0	125.0	0.26	1.27	4.56	0.60	47.1
incl	108.0	145.0	37.0	0.14	1.64	7.92	0.24	61.88
MB-2012-111	145.30	162.5	17.2	0.91	0.13	0.38	0.25	16.1
MB-2012-112	167.2	220.15	52.95	0.49	0.87	1.96	0.85	45.7
MB-2012-113	53.0	94.25	41.25	0.61	0.26	1.24	0.36	16.4
MB-2012-114	56.0	135.5	79.5	0.53	0.98	3.45	0.32	46.5
incl	98.0	126.0	28.0	0.18	2.48	7.59	0.56	102.2
and	153.6	169.0	15.5	0.10	1.63	4.12	0.53	51.9
incl	153.6	163.0	9.4	0.12	2.30	5.41	0.75	71.0
MB-2012-115	23.0	75.0	52.0	0.59	0.24	1.55	0.21	22.4
MB-2012-116	127.0	193.6	66.6	0.35	0.81	1.63	0.91	42.8
MB-2012-117	100.4	185.0	84.6	0.15	1.82	4.62	0.52	69.8
incl	170.0	183.0	13.0	0.29	4.11	10.34	1.39	126.0
MB-2012-118	40.0	59.0	19.0	4.10	0.03	0.12	0.12	12.0
MB-2012-120	14.0	29.0	15.0	3.40	0.08	0.51	0.12	18.5
MB-2012-121	24.0	113.1	89.1	0.43	1.12	2.42	1.14	55.5
incl	42.0	58.0	16.0	0.33	1.19	3.13	2.21	66.0
MB-2012-122	17.0	29.3	12.3	2.60	0.24	1.81	0.12	24.9
incl	14.4	22.0	7.6	4.79	0.10	0.62	0.20	35.6
and	34.7	50.4	15.7	1.92	0.06	0.18	0.02	9.4
MB-2012-123	18.0	54.9	36.9	0.80	0.13	1.22	0.21	19.8
MB-2012-124	29.0	110.0	81.0	0.23	1.35	4.27	0.24	54.0
Incl	29.0	38.0	10.0	0.80	2.55	7.39	0.28	114.8
Incl	56.0	67.0	11.0	0.03	2.00	5.33	0.13	61.6
incl	84.9	92.0	7.1	0.07	1.10	5.41	0.28	46.6
and	128.0	137.0	9.0	0.18	1.08	4.22	0.33	56.7
MB-2012-125	22.0	60.3	38.3	0.54	0.03	0.10	0.17	10.4
MB-2012-126	22.5	94.0	71.5	0.55	0.89	3.65	0.38	38.5
Incl	22.5	34.0	11.5	2.41	2.13	7.10	0.44	87.6

TABLE 10.4 SIGNIFICANT PHASE III DRILL INTERCEPTS (2012)								
Hole ID	**From (m)**	**To (m)**	**Width (m)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Au (g/t)**	**Ag (g/t)**
incl	39.0	74.0	35.0	0.18	0.75	3.60	0.49	32.2
MB-2012-127	26.0	62.3	36.3	0.62	0.36	1.18	0.85	21.5
MB-2012-128	27.0	49.4	22.4	1.05	0.30	0.82	0.13	20.3
MB-2012-129	71.0	77.0	6.0	0.78	0.80	2.67	0.59	28.9
MB-2012-130	46.0	50.6	4.6	1.12	0.06	0.34	0.08	8.7
and	52.7	67.7	15.0	1.19	0.14	0.56	0.20	18.1
MB-2012-131	152.0	173.0	21.0	0.73	83.0	1.64	1.12	33.6
MB-2012-132	89.0	201.0	112.0	0.10	1.92	6.15	0.64	70.6
incl	167.0	200.0	33.0	0.10	3.60	10.50	1.37	126.7
MB-2012-133	183.0	210.2	27.2	0.93	0.03	0.08	0.07	8.9
incl	191.0	210.2	19.2	1.10	0.02	0.10	0.10	7.8
MB-2012-134	139.0	157.4	18.4	0.41	1.54	3.21	1.16	61.9
MB-2012-135	144.8	147.0	2.2	0.42	0.14	0.54	0.03	7.5
MB-2012-136	nsv							
MB-2012-137	nsv							
MB-2012-138	197.6	225.0	45.4	0.18	4.58	8.49	0.59	152.2
incl	214.0	225.0	11.0	0.2	9.6	13.7	1.10	269.7
MB 2012-139	88.0	114.0	26.0	0.47	0.36	2.06	0.28	21.4
MB 2012-140	64.6	91.6	27.0	0.60	0.64	2.73	0.75	36.7
incl	67.0	77.0	10.0	0.10	1.10	4.90	0.18	50.5
MB-2012-141	178.3	180.7	2.4	0.15	1.93	6.27	0.47	99.3
MB-2012-142	53.0	56.9	3.9	0.16	3.69	8.89	0.58	86.4
MB-2012-143	nsv							
MB-2012-144	147.4	159.0	11.6	0.65	1.78	2.82	1.57	63.9
MB-2012-145	nsv							
MB-2012-146	nsv							
MB-2012-147	nsv							
MB-2012-148	nsv							
MB-2012-149	123.3	133.6	10.3	0.51	1.18	2. 30	1.55	45.9
MB-2012-150	26.0	30.15	4.15	4.18	0.21	0.40	0.42	24.3
MB-2012-151	200.8	225.1	24.3	1.4	0.38	1.13	0.61	27.4
and	244.8	248.6	3.8	4.65	0.19	0.41	0.72	30.3
MB-2012-152	21.25	22.75	1.5	0.09	0.67	3.38	0.04	9.00
MB-2012-153	nsv							
MB-2012-154	nsv							
MB-2012-155	14.7	30.05	15.35	1.91	0.05	0.18	0.04	8.3
MB-2012-156	20.25	30.0	9.75	5.26	0.03	0.07	0.18	9.9
MB-2012-157	18.0	27.7	9.7	3.94	0.38	1.82	0.44	45.8
MB-2012-158	24.8	53.0	28.2	1.27	0.22	0.82	0.17	22.7
MB-2012-159	37.45	48.0	10.55	0.36	0.67	1.52	0.75	26.8
MB-2012-160	nsv							
MB-2012-161	175.9	247.0	71.1	0.28	2.12	5098	1.16	91.6
MB-2012-162	219.1	238.15	19.05	1.94	1.04	1.48	2.16	47.0

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The Murray Brook drilling and sampling program was supervised by Votorantim Senior Geologist Garth Graves, P. Geo. Core logging and sample marking was performed by Votorantim staff geologists Laura Coutts, B. Sc., GIT; Denise Martinez, B. Sc. GIT and Barry MacCallum, B. Sc. GIT. Core cutting and sampling were performed by experienced technicians.

Drill core is placed in wooden core boxes beside the drill and secured using rubber bands (pieces of tire inner tubes). Core boxes are picked up once or twice a day from the drill site by MBJV's staff, and delivered directly to Votorantim's secure core logging facility at 1095 Bridge Street, Bathurst, NB.

Core is aligned, measured and checked for core recovery and RQD. Magnetic susceptibility and conductivity are measured by scanning the core using an MPP2 meter from Geophysics GDD of Quebec City. When a certified operator is present, the core may also be scanned using a Niton XL2-500 XRF instrument to gain a qualitative estimate of base metal distribution as well as As and Sb.

Drill core is then logged geologically and results recorded in Excel format. All massive and strongly disseminated sulphide intervals are marked and tagged for sampling, and up to three 'shoulder' samples beyond the limits of the strong sulphide mineralization, depending upon whether the contact is sharp or gradational. Samples are usually 1.0 m long unless lithologic contacts make for more logical breaks. Short intervals (< 30 cm) of country rock may be included in sulphide samples; larger intervals are sampled separately.

Tags are placed in the core boxes to indicate where a standard or blank should be inserted in the sample stream. A line is drawn on the core to indicate to the sampler where to cut the core. When the core has been marked-up and assay tags positioned, it is photographed to preserve a record of the sample numbers and intervals before it is sawn.

Core is sawn in half using a VanCon diamond saw. One half of the core is placed in a standard plastic sample bag and secured by a nylon cable tie and tagged for analysis, and the other half returned to the core box for reference.

A duplicate core sample is taken at random approximately every 20th sample by sawing the remaining core in half, leaving one quarter core for reference. One of three certified reference materials, (a.k.a. standards) and one blank sample is inserted into the sample stream at the rate of one for every 20 samples.

Up to five or six bagged samples are placed in large polypropylene 'rice bags' which are tied with a numbered plastic security tag. These are placed in a 20 litre plastic pail and capped. Samples are shipped on palettes in batches of 30 samples (about six pails), or multiples thereof. These are picked up from the core facility by Day and Ross Inc., a bonded courier, and driven to TSL Laboratories in Saskatoon ("TSL").

TSL was established in 1981 and is an accredited laboratory certified to perform, inter alia, assay and umpire assay work for the five elements routinely assayed for the Murray Brook project.

In April 2004, TSL successfully completed the ISO/IEC 17025 Accreditation, and are Accredited Laboratory No. 538.

Core samples are crushed to 70% passing -10 mesh (1.70 mm), from which a 1,000 gram portion is riffle split and pulverized to 95% passing -150 mesh (106 µm). All equipment is cleaned with compressed air and brushes after every sample. Both pulps and rejects are stored with TSL in Saskatoon.

Samples are assayed for Cu, Pb, Zn and Ag using a 4-acid total digestion followed by AAS. Gold is determined by a standard lead-collection fire assay procedure using a 30 gram aliquot with an AAS finish. Samples exceeding 3,000 ppb are re-analyzed using the fire assay procedure followed by gravimetric weighing.

P&E considers that the sample preparation, security and analytical procedures are in keeping with industry best practises and have produced accurate and precise results for the elements in the resource estimate.

12.0 DATA VERIFICATION

12.1 SITE VISIT AND INDEPENDENT SAMPLING

The Murray Brook Project was visited on October 15 and 16, 2012, by Mr. Gerald Harron, P.Eng., P. Geo., a Qualified Person, ("QP") as defined by Canadian National Instrument NI 43-101 standards of disclosure for mineral projects, for the purposes of completing a site visit and independent sampling program.

Mr. Harron toured the secure core facilities. Core logging, sampling and cutting procedures were reviewed, and it was observed that the core was coherent, showing no intervals of lost core. Five core samples were measured at random to verify for correct location and length as described in the logs. No discrepancies were noted.

Mr. Harron collected 14 samples of NQ core from eight holes by sawing a quarter core from the remaining half core in the box. Samples were selected from a range of grades and placed in a plastic bag with a unique tag. Each bag was sealed, and once all the samples had been collected they were placed in four five-gallon plastic pails and shipped by Day and Ross Inc., a bonded courier to Activation Laboratories, ("Actlabs") a certified analytical laboratory in Ancaster, ON. At no time were any officers or employees of Votorantim notified as to the location of the samples to be collected.

At Actlabs in Ancaster the site samples were analyzed for Ag, Au, Cu, Pb and Zn.

Gold was analyzed by lead-collection fire assay with AA finish. Silver was analyzed using fire assay with a gravimetric finish. Copper, lead and zinc were analyzed using a total acid digestion with ICP-OES finish.

Actlabs has locations in North, Central and South America, Australia, Africa, Greenland and Mongolia.

The Actlabs' Quality System is accredited to international quality standards through the International Organization for Standardization/International Electrotechnical Commission (ISO/IEC) 17025 (ISO/IEC 17025 includes ISO 9001 and ISO 9002 specifications) with CAN-P-1758 (Forensics), CAN-P-1579 (Mineral Analysis) and CAN-P-1585 (Environmental) for specific registered tests by the SCC. The accreditation program includes ongoing audits, which verify the QA system and all applicable registered test methods. Actlabs is also accredited by the National Environmental Laboratory Accreditation Conference (NELAC) program and Health Canada.

Results of the Murray Brook independent sampling program are shown in Figure 12.1 through Figure 12.5.

Figure 12.1 Murray Brook Independent Sampling for Gold

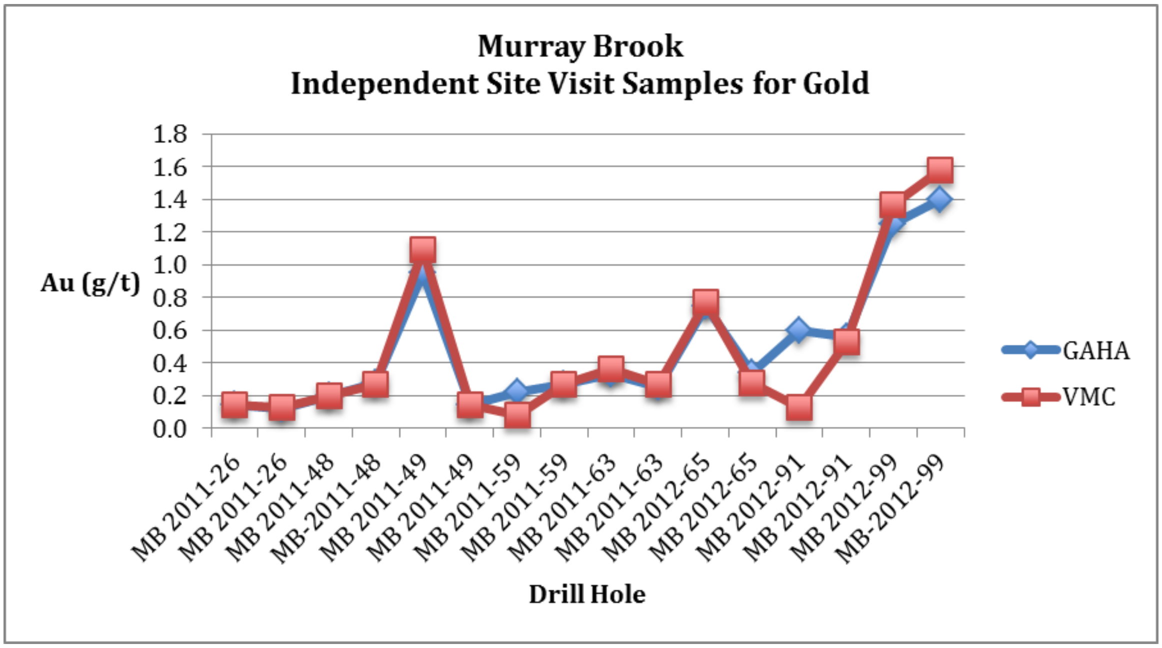


Figure 12.2 Murray Brook Independent Sampling for Silver

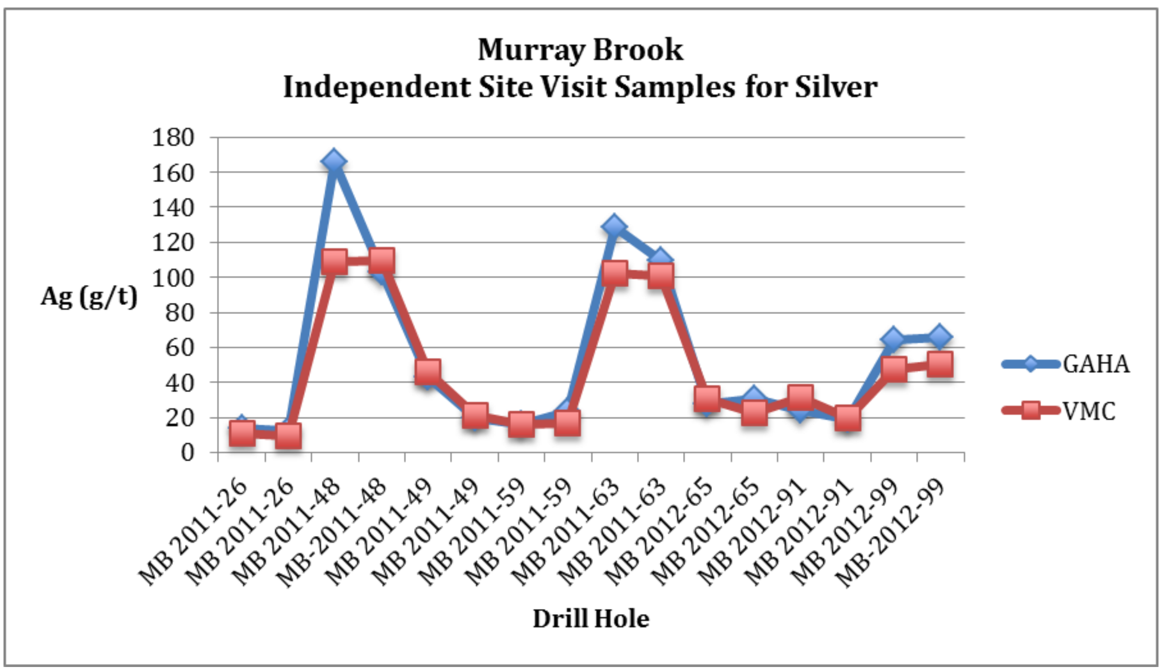


Figure 12.3 Murray Brook Independent Sampling for Copper

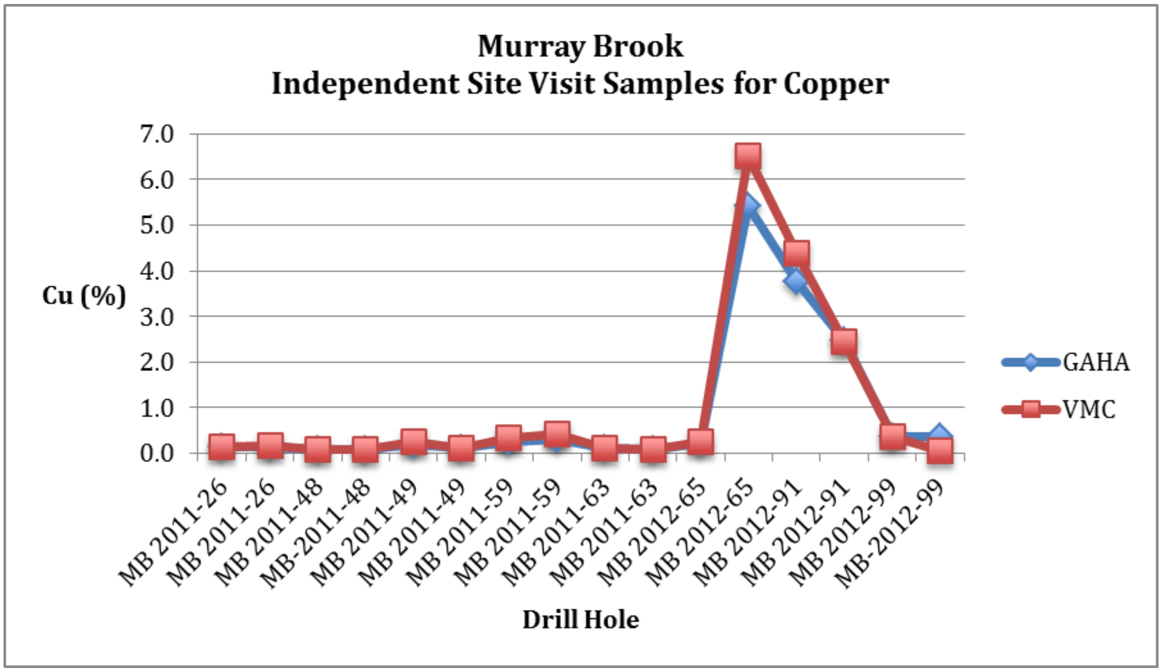


Figure 12.4 Murray Brook Independent Sampling for Lead

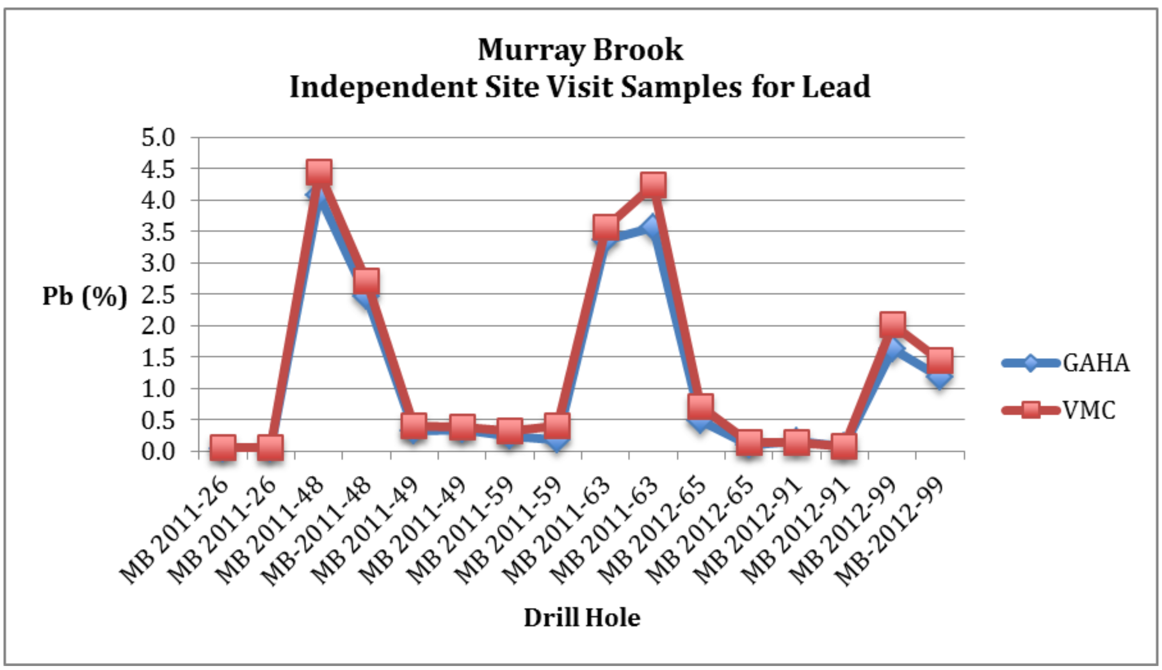
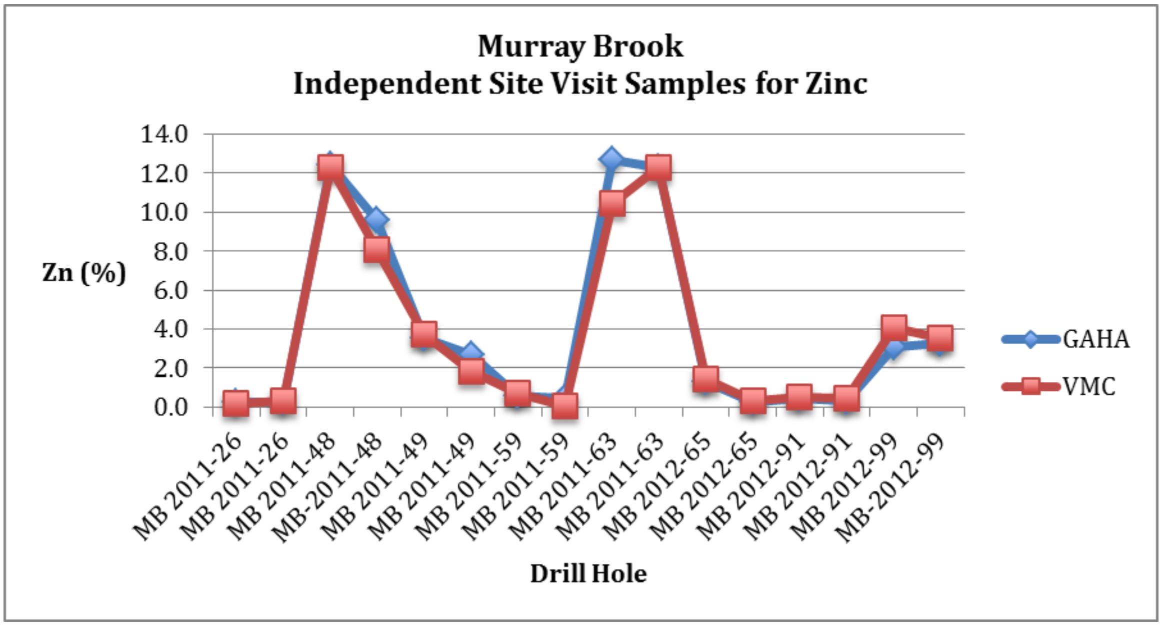


Figure 12.5 Murray Brook Independent Sampling for Zinc



12.2 QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

Votorantim implemented a quality assurance/quality control ("QA/QC" or "QC") program for all phases of drilling including the third phase of diamond drilling which included holes MB-2012-64 to MB-2012-162.

The three Certified Reference Materials, ("CRM" or "standard") used were ME-13, ME-16 and ME-17 supplied by CDN Resource Laboratories Ltd. of Vancouver, BC. ME-13 and ME-17 were prepared from massive and semi-massive sulphides from the Archean-aged Izok Lake VMS deposit; ME-16 was prepared from a "mixture of ores". Standards were inserted into the sample stream at a rate of 1:20.

Sandblasting-grade ground glass purchased in Bathurst was employed as the blank material. It was inserted into the sample stream at a rate of 1:20 (5%).

Core duplicates were produced by ¼ sawing core roughly every 20 samples and sending the ¼ split to the lab as a duplicate of the half core.

Sample pulps were forwarded from the principal lab to a secondary lab for checks.

12.2.1 Performance of Certified Reference Materials

There were 201 standards inserted with the batches sent to the lab. MBJV's geologists monitored the results on a real-time basis as the reports were received from the lab, and for monitoring purposes, two standard deviations above and below the mean were used as warning limits. MBJV also produced a complete and detailed QC report at the end of the drilling phase. The author of this section reviewed all results received from the lab, as well as the MBJV QC report. There were no issues of any concern.

12.2.2 Performance of Blank Material

There were 201 blanks inserted into the sample stream. Gold reported at or below its lower limit of detection (DL) of 5 ppb, with six outliers. The highest Au value was 15 ppb (0.015 g/t). Silver reported almost all values less than 1 g/t, with six outliers. The highest Ag value was 2.0 g/t. Copper and zinc reported all values at or below detection limit with six and seven outliers respectively. Most lead values exceeded the lower detection limit, with a mean value of 220 ppm Pb. Only seven outliers were flagged, with a high value of 800 ppm. It is likely that these assays are most simply explained by the nugget effect of high-lead glass particles, since lead is a common constituent of glass.

None of the outliers was judged to have any impact on the metal value.

12.2.3 Performance of Core Duplicates

A duplicate of the drill core was taken every 20 samples by ¼ sawing the ½ core sent for analyses, leaving a ¼ core sample in the box as a witness in those cases. Two hundred and one (201) duplicate core samples were taken.

Simple scatter graphs for each of the five elements were plotted, showing the correlation between the ¼ and half core sample. Even the gold in the deposit, which would be expected to demonstrate poor precision at this level of homogeneity, in fact demonstrated excellent precision. All the other four elements demonstrated excellent precision close to 1:1.

12.2.4 Secondary Lab Checks

TSL, the primary lab, forwarded 151 pulps at MBJV's request, to ACME labs in Vancouver to verify the performance at TSL. The samples were selected to be representative of the distribution of grades of the massive sulphide body. The correlation coefficients were all very close to one. Silver had the poorest precision, with a correlation coefficient of 0.89. Results for Ag at ACME were on average 11% lower than the results at TSL. This difference is nevertheless completely acceptable, considering that the samples are analyzed at two different labs.

The authors consider that the Murray Brook data were collected using industry best practices, are of good quality, and are suitable for use in the Mineral Resource estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Votorantim Metals Canada Inc. contracted RPC to carry out metallurgical studies on Murray Brook drill core during 2012 and the first quarter of 2013. The Murray Brook deposit is a polymetallic, volcanic hosted massive-sulfide deposit located in the Bathurst Mining Camp of New Brunswick.

13.2 MINERALOGY AND SAMPLES

Sulfides in the deposit are mainly fine grained, massive, weakly laminated pyrite with disseminated and banded sphalerite, chalcopyrite and galena. SEM-EDS mineralogical examination showed that pyrite is the predominant mineral in all samples. In general, sphalerite and galena occur as an interstitial phase and as inclusions, fine veinlets and attachments to pyrite. Chalcopyrite is rare and not found in Hole 132; it occurs interstitial to pyrite. Covellite (CuS) is present in Hole 124. Most of the target mineral occurrences are <20 µm though 50-100 µm. The primary Ag bearing mineral is tetrahedrite.

Composite samples were prepared from three metallurgical drill cores identified as MB-2012-121, 124, and 132. Alteration in the form of near surface oxidation was observed in the core, particularly in the upper regions of Hole 124. Zones showing visible alteration were segregated during compositing.

Average head analyses for the three holes, showing altered zones separately (designated "top" and "middle") are summarized in Table 13.1.

TABLE 13.1
METALLURGICAL SAMPLES

Sample Number and Location	Weight (kg)	Fe (%)	Cu (%)	Pb, (%)	Zn, (%)	Au, (g/t)	Ag, (g/t)
MB-2012-121 - Average Top	21		1.03	0.76	0.95	0.88	36
MB-2012-121 - Average Bottom	1047		0.54	0.83	1.8	0.88	42
MB-2012-124 - Average Top	135		1.07	1.62	5.11	0.33	80
MB-2012-124 - Average Middle	106		0.14	1.72	4.17	0.17	62
MB-2012-124 - Average Bottom	878		0.19	0.74	2.62	0.25	34
MB-2012-132 - Average	1039		1.15	1.64	5.27	0.58	61
Metallurgical Sample Assays							
MB-2012-121		37.55	0.53	0.96	2.02	0.72	32
MB-2012-124		39.38	0.18	0.78	2.77	0.23	27
MB-2012-132		37.41	0.16	1.39	4.34	0.59	53
Composite (121, 124, 132)		39.44	0.33	1.14	3.42	0.51*	47

**calculated*

13.3 GRINDABILITY

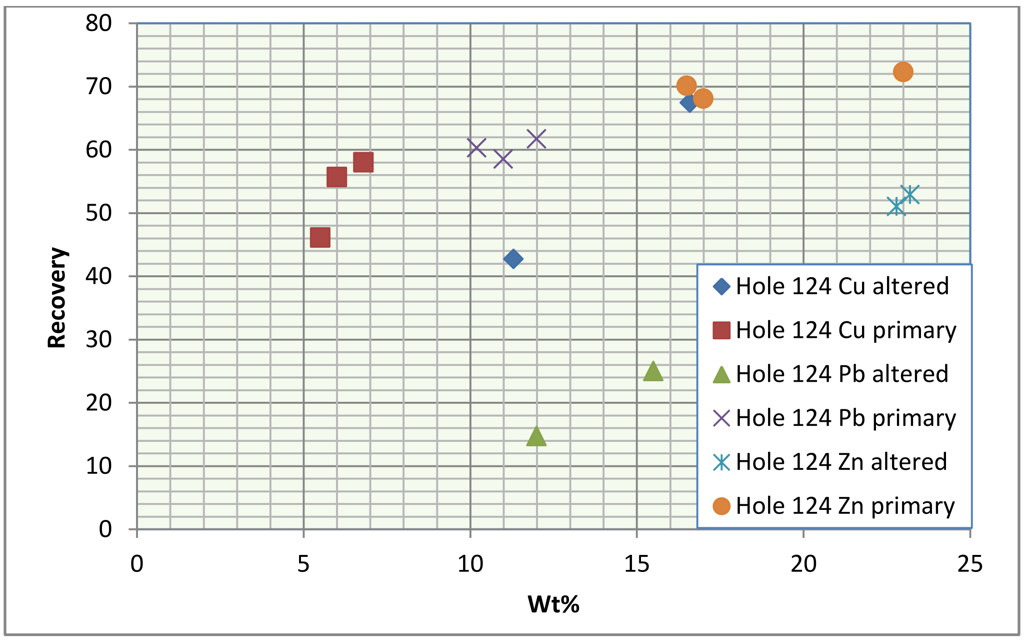
Bond rod mill and ball mill work indices of 14.6 kWh/tonne and 10.7 kWh/tonne were measured on a composite sample of the three cores, indicating a relatively soft material.

13.4 FLOTATION

An initial series of bulk rougher tests were conducted to evaluate the effect of grind. The results indicated that a very fine grind of about 30 microns is required for maximum recovery of all payable metals and this target grind was adopted for further work.

Due to the visible oxidation, particularly in the upper section of the Hole 124 sample, composites of the top and bottom zones were compared in sequential rougher tests. The results are presented graphically in Figure 13.1. Approximately 20% of Hole 124 contained visible oxidation; alteration was absent from Hole 132 and was very minor in Hole 121. The main composite sample, used for subsequent work, was comprised of equal quantities of the three drill hole composites, with visibly altered material excluded.

Figure 13.1 Alteration Effect on Roughing



A comparison was made of two flowsheet alternatives with respect to copper and lead recovery: flotation of a bulk Cu-Pb rougher concentrate followed by cleaning of the concentrate to yield separate copper and lead concentrates and separate (sequential) rougher flotation of copper and lead rougher concentrates. Due to difficulties experienced in separation of copper and lead in the bulk flotation case, sequential flotation was selected for further test-work and locked cycle testing.

The results of sequential rougher flotation tests on composite material are summarized in Table 13.2.

TABLE 13.2
ROUGHER SELECTION

Test	Product	Mass (%)	Analytical Assays						Distribution (%)					
			Fe (%)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Au (g/t)	Fe	Cu	Pb	Zn	Ag	Au
	CuRC	2.8	34.2	6.6	2.7	5.22	545	1.01	2.7	55.3	6.4	4.5	30.3	5.4
	PbRC	10.1	36.3	0.7	7.9	5.16	188	0.78	10.3	20.1	68.1	16	38	15.1
SQ1	ZnRC	16.3	32.6	0.2	0.6	15	40	0.68	15	11.3	7.9	75.6	13.1	21.4
	RT	70.9	36.0	0.1	0.3	0.18	13	0.43	72.1	13.3	17.5	3.9	18.5	58.1
	Calc. Head	100	35.5	0.3	1.2	3.24								
	CuRC	3	30.8	5.9	2.1	4.88	462	0.96	2.4	57.9	5.5	4.4	30.2	5.6
	PbRC	7.4	38.6	0.5	9.7	5.49	188	0.78	7.3	12	64.7	12.2	30.5	11.4
SQ2	ZnRC	20.7	35.1	0.2	0.6	12.82	43	0.62	18.7	15.8	10.3	79.7	19.6	25.4
	RT	69	40.3	0.1	0.3	0.18	13	0.42	71.6	14.4	19.4	3.7	19.8	57.7
	Calc. Head	100	38.8	0.3	1.1	3.32								
	CuRC	4.3	34.8	5.1	2.6	5.53	408	0.96	3.9	65.3	9.8	6.7	39.7	8.2
	PbRC	10.5	39.3	0.3	7.0	4.87	122	0.69	10.8	9.9	65.1	14.4	28.8	14.4
SQ3	ZnRC1(0-3)	12.5	31.6	0.3	0.5	20.93	45	0.56	10.3	9.6	5.7	73.6	12.7	13.8
	ZnRC2(3-5)	6	41.1	0.2	0.5	1.01	28	0.71	6.4	2.6	2.7	1.7	3.8	8.4
	RT	66.7	39.3	0.1	0.3	0.19	10	0.42	68.6	12.6	16.5	3.6	15	55.1
	Calc. Head	100	38.2	0.3	1.1	3.55								
	CuRC	4.5	48.8	6.0	3.0	6.7	354	0.93	7.7	71.9	11.9	10.5	34.4	8.7
	PbRC	9.6	36.5	0.4	7.9	5.03	155	0.63	12.2	10.9	65.8	16.6	31.7	12.5
SQ4-8	ZnRC	19.5	24.3	0.2	0.5	10.4	44	0.67	16.5	8.3	8.4	69.8	18.3	26.9
	RT	66.4	27.5	0.1	0.2	0.14	11	0.38	63.6	9	13.8	3.2	15.6	52
	Calc. Head	100	28.7	0.4	1.2	2.91								

**Cu, Pb & Zn rougher concentrates from SQ4-SQ8 were blended to produce material for cleaning.*

Tests SQ 4-8 concentrates were used as feed for open circuit cleaning tests. The results of cleaning tests were relatively poor; this was attributed to non-optimum selection of collectors in roughing and cleaning.

A partial locked cycle test was conducted incorporating sequential roughing to produce separate copper, lead and zinc concentrates. The production of rougher concentrates as feed for cleaners was campaigned rather than generated for each cycle; thus there were no recycle streams to roughing and first cleaner scavenger tailings were not incorporated in downstream stages. Each of the three cleaner circuits incorporated a regrind stage treating rougher concentrate and first cleaner scavenger concentrate. Accumulated copper and lead first cleaner scavenger tailings were separately floated to produce a zinc concentrate and the contributions of these streams to zinc recovery were mathematically incorporated.

The locked cycle test results are summarized in Table 13.3. These recoveries and grades are the bases for this PEA.

Table 13.3 Summary Grades and Recoveries														
Description	Product	Mass (%)	Assays					Distribution, %				
			Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Cu	Pb	Zn	Ag	Au
Cu Rghr Recovery	Rougher Con	5.7	3.43	2.43	5.1	282	0.8	63.6	12	9	37.6	10
Cu Clnr Recovery	Final Cu Clnr 3 Con	0.9	17.45	6.16	6.04	591	1.1	80.8	37.8	19.1	33.3	20
	Cu Cleaner Tails	4.9	0.82	1.92	4.61	203	0.8	12.9	8	6.9	23	8
	Rougher Tail	99.1										
Overall Cu Recovery								51.4	4.5	1.7	12.5	2
	Cu Cleaner Tails	4.9	0.82	1.92	4.61	203	0.8	12.9	8	6.9	23	8
Pb Rghr Recovery	Rougher Con	8.9	0.37	7.58	5.28	135	0.7	10.6	58.3	14.6	28.1	13
Pb Clnr Recovery	Final Pb Clnr 4 Con	0.8	2.4	50.3	5.27	833	0.9	67.9	62.7	9.5	62	14
	Pb Cleaner Tails	8.1	0.12	3.14	5.26	54	0.5	32.1	37.3	90.5	38	86
	Rougher Tail	98.3										
Overall Pb Recovery								7.2	36.6	1.4	17.5	2
Zn Rghr - Cu ClnrTail	Zn Rghr Con (CuCT)	2.4	1.54	1.66	8.78	369	0.8	12.2	3.5	6.6	21	4
Zn Rghr - Pb ClnrTail	Zn Rghr Con (PbCT)	3.5	0.26	2.72	11.99	82	0.6	3	8.3	13.1	6.7	4
Zinc	Zn Rghr Con (Pb RT)	18	0.21	0.73	13.01	40	0.7	12.4	11.2	72.2	16.9	24
Total Zn Rghr Recovery	Total Rougher	24	0.36	1.11	12.43	80	0.7	27.6	23	91.9	44.6	32
Zn Clnr Recovery	Final Zn Clnr 4 Con	6.1	0.48	1.08	53.78	95	0.4	57.2	35.9	96.6	56.7	17
	Zn Cleaner Tails	17.9	0.12	0.66	0.64	25	0.6	42.8	64.1	3.4	43.3	83
	Zn Rghr Tail (Cu CT)	2.4	0.17	1.97	0.73	21	0.4	1.3	4.1	0.5	1.2	2
	Zn Rghr Tail (Pb CT)	4.6	0.11	2.28	0.73	21	0.5	1.6	8.9	1	2.2	4
Zinc	Rougher Tail	67.3	0.062	0.32	0.2	11	0.4	13.4	18.4	4.2	17.4	53
Calc. Rougher Head		100	0.31	1.16	3.25	43	0.5	100	100	100	100	100
Rougher Head Assays			0.29	1.17	3.12	43	0.6					
Overall Zinc Recovery								15.8	8.3	88.8	25.3	6

ICP analyses of LCT concentrates are recorded in Table 13.4.

Table 13.4 Concentrate Analyses							
Element	Cu Con	Pb Con	Zn Con	Element	Cu Con	Pb Con	Zn Con
Total S, %	36	24.7	33.6	Sr, ppm	3	5.8	1
Na, %	< 0.01	< 0.01	< 0.01	Zr, ppm	19	6	2
Mg, %	0.03	0.02	0.01	Nb, ppm	0.1	< 0.1	0.3
Al, %	0.05	0.05	0.01	Mo, ppm	42.1	17.4	10.4
K, %	< 0.01	0.01	< 0.01	In, ppm	77.2	19.7	> 100
Ca, %	0.135	0.21	0.08	Sn, ppm	153	46	28
Li, ppm	< 0.5	< 0.5	< 0.5	Sb, ppm	> 500	> 500	304
Cd, ppm	153	112	1037	Te, ppm	< 0.1	0.1	0.2
V, ppm	4	2	< 1	Ba, ppm	9	1.5	3
Cr, ppm	73.2	36.4	15.3	La, ppm	< 0.1	< 0.1	< 0.1

TABLE 13.4
CONCENTRATE ANALYSES

Element	Cu Con	Pb Con	Zn Con	Element	Cu Con	Pb Con	Zn Con
Mn, ppm	191	164	211	Ce, ppm	1.8	0.7	0.3
Hf, ppm	0.15	0.15	< 0.1	Pr, ppm	0.2	< 0.1	< 0.1
Hg, ppm	49.5	36.2	> 100	Nd, ppm	0.75	0.3	0.1
Ni, ppm	52.2	25.8	15.7	Sm, ppm	0.2	< 0.1	< 0.1
Er, ppm	0.1	< 0.1	< 0.1	Gd, ppm	0.2	< 0.1	< 0.1
Be, ppm	< 0.1	0.1	0.2	Tb, ppm	< 0.1	< 0.1	< 0.1
Cs, ppm	0.1	0.07	< 0.05	Dy, ppm	0.2	< 0.1	< 0.1
Co, ppm	68.5	19.3	4.6	Ge, ppm	0.95	1.3	0.3
Eu, ppm	0.11	0.07	< 0.05	Yb, ppm	0.15	< 0.1	< 0.1
Bi, ppm	262	751	27.4	Ta, ppm	< 0.1	< 0.1	0.2
Se, ppm	83.2	301	28.6	W, ppm	0.2	0.15	0.2
Ga, ppm	2.3	1.55	9.1	Re, ppm	0.03	0.018	0.011
As, ppm	3335	2080	1460	Tl, ppm	89.1	356	30.6
Rb, ppm	0.8	0.6	< 0.2	Th, ppm	1.4	0.6	0.1
Y, ppm	1.1	0.4	0.1	U, ppm	2.1	1.4	1.3

The test work program on the composite was extended into the first quarter of 2013, culminating in a second locked cycle test. The results of this test were disappointing as shown in Table 13.5.

The relatively poor results compared to the first locked cycle test were attributed to:

- Inclusion of copper circuit cleaner scavenger tails in the lead cleaner circuit, which was not attempted in the first locked cycle test and may have adversely affected performance;
- More aggressive operation of the rougher circuit in an attempt to improve initial recoveries, which may have impacted the cleaner circuits;
- A reported change in reagent additions during the locked cycle test in response to xrf assay results which unfortunately were not accurate (more precise analytical methods were used for metallurgical balancing but were not available during the test).

TABLE 13.5 SUMMARY GRADES AND RECOVERIES												
Description	**Sample/ Product**	**Mass**	**Assays**					**Distribution, %**				
		(%)	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Ag (g/t)**	**Au (g/t)**	**Cu**	**Pb**	**Zn**	**Ag**	**Au**
Hole 121	Bottom	33.3	0.5	0.81	1.83	38	0.82	62.5	25.2	20	30.9	53.4
Hole 124	Bottom	33.3	0.17	0.74	2.46	30	0.21	21.3	23	26.9	24.4	13.5
Hole 132	Whole	33.3	0.13	1.67	4.86	55	0.51	16.3	51.9	53.1	44.7	33.1
Average		100	0.27	1.07	3.05	41	0.51	100	100	100	100	100
Cu Rghr Recovery	Rougher Con	5.8	3.68	2.94	4.38	334	1.26	65.9	14.8	7.5	41.6	13.3
Cu Clnr Recovery	Final Cu Clnr 3 Con	1.4	11.49	6.76	4.14	326	1.59	75.8	51.3	23.9	24.9	34
	Cu Cleaner Tails	4.5	1.16	1.87	4.36	329	1.09	15.9	7.2	5.7	31.2	8.8
	Rougher Tail	98.6										
Overall Cu Recovery								50	7.6	1.8	10.3	4.5
	Cu Cleaner Tails	4.5	1.16	1.87	4.36	329	1.09	15.9	7.2	5.7	31.2	8.8
Pb Rghr Recovery	Rougher Con	10.5	0.17	6.18	4.4	100	1.13	5.5	56.1	13.5	22.4	21.5
	Tot Pb Rghr Recr							21.4	63.3	19.2	53.6	30.3
Pb Clnr Recovery	Final Pb Clnr 4 Con	1.8	2.95	26.08	8.54	702	1.28	74.6	63.3	25.7	76.2	24.9
	Pb Cleaner Tails	13.1	0.14	2.05	3.72	61	0.96	5.4	23.3	14.3	12.7	22.8
	Rougher Tail	96.8										
Overall Pb Recovery								16	40.1	4.9	40.9	7.5
Zn Rghr - Pb Clnr Tails	Zn Rghr Con (Pb CT)	13.1	0.14	2.05	3.72	61	0.96	5.4	23.3	14.3	12.7	22.8
Zinc	Zn Rghr Con (Pb RT)	16.9	0.27	0.78	15.17	50	0.6	14	11.4	75.1	18	18.4
Total Zn Rghr Recovery	Total Rougher	30	0.21	1.33	10.16	55	0.76	19.4	34.7	89.3	35	41.1
Zn Clnr Recovery	Final Zn Clnr 4 Con	6.2	0.52	0.89	48.02	118	0.53	55.3	16.5	94.7	57.1	20
	Zn Cleaner Tails	23.8	0.11	1.17	0.7	23	0.55	44.7	83.5	5.3	42.9	80
Zinc	Rougher Tail	66.8	0.07	0.31	0.2	13	0.39	14.6	17.6	3.9	18	46.8
Calc. Head		100	0.33	1.16	3.41	47	0.551	100	100	100	100	100
Head Assays			0.28	1.13	3.11	42	0.595					
Overall Zinc Recovery								**10.7**	**5.7**	**84.6**	**17.6**	**8.2**

13.5 SUMMARY AND CONCLUSIONS

The test work shows that the Murray Brook deposit is metallurgically difficult, which is a feature of other deposits in the Bathurst camp. A saleable zinc product can be readily made, but copper and lead concentrates typically exhibit low grades and recoveries. Murray Brook material requires very fine primary grinding for adequate liberation of values and comminution costs will be higher than typical.

LCT1 as described above has been selected as the basis for the PEA recoveries and grades as it is believed to represent potentially attainable production scale performance on unaltered Murray

Brook material. The second LCT serves to illustrate the degree of control that will be required to achieve consistent results at full scale. Considerable metallurgical work will be required to optimize and develop confidence in the selected flowsheet and to evaluate possible flowsheet options.

The flotation behaviour of partially altered feed material is inferior to primary material. The treatment of this material has not yet been addressed but should be a component of a future test work program.

14.0 MINERAL RESOURCE ESTIMATE

14.1 INTRODUCTION

The Mineral Resource estimate presented herein is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 and has been performed in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Reported Mineral Resources are not mineral reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into a mineral reserve. The quantity and grade of the reported Inferred Resources may not be realized.

This resource estimate was prepared by Yungang Wu, P.Geo. and Eugene Puritch, P.Eng. of P&E. The effective date of this resource estimate is June 4, 2013.

14.2 DATABASE

All drilling data consisting of collar coordinates, survey, lithology, density, assay and core recovery were provided by Votorantim Metals Canada Inc. (Votorantim) in form of MS Excel files. P&E compiled the drilling data into a Gemcom Access database which contains a total of 161 drill holes completed during 2010-2012. All pre-2010 drill holes were not utilized for this resource estimate due to their non-verified nature. A drill hole plan is shown in Appendix-I.

The drilling database of the Murray Brook Project contains 10,045 samples all of which were analyzed for Cu%, Pb%, Zn%, Au g/t and Ag g/t. A total of 7,964 assays from 141 drill holes have been employed for this resource estimate. Of the 166 holes drilled, five were abandoned and the others were located outside of the sulphide body and not included in the resource estimate. All drill hole survey and assay values are expressed in metric units, while grid coordinates are in the NAD 83 UTM system, zone 19.

14.3 DATA VERIFICATION

Assay data from 2010 and 2011 drilling were verified during the course of the last resource estimate in April 2012. 95.5% (5,710 out of 5,980) of the assay data from 2012 were checked for Cu, Pb, Zn, Au and Ag against the original laboratory certificates from TSL Laboratories Inc. of Saskatoon, Saskatchewan. The finding was that some assays which were below the laboratory detection limits had been set to 0 or half of detection limit in the database, which is acceptable for the resource estimate.

14.4 DOMAIN INTERPRETATION

A single mineralized domain was created with computer screen digitizing on drill hole sections in Gemcom by the author of this report. The domain outline was determined from lithology, structure and NSR value by visually inspecting the drill hole cross sections. Twenty-six (26) drill cross sections were developed on 20-metre spacing looking on an azimuth of 290°. The digitized outlines were influenced by the selection of mineralized material above a cut-off NSR value of C$21/tonne that demonstrated zonal continuity along strike and down dip. In some cases mineralization below C$21/tonne NSR were included for the purpose of maintaining zonal continuity. On each section, polyline interpretations were digitized from drill hole to drill hole but not extended nominally more than 30 metres into untested territory. Minimum constrained true width for interpretation was approximately 2.0 metres. The interpreted polylines from each section were "wireframed" in Gemcom into a 3-dimensional domain. The resulting domain was employed for statistical analysis, grade interpolation, rock coding and resource reporting purposes. The wireframe of the mineralized domain is displayed in Appendix-II.

Surfaces for the topography, overburden and oxidation boundary were generated as well using the data provide by MBJV.

14.5 ROCK CODE DETERMINATION

The mineralized domain solid was assigned rock codes for purpose of resource estimate. The domain was divided into two sub-domains of Oxide and Sulphide by intersecting with the Oxidation surface. The rock codes applied for the modeling are presented in Table 14.1.

TABLE 14.1
ROCK CODE DESCRIPTION FOR MURRAY BROOK RESOURCE ESTIMATE

Rock Type	Rock Codes	Notes
Mineralization	10	Domain
Oxide	21	Sub-Domain
Sulphide	22	Sub-Domain
Air	0	
Overburden	100	
Waste	99	

14.6 COMPOSITING

As shown in Figure 14.1, there appears to be no correlation between sample length and Zn grade. Figure 14.2 illustrates that approximately 86% of the sample lengths within the constrained wireframe were one metre in length. In order to regularize the sample length for grade interpolation, assay compositing to one metre length was carried out down hole within the constraints of the above mentioned domain. The composites were calculated over 1.0 metre length starting at the first point of intersection between drill hole and hanging wall of the 3-D zonal constraint. The compositing process was halted upon exiting from the footwall of the aforementioned constraint. Un-assayed intervals and below detection limit assays were set to 0.001% for Cu, Pb and Zn and 0.001 g/t for Ag and Au. Any composites that were less than 0.25 metres in length were discarded so as not to introduce any short sample bias in the interpolation process.

Figure 14.1 Correlation of Zn% Assay and Sample Length

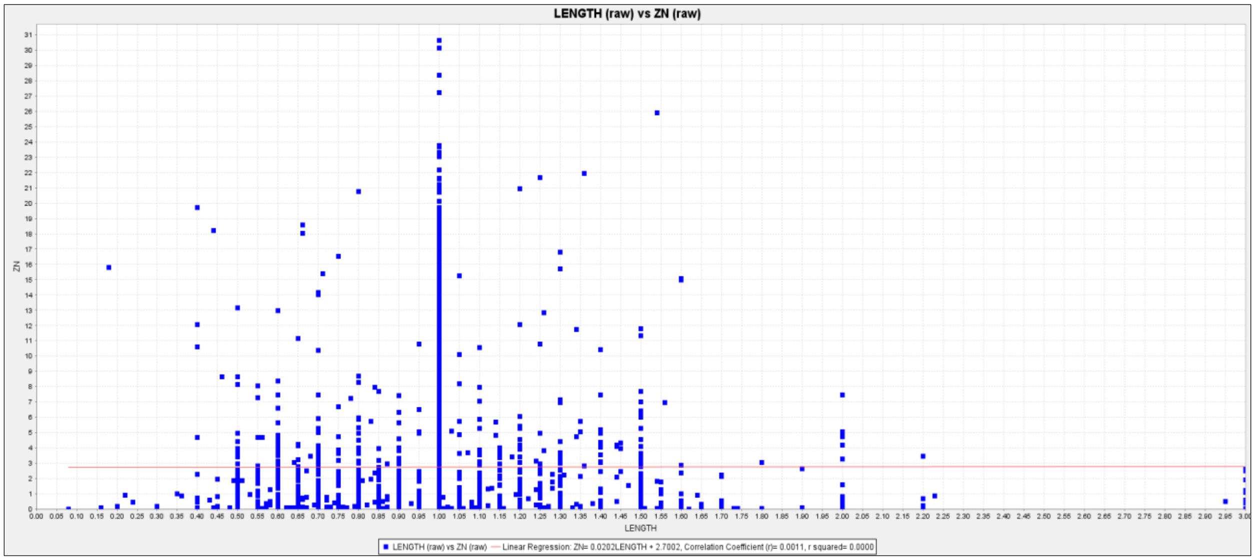
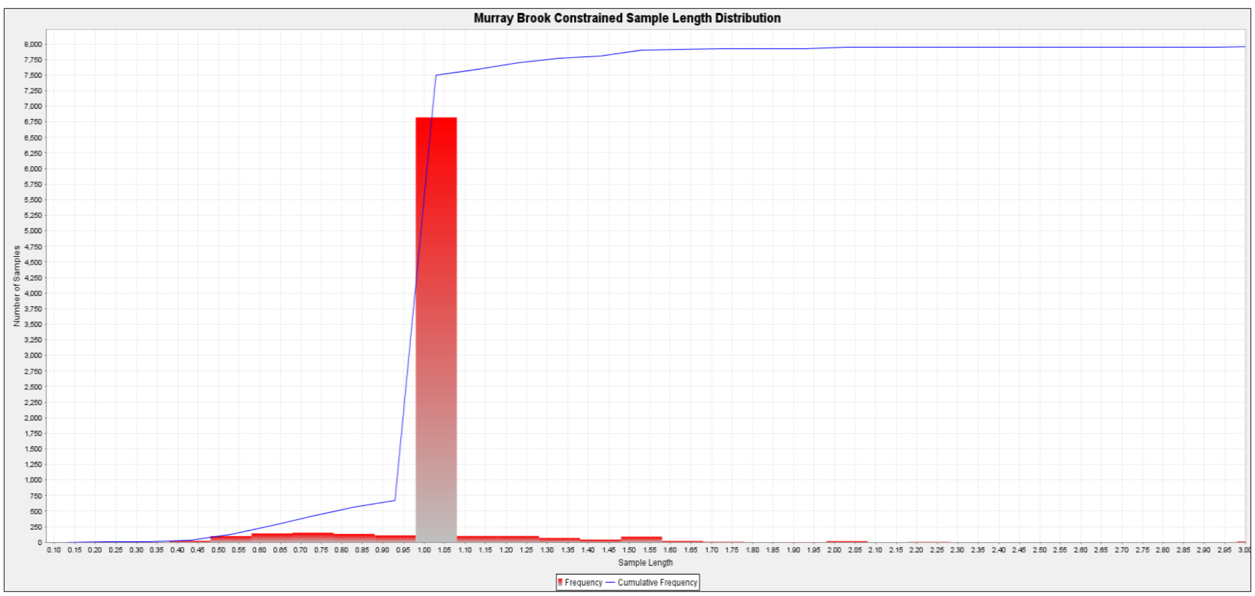


Figure 14.2 Drill Hole Assay Sample Length Distribution



14.7 GRADE CAPPING

Grade capping was investigated on the one metre composite values within the constraining domains to ensure that the possible influence of erratic high values did not bias the database. Log-normal histograms were generated using composites and resulted in the graphs exhibited in Appendix-III. Based on the log-normal histogram performance, as detailed in Table 14.2, Cu was capped at 6%, Pb at 10% and Ag at 250 g/t while no capping was applied for Zn and Au. The capped average grade decreased less than 1% from the grade of the composites. The capped composites were extracted with Gemcom into a point profile, and then utilized for the variogram development and grade interpolation.

TABLE 14.2 MURRAY BROOK GRADE COMPOSITE CAPPING STATISTICS										
	Uncapped					**Capped**				
Element	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)
Number of samples	8,063	8,063	8,063	8,063	8,063	8,063	8,063	8,063	8,063	8,063
Minimum value	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Maximum value	8.781	18.718	29.603	541.34	5.279	6.000	10.000	29.60	250.00	5.279
Mean	0.479	0.985	2.699	41.192	0.560	0.475	0.979	2.699	40.966	0.560
Median	0.245	0.568	1.660	28.775	0.335	0.245	0.568	1.660	28.775	0.335
Variance	0.596	1.705	10.501	1744.6	0.371	0.549	1.569	10.50	1613.555	0.371
Standard Deviation	0.772	1.306	3.241	41.769	0.609	0.741	1.253	3.241	40.169	0.609
Coefficient of variation	1.612	1.325	1.201	1.014	1.087	1.558	1.279	1.201	0.981	1.087
Capping Value	6	10	N/A	250	N/A					
# Composites Capped	29	13	0	23	0					
Percentile Capped	99.6	99.8	100.0	99.7	100.0					

14.8 SEMI-VARIOGRAMS

A semi-variogram study was performed as a guide to the grade interpolation search ellipse parameter strategy. The variography investigation was attempted on the constrained capped composites for Cu, Pb, Zn, Ag and Au respectively. Reasonable variograms were attained along strike, down dip and across dip for all elements. The variogram ranges were used as the spherical search ellipse parameters for grade interpolation. The variograms for Zn are demonstrated in Appendix-IV.

14.9 BULK DENSITY

The bulk density used for this resource model was derived from 1,073 analyses performed on drill core using wet immersion method by TSL. The density varied from 2.62 to 4.86 t/m³ and averaged 4.02 t/m³. The bulk density block model was interpolated with a single pass spherical search ellipse of 50 m x 50 m x 50 m utilizing the constrained bulk density data within the mineralized domain. The average block model mineralized bulk density was calculated to be 4.08 tonnes per cubic metre.

14.10 BLOCK MODELING

The Murray Brook resource block model was constructed using Gemcom modeling software. The block model is oriented with X axis at 110° azimuth (rotated 20° clockwise) parallel to the trend of the mineralization domain. The block model parameters are summarized in Table 14.3.

TABLE 14.3 MURRAY BROOK BLOCK MODEL DEFINITIONS			
Direction	**Origin**	**# of Blocks**	**Block Size (m)**
X	692,511.73	368	3
Y	5,266,507.171	362	3
Z	612	154	3
Rotation	-20^{o} (Clockwise)		

Block models for rock type, density, percent, Zn, Pb, Cu, Ag, Au and NSR, and class were created.

All blocks in the rock type block model were initially assigned a waste rock code of 99, corresponding to the country rocks. The mineralization domain was employed to select all blocks within the rock block model that contain by volume 1 % or greater mineralization. These blocks were assigned rock code 10 representing mineralization. The oxidation surface was utilized to update all mineralization blocks above the surface to oxide and below to sulphide. The overburden and topographic surface were subsequently used to assign rock code 0 for air and 100 for overburden to all blocks 50 % or greater above the surfaces.

A percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside the constraining domain. As a result, the domain boundary was properly represented by the percent model ability to measure individual infinitely variable block inclusion percentages within that domain.

The density models for all mineralized blocks were interpolated using Inverse Distance Squared ($1/d^2$) method. All waste blocks were initialized to a bulk density of $2.7 t/m^3$ while overburden blocks were initialized to $1.8 t/m^3$.

Inverse Distance Squared ($1/d^2$) grade interpolation was utilized for the Cu, Pb and Zn grade interpolation while Inverse Distance Cubed ($1/d^3$) was used for the Au and Ag grade interpolation, with the capped composites. The NSR values of the mineralized blocks were manipulated using formula below:

NSR = (Cu% x 38.54 + Pb% x 9.13 + Zn% x 15.81 + Au x 0.0 + Ag x 0.44) - 11.43

Grade blocks were interpolated using the parameters in Table 14.4.

Factors applied to the metal grades in this formula are based on metal prices and recoveries. The $11.43 is a fixed deduction in the NSR formula that accounts for the fixed cost of transporting the concentrates and fixed smelter treatment costs for all three concentrates.

The domain was divided into two sub-domains to assure the search ellipsoid orientations were aligned with the trend of the mineralized domain. 99.3% of the mineralized blocks were interpolated with the first two passes for Zn. The resulting Zn grade blocks selected are presented on the block model cross-sections and plans in Appendix-V. The Cu, Pb, Zn, Au and Ag grade blocks were combined into an NSR model and can be seen in Appendix VI.

Table 14.4 Block Model Interpolation Parameters								
Elements	**Pass**	**Strike Range (m)**	**Dip Range (m)**	**Across Dip Range (m)**	**Max # per Hole**	**Min # Sample**	**Max # Sample**	**Interpolation Method***
Zn	1	25	40	15	2	7	20	$1/d^2$
	2	40	60	25	2	5	20	
	3	80	120	50	2	1	20	
Cu	1	30	20	15	2	7	20	$1/d^2$
	2	50	30	20	2	5	20	
	3	100	60	40	2	1	20	
Pb	1	25	30	15	2	7	20	$1/d^2$
	2	40	45	25	2	5	20	
	3	80	90	50	2	1	20	
Ag	1	35	40	25	2	7	20	$1/d^3$
	2	55	60	40	2	5	20	
	3	110	120	80	2	1	20	
Au	1	20	15	20	2	7	20	$1/d^3$
	2	35	25	30	2	5	20	
	3	70	50	60	2	1	20	

***Note:** $1/d^2$ means inverse distance squared method*

14.11 RESOURCE CLASSIFICATION

In P&E's opinion, the drilling, assaying and exploration work used for this resource estimate are sufficient to indicate the Murray Brook deposit has reasonable potential for economic extraction and thus qualify it as a Mineral Resource under CIM definition standards. The resource classification was determined with Zn interpolation as Zn generated the highest proportionate NSR value in the block model. Based on the geology determination, semi-variogram performance and density of the drilling data, the Measured Resource category was justified for blocks interpolated by the pass one (Table 14.4) which was using at least seven composites from minimum of four drill holes within spacing of 25m along strike, 40m down dip and 15m on across dip direction. Indicated Resources were classified to the blocks interpolated with the pass two; while Inferred Resources were categorized to all remaining grade populated blocks. The classifications of some blocks have been manually adjusted to represent the resource classification more reasonably. The selected classification block cross-sections and plans are attached in Appendix VII.

14.12 RESOURCE ESTIMATE

The resource estimate was derived from applying an NSR cut-off grade to the block model and reporting the resulting tonnes and grade for potentially mineable areas. The following calculation demonstrates the rationale supporting the block NSR value that is used to determine the open pit potentially economic portions of the constrained mineralization.

14.12.1 Open Pit NSR Calculation

The most recent three year approximate trailing average metal prices as of Jan 31, 2013 are:

Cu Price	US$3.68/lb
Pb Price	US$1.00/lb

Zn Price	US$0.95/lb
Au Price	US$1,500/oz
Ag Price	US$29/oz
$US/$C Exchange Rate	$1.00

The projected recovery, payable, transportation and other parameters used in the calculation are:

Cu Concentrate Recovery	51%
Zn Concentrate Recovery	89%
Pb Concentrate Recovery	37%
Ag Concentrate Recovery	55%
Au Concentrate Recovery	0%
Concentrate Ratio	54:1
Cu Smelter Payable	95%
Pb Smelter Payable	95%
Zn Smelter Payable	85%
Ag Smelter Payable	90%
Au Smelter Payable	0%
Trucking/Storage/Ship Loading	US$30/t per WMT
Zn Smelter Treatment Charge	US$200/t per DMT
Cu Smelter Treatment Charge	US$150/t per DMT
Pb Smelter Treatment Charge	US$170/t per DMT
Humidity Factor	8.0%

These data were derived from the metallurgical reports and other open pit mining operations similar to that anticipated at Murray Brook.

NSR value was calculated based on the above parameters. The formula is

NSR = (Cu% x 38.54 + Pb% x 9.13 + Zn% x 15.81 + Au x 0.0 + Ag x 0.44) - 11.43.

14.12.2 Open Pit NSR Cut-off Basis

In the anticipated open pit operation, Mill Processing and G&A costs combine for a total of ($18 + $3) = $21 per tonne milled which becomes the NSR cut-off value.

14.12.3 Mineral Resource Constraining Parameters

In order for the constrained open pit mineralization in the Murray Brook resource model to be considered potentially economic, a first pass Whittle 4X pit optimization was carried out to create a pit shell for resource reporting purposes (See Appendix VIII) utilizing the criteria below:

Mineralized Material & Waste mining cost per tonne	$2.50
Overburden Mining Cost per tonne	$1.75
Process cost per tonne	$18.00
General & Administration cost per tonne of mill feed	$3.00

Process production rate (ore tonnes per year)	1,750,000
Pit slopes (overall wall angle)	45 degrees
Average Mineralized Rock Bulk Density	4.08/m^3
Waste Rock Bulk Density	2.70t/m^3
Overburden Bulk Density	1.80t/m^3

14.12.4 Mineral Resource Estimate

The resulting In-Pit Mineral Resource estimate for the Murray Brook project is summarized in the Table 14.5.

TABLE 14.5
MURRAY BROOK IN-PIT MINERAL RESOURCE ESTIMATE AT C$21/T NSR CUT-OFF(1-3)

Zone	Category	Tonnes ('000's)	Cu %	Cu M lb	Pb %	Pb M lb	Zn %	Zn M lb	Au g/t	Au K oz	Ag g/t	Ag M oz
Oxide	Measured	981.0	0.90	19.5	0.89	19.2	2.73	59.0	0.33	10.5	39.8	1.3
	Indicated	302.0	1.02	6.8	0.69	4.6	2.05	13.7	0.54	5.3	33.9	0.3
	M+I	1,283.0	0.93	26.3	0.84	23.8	2.57	72.7	0.38	15.8	38.4	1.6
	Inferred	4.0	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.4	0.0
Sulphide	Measured	11,306.0	0.40	100.7	1.04	258.3	2.97	741.2	0.50	182.7	42.5	15.4
	Indicated	6,578.0	0.57	82.9	0.91	131.6	2.32	336.8	0.74	155.5	40.3	8.5
	M+I	17,884.0	0.47	183.6	0.99	389.9	2.73	1,078.1	0.59	338.2	41.7	23.9
	Inferred	284.0	1.57	9.8	0.50	3.1	1.36	8.5	0.47	4.3	28.7	0.3

(1) Mineral Resources which are not mineral reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

(2) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.

(3) The Mineral Resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.

The NSR cut-off sensitivities to the In-Pit Mineral Resource estimate are tabulated in Table 14.6.

TABLE 14.6
OPEN PIT SENSITIVITY TO RESOURCE ESTIMATE OF THE MURRAY BROOK PROJECT

Zone	Category	Cut-Off NSR $/t	Tonnage tonnes	Cu %	Cu M lb	Pb %	Pb M lb	Zn %	Zn M lb	Au g/t	Au K oz	Ag g/t	Ag M oz
Oxide	Measured	100	358,681	1.20	9.5	1.53	12.1	4.76	37.6	0.31	3.5	64.43	0.7
		50	714,514	1.02	16.0	1.09	17.2	3.39	53.4	0.33	7.7	47.66	1.1
		45	770,240	0.99	16.8	1.04	17.7	3.23	54.8	0.34	8.4	45.85	1.1
		40	828,244	0.97	17.7	1.00	18.2	3.07	56.1	0.34	9.0	44.08	1.2
		35	883,714	0.94	18.4	0.96	18.6	2.94	57.3	0.34	9.6	42.55	1.2
		30	929,200	0.92	18.9	0.92	18.9	2.84	58.1	0.34	10.1	41.28	1.2
		25	959,643	0.91	19.3	0.90	19.1	2.77	58.7	0.34	10.3	40.40	1.2
		21	980,755	0.90	19.5	0.89	19.2	2.73	59.0	0.33	10.5	39.78	1.3
		15	1,002,207	0.89	19.6	0.87	19.3	2.68	59.2	0.33	10.7	39.13	1.3
		10	1,010,970	0.88	19.7	0.87	19.3	2.66	59.3	0.33	10.7	38.86	1.3
	Indicated	100	79,901	2.30	4.1	0.77	1.4	2.75	4.8	0.40	1.0	44.77	0.1
		50	219,083	1.25	6.1	0.77	3.7	2.35	11.4	0.54	3.8	38.85	0.3
		45	238,103	1.19	6.3	0.75	4.0	2.28	12.0	0.54	4.1	37.64	0.3
		40	253,796	1.15	6.4	0.74	4.1	2.22	12.4	0.54	4.4	36.60	0.3
		35	272,203	1.10	6.6	0.72	4.3	2.16	13.0	0.54	4.7	35.65	0.3

Zone	Category	Cut-Off NSR $/t	Tonnage tonnes	Cu %	Cu M lb	Pb %	Pb M lb	Zn %	Zn M lb	Au g/t	Au K oz	Ag g/t	Ag M oz
		30	287,446	1.06	6.7	0.71	4.5	2.11	13.4	0.55	5.1	34.82	0.3
		25	296,972	1.03	6.8	0.70	4.6	2.07	13.6	0.54	5.2	34.18	0.3
		21	301,728	1.02	6.8	0.69	4.6	2.05	13.7	0.54	5.3	33.86	0.3
		15	305,211	1.01	6.8	0.69	4.6	2.04	13.7	0.54	5.3	33.60	0.3
		10	306,717	1.01	6.8	0.69	4.6	2.03	13.7	0.54	5.4	33.46	0.3
	Inferred	100	3,706	3.90	0.3	0.18	0.0	0.61	0.0	0.45	0.1	26.40	0.0
		50	4,100	3.73	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.45	0.0
		45	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		40	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		35	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		30	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		25	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		21	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		15	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
		10	4,158	3.69	0.3	0.17	0.0	0.57	0.1	0.43	0.1	25.40	0.0
Sulphide	Measured	100	2,645,129	0.41	24.0	2.12	123.3	6.26	364.9	0.58	48.9	79.59	6.8
		50	7,433,505	0.42	68.3	1.34	219.8	3.87	633.8	0.54	129.9	53.55	12.8
		45	8,150,214	0.42	74.9	1.27	229.0	3.67	659.5	0.54	140.5	51.18	13.4
		40	8,898,154	0.42	81.6	1.21	237.7	3.48	683.5	0.53	151.1	48.91	14.0
		35	9,651,043	0.41	88.2	1.15	245.3	3.31	704.4	0.52	161.5	46.78	14.5
		30	10,362,032	0.41	94.0	1.10	251.4	3.16	721.8	0.51	171.0	44.88	15.0
		25	10,941,894	0.41	98.4	1.06	255.8	3.04	734.2	0.51	178.3	43.39	15.3
		21	11,306,015	0.40	100.7	1.04	258.3	2.97	741.2	0.50	182.7	42.47	15.4
		15	11,696,931	0.40	102.9	1.01	260.5	2.90	747.7	0.50	186.9	41.48	15.6
		10	11,901,885	0.40	103.9	1.00	261.5	2.86	750.4	0.49	188.8	40.94	15.7
	Indicated	100	1,059,076	0.86	20.0	1.83	42.7	5.03	117.4	0.88	30.0	72.47	2.5
		50	4,540,579	0.63	63.5	1.12	111.8	2.84	283.9	0.84	123.0	48.68	7.1
		45	4,983,256	0.62	68.1	1.07	117.5	2.72	298.5	0.82	131.2	46.77	7.5
		40	5,420,390	0.61	72.4	1.02	122.3	2.61	311.4	0.80	138.7	44.94	7.8
		35	5,815,306	0.59	76.3	0.98	126.1	2.51	321.6	0.78	144.9	43.36	8.1
		30	6,135,008	0.59	79.2	0.95	128.7	2.43	328.8	0.76	149.4	42.08	8.3
		25	6,398,280	0.58	81.5	0.93	130.5	2.37	333.8	0.74	153.1	41.05	8.4
		21	6,578,261	0.57	82.9	0.91	131.6	2.32	336.8	0.74	155.5	40.33	8.5
		15	6,766,188	0.56	84.2	0.89	132.6	2.28	339.4	0.72	157.4	39.54	8.6
		10	6,855,652	0.56	84.7	0.88	133.0	2.25	340.5	0.72	158.3	39.14	8.6
	Inferred	100	95,998	2.73	5.8	0.63	1.3	1.75	3.7	0.49	1.5	37.30	0.1
		50	207,120	1.93	8.8	0.56	2.5	1.52	6.9	0.53	3.5	32.53	0.2
		45	223,903	1.84	9.1	0.54	2.7	1.48	7.3	0.52	3.7	31.70	0.2
		40	239,206	1.77	9.3	0.53	2.8	1.45	7.7	0.51	3.9	30.95	0.2
		35	254,458	1.70	9.5	0.52	2.9	1.43	8.0	0.50	4.1	30.18	0.2
		30	268,595	1.63	9.7	0.51	3.0	1.40	8.3	0.49	4.2	29.49	0.3
		25	278,112	1.59	9.8	0.50	3.1	1.38	8.5	0.48	4.3	29.04	0.3
		21	284,487	1.57	9.8	0.50	3.1	1.36	8.5	0.47	4.3	28.74	0.3
		15	292,234	1.53	9.9	0.49	3.2	1.34	8.6	0.46	4.4	28.36	0.3
		10	294,655	1.52	9.9	0.49	3.2	1.33	8.6	0.46	4.4	28.24	0.3

TABLE 14.6
OPEN PIT SENSITIVITY TO RESOURCE ESTIMATE OF THE MURRAY BROOK PROJECT

14.13 CONFIRMATION OF ESTIMATE

The block model was validated using a number of industry standard methods including visual and statistical methods. These included:

- Visual examination of composite and block grades on plans and sections on-screen and review of estimation parameters including:
  - Number of composites used for estimation;
  - Number of holes used for estimation;
  - Distance to the nearest composite;
  - Number of passes used to estimate grade;
  - Mean value for composites used.

- As a test of the reasonableness of the Mineral Resource estimate, the average grade for the block models were compared to the average grade of length weighted assays and capped composites within the constrained solids. As shown in Table 14.7, the block model global mean grade of Cu is slightly higher than the capped composite average grades while Zn, Pb, Ag and Au are slightly lower than their average of capped composites. This is possible due to the local grade spatial effect. In P&E's opinion, the block model grade will be more spatially representative.

TABLE 14.7
COMPARISON OF LENGTH WEIGHTED ASSAYS & CAPPED COMPOSITES TO BLOCK MODEL

Data Type	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Length Weighted Assay	0.48	0.99	2.70	0.56	41.2
Capped Composites	0.48	0.98	2.70	0.56	41.0
Block Model	0.52	0.91	2.51	0.56	38.8

A volumetric comparison was performed with the block model volume of the model blocks versus the geometric calculated volume of the domain solids, as detailed below:

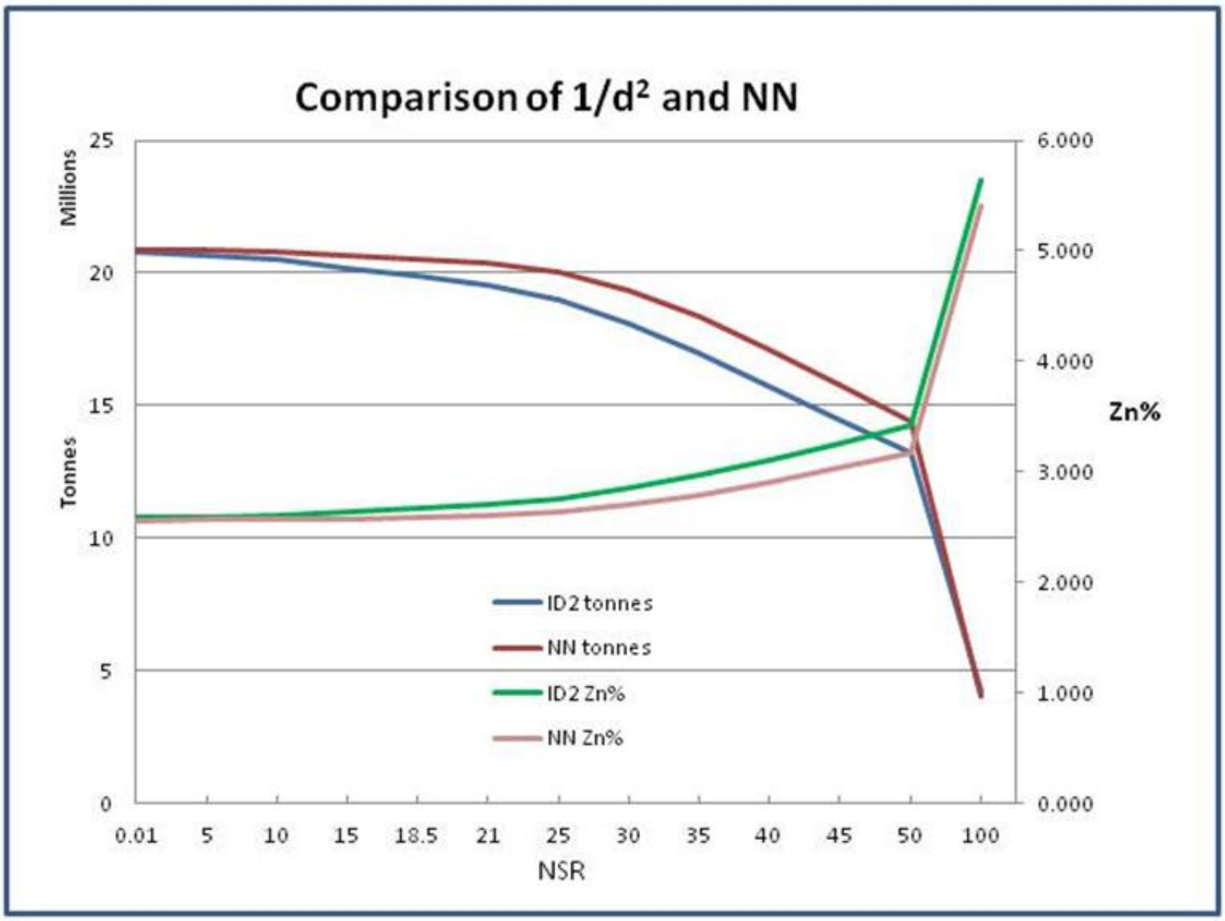
Block Model Volume	5,088,643m^3
Geometric Domain Volume	5,090,429m^3
Difference	0.04%

The sensitivity of the Mineral Resource to NSR cut-off grades was evaluated by constraining the Mineral Resource within an optimized pit shell demonstrated in Appendix VIII. At a cut-off of NSR $21/tonne, within the pit shell, there is a reduction of approximately 0.6% of the global tonnage of the Murray Brook resources.

Comparison of grade models interpolated with Inverse Distance Squared ($1/d^2$) and Nearest Neighbour (NN) on global resource basis at cut-off of NSR $21/tonne, as shown in Table 14.8 and Figure 14.3, the $1/d^2$ method resulted in higher average grades and lower tonnage than that interpolated with NN, while contained metals are similar.

TABLE 14.8 COMPARISON OF RESOURCES INTERPOLATED WITH 1/D^2 AND NN METHODS		
Interpolation Model	**1/d^2**	**NN**
Tonnes ('000s)	19,574	20,397
Zn%	2.71	2.61
Cu%	0.51	0.51
Pb%	0.97	0.94
Ag g/t	41.3	39.5
Au g/t	0.57	0.56

Figure 14.3 Comparison of Tonnes and Zn Grade Interpolated with 1/d^2 and NN Method



15.0 MINERAL RESERVE ESTIMATES

The work undertaken on the Murray Brook Project to date is considered to be at conceptual levels of study only. As such, and according to the NI 43-101 Disclosure Guidelines, it is not possible to declare a mineral reserve.

16.0 MINING METHODS

16.1 INTRODUCTION

The Murray Brook deposit is relatively shallow in depth and lends itself to conventional open pit mining methods. A single open pit will be developed that will have a maximum depth of approximately 340 m from the highest point (550 masl) to lowest point (212 masl).

A production plan has been developed for the Project that has been used in the financial analysis. This production plan utilizes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the Inferred Resources will be upgraded to a higher resource category.

The open pit will require the excavation of four different materials:

- Overburden (placed into an overburden stockpile);
- Waste Rock (placed into a waste rock dump);
- Oxide Waste (mineralized material not defined as processable and placed into a separate stockpile);
- Sulphide Mill Feed (processed through the plant);
- The development of the mine production schedule entailed several sequential steps. These are:
    - Run pit optimizations to select the optimal pit shell;
    - Design an operational pit (with ramps and benches) based on the optimal shell;
    - Develop internal pit phases (push-backs) to smooth the annual production tonnages;
    - Develop a life-of-mine mining schedule;
    - Develop a life-of-mine processing schedule.

16.2 PIT OPTIMIZATIONS

A series of pit optimizations were completed using the CAE NPV Scheduler software package. This optimization process produces a series of nested pit shells containing mineralized material that is economically mineable according to a set of physical and economic design parameters. The pit shell which produces the highest undiscounted cash flow supported by a reasonable incremental pit shell NPV is selected as the optimum shell to be used for mine design.

A series of pit optimizations were run using the parameters shown in Table 16.1 and with a wide range of revenue factors (from 18% to 100%). Metal prices are based on an April 30, 2013 three-year trailing average that has been used in the NSR formula described in Section 14.

The optimization results are shown graphically in Figure 16.1 (NPV0%) and Figure 16.2 (tonnes). These results provide an estimate for the potentially economic portion of the sulphide Mineral Resource mill feed for each revenue factor as well as potential strip ratio. The optimized pit shell forms the basis for the actual pit design and in this case the 81% revenue factor pit was selected as the optimal pit. The sulphide feed tonnage NPV curve is flattening off and minimal NPV or tonnage is gained by selecting a pit any larger than 81%.

The quantities reported represent the potentially economic portion of the mineral resources contained in the optimized pit shell; however, the quantity used in the production schedule will be derived from an operational pit design. The potentially economic portion of the mineral resources consists of varying amounts of both Indicated and Inferred Resources.

Table 16.1 Pit Optimization Parameters		
Copper Price	$US/lb	$3.68
Lead Price	$US/lb	$1.00
Zinc Price	$US/lb	$0.95
Gold Price	$US/oz	$1,500
Silver Price	$US/oz	$29.00
Overburden Mining Cost	$/t	$1.75
Waste Rock Mining Cost	$/t	$2.50
Oxide Waste Mining Cost	$/t	$2.50
Sulphide Mining Cost	$/t	$2.50
Processing	$/t milled	$18.00
G&A	$/t milled	$3.00
NSR Cut-off Value	$/t	$21.00
Pit Slopes for Optimization	Overburden/Rock	30°/45°

Figure 16.1 Pit Optimization NPV0%

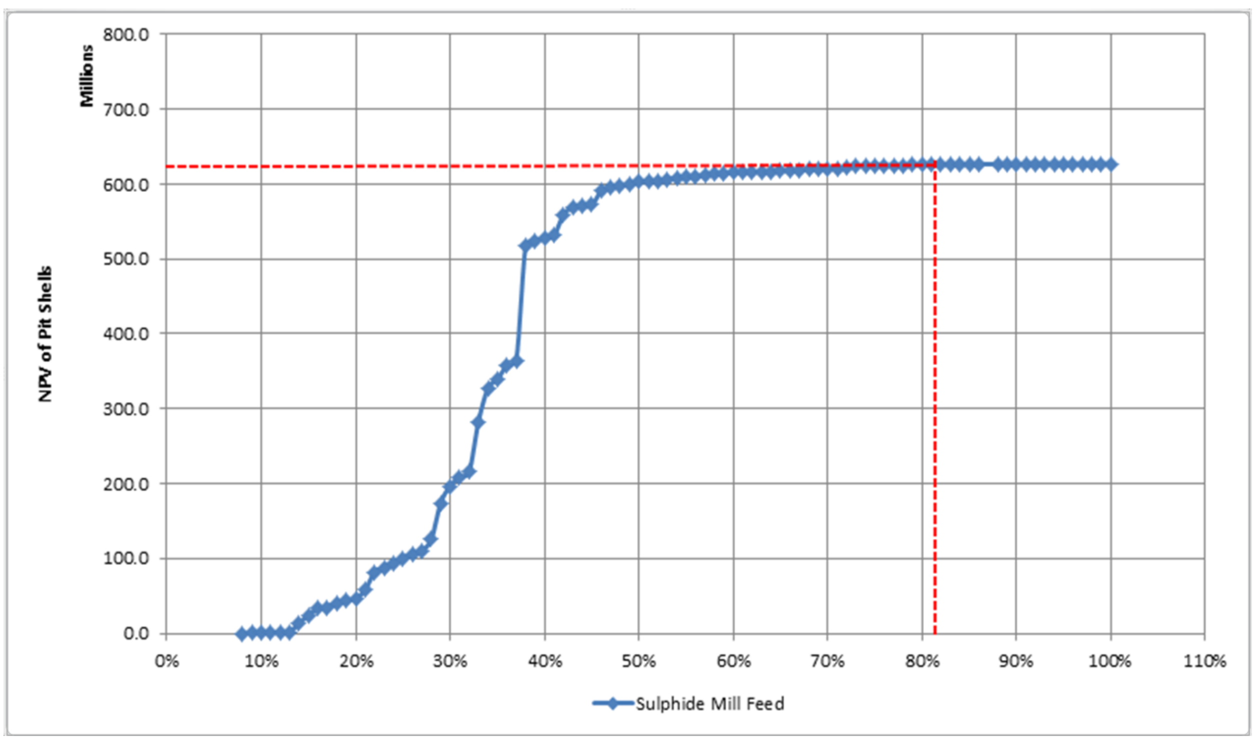
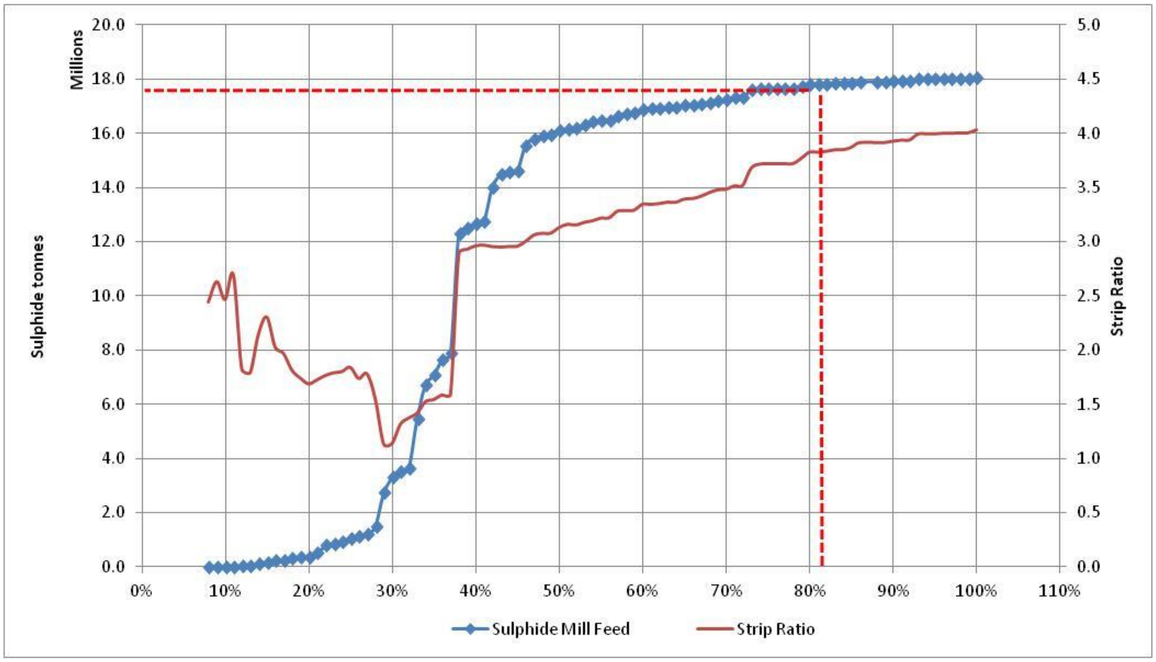


Figure 16.2 Pit Optimization Tonnages

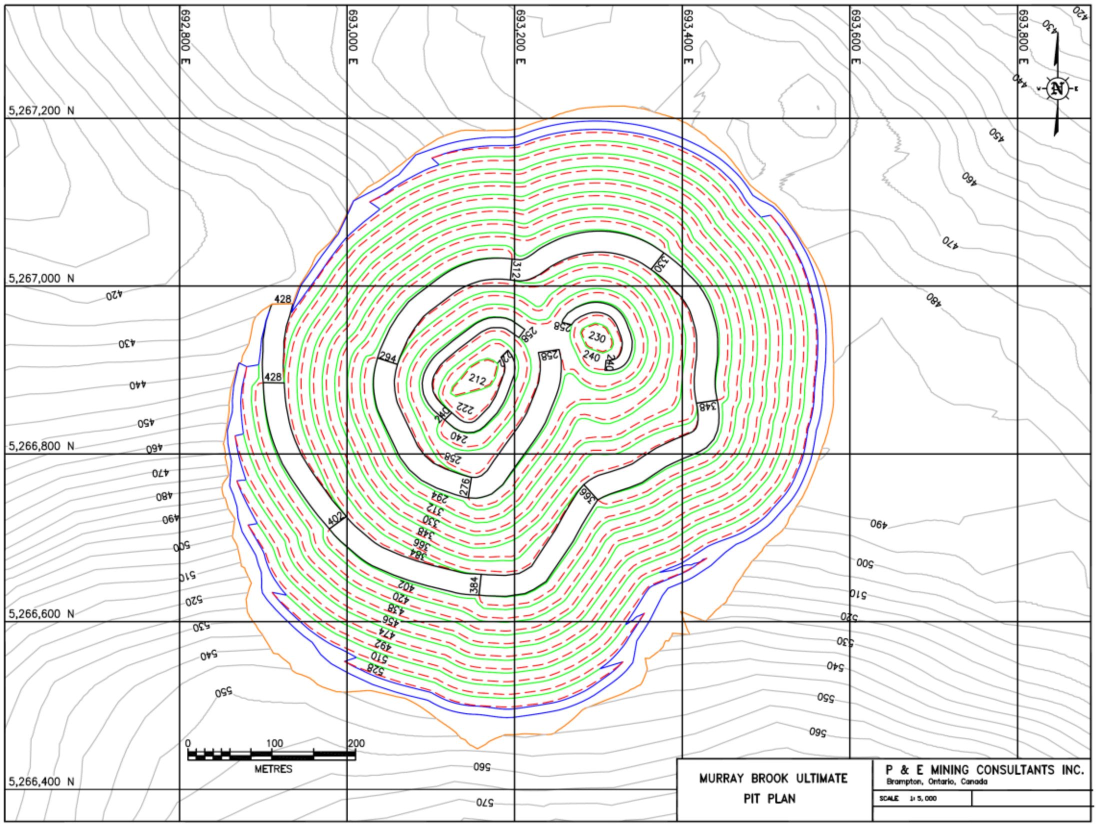


16.3 PIT DESIGNS

An operational pit design was created using the selected optimized shell as the basis. Benches and haul roads were added, according to the guidelines shown in Table 16.2. Figure 16.3 presents a plan view of the final pit shell.

Table 16.2 Pit Design Parameters	
Haul Road Width (double lane)	25 m
Haul Road Width (single lane)	12 m
Haul Road Grade (maximum)	10%
Overburden Slope	
Bench Height	6 m
Bench Face Angle	45º
Catch-bench Width	4.4 m
Inter-ramp Angle	30º
Rock Slope	
Bench Height (triple bench)	18 m
Bench Face Angle	75º
Catch-bench Width	10.3 m
Inter-ramp Angle	50º

Figure 16.3 Final Pit Design



16.4 GEOTECHNICAL STUDIES

No geotechnical studies have been completed at this PEA stage so pit slope angles used (see Table 16.2) were estimated based on P&E's experience with similar rock types.

16.5 HYDROGEOLOGICAL STUDIES

No hydrogeological studies have been completed at this PEA stage to assess groundwater conditions.

16.6 MINING DILUTION AND LOSSES OF MINERALIZED MATERIAL

The amount of dilution that occurs during mining will be dependent on the nature of the mineralized zones being mined.

In order to estimate dilution, several different representative bench plans were selected for analysis. For each bench plan, a 2-metre wide zone (halo) of diluting material was assumed around the mineralized domains. The grade was estimated for this zone and applied as the diluting grade. This average percent dilution was then applied to the in-situ tonnes & grade to arrive at diluted tonnes & grade. Dilution parameters are summarized in Table 16.3. Based on P&E's experience with similar mining operations and rock types, mill feed losses were assumed at 3%.

Table 16.3 Dilution & Mineralized material Loss Criteria	
	Sulphide Mill Feed
Mineralized Material Loss (%)	3.0%
Dilution (%)	11.7%
Diluted Grade - Copper	0.12%
Diluted Grade - Lead	0.07%
Diluted Grade - Zinc	0.20%
Diluted Grade - Gold	0.12 g/t
Diluted Grade - Silver	4.65 g/t

16.7 POTENTIALLY MINEABLE PORTION OF THE MINERAL RESOURCES

After the pit design was created, the potentially mineable portion of the mineral resource and waste tonnages within it were reported. This tonnage is summarized in Table 16.4. This tonnage is used as the basis for the PEA production schedule and incorporates dilution.

Table 16.4 Potentially Mineable Portion of the Resource (Diluted)	
Total Material in Pit (t)	100,833,000
Overburden (t)	8,838,000
Oxides Waste (t)	1,531,000
Waste Rock (t)	71,516,000
Total Waste (t)	73,047,000
Strip Ratio	4.32
Sulphide Feed (t) diluted	18,948,000
NSR ($/t)	$68.70
Au (g/t)	0.53
Ag (g/t)	37.7
Cu (%)	0.43
Pb (%)	0.89
Zn (%)	2.46

Note: The potentially mineable portion of the Mineral Resource tonnage used in the PEA contains both Indicated and Inferred Resources. The reader is cautioned that Inferred 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 no certainty that value from such Resources will be realized either in whole or in part.

16.8 PIT PHASES

In order to better distribute the annual waste mining tonnages and to accelerate the access into the mill feed material, the pit was sub-divided into three phases. These are shown in Figure 16.4 (plan view) and Figure 16.5 (cross-section view). The tonnages and grades contained within each of the phases are shown in Table 16.5.

Figure 16.4 Pit Phase Plan

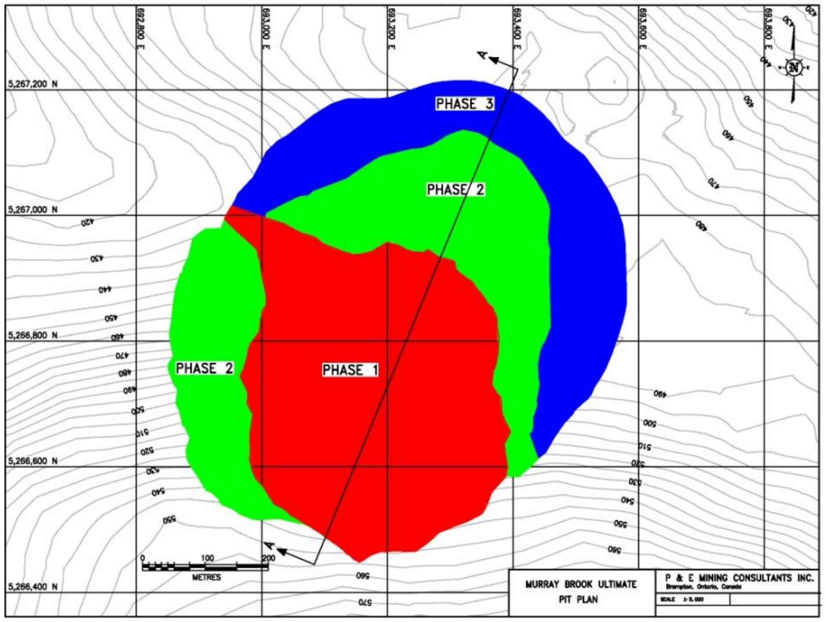


Figure 16.5 Pit Phase Cross-section

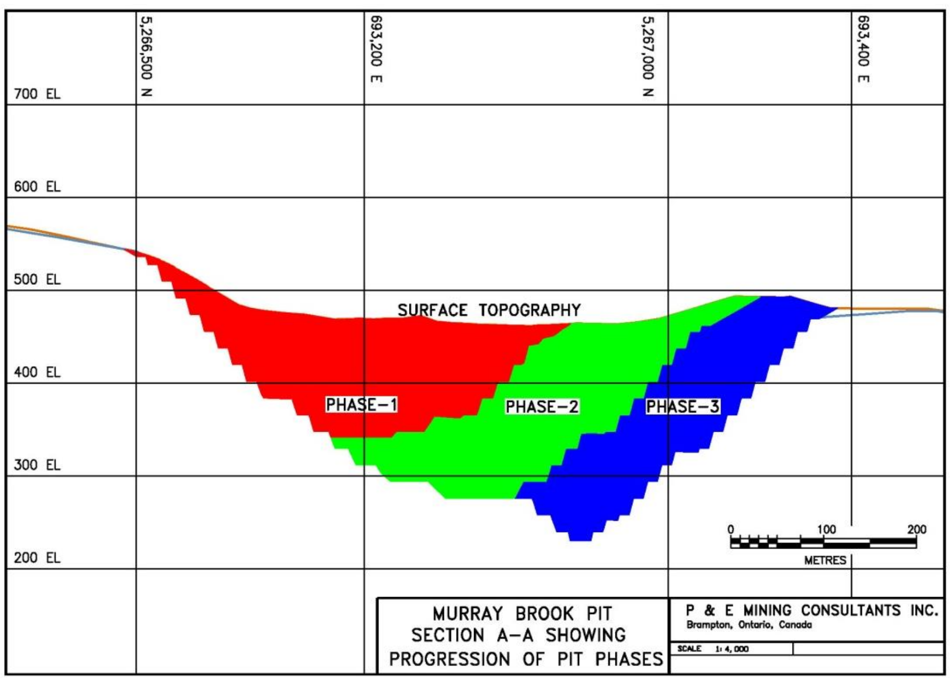


Table 16.5 Pit Phase Tonnages					
	Units	**Phase 1**	**Phase 2**	**Phase 3**	**Total**
Total Material in Pit	'000's of t	22,359	37,471	41,003	100,833
Overburden (t)	'000's of t	5,465	2,099	1,274	8,838
Oxides Waste (t)	'000's of t	1,331	199	1	1,531
Waste Rock (t)	'000's of t	9,136	28,800	33,580	71,516
Total Waste (t)	'000's of t	10,467	28,999	33,581	73,047
Strip Ratio		2.48	4.88	5.67	4.32
Sulphide Feed	'000's of t	6,426	6,373	6,148	18,948
NSR ($/t)	$/t	$66.54	$62.96	$76.91	$68.70
Au (g/t)	g/t	0.31	0.51	0.77	0.53
Ag (g/t)	g/t	31.3	36. 8	45.2	37.7
Cu (%)	%	0.50	0.33	0.46	0.43
Pb (%)	%	0.73	0.87	1.07	0.89
Zn (%)	%	2.42	2.38	2.59	2.46

16.9 PRODUCTION SCHEDULE

The mine production schedule consists of one pre-production year for pre-stripping and then ten years of actual mine production.

The target milling rate is approximately 2,000,000 tonnes per year, or approximately 6,000 t/day. Daily mining rates of mineralized material and waste combined will range from 12,000 t/d to 32,000 t/d and average about 27,000 t/day.

Table 16.6 presents the mine production schedule in detail. Figure 16.6 presents the mill feed tonnes mined from each phase showing the phasing sequence by year.

Figure 16.6 Pit Phase Sequence

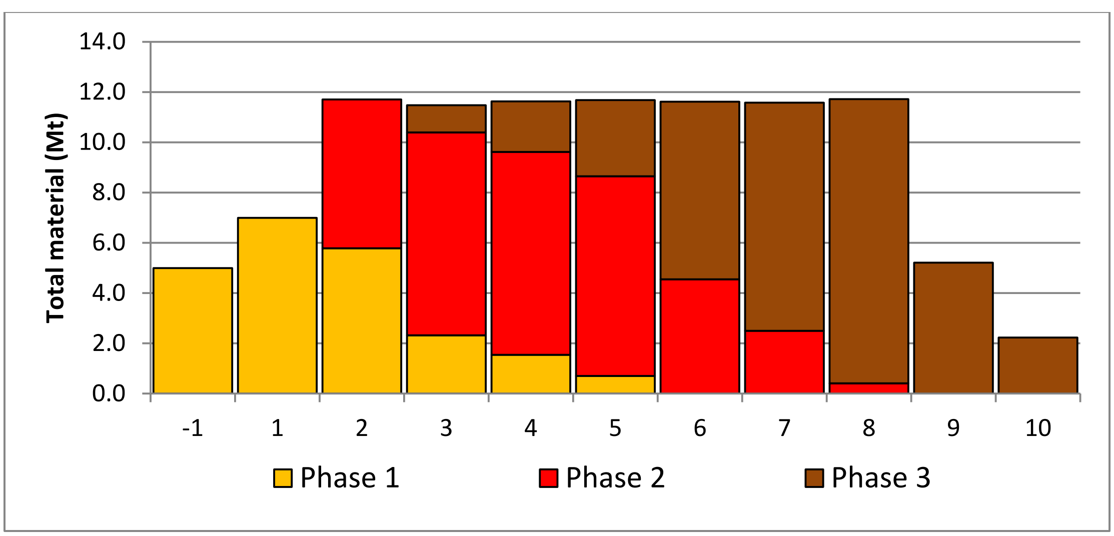


TABLE 16.6
MINE PRODUCTION SCHEDULE

Year	Total	-1	1	2	3	4	5	6	7	8	9	10
Total Material in Pit	100,832,826	5,000,344	6,999,712	11,702,658	11,477,373	11,632,308	11,680,478	11,612,517	11,577,988	11,714,691	5,205,811	2,228,946
Overburden (t)	8,837,667	2,072,741	2,795,181	2,146,079	1,294,412	440,948	35,332	52,974				
Oxides Waste (t)	1,531,392	82,303	750,008	668,913	28,702		1,466					
Waste Rock (t)	71,516,093	2,720,504	2,249,572	6,893,975	8,160,627	9,197,719	9,650,117	9,565,725	9,584,351	9,720,962	3,211,549	560,992
Total Waste (t)	81,885,152	4,875,548	5,794,761	9,708,967	9,483,741	9,638,667	9,686,915	9,618,699	9,584,351	9,720,962	3,211,549	560,992
Strip Ratio	4.32	39.07	4.81	4.87	4.76	4.83	4.86	4.82	4.81	4.88	1.61	0.34
Total Feed Sulphides (t)	18,947,674	124,796	1,204,951	1,993,691	1,993,632	1,993,641	1,993,563	1,993,818	1,993,637	1,993,729	1,994,262	1,667,954
NSR ($/t)	$68.70	$95.38	$51.91	$64.39	$63.26	$67.71	$58.46	$65.04	$67.79	$63.62	$77.55	$104.89
AU (g/t)	0.53	0.33	0.30	0.31	0.46	0.48	0.46	0.43	0.46	0.74	0.73	0.88
AG (g/t)	37.7	21.1	18.8	31.5	35.1	37.5	32.4	37.7	38.6	40.4	43.8	57.8
CU (%)	0.43	2.22	0.89	0.50	0.34	0.38	0.37	0.28	0.26	0.46	0.46	0.42
PB (%)	0.89	0.18	0.32	0.69	0.82	0.90	0.76	0.88	0.95	0.90	1.06	1.51
ZN (%)	2.46	0.66	1.13	2.31	2.44	2.51	2.19	2.61	2.75	1.99	2.67	3.85
PHASE 1												
Total Material in Pit	22,358,567	5,000,344	6,999,712	5,780,311	2,326,993	1,545,655	705,552					
Overburden (t)	5,465,022	2,072,741	2,795,181	596,954	145.8							
Oxides Waste (t)	1,331,218	82,303	750,008	498,473	433.89							
Waste Rock (t)	9,135,999	2,720,504	2,249,572	2,741,784	934,321	402,254	87,564					
Total Waste (t)	15,932,239	4,875,548	5,794,761	3,837,211	934,901	402,254	87,564					
Strip Ratio	2.48	39.07	4.81	1.97	0.67	0.35	0.14					
Total Feed - Sulphides (t)	6,426,328	124,796	1,204,951	1,943,100	1,392,092	1,143,401	617,988					
NSR ($/t)	$66.54	$95.38	$51.91	$64.90	$68.15	$75.87	$73.52					
AU (g/t)	0.31	0.33	0.30	0.30	0.33	0.33	0.31					
AG (g/t)	31.3	21.1	18.8	31.5	33.8	39.1	37.5					
CU (%)	0.50	2.22	0.89	0.50	0.34	0.25	0.19					
PB (%)	0.73	0.18	0.32	0.70	0.84	1.02	1.00					
ZN (%)	2.42	0.66	1.13	2.32	2.78	3.23	3.29					
PHASE 2		0.6648	1.1338	2.3239	2.7844	3.2297	3.2949					
Total Material in Pit	37,471,331			5,922,347	8,068,909	8,069,389	7,948,723	4552011	2500281	409671		
Overburden (t)	2,098,937			1,549,125	474,433	75,379						
Oxides Waste (t)	198,708			170,440	28,268							
Waste Rock (t)	28,800,191			4,152,191	6,964,668	7,143,770	6,604,465	2,826,440	981,295	127,362		
Total Waste (t)	31,097,836			5,871,756	7,467,369	7,219,149	6,604,465	2,826,440	981,295	127,362		
Strip Ratio	4.88			116.06	12.41	8.49	4.91	1.64	0.65	0.45		
Total Feed - Sulphides (t)	6,373,495			50,591	601,540	850,240	1,344,258	1,725,571	1,518,986	282,309		
NSR ($/t)	$62.96			$45.03	$51.93	$56.74	$51.71	$66.89	$71.06	$94.38		
AU (g/t)	0.51			0.76	0.78	0.68	0.52	0.41	0.40	0.41		
AG (g/t)	36.8			31.7	38.1	35.3	30.2	37.9	38.6	54.2		
CU (%)	0.33			0.29	0.35	0.56	0.45	0.27	0.20	0.14		
PB (%)	0.87			0.65	0.78	0.73	0.65	0.90	1.02	1.49		
ZN (%)	2.38			1.61	1.64	1.54	1.69	2.73	3.08	3.97		
PHASE 3												
Total Material in Pit	41,002,928				1,081,471	2,017,264	3,026,203	7060506	9077707	11305020	5205811	2228946
Overburden (t)	1,273,708				819,833	365,569	35,332	52,974				
Oxides Waste (t)	1,466						1,466					
Waste Rock (t)	33,579,903				261,638	1,651,695	2,958,088	6,739,285	8,603,056	9,593,600	3,211,549	560,992
Total Waste (t)	34,855,077				1,081,471	2,017,264	2,994,886	6,792,259	8,603,056	9,593,600	3,211,549	560,992
Strip Ratio	5.67						95.63	25.32	18.13	5.61	1.61	0.34
Total Feed - Sulphides (t)	6,147,851						31,317	268,247	474,651	1,711,420	1,994,262	1,667,954
NSR ($/t)	$76.91						$50.82	$53.16	$57.35	$58.54	$77.55	$104.89
AU (g/t)	0.77						0.51	0.60	0.62	0.79	0.73	0.88
AG (g/t)	45.2						32.0	36.3	38.4	38.2	43.8	57.8
CU (%)	0.46						0.40	0.33	0.47	0.51	0.46	0.42
PB (%)	1.07						0.61	0.77	0.74	0.80	1.06	1.51
ZN (%)	2.59						1.72	1.82	1.71	1.67	2.67	3.85

16.10 OPEN PIT MINING PRACTICES

It is assumed that the Murray Brook pit will be operated as an owner-operated conventional open pit mining operation.

16.10.1 Drilling and Blasting

There are three different competencies of material that must be mined. Similar competencies apply to both waste material or mill feed. The three types are; (a) overburden, (b) oxide, and (c) hard rock, with overburden being the least competent rock and harder rock being the most competent. Sulphide mill feed will consist of hard rock.

It is assumed that the overburden is free digging so no drilling or blasting is required. The oxide and hard rock are more competent so it is assumed that drilling and blasting will be required for both these materials.

Drilling will be carried out using down-hole hammer drills and nominal hole diameters of 100 mm with an operating bench height of 6 metres.

Blasting of the rock will be carried out using an ammonium nitrate fuel oil mixture (ANFO), which will be loaded by a bulk explosives truck directly into the drill holes. Blast initiation will be carried out using non-electric detonators and booster charges. The assumed powder factor is 0.10 kg/t for oxide material and 0.18 kg/t for hard rock materials.

16.10.2 Loading and Hauling

Diesel powered hydraulic front shovel excavators with an 11.5 m^3 heavy rock bucket will be used to free dig the overburden and excavate and load the blasted harder rock. The excavators will load the 90-tonne off-highway haul trucks in a 3 to 7 pass loading match depending on the density of material being handled.

Loading operations will also be supported by a wheel loader with a 12-m^3 rock bucket although only about 15%-20% of the mine truck loading will be done by the wheel loader.

16.10.3 Pit Dewatering

The pit will likely see some groundwater seepage in addition to regular precipitation events and snowmelt. An allowance has been included in the operating and capital costs for a pit dewatering system to pump water from pit sumps. No quantitative information was available to adequately predict the expected water inflow into the pit so this allowance is based on P&E's experience with similar operations.

Skid or trailer mounted centrifugal pumps will be staged up the side of the pit to remove water from the pit sump locations during the pit development.

16.10.4 Auxiliary Pit Services and Support Equipment

The primary mining operations will be supported by a fleet of support equipment consisting of Caterpillar D8 size class bulldozers with ripper attachments, Caterpillar 14 M class graders as well as a Caterpillar 814 class wheel dozer, water truck, maintenance vehicles, and service vehicles. A list of major and support equipment for auxiliary services is provided in Table 16.8.

16.10.5 Waste Dumps

The pit will require the development of several waste disposal locations. These will be of varying size and are listed in Table 16.7.

TABLE 16.7
WASTE DISPOSAL AREAS

	Tonnes	Density Placed (t/m3)	Cubic metres
Overburden Stockpile	8,838,000	1.44	6,137,500
Oxide Waste Stockpile	1,531,000	2.40	637,917
Waste Rock Dump	71,516,000	2.16	33,109,259

16.10.6 Mine Equipment

The mine operations at Murray Brook will employ methods and technologies used at other locations around Canada where similar rock and climatic conditions are found. Table 16.8 lists the mine equipment fleet requirements on a yearly basis.

TABLE 16.8
MINING EQUIPMENT FLEET
Murray Brook, NB

	-1	1	2	3	4	5	6	7	8	9	10
Drill, 100 mm, Crawler	2	2	4	4	4	4	4	4	4	2	2
ANFO Delivery truck, 12 t	1	1	1	1	1	1	1	1	1	1	1
Stemming Truck, 15 t	1	1	1	1	1	1	1	1	1	1	1
Transport for detonators	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Shovel, 11.5 m3	1	1	1	1	1	1	1	1	1	1	1
Wheel Loader 12m3 (C993)	1	1	1	1	1	1	1	1	1	1	1
Haul Truck 90t (C777)	7	5	8	8	8	8	8	9	10	5	4
Personnel van/bus	1	1	1	1	1	1	1	1	1	1	1
Rubber T Dozer 814-class 12' blade	1	1	1	1	1	1	1	1	1	1	1
Flat Deck w Hiab	1	1	1	1	1	1	1	1	1	1	1
Dozer D8	2	2	2	2	2	2	2	2	2	1	1
Welding Truck	1	1	1	1	1	1	1	1	1	1	1
Excavator, 2 cu.m (CAT 336E)	1	1	1	1	1	1	1	1	1	1	1
Fuel Truck	1	1	1	1	1	1	1	1	1	1	1
Grader 14H-class 14' blade	2	2	2	2	2	2	2	2	2	2	2
Dump Truck, 10 t	1	1	1	1	1	1	1	1	1	1	1
Light plant	4	4	4	4	4	4	4	2	2	2	2
Lube truck	1	1	1	1	1	1	1	1	1	1	1
Mechanic truck	1	1	1	1	1	1	1	1	1	1	1
Pickup truck	7	7	7	7	7	7	7	7	7	7	7
Pit Water Pumps	4	4	4	4	4	4	4	4	4	4	4
Tire manipulator	1	1	1	1	1	1	1	1	1	1	1

TABLE 16.8 MINING EQUIPMENT FLEET											
Murray Brook, NB											
	-1	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Wheel Loader 3.8-m3	1	1	1	1	1	1	1	1	1	1	1
Truck & Trailer, 200t	1	1	1	1	1	1	1	1	1	1	1
Water truck HD325 40,000 litre	1	1	1	1	1	1	1	1	1	1	1
Crane, Grove 40t		1	1	1	1	1	1	1	1	1	1

16.10.7 Support Facilities

The Murray Brook mine will require mine offices, change house facilities, maintenance facilities, warehousing and cold storage areas. The mine office will provide for mine management, engineering, geology, mine maintenance services. These are part of the project infrastructure described in Section 18.

A maintenance shop which will provide pit support services will be located near the plant site. The mine maintenance facility will consist of a truck shop which will include a wash facility, welding equipment and a dedicated preventive maintenance bay. The facility will have adjoining indoor parts storage and tool crib. A fuel and lube station will be conveniently located near the maintenance facility and main haul road for equipment access. A mobile truck mounted fuel and lube system will be available to service less mobile equipment in the field.

16.10.8 Mining Manpower

The Murray Brook mining operation will require a workforce ranging approximately 72 to140 personnel, as summarized in Table 16.9. Manpower numbers will fluctuate as mining volumes changes and operating equipment needs change.

The mining operations manning list includes all aspects involved with the open pit operations, including;

- Senior mine and maintenance supervision
- Office technical staff, engineering, geology, surveying, etc.
- Clerical, maintenance planning, training
- Mine operations crews
- Mine support crews
- Mine maintenance crews

TABLE 16.9
MINING MANPOWER

Year	-1	1	2	3	4	5	6	7	8	9	10
Driller	4	5	11	12	13	13	13	13	14	6	3
Driller Helper	4	5	11	12	13	13	13	13	14	6	3
Blasting Foreman	1	1	1	1	1	1	1	1	1	1	1
Blaster	2	2	2	2	2	2	2	2	2	2	2
Bulk Truck Operator	1	1	2	2	2	3	2	3	3	1	1
Laborer	2	2	2	2	2	2	2	2	2	2	2
Truck Drivers	11	15	26	25	26	27	28	30	33	16	7
Shovel Operator	2	3	4	4	4	4	4	4	4	2	1
Loader Operator	1	1	2	2	2	2	2	2	2	1	1
HD Mechanic	7	11	17	17	18	18	18	18	19	11	4
Grader Operator	4	4	4	4	4	4	4	4	4	4	4
Dozer Operator	8	8	8	8	8	8	8	4	4	4	4
Water Truck Operator	4	4	4	4	4	4	4	4	4	4	4
Utility Operators	4	4	4	4	4	4	4	4	4	4	4
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1
Mine General Foremen	1	1	1	1	1	1	1	1	1	1	1
Mine Foremen	4	4	4	4	4	4	4	4	4	4	4
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1
Maintenance General Foreman	1	1	1	1	1	1	1	1	1	1	1
Maintenance Foreman	1	1	1	1	1	1	1	1	1	1	1
Planner	1	1	1	1	1	1	1	1	1	1	1
Welder	2	2	2	2	2	2	2	2	2	2	2
Lead Electrician	1	1	1	1	1	1	1	1	1	1	1
Electrician	2	2	2	2	2	2	2	2	2	2	2
Gas Mechanic	2	2	2	2	2	2	2	2	2	2	2
Tireman	2	2	2	2	2	2	2	1	1	1	1
Partsman	2	2	2	2	2	2	2	2	2	2	2
Laborer	4	4	4	4	4	4	4	4	4	4	4
Equipment Trainer	1	1	1	1	1	1	1	1	1	1	1
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	1
Tailings Engineer	1	1	1	1	1	1	1	1	1	1	1
Geologist	2	2	2	2	2	2	2	2	2	2	2
Surveyor	1	1	1	1	1	1	1	1	1	1	1
Survey Technician	1	1	1	1	1	1	1	1	1	1	1
Mine Technician	1	1	1	1	1	1	1	1	1	1	1
Mill Feed Control Technician	1	1	1	1	1	1	1	1	1	1	1
Total	**90**	**101**	**133**	**134**	**138**	**140**	**140**	**138**	**144**	**98**	**75**

17.0 RECOVERY METHODS

The following general process description outlines the selected flowsheet.

Run-of-mine mill feed is dumped directly to a primary crusher equipped with a rock breaker. Crushed material discharges to a surge pocket and is delivered via an apron feeder and conveyors to a stockpile.

Feed to the semi-autogenous grinding mill (SAG mill) is withdrawn from under the stockpile by belt feeders and conveyed to the mill at a rate controlled by a weightometer located on the mill feed belt.

The grinding circuit consists of a SAG mill followed by a ball mill operating in closed circuit with hydrocyclone classifiers to produce a ground product at approximately 80% passing 30 microns for flotation.

Cyclone overflow slurry from the grinding circuit is fed via a conditioning tank to rougher/scavenger flotation cells comprised of sequential copper, lead, and zinc circuits. Rougher concentrates are reground and cleaned in three stages to produce saleable copper, lead and zinc concentrates. Cleaner scavenger tailings from all circuits are directed to tails along with the final rougher scavenger tailings.

The concentrates are thickened in a conventional thickener and filtered for shipment.

Expected personnel requirements are shown in Table 17.1.

TABLE 17.1
PROCESSING STAFF AND LABOUR

Maintenance Superintendent	1
Maintenance Foreman	1
Planner	1
Mechanics/Machinists/Welders	12
Electricians	4
Instrument Technician	2
Maintenance Helpers/Labour	9
Mill Superintendent	1
Metallurgist	1
Foreman	1
Shifter	4
Chemist	1
Technicians	2
Mill Clerk	1
Crushing Operator	4
Grinding Operator	4
Flotation Operator	8
Tailings/Load-out Operator	4
Training	8
Labour	4
Total	**73**

18.0 PROJECT INFRASTRUCTURE

18.1 OVERVIEW

The Murray Brook Project will have access to the substantial infrastructure, services and skilled labour in the area. There will be minimal infrastructure cost requirements due to its location near Hwy 180 and approximately 60 km from the historic mining town of Bathurst. The regional labour force includes experienced equipment operators, mine workers and material and equipment suppliers.

The mine plan for the Murray Brook Project has an approximate ten year mill production life with a total mill feed to a single mill on the Project of approximately 2.0 Mt per year.

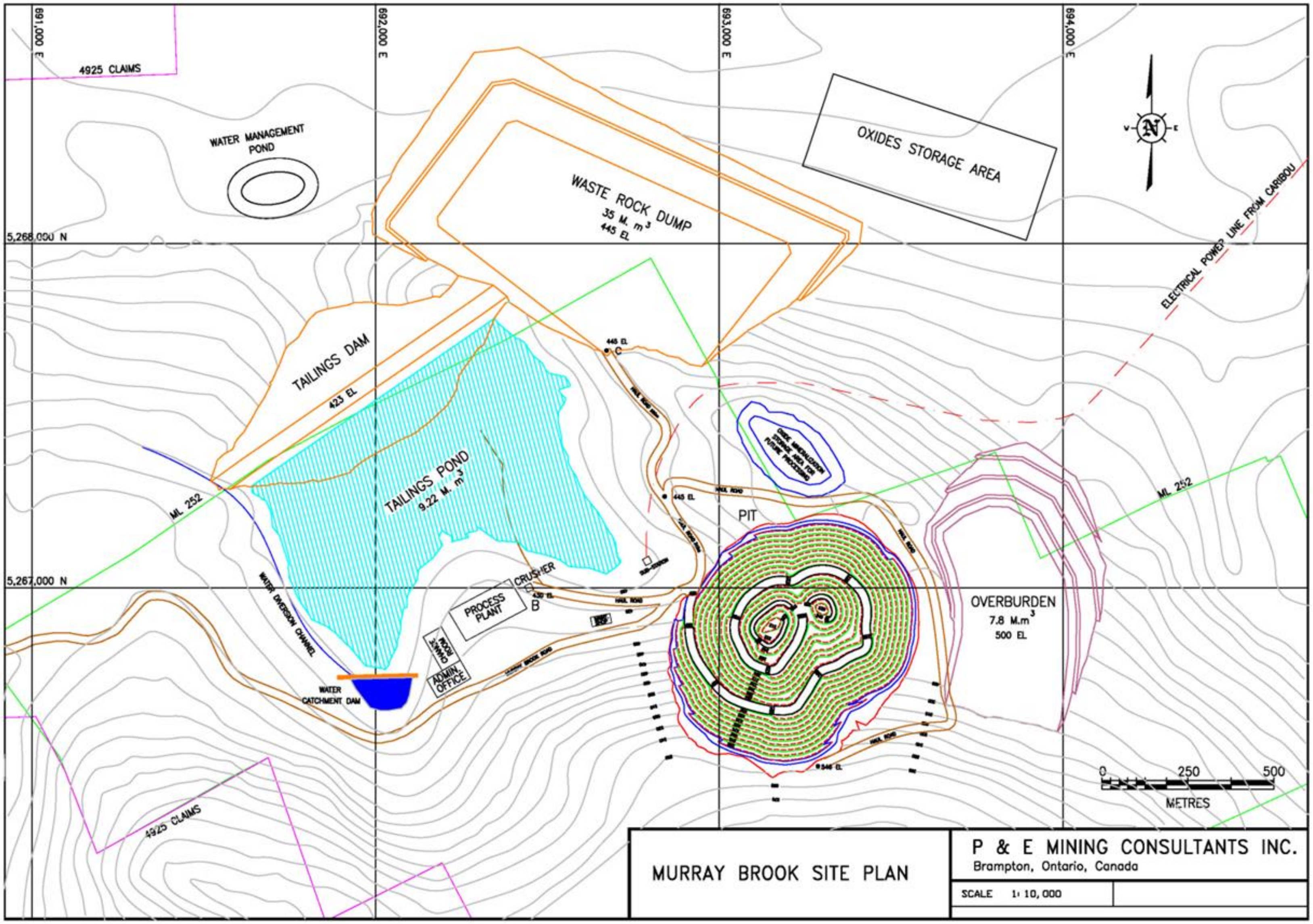
The infrastructure related to the mining and mineral processing facilities have been described in earlier sections.

The services and ancillary facilities required for the Project include the following:

- Project site access road upgrade in order to accommodate highway transport trucks in the construction phase as well as concentrate shipments during production;
- New site service roads and haul roads;
- Power supply connection to the NB Power grid (currently envisaged as coming from a substation at the Caribou Mine site), transmission to the Murray Brook site and power distribution;
- Backup power generating facilities;
- Buildings, including an administration/engineering building, a change house/dry, a warehouse facility and an heavy equipment maintenance and repair shop;
- Site Water management:
- Potable water supply;
- Process water supply
- Fire/fresh water storage and distribution;
- Recycled water collection/storage/distribution;
- Drainage and runoff settling ponds;
- Fuel storage and dispensing;
- Explosives storage facilities;
- Sewage collection and treatment;
- Plant site roads, yard areas and parking;
- Security, safety, and first aid facilities;
- Surface mobile equipment, including a road grader, a service truck, an ambulance; a fire/ rescue truck and pickup trucks.

The site layout showing the proposed configuration of services and ancillary facilities to the mining and milling operations facilities is illustrated in Figure 18.1. This site plan is conceptual and changes will be required as additional data becomes available. This may include information on land ownership and construction permitting, sterilization drilling and geotechnical foundation test work.

Figure 18.1 Murray Brook Site Layout



18.2 SITE ACCESS ROAD

The Murray Brook property is situated in northeastern New Brunswick, approximately 60 kilometres by road from Bathurst, a city of approximately 12,000 people. Access to the property is via Provincial Highway 180 to within 5 kilometres of the mine site, then by a gravel road to the site.

The five kilometre access road will require an upgrade for construction equipment access. No upgrades are assumed or required for any of the provincial roads.

18.3 SITE CLEARING AND GRUBBING

Land clearing will be required in the areas of the open pit, plant sites and the rock and tailings storage areas. Overburden will be collected and stored for later re-vegetation after mine closure. Land clearing will make use of all commercially marketable timber if feasible. No significant value from any marketable resource recovered through land clearing such as harvestable timber is expected in this project. No assumptions about land cost and/or trade-offs for land use have been made in this PEA. The total area of affected land, disturbed and/or cleared, has been estimated at approximately 217 hectares.

18.4 MINE HAULAGE AND SERVICE ROADS

Mine haulage roads will need to be able to accommodate 90 t capacity haulage trucks carrying blasted rock from the pit to the primary crusher, waste rock and overburden storage facilities and to the tailings storage area to build dams. Service roads will be required to connect the mine with the process plant, the office/maintenance/warehouse complex, maintenance and repair facility and site access road.

18.5 POWER SUPPLY

The primary consumption of electric power at the Murray Brook project will occur in the processing plant. The mine will only use marginal amounts of power for the mine maintenance facility and some dewatering.

There is no agreement as yet with the current owners of the Caribou Mine or with NB Power as to the supply of power to the Murray Brook Project. However, for the purpose of this evaluation, it is assumed that power can be provided from the substation at the Caribou mine.

It is understood that a 138kV power line on the NB power grid supplies the nearby Caribou mine site with electricity. A 10 MVA transformer coverts this to 4160 V. Based on preliminary information, electric power may be able to be brought to the Murray Brook site by a powerline over a distance of approximately 12 km. Step down transformers will reduce the voltage to site use levels and on site power will be distributed to locations such as the plant, offices, the maintenance shop and mine dry/change house.

18.6 TAILINGS MANAGEMENT

The Murray Brook project will require the managed disposal of tailings produced from the milling process. It is expected that this material will be potentially acid producing (“PAG”). The conceptual plan for the design of the Tailings Management Facility ("TMF") is to take advantage of the terrain in the vicinity of the mine and the proposed mill site. Separate engineering and environmental studies will be necessary to confirm the adequacy of the site for this purpose. The TMF design would incorporate features to manage the chemical and physical stability of the deposited tailings in accordance with existing and new practices. All of the tailings produced by the milling operation over the life of the project would be placed in this facility.

The capacity of the TMF has been designed to accept a total of 9.2 million m^3 (19 million tonnes) of tailings which represents the currently envisaged LOM tailings production. The containment area will be lined and will be constructed with appropriate spillways and water diversion ditches.

18.7 WASTE ROCK MANAGEMENT

The Murray Brook project will require the managed disposal of mine rock. The total waste rock to be mined from the open pit is estimated at approximately 72 Mt. Waste rock expected to be non-acid producing ("NAG") will be stored in a waste rock storage facility ("WSF") near the mine. The remainder will be stored separately in lined containment areas and deposited in the completed open pit at mine closure. The WSF will be designed, built and closed out so as to minimize long-term impact on the environment.

In addition, approximately 8.8 Mt of overburden will be stripped from the open pit. Some overburden will be used for road and dam construction and the balance will be placed in stockpiles located to the east of the open pit for reclamation of dumps at the end of mine life.

Some NAG waste rock will be hauled to the TMF for dam raising throughout the mine life.

Approximately 1.5 Mt of mineralized oxide material will be mined and placed into stockpile.

Depending on further metallurgical test work, this oxide material may be processed at the end of the mine life or left in the stockpile.

Other waste materials would be recycled (e.g. spent lubricants) or disposed of in accordance with provincial and federal regulations.

18.8 MINE MAINTENANCE AND REPAIR SHOP

The maintenance shop and administration complex will be located adjacent to the processing plant. The maintenance shop will be used to service the open pit and other mobile equipment. The truck shop itself will be comprised of three regular service bays, one welding bays and one preventative maintenance bay. The truck shop and other bays will be serviced by a 50 t bridge crane. The building would be prefabricated from steel structural framing and metal cladding, with concrete floors.

A warehouse will be contained adjacent to the truck maintenance shop and connected via a passageway. It will be used for storage of parts and materials needed for mine and plant operations.

18.9 OPERATIONS AND ADMINISTRATION BUILDING

The Operations and Maintenance building will be a pre-engineered, steel-framed structure with a spread footing foundation and metal deck roof cladding. The building will provide offices for administrative and technical staff, including management, training, accounting, safety, and security. It will also include staff support facilities such as a conference room, print room, and lunch room. It will be connected to a change room/dry complex.

18.10 ON-SITE ACCOMMODATIONS

It is anticipated that Murray Brook Project workers will live in the local area. A temporary camp will be constructed to accommodate contractors and other workers during the construction phase of the Project but no permanent camp is planned for the workers.

18.11 WATER MANAGEMENT

Primary water sources for the processing plant and other uses would be pit dewatering and collection of surface runoff. Reclaim water from the TMF would be recycled to reduce the need for fresh water additions to a minimum.

18.12 WASTE MANAGEMENT

It is expected that the mine will have a waste management program in place to ensure that waste materials are recycled or otherwise disposed in compliance with federal, provincial and local legislation.

Storage facilities for materials such as lubricants, explosives and process chemicals have not been detailed at this preliminary study level. These facilities would be designed to meet relevant codes and regulations in order to protect employees, the public and the environment.

The labour force for the construction and operation of this project are anticipated to be drawn from nearby communities.

18.13 EXPLOSIVES MAGAZINE

The modular powder magazine and explosives magazine will be provided and surrounded by a perimeter security fence with lights.

18.14 FUEL STORAGE

Storage tanks will be provided for both gasoline and diesel fuel. These tanks will be fully enclosed by containment berms to contain leaks and will supply both bulk and independent vehicle dispensing equipment.

18.15 SANITARY SEWAGE SYSTEMS

Sanitary sewage will be collected and treated in two packaged sewage systems. One system will be located adjacent to the Processing Plant/ change house area and the second will service the maintenance facility.

18.16 G&A STAFF AND LABOUR

General and Administration costs for related staff have been based on the number of personnel in the staff classifications listed in Table 18.1.

TABLE 18.1
GENERAL AND ADMINISTRATION PERSONNEL

Project and Mine Staff	Quantity
General Manager	1
Operations Manager	1
Administration Assistant	1
EHS Officer	2
Nurse	2
Warehouse Supervisor	2
Security	4
Total Project	**13**

19.0 MARKET STUDIES AND CONTRACTS

19.1 SUMMARY

There are no existing relevant marketing contracts pertaining to the sale of concentrates produced at the Murray Brook Property.

With respect to commodity prices in general, the general opinion in the industry is that the “super cycle” in commodity prices has come to an end. The market has seen price corrections from the highs in the cycle and the general expectation is that, in view of the higher capital and operating costs prevalent in the industry today, prices in constant dollar terms will reflect today's levels.

19.2 MURRAY BROOK CONCENTRATE MARKETABILITY

Based on available information, it is estimated that the Murray Brook zinc concentrates should provide basis type feed for most zinc refiners. The most likely potential buyer is the CEZ zinc refinery in Québec.

The Murray Brook mineralization is somewhat complex with low copper grades and levels of impurities that make this material likely unattractive to the European smelters. In Canada, it is noted that the Horne smelter in Quebec can be considered to receive feed from Murray Brook, primarily because it previously accepted feed from the recently closed Brunswick Mine located nearby.

The Murray Brook lead concentrates have relatively low lead content and although the contained silver level is reasonable, it is not likely to be sufficiently high to be a first choice for the European smelters. However, the nearby location of the Belledune lead smelter makes it a likely candidate to receive feed from Murray Brook.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section describes the scope of the Project, the regulatory regime, project status, projected environmental and site reclamation requirements and the projected environmental and reclamation costs.

20.1 PROJECT SCOPE

The proposed Project involves open pit mining, on-site processing, mine and mill waste materials storage, concentrate loading and shipping, effluent treatment, project support activities and an integrated site reclamation program.

The proposed open pit would use conventional established open pit drilling, blasting, loading and haulage technologies and equipment. The overburden stripped from the pit area would be separately stockpiled for later use in site reclamation. The mine waste rock would be stockpiled in a designated rock storage area. The development of the proposed open pit would also require the relocation of historic leached gossan that had been encapsulated in two areas located on the mine property.

The mill would process an average of about 6,000 tonnes of mill feed per day. The principal mill processes would include crushing, grinding, flotation, thickening and concentrate storage and concentrate truck loading. The mill tailings would be pumped to the tailings storage management facility where it is assumed that tailings would be kept submerged under a water cover. The tailings pond water would be recycled back to the plant when possible and excess tailings pond water would be treated to prior to release. The Pb-Cu-Zn concentrates that are also likely to contain some Au and Ag values would be stored in the concentrate storage section and be reclaimed using a wheel loader. The mill concentrates would be trucked from site using covered trailers and transported to concentrate treatment facilities or a port for ocean transport.

The PEA assumes that already-reclaimed historic gossan will be relocated from within the proposed pit limits to a new engineered storage cell(s). There is a possibility that residual gold could be recovered from this material – see recommendation in Section 26.

20.2 REGULATORY REGIME

The Project would be subject to federal, provincial and local regulatory requirements. The principal provincial legislation related to the environmental impact assessment (EIA) and approval of mining projects are the Mining Act and General Regulation 86-98, and the Clean Environment Act and EIA Regulation 87-83, and Water Quality Regulation 82-126. Relevant key federal legislation includes the Canadian Environmental Assessment Act, Fisheries Act and Metal Mining Effluent Regulations. The mine approval process in New Brunswick is shown in Figure 20.1. The key steps in the process include:

- Expression of Interest: The mine approval process commences when the proponent submits its Expression of Interest to develop its mine property. The Standing Committee on Mining and the Environment (SCME) provides direction to the proponent on issues, deficiencies and concerns to assist it in developing the required project documentation including a Feasibility Study and Mining and Reclamation Plan.

- Feasibility Study and Mining and Reclamation Plan: This information along with an environmental impact assessment (EIA) Registration Form for screening review is submitted to the SCME. The EIA form is reviewed by a Technical Review Committee comprised of provincial and federal government agencies.

The Project would be subject to a federal-provincial coordinated EIA requiring public consultation and respecting federal and provincial environmental assessment requirements.

20.3 FINANCIAL SECURITY

20.3.1 Rehabilitation and Site Reclamation

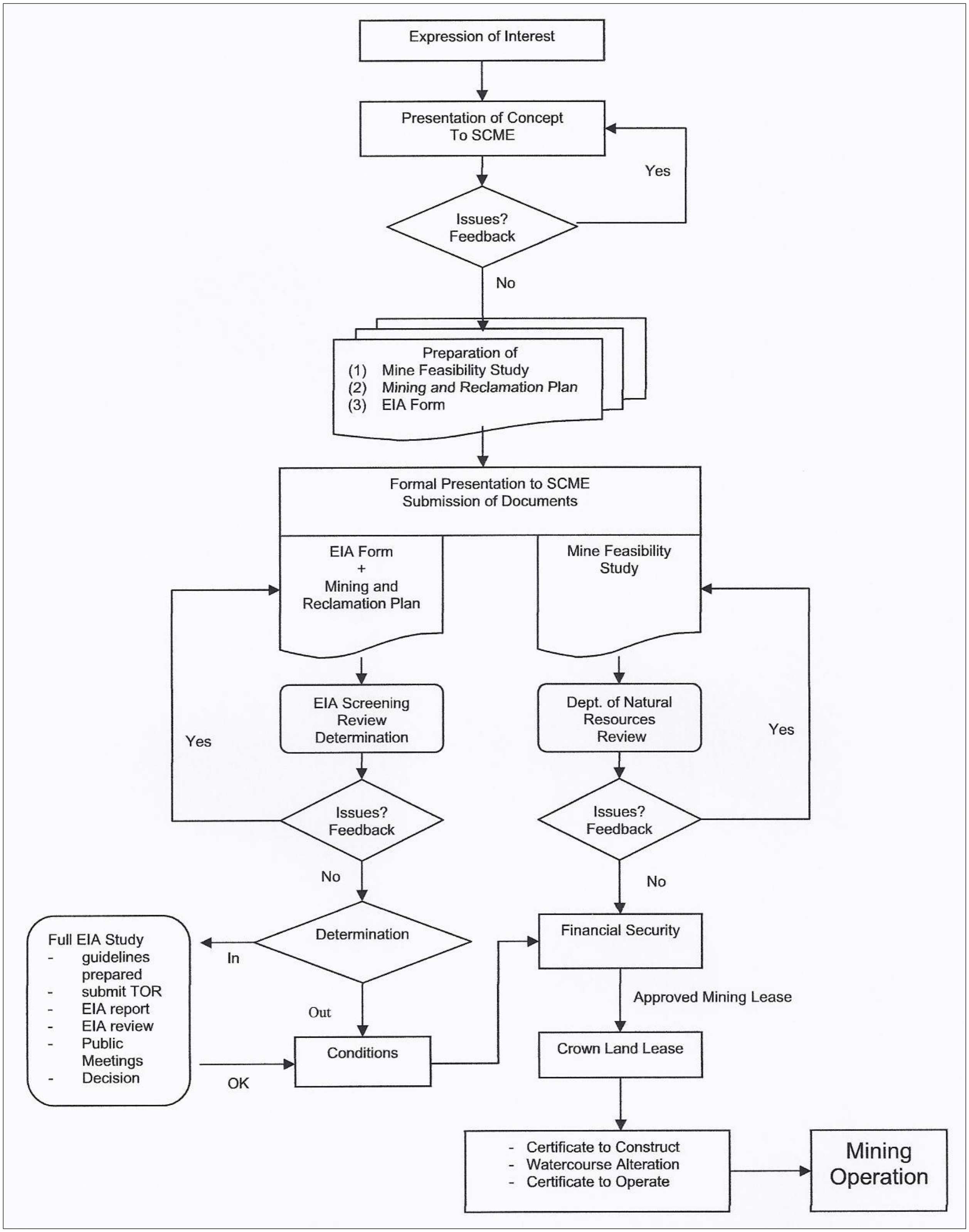
The New Brunswick Minister of the Department of Energy and Mines (DEM) requires financial security for site reclamation. The acceptable forms of security are: 1) money deposit; 2) a negotiable bond assigned to the Province; 3) an irrevocable letter of credit; 4) an insurance company bond; or 5) other form of security acceptable to the Minister of Energy and Mines (ref: General Regulation 86-98, s.47). A reclamation security holdback is typically required during the post-reclamation monitoring stage when the success of the reclamation program is assessed. A walk-away condition is possible following the completion of the rehabilitation program when a site does not pose a public safety hazard and long term water treatment and site maintenance are not required.

Rehabilitation security is required for an Approval to Operate under the Clean Environment Act, Water Quality Regulation 82-126 for the purpose of ensuring the operation, modification, repair or rehabilitation of any source, wastewater works or waterworks or areas affected thereby at any time, whether before or after abandonment of the source, wastewater works or waterworks.

20.3.2 Approvals, Permits and Leases

The Minister of the DEM would issue final approval for an application for a mining lease after receiving approval for the Mine and Reclamation Plan from the Minister of Agriculture, Aquaculture and Fisheries and the Minister of the Environment and Local Government. The proponent would also need to obtain a surface industrial lease from the Department of Natural Resources (DNR), Crown Lands Branch, for project components situated on Crown land. The Approval to Operate for the Project would set out water and air quality limits as well as monitoring and reporting requirements.

Figure 20.1 Mine Approval Process



after NBDEM, 2013a

20.4 STATUS

MBJV's recent focus has been to assess the potential economic viability of the Project by way of the present PEA. It has not yet initiated the mine approvals process for the Project or commenced its formalized public consultation program.

Approximately 1.2 Mt of gossan were mined from 1989 to 1992 by others from an open pit located within the proposed ultimate pit limits.

Jacques Whitford (1999) reports that the historic on-site processing operations involved the cement and hydrated lime agglomeration of the gossan which was then placed in concrete vats in an enclosed facility. Gold, silver and mercury were recovered using a cyanide dissolution with Merryl-Crowe zinc precipitation process.

Pilot leach testing for copper recovery was carried out at the site until 1997. The pilot bioleaching plant was dismantled in 1998 and the spent heap leach material was returned to the pit and gossan-capped prior to its flooding. The flooded pit was lime treated commencing in 1998 to help reduce dissolved copper concentrations in the pit water.

The historic treated-gossan material was reclaimed and is located in two reclaimed stockpiles within and outside of the footprint of the proposed new open pit. Harris and Pheeney (2001) describe the work that was done to characterize the historic materials and reclaim the historic site. In brief, testing was conducted to characterize the pH and residual copper, cyanide and mercury concentrations and permeability of the treated gossan; and provide a technical basis for the development of the site reclamation plan which was approved and implemented in 2000.

The reclamation plan for the historic operation addressed concerns over the migration of residual cyanide and soluble mercury from the treated gossan to ground and surface waters and the receiving environment. The reclamation plan involved the dewatering of the historic pit and the relocation of ~50 kt of sulphide material mined in 1992 for a heap leach demonstration project and 600 kt of treated gossan to the pit. Harris and Pheeney (2001) report that the 600 kt of gossan placed in the pit was placed in compacted lifts and that both the remaining treated gossan pile and the gossan relocated to the pit were capped with compacted till. In the reclamation plan, the pit groundwater would preferentially flow through bedrock instead of the low permeability gossan placed in the pit, and the gossan stockpile seepage would be controlled by the till cover and a bottom liner.

Harris and Pheeney (2001) indicate that the reclamation measures reduced contaminated groundwater inputs to Gossan Creek. MBJV's environmental consultants have since conducted groundwater and downstream water quality sampling, completed environmental effects monitoring in downstream receiving waters, and met with provincial regulators to gain an understanding the Province's objectives for the mitigation of historic impacts to water quality in Gossan Creek where water quality is understood to be insufficient to support a fish population.

Jacques Whitford (2007) carried out a technical study and assessed the concentrations of Cu and Zn in the reclaimed open pit and tailings pile and natural surficial soils and assessed the Cu and Zn concentrations of groundwater samples obtained from monitoring wells located between the materials and Gossan Creek, and Cu and Zn concentrations at selected monitoring stations along Gossan Creek and downstream of its confluence with Copper Creek. The monitoring data indicated that there had been a downward trend in Cu and Zn concentrations in a path from the reclaimed pit to Gossan Creek since pit backfilling. Jacques Whitford (2007) concluded that further monitoring and a hydrogeological technical review would be needed to understand the preferential groundwater flow path, the boundaries of the historic groundwater plume and if the peak of the plume had passed, and indicated that in the absence of any change in the trend discharge to streams below the headwaters of Gossan Creek was not viewed as a concern.

The historic mine site operator had put \$0.5M in rehabilitation security in place for an Approval to Operate (ATO) for the reclaimed Murray Brook historic mine site. That ATO would have expired in October 2014. During the preparation of the present PEA, MBJV was in negotiations with provincial regulators to obtain an Approval to Operate. The present PEA has assumed that MBJV would increase the rehabilitation security for the reclaimed site from the historic \$0.5M level to \$2M. Votorantim has since reported that it has received Approval to Operate (ATO) No. I-8297 valid to March 31, 2018 for the operation of the Reclaimed Murray Brook Mine Site identified as a "source" including the historic tailings (e.g. processed gossan) pile, the reclaimed open pit, and groundwater monitoring wells. It also reports that it has replaced the historic \$0.5M security with a \$0.5M cash security with the understanding that its cash security will be returned when it provides a \$2M rehabilitation bond in the form of an irrevocable letter of credit to the Government of New Brunswick. ATO No. I-8297 supersedes the historic ATO (I-6920). ATO I-8297 does not include approval to remove bulk material from the historic in-filled pit or disturb the historic tailings (e.g. processed gossan) pile. If by December 31, 2016 MBJV has not registered a project under the EIA Regulation to re-open the Murray Brook Mine, MBJV is to carry an additional investigation of the groundwater plume to Gossan Creek and Copper Creek; develop a mitigation strategy to reduce metal concentrations and estimate the costs to implement the plan for review by the Industrial Processes Section; and implement the accepted mitigation strategy.

The present PEA assumes that MBJV would register the Project under the Environmental Impact Assessment Regulation, undertake the mine approvals and environmental impact assessment processes including public input and consultation, and obtain the necessary approvals and permitting for the Project, and operate and reclaim the site based on an integrated reclamation plan for the Project that includes the historic Murray Brook mine site.

20.5 PROJECTED SITE RECLAMATION REQUIREMENTS

The Project, as currently envisaged would make use of best management practices and engineered controls to eliminate or mitigate potential environmental impacts and be designed taking reclamation requirements into consideration. P&E has assumed, as examples, that the mill tailings would be disposed underwater to inhibit sulphide oxidation and that an active treatment system would be used to treat final effluent.

The following conceptual reclamation plan for the Project, including the reclaimed historic gossan has been developed for the purposes of the present PEA based on available information and assumptions. It is expected that MBJV would further develop and refine the conceptual plan to create its reclamation plan for the Project, which would be required as part of the mine permitting process. P&E has assumed that the reclamation plans for the proposed Project and the historic treated gossan would be combined into an integrated reclamation plan where:

- The historic treated gossan would be relocated to develop the proposed pit. It is assumed that the historic treated gossan would be placed into an engineered containment area.
- Till and non-treated/non-processed gossan excavated from the proposed new open pit would be stockpiled separately. It is assumed that a portion of the waste rock excavated from the proposed pit would be acid generating / metal leaching and that these materials would be segregated and stockpiled in a containment area over the mine life and then relocated to the pit and kept underwater. The slopes of non-acid generating waste rock piles would be re-sloped and vegetated. Sulphide tailings would be disposed underwater in the tailings management storage area and kept saturated over the long term by maintaining a water cover over the tailings or elevated water table conditions within the tailings.
- The processing plant, site infrastructure and equipment would be salvaged or otherwise demolished. Unused or waste chemicals and hazardous materials would be disposed in accordance with regulatory requirements. Disturbed land areas would be reclaimed and left in a safe and stable condition.

The performance of the reclamation plan would be assessed over a three to five year post-reclamation monitoring program.

20.6 PROJECTED ENVIRONMENTAL PROGRAM AND RECLAMATION COSTS

The projected environmental capital costs are shown in Table 20.1.

TABLE 20.1
INITIAL ENVIRONMENTAL COSTS

Item	Projected Cost
Mine material acid rock drainage and metal leaching characterization and additional groundwater plume investigations.	$0.7M
Develop the integrated reclamation plan for the Project.	$0.3M
Construct a treated gossan storage cell. [1.2 Mt storage capacity x $2.50/t capacity]	$3.0M
Relocate reclaimed historic gossan to the new storage cell. [(0.6 Mt x $2.90/t in pit) + $60k lime]	$1.8M
Construct the tailings management area starter dam and spillway. [Cost allowance]	$12M
Construct effluent treatment plant. [Cost allowance assuming lime treatment with settling and polishing ponds]	$5M
Construct waste rock storage pad.	$0.5M
Total	**$23.3M**

20.7 CLOSURE SECURITY

The PEA includes closure security cost allowances amounting to $6.5M. This amount includes a reclamation security cost allowance of $3M, a $1.5M top-up of the rehabilitation security for the existing reclaimed site and a $2M rehabilitation security allowance for the envisaged effluent treatment plant. Projected environmental costs that would be incurred during operations are shown in Table 20.2.

TABLE 20.2
ENVIRONMENTAL COSTS DURING MINE PRODUCTION PHASE

Item	Projected Costs
Tailings management storage facility expansion / dam raising	$18 M over LOM
Effluent recycle and treatment cost allowance. [18.9 Mt milled x $0.20/t milled]	$4M over LOM
Reclaim clean waste rock, untreated gossan and disturbed areas. [Cost allowance]	$2M over LOM
Ongoing environmental monitoring.	See G&A costs.
Total	**$24M over LOM**

<u>Environmental Costs during Mine Reclamation</u>

The projected mine reclamation costs are shown in Table 20.3.

TABLE 20.3
PROJECTED MINE RECLAMATION COSTS

Item	Projected Costs
Mill and infrastructure salvage and demolition.	$3M
Relocate acid waste rock to pit. [71.5 Mt waste rock x (assume up to 25%) x $1.20/t]	up to $22M*
Reclaim tailing management storage area.[Cost allowance]	$2M
Post-operational monitoring and care and maintenance.	$2M
Subtotal	up to $29M
Less: projected reclamation security including existing $0.5M security.	($7M)
Reclamation and Rehabilitation Cost (excluding reclamation security)	**Up to $22**

**Preliminary cost allowance. MBJV has not yet commenced acid rock and metal leaching test work. For the purposes of the PEA, it is assumed that up to 25% of the pit waste rock would be acid generating or potentially acid generating and that this material would be relocated to the pit at the end of the mine life and kept submerged over the long term.*

21.0 CAPITAL AND OPERATING COST

21.1 CAPITAL COSTS

21.1.1 Summary

The capital cost of the project includes engineering, procurement, construction and start-up of the Murray Brook Project, will comprise an open-pit mine, a concentrator capable of processing 6,000 tpd, and associated ancillary facilities.

The capital cost estimate was developed to a level commensurate with that of a Preliminary Economic Assessment. After inclusion of the contingency, the capital cost estimate is considered to have an accuracy of ±35%.

Table 21.1
Capital Cost Summary ('000's of $)

Year	Mining	Process Plant	Infrastructure	Environmental & Reclaim	Indirects	Contingency	Total
Preproduction							
-2	$33,706	$52,092	$0		$20,800	$16,264	$122,862
-1	$8,707	$52,092	$35,000		$23,200	$18,994	$137,993
Subtotal	$42,413	$104,184	$35,000		$44,000	$35,257	$260,854
Production							
1	$8,749	$300		$9,050		$497	$18,596
2	$139	$300		$3,050		$67	$3,556
3	$433	$300		$3,050		$82	$3,865
4	$139	$300		$3,050		$67	$3,556
5	$244	$300		$3,050		$72	$3,666
6	$433	$300		$3,050		$82	$3,865
7		$300		$3,050		$60	$3,410
8		$0		$3,050		$0	$3,050
9		$0		$3,050		$0	$3,050
10		$0		$3,050		$0	$3,050
11		$0				$0	$0
12		$0		$24,300		$0	$24,300
Subtotal	$10,137	$2,100	$0	$60,800	$0	$927	$73,964
Contingency	5%	20%	10%	0%	20%	----	----
Total	**$52,550**	**$106,284**	**$35,000**	**$60,800**	**$44,000**	**$36,184**	**$334,818**

The total estimated cost to design, procure, construct and start-up the facilities described in this report is $260.8 million. Sustaining capital costs will be in the order of $3.5 million per year, after achieving steady-state production. Sustaining capital represents capital expenses for additional costs and equipment purchases that will be necessary during the operating life of the project, and are not included in the normal operating costs. The total sustaining capital cost is approximately $74.0 million (Table 21.1).

An exchange rate of US$1.00= C$1.00 has been used for the capital cost estimate.

The preproduction capital cost estimate includes an allowance for contingency of approximately 15.6% or $36.2 million.

No provision has been included in the capital cost to offset future escalation.

Items not included in the capital estimate are:

- Sunk costs and costs prior to the start of basic engineering phase
- Cost escalation;
- Working capital;
- Interest and financing costs;
- Taxes.

21.1.2 Presentation of Costs

The capital cost estimate has been built up by cost areas. Costs are based on the assumption that equipment and materials will be purchased on a competitive basis and installation contracts will be awarded in defined packages for lump sum or unit rate contracts.

21.1.3 Indirect Costs

These costs have been calculated using percentages based on historical data from similar projects. Indirects include overhead staff and support facilities; bonding; insurance; construction permits; contract administration; schedule management; management of subcontractors; onsite busing; surveying; mobilization and demobilization; construction equipment and small tools; supervision; safety; temporary power, toilets and communication; warehousing; cleanup and waste removal; construction vehicles, fuel and maintenance.

21.1.4 Spare Parts and Initial Fills

An allowance has been made for spare parts required for start-up and commissioning of the Project. A percentage of the equipment value has been assigned.

21.1.5 EPCM Services

EPCM services for basic and detailed engineering design, procurement, and construction management of the processing and ancillary facilities have been included. These percentages are based on past experience with similar work.

21.1.6 Freight

Transportation costs have been included in the direct costs for delivery of equipment and materials to the jobsite. In general, it has been assumed that most equipment and bulk materials will be purchased in North America and can be trucked to the site.

21.1.7 Contingency

Contingency has been included in the capital cost in recognition of the degree of detail upon which the estimate is based.

21.1.8 Mining Capital Cost

The mine capital cost has been subdivided into four areas; (i) mining equipment, (ii) mine infrastructure, (iii) freight, and (iv) pre-stripping. Table 21.2 summarizes the initial mine capital cost of $52.5 million which includes a 5% contingency allowance.

TABLE 21.2
MINE CAPITAL COST SUMMARY (LIFE OF MINE)

	Initial ($M)	Sustaining ($M)	Total ($M)
Mining Equipment	30.8	8.9	39.6
Mine Infrastructure	1.4	0.8	2.2
Freight	1.5	0.4	2.0
Sub-total	33.7	10.1	43.8
Pre-strip	8.7	0.0	8.7
Total	**42.4**	**10.1**	**52.5**

On-going mine equipment additions and replacements will add another $10.1 million over the life of the project.

Mine infrastructure consists of mine road construction, dump area preparation, shop and office supplies, and explosive storage facilities.

The pre-stripping will be done by the owner mining fleet during year -1. Hence a large portion of the mining fleet will be purchased in Year -2. While pre-stripping will be done by the Murray Brook operations fleet, it has been assumed that year -1 mining operating costs will be capitalized.

Freight is included and has been estimated at 5% of the mining equipment capital cost.

21.2 PROCESS PLANT CAPITAL COST

21.2.1 Basis

Capital costs are based on an average daily throughput of approximately 6,000 t/d and the general flowsheet described in Section 17. All costs are quoted in fourth quarter 2012 Canadian dollars.

Equipment costs are developed from EHA Engineering Ltd. (“EHA”) in-house cost data and correlations. Direct costs other than equipment are factored on equipment or direct costs on a process area basis, using factors derived from historical projects.

Building costs are based on direct cost factors derived from area/volume unit cost data for historical projects and include space allowance for facilities such as mill offices, change rooms and laboratory.

Indirect costs are factored on direct costs using information derived from historical projects and selected to relate to the project location. A 20% contingency allowance is added, based on total direct plus indirect costs.

The following costs are included in the estimate:

- All process equipment cost;
- Site development costs, based on general site conditions expected for the project and normal site services;
- All direct costs related to the process;
- Serviced process building;
- Construction indirect and EPCM costs;
- Spare parts allowance;
- Start-up allowance;
- Freight allowance.

The following costs are excluded from the estimate:

- Ancillary buildings and services, including any mine related facilities, camp and general administration;
- All off-site costs including services to the site, except as noted above.\;
- Tailings disposal, tailings line, reclaim and fresh water pumps and pipelines;
- Backfill plant;
- Any secondary effluent treatment facilities;
- Taxes and duties;
- Mobile equipment;
- Owner's costs including first fill and product inventory;
- Cost escalation;
- The accuracy of the estimate is +/- 30 %

21.2.2 Capital Cost Summary

Process plant capital costs are summarized in Table 21.3.

TABLE 21.3
PROCESS PLANT CAPITAL COST SUMMARY ('000'S OF $)

Direct Costs			
Equipment		$43,920	
Other Direct Costs		$60,263	
Total Direct Costs			$104,183
Indirect Costs			
Construction			
Field Supervision	$3,594		
Field Expense	$4,480		

TABLE 21.3 **PROCESS PLANT CAPITAL COST SUMMARY ('000'S OF $)**			
Temporary Facilities	$1,563		
Construction Equipment	$2,605		
Craft Benefits	$6,564		
Total Construction Indirect Costs		$18,805	
Other Indirect Costs			
Engineering		$11,981	
Freight		$2,084	
Spare Parts		$1,563	
Start-up		$521	
Total Indirect Costs			$34,953
Total Direct and Indirect Costs			$139,136
Contingency Allowance @ 20%			$27,827
Total Capital Cost			$166,964

21.3 INFRASTRUCTURE CAPITAL COST

Infrastructure capital costs include site facilities, buildings, furnishings and surface mobile equipment.

The capital cost of site facilities includes: site roads; surface parking areas; the fuel farm; lubrication and oil storage facilities; surface explosive magazines; yard piping; the fire prevention and fighting system; the potable water treatment plant and storage tanks; the tailings water treatment plant and pond; and the water management pond building and site run-off.

Buildings capital costs include: the main gate building; the surface mine shop; the warehouse and warehouse equipment; the office facility and the dry. The buildings furnishings include; the surface mine shop equipment and tools; the office furniture, computers, etc.; environmental equipment; dry equipment; site communications and medical centre equipment.

Surface mobile equipment capital costs include: a road / ramp grader; an integrated tool carrier; a fuel/lube truck; a service truck; a garbage truck; an ambulance; a fire/ rescue truck; and pickup trucks. The surface infrastructure capital cost summary is presented in Table 21.4.

TABLE 21.4 PROJECT INFRASTRUCTURE CAPITAL EXPENDITURES ('000'S OF $)						
Area	**Quantity**	**Units**	**Unit Cost**	**Total Cost**	**Year**	
					-2	**-1**
Initial Environmental Costs (*see Section 20*)	1	Lump Sum	$23,300	$23,300	$0	$23,300
Site Clearing and Grubbing	1	Lump Sum	$1,000	$1,000	$0	$1,000
Site Roads	5	km	$100	$500	$0	$500
Shop/Warehouse	1	Lump Sum	$1,000	$1,000	$0	$1,000
Office	1	Lump Sum	$1,000	$1,000	$500	$500
Dry	1	Lump Sum	$500	$500	$0	$500
Powerline	12	km	$100	$1,200	$0	$1,200
Services Substation	1	Lump Sum	$500	$1,000	$0	$1,000
Water Treatment Plant	1	Lump Sum	$2,000	$2,000	$0	$2,000
Site Mobile Equipment		allowance	$500	$1,000	$0	$1,000
Miscellaneous		allowance		$2,500	$0	$2,500
Subtotal Infrastructure Capital				$35,000	$500	$34,500
EPCM	12%			$4,200	$0	$4,200
Contractors Overhead	8%			$2,800	$0	$2,800
Commissions, Vendor Reps	8%	*of EPCM*		$336	$0	$336
Spare Parts for Construction	1.00%			$350	$0	$350
EIA and Supporting Studies	all		$2,000	$2,000	$500	$1,500
Contingency	10%			$4,469	$100	$4,369
Total Infrastructure Cost				**$49,155**	**$1,100**	**$48,055**

Note: *The environmental and reclamation costs are described in Section 20.*

21.4 OPERATING COSTS

The operating costs estimate includes the cost of mining, processing, waste management, and G&A services. The life-of-mine average operating cost for the Murray Brook Project is summarized in Table 21.5.

Table 21.5 Operating Cost Summary	
Description	**$/t milled**
Mining	$12.23
Processing	$14.25
G&A	$1.32
Total	**$27.80**

21.4.1 Mining

Mine operating costs are derived from in-house equipment databases for all major and supporting equipment operating parameters, and include fuel, consumables, labor ratios, and general parts costs. The mine operating cost is summarized in Table 21.6 and averages at $2.30/tonne mined over the life of the project.

Table 21.6 Mine Operating Cost Summary		
Mine Operating Cost (Life of Mine Average)		
Drilling	$/t total material mined	$0.44
Blasting		$0.32
Loading		$0.20
Hauling		$0.78
Services/Roads/Dumps		$0.38
General, Supervision & Technical		$0.18
Allowance		
Total Operating Cost	**$/t total material mined**	**$2.30**
Total Operating Cost	**$/t milled**	**$12.23**

Annual production tonnes, waste tonnes and loading and hauling hours are calculated based on the capacities of the loading and hauling fleet. These tonnes and hours provide the basis for drilling, blasting, and support fleet inputs. Based on the tonnes scheduled, a requirement for production drilling hours is calculated based on hole size and pattern, bench height, material density and, penetration rate of the drill.

The quantity of explosives is calculated, priced, and contractor labour and fees added. An estimate for initiation systems and blasting accessories is provided on a per hole basis. Drilling and blasting inputs (pattern area, powder factor, etc.) have been included.

Fleet requirements for loading, hauling and support are derived from the loading and hauling operating hours. The support fleet of dozers, front-end loaders, graders, service and welding trucks, etc., is added in.

All equipment cost is based on estimated fuel consumption rate, consumables cost, GET estimate, and general parts and preventative maintenance costs on a per-hour or per-metre interval basis.

Operating labor man-hours are categorized for the different labor categories such as operators, mechanics, electricians, etc. The mining cost also includes costs for all mine salaried staff, consumables, and software and fleet management systems' licensing and maintenance. It is essentially a fixed cost component.

21.4.2 Processing

Operating costs include all processing costs from receipt of feed from the mine through to concentrate production and disposal to tailings. Labour costs are based on estimated current rates and manning levels. Power unit cost is based on a provisional estimate of $0.065/kWh. Reagent prices are based primarily on vendor budget quotations for other projects and include an allowance for freight. Indirect costs such as the following are not included in this section:

- Insurance;
- Taxes;
- Safety and security;
- Research and development;
- General administration and head office expenses;
- Depreciation and amortization.

Table 21.7 summarizes estimated process operating costs.

TABLE 21.7
MINE OPERATING COST SUMMARY

Item	$C/t	$C/a
Operating Labour	1.53	3,048,500
Power	3.27	6,523,000
Reagents	6.08	12,124,800
Operating Supplies	1.29	2,568,000
Maintenance Labour	1.03	2,059,100
Maintenance Supplies	1.05	2,087,000
Total	**14.25**	**28,410,400**

21.4.3 General and Administrative (G&A)

General and Administration ("G&A") costs include costs for staff, general maintenance, office administration, safety equipment and personal protective equipment (PPE), and engineering tools and professional services cost.

The estimated cost for G&A is approximately $1.32 per tonne milled (Table 21.8)

TABLE 21.8
GENERAL AND ADMINISTRATIVE COSTS

Description	Yearly Cost
Staff	$1,154,000
General Maintenance	$60,000
Office Administration	$588,000
Safety Equipment and PPE	$100,000
Environment, Health and Safety/Training	$598,000
Total G & A cost per year	**$2,500,000**
Total G&A per Tonne Rock Processed	**$1.32**

22.0 ECONOMIC ANALYSIS

22.1 SUMMARY

The Murray Brook Project's financial results are summarized in Table 22.1 and indicate an after-tax net present value ("NPV") of $96.4 million at a 5% discount rate, an internal rate of return ("IRR") of 11.4% and a 5.4 year payback. The initial capital expenditure would be $260.8 million with a life-of-mine capital cost of $334.8 million. All currency values are expressed in Canadian dollars unless otherwise noted.

TABLE 22.1
FINANCIAL RESULTS SUMMARY

NPV (0%)	$228.67	millions
NPV (5%)	$96.4	millions
NPV (7%)	$59.7	millions
IRR	11.4%	
Payback	5.4	years
Total LOM Capital	**$334.8**	**millions**

The financial results are based on April 30, 2013 three year trailing average metal prices of $US 3.70/lb copper, $US 1.00/lb lead, $US 0.94/lb zinc, $US 1,540/oz gold, $US 30.09/oz silver. The assumed exchange rate is $US 1.00=$C 1.00.

The projected cash flow calculation spreadsheet is provided in Appendix IX

The Murray Brook production results are summarized in Table 22.2. Due to low gold recovery, no payable gold is assumed to be contained in the concentrates.

TABLE 22.2
METAL PRODUCTION RESULTS

	Annual Average		LOM Total	
Copper	9.2	M lbs	92.0	M lbs
Lead	13.6	M lbs	135.7	M lbs
Zinc	91.4	M lbs	913.6	M lbs
Gold	0	M oz	0	M oz
Silver	1.3	M oz	12.7	M oz

22.2 ASSUMPTIONS

A discounted cash flow analysis of the Murray Brook has been prepared based on technical and cost inputs developed by the P&E engineering team.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted. The financial evaluation uses a discount rate of 5% and was performed at commencement of construction (Year -2 of the Project).

22.2.1 Metal Prices Assumptions

The Murray Brook's key financial input assumptions are summarized in Table 22.3. Given the Project being located in Canada, operating and sustaining costs will be predominantly denominated in Canadian dollars with revenues from metals being US dollar denominated. The economics of the project will therefore be sensitive to US currency fluctuations relative to the Canadian dollar. Capital costs have been quoted in the Study based on an exchange rate of 1 US dollar to 1 Canadian dollar.

TABLE 22.3
METAL PRICE ASSUMPTIONS

Copper	3.70	$US/lb
Lead	1.00	$US/lb
Zinc	0.94	$US/lb
Gold	1,540	$US/oz
Silver	30.09	$US/oz
Exchange Rate	1:1.00	$US:$C

22.2.2 Recoveries

The Murray Brook Project's recovery assumptions are outlined in Table 22.4.

TABLE 22.4
RECOVERY ASSUMPTIONS FOR PAYABLE METALS

	Copper Concentrate	Lead Concentrate	Zinc Concentrate
Copper recovery	51.4%		
Lead recovery		36.6%	
Zinc recovery			88.8%
Gold recovery			
Silver recovery	12.5%	17.5%	25.3%

22.2.3 Capital Costs

Total capital costs are estimated at $260.8 million as outlined in the Capital and Operating Cost Section 21. Most of the initial capital costs are incurred over a two year construction period. Sustaining capital costs for mining equipment and waste management for the Murray Brook Project totals about $80.0 million, resulting in a total life-of-mine capital of $334.8 million.

Ramp-Up Assumptions

In the first year of production (Year+1), the plant is assumed to achieve only 67% of the nameplate throughput capacity, or about 1.33 Mt processed instead of the 2.0 Mt capacity.
Net Smelter Return (NSR)

Table 22.5 gives a summary of the NSR assumptions of the Murray Brook Project.

Table 22.5 Net Smelter Return Parameters			
	Copper Concentrate	**Lead Concentrate**	**Zinc Concentrate**
Deduction (units)	1%	0%	0%
Payable metal	96.5%	95.0%	90.0%
Payable Ag	90%	95%	90%
Refining Copper ($/lb)	$0.10	-	-
Refining Ag ($/oz)	$0.50	$1.00	$1.00
Treatment Cost ($/dmt)	$105	$275	$250
Marketing ($/dmt)	$7.00	$7.00	$7.00
Insurance ($/dmt)	$2.00	$2.00	$2.00
Assaying, Supervision ($/dmt)	$1.00	$1.00	$1.00
Penalties	not included	not included	not included
Transport Cost ($/wmt)	$30	$30	$30

22.3 CASH FLOW SUMMARY

The estimated annual LOM cash flow for the Murray Brook Project is summarized in Table 22.6.

Table 22.6 Cash Flow Summary		
Mine Production		
Overburden	t	8,837,667
Oxides Waste	t	1,531,392
Waste Rock	t	71,516,093
Total Waste	t	81,885,152
Pit Strip Ratio		4.3
Total Feed Sulphides	t	18,947,674
Total Material Mined	t	100,832,826
Processing		
Mill Feed tonnage	tpy	18,947,600
Grade - Au	g/t	0.53
Grade - Ag	g/t	37.7
Grade - Cu	%	0.43
Grade -Pb	%	0.89
Grade -Zn	%	2.46
Revenue		
Copper Concentrate	C$(‘000)	346,073.7
Lead Concentrate	C$(‘000)	200,846.4
Zinc Concentrate	C$(‘000)	699,106.1
Total NSR Revenue	C$(‘000)	1,246,026.2
0.25% Royalty Payable*	C$(‘000)	3,115.1
Operating Cost		
Mining Cost	$/t material	2.30
Processing Cost	$/t milled	14.25
G&A	$M/year	2.50
Unit Operating	$/t milled	$27.80

TABLE 22.6 CASH FLOW SUMMARY		
Unit Mining Cost	$/t milled	12.23
Capital Costs		
Mine Pre-Stripping	C$(‘000)	8,707.0
Mining Capital Cost	C$(‘000)	33,705.9
Process Plant	C$(‘000)	104,184.0
Infrastructure	C$(‘000)	35,000.0
Indirects	C$(‘000)	44,000.0
Contingency	C$(‘000)	35,257.4
Initial Project Capital	C$(‘000)	260,854.3
Sustaining Capital		
Mine	C$(‘000)	10,137.0
Process Plant	C$(‘000)	2,100.0
Environment & Reclamation	C$(‘000)	60,800.0
Contingency (sustaining)	C$(‘000)	926.9
Total Sustaining Capital	C$(‘000)	73,963.9
Total Capital (LOM)	C$(‘000)	334,818.1
Cash Flow		
Revenue from Concentrate	C$(‘000)	1,246,026.2
Operating Cost	C$(‘000)	(518,076.0)
Royalties	C$(‘000)	(3,115.1)
Taxes	C$(‘000)	(161,343.5)
Capital Spending	C$(‘000)	(334,818.1)
Cash Flow	C$(‘000)	228,673.4

Note: *The financial analysis includes the payment of the 0.25% royalty at the start of production. In fact, the 0.25% Royalty payment commences on the first anniversary of production, or effectively one year later*.

22.4 INCOME TAXES AND MINING TAXES

Mining operations in New Brunswick are subject to three tiers of taxes: a federal income tax under the Income Tax Act (Canada); a provincial income tax under the New Brunswick Income Tax Act and a provincial mining tax under the New Brunswick Metallic Minerals Act (MMA). The following is a summary of the significant taxes applicable to the Murray Brook Project.

Federal Income Tax

Federal income tax is applied to the project’s taxable income (generally being net of operating expenses, depreciation on capital asset and the deduction of exploration and pre-production development costs). The current federal income tax rate in Canada is 15%.

Provincial Income Tax

A New Brunswick provincial income tax is based on a similar taxable income as the federal calculation of taxable income. The current provincial income tax rate in New Brunswick is 10%.

NB Provincial Mining Tax

The Province of New Brunswick levies a two-tier mining tax: a 2% royalty based on “net revenue”; and a 16% levy on “net profits”.

The 2% royalty comes into effect two years after a new mine starts production. The royalty is based on 2% of the net revenue generated by the operation, which is generally equal to the revenue generated from the sale of mine output less transportation and processing costs (including refining & smelting costs). A processing allowance of 8% for milling or concentrating assets can be deducted from net revenue. The total deduction cannot exceed 25% of the net revenue before the processing allowance has been deducted.

The net profit tax is calculated as 16% of the gross revenue in excess of $100,000 less allowable costs, eligible exploration expenditures and specified allowances for depreciation, financing, and processing. The 2% royalty paid is also deductible in determining net profits.

New Brunswick mining taxes paid are deductible for federal and provincial income tax purposes.

22.5 SENSITIVITIES

The Murray Brook Project sensitivity analysis was conducted to the following key variables:

- Zinc vs. Copper Price (Table 22.7);
- Capital and Operating costs (Table 22.8);
- Before and After Tax NPV (Table 22.9).

The results of the sensitivity analysis for the key variables on the After-Tax NPV are shown in Table 22.7 and Table 22.8.

TABLE 22.7
SENSITIVITY - ZINC VERSUS COPPER PRICE

NPV5%		Zinc Price								
	$96.4	$0.54	$0.64	$0.74	$0.84	$0.94	$1.04	$1.14	$1.24	$1.34
	$2.90	$ (95.5)	$ (52.0)	$ (11.3)	$ 26.9	$ 64.6	$ 101.0	$ 137.4	$ 172.7	$ 209.2
	$3.10	$ (86.1)	$ (43.1)	$ (3.1)	$ 35.8	$ 72.4	$ 109.9	$ 145.0	$ 180.5	$ 216.8
	$3.30	$ (76.6)	$ (33.9)	$ 5.0	$ 44.6	$ 80.4	$ 117.5	$ 152.6	$ 189.5	$ 224.3
Cu	$3.50	$ (67.1)	$ (25.2)	$ 13.5	$ 52.3	$ 88.5	$ 125.2	$ 160.4	$ 197.0	$ 231.8
Price	$3.70	$ (57.4)	$ (16.7)	$ 22.3	$ 60.1	$ 96.4	$ 132.8	$ 168.3	$ 204.5	$ 239.3
	$3.90	$ (48.6)	$ (7.7)	$ 31.3	$ 67.8	$ 105.3	$ 140.4	$ 177.3	$ 212.0	$ 246.8
	$4.10	$ (39.5)	$ 0.5	$ 40.1	$ 75.8	$ 112.9	$ 148.2	$ 184.8	$ 219.5	$ 254.3
	$4.30	$ (30.2)	$ 9.0	$ 47.8	$ 84.0	$ 120.6	$ 156.0	$ 192.3	$ 227.0	$ 261.8

TABLE 22.8
SENSITIVITY – CAPITAL AND OPERATING COST FACTORS

NPV5%		Capital Cost Factor				
	$96.4	80%	90%	100%	110%	120%
	80%	$ 188.9	$ 168.4	$ 146.1	$ 125.0	$ 102.5
Op Cost	90%	$ 165.2	$ 143.2	$ 121.8	$ 99.4	$ 77.1
Factor	100%	$ 140.3	$ 118.8	$ 96.4	$ 74.0	$ 52.5
	110%	$ 115.9	$ 93.6	$ 71.1	$ 49.5	$ 25.4
	120%	$ 90.9	$ 68.3	$ 46.6	$ 22.2	$ (1.6)

TABLE 22.9
SENSITIVITY – BEFORE AND AFTER TAX NPV5%

NPV5%	$Millions
Before Tax	196.7
After Tax	96.4

A sensitivity analysis of the project to zinc price was also performed, at zinc prices of US$1.50 and US$2.00. Table 22.10 demonstrates that the project performance is very sensitive to zinc prices.

TABLE 22.10
SENSITIVITY – ZINC PRICE

Zinc Price	NPV @ 5% Discount rate	IRR
US$0.94	$96.4	11.4%
US$1.50	$294.9	21.7%
US$2.00	$468.7	29.0%

23.0 ADJACENT PROPERTIES

The Murray Brook Deposit is hosted by the Ordovician California Lake Group, an important host to mineralization in the Bathurst mining camp. Murray Brook is in the northwest part of the Bathurst camp and is located approximately 42 km west-northwest of the recently decommissioned Bathurst #12 Mine, which was formerly the largest producing mine in the camp.

In the vicinity of Murray Brook, the California Lake Group has a regional strike direction of approximately 70°. The Caribou Mine is located approximately 11 km east-northeast of Murray Brook and the past producing Restigouche Mine is located 10 km to the west-southwest. The geology and mineralization at both of these mines is broadly similar to the Murray Brook deposit.

Trevali Mining Corporation ("Trevali") owns the Caribou mine and 3,000 tpd flotation mill that will produce zinc, lead and copper concentrates. Trevali received approval to operate the Caribou underground mine, crushing facility, concentrator, mine water treatment plant and tailings impoundment on May 1, 2013 and anticipates commencing operations at the Caribou mill in early 2014 (Trevali Press Release May 1, 2013). Arsenault (2013) has provided a Mineral Resource statement for the Caribou Property that is reported in Table 23.1.

TABLE 23.1
MINERAL RESOURCE STATEMENT FOR THE CARIBOU PROPERTY

Mineral Resource Statement*, Caribou Project, Bathurst New Brunswick, SRK Consulting, January 17, 2013.

Category	Quantity	Grade					Metal				
		Au	Ag	Pb	Zn	Cu	Au	Ag	Pb	Zn	Cu
	Mt	g/t	g/t	%	%	%	M oz	M oz	M lbs	M lbs	M lbs
Underground**											
Measured	5.61	0.84	84.64	2.93	6.91	0.46	0.15	15.28	362.69	855.36	56.94
Indicated	1.62	1.06	83.68	2.94	7.28	0.34	0.06	4.36	104.95	259.87	12.14
Measured and Indicated	**7.23**	**0.89**	**84.43**	**2.93**	**6.99**	**0.43**	**0.21**	**19.64**	**467.64**	**1,115.23**	**69.08**
Inferred	3.66	1.23	78.31	2.81	6.95	0.32	0.14	9.21	226.60	560.44	25.80

* Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.
** Underground mineral resources are reported at a cut-off grade of 5% Zn equivalent. Cut-off grades are based on Price for Au of $1470 per ounce, Ag is $26 per ounce, Cu is $3.39 per pound, Pb is $1.18 per pound, and Zn is $1.14 per pound. A recovery of 83% was applied to Zn, 71% was applied to Pb, 57% was applied to Cu, 45% was applied to Ag, and 40% was applied to Au.

The property containing the past-producing Restigouche Mine corresponds to mining lease number 255. The Restigouche massive sulphide deposit comprises at least two separate lenses of massive sulfide, which coalesce in the central part of the deposit and are underlain by a chlorite-pyrite stringer zone. Pelletier and Beausoleil (2006) report that the Restigouche deposit has Measured and Indicated Resources totaling 756,000 tonnes grading 7.11% Zn, 5.55% Pb, and 101.8 g/t Ag. The mineral resources for the Restigouche deposit are considered historical mineral resources. P&E cautions that a Qualified Person has not done the work necessary to verify their validity or reliability. The key assumptions and parameters used to prepare the estimate have not been verified by P&E and as such the estimates should not be relied upon.

P&E has not done sufficient work to verify the information about the mineral properties discussed in this section of the technical report and the information in this section of the report is not necessarily indicative of the mineralization on the Murray Brook Property.

24.0 OTHER RELEVANT DATA AND INFORMATION

P&E is not aware of any other relevant data or information as of the effective date of this report.

25.0 INTERPRETATION AND CONCLUSIONS

Due to the nature of a PEA, no mineral reserves have been defined at this stage of study.

The economic criteria used to distinguish between mill feed material and waste rock was based on a NSR value approach due to the poly-metallic nature of the mineralization.

The Murray Brook mining operation would be a conventional open pit operation using standard hard rock mining methods. A truck fleet consisting of 90-tonne trucks would be optimal for the size of this project.

Three types of waste materials would be generated by the mining operation: overburden; oxide waste rock; and hard waste rock.

P&E has developed a conceptual reclamation plan for the proposed Project that includes the relocation of previously reclaimed, treated gossan materials, and has developed preliminary reclamation cost estimates based on available information and stated assumptions. There is a possibility that the estimated environmental protection and reclamation costs could be higher than estimated. Key identified aspects that have the potential to affect the projected economic outcome include the quantity of acid generating, potential acid generating and non-acid generating, mine materials; and mine waste and water management. Further investigations will be required in these areas.

Public support and acceptance of the project would also be crucial for the permitting and development and eventual closure of the Project. MBJV reports that it has received generally positive support for the Project and has indicated that it is working towards the commencement of a formalized, wider scope community communication program, and that it understands the importance of obtaining input and maintaining good relationships with communities, First Nations and others that could be impacted by the Project. It may be beneficial for MBJV to consider developing its community communication program in consultation with regulatory authorities, community consultation specialists and others as appropriate (see Section 26.0). P&E is not aware of any current public acceptance obstacles to the development of the Project.

The test work shows that the Murray Brook deposit is metallurgically difficult, which is a feature of other deposits in the Bathurst camp. A saleable zinc product can be readily made, but copper and lead concentrates typically exhibit low grades and recoveries. Murray Brook mineralized material requires very fine primary grinding for adequate liberation of values and comminution costs will be higher than typical.

The test work carried out to date indicates that a high degree of control in the metallurgical process will be required to achieve consistent results in full scale production. Considerable metallurgical work will be required to optimize and develop confidence in the selected flowsheet and to evaluate possible flowsheet options.

26.0 RECOMMENDATIONS

The Base Case of this PEA shows that the Project has economic potential for producing copper, lead and zinc concentrates.

Note: This PEA is preliminary in nature and its mineable tonnage includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves and there is no certainty that the preliminary assessment will be realized. Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

P&E recommends that MBJV advance the project with extended and advanced technical studies, particularly in metallurgical, geotechnical and environmental matters with the intention to advance the project to a feasibility stage.

Specifically, it is recommended that MBJV take the following actions to develop the project to a Pre-Feasibility Study level.

- Continue metallurgical test work on larger scale to confirm and improve the process design with the goal of improving metal recoveries;
- Assess the possibility of re-processing the reclaimed gossan, while taking metallurgical, environmental protection and legal aspects into consideration;
- Characterize the acid generation / acid consuming potential and characteristics of the mine materials likely to be produced by the Project;
- Develop a preliminary mine waste plan and site water management plan at the next technical assessment stage of the Project;
- Carry out preliminary geotechnical investigations in the area of the proposed open pit;
- Carry out a preliminary hydrogeological investigation and modelling study for the Project;
- Review the envisaged Project with regulatory authorities including possible environmental and social impact assessment study requirements and related public consultation aspects, time lines, etc. and consider proactively commencing studies that are likely to be required or that may require an extended time to complete. The environmental assessment requirements are established as part of the environmental impact assessment process.
- Investigate and negotiate preliminary commercial parameters of key project components such as power supply, fuel and grinding media and key reagents;
- The flotation behaviour of partially altered feed material in the mill is inferior to primary material. The treatment of this material has not yet been addressed but should be a component of a future test-work program.

P&E also recommends that other exploration targets in the area continue to be identified and investigated to provide supplemental mill feed in the future.

27.0 REFERENCES

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El Nino Ventures Inc., News Release February 28, 2012, El Nino Ventures Inc. Announces an Initial NI 43-101 Mineral Resource Estimate for the Murray Brook Zn-Cu-Pb-Ag-Au Deposit, Bathurst Mining Camp, NB

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El Nino Ventures Inc., News Release June 5, 2012, El Nino Ventures Inc. Announces High Grade Zinc, Gold and Silver intersections over significant intervals from the Phase II 18,000 metre Drill Program at Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick

El Nino Ventures Inc., News Release June 27, 2012, El Nino Ventures Inc. continues to intersect High Grade Zinc, Gold and Silver over significant intervals at Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick

El Nino Ventures Inc., News Release July 17, 2012, El Nino Ventures Inc. Drills 9.23% Zinc, 108.7 gpt Silver, 0.18% Copper, 0.336 gpt gold and 2.58% lead over 31 metres at the Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick

El Nino Ventures Inc., News Release July 30, 2012, El Nino Ventures Inc. Announces a 10.34%Zinc, 126.0 gpt Silver, 0.29%Copper, 1.396 gpt Gold and 4.11% Lead drill intersection over 13 Metres at Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick

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El Nino Ventures Inc., News Release August 28, 2012, El Nino Ventures Inc. Announces a 122 metre thick mineralized interval of massive sulphide mineralization assaying 6.15% Zn, 0.10% Cu, 1.92% Pb, 0.64 gpt Au and 70.6 gpt Ag at the Murray Brook Deposit, New Brunswick

El Nino Ventures Inc., News Release September 6, 2012, El Nino Ventures Inc. Announces completion of its 35% earn-in; Receives Notice of option to earn additional 15% (for a total of 50%) in Murray Brook Polymetallic Project, New Brunswick, Canada

El Nino Ventures Inc., News Release October 2, 2012, El Nino Ventures Inc. announces a 45.4 metre drill intersection assaying 8.49% Zinc, 0.18% Copper, 4.58% Leas, 0.59 gpt Gold and 152.2 gpt Silver at the Murray Brook Deposit, New Brunswick

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28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

EUGENE J. PURITCH, P.ENG.

1. I, Eugene J. Puritch, P. Eng., residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:
2. I am an independent mining consultant and President of P & E Mining Consultants Inc.
3. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada" (the "Technical Report") with an effective date of June 4, 2013.
4. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for Bachelor's Degree in Engineering Equivalency. I am a mining consultant currently licensed by the Professional Engineers of Ontario (License No. 100014010) and registered with the Ontario Association of Certified Engineering Technicians and Technologists as a Senior Engineering Technologist. I am also a member of the National and Toronto Canadian Institute of Mining and Metallurgy.

   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 have practiced my profession continuously since 1978. My summarized career experience is as follows:

   - Mining Technologist - H.B.M.& S. and Inco Ltd.,..1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd.,..1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine,..1984-1986
   - Self-Employed Mining Consultant – Timmins Area,...1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, ...1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator,...1995-2004
   - President – P & E Mining Consultants Inc,... 2004-Present

1. I have visited the Property that is the subject of this report on March 18, 2013.
2. I am responsible for authoring co-authoring Sections 14, 21, 22, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
3. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
4. I co-authored the NI 43-101 compliant Technical report on the Murray Brook property titled "Technical Report On The Murray Brook Property, Restigouche County New Brunswick, Canada For El Nino Ventures Inc.", and dated April 13, 2013.
5. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
6. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 4, 2013
Signed Date: July 17, 2013

{SIGNED AND SEALED}
[Eugene Puritch]

____
Eugene J. Puritch, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

GERALD A. HARRON, P.ENG.

I, Gerald A. Harron, M.Sc., P.Eng. do hereby certify that:

I am the President of: G.A. Harron & Associates Inc. Suite 501, 133 Richmond Street West Toronto, Ontario, Canada M5H 2L3 Tel.: (416) 865-1060 Fax.: (416) 865-0213 Email: gaharron@bellnet.ca

1. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada" (the "Technical Report") with an effective date of June 4, 2013.
2. I graduated with a Bachelor of Science degree in Geology from Carleton University in 1969 and also graduated from the University of Western Ontario with a Master of Science degree in Economic Geology in 1972.
3. I am a member of the Association of Professional Engineers of Ontario, the Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories and Nunavut.
4. I have worked as a geologist for over 35 years since my graduation from university and have been involved in minerals exploration for base, precious and noble metals and uranium throughout North America, South America and Africa, during which time I directed, managed and evaluated regional and local exploration programs.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for authoring Sections 4 through 10 and 23 and co-authoring Section 11 and 12 of the Technical Report along with those sections of the Summary pertaining thereto.
7. I have conducted a site visit to the properties, on October 15 and 16, 2012.
8. I co-authored the NI 43-101 compliant Technical report on the Murray Brook property titled "Technical Report On The Murray Brook Property, Restigouche County New Brunswick, Canada For El Nino Ventures Inc.", and dated April 13, 2013.
9. I acknowledge that as of the date of the certificate, and to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in section 1.4 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Effective Date: June 4, 2013
Signed Date: July 17, 2013

{SIGNED AND SEALED}
[Gerald Harron]

Gerald A. Harron P.Eng.

CERTIFICATE OF QUALIFIED PERSON

ALFRED S. HAYDEN, P.ENG.

I, Alfred S. Hayden, P. Eng., residing at 284 Rushbrook Drive, Ontario, L3X 2C9, do hereby certify that:

1. I am currently President of:
   EHA Engineering Ltd.,
   Consulting Metallurgical Engineers
   Box 2711, Postal Stn. B.
   Richmond Hill, Ontario, L4E 1A7
2. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada" (the "Technical Report") with an effective date of June 4, 2013.
3. I graduated from the University of British Columbia, Vancouver, B.C. in 1967 with a Bachelor of Applied Science in Metallurgical Engineering. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum and a Professional Engineer and Designated Consulting Engineer registered with Professional Engineers Ontario. I have worked as a metallurgical engineer for a total of 46 years since my graduation from university.

   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.

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring of Sections 13 and 17 and co-authoring Section 21 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 4, 2013
Signed Date: July 17, 2013

{SIGNED AND SEALED}
[Alfred Hayden]

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Alfred S. Hayden, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

YUNGANG WU, P.GEO.

I, Yungang Wu, P. Geo., residing at 4334 Trail Blazer Way, Mississauga, Ontario, L5R 0C3, do hereby certify that:

1. I am an independent consulting geologist contracted by P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada" (the "Technical Report") with an effective date of June 4, 2013.
3. I am a graduate of Jilin University, China with a Master Degree in Mineral Deposits (1992). I am a geological consultant and a registered practising member of the Association of Professional Geoscientist of Ontario (Registration No. 1681). I am also a member of the Ontario Prospectors Association.

   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.
   My relevant experience for the purpose of the Technical Report is as follows:

   - Geologist –Geology and Mineral Bureau, Liaoning Province, China…………………….. 1992-1993
   - Senior Geologist – Committee of Mineral Resources and Reserves of Liaoning, China… 1993-1998
   - VP – Institute of Mineral Resources and Land Planning, Liaoning, China……………….. 1998-2001
   - Project Geologist–Exploration Division, De Beers Canada……………………………….. 2003-2009
   - Mine Geologist – Victor Diamond Mine, De Beers Canada………………………………. 2009-2011
   - Resource Geologist– Coffey Mining Canada……………………………………………….2011-2012
   - Consulting Geologist…………………………………………………………………..Present

4. I have not visited the property that is the subject of this Technical Report.
5. I am responsible for co-authoring Section 14 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: June 4, 2013
Signed Date: July 17, 2013

{SIGNED AND SEALED}

[Yungang Wu]

Yungang Wu, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

KEN KUCHLING, P.ENG.

I, Ken Kuchling, P. Eng., residing at 33 University Ave., Toronto, Ontario, M5J 2S7, do hereby certify that:

2. I am a senior mining consultant with KJ Kuchling Consulting Ltd. located at #1903-33 University Ave, Toronto, Ontario Canada contracted by P&E Mining Consultants Inc.
3. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada" (the "Technical Report") with an effective date of June 4, 2013.

   I graduated with a Bachelor degree in Mining Engineering in 1980 from McGill University and a M. Eng degree in Mining Engineering from UBC in 1984. I have worked as a mining engineer for a total of 31 years since my graduation from university. My relevant work experience for the purpose of the Technical Report is 12 years as an independent mining consultant in commodities such as gold, copper, potash, diamonds, molybdenum, tungsten, and bauxite. I have practiced my profession continuously since 1980. I am a member of the Professional Engineers of Ontario.

   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.

   My relevant experience for the purpose of the Technical Report is:

   - Associate Mining Engineer, P&E Mining Consultants Inc. ....2011 – Present
   - Mining Consultant, KJ Kuchling Consulting Ltd. ....2000 – Present
   - Senior Mining Engineer, Diavik Diamond Mines Inc., ... 1997 – 2000
   - Senior Mining Consultant, KJ Kuchling Consulting Ltd., ... 1995 – 1997
   - Senior Geotechnical Engineer, Terracon Geotechnique Ltd., ...1989 - 1995
   - Chief Mine Engineer, Mosaic, Esterhazy K1 Operation. .... 1985 – 1989
   - Mining Engineering, Syncrude Canada Ltd.. .... 1980 – 1983

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 16, 21, 22, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of the Issuer applying all of the tests in section 1.5 of National Instrument 43-101.
7. I have not had prior involvement with the project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 4, 2013
Signed Date: July 17, 2013

{SIGNED AND SEALED}
[Ken Kuchling]

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Ken Kuchling P.Eng.

CERTIFICATE OF QUALIFIED PERSON

DAVID A. ORAVA, P.ENG.

I, David A. Orava, M. Eng., P. Eng., residing at 19 Boulding Drive, Aurora, Ontario, L4G 2V9, do hereby certify that:

1. I am an Associate Mining Engineer at P&E Mining Consultants Inc. and President of Orava Mine Projects Ltd.
2. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada" (the "Technical Report") with an effective date of June 4, 2013.
3. I am a graduate of McGill University located in Montreal, Quebec, Canada at which I earned my Bachelor Degree in Mining Engineering (B.Eng. 1979) and Masters in Engineering (Mining - Mineral Economics Option B) in 1981. I have practiced my profession continuously since graduation. I am licensed by the Professional Engineers of Ontario (License No. 34834119).

   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.

   My summarized career experience is as follows:

   - Mining Engineer – Iron Ore Company of Canada. 1979-1980
   - Mining Engineer – J.S Redpath Limited / J.S. Redpath Engineering. 1981-1986
   - Mining Engineer & Manager Contract Development – Dynatec Mining Ltd. 1986-1990
   - Vice President – Eagle Mine Contractors 1990
   - Senior Mining Engineer – UMA Engineering Ltd. 1991
   - General Manager - Dennis Netherton Engineering 1992-1993
   - Senior Mining Engineer – SENES Consultants Ltd. 1993-2003
   - President – Orava Mine Projects Ltd. .... 2003 to present
   - Associate Mining Engineer – P&E Mining Consultants Inc. .... 2006 to present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 20 and co-authoring Sections 18, 21, 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
6. I am an independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 4, 2013
Signed Date: July 17, 2013

{SIGNED AND SEALED}
[David Orava]

____
David Orava, M.Eng., P.Eng.

CERTIFICATE OF AUTHOR

TRACY J. ARMSTRONG, P.GEO.

I, Tracy J. Armstrong, residing at 2007 Chemin Georgeville, res. 22, Magog, QC J1X 0M8, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc. and have worked as a geologist continuously since my graduation from university in 1982.
2. This certificate applies to the technical report titled “Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada” (the “Technical Report”) with an effective date of June 4, 2013.
3. I am a graduate of Queen’s University at Kingston, Ontario with a B.Sc. (HONS) in Geological Sciences (1982). I am a geological consultant currently licensed by the Order of Geologists of Québec (License 566), the Association of Professional Geoscientists of Ontario (License 1204) and the Association of Professional Engineers and Geoscientists of British Columbia, (Licence No. 34720).

   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 and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. This report is based on my personal review of information provided by the Issuer and on discussions with the Issuer’s representatives. My relevant experience for the purpose of the Technical Report is:

   - Underground production geologist, Agnico-Eagle Laronde Mine 1988-1993
   - Exploration geologist, Laronde Mine 1993-1995
   - Exploration coordinator, Placer Dome 1995-1997
   - Senior Exploration Geologist, Barrick Exploration 1997-1998
   - Exploration Manager, McWatters Mining 1998-2003
   - Chief Geologist Sigma Mine 2003
   - Consulting Geologist .... 2003-to present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 11 and 12 of this Technical Report along with those sections of the Summary pertaining thereto.
6. I am independent of issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 4, 2013
Signed Date: July 17, 2013

{SIGNED AND SEALED}
[Tracy J. Armstrong]

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Tracy J. Armstrong, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

KIRK RODGERS, P.ENG.

I, Kirk H. Rodgers, P. Eng., residing at 378 Bexhill Rd., Newmarket, Ontario, do hereby certify that:

1. I am an independent mining consultant, contracted as Vice President, Engineering by P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada" (the "Technical Report") with an effective date of June 4, 2013.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining. I subsequently attended the mining engineering programs at Laurentian University and Queen's University for a total of two years. I have met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for Bachelor's Degree in Engineering Equivalency. I have been licensed by the Professional Engineers of Ontario (License No. 39427505), from 1986 to the present. I am also a member of the National and Toronto Canadian Institute of Mining and Metallurgy.

   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. My relevant experience for the purpose of the Technical Report is:

   - Underground Hard Rock Miner, Denison Mines, Elliot Lake Ontario..1977-1979
   - Mine Planner, Cost Estimator, J.S Redpath Ltd., North Bay Ontario 1981-1987
   - Chief Engineer, Placer Dome Dona Lake Mine, Pickle Lake Ontario 1987-1988
   - Project Coordinator, Mine Captain, Falconbridge Kidd Creek Mine, Timmins, Ontario 1988-1990
   - Manager of Contract Development, Dynatec Mining, Richmond Hill, Ontario.........................1990-1992
   - General Manager, Moran Mining and Tunnelling, Sudbury, Ontario 1992-1993
   - Independent Mining Engineer 1993
   - Project Manager - Mining, Micon International, Toronto, Ontario ....1994 - 2004
   - Principal, Senior Consultant, Golder Associates, Toronto, Ontario .... 2004 – 2010
   - Independent Consultant, VP Engineering to P&E Mining Consultants Inc, Brampton Ontario 2011 – present

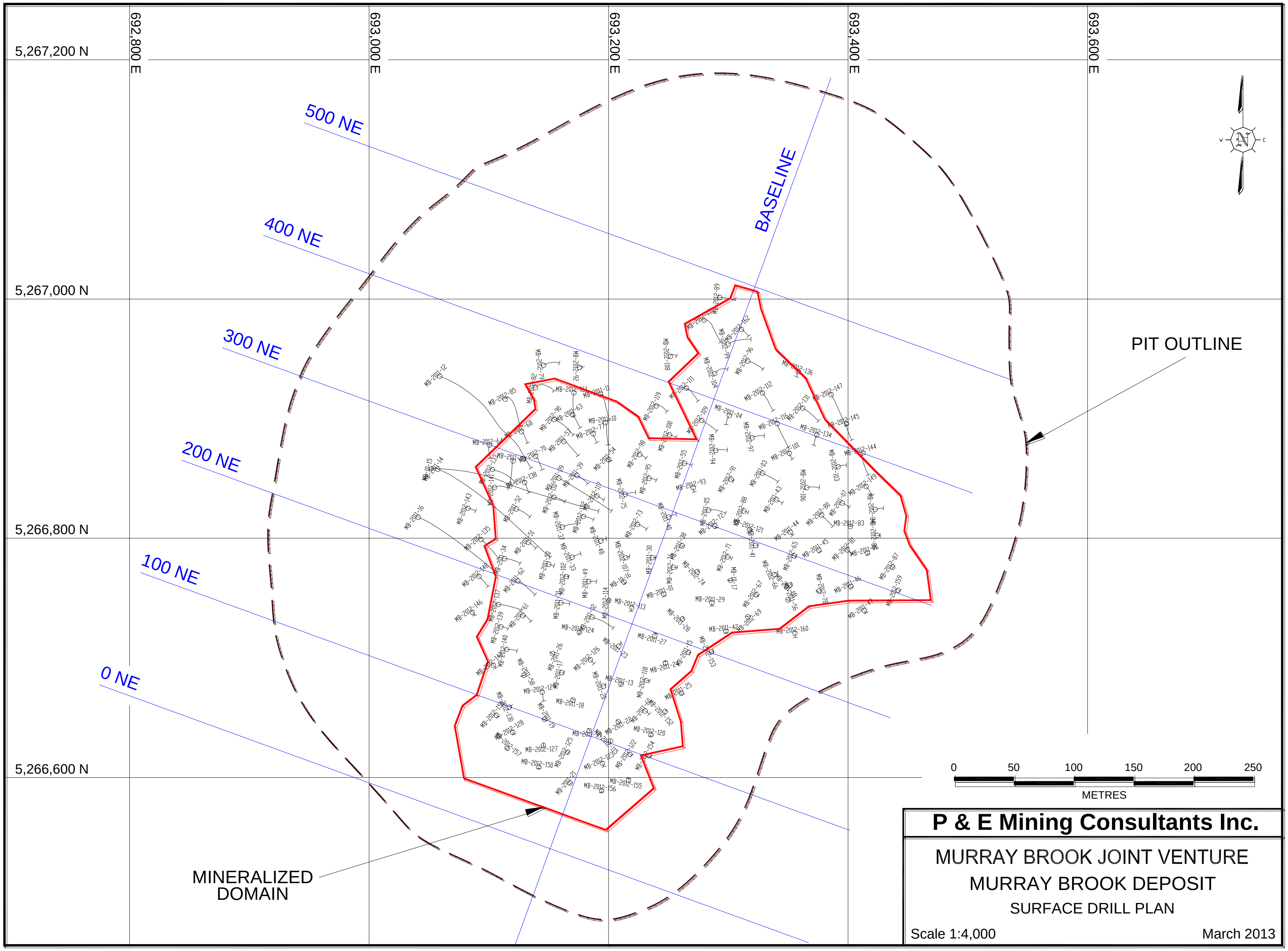
4. I am responsible for authoring Sections 2, 3, 19 and 24 and co-authoring Sections 15, 18, 22, 25 and -26 of this Technical Report along with those sections of the Summary pertaining thereto.
5. I have not visited the Property that is the subject of this Technical Report.
6. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
7. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the Property that is the subject of this Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.

Effective Date: June 4, 2013
Signed Date: July 17, 2013

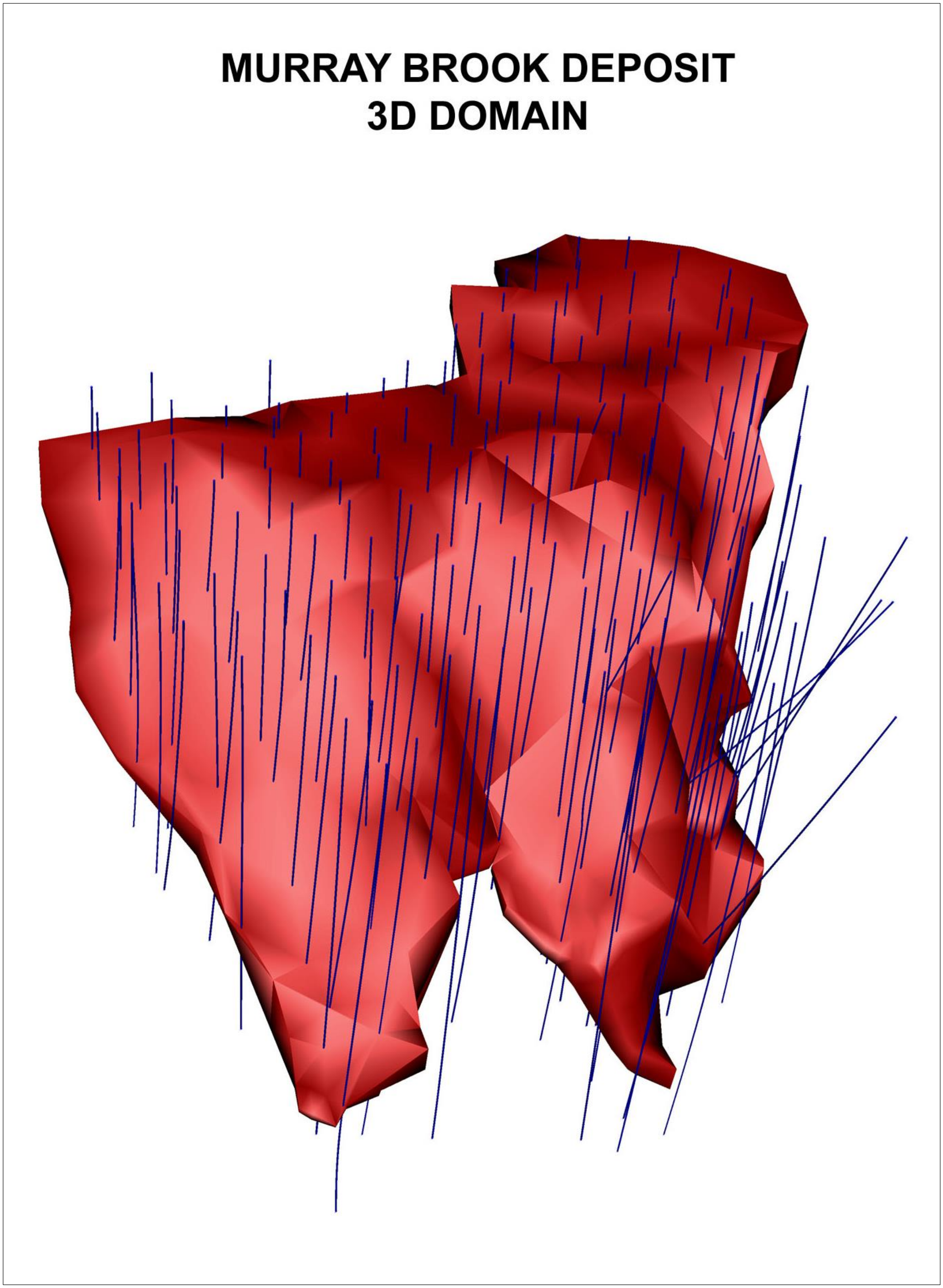
{SIGNED AND SEALED}
{Kirk Rodgers}

____
Kirk Rodgers, P.Eng.

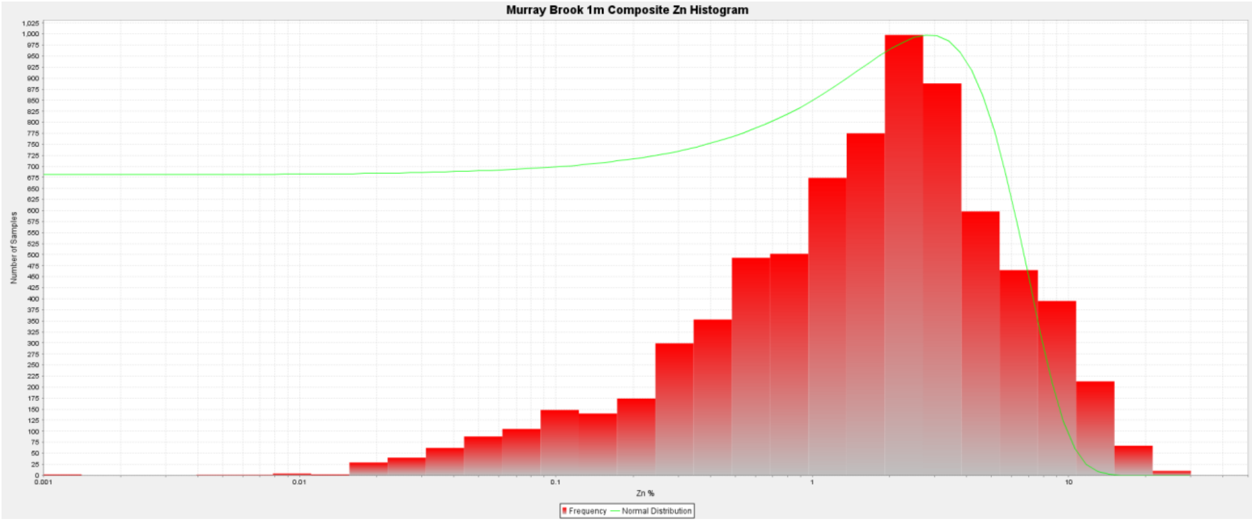
APPENDIX I. SURFACE DRILL HOLE PLAN

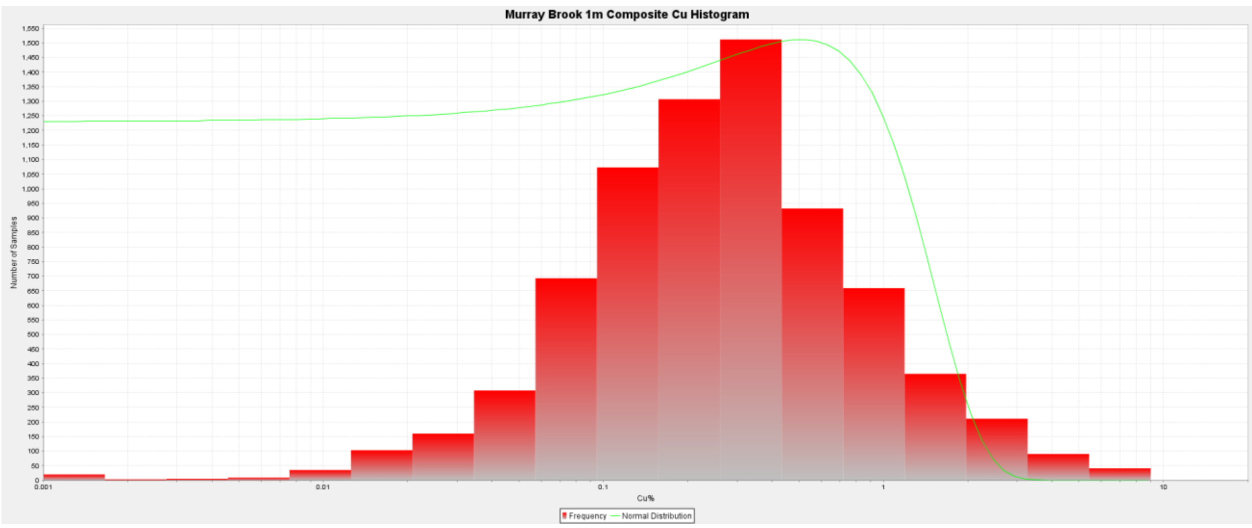

5,267,200 N
5,267,000 N
5,266,800 N
5,266,600 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
500 NE
400 NE
300 NE
200 NE
100 NE
0 NE
BASELINE
PIT OUTLINE
MINERALIZED DOMAIN
0 50 100 150 200 250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
SURFACE DRILL PLAN
Scale 1:4,000
March 2013

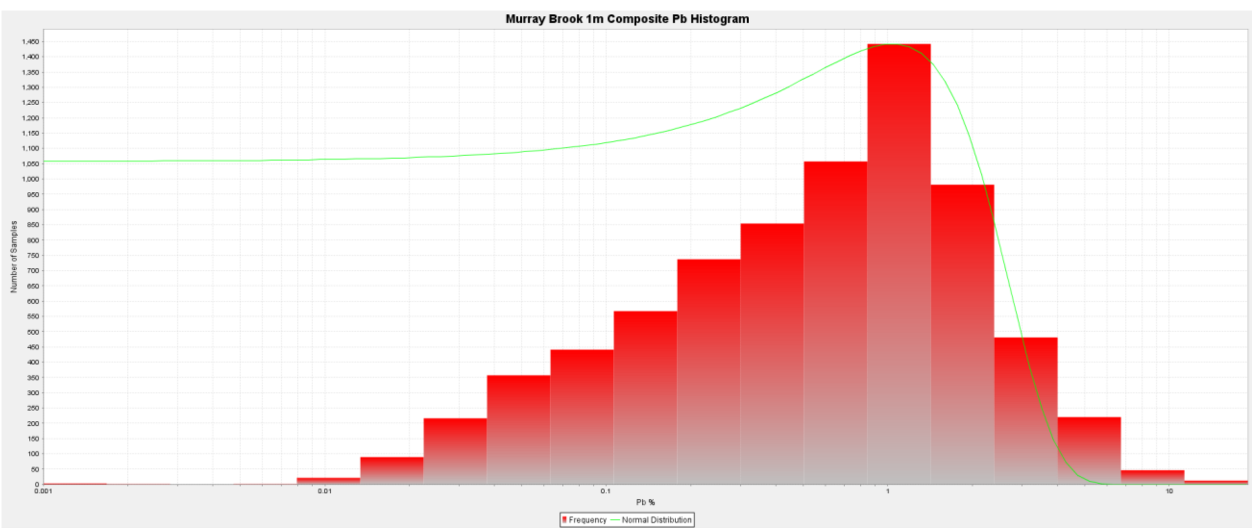
APPENDIX II. 3D DOMAIN

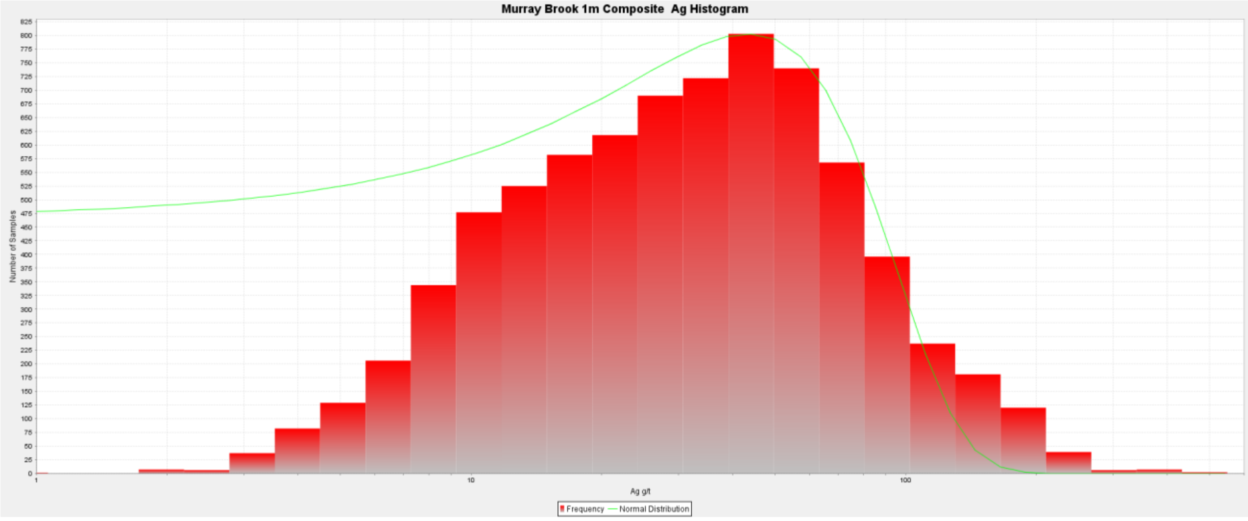

MURRAY BROOK DEPOSIT
3D DOMAIN

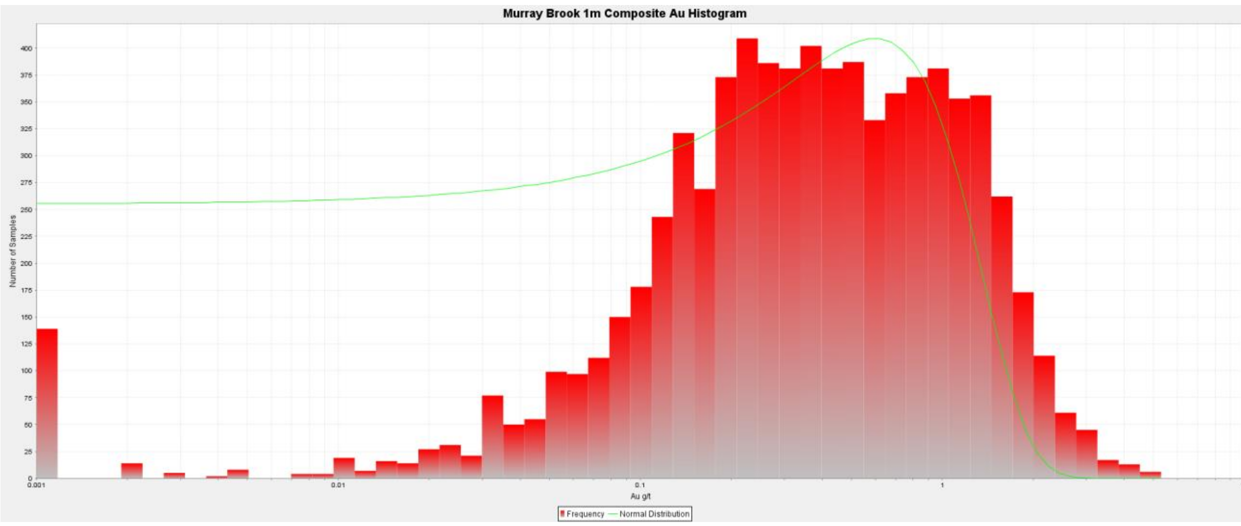
APPENDIX III. LOG NORMAL HISTOGRAMS


Murray Brook 1m Composite Zn Histogram
Number of Samples
Zn %
0.001
0.01
0.1
1
10
Frequency
Normal Distribution

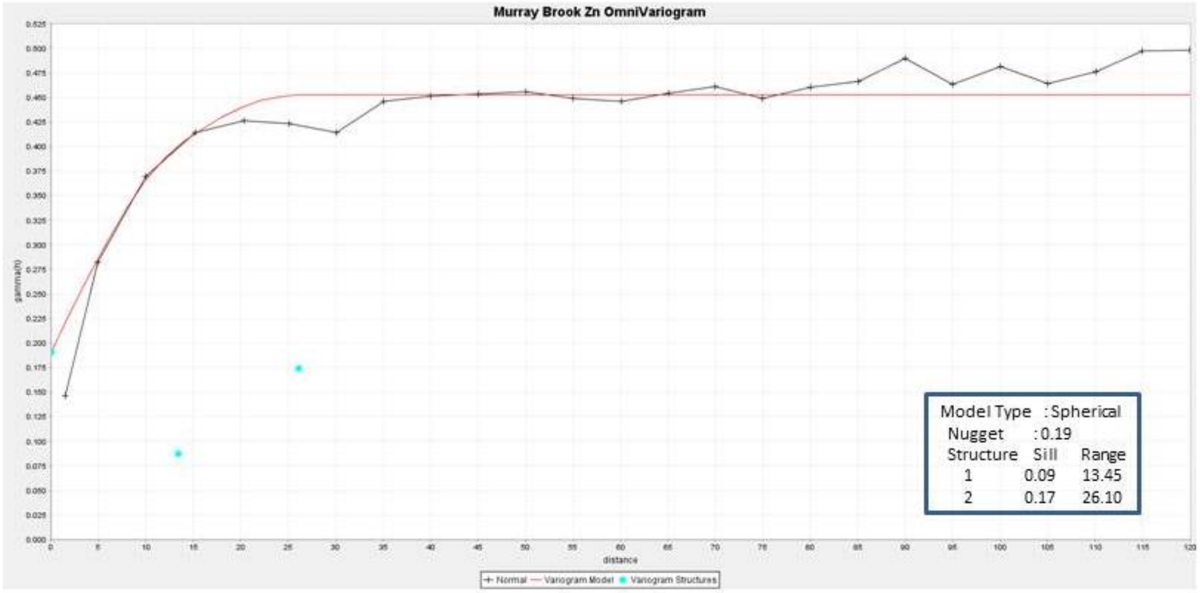

Murray Brook 1m Composite Cu Histogram
Number of Samples
Cu%
0.001
0.01
0.1
1
10
Frequency
Normal Distribution

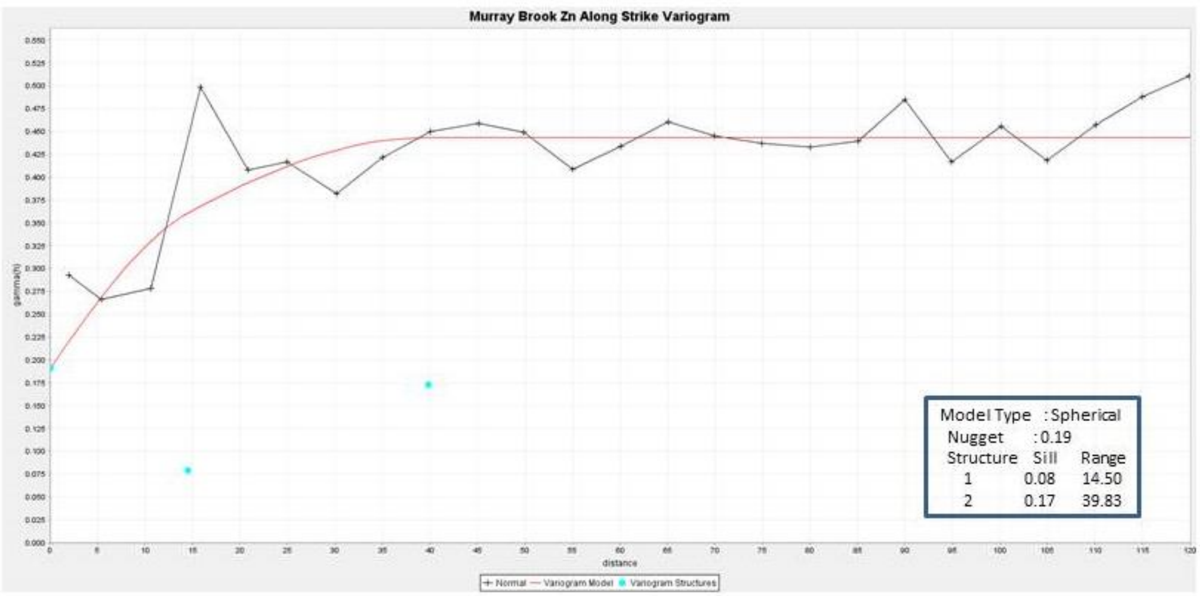

Murray Brook 1m Composite Pb Histogram
Number of Samples
Pb %
0.001
0.01
0.1
1
10
Frequency
Normal Distribution

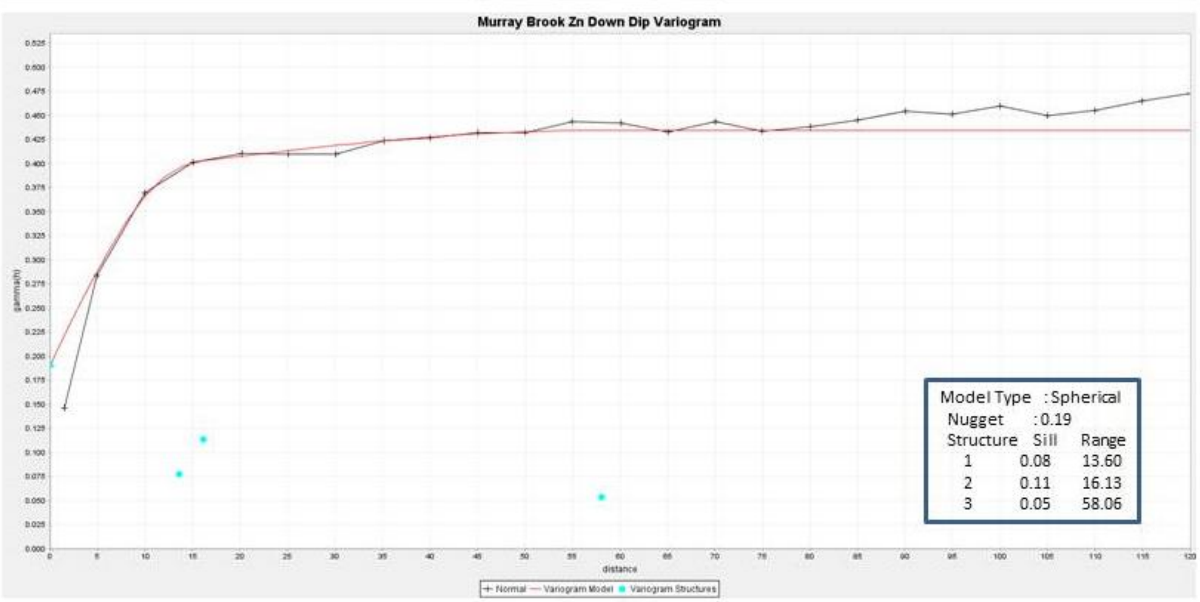

Murray Brook 1m Composite Ag Histogram
Number of Samples
Ag g/t
Frequency
Normal Distribution

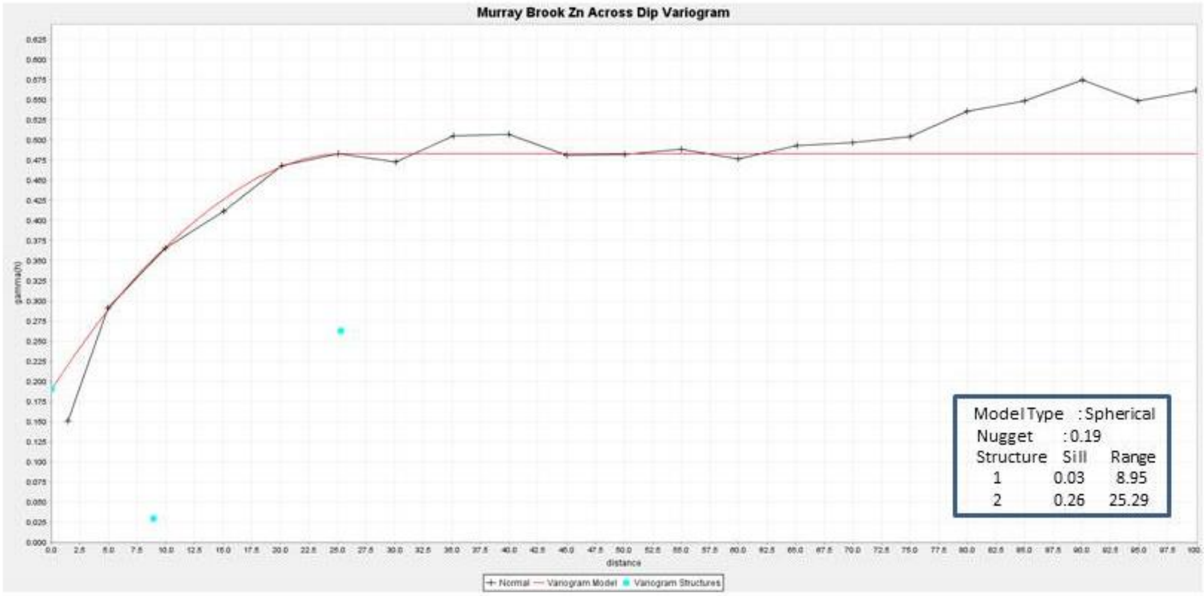

Murray Brook 1m Composite Au Histogram
Number of Samples
Au g/t
Frequency
Normal Distribution

APPENDIX IV. VARIOGRAMS

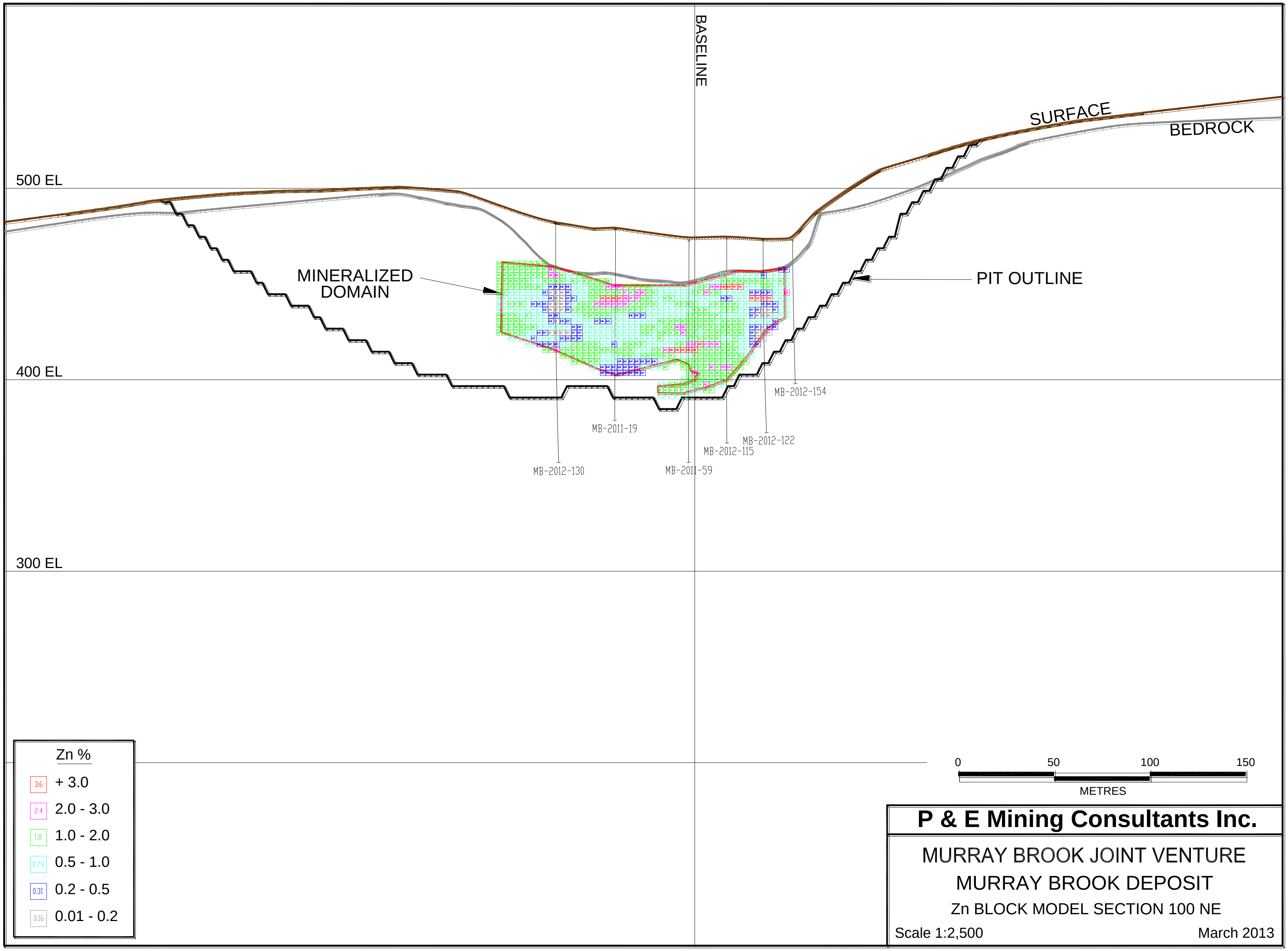

Murray Brook Zn OmniVariogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures
Model Type : Spherical
Nugget : 0.19
Structure Sill Range
1 0.09 13.45
2 0.17 26.10

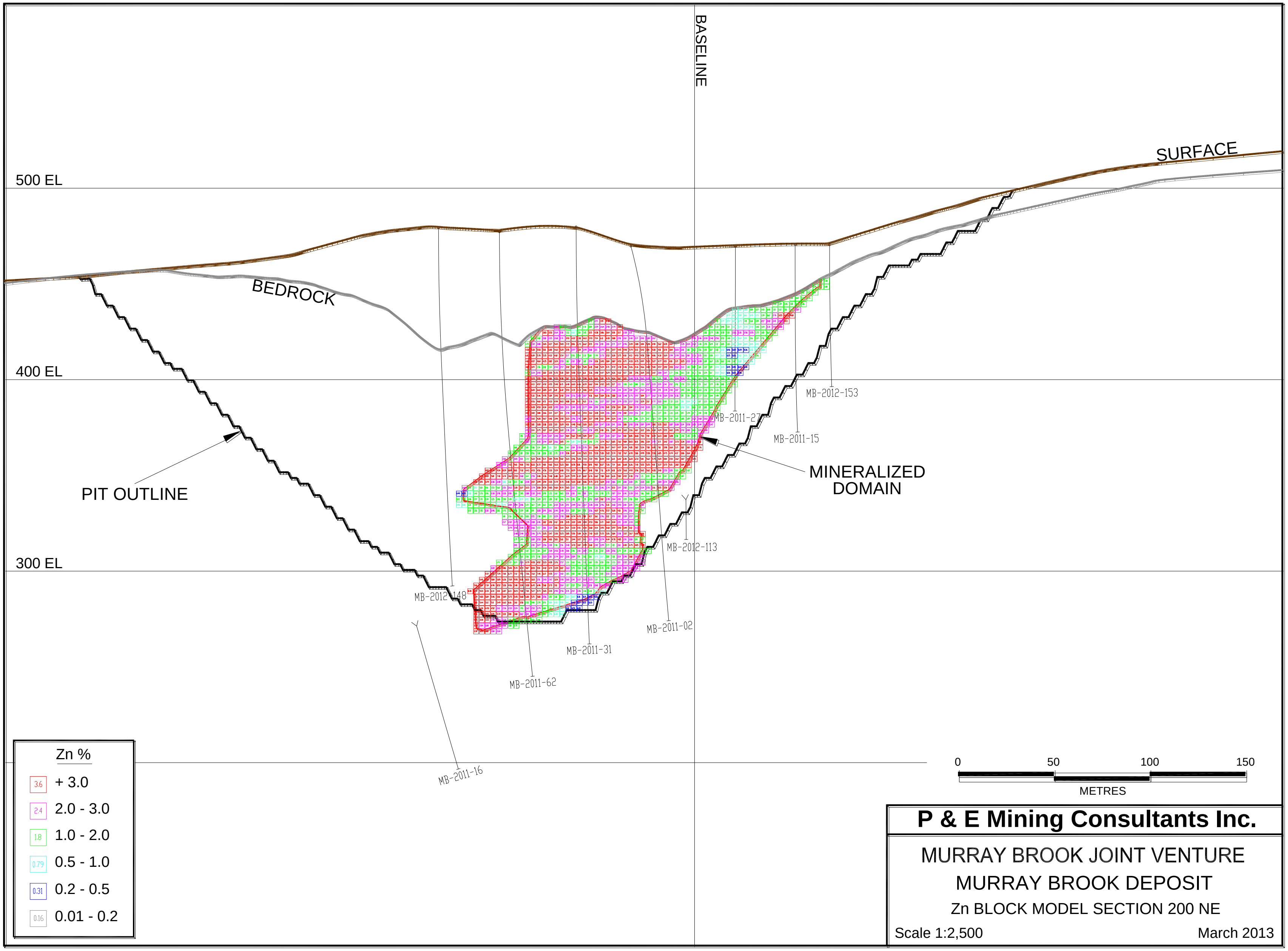

Murray Brook Zn Along Strike Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures
Model Type : Spherical
Nugget : 0.19
Structure Sill Range
1 0.08 14.50
2 0.17 39.83

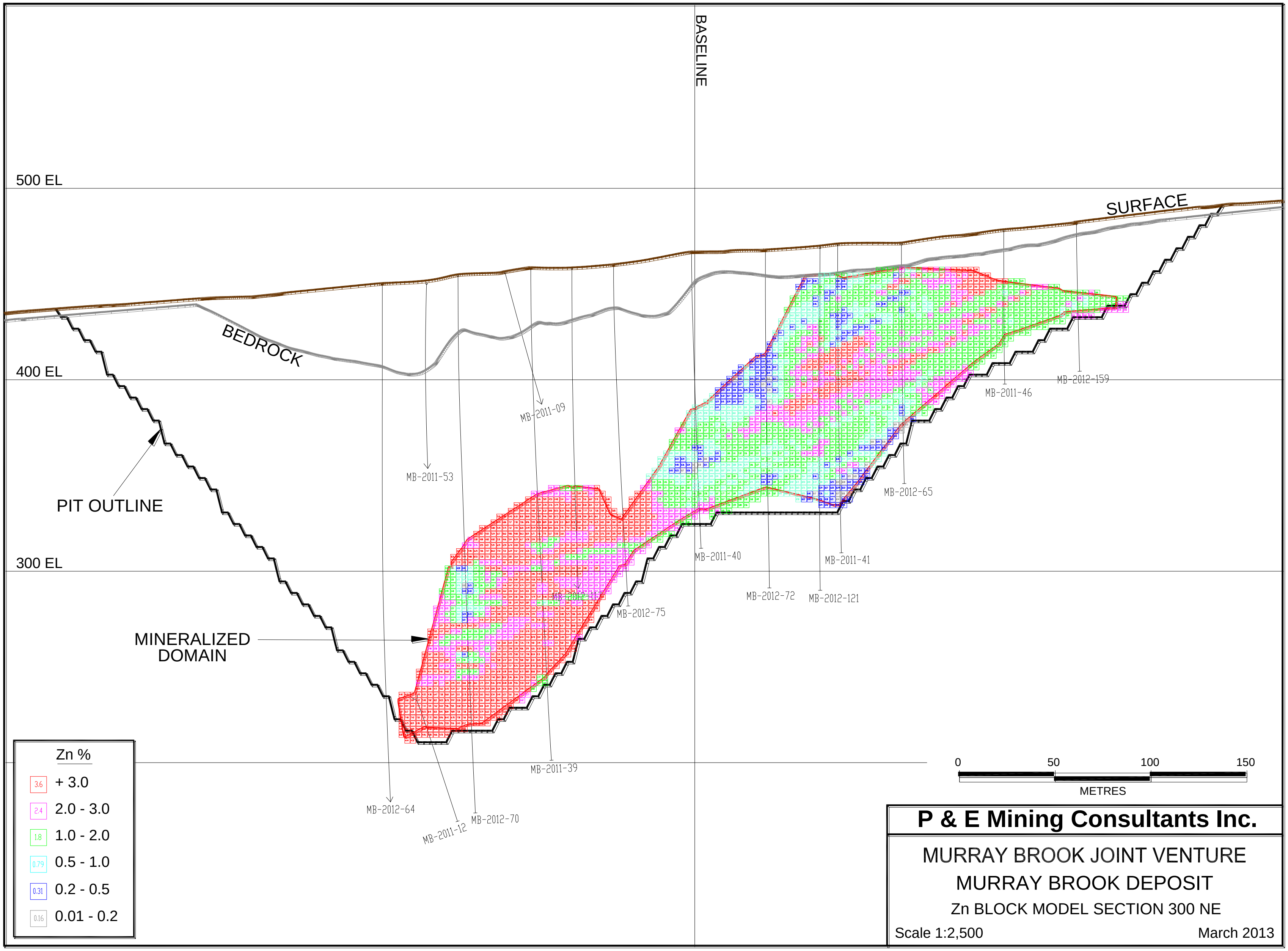

Murray Brook Zn Down Dip Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures
Model Type : Spherical
Nugget : 0.19
Structure Sill Range
1 0.08 13.60
2 0.11 16.13
3 0.05 58.06


Murray Brook Zn Across Dip Variogram
Model Type : Spherical
Nugget : 0.19
Structure Sill Range
1 0.03 8.95
2 0.26 25.29
distance
Normal
Variogram Model
Variogram Structures

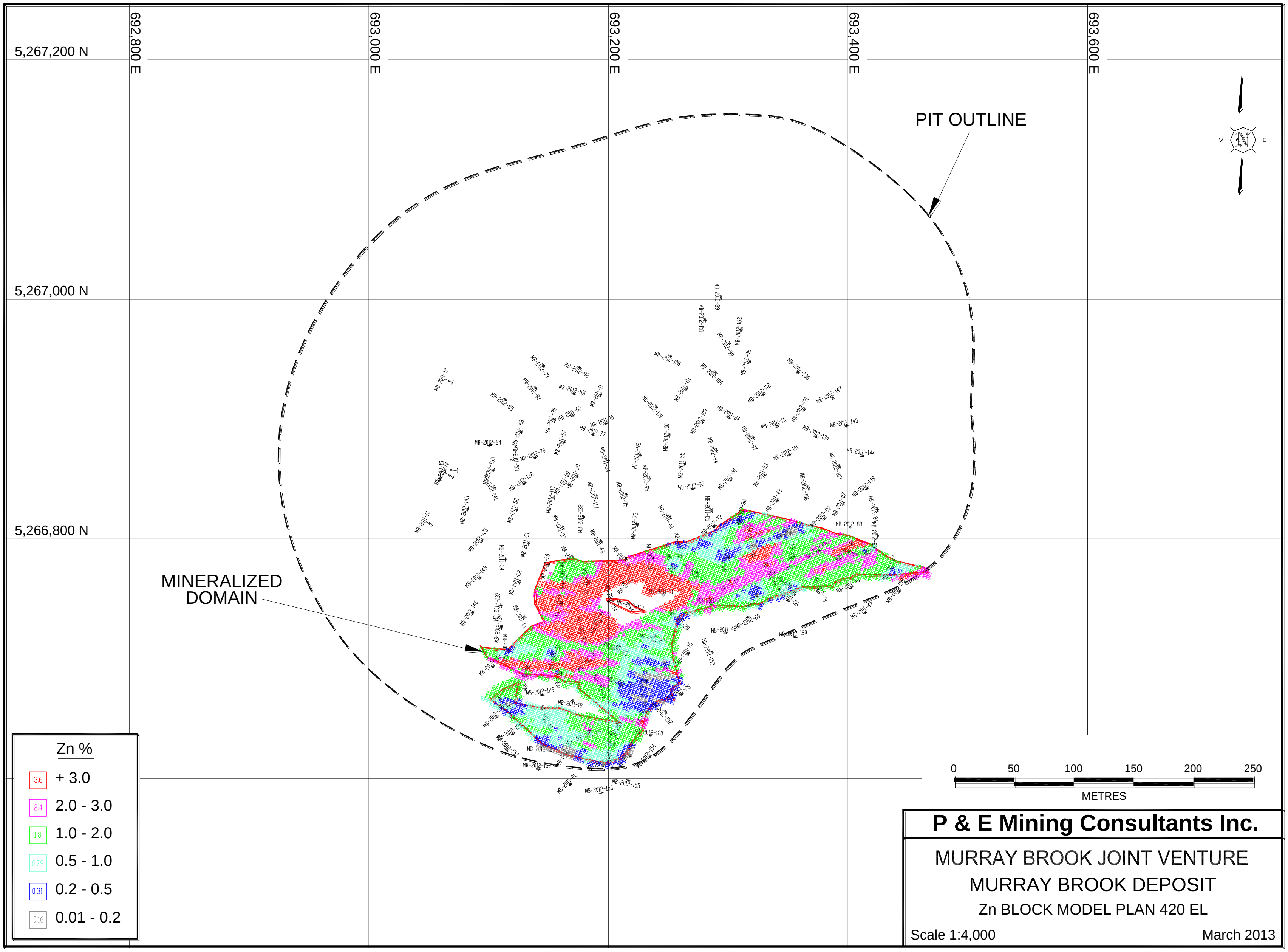
APPENDIX V. ZN BLOCK MODEL CROSS SECTIONS AND PLANS


BASELINE
SURFACE
BEDROCK
500 EL
400 EL
300 EL
MINERALIZED
DOMAIN
PIT OUTLINE
MB-2012-154
MB-2011-19
MB-2012-122
MB-2012-115
MB-2012-130
MB-2011-59
Zn %
+ 3.0
2.0 - 3.0
1.0 - 2.0
0.5 - 1.0
0.2 - 0.5
0.01 - 0.2
0
50
100
150
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
Zn BLOCK MODEL SECTION 100 NE
Scale 1:2,500
March 2013

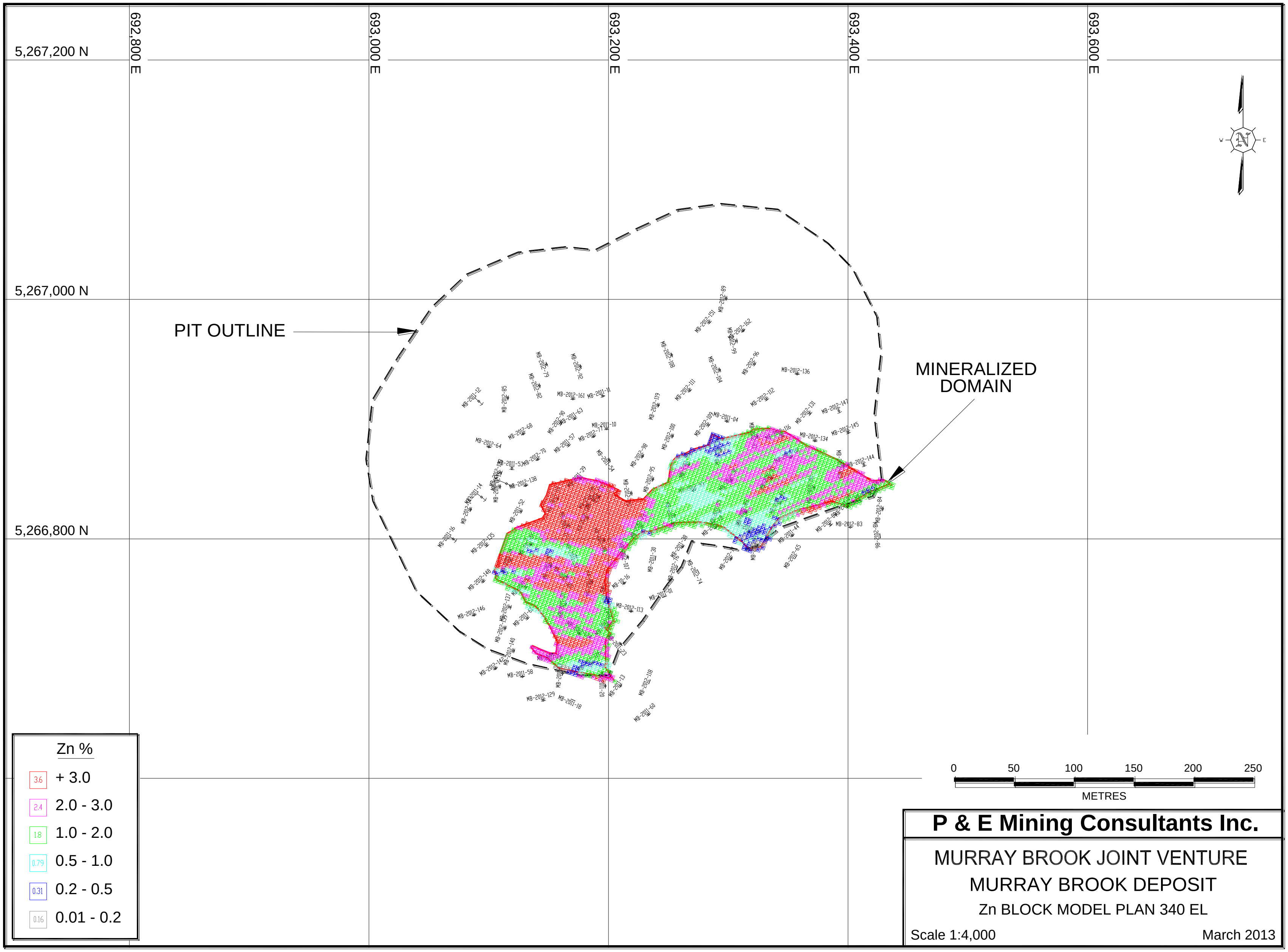

BASELINE
SURFACE
500 EL
400 EL
300 EL
BEDROCK
PIT OUTLINE
MINERALIZED DOMAIN
MB-2012-153
MB-2011-27
MB-2011-15
MB-2012-113
MB-2012-148
MB-2011-02
MB-2011-31
MB-2011-62
MB-2011-16
Zn %
+ 3.0
2.0 - 3.0
1.0 - 2.0
0.5 - 1.0
0.2 - 0.5
0.01 - 0.2
0
50
100
150
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
Zn BLOCK MODEL SECTION 200 NE
Scale 1:2,500
March 2013

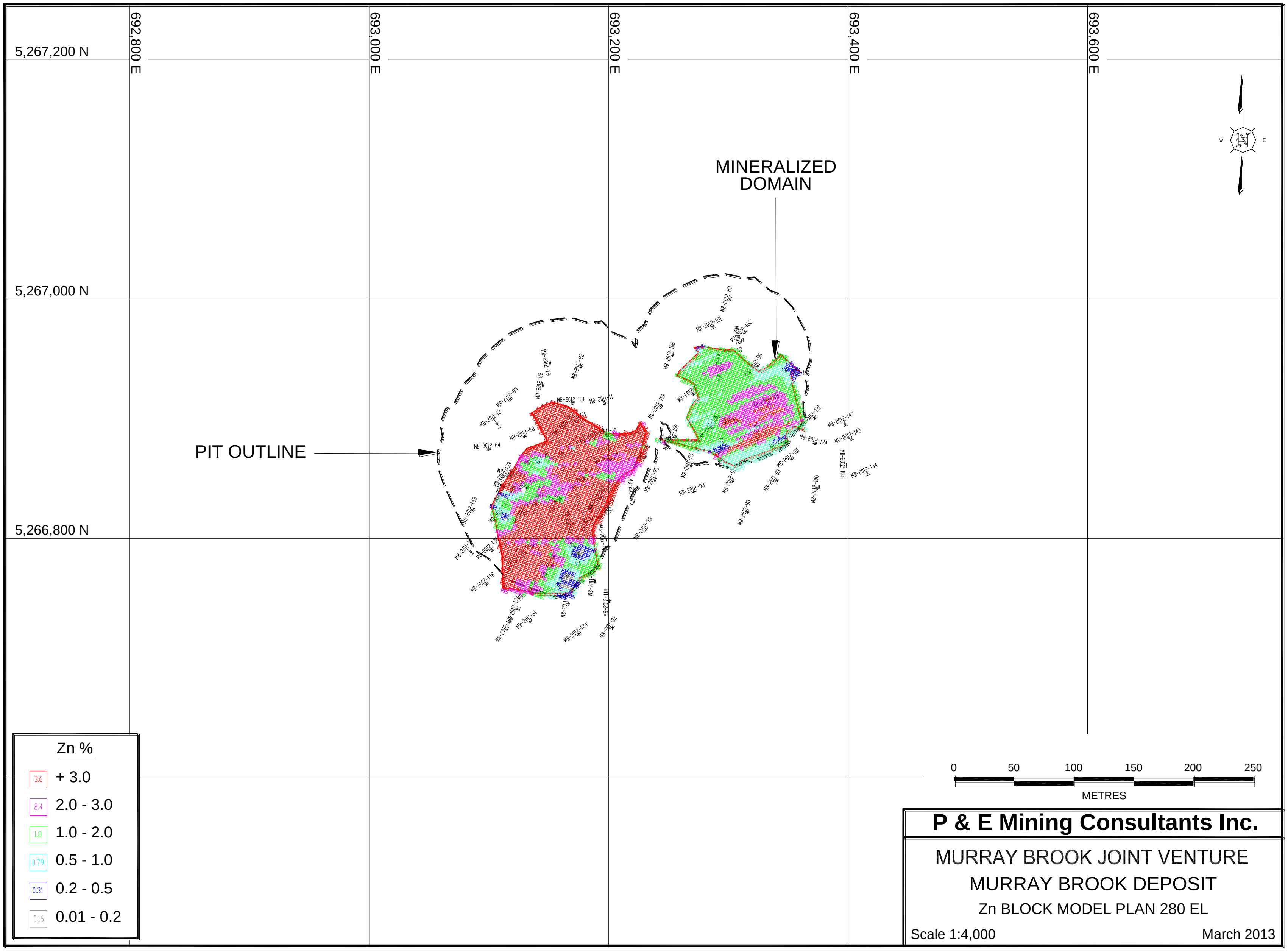


BASELINE
500 EL
400 EL
300 EL
SURFACE
BEDROCK
PIT OUTLINE
MINERALIZED
DOMAIN
MB-2011-09
MB-2011-53
MB-2011-40
MB-2011-41
MB-2011-46
MB-2012-159
MB-2012-65
MB-2012-72
MB-2012-121
MB-2012-75
MB-2011-39
MB-2012-64
MB-2011-12
MB-2012-70
Zn %
+ 3.0
2.0 - 3.0
1.0 - 2.0
0.5 - 1.0
0.2 - 0.5
0.01 - 0.2
0
50
100
150
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
Zn BLOCK MODEL SECTION 300 NE
Scale 1:2,500
March 2013



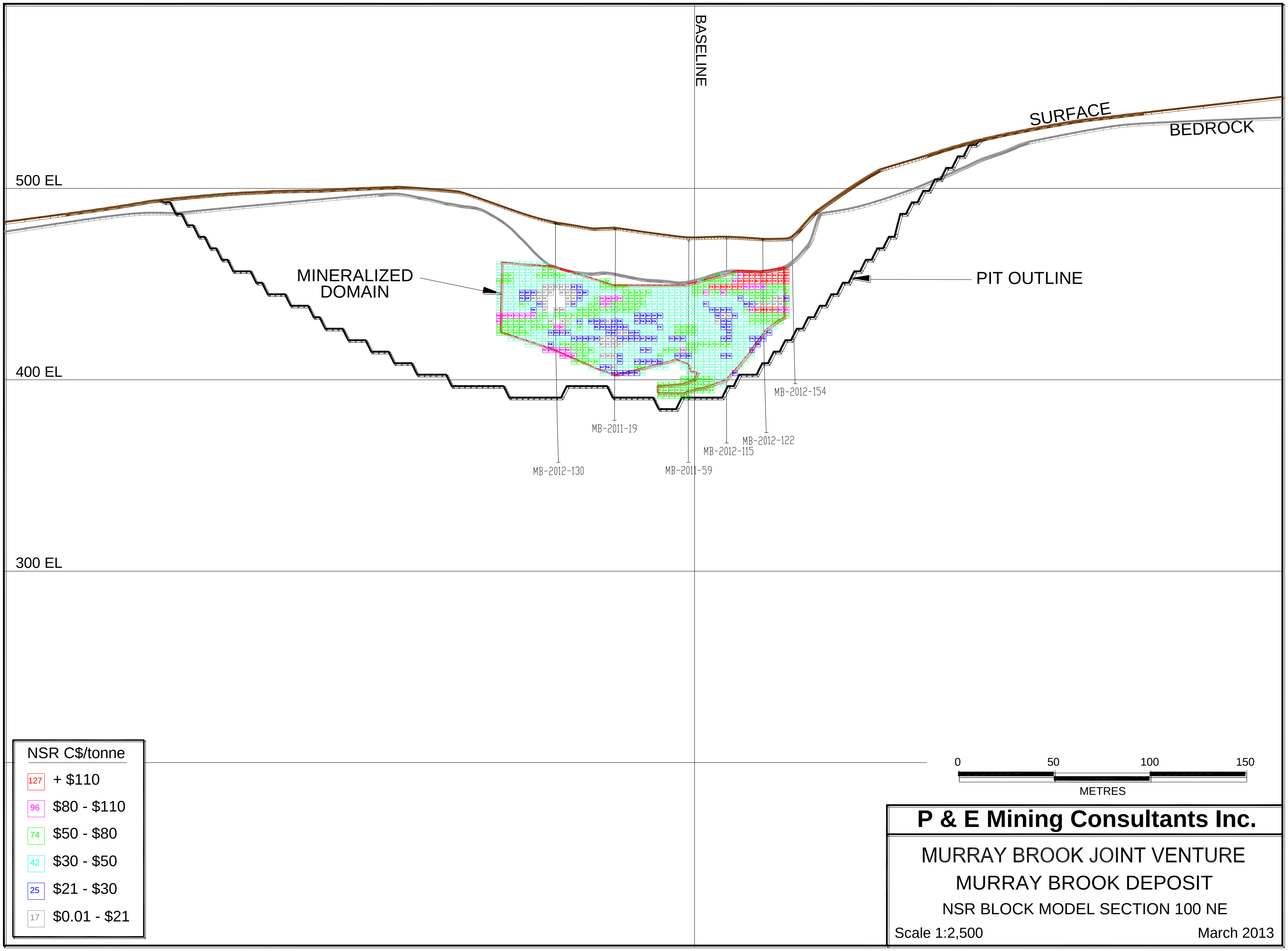
5,267,200 N
5,267,000 N
5,266,800 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
PIT OUTLINE
MINERALIZED DOMAIN
Zn %
+ 3.0
2.0 - 3.0
1.0 - 2.0
0.5 - 1.0
0.2 - 0.5
0.01 - 0.2
0
50
100
150
200
250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
Zn BLOCK MODEL PLAN 420 EL
Scale 1:4,000
March 2013

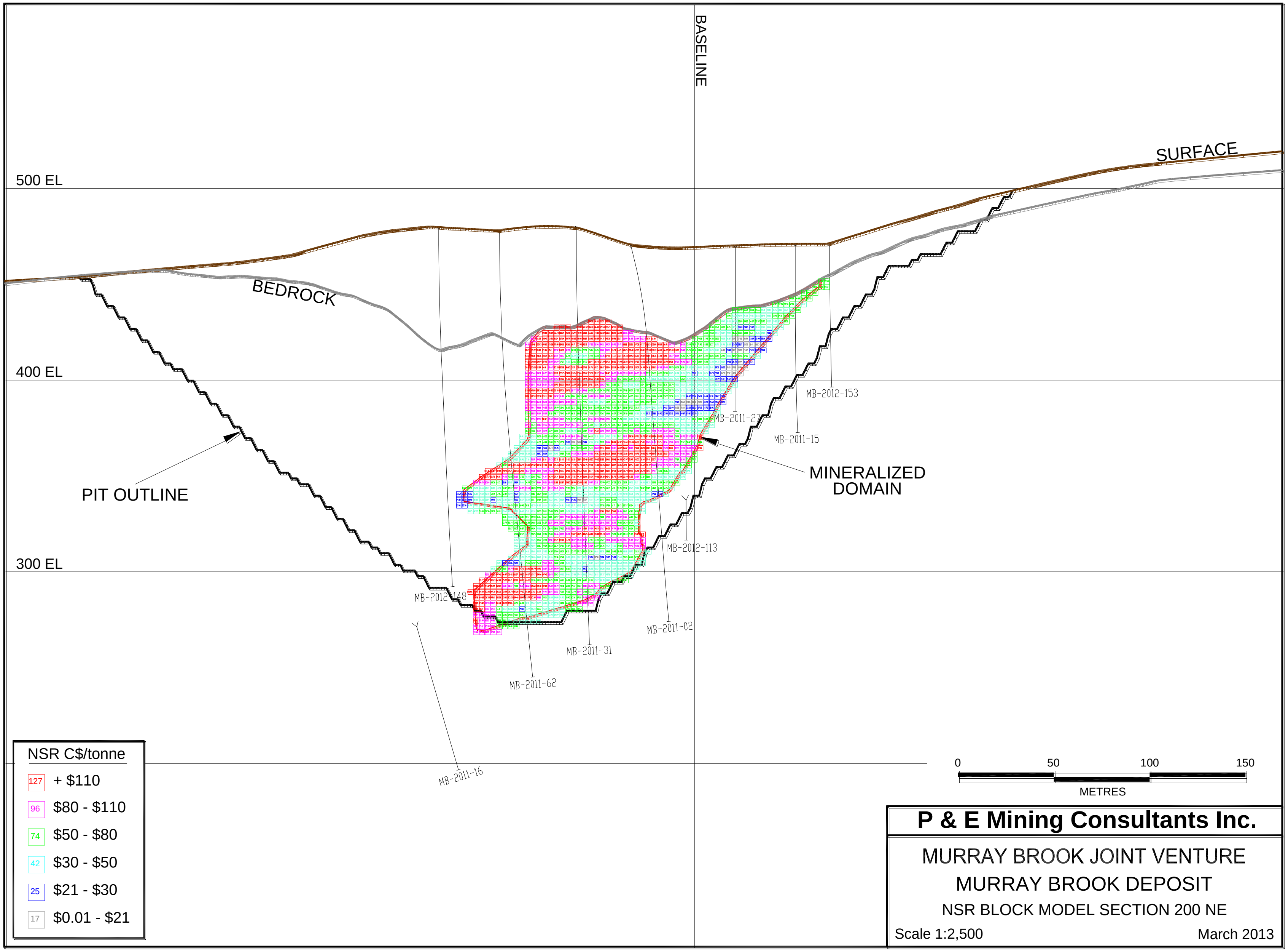

5,267,200 N
5,267,000 N
5,266,800 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
PIT OUTLINE
MINERALIZED DOMAIN
Zn %
+ 3.0
2.0 - 3.0
1.0 - 2.0
0.5 - 1.0
0.2 - 0.5
0.01 - 0.2
0
50
100
150
200
250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
Zn BLOCK MODEL PLAN 340 EL
Scale 1:4,000
March 2013

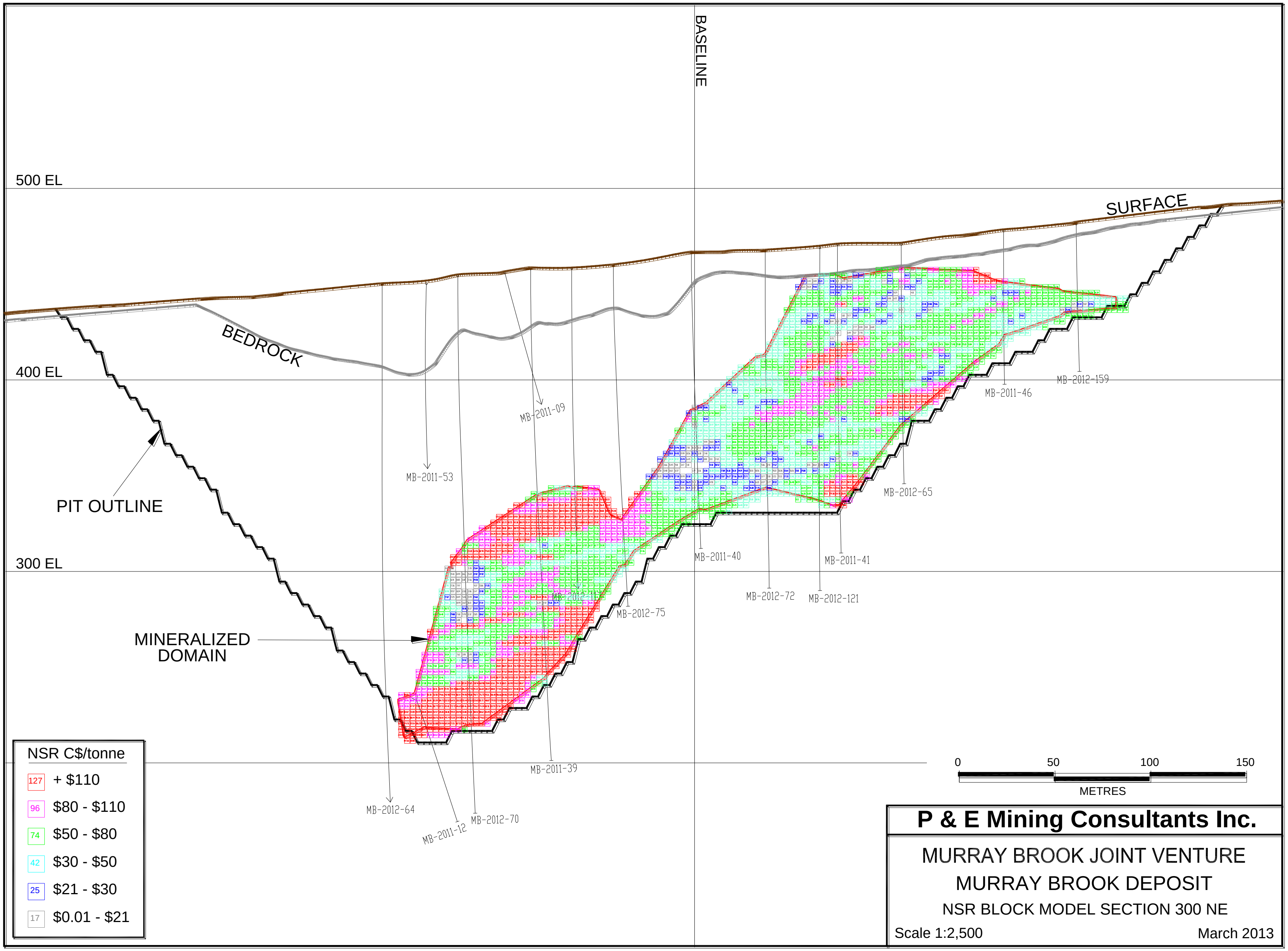


5,267,200 N
5,267,000 N
5,266,800 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
MINERALIZED DOMAIN
PIT OUTLINE
Zn %
+ 3.0
2.0 - 3.0
1.0 - 2.0
0.5 - 1.0
0.2 - 0.5
0.01 - 0.2
0
50
100
150
200
250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
Zn BLOCK MODEL PLAN 280 EL
Scale 1:4,000
March 2013

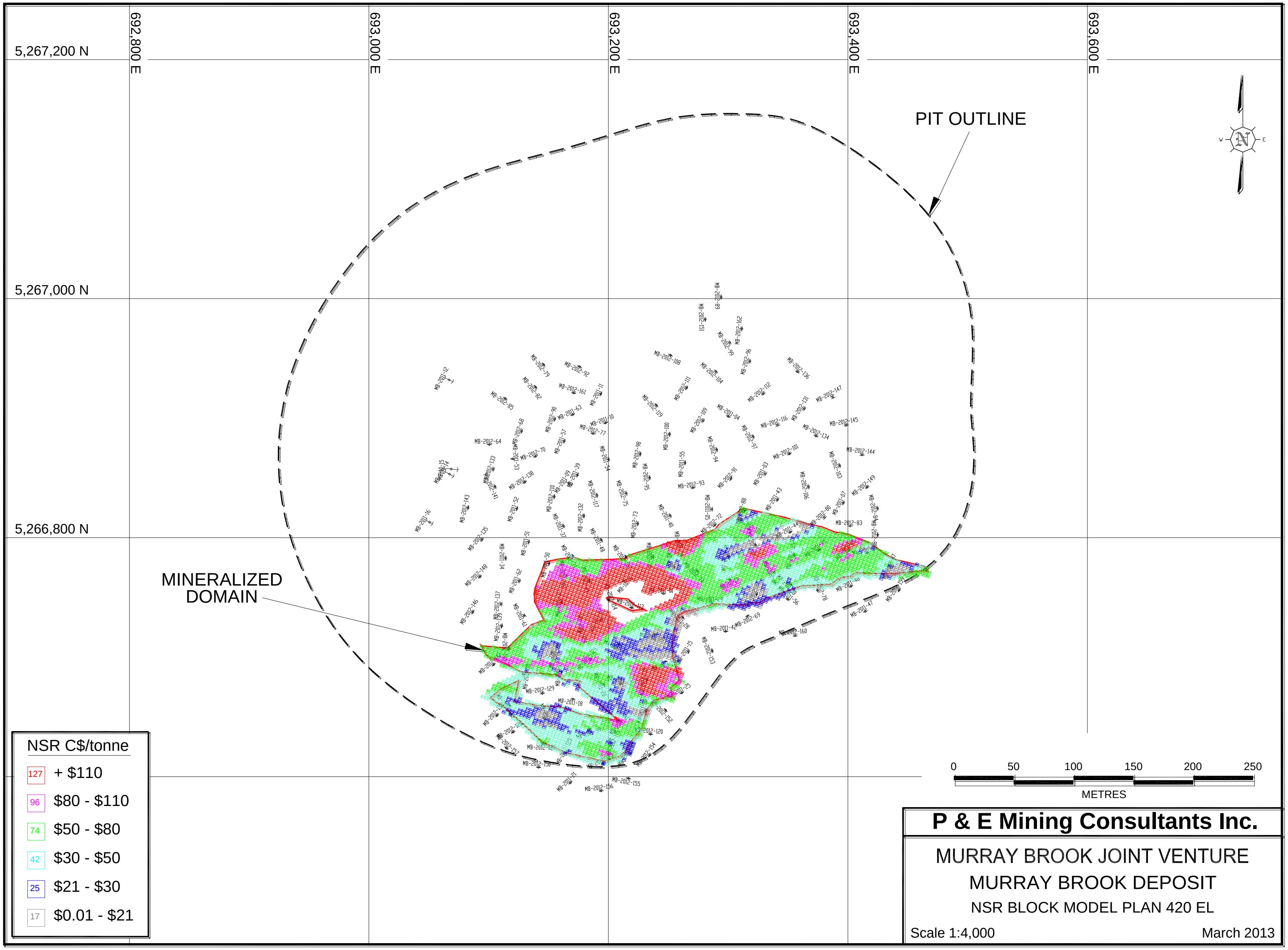
APPENDIX VI. NSR BLOCK MODEL CROSS SECTIONS AND PLANS


BASELINE
SURFACE
BEDROCK
500 EL
400 EL
300 EL
MINERALIZED DOMAIN
PIT OUTLINE
MB-2012-154
MB-2011-19
MB-2012-122
MB-2012-115
MB-2012-130
MB-2011-59
NSR C$/tonne
127 + $110
96 $80 - $110
74 $50 - $80
42 $30 - $50
25 $21 - $30
17 $0.01 - $21
0 50 100 150
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
NSR BLOCK MODEL SECTION 100 NE
Scale 1:2,500
March 2013

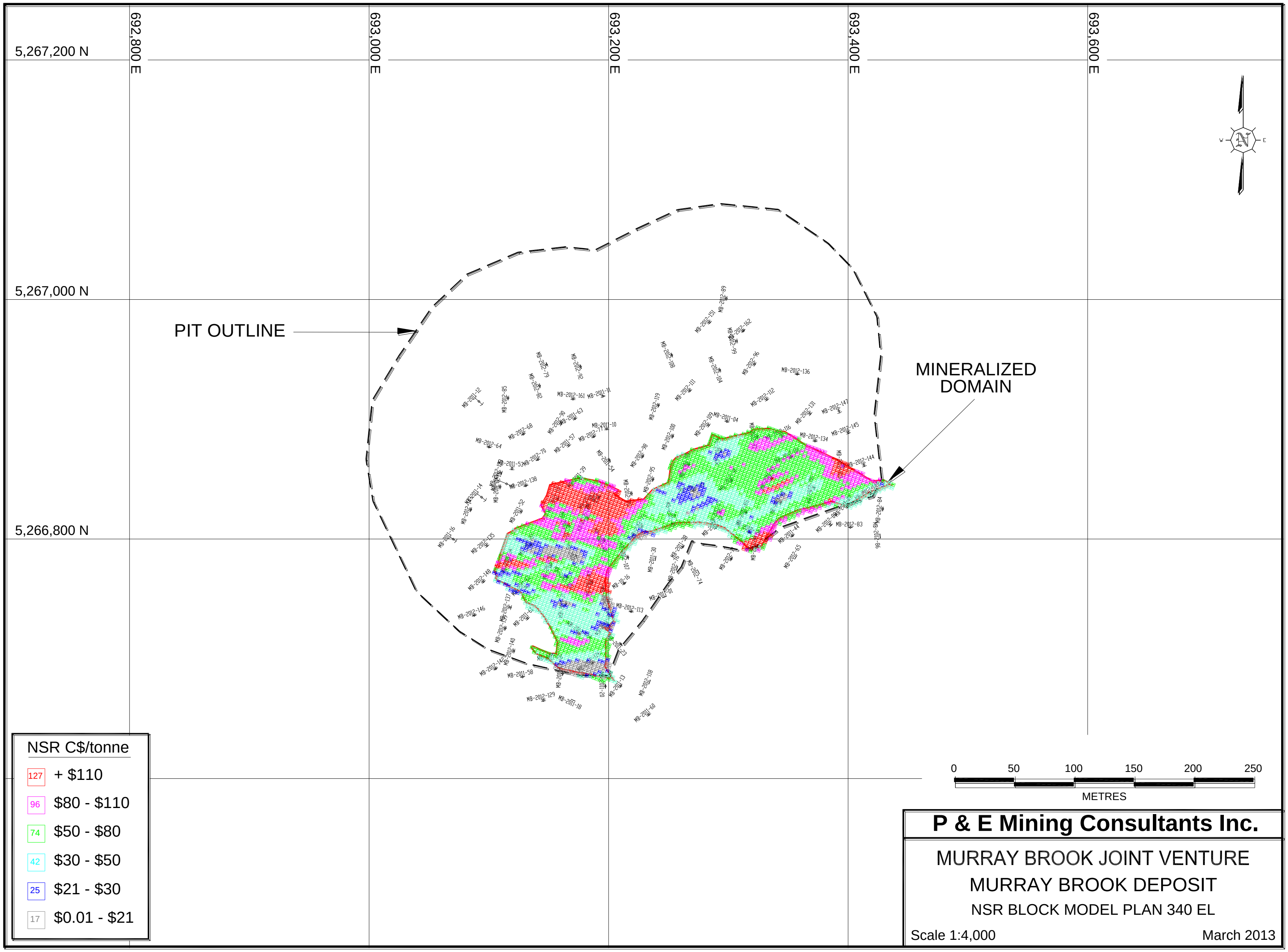

BASELINE
SURFACE
500 EL
400 EL
300 EL
BEDROCK
PIT OUTLINE
MINERALIZED DOMAIN
MB-2012-153
MB-2011-27
MB-2011-15
MB-2012-113
MB-2012-148
MB-2011-02
MB-2011-31
MB-2011-62
MB-2011-16
NSR C$/tonne
127 + $110
96 $80 - $110
74 $50 - $80
42 $30 - $50
25 $21 - $30
17 $0.01 - $21
0
50
100
150
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
NSR BLOCK MODEL SECTION 200 NE
Scale 1:2,500
March 2013



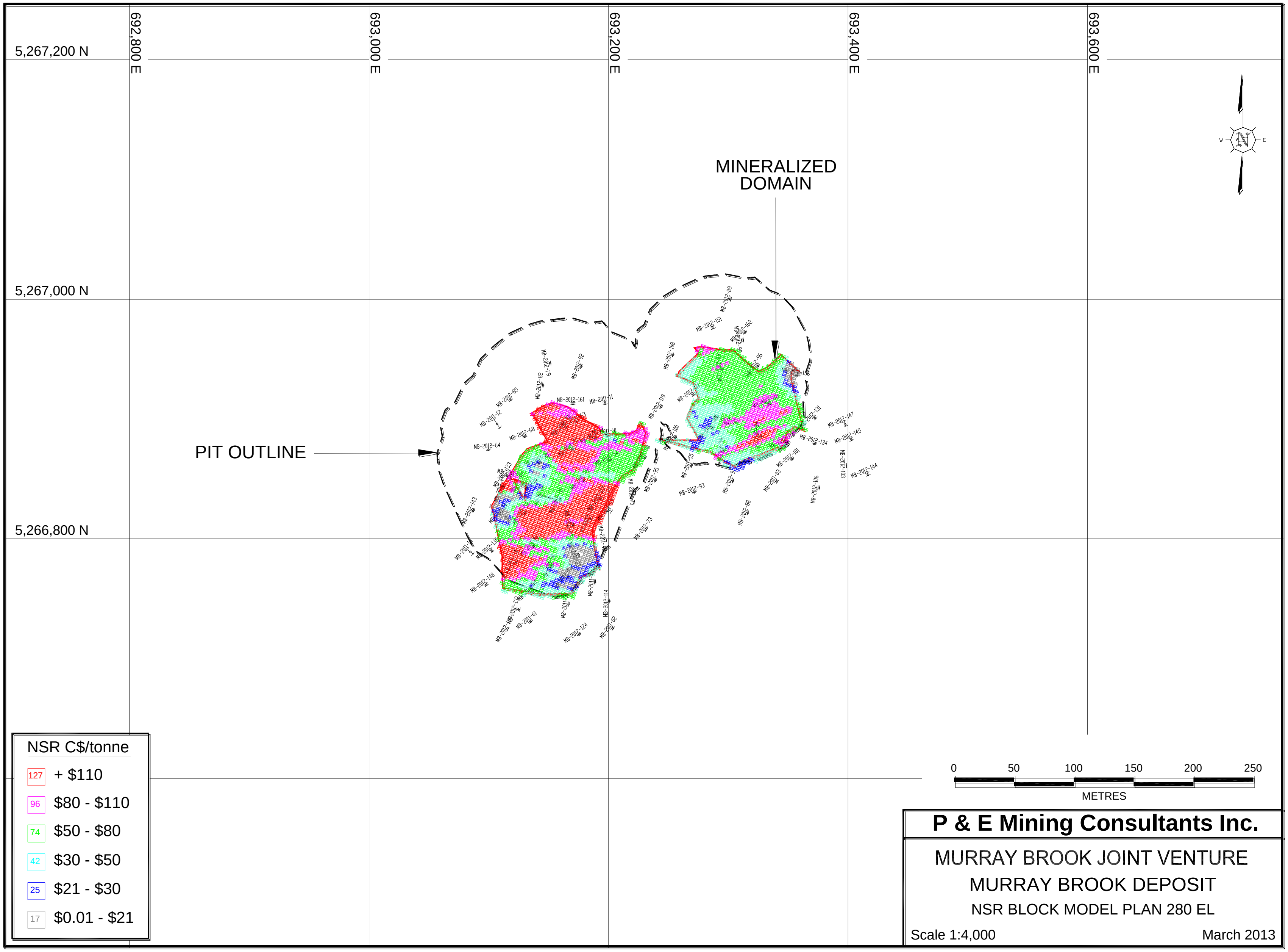
BASELINE
500 EL
400 EL
300 EL
SURFACE
BEDROCK
PIT OUTLINE
MINERALIZED
DOMAIN
MB-2011-09
MB-2011-53
MB-2011-46
MB-2012-159
MB-2012-65
MB-2011-40
MB-2011-41
MB-2012-72
MB-2012-121
MB-2012-75
MB-2011-39
MB-2012-64
MB-2011-12
MB-2012-70
NSR C$/tonne
127 + $110
96 $80 - $110
74 $50 - $80
42 $30 - $50
25 $21 - $30
17 $0.01 - $21
0
50
100
150
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
NSR BLOCK MODEL SECTION 300 NE
Scale 1:2,500
March 2013



5,267,200 N
5,267,000 N
5,266,800 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
PIT OUTLINE
MINERALIZED DOMAIN
NSR C$/tonne
127 + $110
96 $80 - $110
74 $50 - $80
42 $30 - $50
25 $21 - $30
17 $0.01 - $21
0 50 100 150 200 250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
NSR BLOCK MODEL PLAN 420 EL
Scale 1:4,000
March 2013

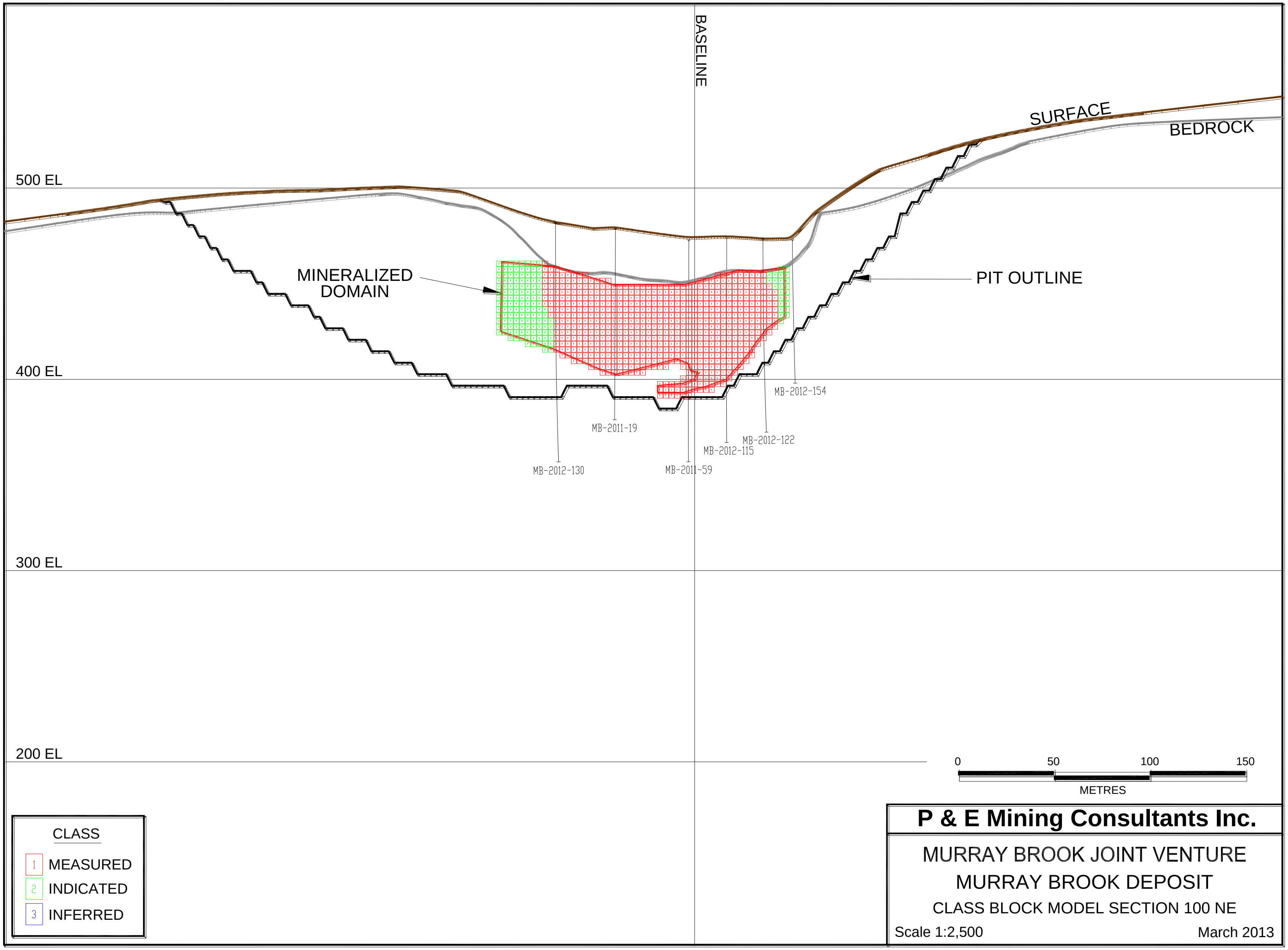


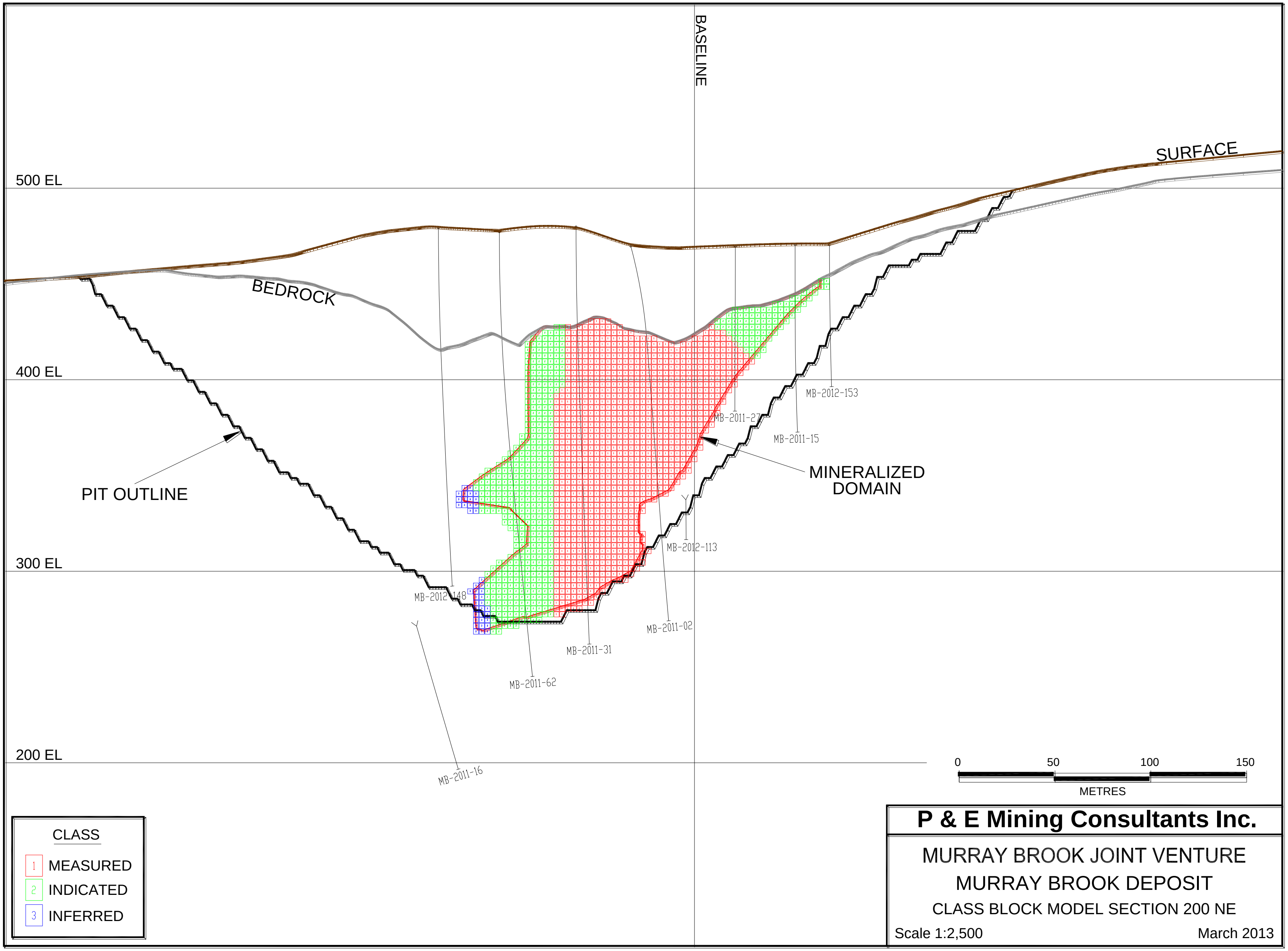
5,267,200 N
5,267,000 N
5,266,800 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
PIT OUTLINE
MINERALIZED DOMAIN
NSR C$/tonne
127 + $110
96 $80 - $110
74 $50 - $80
42 $30 - $50
25 $21 - $30
17 $0.01 - $21
0 50 100 150 200 250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
NSR BLOCK MODEL PLAN 340 EL
Scale 1:4,000
March 2013

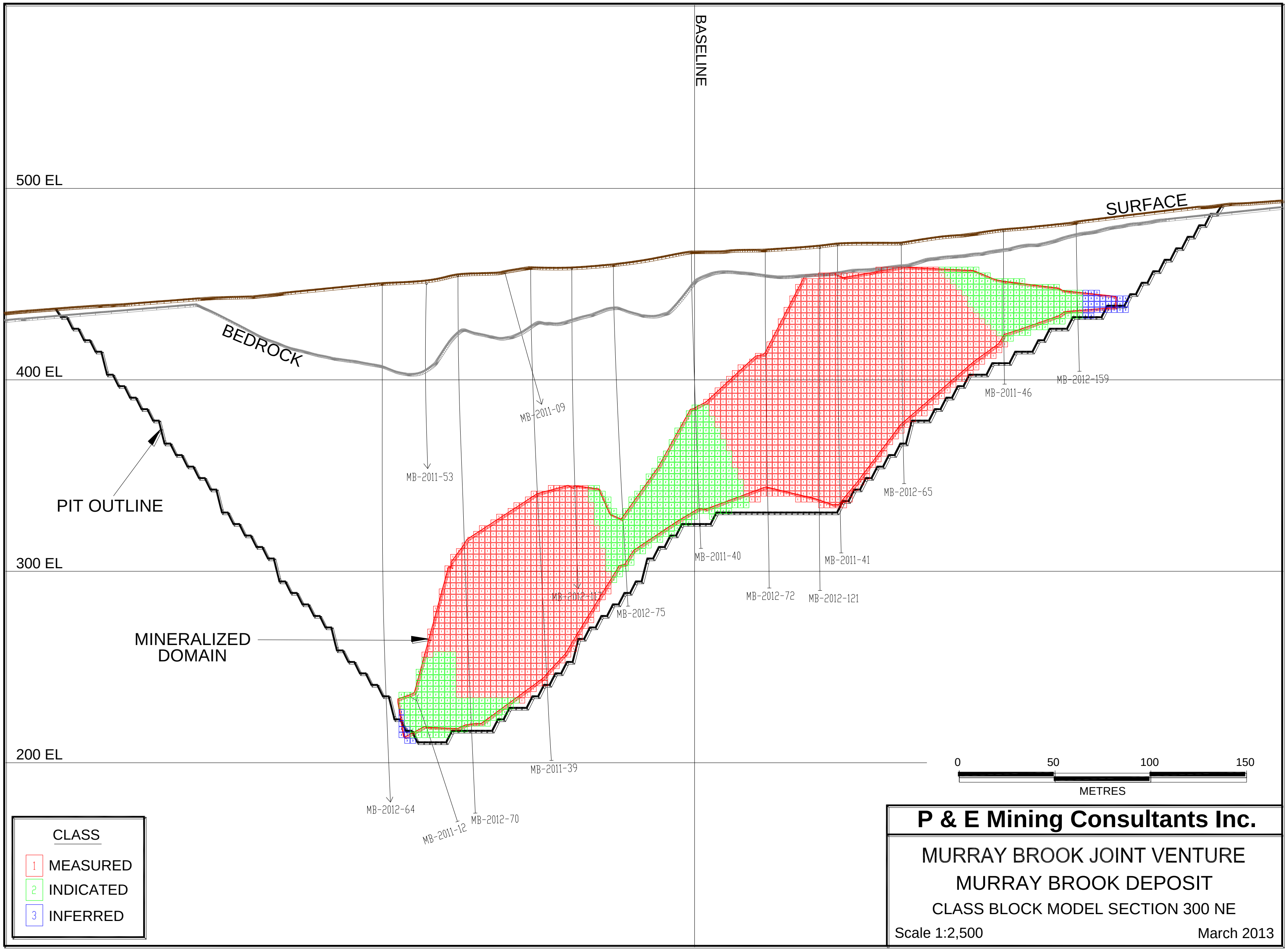


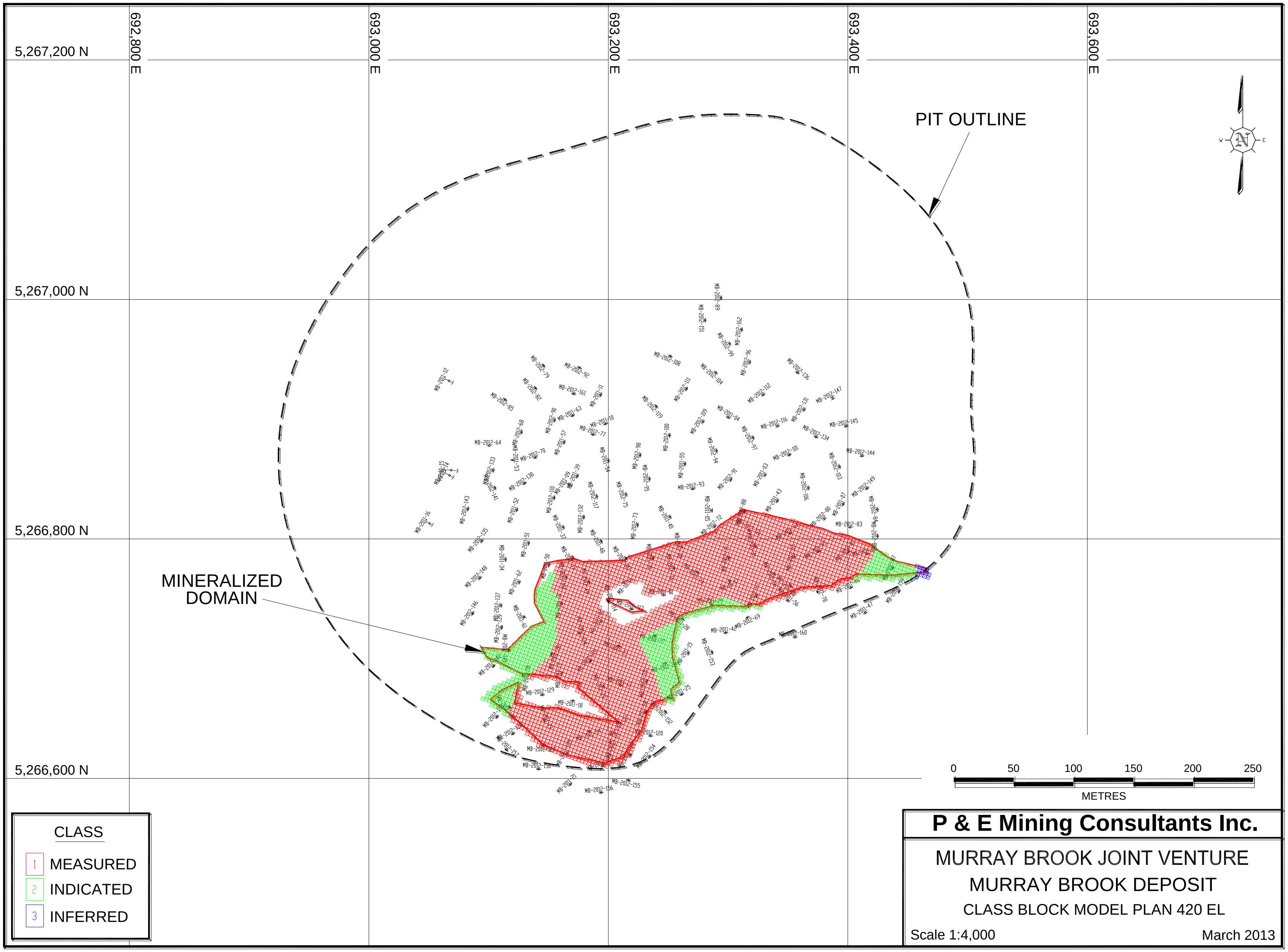
5,267,200 N
5,267,000 N
5,266,800 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
MINERALIZED DOMAIN
PIT OUTLINE
NSR C$/tonne
127 + $110
96 $80 - $110
74 $50 - $80
42 $30 - $50
25 $21 - $30
17 $0.01 - $21
0 50 100 150 200 250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
NSR BLOCK MODEL PLAN 280 EL
Scale 1:4,000
March 2013

APPENDIX VII. CLASSIFICATION BLOCK MODEL X-SECTIONS & PLANS

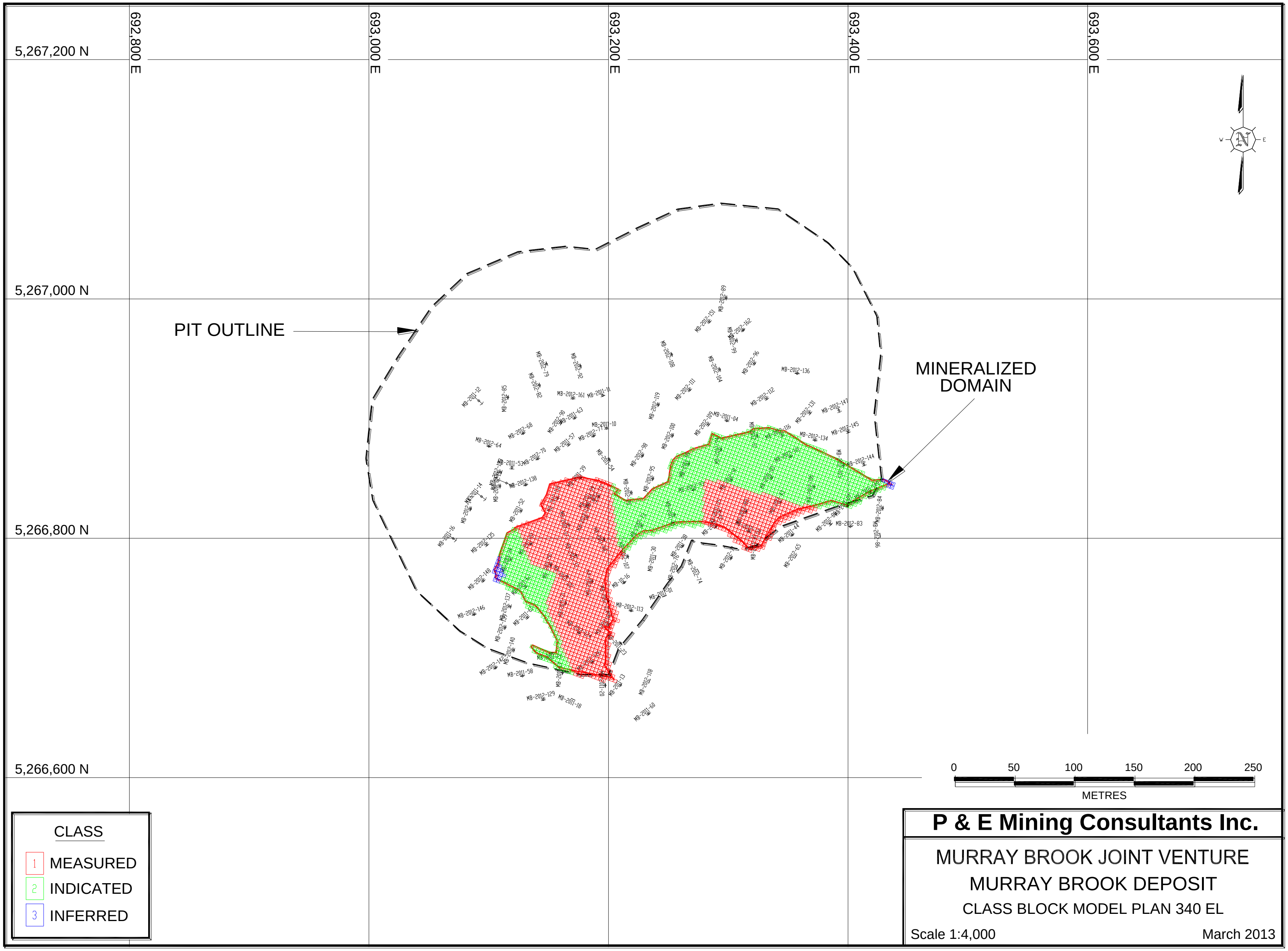

BASELINE
SURFACE
BEDROCK
500 EL
400 EL
300 EL
200 EL
MINERALIZED DOMAIN
PIT OUTLINE
MB-2012-154
MB-2011-19
MB-2012-122
MB-2012-115
MB-2012-130
MB-2011-59
0
50
100
150
METRES
CLASS
MEASURED
INDICATED
INFERRED
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
CLASS BLOCK MODEL SECTION 100 NE
Scale 1:2,500
March 2013


BASELINE
SURFACE
500 EL
BEDROCK
400 EL
300 EL
200 EL
PIT OUTLINE
MINERALIZED DOMAIN
MB-2012-153
MB-2011-27
MB-2011-15
MB-2012-113
MB-2012-148
MB-2011-02
MB-2011-31
MB-2011-62
MB-2011-16
0
50
100
150
METRES
CLASS
1 MEASURED
2 INDICATED
3 INFERRED
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
CLASS BLOCK MODEL SECTION 200 NE
Scale 1:2,500
March 2013

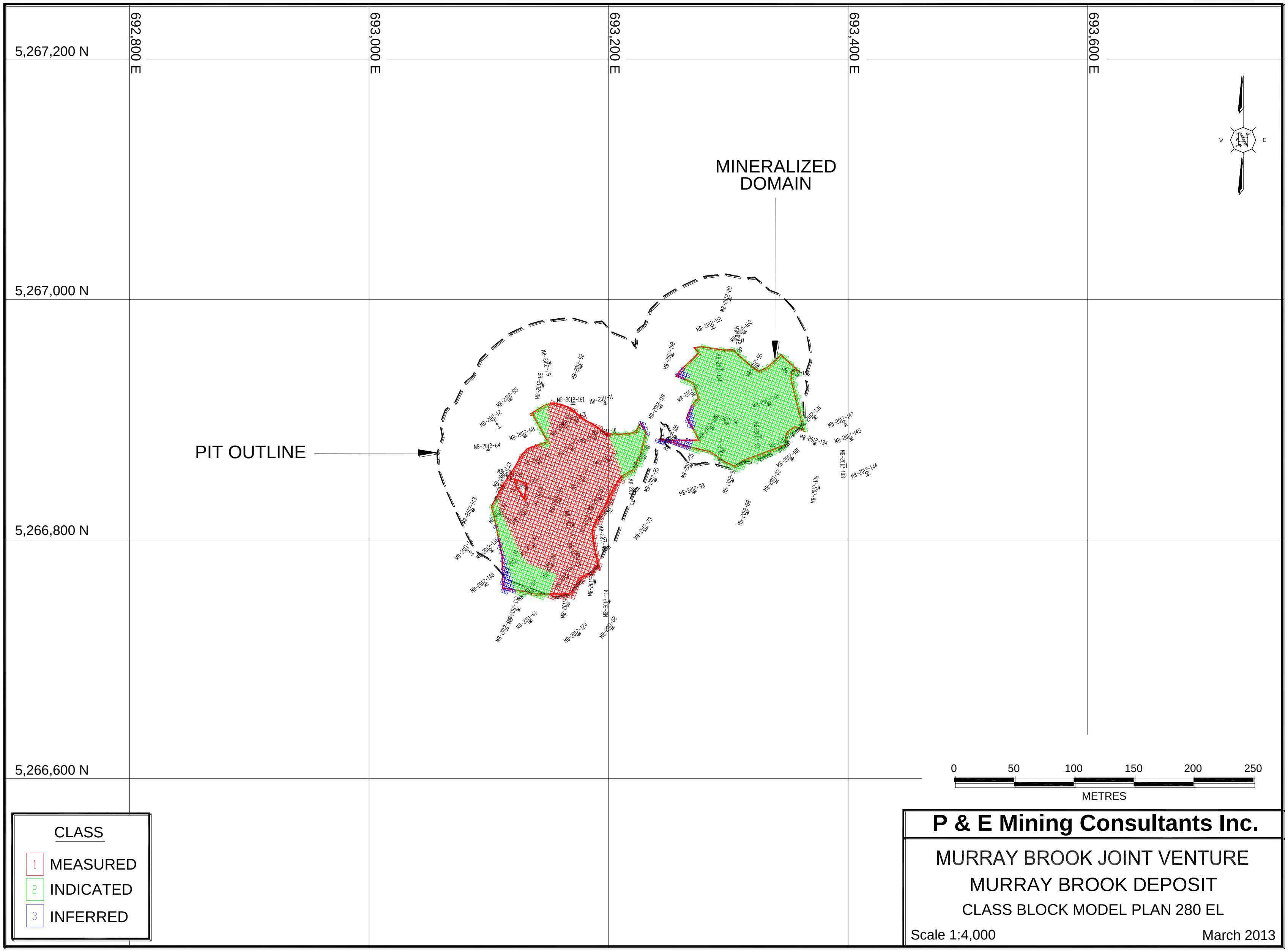

BASELINE
500 EL
400 EL
300 EL
200 EL
SURFACE
BEDROCK
PIT OUTLINE
MINERALIZED
DOMAIN
MB-2011-09
MB-2011-53
MB-2011-40
MB-2011-41
MB-2012-72
MB-2012-121
MB-2012-75
MB-2012-65
MB-2011-46
MB-2012-159
MB-2011-39
MB-2012-64
MB-2011-12
MB-2012-70
0
50
100
150
METRES
CLASS
1 MEASURED
2 INDICATED
3 INFERRED
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
CLASS BLOCK MODEL SECTION 300 NE
Scale 1:2,500
March 2013



5,267,200 N
5,267,000 N
5,266,800 N
5,266,600 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
PIT OUTLINE
MINERALIZED DOMAIN
CLASS
1 MEASURED
2 INDICATED
3 INFERRED
0 50 100 150 200 250
METRES
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
CLASS BLOCK MODEL PLAN 420 EL
Scale 1:4,000
March 2013



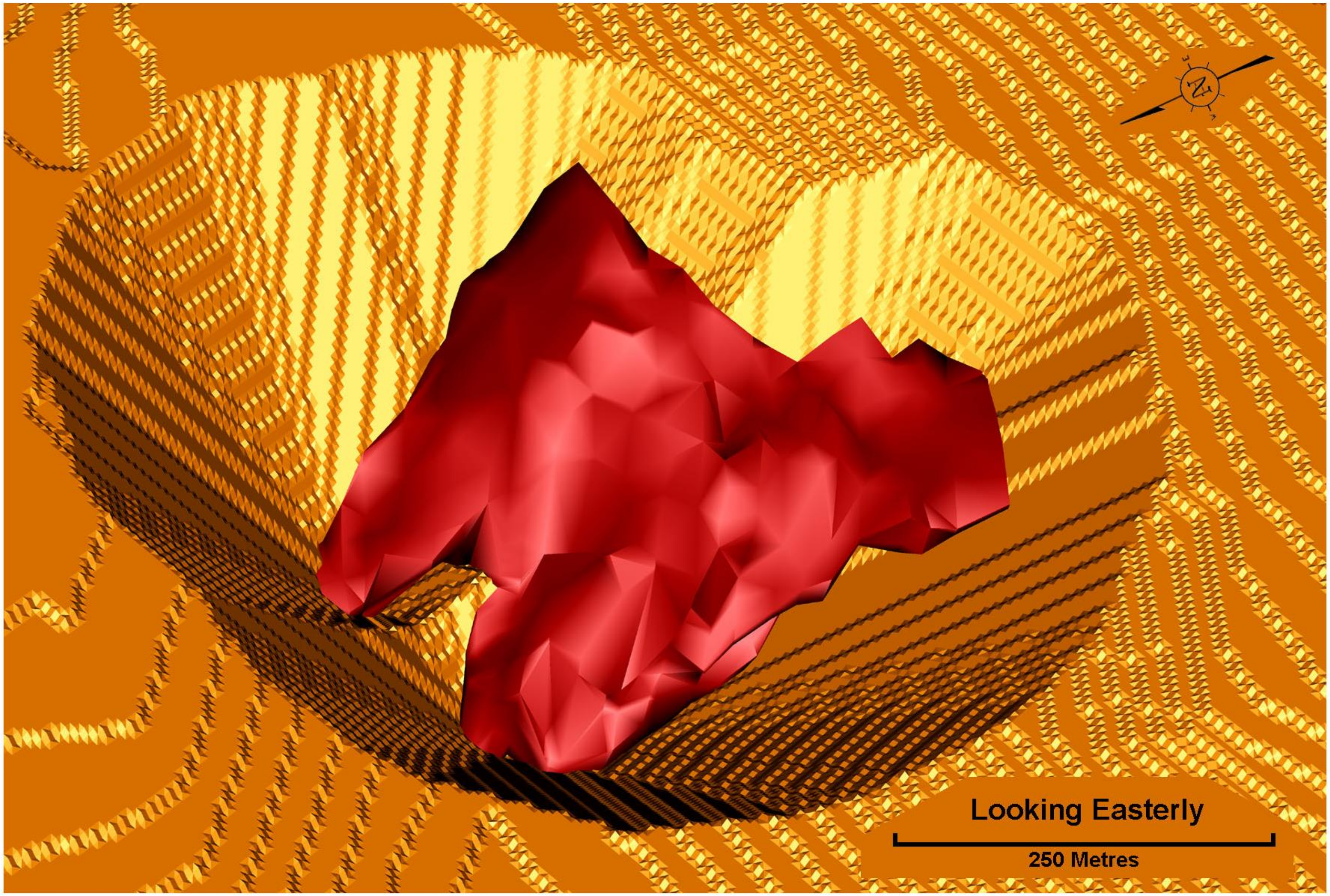
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
5,267,200 N
5,267,000 N
5,266,800 N
5,266,600 N
PIT OUTLINE
MINERALIZED DOMAIN
0 50 100 150 200 250
METRES
CLASS
1 MEASURED
2 INDICATED
3 INFERRED
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
CLASS BLOCK MODEL PLAN 340 EL
Scale 1:4,000
March 2013



5,267,200 N
5,267,000 N
5,266,800 N
5,266,600 N
692,800 E
693,000 E
693,200 E
693,400 E
693,600 E
MINERALIZED DOMAIN
PIT OUTLINE
0
50
100
150
200
250
METRES
CLASS
1 MEASURED
2 INDICATED
3 INFERRED
P & E Mining Consultants Inc.
MURRAY BROOK JOINT VENTURE
MURRAY BROOK DEPOSIT
CLASS BLOCK MODEL PLAN 280 EL
Scale 1:4,000
March 2013

APPENDIX VIII. OPTIMIZED PIT SHELL

MURRAY BROOK DEPOSIT - OPTIMIZED PIT SHELL



APPENDIX IX. PROJECTED CASH FLOWS

Murray Brook Joint Venture

Murray Brook Project

Projected Cash Flow Summary

19-Jul-13

				2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
	Units	Inputs	Totals	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Months Operating Target	months			-	-	9	12	12	12	12	12	12	12	12	10	12	12
Mill Processing Rate	t/day	92.0% availability				5,280	5,937	5,937	5,937	5,937	5,937	5,937	5,937	5,939	5,961	-	-
MINE PRODUCTION																	
		years=	11														
Overburden (t)	t		8,837,667		2,072,741	2,795,181	2,146,079	1,294,412	440,948	35,332	52,974	-	-	-	-		
Oxides Waste (t)	t		1,531,392		82,303	750,008	668,913	28,702	-	1,466	-	-	-	-	-		
Waste Rock (t)	t		71,516,093		2,720,504	2,249,572	6,893,975	8,160,627	9,197,719	9,650,117	9,565,725	9,584,351	9,720,962	3,211,549	560,992		
Total Waste (t)	t		81,885,152		4,875,548	5,794,761	9,708,967	9,483,741	9,638,667	9,686,915	9,618,699	9,584,351	9,720,962	3,211,549	560,992		
Strip Ratio	t		4.322		39.1	4.8	4.9	4.8	4.8	4.9	4.8	4.8	4.9	1.6	0.3		
Total Feed Sulphides (t)	t		18,947,674		124,796	1,204,951	1,993,691	1,993,632	1,993,641	1,993,563	1,993,818	1,993,637	1,993,729	1,994,262	1,667,954		
Total Material Mined	t		100,832,826	-	5,000,344	6,999,712	11,702,658	11,477,373	11,632,308	11,680,478	11,612,517	11,577,988	11,714,691	5,205,811	2,228,946	-	-
PROCESSING																	
		years=	10														
Mill Feed tonnage	tpy		18,947,600			1,329,700	1,993,700	1,993,600	1,993,600	1,993,600	1,993,800	1,993,600	1,993,700	1,994,300	1,668,000		
NSR (from formula)	$/t		$68.70			$55.99	$64.39	$63.26	$67.71	$58.46	$65.04	$67.79	$63.62	$77.55	$104.89		
Au (g/t)	g/t-dil		0.53			0.30	0.31	0.46	0.48	0.46	0.43	0.46	0.74	0.73	0.88		
Ag (g/t)	g/t-dil		37.7			19.0	31.5	35.1	37.5	32.4	37.7	38.6	40.4	43.8	57.8		
Cu (%)	%-dil		0.43			1.01	0.50	0.34	0.38	0.37	0.28	0.26	0.46	0.46	0.42		
Pb (%)	%-dil		0.89			0.30	0.69	0.82	0.90	0.76	0.88	0.95	0.90	1.06	1.51		
Zn (%)	%-dil		2.46			1.09	2.31	2.44	2.51	2.19	2.61	2.75	1.99	2.67	3.85		
Copper Concentrate																	
Mass Pull (calculated)						2.99%	1.46%	1.01%	1.13%	1.08%	0.81%	0.77%	1.34%	1.36%	1.24%		
Concentrate Tonnes (dry)			239,144	-	-	39,702	29,201	20,120	22,461	21,508	16,210	15,326	26,723	27,210	20,685	-	-
Concentrate tonnes (wet)	Moist=	8.0%	258,276	-	-	42,878	31,537	21,730	24,258	23,229	17,507	16,552	28,860	29,387	22,339	-	-
Recovery (Cu)		51.4%				51.4%	51.4%	51.4%	51.4%	51.4%	51.4%	51.4%	51.4%	51.4%	51.4%		
Recovery (Au)																	
Recovery (Ag)		12.5%				12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%		
Cu to concentrate	t		41,731	-	-	6,928	5,096	3,511	3,919	3,753	2,829	2,674	4,663	4,748	3,609	-	-
Grade (Cu)	%Cu	17.5%				17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%		
Grade (Au)	g/t			-	-	-	-	-	-	-	-	-	-	-	-	-	-
Grade (Ag)	g/t	373		-	-	79.4	268.8	435.2	416.0	375.9	579.1	627.3	377.1	401.0	582.9	-	-
Copper Metal Produced	M-lbs		92	-	-	15.3	11.2	7.7	8.6	8.3	6.2	5.9	10.3	10.5	8.0	-	-
Gold Metal Produced	oz		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver Metal Produced	oz		2,868,947	-	-	101,409	252,393	281,515	300,369	259,950	301,778	309,105	323,985	350,821	387,621	-	-
Lead Concentrate																	
Mass Pull (calculated)						0.22%	0.51%	0.60%	0.65%	0.55%	0.64%	0.69%	0.65%	0.77%	1.10%		
Concentrate Tonnes (dry)			122,337	-	-	2,941	10,075	11,883	13,032	11,022	12,802	13,788	13,049	15,393	18,351	-	-
Concentrate tonnes (wet)	Moist=	8.0%	132,124	-	-	3,177	10,881	12,833	14,075	11,904	13,826	14,891	14,093	16,625	19,819	-	-
Recovery (Pb)	Recovery =	36.6%				36.6%	36.6%	36.6%	36.6%	36.6%	36.6%	36.6%	36.6%	36.6%	36.6%		
Recovery (Ag)		17.5%				17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%	17.5%		
Pb to conc	t		61,536	-	-	1,480	5,068	5,977	6,555	5,544	6,439	6,936	6,564	7,743	9,231	-	-
Ag to conc	oz		4,016,526	-	-	141,973	353,350	394,121	420,517	363,930	422,489	432,748	453,579	491,149	542,670	-	-
Grade (Pb)	%Pb	50.3%				50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%		
Grade (Ag)	g/t	1021		-	-	1,501.3	1,090.9	1,031.7	1,003.6	1,027.0	1,026.5	976.2	1,081.2	992.4	919.8	-	-
Lead Metal Produced	t		61,536	-	-	1,480	5,068	5,977	6,555	5,544	6,439	6,936	6,564	7,743	9,231	-	-
Lead Metal Produced	M-lbs		135.66	-	-	3.26	11.17	13.18	14.45	12.22	14.20	15.29	14.47	17.07	20.35	-	-
Silver Metal Produced	oz		4,016,526	-	-	141,973	353,350	394,121	420,517	363,930	422,489	432,748	453,579	491,149	542,670	-	-

Note: *The financial analysis includes the payment of the 0.25% royalty at the start of production. In fact, the 0.25% Royalty payment commences on the first anniversary of production, or effectively one year later.*

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Zinc Concentrate																	
Mass Pull (calculated)						1.80%	3.81%	4.02%	4.14%	3.61%	4.30%	4.54%	3.29%	4.40%	6.35%		
Concentrate Tonnes (dry)			770,247	-	-	23,919	75,880	80,219	82,630	71,922	85,764	90,560	65,572	87,829	105,951	-	-
Concentrate tonnes (wet)	Moist=	8.0%	831,866	-	-	25,832	81,950	86,637	89,240	77,676	92,625	97,804	70,818	94,856	114,428	-	-
Recovery (Zn)	Recovery =	88.8%				88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%		
Recovery (Ag)		25.3%				25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%		
Zn to conc	t		414,393	-	-	12,868	40,823	43,158	44,455	38,694	46,141	48,721	35,278	47,252	57,002	-	-
Ag to conc	oz		5,806,749	-	-	205,252	510,843	569,786	607,948	526,138	610,798	625,629	655,746	710,062	784,546	-	-
Grade (Zn)	%Zn	53.8%				53.8%	53.8%	53.8%	53.8%	53.8%	53.8%	53.8%	53.8%	53.8%	53.8%		
Grade (Ag)	g/t	234		-	-	266.9	209.4	220.9	228.8	227.5	221.5	214.9	311.1	251.5	230.3	-	-
Zinc Produced	t		414,393	-	-	12,868	40,823	43,158	44,455	38,694	46,141	48,721	35,278	47,252	57,002	-	-
Zinc Produced	M-lbs		913.57	-	-	28.37	90.00	95.15	98.01	85.31	101.72	107.41	77.77	104.17	125.67	-	-
Silver Metal Produced	oz		5,806,749	-	-	205,252	510,843	569,786	607,948	526,138	610,798	625,629	655,746	710,062	784,546	-	-
REVENUE																	
Exchange =		$ 1.00															
Concentrate Values (per dmt)																	
Copper Concentrate	$US/lb	$ 3.70	$CAD / dmt	$ -	$ -	$ 1,182	$ 1,353	$ 1,503	$ 1,486	$ 1,450	$ 1,633	$ 1,677	$ 1,451	$ 1,472	$ 1,637	$ -	$ -
Lead Concentrate	$US/lb	$ 1.00	$CAD / dmt	$ -	$ -	$ 2,068	$ 1,704	$ 1,651	$ 1,626	$ 1,647	$ 1,646	$ 1,602	$ 1,695	$ 1,616	$ 1,552	$ -	$ -
Zinc Concentrate	$US/lb	$ 0.94	$CAD / dmt	$ -	$ -	$ 935	$ 887	$ 896	$ 903	$ 902	$ 897	$ 891	$ 972	$ 922	$ 904	$ -	$ -
Gold Dore	$US/oz	$ 1,627.00															
Silver	$US/oz	$ 30.09															
Metal Values Recovered (Theoretical)																	
Copper	C$(000)		340,398	$ -	$ -	$ 56,511.2	$ 41,564.9	$ 28,638.7	$ 31,970.5	$ 30,615.2	$ 23,073.2	$ 21,814.4	$ 38,036.8	$ 38,730.6	$ 29,442.4	$ -	$ -
Lead	C$(000)		135,661	$ -	$ -	$ 3,261.8	$ 11,172.6	$ 13,176.7	$ 14,451.7	$ 12,222.9	$ 14,195.8	$ 15,290.1	$ 14,470.1	$ 17,070.1	$ 20,349.7	$ -	$ -
Zinc	C$(000)		858,756	$ -	$ -	$ 26,667.4	$ 84,599.1	$ 89,437.1	$ 92,125.1	$ 80,186.8	$ 95,619.3	$ 100,965.7	$ 73,107.2	$ 97,921.8	$ 118,126.4	$ -	$ -
Gold	C$(000)		-	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Silver	C$(000)		261,052	$ -	$ -	$ 9,227.5	$ 22,965.7	$ 25,615.7	$ 27,331.3	$ 23,653.4	$ 27,459.4	$ 28,126.2	$ 29,480.1	$ 31,922.0	$ 35,270.5	$ -	$ -
Total Value (theoretical)	C$(000)		$ 1,595,867	$ -	$ -	$ 95,668	$ 160,302	$ 156,868	$ 165,879	$ 146,678	$ 160,348	$ 166,196	$ 155,094	$ 185,644	$ 203,189	$ -	$ -
Theoretical Value per tonne ore	$/t milled		$84.23			$71.95	$80.40	$78.69	$83.21	$73.57	$80.42	$83.36	$77.79	$93.09	$121.82		
Revenues																	
Copper Concentrate	C$(000)	28%	346,074	-	-	46,924.8	39,507.7	30,243.6	33,371.8	31,179.6	26,471.8	25,695.8	38,766.6	40,061.4	33,850.5	-	-
Lead Concentrate	C$(000)	16%	200,846	-	-	6,081.5	17,163.4	19,618.4	21,192.8	18,152.4	21,077.4	22,086.7	22,117.1	24,879.3	28,477.4	-	-
Zinc Concentrate	C$(000)	56%	699,106	-	-	22,360.0	67,275.3	71,897.9	74,607.4	64,860.3	76,910.2	80,706.6	63,725.7	80,967.5	95,795.3	-	-
Total NSR Revenue	C$(000)	100%	$ 1,246,026	$ -	$ -	$ 75,366	$ 123,946	$ 121,760	$ 129,172	$ 114,192	$ 124,459	$ 128,489	$ 124,609	$ 145,908	$ 158,123	$ -	$ -
NSR per tonne ore	$/t milled		$65.76			$56.68	$62.17	$61.08	$64.79	$57.28	$62.42	$64.45	$62.50	$73.16	$94.80		
Overall Payable Factor	%					78.8%	77.3%	77.6%	77.9%	77.9%	77.6%	77.3%	80.3%	78.6%	77.8%		
OPERATING COST	100%																
Mining Cost	$/t matl	$2.30	$231,779.7	$ -	$ 8,707.0	$ 16,090.1	$ 24,536.4	$ 25,047.1	$ 26,121.1	$ 26,697.8	$ 26,809.3	$ 27,117.1	$ 28,283.1	$ 16,099.0	$ 6,271.8	$ -	$ -
Processing Cost	$/t ore	$14.25	$270,003.3	$ -	$ -	$ 18,948.2	$ 28,410.2	$ 28,408.8	$ 28,408.8	$ 28,408.8	$ 28,411.7	$ 28,408.8	$ 28,410.2	$ 28,418.8	$ 23,769.0	$ -	$ -
G&A	$ M/yr	$2.50	$25,000.0	$ -	$ -	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ 2,500.0	$ -	$ -
Total operating Cost	C$(000)		$ 526,783	$ -	$ 8,707	$ 37,538	$ 55,447	$ 55,956	$ 57,030	$ 57,607	$ 57,721	$ 58,026	$ 59,193	$ 47,018	$ 32,541	$ -	$ -
Unit Operating	$/t ore		$27.80			$28.23	$27.81	$28.07	$28.61	$28.90	$28.95	$29.11	$29.69	$23.58	$19.51		
Unit Mining Cost	$/t ore		$12.23			$12.10	$12.31	$12.56	$13.10	$13.39	$13.45	$13.60	$14.19	$8.07	$3.76		
Unit Mining Cost	$/t material		$2.30		$1.74	$2.30	$2.10	$2.18	$2.25	$2.29	$2.31	$2.34	$2.41	$3.09	$2.81		
Operating Margin	C$(000)		$ 719,243			$ 37,828	$ 68,500	$ 65,804	$ 72,142	$ 56,586	$ 66,738	$ 70,463	$ 65,416	$ 98,890	$ 125,582	$ -	$ -

Note: *The financial analysis includes the payment of the 0.25% royalty at the start of production. In fact, the 0.25% Royalty payment commences on the first anniversary of production, or effectively one year later.*

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ROYALTIES																	
0.25% Royalty Payable	C$(000)	**0.25%**	3,115.1	-	-	188.4	309.9	304.4	322.9	285.5	311.1	321.2	311.5	364.8	395.3	-	-
Total Royalty	C$(000)		3,115.1	-	-	188.4	309.9	304.4	322.9	285.5	311.1	321.2	311.5	364.8	395.3	-	-
CAPITAL COSTS	100%	Contingency															
Mine Pre-Stripping	C$(000)	5.0%	$ 8,707	$ -	$ 8,707.0												
Mining Capital Cost	C$(000)	5.0%	$ 33,706	$ 33,705.9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	C$(000)	20.0%	$ 104,184	$ 52,092.0	$ 52,092.0												
Infrastructure	C$(000)	10.0%	$ 35,000		$ 35,000.0												
Indirects	C$(000)	20.0%	$ 44,000	$ 20,800.0	$ 23,200.0												
Contingency	C$(000)	15.6%	$ 35,257	$ 16,263.7	$ 18,993.7	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Initial Project Capital	C$(000)		$ 260,854	$ 122,861.5	$ 137,992.7	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustainining		Contingency															
Mine	C$(000)	5.0%	$ 10,137	$ -	$ -	$ 8,749.0	$ 139.0	$ 433.0	$ 139.0	$ 244.0	$ 433.0	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	C$(000)	20.0%	$ 2,100			$ 300.0	$ 300.0	$ 300.0	$ 300.0	$ 300.0	$ 300.0	300.0					
Enviro & Reclamation	C$(000)		$ 60,800			$ 9,050.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ -	$ 24,300.0
Contingency (sustaining)	C$(000)	15.6%	$ 927	$ -	$ -	$ 497.5	$ 67.0	$ 81.7	$ 67.0	$ 72.2	$ 81.7	$ 60.0	$ -	$ -	$ -	$ -	$ -
Total Sustaining Capital	C$(000)		$ 73,964	$ -	$ -	$ 18,596.5	$ 3,556.0	$ 3,864.7	$ 3,556.0	$ 3,666.2	$ 3,864.7	$ 3,410.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ -	$ 24,300.0
Total Capital	C$(000)		$ 334,818	$ 122,861.5	$ 137,992.7	$ 18,596.5	$ 3,556.0	$ 3,864.7	$ 3,556.0	$ 3,666.2	$ 3,864.7	$ 3,410.0	$ 3,050.0	$ 3,050.0	$ 3,050.0	$ -	$ 24,300.0
DEPRECIATION																	
Capital Additions	C$(000)		$ 334,818	$ 122,862	$ 137,993	$ 18,596	$ 3,556	$ 3,865	$ 3,556	$ 3,666	$ 3,865	$ 3,410	$ 3,050	$ 3,050	$ 3,050	$ -	$ 24,300
EBITDA	C$(000)		$ 727,950	$ -	$ -	$ 37,828	$ 68,500	$ 65,804	$ 72,142	$ 56,586	$ 66,738	$ 70,463	$ 65,416	$ 98,890	$ 125,582	$ -	$ -
Depreciation (max)	C$(000)	33.0%	$ 354,056	$ -	$ -	$ 86,082	$ 79,798	$ 58,469	$ 40,449	$ 28,274	$ 20,154	$ 14,778	$ 11,027	$ 8,394	$ 6,631	$ -	$ -
Depreciation (actual)	C$(000)		$ 294,006	$ -	$ -	$ 37,639	$ 68,190	$ 58,469	$ 40,449	$ 28,274	$ 20,154	$ 14,778	$ 11,027	$ 8,394	$ 6,631	$ -	$ -
Book Value Start of Yr	C$(000)		$ 1,228,783	$ -	$ 122,862	$ 260,854	$ 241,811	$ 177,177	$ 122,574	$ 85,680	$ 61,072	$ 44,783	$ 33,415	$ 25,438	$ 20,093	$ 16,512	$ 16,512
(+) Capital Additions	C$(000)		$ 334,818	$ 122,862	$ 137,993	$ 18,596	$ 3,556	$ 3,865	$ 3,556	$ 3,666	$ 3,865	$ 3,410	$ 3,050	$ 3,050	$ 3,050	$ -	$ 24,300
(-) Depreciation (Actual)	C$(000)		$ 294,006	$ -	$ -	$ 37,639	$ 68,190	$ 58,469	$ 40,449	$ 28,274	$ 20,154	$ 14,778	$ 11,027	$ 8,394	$ 6,631	$ -	$ -
= Book Value End of Yr	C$(000)			$ 122,862	$ 260,854	$ 241,811	$ 177,177	$ 122,574	$ 85,680	$ 61,072	$ 44,783	$ 33,415	$ 25,438	$ 20,093	$ 16,512	$ 16,512	$ 40,812
TAXES	100%																
Net Income	C$(000)		727,950	$ -	$ -	$ 37,827.9	$ 68,499.6	$ 65,804.0	$ 72,142.3	$ 56,585.7	$ 66,738.5	$ 70,463.2	$ 65,416.1	$ 98,890.4	$ 125,582.5	$ -	$ -
deduct Royalty	C$(000)		- 3,115	$ -	$ -	-$ 188.4	-$ 309.9	-$ 304.4	-$ 322.9	-$ 285.5	-$ 311.1	-$ 321.2	-$ 311.5	-$ 364.8	-$ 395.3	$ -	$ -
Depreciation	C$(000)		-$ 294,006	$ -	$ -	-$ 37,639.5	-$ 68,189.8	-$ 58,468.5	-$ 40,449.3	-$ 28,274.5	-$ 20,153.7	-$ 14,778.3	-$ 11,026.8	-$ 8,394.4	-$ 6,630.8	$ -	$ -
Other	C$(000)		$ -														
Taxable Income	C$(000)		$ 430,829	$ -	$ -	$ -	$ -	$ 7,031.1	$ 31,370.1	$ 28,025.7	$ 46,273.6	$ 55,363.6	$ 54,077.8	$ 90,131.2	$ 118,556.4	$ -	$ -
Federal Income Tax	C$(000)	15.0%	$ 53,897	$ -	$ -	$ -	$ -	$ 714.4	$ 4,015.1	$ 3,584.2	$ 5,826.3	$ 6,939.6	$ 6,772.1	$ 11,250.9	$ 14,794.8	$ -	$ -
New Brunswick Income Tax	C$(000)	10.0%	$ 35,931	$ -	$ -	$ -	$ -	$ 476.3	$ 2,676.7	$ 2,389.4	$ 3,884.2	$ 4,626.4	$ 4,514.7	$ 7,500.6	$ 9,863.2	$ -	$ -
NB Net Profit Mining Tax	C$(000)	16.0%	$ 51,914	$ -	$ -	$ -	$ -	$ -	$ 2,186.3	$ 2,014.1	$ 5,109.4	$ 6,696.4	$ 6,605.2	$ 12,374.0	$ 16,928.8	$ -	$ -
NB Net Revenue Tax	C$(000)	2.0%	$ 19,601			$ -	$ -	$ 2,268.5	$ 2,416.7	$ 2,117.2	$ 2,322.5	$ 2,403.1	$ 2,325.5	$ 2,751.5	$ 2,995.8	$ -	$ -
Deduction for NBNRT	C$(000)			$ -	$ -	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ 8,334.7	$ -	$ -
Total Tax Payable	C$(000)		$ 161,344	$ -	$ -	$ -	$ -	$ 3,459.1	$ 11,294.8	$ 10,104.9	$ 17,142.3	$ 20,665.5	$ 20,217.5	$ 33,876.9	$ 44,582.5	$ -	$ -
Effective Tax Rate	% of Tax Inc		37.4%					49.2%	36.0%	36.1%	37.0%	37.3%	37.4%	37.6%	37.6%		
Effective Tax Rate	% of Net Inc		22.2%					5.3%	15.7%	17.9%	25.7%	29.3%	30.9%	34.3%	35.5%		
CASH FLOW				-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Revenue from Concentrate	C$(000)		$ 1,246,026	$ -	$ -	$ 75,366.3	$ 123,946.3	$ 121,759.9	$ 129,172.1	$ 114,192.3	$ 124,459.4	$ 128,489.1	$ 124,609.4	$ 145,908.1	$ 158,123.2	$ -	$ -
Operating Cost	C$(000)		-$ 518,076			-$ 37,538.3	-$ 55,446.6	-$ 55,955.9	-$ 57,029.9	-$ 57,606.6	-$ 57,720.9	-$ 58,025.9	-$ 59,193.3	-$ 47,017.7	-$ 32,540.8	$ -	$ -
Working Capital	C$(000)	$ 6,300	$ -	$ -	-$ 6,300.0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 6,300.0	
Royalties	C$(000)		-$ 3,115	$ -	$ -	-$ 188.4	-$ 309.9	-$ 304.4	-$ 322.9	-$ 285.5	-$ 311.1	-$ 321.2	-$ 311.5	-$ 364.8	-$ 395.3	$ -	$ -
Taxes	C$(000)		-$ 161,344	$ -	$ -	$ -	$ -	-$ 3,459.1	-$ 11,294.8	-$ 10,104.9	-$ 17,142.3	-$ 20,665.5	-$ 20,217.5	-$ 33,876.9	-$ 44,582.5	$ -	$ -
Capital Spending	C$(000)		-$ 334,818	-$ 122,861.5	-$ 137,992.7	-$ 18,596.5	-$ 3,556.0	-$ 3,864.7	-$ 3,556.0	-$ 3,666.2	-$ 3,864.7	-$ 3,410.0	-$ 3,050.0	-$ 3,050.0	-$ 3,050.0	$ -	-$ 24,300.0
Annual Cash Flow	C$(000)	$ 43,344	$ 228,673	-$ 122,861.5	-$ 144,292.7	$ 19,043.0	$ 64,633.8	$ 58,175.8	$ 56,968.6	$ 42,529.1	$ 45,420.3	$ 46,066.4	$ 41,837.1	$ 61,598.7	$ 77,554.6	$ 6,300.0	-$ 24,300.0
Cumulative Cash Flow	C$(000)			-$ 122,861.5	-$ 267,154.3	-$ 248,111.2	-$ 183,477.4	-$ 125,301.6	-$ 68,333.0	-$ 25,803.9	$ 19,616.5	$ 65,682.9	$ 107,520.0	$ 169,118.8	$ 246,673.4	$ 252,973.4	$ 228,673.4
Disc Annual Cash Flow	C$(000)	7%		-$ 114,823.9	-$ 126,030.9	$ 15,544.8	$ 49,308.8	$ 41,478.6	$ 37,960.6	$ 26,485.0	$ 26,435.1	$ 25,057.1	$ 21,267.9	$ 29,265.1	$ 34,435.2	$ 2,614.3	-$ 9,424.0
Disc Cumulative Cash Flow	C$(000)			-$ 114,823.9	-$ 240,854.7	-$ 225,309.9	-$ 176,001.1	-$ 134,522.5	-$ 96,562.0	-$ 70,077.0	-$ 43,641.9	-$ 18,584.8	$ 2,683.1	$ 31,948.2	$ 66,383.4	$ 68,997.6	$ 59,573.7

Note: *The financial analysis includes the payment of the 0.25% royalty at the start of production. In fact, the 0.25% Royalty payment commences on the first anniversary of production, or effectively one year later.*