TECHNICAL REPORT
AND
PRELIMINARY ECONOMIC ASSESSMENT
OF THE
UPPER BEAVER GOLD-COPPER DEPOSIT
KIRKLAND LAKE, ONTARIO, CANADA

For

Queenston Mining Inc.

By

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TECHNICAL REPORT No. 239

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1.0 SUMMARY

The following Technical Report and Preliminary Economic Assessment ("PEA") of the gold / copper mineralization contained in the Upper Beaver Property ("Property") located in Gauthier township, North-Eastern Ontario, Canada, was prepared pursuant to National Instrument 43-101 ("NI 43-101") regulations and guidelines.

The Upper Beaver property is held 100% by Queenston Mining Inc ("Queenston").

This report was prepared by P&E Mining Consultants Inc. ("P&E") with contributions from Watts, Griffis and McOuat Limited, at the request of Mr. Charles E. Page, P. Geo., President and CEO of Queenston.

The Upper Beaver Property comprises 35 patented claims covering 572.556 ha and 3 leased claims (one lease) covering 53.584 ha with surface and mining rights and 9 unpatented mining claims (49 claim units – 784 ha) for a total of 1,410.14 ha. Queenston owns 100% interest in the Property and all 35 patented claims are in good standing in perpetuity.

Two of the Upper Beaver patented claims are subject to a NSR of 2% payable to Timmins Forest Products.

The Property is located in north-eastern Gauthier Township and north-western McVittie Township in the Larder Lake Mining Division in north-eastern Ontario. The Property can be operated on a year-round basis.

The Property is accessible from Highway 66 and Beaverhouse Road, which crosses Highway 66, 11 km west of the village of Larder Lake.

There are excellent local resources and infrastructure to support exploration and mining activities and mining equipment and personnel are readily available from the towns of Kirkland Lake, Matachewan, Ontario (approximately 50 kilometres west of Kirkland Lake) and Rouyn-Noranda, Quebec (approximately 60 kilometres east of the property).

There is low topographic relief on the Upper Canada Property, within the order of several metres and the terrain is characterized by relatively flat plateaus and glacial deposits, such as eskers and moraines.

All dollar amounts presented in this report are in Canadian dollars, unless otherwise stipulated.

1.1 MINERAL RESOURCES

The mineral resources ("Mineral Resources") referred to in this report have been taken from the mineral resource estimates contained in the Watts, Griffis and McOuat Limited ("WGM"), June 2011 Technical Report and Mineral Resource Estimate for the Upper Beaver Property, Ontario, (the "2011 WGM Report") prepared for Queenston by Kurt Breede, P.Eng., Richard W. Risto M.Sc., P.Geo.; and Michael W. Kociumbas, B.Sc., P.Geo. of WGM.

Gold was first discovered on the Property in 1912 and two shafts were sunk exploring a series of gold-copper veins. In 1919 Argonaut Gold Mines built a small mill with limited production occurring until 1928. Between 1935 and 1964 a variety of companies conducted exploration on the Property with no reported production. In 1964, Upper Canada Gold Mines Limited acquired the Property and resumed production, mining to a depth of 365 metres until closure in 1971. In 1977 Queenston Gold Mines Limited purchased the Kirkland Lake assets held by Upper Canada Resources Limited. Total production from the mine was 140,000 oz. of gold and 11.9 M lbs. of copper from 526,678 tonnes grading 8.3 g/t Au and 1% Cu. When the mine was closed in 1971 there remained underground resources of approximately 200,000 t grading 7.9 g/t Au with 1.2% Cu.

The majority of the Mineral Resources occur in a series of breccia zones that dip steeply north (75°) below the old mine workings. These zones contain chalcopyrite, magnetite, pyrite and visible gold within a mineralized corridor that extends over a horizontal length of approximately 500 metres and a dip length of approximately 1,300 m. The most prominent are the Porphyry Zones that contain approximately 80% of the Mineral Resource.

From 2005 to 2010 Queenston completed 201 drill holes outlining multiple gold-copper zones adjacent and below the Upper Beaver mine. Two NI 43-101 mineral resource studies were completed by WGM, one dated September 2008 and the second May 2011. During 2011 an additional 84 drill holes and wedge holes (41,019 m) were drilled on the Property both upgrading the inferred category towards indicated as well as extending the footprint of the deposit. A resource update is planned for the third quarter of 2012 and is expected to include all of the drilling conducted in calendar 2011 as well as the early part of 2012. This PEA is based on the 2011 Mineral Resource and does not take into account drill results after December 31, 2010.

There are three shafts on the Property. The No.3 Shaft, which surfaces on the west shore of York Lake, was the main production shaft for the previous underground operation. It extends to a depth of 605 feet (184 m), with an internal winze from the 500 to the 1250-ft level. Levels are established at 80, 200, 350 and 500 feet, and, at 125-foot intervals from the 500 level to 1,250 feet (381 m). The No.3 Shaft is capped.

P&E understands that there are no significant environmental liabilities on the Property and that Queenston is engaging with potentially affected aboriginal communities regarding the exploration and development of the project.

This PEA is based on P&E's evaluation of the NI 43-101 compliant Mineral Resource estimate for the Upper Beaver ("the Deposit") of 3,074,000 t @ 6.98 g/t Au or 690,000 oz. in the Indicated category and 3,093,000 tonnes @ 6.19 g/t Au or 616,000 oz. in the Inferred category. The Upper Beaver resource includes 36.6 M lbs Cu (0.54%) Indicated and 28 M lbs Cu (0.41%) Inferred. The Mineral Resource estimates were prepared by Kurt Breede, P.Eng. of Watts, Griffis and McOuat Limited ("WGM") an independent Qualified Person ("QP") as defined by NI 43-101 in the 2011 WGM Report dated June 15, 2011 and filed on SEDAR. Please refer to Table 1.1 for a summary of these Mineral Resources.

TABLE 1.1 SUMMARY OF UPPER BEAVER MINERAL RESOURCES(1-8)						
Resource Category	**Tonnes**	**Cu (%)**	**Au (g/t) (Uncapped)**	**Au (Ounces) (Uncapped)**	**Au (g/t) (Capped)**	**Au (Ounces Capped)**
Indicated	3,074,000	0.54	8.84	874,000	6.98	690,000
Inferred	3,093,000	0.41	7.15	711,000	6.19	616,000

(1) These Mineral Resources 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.

(2) Mineral Resources were estimated at a cut-off grade of 2.5 g/t Au and a minimum true width of 2.0 m

(3) Mineral Resources were estimated using a three-year rolling average gold price of US$1,050/oz., an exchange rate of US$0.95=CDN$1.00 and metallurgical recoveries of 95%.

*(4) *Individual assays were capped at 50 g/t Au.*

(5) A bulk density of 2.9 t/m³ was used.

(6) Kurt Breede, P.Eng. of Watts, Griffis and McOuat Limited (“WGM”) is the independent Qualified Person ("QP") under NI 43-101 who completed this Mineral Resource estimate.

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

(8) The quantity and grade of reported Inferred Mineral 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.

1.2 POTENTIALLY ECONOMIC PORTION OF THE MINERAL RESOURCES

A Potentially Economic portion of the Mineral Resources was estimated as a basis for this Preliminary Economic Assessment of the Deposit. The envisaged underground longhole mining method is estimated to experience mining dilution in the order of 20% at zero grade. Mine recovery (extraction) is estimated to be 95%. A summary of Potentially Economic Portion of the Mineral resources, including dilution and recovery, is presented in Table 1.2.

TABLE 1.2 SUMMARY OF UPPER BEAVER POTENTIALLY ECONOMIC PORTION OF THE RESOURCE ESTIMATE(1)(2)(3)				
Category	**Tonnes**	**Cu (%)**	**Au (g/t) (Capped)**	**Au (Ounces) (Capped)**
Potentially Economic Portion of the Indicated Resources	3,713,000	0.41	5.24	625,000
Potentially Economic Portion of the Inferred Resources	3,181,000	0.32	4.97	508,000

(1) Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. Environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues may materially affect the estimate of mineral resources. 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.

(2) The Potentially Economic portion of the Mineral Resource estimate was prepared by Eugene Puritch, P. Eng and James L. Pearson P.Eng. of P&E Mining Consultants Inc. Mineral Resource estimates reported in this press release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions Values have been rounded.

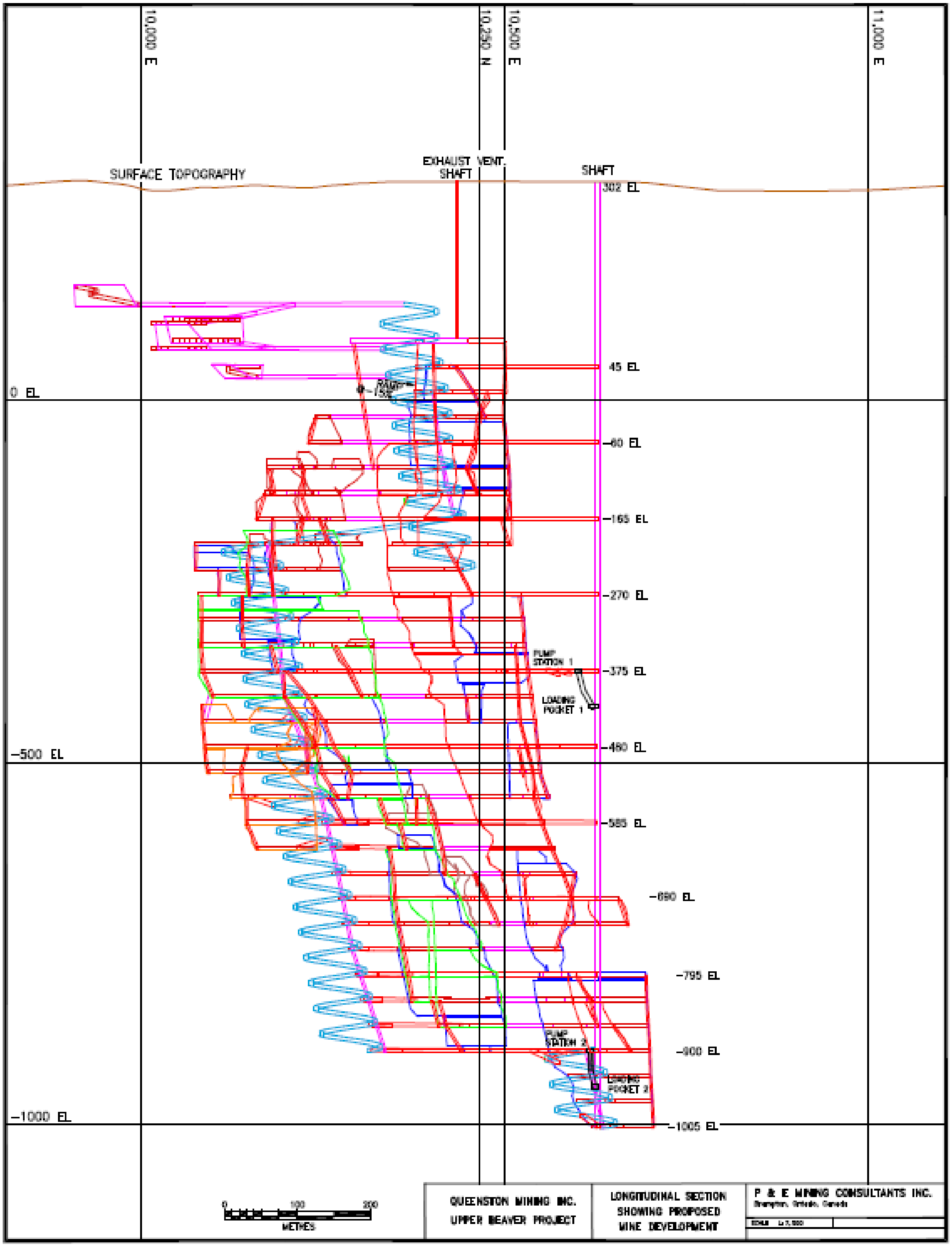
(3) Mine recovery and dilution are included in these quantities and average metal grades

The Potentially Economic Portion of the Mineral Resources contains Inferred Mineral Resources which have not been sufficiently drilled to confidently demonstrate economic viability. In addition, the work undertaken to date on the potential mining and milling operation at the Property ("the Project") is considered to be at conceptual levels of study only. As such, and according to the NI 43-101 Regulations, it is not possible to declare a mineral reserve of any kind.

1.3 CONCEPTUAL MINING AND PROCESSING PLAN

A conceptual mining and processing plan has been developed to assess the potential of economically extracting metals from the Deposit. This PEA envisages the development of an underground trackless mining operation with a steady state production rate of 2,000 tpd of mill feed. A longitudinal section of the proposed mine is shown in Figure 1.1.

Figure 1.1 Longitudinal Section



Access to the Deposit would be via a 6.5 metre diameter, concrete lined 1,300 m deep fresh air shaft. Two hoists would be configured to transport workers and materials between surface and the underground levels. A series of three internal declines would be located in the vicinity of the stoping operations. The primary mining method would be conventional longitudinal longhole retreat with paste backfill. Sub-levels would be developed at 35 metre vertical intervals. Drifts-in-ore would be developed to the full width of the Deposit. These drifts would provide access for the successive operations of slot raise development, blasthole drilling and blasting and backfill placement. Remotely operated underground load/haul/dump ("LHD") units would remove broken mineralization from the stope and from the excavated drifts-in-ore. The stopes would be backfilled primarily with cemented paste backfill, supplemented with waste rock. Stope mining would commence at the -375m and -900m loading pocket levels and proceed upwards through the mineralization.

It is estimated that 217 stopes would be mined over the mine life. This would generate an average of 2,000 tonnes per day ("tpd") composed of 1,749 stoping tonnes ("t") and 251t from the drift-in-ore and slot raise development.

Gold and copper mineralization would be processed in a 2,000 tpd expandable mill and paste backfill plant using conventional crushing, grinding, flotation and CIL processes. The current flow sheet does not include a gravity circuit as more testing is required to determine if this step is warranted, particularly as the design considerations would examine this mill as a future central facility for all of Queenston's Projects within the Kirkland Lake gold camp. P&E has included the capital cost of a gravity circuit in the process plant estimate. Metallurgical testwork completed by SGS Lakefield Research Limited indicates gold recovery of 98% and copper recovery of 90% using simple floatation and cyanidation. Payable gold and copper are estimated at 95% for gold and 90% for copper. Approximately, 80% of the gold is recovered in flotation with the balance being recovered from CIL. The projected gold-rich copper concentrate would be shipped to a smelter off site.

Ore extraction and processing commences in the third year following the commencement of project development with commercial production during the fourth year.

Power to the Property would be supplied by extending the existing 115-kV line 2 km to a substation then through a new 7 km long 44-kV transmission and communications line to the Property. Overall site power consumption during potential mining and milling operations is estimated to be approximately 15 MW.

Tailings generated by the processing of the mineralized rock from the mine, will disposed into the existing historical tailings management facility ("TMF"), approximately four kilometers from the site. Separate engineering and environmental studies are currently underway on this facility. The TMF design would incorporate features to manage the chemical and physical stability of the deposited tailings in accordance with existing and new practices. Approximately 45%-65% of the tailings would be deposited in the TMF. The remainder would be converted to paste backfill and deposited underground during the stoping operations

Major surface facilities to support the Upper Beaver mine would include an administration/engineering building, mineral process and paste backfill plant, warehouse, fuel storage, explosive storage, effluent treatment facility, fire protection and maintenance shop. While a construction camp for the project development phase is included in this study, it would not be required during operations.

1.4 ENVIRONMENTAL IMPACT AND REHABILITATION

Rehabilitation measures will be designed to ensure the long-term physical and chemical stability of the site in accordance with Ontario's closure plan approval process. The rehabilitation measures would return the site to a productive land use.

Environmental baseline studies to support the advanced exploration project permitting process and permit applications are underway. The terms of reference for the environmental assessment of the proposed producing mine and mill have yet to be established.

The current development plan envisions the expansion of the historic tailings impoundment site in order to support future mining. Testing to date indicates that the Upper Beaver mill tailings would be non-acid generating. The Project would be developed, operated and closed in accordance with environmental and health and safety regulatory requirements.

1.5 CAPITAL AND OPERATING COSTS

The estimated total capital costs for the Project is $418.1 million (see Table 1.3). This is composed of approximately $240.1 million in preproduction capital costs and $178.0 million in sustaining capital costs. Note that Canadian dollars are shown throughout this Technical report, unless otherwise described

TABLE 1.3
SUMMARY OF CAPITAL COSTS (LIFE OF MINE)

Description	Total ($M)
Mine & Stope Development	180.3
Shaft Development	50.0
Shaft Headframe, Hoist & Hoist Room, LP	14.5
Mine Equipment	18.3
U/G Infrastructure	5.3
Surface Infrastructure	28.7
Process Plant	68.5
Closure Bond & Salvage	-2.0
Contingency (15%)	54.5
Total Capital	**418.1**

***Note:** Some values have been rounded. The totals are accurate summations of the columns of data.*

The estimated total average operating cost of the mine is $73.06 per tonne of rock milled. This is composed of the components listed in Table 1.4.

TABLE 1.4 SUMMARY OF OPERATING COSTS	
Description	**Total ($/T Milled)**
Stope Mining	20.70
Paste Backfill	7.00
Tailings to Tailings Dam	1.00
Tailings Pond Water Treatment	0.32
Process Plant	17.81
U/G Haulage	3.50
U/ G Hoisting Services Costs	1.50
Mine Air Heating	3.05
G&A COSTS	6.00
Contingency (20%)	12.18
Total Operating	**73.06**

1.6 FINANCIAL EVALUATION

The Project was evaluated on an after-tax cash flow basis and it generates a net cash flow of $413.9 million. This results in an after-tax Internal Rate of Return (IRR) of 22.1% and an after-tax Net Present Value (NPV) of $233.4 million when using a 5% discount rate. In the base case scenario, the Project has a payback period of approximately 2.5 years from the start of commercial production. The gold and copper prices used in this PEA are US$1,275/oz. and US3.00/lb, respectively, and the US$/CAN$ exchange rate used in the PEA is 0.96. The average life-of-mine cash costs is US$415.99/oz. Au, net of copper credits, at an average operating cost of $73.06 per ore tonne processed.

This after-tax base case NPV is most sensitive to the Au metal price followed by the capital cost, operating costs and discount rate.

1.7 CONCLUSIONS AND RECOMMENDATIONS

P&E concludes that the Upper Beaver Project has economic potential as an underground mining and mineralized material processing operation producing gold doré and copper concentrate.

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

Specifically, it is recommended that Queenston take the following actions to develop the Project to a preliminary feasibility study level

- Complete detailed engineering and develop an exploration shaft which will provide access for bulk sampling and confirm the mineability/continuity of the deposit. This will include shaft sinking contractor selection and hoists procurement;
- Update current mineral resource by incorporating all new drilling that was not included in the 2011 WGM mineral resource;

- Complete the permitting procedure to procure an Advanced Exploration Permit for shaft sinking.
- Continue with baseline studies to support the environmental permitting process;
- Continue to engage the community and aboriginal groups in the project development. It is expected that Queenston will continue to work cooperatively with aboriginal communities to communicate the project's scope, impacts and benefits during the Advanced Exploration and Production stages;
- Carry out additional metallurgical testwork to improve metallurgical recoveries and process optimization. It is also recommended that tests on direct cyanidation of the mineralization be carried out.

Queenston should also continue with infill and step-out drilling for further exploration and mineral resource definition, as well as permitting and community matters. A proposed budget for this work in 2012 is provided in Table 1.5.

TABLE 1.5
PROPOSED BUDGET

Description	Cost
Drilling	$11,000,000
Environmental Work	$250,000
First Nation Consultation	$500,000
Metallurgical Testwork	$100,000
Resource Estimation	$100,000
Hydrogeology Study	$100,000
Archaeological Study	$75,000
Advance Exploration Closure Report	$250,000
Geotechnical and Condemnation Drilling	$1,000,000
Housing and Accommodation	$500,000
Site Preparation	$500,000
Total	**$14,375,000**

P&E also recommends that the Mineral Resource estimates be updated to incorporate any additional information that has become available since the WGM (2011) Report, including any material results from exploration and diamond drilling work that has been underway during and prior to this period.

2.0 INTRODUCTION

2.1 TERMS OF REFERENCE

The following report was prepared pursuant to the NI 43-101 regulations and guidelines in order to provide a Technical Report and Preliminary Economic Assessment of the gold and copper mineralization contained in the Upper Beaver Copper Property ("the Property"), a part of the Kirkland Lake Gold Project, north-eastern Ontario, Canada. The Upper Beaver Property is held 100% by Queenston Mining Inc.

This report was prepared by P&E Mining Consultants Inc. ("P&E"), with contributions from Watts, Griffis and McOuat Limited, at the request of Mr. Charles E. Page, P. Geo., President and CEO of Queenston.

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This report has an effective date of February 16, 2012.

Mr. Kurt Breede, P.Eng., of Watts, Griffis and McOuat Limited, a Qualified Person ("QP") under the regulations of NI 43-101, conducted a site visit and independent assay verification sampling program at the Property on March 30, 2011. Mr. Gene Puritch, P.Eng. and Mr. James Pearson, P.Eng. of P&E visited the property October 5, 2011 to examine engineering aspects of the project.

In addition to the site visit, P&E has held discussions with technical personnel from the Company regarding all pertinent aspects of the project and carried out a review of all available literature and documented results concerning the Property. The reader is referred to those data sources, which are outlined in the References, Section 27.0 of this report, for further detail.

The present Technical Report is prepared in accordance with the requirements of NI 43-101F1 of the Ontario Securities Commission ("OSC") and the Canadian Securities Administrators ("CSA").

The Mineral Resources in the estimate are considered compliant with 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.

The purpose of the current report is to provide an independent, NI 43-101 compliant, Technical Report and Preliminary Economic Assessment ("PEA") of the Upper Beaver Property. P&E understands that this report will be used for internal decision making purposes and may be filed as required under regulatory filing requirements. The report may also be used to support public equity financings.

2.2 SOURCES OF INFORMATION

This report is based, in part, on internal company technical reports, maps and technical correspondence, published government reports, press releases and public information as listed in the References, Section 27, at the conclusion of this report. Several sections from reports authored by other consultants have been directly quoted or summarized in this report, and are so indicated where appropriate.

With regard to certain sections of the current report the authors have drawn heavily upon selected portions or excerpts from material contained in a NI 43-101 Technical Report prepared by WGM as noted below:

Watts, Griffis and McOuat Limited, June 2011 Technical Report and Mineral Resource Estimate for the Upper Beaver Property, Ontario for (“the 2011 WGM Report”) Queenston Mining Inc.

2.3 UNITS AND CURRENCY

Unless otherwise stated all units used in this report are metric. Gold values are reported in grams per tonne (“g Au/t”) and copper values are reported as a percentage unless some other unit is specifically stated. The CDN$ is used throughout this report unless otherwise specifically stated. The US$/CAN$ exchange rate used in the PEA is 0.96.

2.4 GLOSSARY AND ABBREVIATION OF TERMS

In this document, the following terms have the meanings set forth below unless the context otherwise requires.

“$” and “CD$”	means the currency of Canada
“AAS”	means Atomic Absorption Spectroscopy
“AA”	is an acronym for Atomic Absorption, a technique used to measure metal content subsequent to fire assay
“asl”	means above sea level
“Au”	means gold
“C”	means degrees Celsius
“CIM”	means the Canadian Institute of Mining, Metallurgy and Petroleum
“cm”	means centimetres
“Cu”	means Copper
“CSA”	means the Canadian Securities Administrators
“Dicom”	means Dicom Express Inc.
“DMT”	means dry metric tonne
“E”	means east
“el”	means elevation level
“Franco-Nevada”	means the Franco-Nevada Mining Corporation
“Ga”	means gigayear, a unit of a billion years
“g Au/t”	means grams of gold per tonne
“ha”	means Hectare
“IETS”	Inco Exploration and Technical Services Inc.

"Inco"	means Inco Ltd.
"KLGP"	means the Kirkland Lake Gold Project
"KLGC"	means the Kirkland Lake Gold Camp
"KLMB"	means the Kirkland Lake Main Break
"km"	means kilometre
"lbs Cu/t"	means pounds of copper per tonne
"LLB"	means Larder Lake Break
"m"	means metre
"M"	means million
"Ma"	means millions of years
"mm"	means millimetres
"MNDM"	means Ministry of Northern Development and Mines
"Mt"	means millions of tonnes
"N"	means north
"NE"	means northeast
"NI 43-101"	means National Instrument 43-101
"NPI"	means Net Profit Interests
"NTS"	means National Topographic System
"NW"	means northwest
"NSR"	means an acronym for net smelter return, which means the amount actually paid to the mine or mill owner from the sale of ore, minerals and other materials or concentrates mined and removed from mineral properties, after deducting certain expenditures as defined in the underlying smelting agreements
"OGS"	means Ontario Geological Survey
"oz. Au/T"	means ounces per short ton
"P&E"	means P&E Mining Consultants Inc.
"oz. Au/t"	means ounces of gold per tonne
"PEA"	means a Preliminary Economic Assessment
"Property"	means the Upper Beaver Property
"ppb"	means parts per billion
"ppm"	means parts per million
"Queenston"	means Queenston Mining Inc.
"S"	means south
"SE"	means southeast
"SEDAR"	means the System for Electronic Document Analysis and Retrieval
"SGS"	means SGS Laboratories Ltd., in Rouyn-Noranda, Quebec
"SW"	means southwest
"Swastika Lab"	means Swastika Laboratories Ltd., in Swastika, Ontario
"t"	means tonnes (metric measurement)
"t/a"	means tonnes per year
"TN"	means True North
"tpd"	means tonnes per day
"TSX-V"	means the TSX Venture Exchange
"US$"	means the currency of the United States
"UTM"	means Universal Transverse Mercator
"WGM"	means Watts, Griffis and McOuat Limited
"W"	means west

3.0 RELIANCE ON OTHER EXPERTS

The authors of this report have assumed, and relied on the fact, that all the information and existing technical documents listed in the References section of this report are accurate and complete in all material aspects. While all the available information presented was carefully reviewed, its accuracy and completeness cannot be guaranteed. The authors reserve the right, but will not be obligated to revise their report and conclusions if additional information becomes known subsequent to the date of this report.

Copies of the tenure documents, operating licenses, permits, and work contracts were not reviewed but an independent verification of claim title was performed using the MNDM's CLAIMaps web application. It should be noted that patented claims cannot be verified in this manner. P&E has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has relied on, and believes it has a reasonable basis to rely upon, William McGuinty, Vice President Exploration for Queenston, to have conducted the proper legal due diligence.

A draft copy of the report has been reviewed for factual errors by the clients and P&E has relied on Queenston's knowledge of the Property in this regard. All statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The Property is located in north-eastern Gauthier Township and north-western McVittie Township in the Larder Lake Mining Division in north-eastern Ontario (Figure 4.1). The claim group lies 8 km northwest of the village of Larder Lake and is approximately 25 km from Kirkland Lake. The geographic centre of the Upper Beaver Property is approximately 48^{o} 10' 21" north latitude and 79^{o} 45' 20" west longitude. The property is located approximately 500 km north of Toronto, 200 km north-northeast of Sudbury and 120 km southeast of Timmins. The Property co-ordinates used in this report are located relative to the NAD 82 UTM coordinate system.

4.2 PROPERTY DESCRIPTION AND OWNERSHIP

The Upper Beaver Property is contiguous with 5 other Queenston properties; the Gauthier Property and Upper Canada Property to the west, the Mary Ann Property to the south and the Fort Lake Property to the southwest as well as the Lac McVittie property to the east which is a joint venture between Queenston, Barrick and Contact Diamond Corp.

The Property consists of 35 patented claims covering 572.6 ha and 3 leased claims (one lease) covering 53.6 ha with surface and mining rights and 9 unpatented mining claims (49 claim units – 784 ha) for a total 1,410.1 ha as listed in Table 4.1 and shown on Figure 4.2.

Each patented claim would have had a legal land survey when it was registered, however, P&E has not seen these surveys. The Ontario Mining Act requires that unpatented claims must be surveyed by a licensed Ontario surveyor before a lease can be granted. All unpatented claims were verified using the MNDM's CLAIMaps web application. Patented claims cannot be verified in this manner. All survey documents for the Upper Beaver Property leased and patented claims are registered and filed at the Ontario Land Registry Office located in Haileybury, Ontario.

The unpatented mining claims have not had a legal land survey.

The Property is owned 100% by Queenston with certain claims subject to royalties and interests to other parties (see Property Agreements).

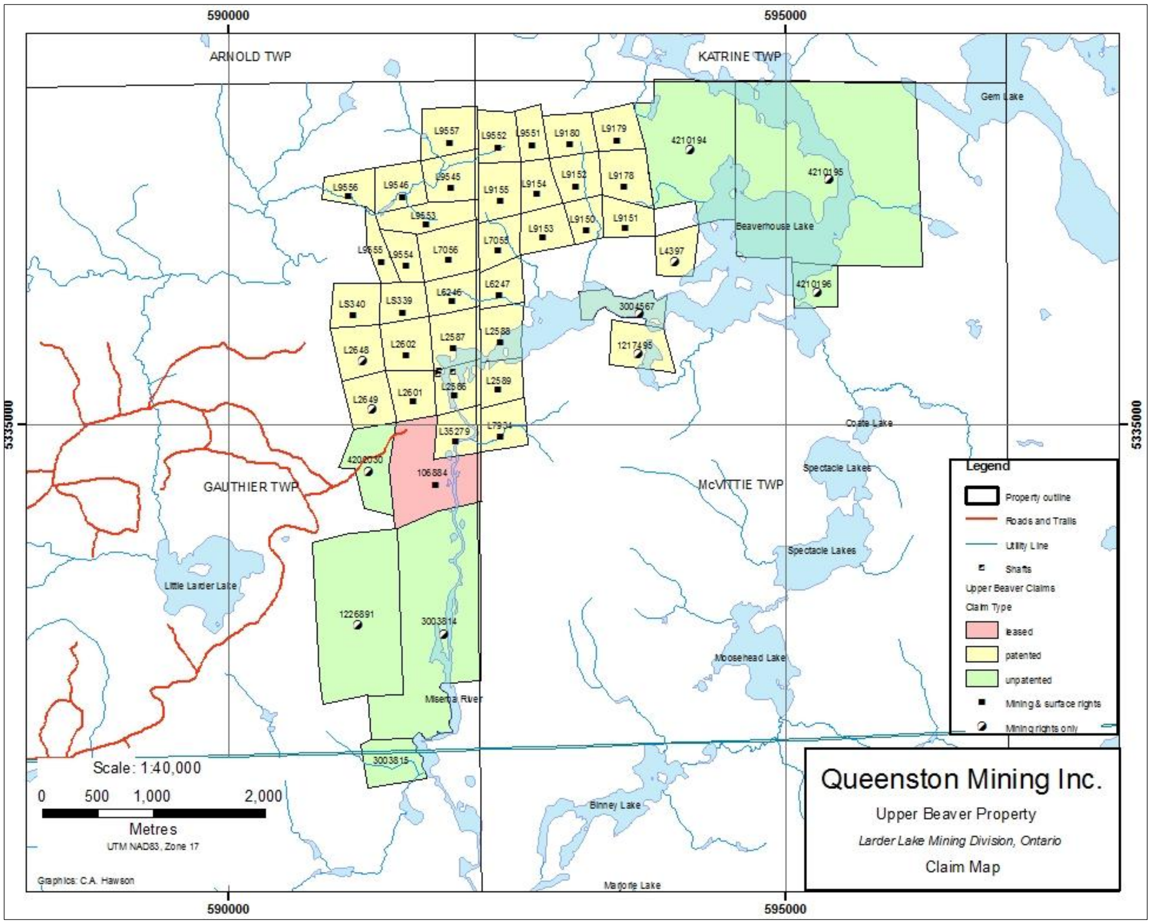
Queenston pays a land tax to maintain the patented claims in good standing. The 21-year Lease, 106884, covering three claims requires annual rental payments. To maintain unpatented claims in good standing, approved exploration work of required dollar value must be completed and filed with MNDM. As prescribed by the Ontario Mining Act and Regulations, work to a value of $400 per year is required per claim except for the first year, when no assessment work is required. Assessment work must be performed and applied to each of the mining claims until the holder applies for a Mining Lease. The earliest due date for Queenston's mining claims is August 1, 2013 (see Table 4.1). P&E understands that Queenston has abundant excess credits from its exploration programs to renew the claims when they become due.

Table 4.1 **Upper Beaver Property Claims and Leases**							
Township	**Claim Number**	**Claim Type**	**Due Date**	**Rights**	**Units**	**Area (ha)**	**Royalty**
McVittie	L9551	patented	Annual tax	M&SR	1	10.927	
McVittie	L9552	patented	Annual tax	M&SR	1	17.159	
Gauthier	L9553	patented	Annual tax	M&SR	1	19.83	
Gauthier	L9554	patented	Annual tax	M&SR	1	12.141	
Gauthier	L9555	patented	Annual tax	M&SR	1	8.903	
Gauthier	L9556	patented	Annual tax	M&SR	1	14.204	
Gauthier	L9557	patented	Annual tax	M&SR	1	20.639	
McVittie	L9150	patented	Annual tax	M&SR	1	11.938	
McVittie	L9151	patented	Annual tax	M&SR	1	14.366	
McVittie	L9152	patented	Annual tax	M&SR	1	16.106	
McVittie	L9153	patented	Annual tax	M&SR	1	16.187	
McVittie	L9154	patented	Annual tax	M&SR	1	13.152	
McVittie	L9155	patented	Annual tax	M&SR	1	18.616	
McVittie	L9178	patented	Annual tax	M&SR	1	21.448	
McVittie	L9179	patented	Annual tax	M&SR	1	15.985	
McVittie	L9180	patented	Annual tax	M&SR	1	15.783	
Gauthier	L9545	patented	Annual tax	M&SR	1	19.587	
Gauthier	L9546	patented	Annual tax	M&SR	1	17.887	
Gauthier	L2601	patented	Annual tax	M&SR	1	17.604	
Gauthier	L2602	patented	Annual tax	M&SR	1	19.668	
Gauthier	LS339	patented	Annual tax	M&SR	1	14.569	
Gauthier	LS340	patented	Annual tax	M&SR	1	16.187	
Gauthier	L2648	patented	Annual tax	MRO	1	16	2% NSR
Gauthier	L2649	patented	Annual tax	MRO	1	16	2% NSR
McVittie	L7934	patented	Annual tax	M&SR	1	16.39	
McVittie	L7055	patented	Annual tax	M&SR	1	15.257	
Gauthier	L7056	patented	Annual tax	M&SR	1	23.229	
Gauthier	L35279	patented	Annual tax	M&SR	1	16.066	
Gauthier	L2586	patented	Annual tax	M&SR	1	16.835	
Gauthier	L2587	patented	Annual tax	M&SR	1	21.1	
McVittie	L2588	patented	Annual tax	M&SR	1	18.575	
McVittie	L2589	patented	Annual tax	M&SR	1	15.216	
Gauthier	L6246	patented	Annual tax	M&SR	1	15.095	
McVittie	L6247	patented	Annual tax	M&SR	1	14.65	
McVittie	L4397	patented	Annual tax	MRO	1	15.257	
Gauthier	L106884 (67180)	lease	08/01/2013	M&SR	3	53.584	
Gauthier	L106884 (72883)						
Gauthier	L106884 (67288)						
McVittie	1217495	unpatented	27/05/2016	MRO	1	16	
Gauthier	1226891	unpatented	report pending	MRO	8	128	
Gauthier	4202030	unpatented	report pending	MRO	2	32	
Gauthier	3003814	unpatented	28/06/2016	MRO	10	160	
Gauthier	3003815	unpatented	28/06/2016	MRO	2	32	
McVittie	3004567	unpatented	30/10/2016	MRO	1	16	
McVittie	4210194	unpatented	24/03/2016	MRO	8	128	
McVittie	4210195	unpatented	24/03/2016	MRO	16	256	
McVittie	4210196	unpatented	24/03/2016	MRO	1	16	
				Total	**87**	**1,410**	

Note: *2% NSR = NSR royalty to Timmins Forest products Ltd.*

Figure 4.1 Property Location

Figure 4.2 Claim Map



4.3 PROPERTY AGREEMENTS

Contact Diamond Mines Corp., formerly Sudbury Contact Mines Limited, holds 100% of the diamond rights only on the 35 leased and patented claims.

On claims L2648 and L2649, Timmins Forest Products holds a 2% Net Smelter Return ("NSR") royalty (see Table 4.1). Queenston has the right to purchase 50% of the royalty, at any time, for C$1,000,000 and retains a First Right of Refusal on any third party offer to purchase the royalty.

4.4 ENVIRONMENTAL ISSUES

The Property was not subject to any known environmental liabilities as of the effective date of this report. There may be some mill tailings from the 1920s era stamp mills, but their location is unknown due to re-vegetation of the mine site. The last production (1965-1972) from the Property was trucked to the Upper Canada mill located 7 km to the southwest.

Three shafts are located on the Property. The #3 Shaft on the west shore of York Lake was the main production shaft for the previous underground operation. It extends to a depth of 605 feet (184 m), with an internal winze from the 500 to the 1,250-ft level. Levels are established at 80, 200, 350 and 500 feet, and, at 125-foot intervals from the 500 level to 1,250 feet (381 m). The shaft is capped. A waste pile from the early 1919-1935 underground development is located east of the #3 Shaft at the edge of Beaverhouse Lake. This waste material is non-acid generating and about 60% was used in 2003 to build roads.

The #1 Shaft is located further east, on the east shore of York Lake. It is 102 feet (31 m) deep and rock filled. Its perimeter is fenced. Less is known about the #2 Shaft, but historic plans show it to be 68 m SSW of the #3 Shaft at the northern end of the 'g' Vein. The shaft (estimated at 15 m deep) is now incorporated into the g Vein open cut, which is backfilled with waste rock.

In addition to the three shafts, two adits dating to 1912-1919 on the H and K veins are present. Both are backfilled. As noted above, an open cut on the 'g' Vein was backfilled with mine rock, along with capping of various raises, and refurbishment of the fencing and timber at the remaining hazards between 2001 and 2004.

4.5 FIRST NATION ISSUES

P&E is not aware of any First Nation issues pertaining to the Upper Beaver Property. Preliminary discussions have been held with local First Nations communities with regard to possible future mining developments on the property.

4.6 PERMITS AND OBLIGATIONS

The development of a mining project can require a number of environmental permits and approvals depending on the size, type of project and facilities required. Early stage exploration projects require few permits or approvals but environmental regulations still apply regardless of the need for specific approvals.

Accommodation for project personnel is located in the several local communities therefore no permit from the Ontario Ministry of Municipal Affairs and Housing will be required. The Ontario Ministry of Labour (MOL) needs to be advised of drilling programs but no permits are required.

For Queenston Mining Inc., asset retirement obligations relate to the dismantling of the McBean Mine site as well as a closure plan on the newly acquired Victoria Creek property. In accordance with the Ontario Ministry of Northern Development, Mines and Forestry, the company developed closure plans for the McBean Mine. The present value of the estimated site closure and restoration cost is recorded as a liability on the company's books and the company is providing financial assurance for the mine closure with a letter of credit. Queenston reviews all its property holdings with regard to environmental issues and risks. The Company is not aware of any other material environmental risks on its properties.

4.7 QUEENSTON HOLDINGS IN THE AREA

Queenston's land holdings in the Kirkland Lake gold camp area approaches 20,200 hectares containing 1,264 claim units comprising 36 mineral leases, 369 patented claims and 636 unpatented claims in 31 properties. Of these 31 properties, Queenston owns varying interests ranging from 41% to 100%. These properties occur primarily in three townships: Teck, Lebel and Gauthier (Figure 4.3).

Queenston's properties in the Kirkland Lake Area contain current and historic mineral resources in seven gold deposits: Upper Beaver, Anoki, Anoki South, McBean, Upper Canada, AK and 180 East.

Information on the historic resources on Queenston's properties can be examined in the technical report prepared by Dale R. Alexander, P.Geo. titled, "Technical Report on the Mineral Properties of Queenston Mining Inc. in the Kirkland Lake Gold Camp," dated November 17, 2007. This report can be found on the SEDAR website.

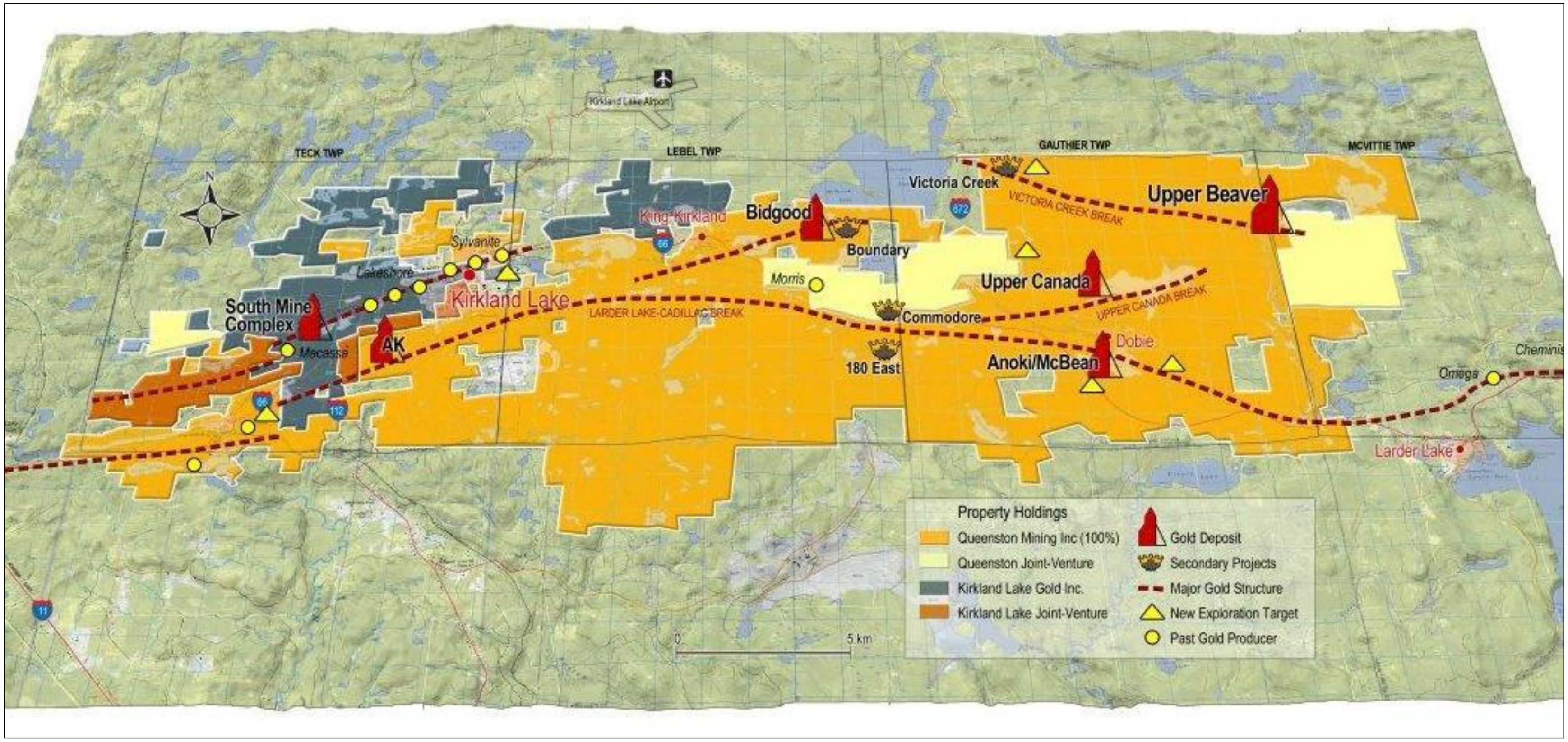
In 2008 a NI 43-101 mineral resource for the Upper Beaver property was presented in a report dated November 6, 2008 by Michael Kociumbas, P.Geo., of Watts, Griffis and McOuat Limited (WGM) of Toronto.

In 2009 P&E prepared a new NI 43-101 resource estimates for both the McBean and Anoki deposits in a report dated January 29, 2010 and in 2011, P&E prepared a new NI43-101 resource estimate for the Upper Canada Property. These reports can be found on the SEDAR website.

In the western side of the camp, the South Claims Joint Venture is continuing underground exploration. In 2008 an NI 43-101 compliant mineral resource was outlined by joint venture partner Kirkland Lake Gold Inc. and verified by Glenn R. Clark, P.Eng. of Glenn R. Clark and Associates Limited. The report, dated August 25, 2008 can be found on SEDAR.

In 2011, a NI 43-101 mineral resource update for the Upper Beaver property was presented in a report dated June 15th, 2011 by Kurt Breede, P.Geo., of Watts, Griffis and McOuat Limited (WGM) of Toronto.

Figure 4.3 Property Map of the Kirkland Lake Gold Camp



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Property is accessible from Highway 66. Beaverhouse Road crosses Highway 66, 11 km west of the village of Larder Lake. Beaverhouse Road is a gravel road that extends from the village of Dobie to Beaverhouse Lake, a distance of 7 km. Numerous old drill roads and recently constructed logging roads provide excellent access to the Property (see Figure 4.2).

5.2 CLIMATE

The climate is northern temperate with warm summers and cold winters. Temperatures vary from +30º Celsius in the summer to –40º Celsius in the winter. The ground is usually snow covered between mid-November and mid-April.

Vegetation is mixed bush with spruce, fir, larch, jack pine, poplar, birch, ash and alders. The patented claims were recently logged. Soil conditions and drainage tend to dictate the type of vegetation from open wet swamps to bare outcrop scarps.

The climate information presented in Figure 5.1 was taken from the weather station in Earlton, Ontario, located approximately 35 km south of the Upper Beaver Property.

Exploration activities would be hindered in snowmelt conditions but it is expected that any mining activity on the property could be conducted year-round.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The Property is located approximately 25 km east of the town of Kirkland Lake, Ontario. Kirkland Lake is the main commercial centre for the north part of the Timiskaming District and there is a skilled and capable workforce with experience in mining and mineral exploration in the immediate area.

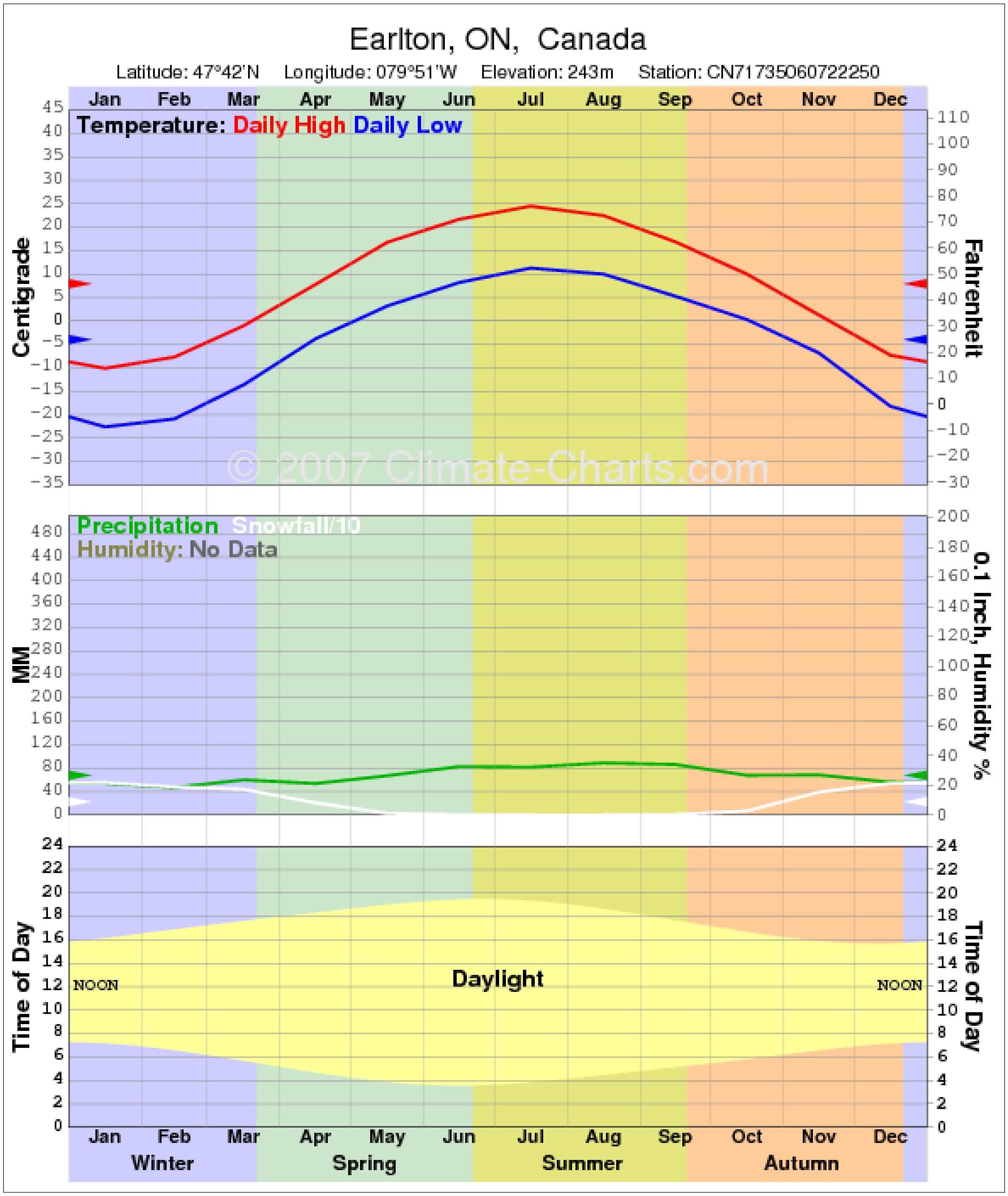
There is no power into the Property. The closest power line from which adequate power for mine operations is available is located 7 km to the south-southwest near the Upper Canada mine site at Dobie, Ontario.

Water is available from rivers, ponds and creeks within the Upper Beaver property.

5.4 PHYSIOGRAPHY

The topography is hummocky. Relief is in the order of 50 m from lakes, rivers and alder swamps at waterway margins, to higher outcrop knobs with local jack pine. Overburden depths range up to 30 m of clay till. Outcrop exposure averages 10-15% from low-lying exposures to more prominent knobs.

Figure 5.1 Climate Trend Data Earlton, Ontario



6.0 HISTORY

Note: *The authors of this section have drawn heavily upon selected portions or excerpts from material contained in the WGM 2011 Report.*

Gold was discovered west of Beaverhouse Lake in 1912 by Alfred Beauregard. A summary of previous work on the Property follows:

1912-1919 La Mine d'Or Huronia: shaft sinking, Nos. 1 and 3 shafts, development and production. No. 1 shaft 102 feet deep located on the east shore of York Lake. No 3 Shaft, 500 feet deep and winze, from 500 feet to 1,250 feet on the west shore of York Lake. Ten levels of mine developed. 15 ton stamp mill constructed;

1919-1928 Argonaut Gold Mines Limited leased the Property, constructed a 200 tpd mill and continued production. Mine was closed in 1928 when lower levels failed to develop sufficient ore. Production from 1912 to 1928, 131,000 tons at 0.20 opt Au (6.9 g Au/t and 0.60% Cu);

1935 Beaverhouse Lake Mines acquired Property and carries out surface exploration program, which resulted in the discovery of new veins;

1937-1939 Toburn Mines ("Toburn") options the Property. Underground development and mining to 350-level resumed;

1939 Ventures Ltd. dewatered the mine to the 500-level, 800 feet of new development;

1951 Toburn initiates surface drilling and geological mapping program;

1961 Augustus Exploration Ltd. acquires the Property. De-waters the mine, completes surface and underground drilling;

1964 Upper Canada Mines Ltd. ("Upper Canada") becomes manager of the Property, conducts AEM ("airborne electromagnetic") survey and geological mapping program;

1965 Upper Canada dewaters mine and carries out underground development. Mine put into production at 100 tpd. Mining rate then increased to 750 tpd, ore trucked to Upper Canada mill at Dobie;

1966 Upper Canada/Canico conducts geophysical test surveys, magnetometer, self potential and VLEM ("vertical loop electromagnetic") surveys completed over known veins;

1967 Upper Canada conducts Turam EM survey and surface drill program to test three AEM anomalies from the 1964 survey. Discovery of pyrite-pyrrhotite-graphite mineralization in Gauthier felsic volcanics;

1968	Upper Canada geophysical test surveys were conducted over the known veins to the west of No 3 Shaft, (IP ("induced polarization"), HLEM ("horizontal loop electromagnetic"), VLEM and magnetometer surveys;
1970	Upper Canada geological report by G.E. Parsons. Surface and underground mapping by R.G. Roberts and J.H. Morris. Geochemical mercury survey completed. Surface drillholes 71-1 to 71-4 completed;
1971	Mine closes after producing 106,750 ounces of gold Au (427,000 tons grading 0.25 opt Au (8.6 g Au t) and 1.28% Cu;
1974	Upper Canada surface diamond drilling, two holes (74-1, 74-2), aggregating 1,588 ft. Eighty-five (85) line miles of magnetometer survey, HLEM, and VLF-EM survey over claims in McVittie Twp. M.Sc. thesis concerning Property completed by J.H. Morris;
	Upper Canada study of Property completed by L.J. Cunningham, consultant. Inferred mineral resource estimate completed totalling 200,000 tons grading 0.23 opt Au, 1.23% Cu; mainly as a salvage operation;
1985	Queenston Gold Mines Ltd. conducts magnetometer surveys, detailed surface mapping, rock geochemical survey and limited stripping;
1989-1990	Pamorex Minerals Inc. - Queenston Mining Inc. JV formed. Program of detailed geological mapping and sampling, overburden stripping and trenching, geophysical surveys; HLEM (Horizontal Loop Electromagnetic) and magnetometer. Diamond drilling of 12 holes and 2 wedges aggregating 20,844 feet;
1991	Beaverhouse Resources Ltd., a subsidiary of Royal Oak Mines Ltd. ("Royal Oak") - Queenston Mining Inc. ("Beaverhouse-Queenston") JV formed. Diamond drilling of 17 holes aggregating 24,693 feet;
1995	Beaverhouse-Queenston continues exploration with diamond drilling of 10 holes aggregating 12,833 ft. IP and down-hole EM survey completed in drillhole 91-9;
2000	Queenston re-acquires 100% interest in the Property from Royal Oak receiver. Completes diamond drilling of one hole to 596 m;
2005	Queenston continues surface exploration with linecutting and IP survey;
	Queenston diamond drilling of 20 holes aggregating 8,334 m;
2006	Queenston extends drill program. Fifty-four holes aggregating 40,720 m completed;
2007	Queenston mandates Aeroquest International Limited ("Aeroquest") to complete a helicopter AeroTEM electromagnetic and magnetic survey of the Property;

	Quantec Geoscience Inc. ("Quantec") Titan-24 Array-DCIP & magnetotelluric survey completed for Queenston 49 drill holes completed for a total of 40,950 m are completed.
2008	Completed diamond drilling of aggregating 22,400 m in 20 holes and 2 wedges. Release of initial NI 43-101 compliant Mineral Resource estimate.
2009	During 2009, drilling continued at the Upper Beaver Property with the objective of improving resource quality and extending the limits of the 2008 resource envelope. Drilling focused mainly on an infill drilling campaign to test continuity between holes included in the 2008 resource estimate. During the year several holes were drilled to test for deep extension of the deposit from -800 m to -1200m. A total of 18 drill holes and 23 wedge cuts were drilled during 2009.
2010	In 2010, drilling of the deposit was focused on developing mineral resources to the indicated category below the base of the resource defined in the 2008 estimate, located at 800 m below surface. Work in this area continues to focus on the steeply plunging Porphyry Zones which represent the bulk of economic mineralization identified at the Property to date. Drilling was conducted throughout the year with three drills targeting the Porphyry Zones on both sides of the north-trending diabase dyke which bisects the deposit. A total of 13 drill holes and 29 wedge cuts were drilled during 2010 (22,533 m).
2011	Drilling continued at the Upper Beaver project for the duration of 2011 with three drills testing the Upper Beaver resource and one drilling a test hole in the proposed shaft location and other exploration targets which amounted to 44,313 m of diamond drilling in 84 holes and wedge cuts. In addition to resource extension and infill in the 800 m to 1200 m depth range, the exploration drilling targeted both the eastern and western portions of the deposit for shallow mineralization above the mineral resource. The shallow drilling program targeted an area 100-400 m northeast of the historic Upper Beaver Mine workings where only limited drilling was completed during the operation of the mine. The drilling completed in this area intersected narrow high-grade gold-copper mineralization often within a broader low-grade zone hosted in an altered mafic volcanic assemblage intruded by narrow feldspar porphyry and syenite dykes, Q-Zone.

An updated Resource estimate was completed in June, 2011 by Watts, Griffis and McOuat Limited.

A large diameter pilot hole has been completed to a depth of 1,200 m at the proposed future site of an exploration shaft. The entire hole was in massive syenite-porphyry and is currently undergoing detailed rock engineering studies.

Application for permitting of the proposed shaft and an advanced underground exploration program has been submitted to the Ontario Ministry of Northern Development and Mines with a schedule to begin breaking ground in mid-2012.

6.1 HISTORIC PRODUCTION

The main periods of production from the Property were 1912 to 1919, 1919 to 1928 and 1965 to 1971. Minor sporadic production also occurred from 1928 through 1944. Table 6.1 summarizes production after Lovell, 1979.

TABLE 6.1
SUMMARY OF HISTORIC MINE PRODUCTION

Period	Source	Production
1912-1944	La Mine d'Or Huronia, Argonaut Gold Mines Limited and Toburn Mines	38,347 ounces of gold and 1,030,783 pounds of copper from 119,372 t grading 9.99 g Au/t and 0.39% Cu.
1965-1971	Upper Canada/Upper Beaver Mines	102,362 ounces gold and 10,924,529 pounds of copper from 407,306 t grading 7.82 g Au/t and 1.22% Cu.
Total		**140,709 ounces gold and 11,955,312 pounds of copper from 526,678 t grading 8.31 g Au/t and 1.03% Cu.**

6.2 HISTORIC MINERAL RESOURCE/RESERVE ESTIMATES

A historic "inferred resource" of 200,000 tons (181,437 t) grading 7.89 g Au/t and 1.2% copper was estimated by Cunningham in 1974 on behalf of Upper Canada Resources Ltd. The estimate includes 68,039 t outlined at the time of closure in 1971, and 113,398 t of an inferred potential resource based on a minimum of 40 drill intersections accessible from the mine workings.

Cunningham (1977) stated that: "the bulk of the resources occur in veins U, X, XW and Y, which lie at the extreme north-western end of the mine workings". WGM understands that a list of the individual blocks that constitute the "resource" are no longer available with the 1974 report. The threads of the calculations are available, but the method and supportive data are missing. Thus, this historic "resource" estimate cannot currently be validated and should not be relied on.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

***Note:** The authors of this section have drawn heavily upon selected portions or excerpts from material contained in the WGM 2011 Report.*

The Upper Beaver Property is located in the Abitibi greenstone belt in the Superior Province of the Canadian Shield (Figure 7.1). Past gold production in the Kirkland Lake area has exceeded 75 million ounces.

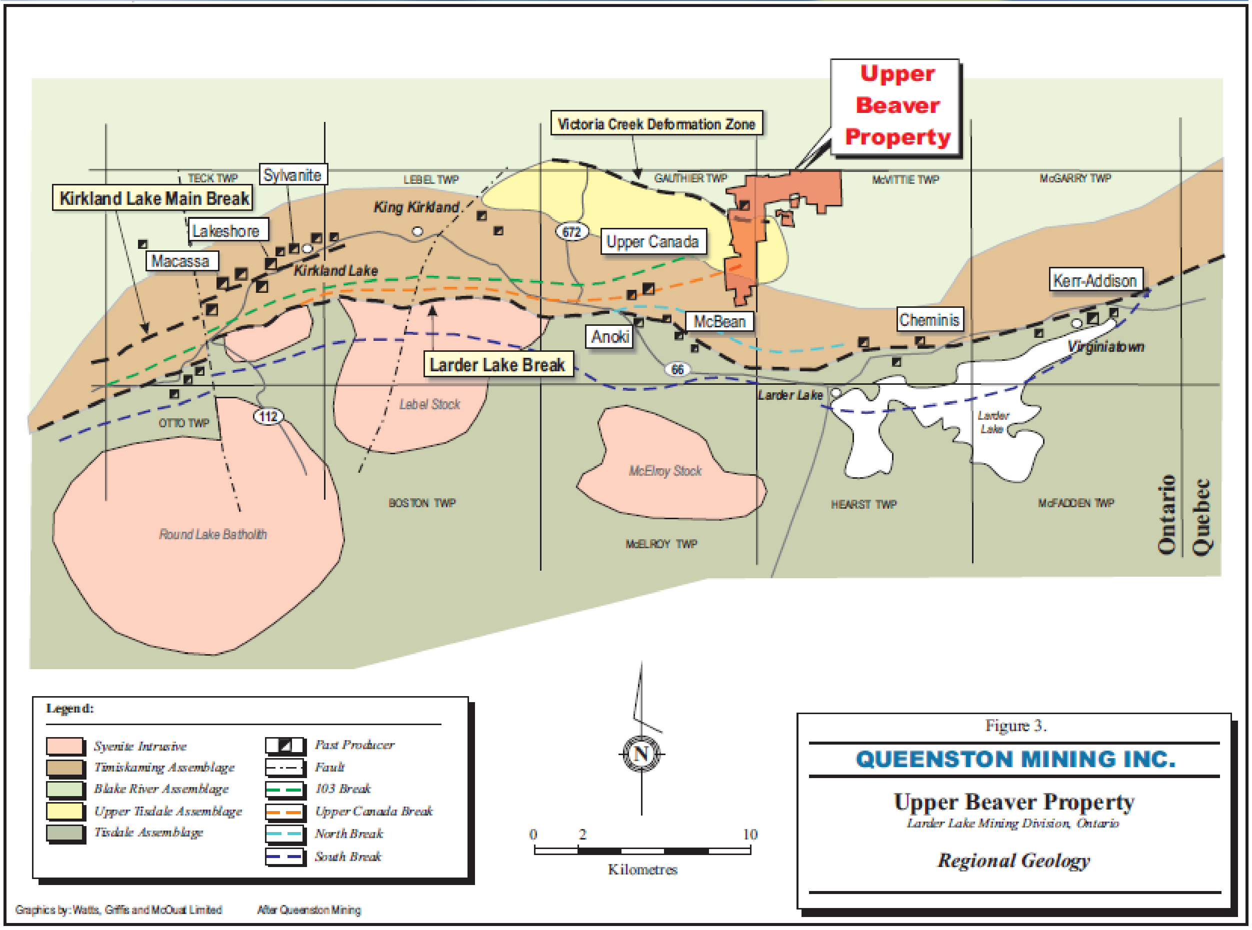
The Upper Beaver area is underlain by a succession of Archean assemblages:

Timiskaming	2676-2670 Ma. Clastic sedimentary rocks and some intercalated alkaline volcanic rocks. Syenite intrusions.
Unconformity	Upper Blake River: 2701- 2696 Ma; calc-alkaline basalt and andesite with some areas underlain by bimodal tholeiitic basalt and rhyolite.
Blake River	Lower Blake River: 2704-2701 Ma, Tholeiitic mafic volcanics with lesser amounts felsic volcanic rocks and turbiditic sedimentary rocks.
Victoria Creek Deformation Zone	
Tisdale	Upper Tisdale: 2704-2706 Ma Gauthier Group; Mainly calc-alkaline felsic to intermediate volcanic rocks with volcaniclastic sedimentary units. Lower Tisdale: 2707-2710 MA, Larder Lake Group, mainly tholeiitic mafic volcanic rocks with some komatiite, intermediate to felsic cal-alkaline volcanic rocks and iron formation

The Upper Beaver area is underlain by volcanic and volcaniclastic rocks of the Tisdale and Blake River assemblages. The dominant regional structural feature is the east-west trending Cadillac-Larder Lake Deformation Zone ("CLLDZ").

The locus of the CLLDZ is approximately 8 km south of the Upper Beaver mine. This deformation zone includes a number of component faults or breaks which are main controls for gold mineralization. The northeast-trending Upper Canada Break is one such component and likely is a splay fault off the CLLDZ. The projection of the Upper Canada Break, and its parallel Upper Canada Break South Branch, flank the shafts on the Property and appear to control, to some extent, syenite intrusions on the Property. The Victoria Creek Deformation Zone lies along the contact between the Tisdale and Blake River assemblages in the Property area and also likely represents a component of movement related to the CLLDZ.

Figure 7.1 Regional Geology Map



7.2 PROPERTY GEOLOGY

The central part of the Property is underlain by felsic and intermediate volcaniclastic rocks of the Upper Tisdale assemblage (Figure 7.2). These rocks are interpreted to occur in the core of an east to east southeast-trending, south-easterly plunging anticline – the Spectacle Lake anticline. The uppermost unit of the felsic volcanic sequence is a chert-pyritic tuff-carbonaceous sedimentary horizon. The Tisdale assemblage is conformably overlain by the Lower Blake River assemblage. This contact is located immediately south of the Upper Beaver mine shafts. The Victoria Creek Deformation Zone is in part spatially coincident with this contact between the older Tisdale and younger Lower Blake River assemblages.

The majority of the north part of the Property is underlain by the Lower Blake assemblage. These rocks, (previously known as the Kinojevis) consist of an alternating sequence of strongly magnetic iron and magnesium-rich tholeiitic basalts. The southern part of the Property is underlain by Timiskaming volcanics, volcaniclastics and sediments; however, the age of these sediments is currently being debated. This sequence is in fault contact with Tisdale assemblage rocks.

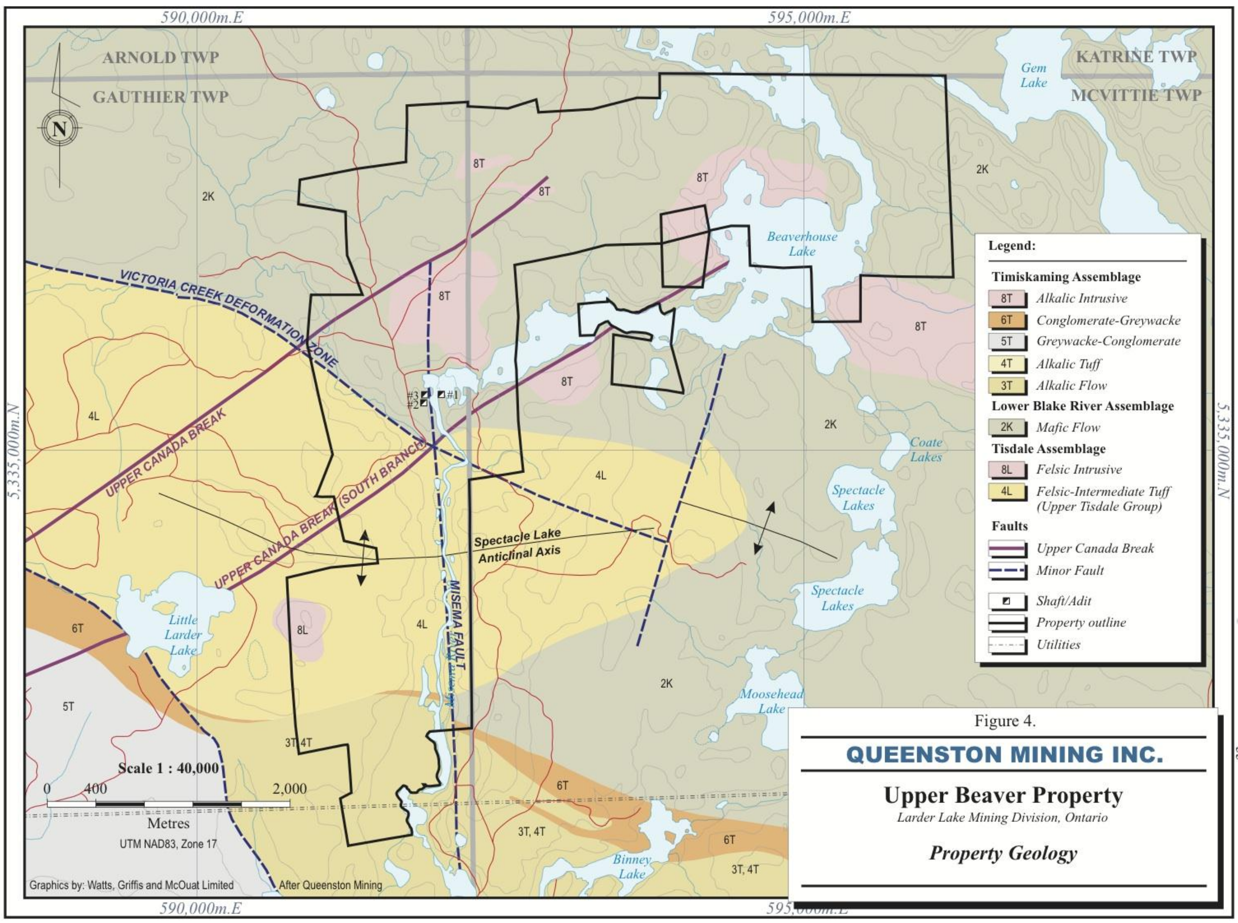
Syenite complexes of Timiskaming age intrude both the Tisdale, Blake River and Timiskaming assemblages. Various intrusive phases are present. The two main syenite phases are a dark grey mafic syenite and a red-brown to dark grey feldspar phyric syenite with an aphanitic matrix. Feldspar porphyry phases are also present. A prominent plug of syenite and mafic syenite, 600 m in diameter, occurs 250 m north of the #3 Shaft. A feldspar porphyry intrusion lies adjacent to its margin.

Matachewan diabase dykes cut all other rock units. The north-trending Misema Fault follows the Misema River. A diabase dyke, 30 to 40 m thick follows this structure.

The mafic volcanics east, west and north of the syenite plug strike east-west and dip 70-80º to the north. Three sets of faults have been mapped as follows:

- Northwest-trending and steeply dipping northeast;
- Northeast-trending and steeply dipping northwest; and
- East-west striking faults, dipping steeply north through the syenite plug and mafic volcanics.

Figure 7.2 Property Geology Map



8.0 DEPOSIT TYPES

The Deposit has been described as an Archean gold lode deposit where mineralized zones are structurally controlled and consist of brittle to ductile discontinuous, anatomising structures.

Such deposit types are common along the CLLDZ in the Kirkland Lake area where precious metal production has exceeded 40 million ounces. Details for these deposits are, however, highly variable. Common features include regional and local structural control and spatial and temporal relationship with felsic to alkalic intrusives.

Not-typical for the Kirkland Lake camp is the copper-gold association at Upper Beaver with the widespread and pervasive development of magnetite-feldspar-actinolite-epidote and carbonate-sericite. These features are more consistent with some deposits in the Timmins camp (such as the McIntyre Mine) along the Destor-Porcupine Fault Zone. Kontact, Dube and Benham (personal communication) have suggested that the Upper Beaver deposits are consistent with an alkali porphyry copper-gold model.

8.1 MINERALIZATION

Note: *The authors of this section have drawn heavily upon selected portions or excerpts from material contained in the WGM 2011 Report.*

Mineralization at Upper Beaver, as described by Queenston (Kontack, Dube and Benham, unpublished):

- Occurs both in flat and steeply dipping zones;
- Is of replacement-type with rare vein-type mineralization;
- Is associated with minor to pervasive alteration which includes feldspar, epidote, carbonate, sericite, silica and magnetite with trace hematite; and
- Has an element association of cu, au, or au-cu with associated molybdenum.

Queenston classifies the mineralization as three main groups of zones (from south to north):

- South Contact Zones;
- Beaver North Zones; and
- North Basalt Zones.

The vein systems are complex. Sufficient data is often not available to define a true width. As a rule of thumb, the more steeply dipping zones in the Beaver North and North Basalt Zones are estimated to have a true width factor of 70 to 77% of the core length interval, while the more flatly dipping South Contact mineralization ranges from 90 to 100% of the original intersection. The composite cross section from the 2008 NI 43-101 report illustrates schematically the South Contact and Beaver North zones, their orientations / structures and host rocks that contain the mineralization.

8.1.1 South Contact Zones

The South Contact Zones' disseminated mineralization consists of two, relatively flat-lying zones. These occur below and south of the mine workings in the Upper Tisdale contact area, marked by the roll in the stratigraphy from a north-westerly to north-easterly strike. Gold and copper contents increase where steeply dipping quartz-chalcopyrite-quartz veins and stringers intersect the flat-lying disseminated zones. The host is mafic breccia and volcaniclastic conglomerate with variable silica, epidote and calcite alteration, along with magnetite, chalcopyrite, pyrite, pyrrhotite, and visible gold.

8.1.2 Beaver North Zones

The Beaver North Zones include a series of east-northeast striking, north-dipping, fracture, vein and stringer systems containing chalcopyrite, magnetite, pyrite and visible gold. They occur below and north of the mine workings near the south contact of the large (600 m) syenite plug. The fracture systems crosscut a variety of rock types and are tentatively named by their position in the stratigraphy when first identified as: Syenite Zones, North Contact Zone (the basalt / syenite contact area), Porphyry Zones (associated with feldspar porphyry), Syenite Breccia Zones, and Lower Gauthier Zone (in Upper Tisdale assemblage rocks).

8.1.3 North Basalt Zones

The North Basalt Zones are located at the north contact of the 600 m, syenite plug. They are also characterized by a series of fractures and stringers with chalcopyrite and magnetite crosscutting syenite to mafic syenite and basalt. In all, some five zones (lettered A to E) are currently indicated, however, drill information is sparse and no Mineral Resources are yet defined for these zones. The fracture systems strike east-northeasterly and dip steeply north. They are primarily found in altered and brecciated basalt. Although no major faulting is indicated, the North Basalt Zones track close to the proposed trace of the regional Upper Canada Break.

The controlling structures for these zones vary. For the North Basalt Zones, the controlling structures are probably a combination of the Upper Canada Break, folded primary volcanic stratigraphy and intrusion of the syenite complex. For the South Contact Zones, multistage deformation along the contact between the Lower Blake River and Upper Tisdale assemblages is important. This deformation likely includes the Victoria Creek Deformation Zone, the feldspar Porphyry and progressive deformation prior to, during and postdating the feldspar porphyry.

For the Beaver North Zones, these same controls seem likely, plus the central syenite plug and continual deformation postdating intrusion of the central syenite plug are probably important. Mineralization also is zoned both with depth and laterally towards the central parts of individual zones. Early 1920s-1935 historic production came from gold quartz veins with low copper ratios, however, historic 1965-1972 production was from gold-bearing quartz-chalcopyrite-magnetite veins with high copper ratios. The central portions of the Porphyry Zones are chalcopyrite-magnetite rich. Laterally towards the east and west margins of the zones, pyrite becomes the dominant sulphide while chalcopyrite and magnetite decrease. Near the margins, the zone width is less than 1 m quartz-calcite veins, usually with visible gold. Outside the margins of the mineralized zones there is a chlorite-epidote-carbonate altered fractured to brecciated zones.

Vertically, the width of zones typically increase from an average of less than 1.5 m in the volcanics, to greater than 5 m in the syenite and mafic syenite porphyry rocks most, likely due to the more brittle nature of the intrusive rocks. There is an apparent increase in gold grades with depth from 3 g Au/t at the -400 m level to +10 g Au/t below the -500 m level. The Porphyry Zone are still open at depth, so it is not known if there is a similar quartz-sulphide-magnetite zoning towards the bottom of the zones, as is the case laterally. High gold ratios are not directly related to the chalcopyrite and magnetite content of the mineralized zones

9.0 EXPLORATION

Note: *The authors of this section have drawn heavily upon selected portions or excerpts from material contained in the WGM 2011 Report.*

Queenston exploration programs since reacquisition of the Property in 2000 have consisted mostly of diamond drilling and some geophysical surveying.

In 2000, Queenston drilled one drillhole. In early-2005, Queenston re-established a north-south cutline grid over the north-central part of the Property with lines spaced at 100 m intervals. Subsequently, Remy Belanger Geophysics from Rouyn-Noranda, Quebec was mandated to conduct a frequency domain Induced Polarization survey over the grid. A number of anomalies were defined that were drilled later in Phase I, 2005. Most anomalies were attributed to flowtop breccias and iron-rich (magnetite-enriched) tholeiitic basalts; some to mineralized zones.

The exploration programs in 2006 consisted mostly of drilling as described under Drilling. Preceding 2007 drilling Aeroquest International Limited was contracted to carry out a helicopter-borne geophysical survey over the Property and the adjacent Lac-McVittie JV Property. The survey was conducted using an AreoTEM II (Echo) time domain system and a high-sensitivity caesium vapour magnetometer. The total survey coverage was 297.8 line kilometres flown at 100 m line spacing in a 147 degree survey flight direction. The purpose of the survey was to determine the geophysical signature or "footprint" of the Upper Beaver gold-copper deposit and identify other potential targets on the properties.

The magnetometer survey was successful in outlining the geological characteristics of the properties. In the western portion of the survey area, on the Upper Beaver Property, the syenite plug that lies north of the mine workings and hosts the gold-copper mineralization at depth is identified by an oval shaped magnetic-low feature. This feature is surrounded by a high magnetic response occurring in the Lower Blake River metavolcanic basalts indicating the presence of magnetite, an important component of the mineralized system that hosts the Upper Beaver deposit. A similar magnetic-high response in the same package of rocks located 4 km to the east has been identified by the survey.

The most significant electromagnetic responses are located in the southern portion of the survey area within a magnetic-low feature that outlines the Upper Tisdale metavolcanic felsic pyroclastic assemblage. Here the survey has located a cluster of AEM anomalies in an area where previous drilling has intersected semi-massive pyrite, minor chalcopyrite, sphalerite and arsenopyrite with trace gold.

In September and October 2007, Quantec Geoscience Ltd. completed a four-line Titan 24 DCIP (DC Resistivity and Induced Polarization) and MT (Tensor-Magnetotelluric) survey over the Property. The purpose of the survey was to determine the geophysical characteristics of the new gold-copper mineralization discovered and to identify other, deeper targets on the Property that display similar characteristics.

The Titan 24 inversion results over the Upper Beaver mineralization identified responses (strong chargeability with coincident DC and MT low resistivity) for the South Contact, Beaver North and North Basalt Zones. The survey also identified at least 5 other anomalies that could represent significant sulphide mineralization, alteration and/or structure.

After completion of the 2008 Mineral Resource estimate, Queenston embarked on a program of deep exploration to explore and extend the Deposit below a depth of 800m. A Titan 24 geophysical survey identified a deep anomaly within the mineralized corridor below the resource. The first results of this program were reported in December 2008 (see Queenston news release dated December 16, 2008) with the highlight in deep hole UB08-139 that intersected a high grade, wide intersection in the Porphyry Zone assaying 30.3 g Au/t with 1.0% Cu over 20.8 m.

10.0 DRILLING

10.1 PRE-2000 DRILLING

Note: *The authors of this section have drawn heavily upon selected portions or excerpts from material contained in the WGM 2011 Report.*

WGM has not reviewed pre-2000 drilling on the Property except for what is listed in the History of the Property section. No pre-2000 drillholes are used for the current Mineral Resource estimate.

10.1.1 Queenston 2000 - 2008 Drilling

General

The magnetite-chalcopyrite-gold mineralization intersected in altered mafic breccias in the Pamorex-Beaverhouse Resources and Queenston drilling was considered to be possibly representing chalcopyrite-magnetite stringer mineralization related to a hydrothermal feeder zone to a nearby blind VMS deposit similar to the Corbet and Ansil VSM deposits which were mined at Rouyn-Noranda. During the winter of 2005, an IP survey was conducted to search for sulphide zones along east-west striking interflow contacts within the mafic volcanics overlying the felsic volcanics to the north and west of the old mine workings. Several IP anomalies of interest were detected.

Drilling to test the IP anomalies was started in May 2005. Phase I consisted of 15 drillholes (UB-05-01 to UB-05-15) totalling 5,913.4m, was planned to test IP and magnetic anomalies and to follow-up anomalous gold-copper zones intersected in the 1989-1995 Pamorex/Beaverhouse/Queenston joint venture drill programs. A phase II program, which consisted of five holes (UB-05-16 to UB-05-20) totalling 2,420.9 metres, was planned to follow-up the results of the first phase. Phase II was completed on August 27, 2005. Phase III extended from September 25, 2005 to November 03, 2006. It consisted of 54 drillholes (UB-05-21 to UB-06-74 totalling 40,720m.

Drilling in 2006 continued to encounter high grade mineralization over wide intervals. After completing a preliminary in-house resource estimation, it was decided to carry out an infill definition drilling program in preparation for an "NI 43-101" Mineral Resource estimate.

The Phase IV infill drill program started January 3, 2007 and was completed March 19, 2008. The purpose of this work was to drill off the Upper Porphyry gold-copper zone at 50-metre spacing's between the 400-700 metre levels. After intersecting a high grade zone in hole UB07-100 at -810 metre level, the infill drilling was extended to the 800 metre level. The drilling also tested the Syenite, North Contact, Lower Porphyry, Lower Gauthier and Syenite Breccia zones which occur in a broad alteration corridor above and below the main Upper Porphyry Zone. The program consisted of 60 drillholes, including wedge holes (UB 07 75 to UB-08-128), aggregating 49,060m.

The Phase I to Phase IV programs aggregated 100,672 m. All drilling was nominally NQ and carried out by Benoit Diamond Drilling Ltd. from Val d'Or, Quebec. The drill programs were planned and supervised by Wayne R. Benham P.Geo., Queenston. The core through the four phases was logged and sampled by W. Benham, F. Ploeger, P.Geo.; D. Alexander, P.Geo.; M. Leblanc, P.Geo., and Eric. von Bloedau (Temp. Geo.,) at Queenston's Upper Canada mine site.

10.1.2 Queenston 2008 - 2011 Drilling

General

Since the release of the 2008 NI 43-101 report, three additional phases of diamond drilling have been conducted on the Upper Beaver Property. The following is a brief summary of these drilling programs.

All diamond drilling was conducted by Benoit Diamond Drilling of Val d'Or, Quebec, and Major Diamond Drilling Group of Winnipeg, Manitoba who had acquired Benoit. Drill core was nominal NQ diameter during all phases of the drilling programs. Three HQ diameter drill holes were also completed during the period. The drill core, plus rejects and pulps from the sampling programs, are stored at the Upper Canada mine site.

Swastika Laboratories, of Swastika, Ontario was the primary lab used for assaying of samples for geochemical gold in ppb (Fire assay-one assay ton) and for copper in ppm. Samples with >1,000 ppb gold (1 g Au/t) were checked by fire assay using a gravimetric finish. Copper assays returning >10,000 ppm (1% Cu) were re-assayed for percent copper.

All drill hole collar locations were spotted by Northland Technical Surveys, Kirkland Lake, Ontario using a Total Station, NAD 83 UTM co-ordinates and geodetic elevation system. During drilling, down-hole attitude surveying was conducted using a Reflex instrument. Upon completion the holes were re-surveyed by Halliburton Sperry Drilling Services, North Bay, Ontario, using a north-seeking gyroscopic system. A small percentage of holes which experienced technical difficulties such as broken rod-strings, or which were otherwise abandoned before reaching targeted areas were subsequently not surveyed by gyro. Drill hole casings are left in place, capped and marked with 2x2 posts with aluminum tags or with metal flag casing caps identified with metal tags.

Since the 2008 resource estimation (up to and including drillhole UB08_128), a total of 60,228m of drilling in 102 holes, has been done on the Upper Beaver Project, including 48 holes from surface and 53 wedge cuts. The programs are subdivided below into the 2008-2009 drilling, infill drilling program, and 2009-2011 deep exploration and definition drilling.

The current June 2011 WGM resource estimate uses drill holes up to an including drill hole UB10_170W2.

10.1.3 2008 - 2009 Drill Program

During the period of March 19, 2008 to April 2009 a total of 16,572.5m of drilling was completed in 19 holes on the Upper Beaver Project, including 12 holes from surface, 6 wedge cuts and one extension to existing hole UB06-63. **Error! Reference source not found.** summarises the drill hole locations while detailed lithological descriptions and results are available in the drill logs for holes UB08_129 to UB08_139, UB09_135W1-W3, and UB09_63E, 63W1, 63W2.

Drill hole targets for the 2008-2009 drill campaign were two-fold; drill holes 129, 130, 131, 132, 132W, 133, 134 and 134A were designed to test Titan-24 anomalies outlined in the north end of the Property, north of the North Basalt Zone, and in the area south of the old mine workings, beneath and along the south shore of Beaver House Lake. No significant auriferous zones were outlined during this exploratory drilling.

The remainder of the drilling from the period (drill holes 135, 135W1, 135W2, 136, 137, 138, 139, 63E, 63W1 and 63W2) focused on exploring for the deep, down-dip extension of the East and West Porphyry Zones below the -800 metre elevation. Results of this drilling are tabulated in **Error! Reference source not found.**. Figure 10.1 is a surface plan showing drill hole locations for all surface drilling to-date on the Upper Beaver Property which were incorporated into the updated resource estimate. It also shows the location of cross sections as discussed in Section 17 of this report.

10.1.4 2009 - 2010 In-Fill Drilling Program

From June 2009 to January 2010 a second in-fill drilling program was undertaken, in order to upgrade the current resource block between the -400 to -800 metre elevations outlined in the 2008 NI 43-101 compliant report, from the inferred to the indicated categories. Drilling was focused primarily on the East and West Porphyry Zones which accounted for approximately 82% of the resource outlined by WGM. Pierce point locations were developed from 3-D modelling of the zones and longitudinal sections, in order to achieve approximately 25 metre spacing relative to previous drilling within the resource block. Pierce point locations were outlined by Manuel Ng Lai, P. Eng., Queenston Mining staff. It was determined that the most efficient way of ensuring accurate intersection of outlined pierce point locations was primarily from wedging from pre-existing holes. In about half of the cases, new holes were required to be drilled from surface, using controlled drilling techniques necessary to hit the tight target areas outlined. Sixteen holes were completed in the in-fill program, 8 from surface and 8 wedge cuts, totalling 8,950.3 m. All but two of the holes, which intersected post-mineralization dykes, successfully intersected the mineralized zones and the results are tabulated in **Error! Reference source not found.**.

10.1.5 2009 - 2011 Deep Exploration & Definition Drill Program

During the period of April 2009 to February 2011 (UB09_140 to UB11_171) exploration and definition drilling continued to explore the mineralized Au±Cu zones on the Upper Beaver Property. A total of 43,566m in 83 holes (including 8,950.3 metres for in-fill program, surface holes and wedge cuts) was completed during this period and are outlined in **Error! Reference source not found.**. Diamond drilling is still on-going, further delineating the Porphyry Zones.

Drilling has primarily focused upon further exploring and expanding the Porphyry Zones at depth, below the -800 metre elevation and the limits of the current resource, with the goal of further expanding the resource within these zones.

Drill holes are designed to target the down-dip/down-plunge extension of the Porphyry Zones based on an interpreted attitude of the vein system striking 50^o to 60^o and dipping 60^o to 70^o to the northwest. Drilling to-date has defined a plunge of the core of the vein system to be approximately 50^o to the northeast.

Historically, within the old mine workings from surface to the -385 metre elevation, at least six distinct types of vein morphologies were recognized:

1) Chalcopyrite-magnetite±quartz-calcite-pyrite.
2) Quartz-Calcite-Chalcopyrite±VG-molybdenite-specularite-pyrite.
3) Quartz+Molybdenite veins.
4) Calcite±Chalcopyrite-pyrite-specularite-molybdenite-VG.
5) Quartz-feldspar stringers in basalt.
6) Quartz±Chalcopyrite stringers in syenite.

TABLE 10.1
SELECTED ASSAY RESULTS FROM THE 2011 UPPER BEAVER DRILLING PROGRAM

Hole #	From (m)	To (m)	Interval (m)	Cu (%)	Au (g/t)	Zone
UB11-170W3	929.0	931.0	2.0	trace	6.74	New
	951.0	956.0	5.0	trace	3.12	North Contact
	981.0	984.0	3.0	trace	11.00	North contact
	1001.0	1005.5	4.5	0.20	11.35	Porphyry Zone
	1015.3	1020.0	4.7	0.69	24.59	Porphyry Zone
UB11-170W4	964.5	965.5	1.0	trace	5.45	New
	968.5	969.5	1.0	trace	5.76	New
	1023.0	1034.2	11.2	0.11	3.54	Porphyry Zone
including	1025.0	1032.2	7.2	0.10	5.00	Porphyry Zone
	1049.0	1052.0	3.0	trace	2.00	Porphyry Zone
UB11-170W5	818.0	822.0	4.0	trace	1.58	North Contact
	970.5	971.0	0.5	trace	25.96	North Contact
	1012.0	1013.0	1.0	0.30	11.01	Porphyry Zone
	1020.8	1023.0	2.2	trace	10.45	Porphyry Zone
UB11-170W6	939.2	940.5	1.3	trace	5.26	North Contact
	969.5	970.2	0.7	trace	75.45	Porphyry
	1028.9	1030.0	1.1	trace	3.45	Porphyry
UB11-171	1091.0	1095.0	4.0	trace	2.79	Porphyry Zone
including	1092.0	1093.7	1.7	trace	7.40	Porphyry Zone
	1230.3	1231.3	1.0	1.42	2.20	New
	1264.0	1264.5	0.5	3.08	1.34	New
	1269.0	1270.6	1.6	4.77	3.37	New
UB11-171W1	1085.5	1089.5	4.0	trace	5.06	Porphyry
including	1089.0	1089.5	0.5	trace	31.95	Porphyry
UB11-171W2	1047.0	1061.0	14.0	Trace	1.16	Porphyry
UB11-174	1129.0	1129.5	0.5	trace	12.62	North Contact
	1138.0	1139.0	1.0	trace	12.45	North Contact
	1198.0	1209.0	11.0	0.71	8.07	Porphyry
including	1199.0	1208.5	9.5	0.79	9.04	Porphyry
UB11-174W1	1098.0	1100.0	2.0	trace	2.83	North Contact
	1195.0	1202.0	7.0	0.21	1.70	Porphyry
	1209.0	1211.0	2.0	0.22	4.17	Porphyry
UB11-174W2	1182.0	1192.3	10.3	0.52	5.61	Porphyry
including	1182.0	1185.0	3.0	0.69	6.80	Porphyry

Table 10.1 Selected Assay Results from the 2011 Upper Beaver Drilling Program						
Hole #	**From (m)**	**To (m)**	**Interval (m)**	**Cu (%)**	**Au (g/t)**	**Zone**
	1189.0	1192.3	3.3	0.41	6.20	Porphyry
UB11-174W3	1179.5	1193.0	13.5	1.14	13.15	Porphyry
including	1179.5	1189.0	9.5	1.40	17.74	Porphyry
including	1182.5	1189.0	6.5	1.66	20.50	Porphyry
UB11-174W5	1182.0	1190.0	8.0	0.75	12.9	Porphyry
including	1184.5	1187.0	2.5	0.94	30.1	Porphyry
UB11-174W6	1085.0	1087.0	2.0	Trace	11.4	
	1173.0	1183.0	10.0	0.31	3.07	Porphyry
UB11-174W7	1162.0	1184.0	22.0	0.84	6.44	Porphyry
Including	1162.0	1166.0	4.0	1.22	10.4	Porphyry
	1172.0	1173.4	1.4	0.30	3.67	Porphyry
and	1176.0	1184.0	8.0	1.60	11.6	Porphyry
UB11-174W8	1159.0	1182.0	22.4	0.30	6.71	Porphyry
	1166.0	1173.8	14.0	0.39	8.21	Porphyry
Including	1117.0	1173.8	3.8	0.85	22.9	Porphyry
UB11-174W9	1073.0	1085.0	12.0	trace	1.53	NC
	1149.0	1167.4	18.4	0.53	9.15	Porphyry
Including	1156.0	1159.0	3.0	0.23	13.0	Porphyry
and	1164.5	1167.4	2.9	2.05	35.9	Porphyry
UB11-174W10	1198.0	1203.0	5.0	0.25	18.7	Porphyry
UB11-174W12	1162.0	1169.0	7.0	0.78	49.9	East Porphyry
Including	1164.0	1169.0	5.0	0.92	69.5	East Porphyry
UB11-175	1036.0	1042.0	6.0	0.70	4.56	Porphyry
including	1036.0	1039.0	3.0	1.40	8.00	Porphyry
	1027.0	1028.0	1.0	0.20	1.90	Porphyry
UB11-175W1	1018.0	1019.0	1.0	trace	122.82	Porphyry
	1030.0	1036.0	6.0	0.54	21.42	Porphyry
including	1033.0	1035.0	2.0	1.18	61.14	Porphyry
	1173.2	1173.7	0.5	0.15	11.52	New
	1181.8	1182.3	0.5	Trace	10.63	New
UB11-175W3	971.5	978.0	6.5	Trace	2.60	North Contact
	1012.0	1013.0	1.0	0.35	6.35	North Contact
	1037.0	1041.0	4.0	0.52	2.10	Porphyry
	1045.0	1070.0	25.0	0.40	11.5	Porphyry
Including	1045.0	1050.0	5.0	0.45	8.79	Porphyry
and	1057.2	1069.3	12.1	0.49	19.1	Porphyry
UB11-175W4	998.7	1006.7	8.0	0.15	0.90	Porphyry
UB11-180W1	855.0	911.0	56.0	Trace	2.75	Porphyry
Including	888.0	909.0	21.0	Trace	3.64	Porphyry
Including	901.0	909.0	8.0	Trace	8.35	Porphyry
UB11-180W2	884.0	887.0	3.0	Trace	13.8	NC
	907.0	917.0	10.0	Trace	3.55	Porphyry
Including	913.0	916.0	3.0	Trace	9.62	Porphyry
	928.0	930.0	2.0	Trace	6.59	Porphyry
UB11-180W3	825.0	827.0	2.0	Trace	10.9	Porphyry

TABLE 10.1 SELECTED ASSAY RESULTS FROM THE 2011 UPPER BEAVER DRILLING PROGRAM						
Hole #	**From (m)**	**To (m)**	**Interval (m)**	**Cu (%)**	**Au (g/t)**	**Zone**
	843.0	848.0	5.0	Trace	3.17	Porphyry
	935.0	937.0	2.0	Trace	3.56	New
UB11-180W4	912.0	929.0	17.0	Trace	3.00	Porphyry
Including	912.0	916.0	4.0	Trace	6.88	Porphyry
UB11-176	767.0	768.0	1.0	Trace	6.23	North Contact
	787.0	791.0	4.0	Trace	2.56	North Contact
	796.9	802.0	5.1	Trace	1.37	North Contact
	826.5	828.0	1.5	0.79	3.08	North Contact
	1156.0	1157.0	1.0	Trace	15.7	Syenite Breccia
UB11-177	313.0	411.0	98.0	0.10	2.41	New
including	320.0	346.5	26.5	trace	4.97	New
and	330.1	336.0	5.9	trace	11.2	New
and	362.0	366.7	4.7	trace	8.45	New
	453.7	455.8	1.1	2.02	8.05	New
	616.0	618.0	2.0	3.49	9.59	New
UB11-179	68.0	123.0	55.0	Trace	1.00	Q Zone
including	94.8	95.3	0.5	Trace	53.7	Q Zone
and	116.2	118.0	1.8	Trace	2.85	Q Zone
	138.0	146.0	8.0	Trace	1.00	Q Zone
	203.0	229.0	26.0	0.24	1.00	Q Zone
including	217.3	219.0	1.7	0.91	3.05	Q Zone
and	221.9	228.0	6.1	0.64	2.32	Q Zone
	276.4	320.7	44.3	0.27	2.06	Q Zone
including	285.5	286.0	0.5	3.31	7.49	Q Zone
and	302.5	303.0	0.5	2.69	119.6	Q Zone
and	316.5	317.0	0.5	1.70	12.0	Q Zone
	325.0	326.0	1.0	0.46	16.0	Q Zone
UB11-180	901.0	911.0	10.0	Trace	2.99	Porphyry
	1085.0	1095.0	10.0	1.64	0.77	
UB11-181	47.0	103.0	56.0	Trace	1.62	Q Zone
including	50.0	57.5	7.5	Trace	2.06	Q Zone
and	75.7	77.0	1.3	Trace	6.45	Q Zone
and	82.9	84.0	1.1	Trace	27.7	Q Zone
and	97.0	103.0	6.0	Trace	4.00	Q Zone
	160.7	185.4	24.7	Trace	0.63	Q Zone
	191.8	238.0	46.2	0.2	0.89	Q Zone
including	191.8	198.6	6.8	Trace	3.03	Q Zone
	207.0	238.0	31.0	0.25	0.58	Q Zone
UB11-182	57.0	59.4	2.4	Trace	2.44	Q Zone
	137.0	142.0	5.0	0.43	3.49	Q Zone
	164.0	166.0	2.0	Trace	13.6	Q Zone
	196.0	212.0	16.0	Trace	1.41	Q Zone
	227.0	228.0	1.0	Trace	4.27	Q Zone
	290.0	305.0	15.0	Trace	1.30	Q Zone
UB11-183	108.0	110.5	2.5	Trace	3.38	Q Zone

TABLE 10.1 SELECTED ASSAY RESULTS FROM THE 2011 UPPER BEAVER DRILLING PROGRAM						
Hole #	**From (m)**	**To (m)**	**Interval (m)**	**Cu (%)**	**Au (g/t)**	**Zone**
	125.2	129.0	3.8	Trace	2.26	Q Zone
	250.0	256.0	6.0	Trace	2.35	Q Zone
including	255.5	256.0	0.5	Trace	13.2	Q Zone
	434.0	435.5	1.5	1.84	4.21	Q Zone
	444.3	445.3	1.0	1.18	1.96	Q Zone
UB11-184	92.0	99.0	7.0	Trace	6.75	Q Zone
	217.0	232.0	15.0	0.33	1.02	NC
UB11-185	354.0	355.0	1.0	Trace	22.3	Porphyry
	369.0	370.0	1.0	Trace	16.7	Porphyry
	410.5	423.0	12.5	0.60	1.26	FW Copper
	602.0	604.0	2.0	2.42	3.09	New
UB11-186	132.0	175.0	43.0	0.13	1.94	NC
Including	132.0	136.0	4.0	Trace	4.70	NC
And	167.0	168.0	1.0	0.95	22.7	NC
	185.0	202.0	17.0	0.13	0.93	NC
UB11-187	43.0	97.0	54.0	0.26	3.41	Q Zone
Including	54.0	72.0	18.0	0.47	4.12	Q Zone
	66.0	69.0	3.0	0.99	10.1	Q Zone
	79.0	82.0	3.0	0.12	5.43	Q Zone
And	92.0	97.0	5.0	0.35	16.8	Q Zone
	141.0	152.0	11.0	Trace	1.77	Q Zone
UB11-188	289.0	390.0	101.0	0.10	2.01	
Including	293.0	296.0	3.0	Trace	16.7	NC
And	337.0	342.3	5.3	Trace	7.79	Porphyry
And	378.0	390.0	12.0	0.31	2.37	SC
Including	385.0	386.0	1.0	0.72	11.3	FW Copper
	546.0	547.0	1.0	Trace	12.4	FW Copper
	557.0	558.0	1.0	0.18	5.47	FW Copper
UB11-189	40.0	56.0	16.0	0.33	1.31	Q Zone
Including	54.0	56.0	2.0	0.35	5.79	Q Zone
	71.0	75.0	4.0	0.08	3.19	Q Zone
	114.0	118.0	4.0	1.15	26.5	Q Zone
	134.0	136.0	2.0	0.12	6.70	Q Zone
	175.5	179.1	3.6	2.35	2.89	NC
UB11-190	41.0	45.0	4.0	0.26	4.31	Q Zone
	58.0	65.0	7.0	0.19	1.56	Q Zone
	82.0	86.0	4.0	0.24	1.69	NC
	136.0	136.5	0.5	2.08	4.90	NC
	166.0	172.8	6.8	0.18	1.56	NC
	522.0	528.0	6.0	Trace	3.05	New
Including	527.0	528.0	1.0	Trace	13.4	New
UB11-191	171.0	185.0	14.0	Trace	2.25	East Shallow
Including	174.0	176.0	2.0	Trace	11.7	East Shallow
	188.0	191.0	3.0	Trace	1.67	East Shallow
	233.0	260.	27.0	Trace	1.93	East Shallow

TABLE 10.1
SELECTED ASSAY RESULTS FROM THE 2011 UPPER BEAVER DRILLING PROGRAM

Hole #	From (m)	To (m)	Interval (m)	Cu (%)	Au (g/t)	Zone
Including	234.0	239.0	5.0	Trace	3.00	East Shallow
And	245.0	248.0	3.0	0.14	5.20	East Shallow
And	256.0	259.0	3.0	Trace	4.66	East Shallow
	270.0	271.0	1.0	Trace	5.91	East Shallow
	297.0	304.0	7.0	0.22	1.64	East Shallow
	308.0	310.5	2.5	0.46	2.52	East Shallow
	342.0	355.0	13.0	Trace	1.6	East Shallow
UB11-192	73.5	74.5	1.0	Trace	29.2	NC
	169.4	188.0	18.6	0.26	0.66	Porphyry
	263.0	323.0	60.0	0.15	2.51	SC
Including	280.0	292.0	12.0	0.30	4.94	SC
And	296.0	308.0	12.0	0.16	3.50	SC
And	318.0	321.0	3.0	0.42	6.49	SC
UB11-193	78.0	95.0	17.0	Trace	1.15	NC
	102.0	107.0	5.0	0.13	1.08	NC
	128.0	131.0	3.0	Trace	5.40	NC
	211.0	221.0	10.0	0.54	1.11	SC
	228.0	234.0	6.0	0.25	8.55	SC
Including	232.0	234.0	2.0	0.28	23.7	SC
UB11-194	44.0	53.0	9.0	Trace	5.18	NC
	90.0	97.0	7.0	0.29	5.83	NC
UB11-195	127.0	175.0	48.0	Trace	2.07	East Shallow
	227.0	236.0	9.0	Trace	3.81	East Shallow
	250.0	256.0	6.0	Trace	3.48	East Shallow
	298.0	351	53.0	Trace	2.26	East Shallow
Including	305.0	316.0	11.0	0.10	2.75	East Shallow
And	326.0	338.0	12.0	0.11	1.75	East Shallow
And	346.0	351.0	5.0	Trace	11.7	East shallow
UB11-196	18.0	20.0	2.0	0.39	2.45	West Shallow
	51.0	53.0	2.0	Trace	2.73	West Shallow
	104.0	108.3	4.3	0.66	4.26	West Shallow
	155.0	166.0	11.0	0.26	1.02	West Shallow
	262.0	263.0	1.0	4.85	24.2	West Shallow
	286.0	298.0	12.0	0.11	1.40	West Shallow
	514.0	515.0	1.0	0.89	14.4	West Shallow
UB11-197	193.0	194.0	1.0	Trace	9.03	East Shallow
	312.0	325.4	13.4	0.13	1.42	East Shallow
	343.0	399.0	56.0	0.12	2.49	East Shallow
Including	344.0	355.0	11.0	Trace	10.1	East Shallow
	552.0	567.0	15.0	Trace	1.52	East Shallow
	573.0	574.0	1.0	0.14	12.4	East Shallow
UB11-198	50.0	52.0	2.0	0.39	2.02	West Shallow
	92.0	96.0	4.0	0.74	0.96	West Shallow
	110.0	120.0	10.0	0.14	42.2	West Shallow
	137.0	144.0	7.0	0.31	3.02	West Shallow

TABLE 10.1 SELECTED ASSAY RESULTS FROM THE 2011 UPPER BEAVER DRILLING PROGRAM						
Hole #	**From (m)**	**To (m)**	**Interval (m)**	**Cu (%)**	**Au (g/t)**	**Zone**
	200.0	205.0	5.0	0.27	1.75	West Shallow
UB11-200W1	1489.7	1490.2	0.5	1.27	10.7	East Porphyry
	1520.0	1526.0	6.0	0.60	13.7	East Porphyry
Including	1520.5	1523.0	2.5	1.15	30.2	East Porphyry
	1534.5	1535.0	0.5	1.59	16.2	East Porphyry
UB11-202	43.0	50.0	7.0	0.31	3.02	West Shallow
	101.0	105.0	4.0	0.25	6.33	West Shallow
	150.0	152.0	2.0	Trace	7.14	West Shallow
UB11-203	40.0	42.0	2.0	0.37	11.2	West Shallow
	151.0	156.0	5.0	0.11	2.26	West Shallow
	177.0	177.7	0.7	Trace	7.14	West Shallow
UB11-204	35.7	52.0	16.3	0.30	0.78	West Shallow
	119.0	145.0	26.0	0.20	4.05	West Shallow
Including	132.0	142.0	10.0	0.24	8.92	West Shallow
And	132.0	134	2.0	0.59	6.27	West Shallow
	188.0	194.0	6.0	0.56	28.5	West Shallow
Including	190.0	194.0	4.0	0.76	42.2	West Shallow
UB11-205	94.0	112.0	18.0	0.47	1.27	West Shallow
	136.0	163.0	27.0	0.31	5.78	West Shallow

TABLE 10.2 SUMMARY OF DRILLHOLES						
Hole ID	**Location X**	**Location Y**	**Location Z**	**Azimuth (o)**	**Dip (^{o})**	**Length (m)**
2011						
UB11-170W3	591534.0	5336268.0	302.36	131.1	-60	664.1
UB11-170W4	591534.0	5336268.0	302.36	131.1	-60	435.0
UB11-170W5	591534.0	5336268.0	302.36	131.1	-60	525.0
UB11-170W6	591534.0	5336268.0	302.36	131.1	-60	617.6
UB11-171	591745.0	5336268.0	306.28	139.8	-64.6	1326.0
UB11-171W1	591745.0	5336268.0	306.28	139.8	-64.6	507.8
UB11-171W2	591745.0	5336268.0	306.28	139.8	-64.6	627.0
UB11-174	591715.9	5336495.9	306.4	142	-70.5	1410.0
UB11-174W1	591715.9	5336495.9	306.4	142	-70.5	445.2
UB11-174W2	591715.9	5336495.9	306.4	142	-70.5	492.0
UB11-174W3	591715.9	5336495.9	306.4	142	-70.5	360
UB11-174W5	591715.9	5336495.9	306.4	142	-70.5	264.0
UB11-174W6	591715.9	5336495.9	306.4	142	-70.5	514.0
UB11-174W7	591715.9	5336495.9	306.4	142	-70.5	558
UB11-174W8	591715.9	5336495.9	306.4	142	-70.5	579.0
UB11-174W9	591715.9	5336495.9	306.4	142	-70.5	612.0
UB11-174W10	591715.9	5336495.9	306.4	142	-70.5	703.3
UB11-174W12	591715.9	5336495.9	306.4	142	-70.5	335.0
UB11-175	591759.9	5336345.0	302.4	140.0	-71.0	1245.0
UB11-175W1	591759.9	5336345.0	302.4	140.0	-71.0	514.8

TABLE 10.2 SUMMARY OF DRILLHOLES						
Hole ID	**Location X**	**Location Y**	**Location Z**	**Azimuth (o)**	**Dip (^{o})**	**Length (m)**
UB11-175W3	591759.9	5336345.0	302.4	140.0	-71.0	609.4
UB11-175W4	591759.9	5336345.0	302.4	140.0	-71.0	251.0
UB11-180W1	591749.8	5336274.9	301.8	140	-66	526.0
UB11-180W2	591749.8	5336274.9	301.8	140	-66	567.6
UB11-180W3	591749.8	5336274.9	301.8	140	-66	132.30
UB11-180W4	591749.8	5336274.9	301.8	140	-66	610.40
UB11-176	591486.0	5335920.0	303.3	140	-60	1244.0
UB11-177	591844.0	5335920.0	303.3	140.0	-60	525.0
UB11-179	591661.0	5335740.9	299.1	130	-55	326.0
UB11-180	591749.8	5336274.9	301.8	140	-66	1199.4
UB11-181	591661.0	5335740.9	299.1	140	-40	402
UB11-182	591712	5335699.9	298.5	130	-46	325
UB11-183	591695.9	5335778.9	301.9	132	-55	492
UB11-184	591659.9	5335778.9	301.9	132	-46	313
UB11-185	591825.0	5335905.9	303.8	140	-60	735
UB11-186	591745.0	5335737.0	300.4	130	-45	222
UB11-187	591630.9	5335703.0	296.7	130	-60	696
UB11-188	591825.0	5335905.9	303.8	140	-55	567.0
UB11-189	591630.9	5335703.0	296.7	130	-45	179.1
UB11-190	591688	5335655.0	295.6	130	-45	414.0
UB11-191	591857.9	5335859.9	302.5	140	-47	576.0
UB11-192	591857.9	5335614.0	289.0	130	-45	327.0
UB11-193	591737.0	5335649.9	289.1	130	-45	252.0
UB11-194	591826	5335670	286	130	-45	121.0
UB11-195	591891	5335855	302.7	140	-52	528.0
UB11-196	591499.9	5335747.4	303.7	127.8	-60.6	671.0
UB11-197	591844	5335920.0	303.3	140	-57	597.7
UB11-198	591499.9	5335747.4	303.7	140	-45	828.4
UB11-199	591665.9	5336825.9	315.9	130	-70	189.0
UB11-200W1	591665.9	5336825.9	301.8	135	-63	338.0
UB11-202	591538.0	5335716.0	301.8	135	-63	338.0
UB11-203	591600.0	5335664.0	298.4	130	-45	177.7
UB11-204	591599.9	5335730	300.7	130	-55	225.0
UB11-205	591599.9	5335768.9	300.7	130	-62	687.00
Subtotal	54	Holes				28,926.8

Within the upper levels of the mine the predominant vein set was hosted within mafic volcanic flows of the Blake River Group. These veins had an attitude striking NNE (20°) and dipping steeply to the west (70°). In the middle and lower levels of the mine the predominant vein system encountered strike at 50 to 60° and dip 60 to 70° to the north-west, which is the interpreted attitude of the Porphyry Zone(s) and the hanging wall North Contact Zones. It is postulated that a number of these north-north-easterly striking subsidiary veins occur at depth, but have not been specifically targeted with the recent drilling. Due to their orientation at an acute angle to the drilling direction (145°), intersection and hence correlation of these vein systems is problematic.

During the course of drilling subsidiary vein systems within the hanging wall and footwall of the Porphyry Zone(s) were also intersected, but were not specifically targeted in lieu of any determined plunges or potential attitude variations within these secondary zones. These include the North Basalt Zone, North Contact Zones and the South Breccia Zone located proximal to or within the underlying intermediate pyroclastic rocks of the Upper Tisdale Group. Most of the deep holes done to-date have been taken well into the footwall of the Porphyry Zone in order to define this underlying contact with the Tisdale Group and to further define, if possible, the South Breccia Zone. No further drilling of the South Contact Zone was undertaken during the period.

At depth the mineralized system manifests itself as a sheeted vein array within a broad brittle-ductile deformation corridor 200 to 300 metres wide associated with wide spread Sericite±Hematite±Carbonate alteration. Mineralization within the Porphyry Zone(s) is predominantly Quartz-Calcite-Magnetite-Chalcopyrite-Pyrite veining and fracture fillings. The Porphyry Zone(s) and North Contact Zone(s) are hosted primarily within micro-phyric hornblende porphyry and feldspar porphyry dykes of the Beaver House Intrusive complex, although veining is also recognized within basaltic flows and a limited amount of volcaniclastic rocks of the Blake River Group.

10.2 SURVEYS

For the 2005 Phase I and II programs, drillholes were spotted using global positioning system ("GPS") and the north trending (100 m spaced lines) cut grid on the Property established for the IP survey. Casings for most of the drill holes were subsequently surveyed in 2005 by Northland Technical Surveys ("Northland") of Kirkland Lake, Ontario using Total Station, NAD 83 UTM co-ordinates and geodetic elevation. Phase III and IV program drillhole sites were all (except for one drillhole) spotted directly by Northland using Total Station.

Two fore sites were used to spot the holes because of the configuration of the drill shack. Drillers lined up the drills for azimuth. The drillers will contact the project geologist by phone when the first downhole survey test results were taken 15 m below the casing. As a guideline, if the test results were within ± 0.5° of the planned dip and within ± 2° of the planned azimuth, the hole was continued. If the results were unsatisfactory, the drillers were instructed to pull the casing and restart the hole. The drillers submitted daily work reports for day and night shifts for each drill rig. The drillers are in radio and/or cell phone contact with their foreman, Queenston's Kirkland Lake exploration office and/or the project geologist at his local residence in case of any problems or questions.

Downhole attitude surveys for Phase I, UB-05-01 to UB-05-15 were by Reflex EZ-SHOT. For subsequent drilling, EZ-SHOT was largely used for surveying only during drilling. After the holes were completed they were resurveyed using a north seeking gyroscopic system by Halliburton Sperry Drilling Services ("Halliburton") North Bay, Ontario. However, for a number of drillholes, Halliburton was not available in a timely manner and some holes were lost in faults or were blocked by cave after the drill was dismounted. A number of drillholes therefore only have EZ-SHOT surveys.

Beginning in 2006 and continuing to the present, all drill holes are spotted by Northland and subsequently down hole surveyed by Halliburton.

Figure 10.1 Drillhole Location Map (Holes to UB 10 1700W2)

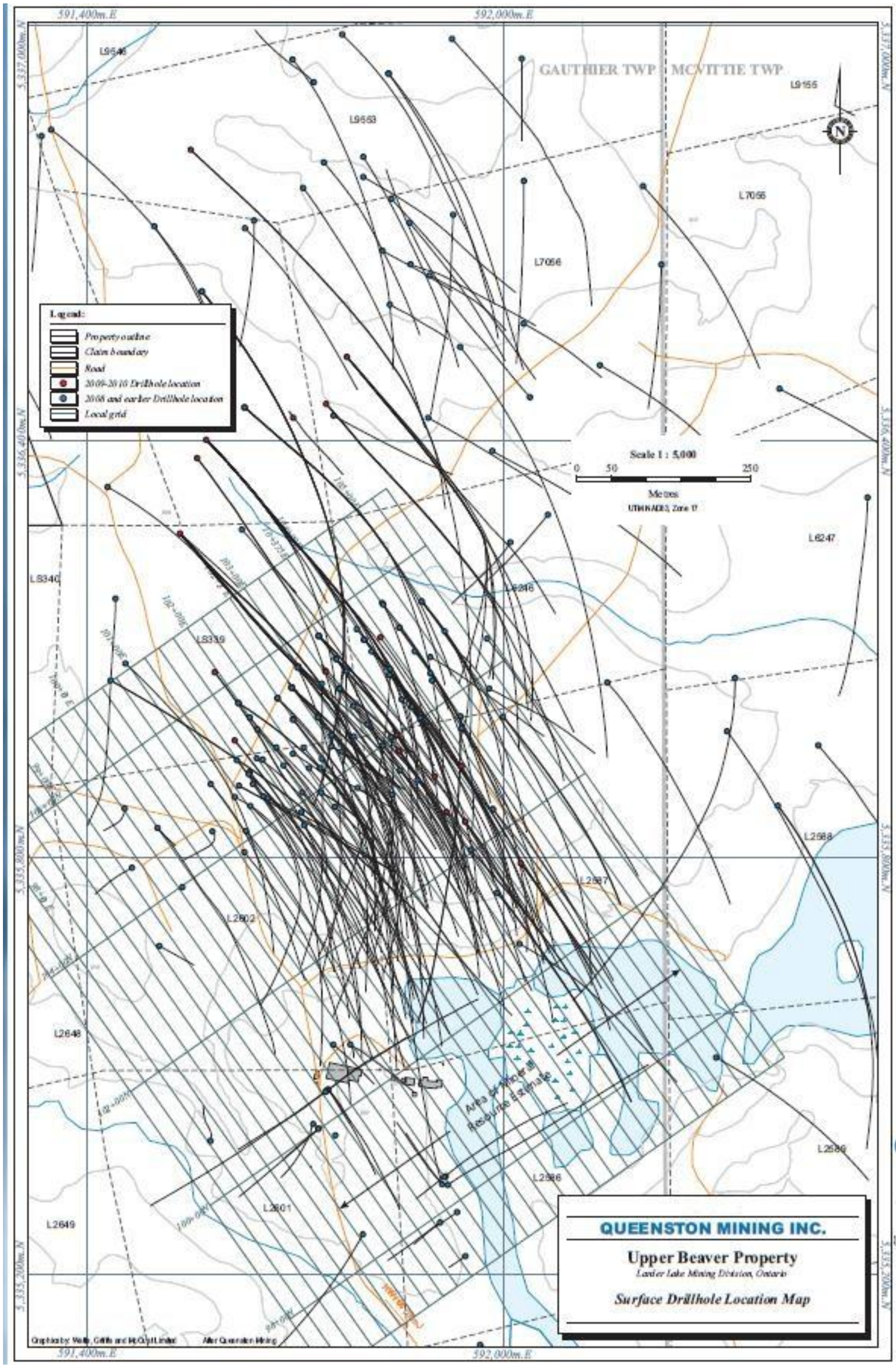
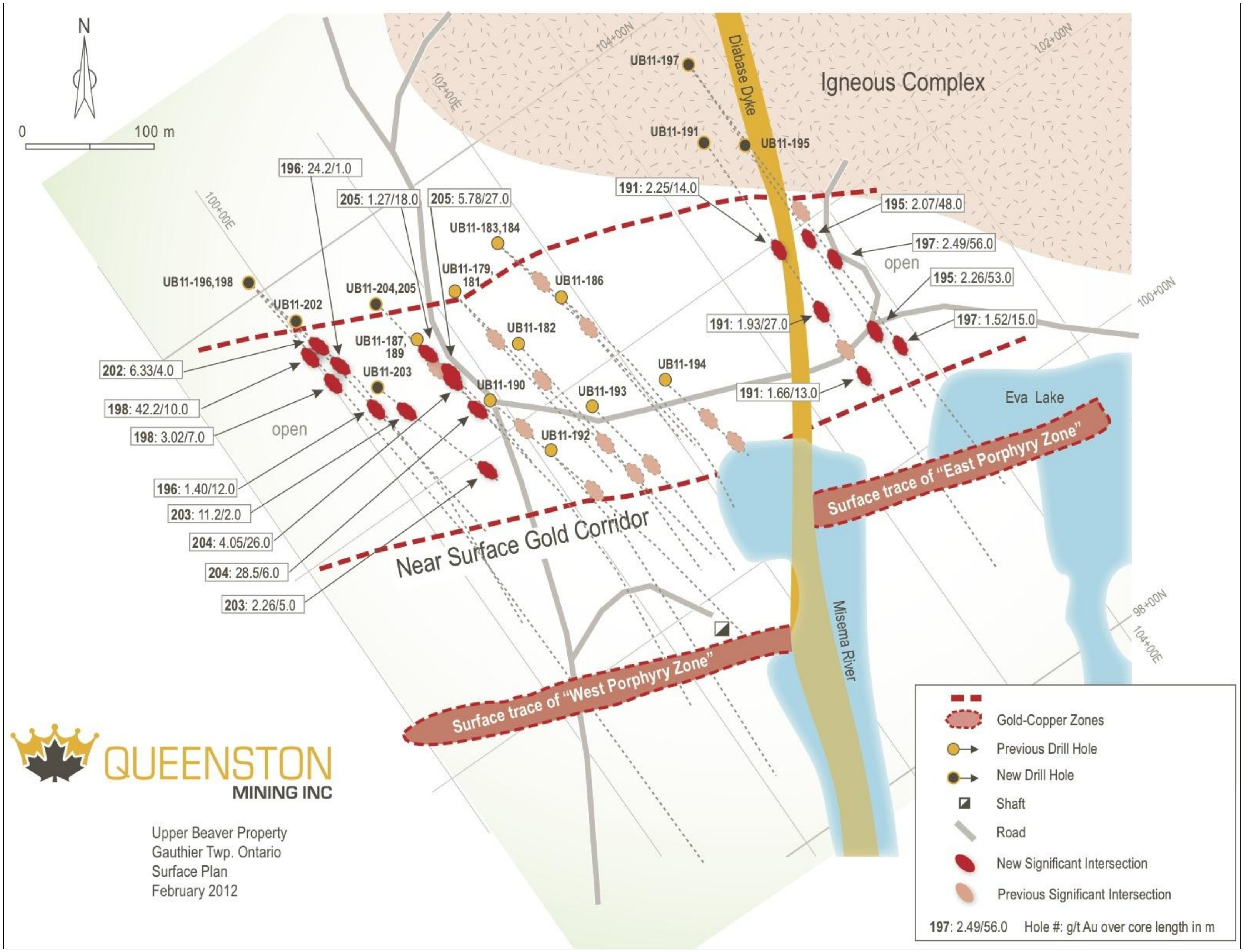


Figure 10.2 Upper Beaver 2011 Bore Hole Locations



Drilling continued at the Upper Beaver project for the duration of 2011 with three drills testing exploration targets and one drilling a test hole in the proposed shaft location. An 84 borehole drill program, totaling 44,313 m of diamond drilling, was completed. The exploration drilling targeted both the eastern and western portions of the deposit and tested for shallow mineralization above the mineral resource. The shallow drilling program targeted an area 100-400 m northeast of the historic Upper Beaver Mine workings where only limited drilling was completed during the operation of the mine. The drilling completed in this area intersected narrow high-grade gold-copper mineralization often within a broader low-grade zone hosted in an altered mafic volcanic assemblage intruded by narrow feldspar porphyry and syenite dykes, Q-Zone. The deposit remains open along strike to depth and new shallow zones have been

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section was completed by Richard Risto of WGM.

Statements below are current as at the release of WGM's Technical Report, June 15, 2011. Work completed on the Property subsequent to this date has not been reviewed, nor has it been determined if said work has any material effect on WGM's conclusions as at the date of its report.

11.1 SAMPLING METHOD

11.1.1 Pre-2002 Programs

WGM has not reviewed any pre-2002 program data for the Property. No pre-2002 drillhole data is used in the current Mineral Resource estimate. Information relating to Queenston's drilling programs between 2005 and 2008 are in WGM's 2008 report.

11.1.2 2008 To 2011 Programs

11.2 CORE HANDLING, LOGGING AND SAMPLING PROCEDURES

11.2.1 Core Logging

During the period March 2008 to February 2011, most surface diamond drillholes were NQ in diameter and three in HQ diameter (Hole UB09_144, UB09_146, and UP09_147). After pulling the rods, the core is placed in wooden core boxes by the drillers. The boxes are picked up by Queenston technicians at the drill site and delivered to the core logging facility at the former Upper Canada mine site.

The core logging protocol by Queenston geologists is summarized as follows:

A rock quality designation ("RQD") technician looks at the core, measures runs and core lengths, joint angles and records the data. The core is re-measured by the geologist and also checked that the drillers' metre blocks are correct, and the metreage is marked at the start of each box. Any lost or ground core, zones of poor RQD (i.e. <75%) or reaming done by wedging are noted within the log.

The core is logged in detail and recorded in a digital format using a Microsoft Excel spreadsheet.

11.2.2 Sampling

Core displaying obvious mineralization and alteration is sampled. The samples are marked by the geologist and sample tickets are inserted in the core box. Depending on the lithology, alteration and mineralization, sample widths are predominantly confined to 0.5m or 1.0m lengths.

The samples are entered on the drill logs and for each sample the percentage of quartz- carbonate veining, % pyrite/pyrrhotite, % magnetite and % chalcopyrite are estimated and entered on the log. After logging is completed the boxes are photographed before being returned to the racks. Digital photographs are stored in folders by hole along with the digital logs. The samples are then cut in half by a Queenston technician using a diamond core saw. Half the core is placed in a plastic bag with a sample ticket and the other half is put back in the box with a duplicate sample ticket at the end of the sampled interval. Samples with visible gold have blanks inserted following the sample and are flagged for the core cutter to take special care to clean the saw blade after cutting the potentially high grade sample in order to avoid contamination of the next sample. The assay lab is also advised of visible gold samples to avoid batch contamination. The bagged samples are placed in rice bags, a lab work order is prepared and the samples are delivered by truck to Swastika Laboratories Ltd. ("Swastika") of Swastika, Ontario.

Metal tags with the drillhole number and the depth of hole for the contained core interval are nailed onto the end of each core box. The boxes of mineralized zones are placed in racks outside for future reference including a few uncut boxes above and below the zone. Boxes which have not been sampled are stored on pallets. Starting in 2007, some old holes and the unmineralized tops of drillholes with no samples were stacked on wooden pallets to save core rack space.

WGM believes that Queenston's logging and sampling methods are to industry standard and appropriate.

11.3 SAMPLE PREPARATION, ASSAYING AND SECURITY

11.3.1 2002-2011 Programs

Queenston's assessment program on the Upper Beaver Property was initiated with one drillhole in 2002. Additional drilling was completed from 2005 through 2011 and is currently ongoing. Swastika was the Primary laboratory used for all assay work. Secondary laboratories for external check assaying were used for the 2007 to 2010 programs and are ongoing. The Secondary labs were Polymet Laboratory ("Polymet") of Cobalt, Ontario and Laboratoire Expert Inc., ("Expert") of Rouyn-Noranda, Quebec. From 2008 to 2011 the secondary labs used were Expert and SGS Laboratories ("SGS") of Lakefield, Ontario. SGS is accredited under the Standards Council of Canada. Their scope of accreditation conforms to the requirements of CAN−P−1579 Guidelines for the Accreditation of Mineral Analysis Testing Laboratories and CAN−P−4E (ISO/IEC 17025:2005), General Requirements for the Competence of Testing and Calibration Laboratories for individual analytical and sample preparation methods. None of these other labs are completely accredited, but Swastika, Expert and Polymet do have certificates of laboratory proficiency issued by the Standards Council of Canada and participate in the Proficiency Testing Program for Mineral Analysis Laboratories ("PTP-MAL") round robin assaying for gold and other elements operated by the Canada Centre for Mineral and Energy Technology, Natural Resources Canada.

For Queenston's programs prior to 2006, there were no field-inserted Standards and/or Blanks. For its 2006 to 2011 programs, field-inserted Certified Reference Standards and Blanks supplemented Swastika's internal Quality Assurance / Quality Control ("QA/QC") programs on Blanks and Standards (Table 11.1).

TABLE 11.1 SUMMARY OF ASSAY METHODS	
Sample Type	**Number of Assays**
Routine Au Sample Assays	46,639
Metallic Screen Assays	1,456
Assays of Field-inserted Blanks	1,130
Assays of Field-inserted Gold Assay Control Certified Reference Standards	1,318
Secondary Lab Gold Check Assays (pulps and rejects)	1,624
Secondary Lab Copper Check Assays (pulps and rejects)	42

In addition to the details in Table 11.1, as aforementioned, Swastika, SGS and Expert's internal QA/QC procedures call for the insertion of Blanks and Standards. This data has been compiled by Queenston. The Secondary laboratories also conduct internal QA/QC programs involving insertion of Blanks and Standards.

11.4 ROUTINE ASSAYING AND TESTWORK

At Swastika, all samples were assayed for gold by fire assay using a 1 assay ton charge and for copper using Atomic Absorption spectroscopy ("AAS"). Routine sample preparation includes sample drying, crushing to 6 to 10 mesh, and splitting out a 400 g sub-sample using a Jones Riffler. The excess is stored as a reject. The 400 g sub-sample is pulverized using a ring and puck pulverizer for sufficient time enabling 90–95% of the material to pass through a 100 mesh screen. The sample is then blended and mixed well.

For gold analysis by fire assay, a charge of 29.17 g is obtained by sub-sampling. Assay finish is routinely by AAS but some samples go directly to gravimetric ("GRAV") finish after cupellation based on visual assessment of the bead. For copper assay, digest is by aqua regia (nitric and hydrochloric acids) in a hot water bath until the pulp is all dissolved.

Samples that on initial assay return results greater than 1 g Au/t are re-assayed using a new pulp from the 6-10 mesh reject. These assays are then finished gravimetrically.

Samples that on initial assay return greater than 1% Cu are re-assayed using a smaller charge of sample.

Swastika procedures call for:

- Cleaning the crushers with compressed air after each sample pass. Barren material is crushed subsequent to each customer run to minimize sample contamination;
- Compressed air is used to clean the riffle divider after the final split of each sample;
- Compressed air is used to clean the bowl, ring, puck and rubber mat after each sample is pulverized; and
- A screen test is performed on pulverized samples at the beginning of each shift, or more frequently when material hardness is in question, to ensure particle size remains within prescribed limits.

During the period of 2005 to 2009 (up to an including Hole UB09_147), the final gold assay in the database is the metallic screen assay (see below), where such assays were completed. Where gravimetric gold fire assays and AAS finished assays were both completed on a sample, Gold-Final was the average of both AAS and gravimetric finished assays. Where two AAS finished gold assays are completed on the same pulp, the average result of the two assays is Gold-Final. From 2009 onward, the practice of averaging gold assays was abandoned and the first gold value reported, whether it was ASS and GRAV finished, was the value used. Beginning in 2011 if there was a gold value reported for both AAS and GRAV, then the GRAV value would be used in the Gold-Final column. Check assays completed at the Secondary labs are not used in the calculations of final assays for the assay database used for the Mineral Resource estimate. Where a second copper assay is completed, Copper-Final is the second assay determined using the higher reporting limit. Initial copper assays in such cases are expressed as >10,000 ppm and therefore are not averaged in. No copper repeat assays are done if initial results are less than 10,000 ppm.

11.5 ADDITIONAL ASSAYING

A total of 1,180 samples from the 2005-2006 drill programs, in addition to routine assaying, were re-assayed by the screened pulp metallic method Metallic screen assaying is an assaying strategy used to help mitigate the effects of coarse gold towards obtaining more representative assays.

The samples for metallic screen assaying were selected using a variety of criteria. For programs up to the end of 2006, all samples within designated mineralized zones were sent for metallic screen assaying. Early in the program, samples with visible gold were also sent for screen assaying. A number of samples were also selected based on initial high copper assays. Metallic screen assaying was discontinued after the 2006 program, except for one sample that was Check Assayed in 2007. Swastika's metallic screen assaying procedure entails crushing and pulverizing the entire reject sample and dry screening at 100 mesh. The +100 mesh (coarse) fraction is weighed, fire assayed using a gravimetric finish. The -100 mesh (fine) fraction is also fire assayed using a gravimetric finish and a 1 assay ton charge. The gold content for the original samples is calculated using the weighted average assay results for the coarse and fine fractions.

Results for metallic screen fire assays compared to routine fire assays are shown in Figure 11.1 and Figure 11.2, and Table 11.2.

TABLE 11.2
SUMMARY STATISTICS FOR METALLIC SCREEN AND ROUTINE FIRE ASSAY PAIRS

Description	Number
Count of Samples	1,180
Average Original Regular Fire Assay (g Au/t)	2.779
Average Metallic Screen Fire Assay (g Au/t)	2.906
% Difference Between Averages	4.47

Figure 11.1 Comparison of Metallic Screen Assays to Original Regular / Routine Fire Assays

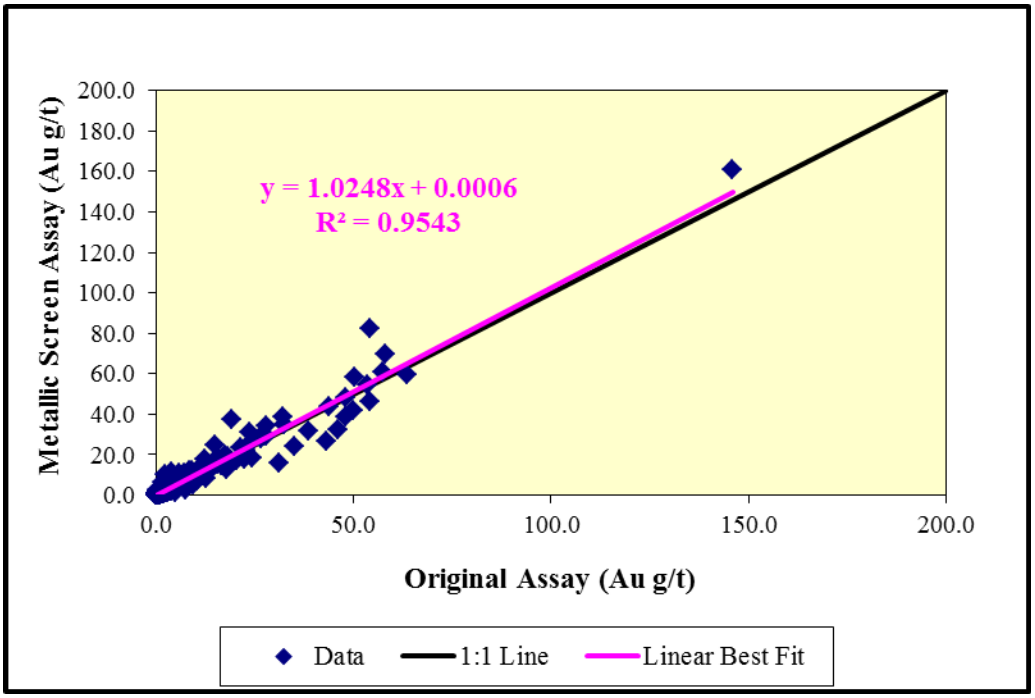
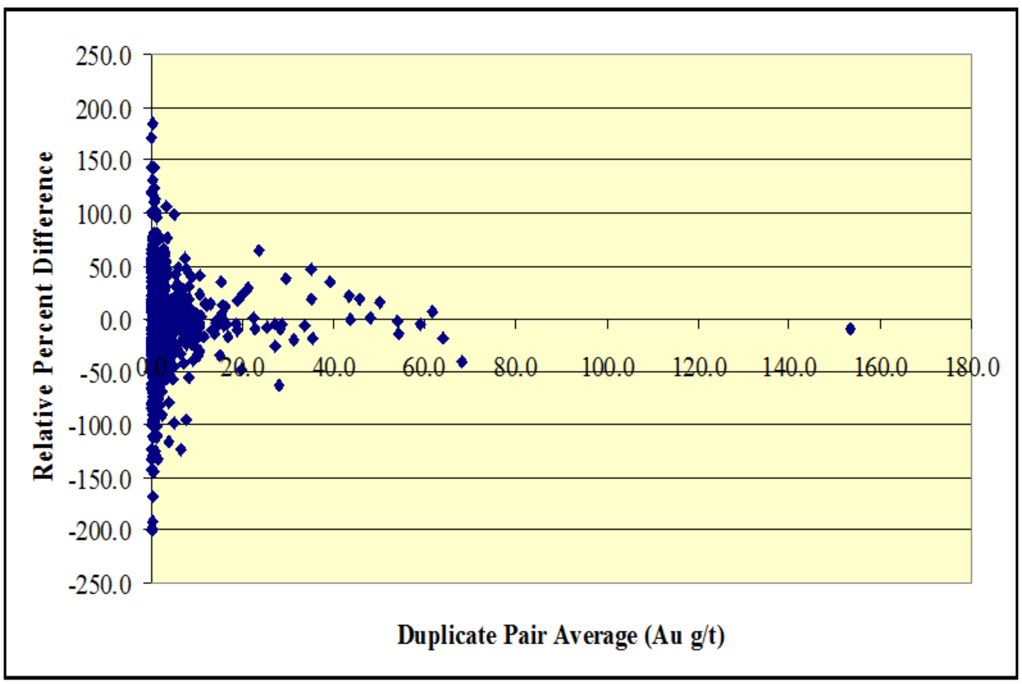


Figure 11.2 Relative Percent Difference Plot for Metallic Screen Fire Assays vs. Original Regular / Routine Fire Assays



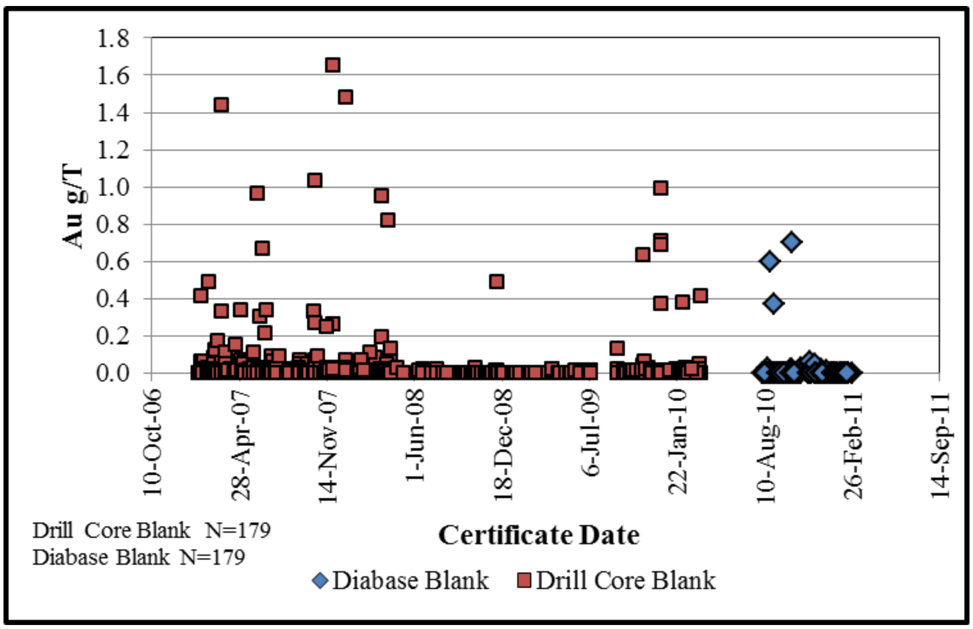
11.6 QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

QA/QC for assays includes components initiated by Queenston and also components conducted by its Primary and Secondary assay laboratories. Swastika is Queenston's Primary assay laboratory. Secondary assay laboratories used for periodic check assaying of sub-samples previously assayed at Swastika have included Polymet, Expert and SGS. These laboratories carry out their own internal QA/QC programs consisting of the insertion of Blanks, and Certified Reference Standards into the sample stream.

11.6.1 Queenston's in-field QA/QC Protocol

Starting with the infill definition drilling program in January 2007 (Hole UB07-75), Queenston's QA/QC program was implemented. Queenston initiated insertion of Certified Gold Reference Standards and Blanks into the sample stream at frequencies of one control sample every 25th regular/routine sample. Blank samples were drill core of un-mineralized basalt and interflow sediments from a previous Queenston drill program and starting in late 2009, barren diabase from the Upper Beaver was used. These Blanks were also inserted following samples containing coarse visible gold for the purpose of determining if there was any contamination between samples. A value of 200 ppb Au was designated by Queenston as the upper limit threshold for separating anomalous from non-anomalous values. This value may have been selected based on previous experience with similar samples. Figure 11.3 shows assay results for field-inserted Blanks since start of program in early 2007.

Figure 11.3 Gold Assay Results for Field-Inserted Blanks 2007 to 2011



The Certified Reference Standards for gold control were purchased from Rocklabs Ltd. ("Rocklabs") Auckland, New Zealand. Eight different Standards have been used since 2007. The certified copper-molybdenum ore reference material (HV-2) was purchased from CANMET. These control samples were inserted in the field by the sampler as requested by the core logging geologist. About 50 g was scooped from the supplier's container and placed in a sample bag. The sample bags were numbered in accordance with the routine sampling scheme. The identity of the control material was not provided to Swastika.

Figure 11.4 shows results Queenston's eight field-inserted Au Standards since program reception. Table 11.3 summarizes statistical results. Figure 11.15 shows results for the copper field inserted Standard HV-2 and Table 11.4 summarizes statistical results.

Figure 11.4 Gold Assay Results for Field-Inserted Certified Reference Standards 2007 to 2011

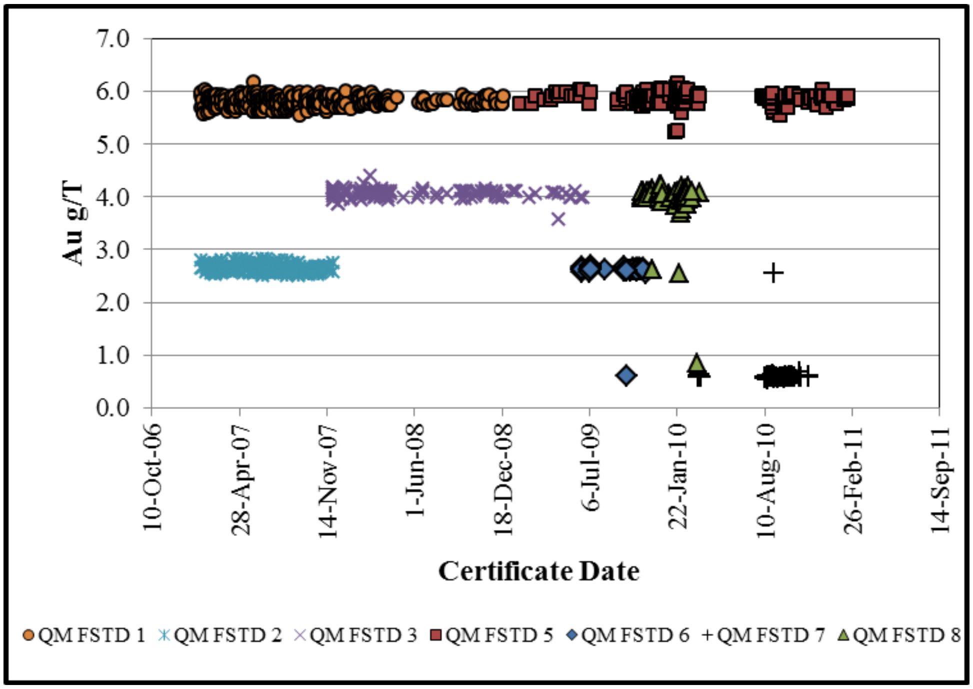


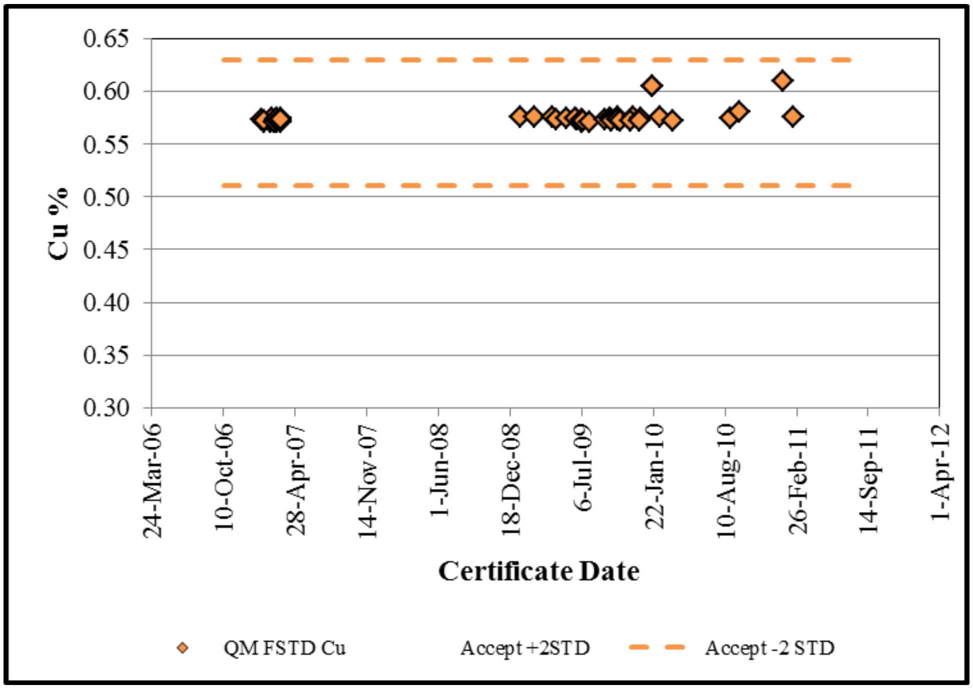
TABLE 11.3
STATISTICAL SUMMARY FOR GOLD ASSAYS FOR FIELD-INSERTED CERTIFIED REFERENCE STANDARDS

Queenston Standard ID	Standard	Provider	Certified Value (g Au/t)	95% Cofid	Standard Deviation	Count	Avg (g Au/t)	Median (g Au/t)	Min (g Au/t)	Max (g Au/t)	Date Usage
QM Std 1	OxL51	Rocklabs	5.85	0.051	0.123	317	5.80	5.82	5.53	6.17	Feb 2007 to Dec 2008
QM Std 2	SJ32	Rocklabs	2.645	0.027	0.068	192	2.64	2.63	2.51	2.81	Sept 2007 to Nov 2007
QM Std 3	SK33	Rocklabs	4.041	0.041	0.103	119	4.05	4.05	3.57	4.39	Nov 07 to Jun 09
QM Std 5	SL46	Rocklabs	5.867	0.066	0.17	206	5.80	5.89	0.58	6.15	Jan 09 to Feb 11
QM Std 6	SJ39	Rocklabs	2.641	0.033	0.083	38	2.57	2.61	0.60	2.69	Jun 09 to Nov 09
QM Std 7	OxE74	Rocklabs	0.615	0.006	0.017	50	0.73	0.59	0.54	5.75	Aug 10 to Nov 10
QM Std 8	SK43	Rocklabs	4.086	0.036	0.093	81	3.93	4.00	0.85	4.23	Nov 09 to Mar 10
QM Std 11	SH41	Rocklabs	1.344	0.015	0.041	40	1.3	1.3	1.21	1.38	Dec 10 to Feb 11

TABLE 11.4
STATISTICAL SUMMARY FOR COPPER ASSAYS FOR FIELD-INSERTED CERTIFIED REFERENCE STANDARDS

StandardID	Standard	Provider	Certified Value (% Cu)	95% Confid	Standard Deviation	Count (%Cu)	Avg (%Cu)	Median (%Cu)	Min (%Cu)	Max (%Cu)	Date Usage
QM Cu Std (%)	HV-2	CANMET	0.57	0.02	0.03	49	0.57	0.57	0.57	0.57	Feb 07 to Feb 11

Figure 11.5 Copper Assay Results for Field-Inserted Certified Reference Standards 2007 to 2011



No re-assaying was done by Queenston on the basis of the results for field-inserted Blanks and Standards, however, a re-assaying of all samples on certificate 10-1037 (19/04/2011) was requested by Queenston due to a discrepancy between the digital file and the signed hard copy from the lab, as well as a >20g difference between several Au values due to a nugget effect.

11.6.2 Swastika's Internal QA/QC Protocol

Swastika's lab internal QAQC protocol includes analytical duplicates and assaying of Certified Reference Standards. Figure 11.6 shows the results for Certified Reference Standards for gold and Table 11.5 summarizes statistical results for these Standards versus certified values.

Figure 11.6 Gold Assay Results for Swastika-Inserted Certified Reference Standards 2005 to 2011

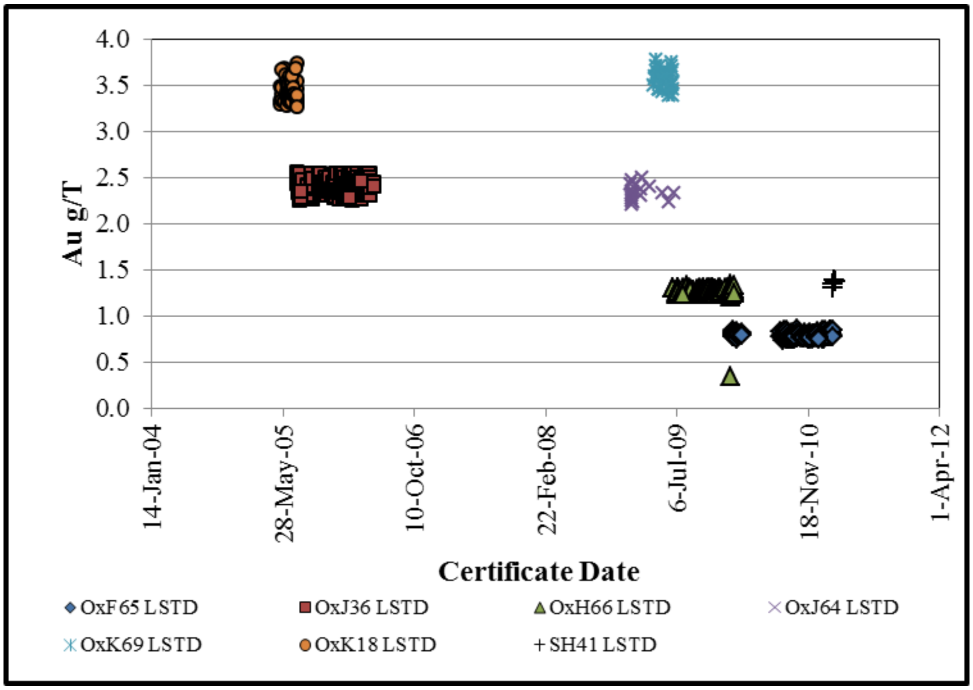


TABLE 11.5 STATISTICAL SUMMARY FOR GOLD ASSAYS FOR SWASTIKA-INSERTED CERTIFIED REFERENCE STANDARDS										
Standard	**Provider**	**Certified Value (g Au/t)**	**95% Cofid**	**Standard Deviation**	**Count**	**Avg (g Au/t)**	**Median (g Au/t)**	**Min (g Au/t)**	**Max (g Au/t)**	**Date Usage**
OxF65	Rocklabs	0.805	0.014	0.034	277	0.79	0.79	0.74	0.86	Feb 10 to Feb 11
OxJ36	Rocklabs	2.398	0.031	0.073	487	2.39	2.39	2.24	2.55	Jul 05 to May 06
OxJ64	Rocklabs	2.366	0.031	0.079	17	2.34	2.33	2.21	2.50	Jan 09 to Jun 09
OxK69	Rocklabs	3.583	0.033	0.086	35	3.57	3.60	3.39	3.77	Apr 09 to Jun 09
OxH66	Rocklabs	1.285	0.012	0.032	274	1.27	1.27	0.35	1.34	Jun 09 to Feb 10
OxK18	Rocklabs	3.463	0.058	0.132	57	3.43	3.40	3.26	3.73	May 05 to Jul 05
SH41	Rocklabs	1.344	0.015	0.041	5	1.35	1.34	1.30	1.39	Feb-11

Queenston has not compiled similar results for copper so this data is not available for WGM's review. Assay results for Analytical Duplicates and re-assays with gravimetric finish versus AAS finish have also not been compiled and have not been reviewed by WGM.

The Secondary assay laboratories also use Certified Reference Standards, Blanks and Duplicates but WGM has not reviewed this data.

11.6.3 Check Assay Program

Selected pulps and rejects from mineralized intervals were pulled from the initial sample populations approximately every two to three months starting with the definition drilling program in 2007 for Check Assaying at a Secondary laboratory. From 2007 through the 2008 program, Queenston used Polymet and Expert for Check assaying a selection of samples originally assayed by Swastika. From 2009 into 2011, Expert and SGS are being used for check assaying. Queenston's aim was to complete Check assaying on 5% of rejects and 5% of pulps from the gold-copper mineralized zones. Rejects were bagged in larger plastic bags, sealed and labelled. Pulps were placed in cardboard boxes sealed and labelled. Sample numbers remained the same as the original sample numbers. The rejects and pulps were delivered by truck by a Queenston employee to the Secondary lab or picked up by a laboratory vehicle. After results were received from the Secondary labs, the pulps and rejects were picked up and returned to the storage containers at the Upper Canada mine site.

Table 11.6 and Table 11.7, and Figure 11.7 through Figure 11.12 illustrate and summarize pre-2008 Check assaying results for gold and copper for pulps and rejects at Polymet and Expert.

TABLE 11.6
SUMMARY STATISTICS FOR GOLD CHECK ASSAYS PRE 2008

Description	Polymet Pulps	Expert Pulps	Polymet Rejects	Expert Rejects
Count of Samples	93	164	123	247
Average Swastika Original Assays (g Au/t)	12.017	10.416	7.684	7.502
Average Check Assays (g Au/t)	12.200	11.257	7.541	7.247
% Difference Between Averages	1.52	7.76	1.88	3.46

Similar to the Check assaying completed to verify gold values, selected rejects and pulps were also check assayed at Polymet and Expert for copper (Table 11.7).

TABLE 11.7
SUMMARY STATISTICS FOR COPPER CHECK ASSAYS PRE 2008

Description	Polymet Pulps	Expert Pulps	Polymet Rejects	Expert Rejects
Count of Samples	93	161	123	247
Average of Swastika Original Assays (% Cu)	0.479	0.501	0.406	0.730
Average of Check Assays (% Cu)	0.453	0.569	0.381	0.843
% Difference Between Averages	5.68	12.61	6.47	14.42

Figure 11.7 Polymet Gold Assay of Duplicate Pulp vs. Original Swastika Assay

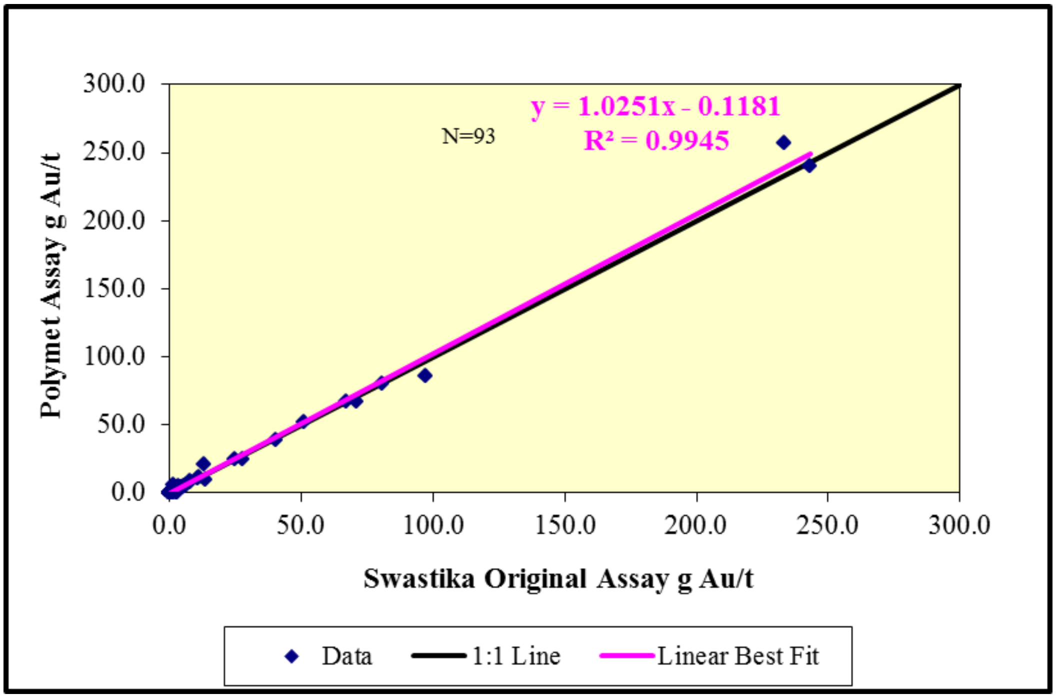


Figure 11.8 Relative Percentage Difference Chart For Swastika And Polymet Gold Assays on Duplicate Pulps

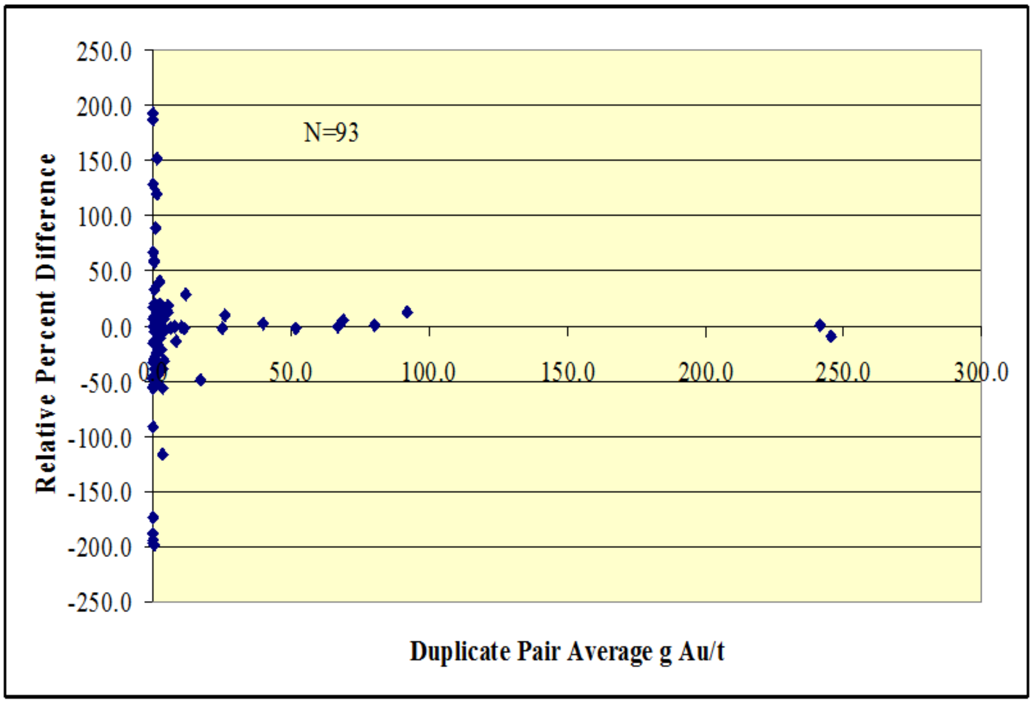


Figure 11.9 Expert Gold Assay of Duplicate Pulp vs. Original Swastika Assay (Truncated Distribution)

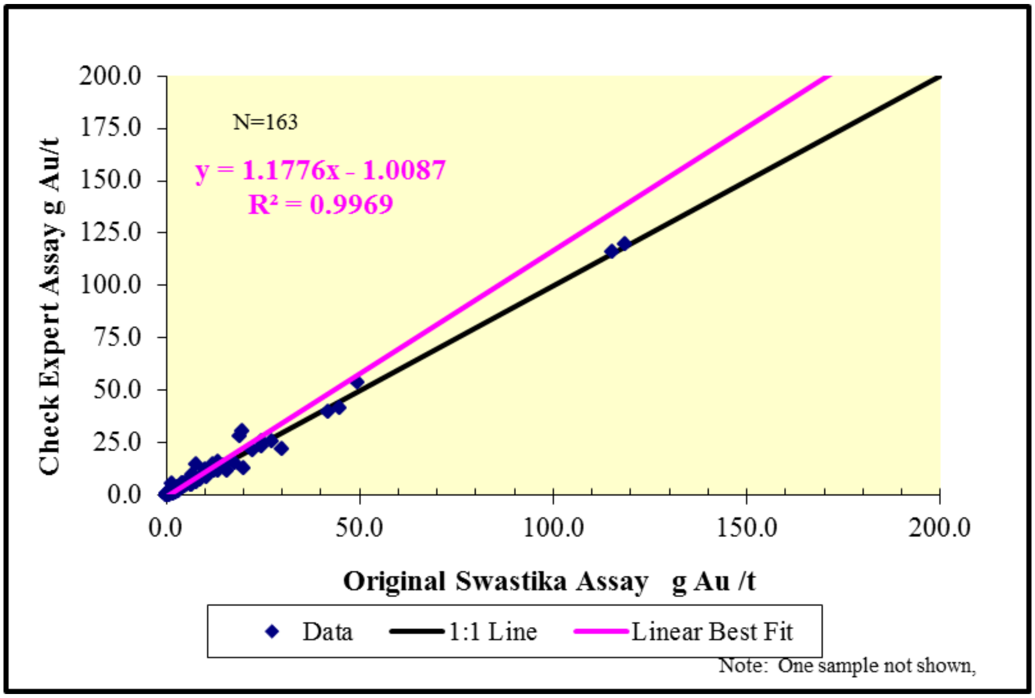


Figure 11.10 Relative Percentage Difference Chart for Swastika And Expert Gold Assays on Duplicate Pulps (Truncated Distribution)

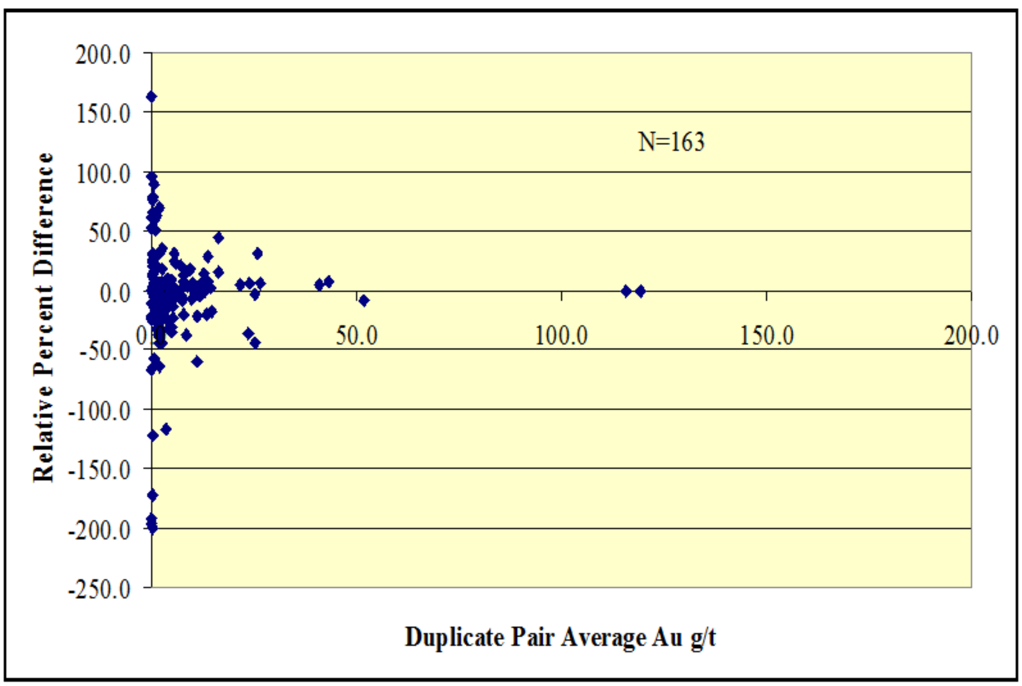


Figure 11.11 Polymet Check Copper Assays vs. Original Swastika Assays on Same Pulps

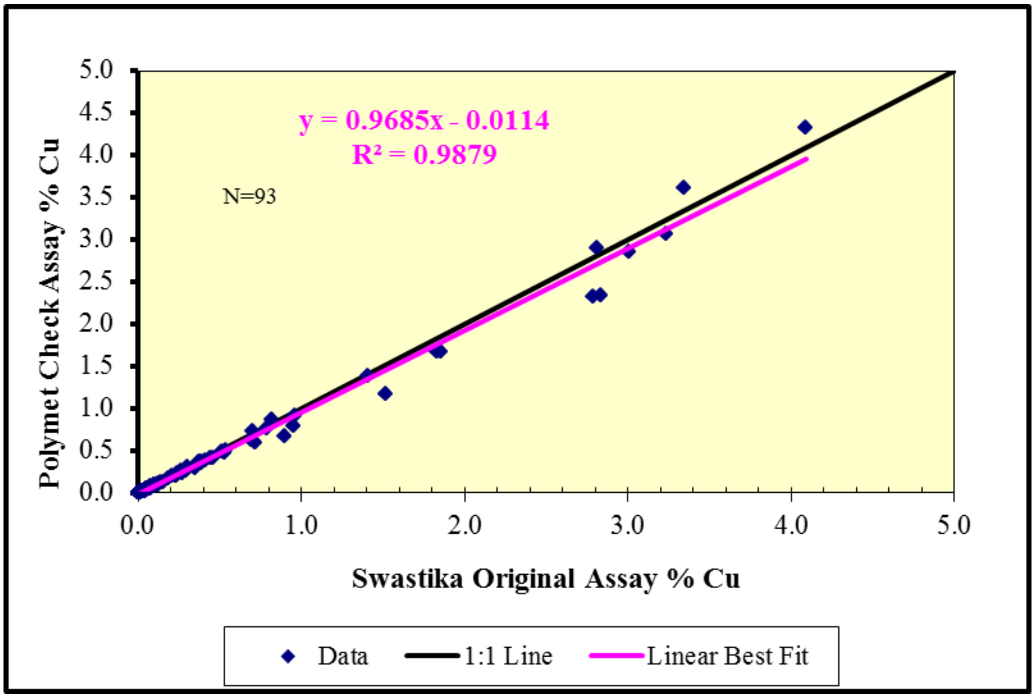
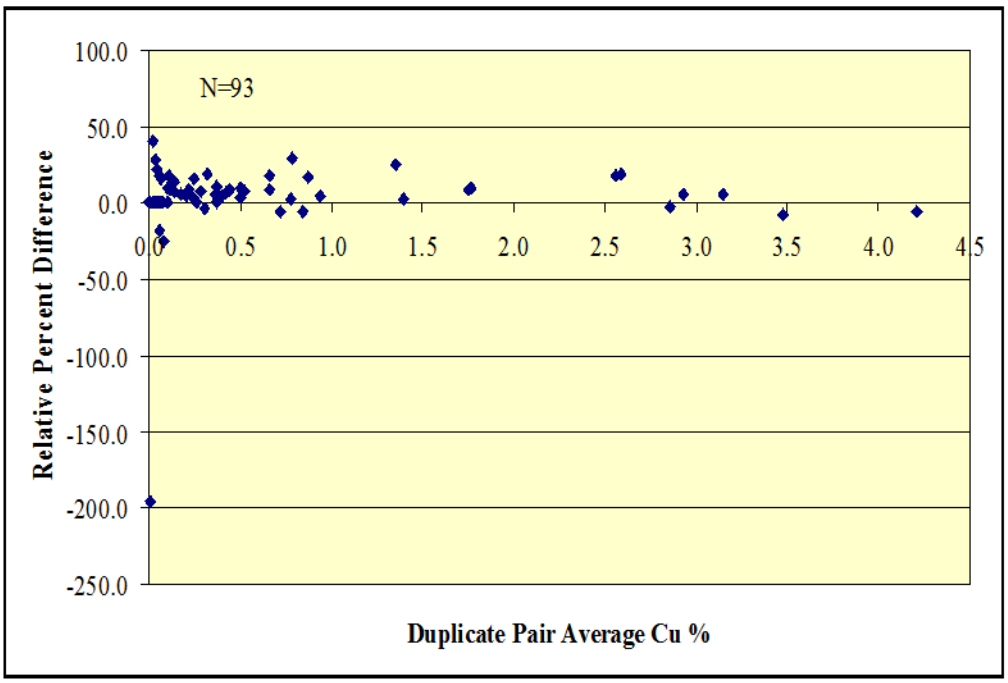


Figure 11.12 Relative Percent Difference Chart for Polymet Check Assays and Original Swastika Assays on Same Pulps



In 2009, check assaying for 2008 and 2009 drillhole samples was completed at SGS. Results are shown on Figure 11.13 and Figure 11.14 and summarized in Table 11.8 however, not all check assaying results are shown due to incomplete compilation of data.

Figure 11.13 Gold Check Assay of Rejects by SGS vs. Original Assays by Swastika

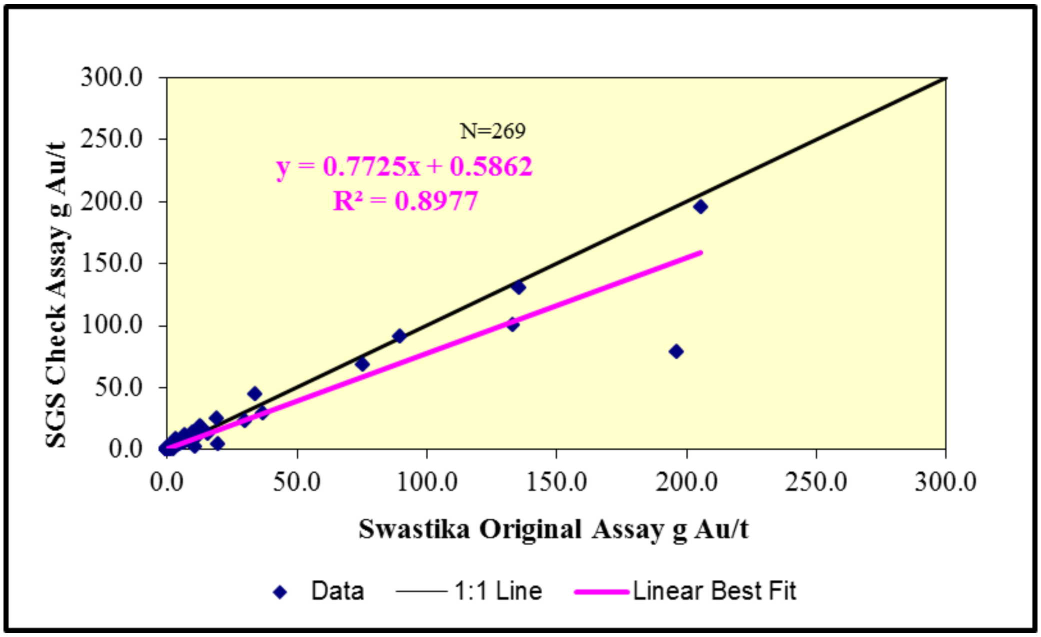


Figure 11.14 Relative Percent Difference Chart for SGS Check Gold Assays and Original Swastika Assays on New Pulps

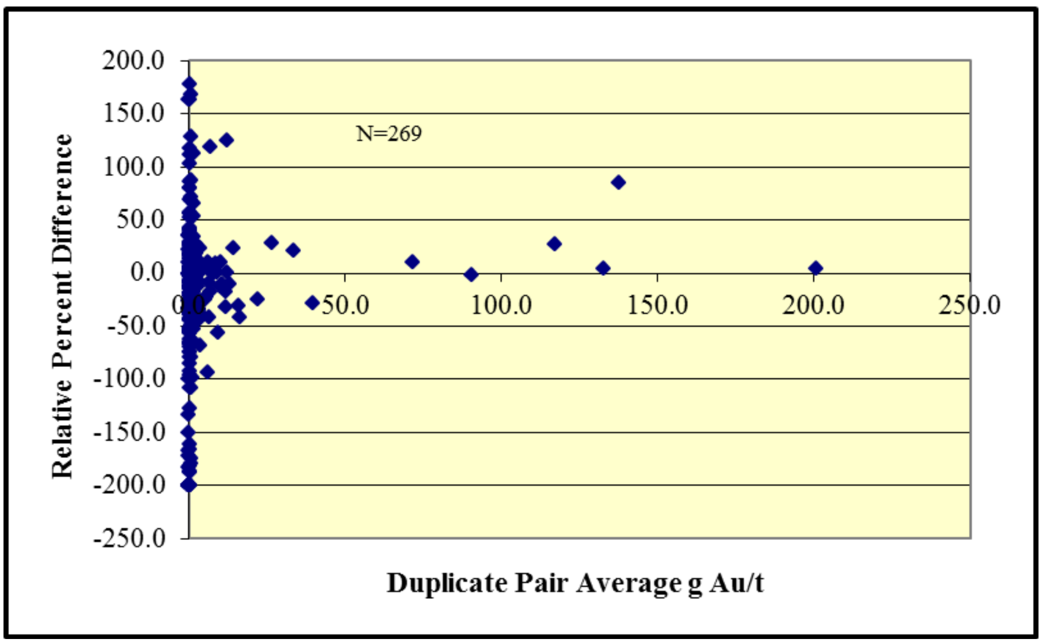


TABLE 11.8 **SUMMARY STATISTICS FOR SGS CHECK ASSAYING OF REJECTS FOR GOLD**	
Count of Samples	269
Average Assay Swastika	4.951
Average Assay SGS	4.394
% Difference Between Averages	-11.91

In early 2011, approximately 700 pulps for samples originally assayed at Swastika in 2009 and 2010 were Check Assayed at Expert. Results for gold check assaying are shown on Figure 11.15 and Figure 11.16. Table 11.9 summarizes assay results for gold.

Figure 11.15 Gold Check Assays of Pulps by Expert vs. Original Assays by Swastika

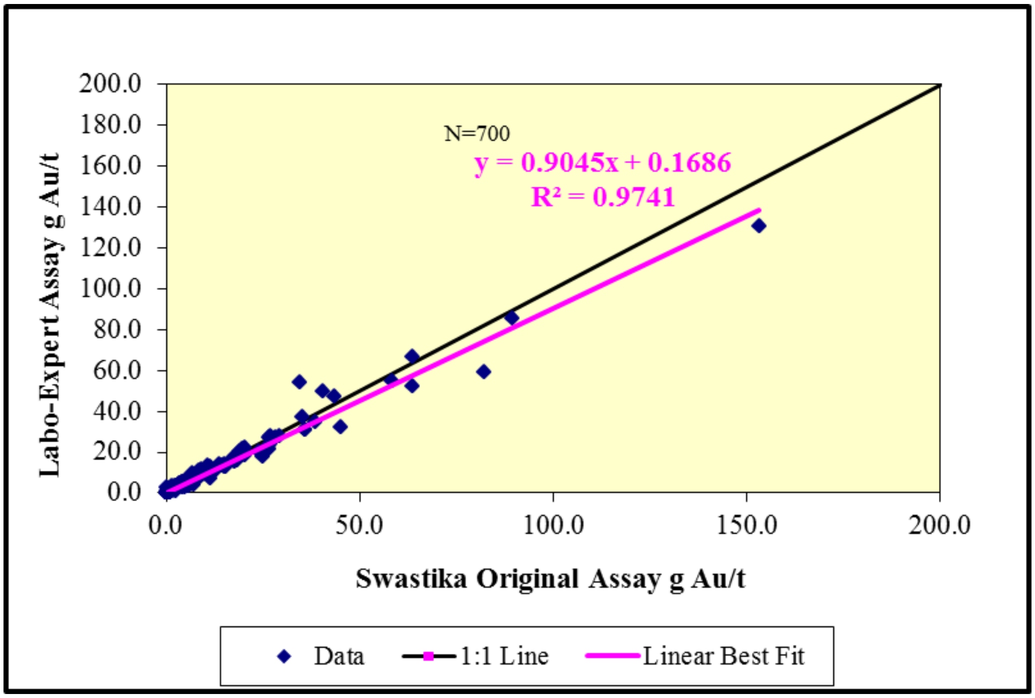
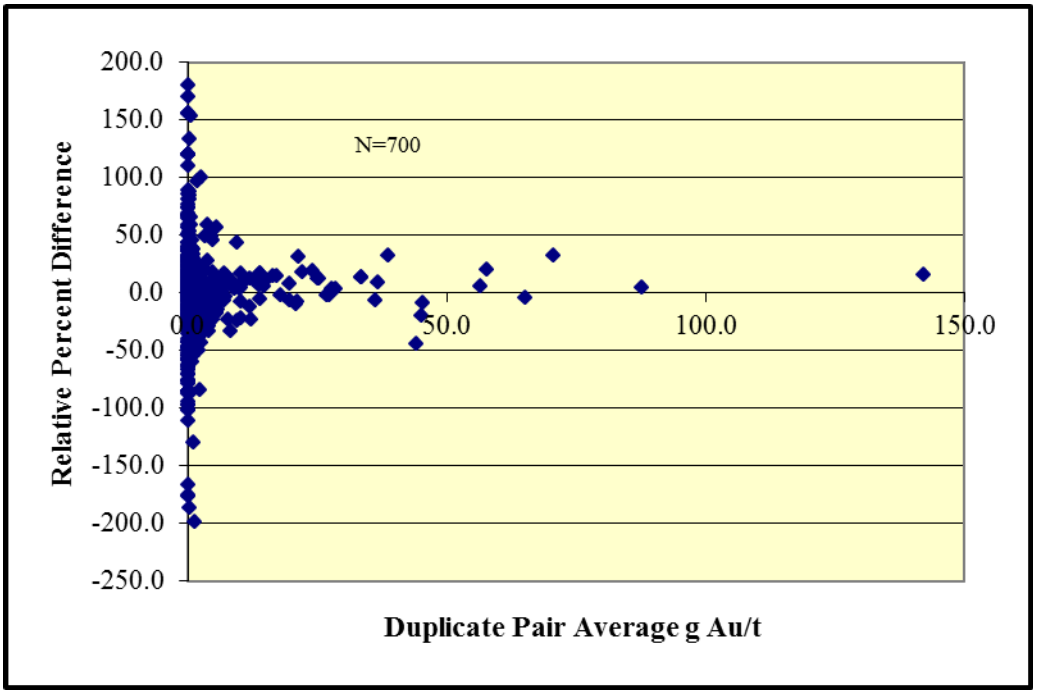


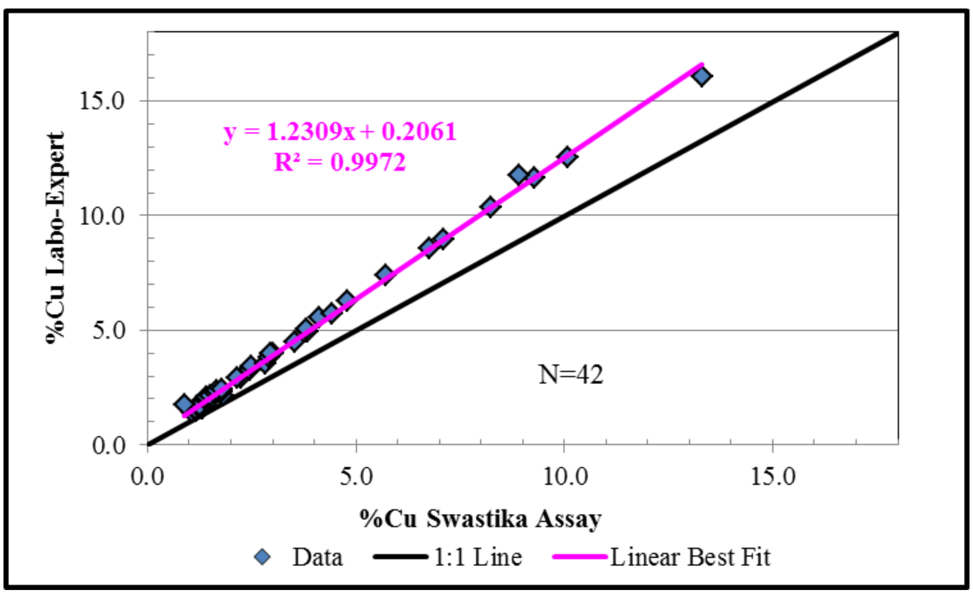
TABLE 11.9 SUMMARY STATISTICS FOR EXPERT CHECK ASSAYING FOR GOLD ON PULPS	
Count of Samples	700
Average Assay Swastika	2.986
Average Assay Labo-Expert	2.870
% Difference Between Averages	-3.96

Figure 11.16 Relative Percent Difference Chart for Expert Check Gold Assays and Original Swastika Assays on Same Pulps



Forty-two of these 700 samples were analysed for copper. Figure 11.17 shows the copper check assay results.

Figure 11.17 Copper Check Assays of Pulps by Expert vs. Original Assays by Swastika



11.7 SAMPLE SHIPPING AND SECURITY

Samples are delivered by truck to the Swastika Laboratories Ltd. The Upper Canada mine site, where the core is stored and the Queenston office is located, is surrounded by fences and locked gates are in place at all road access points to the site.

WGM agrees that Queenston's current sampling, assaying and QA/QC protocols represent good industry practice. Analytical results for Certified Reference Standards inserted by Queenston into the sample stream in the field and Check assaying completed at its Secondary labs indicates Swastika laboratory results are, in general, reasonably accurate and precise and suitable for the purposes of a Mineral Resource estimate. Metallic screen assaying indicated no evident significant bias between routine fire assaying and metallic screen assaying. The sample and assay database is currently inadequate for the project as assay information is not readily accessible for review, validation and auditing.

Check assaying results between labs are generally mixed indicating Swastika assays are generally reasonably reliable. For Check gold assaying of pre-2008 samples, two of the four sets of Secondary lab Checks, (Polymet rejects, Expert rejects) returned slightly lower assay averages than Swastika originals, while the pulp Checks returned slightly higher average assay than Swastika originals. SGS Check gold assays of 2008 program rejects were generally lower than original Swastika assays. The Expert Check assay results for 2009 and 2010 sample pulps were also slightly lower than Swastika original assays, but bias is minimal and a high degree of correlation exists between original and Check assays. Check assaying by Expert in 2011 returned significantly higher copper assays than Swastika. Earlier copper Check assaying returned mixed results. Copper assay results by Swastika used in the Mineral resource estimate are therefore probably conservative.

Assay results for field-inserted and lab-inserted Blanks and Standards also indicate accurate assaying. A few of the field and lab inserted Standards report erroneous values likely due to sample mix-ups in the field. Apparently, however, anomalous results are not followed up.

WGM recommends that Queenston needs to improve its sampling and assaying database and data handling capabilities. It should compile all of its assay records. Presently it is difficult to audit and process assay data except on a one by one basis as all assay records are not compiled. A relational database is ideal to accomplish these purposes and to enhance data review and validation. The databases should include all assays, not just the Finals computed from component assays and also include assays at Secondary labs. The database also should include results for all QA/QC materials both for Queenston inserted materials and also laboratory inserted materials. Tables should also contain results for specific gravity measurements. Basic tables for a relational sample and assay database are: Sample table listing and classifying all samples by type and location, various Assay tables (perhaps one for the Primary lab and one each for the Secondary labs) and a Certificate table listing all certificates, by laboratory and date of issue.

Queenston should also strive to avoid repeating sample numbers, as sample number repeats complicate tracing assays to certificates and archived core and processing and interpreting results. Towards building an assay database in a relational database system historic sample it is very advantageous for Sample IDs to be unique. One possible strategy to accomplish this would be to add a year prefix to the existing Sample IDs. Where samples have no pre-existing Sample ID, the best policy might be to create one by combining Hole ID and From meterage.

WGM also recommends that Queenston develop a written protocol to specify definition and practice of QA/QC failures. The assay database should also include a table to track QA/QC issues and responses. Going forward some simplification of assay certificates may be possible to simplify data entry.

12.0 DATA VERIFICATION

This section was completed by Richard Risto of WGM

Statements below are current as at the release of WGM's Technical Report, June 15, 2011. Work completed on the Property subsequent to this date has not been reviewed, nor has it been determined if said work has any material effect on WGM's conclusions as at the date of its report.

Information relating to past work on the Property was primarily obtained from Queenston. On March 30, 2011, WGM Resource Engineer and Vice-President, Marketing, Kurt Breede, P.Eng. and Qualified Person, visited the Property, the Queenston field office and core storage facilities at the old Upper Beaver mine site. During WGM's original site visits in 2007 and 2008, the old shaft areas, trenches/pits and some old showing areas were visited. Several drills were in operation on the Upper Beaver Property during WGM's recent site visit, of which one was visited, along with several newer drillhole collars locations which were located with a GPS instrument.

Discussions were held with Mark Masson, project lead for Queenston, Christal Hanuszczak, and Manuel NgLai, Project Engineer. WGM observed that logging and sampling procedures were meticulous and "general housekeeping" at the site, core shack and field office was very good. While at the site, WGM reviewed numerous intersections of drillholes completed by Queenston throughout the various newer phases of drilling. Drill core was examined and compared with drill log descriptions and representations on drill cross sections.

12.1 DATABASE VALIDATION

Prior to the site visit and Mineral Resource estimation, WGM carried out an internal validation of the drill holes in the digital drill hole database used in this Mineral Resource estimate. Holes were selected for validation according to the following criteria:

- Distribution in the various zones;
- Representative selection based on the drilling year; and
- Grade distribution.

As listed in Table 12.1, a total of 6 holes were selected for validation. Laboratory assay certificates were requested from Swastika and received in digital PDF and Microsoft Excel format, and values matched against the database entry.

No discrepancy was noted.

12.2 COLLAR COORDINATE VALIDATION

Collar coordinates for 4 holes selected for validation were checked against the printed drill logs with no discrepancy noted. These same 4 collars were validated in the field with the aid of a hand-held GPS. Collar stakes were generally wooden with spray painted and flagged tips, although newer holes have been marked with stronger steel stakes. It was noted that some areas in proximity to the current and historical drill collars were designated for foresting.

As shown on Table 12.1, results indicated an average difference in the X-Y plane of ± 5.0 m for the 4 hole collars where the instrument was located near the top of the casing. The calculated differences in the X-Y plane are in close agreement to Queenston's reported measurements and drill hole design considering a conversion was applied to the hand held GPS coordinate to record the information in mine grid.

TABLE 12.1
WGM DRILL HOLE COLLAR FIELD VERIFICATION

Hole-ID	WGM (UTM)		Queenston (UTM)		±	
	North	**East**	**North**	**East**	**North**	**East**
UB10-165	591747	5336458	591745	5336454	2	4
UB08-135	591622	5336452	591627	5336449	-5	3
UB09-142A	591700	5336056	591695	5336045	5	11
UB09-100W1	591731	5336130	591734	5336120	-3	10

12.3 ASSAY VALIDATION

Assay validation was undertaken by comparing entries in the GEMS database entry against the laboratory certificate from the signed PDF copy and certificate in XLS spreadsheet format, for a total of 6 holes. Validation results showed no erroneous data. WGM regards the sampling, sample preparation, security, and assay procedures as adequate to form the basis of the Mineral Resource estimation.

Twelve independent samples of mineralized split drill core (the remaining half) were taken for check assaying. They were bagged, sealed on site and were transported personally by car to WGM's Toronto office by Mr. Breede. On arrival at the office, the samples were boxed up and couriered to the SGS Mineral Services Inc. ("SGS") ISO 9001:2000 accredited laboratory in Toronto for independent assaying. The samples were analyzed for Au and Cu using a similar analysis package offered by Swastika to Queenston (SGS codes FAI515 and FAG505 for Au and ICP90Q for Cu), however, WGM decided to use a 50 g sample size instead of one assay ton due to the high grade nature of the mineralization being tested. Please see previous sections in this report for a more complete description of the Swastika analytical procedures.

The WGM samples were taken as characterization samples to confirm that gold and copper was present and the general nature/tenure of the mineralization. All samples returned gold/copper values and our sampling results, along with those of the original Queenston assays for the same intervals, are shown in Table 12.2.

TABLE 12.2 WGM INDEPENDENT SAMPLING RESULTS						
Sample Number	**Hole-ID**	**Original Au (Queenston) (g Au/t)**	**WGM Au FAI515 (ppb)**	**WGM Au FAG505 (g Au/t)**	**Original Cu (Queenston) (ppm)**	**WGM Cu ICP90Q (ppm)**
8784	UB09-148W1	8.64	6,050	N.A.	0.61	0.76
8785	UB09-148W1	10.79	>10,000	10	0.57	0.35
8786	UB010-148W4	11.1	>10,000	14	0.28	0.22
8787	UB010-148W4	48.97	>10,000	39	1.76	1.43
8788	UB010-161W5	9.94	8,850	N.A.	0.52	0.21
8789	UB010-161W5	43.16	>10,000	65	2.21	2.16
8790	UB010-139W3	5.31	768	N.A.	0	0.02
8791	UB010-139W3	8.23	>10,000	11	0.58	0.49
8792	UB09-143W1	9.63	>10,000	9	1.44	1.13
8793	UB09-143W1	4.46	3,750	N.A.	3.01	2.19
8794	UB09-143W1	5.11	5,670	N.A.	0	0.01
8795	UB09-153	6.55	3,600	N.A.	1.59	1.74

WGM's sampling results generally corroborated those obtained by Queenston. The variance in assays from one half of the core to the other is typical of gold mineralization and, in particular Upper Beaver-style deposit mineralization, where there may be coarse gold particles present.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 GENERAL

Note: *The authors of this section have drawn heavily upon selected portions or excerpts from material contained in the WGM 2011 Report.*

Early testwork on the Deposit was conducted from 1963 to 1965 at Upper Canada Mines and Faraday Mines which yielded high copper and gold recoveries to flotation concentrates. Cyanidation of flotation tailings recovered most of the remaining gold. A 150 ton per day flotation – cyanidation process operated at Upper Canada Mines in 1965 with a total throughput tonnage of 37,277 tons, milled at 12.3 g/t gold and 0.64% copper and yielding recoveries of 90% copper and 93.6 % gold.

13.2 RECENT TESTWORK

13.2.1 Samples

In 2008-9 a limited scoping program of metallurgical tests was conducted by SGS-Lakefield on two Upper Beaver ore composites representing high (H) and low (L) copper-gold grades. This program was followed by additional testwork in 2010 -11 at SGS evaluating a high grade composite sample (UB-MET) from Upper Beaver, as well as samples from other deposits.
The results of the analysis of two samples are presented in Table 13.1, with the current study head values included for comparison.

TABLE 13.1
SGS SAMPLE HEAD GRADES

SGS report	Sample	Au, g/t	Ag, g/t	Cu, %
2009	H	9.64	6.54	1.17
2009	L	4.82	0.8	0.16
2011	UB-MET	28.1	13.9	2.37
Current Study		6		0.44

13.2.2 Grinding

Two Bond ball mill index measurements were made with one for each report. The “H” sample yielded an index of 17.0 kWh/t and the UB-MET sample reetuned14.1kWh/t. It was noted that the “L” sample took twice as long to grind as did the “H” sample, suggesting that rock hardness increases with decreasing grade. The highest measured value of 17 kWh/t is selected for this study.

13.2.3 Gravity Separation

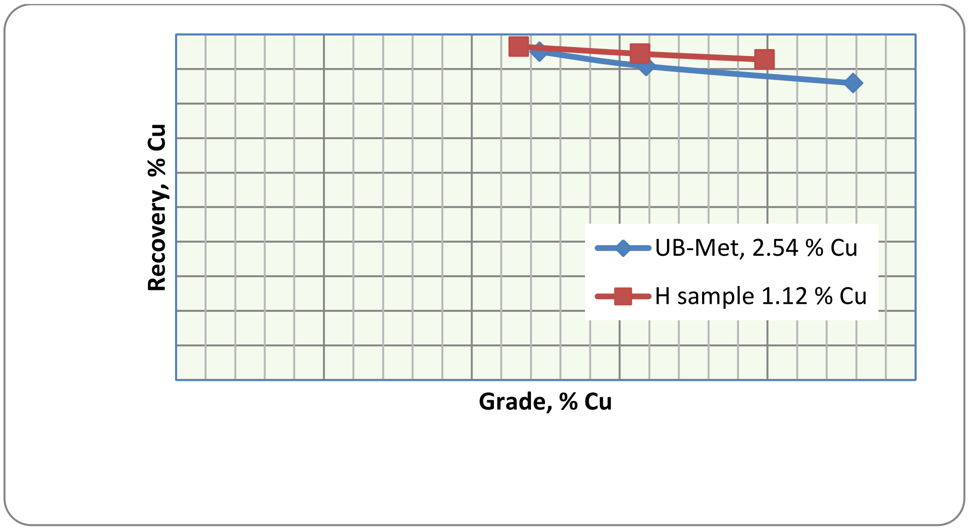
Gravity separation was conducted prior to flotation as a matter of course in almost all tests. The results indicate that a gravity concentrate could be produced containing about one third of the gold and a relatively minor percentage of the copper. Although not definitive, there was no indication that coarse gold is present.

One flotation test was conducted on “H” sample without prior gravity concentration. The results indicated no net benefit to gravity concentration in terms of overall gold recovery. Subject to more definitive testwork, gravity concentration is not included in the flowsheet.

13.2.4 Flotation

Copper floats very well as indicated in Figure 13.1, which summarizes the results of cleaner flotation tests at a primary grind of approximately 70 microns. (The slightly inferior result for the higher grade sample was attributed to minor sample oxidation).

Figure 13.1 Copper Recovery vs. Grade



At these high grades, there should be no difficulty in producing a saleable concentrate. The rougher flotation tests on the “L” sample at a head grade of 0.15% Cu suggest that it may be difficult to produce a saleable grade from that material.

Production from the Upper Canada mill treating Upper Beaver mineralization by flotation, yielded 90% recovery to a 23.3% Copper concentrate from a head grade of 0.64 % Cu, which is close to the current planned mine grade. These data are generally consistent with the SGS results and it is therefore assumed that a copper recovery of 90% to a saleable concentrate can be obtained.

13.2.5 Cyanidation

There are no test data on direct cyanidation of the mineralization in the Deposit and a future test program should include this option.

Cyanidation of flotation tailings was conducted in both SGS testwork programs. Tests at a typical grind of about 70 microns for 72 hours yielded excellent gold extractions. Interpretation is made somewhat difficult by the use of gravity in all tests and the grades of the tested samples but in general overall gold recoveries exceeding 98% were obtained. At the proposed head grade of 6 g/t, and based on residue assay correlation with allowance for soluble losses, an overall gold recovery of 98% is predicted for a circuit comprising flotation of a copper concentrate plus cyanidation of rougher flotation tailings. The amount of gold that reports to the copper concentrate may be very high, at perhaps 80%, but there is potential to modify this through further testwork if economics warrant.

14.0 MINERAL RESOURCE ESTIMATE

Subsections 14.1 through 14.9 were completed by Kurt Breede of WGM.

Statements below are current as at the release of WGM's Technical Report, June 15, 2011. Work completed on the Property subsequent to this date has not been reviewed, nor has it been determined if said work has any material effect on WGM's conclusions as at the date of its report.

14.1 WGM MINERAL RESOURCE ESTIMATE STATEMENT

WGM has prepared an updated Mineral Resource estimate for the Upper Beaver Property mineralized zones that have sufficient data to allow for continuity of geology and grades. A summary of the Mineral Resources is provided in Table 14.1.

TABLE 14.1
SUMMARY OF UPPER BEAVER PROPERTY UPDATED MINERAL RESOURCE ESTIMATE(CUT-OFF OF 2.5 G AU/T)(1-8)

Category	Tonnes	Cu (%)	Au (g/t) (uncapped)	Ounces (uncapped)	Au (g/t) (capped)	Ounces (capped)
Indicated	3,074,000	0.54	8.84	874,000	6.98	690,000
Inferred	3,093,000	0.41	7.15	711,000	6.19	616,000

(1) Mineral Resources are as of June 15, 2011.
(2) Mineral Resources were estimated using a block model. A grade capping factor of 50 g Au/t was applied. A lower cut-off grade of 2.5.0 g Au/t, a minimum horizontal width of 2 m, and a global specific gravity of 2.90 is assumed.
(3) Mineral Resources were estimated using a three-year rolling average of US$1,050/ounce, an exchange rate of US$0.95=C$1.00, and assumed metallurgical recovery of 95%;
(4) 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, socio-political, marketing, or other relevant issues;
(5) The quantity and grade of reported Inferred Mineral 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;
(6) 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 CIM Council December 11, 2005.

The classification of Mineral Resources used in this report conforms with the definitions provided in the final version of NI 43-101, which came into effect on February 1, 2001, as revised on June 30, 2011. We further confirm that, in arriving at our classification, we have followed the guidelines adopted by the Council of the Canadian Institute of Mining Metallurgy and Petroleum ("CIM") Standards. The relevant definitions for the CIM Standards/NI 43-101 are as follows:

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

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

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

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

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

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

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

Mineral Resource classification is based on certainty and continuity of geology and grades. In most deposits, there are areas where the uncertainty is greater than in others. The majority of the time, this is directly related to the drilling density. Areas more densely drilled are usually better known and understood than areas with sparser drilling.

14.2 GENERAL MINERAL RESOURCE ESTIMATION PROCEDURES

The block model Mineral Resource estimate procedure included:

- Importing/compiling and validation of data from Microsoft Excel to Gemcom GEMS v6.2.4 to create a Project database;
- Generation of cross sections and plans to be used for validation of geological interpretations;
- Basic statistical and decile analyses to assess cut-off grades, compositing and cutting (capping) factors;
- Validation of 3-D wireframe models for zones with continuity of geology/mineralization, using available geochemical assays for each drillhole sample interval; and
- Generation of block models for Mineral Resource estimates for each defined zone and categorizing the results according to NI 43-101 and CIM definitions.

14.3 DATABASE

Data used to generate the Mineral Resource estimates originated from Microsoft Excel files supplied to WGM by Queenston via ftp. A GEMS project was established to hold all data and to be used for the manipulations necessary for the Mineral Resource estimate.

The Property drillhole database consisted of 270 drillholes, geological codes, and 52,462 assay intervals for Au and Cu averaging 1m in length. Additional information, including copies of the geological logs, summary reports, mine workings, and geological interpretations were supplied as DXF or similar electronic files.

14.3.1 Data Validation

Upon receipt of the data, WGM performed the following validation steps:

- Checking for location and elevation discrepancies by comparing collar coordinates with the copies of the original drill logs received from the site;
- Checking minimum and maximum values for each quality value field and confirming/modifying those outside of expected ranges;
- Checking for inconsistency in lithological unit terminology and/or gaps in the lithological code;
- Spot checking original assay certificates with information entered in the database; and
- Checking for gaps, overlaps and out of sequence intervals for both assays and lithology tables.

The assay table contained no errors when compared to the original certificates, and were deemed appropriate for use in the subsequent Mineral Resource estimate. Some gaps or missing intervals identified were due to unsampled / unassayed intervals outside of the mineralized zones. WGM found the database to be in good order and accurate and no errors were identified that would have a significant impact on the Mineral Resource estimate.

14.3.2 Database Management

The drillhole data were imported into a GEMS multi-tabled workspace specifically designed to manage collar and interval data. The line work for the geological interpretations and the resultant 3-D wireframes were also stored within the GEMS project. The project database stored cross section and level plan definitions and the block models, such that all data pertaining to the project are contained within the same project database. A copy of the project database is stored in WGM's servers in Toronto.

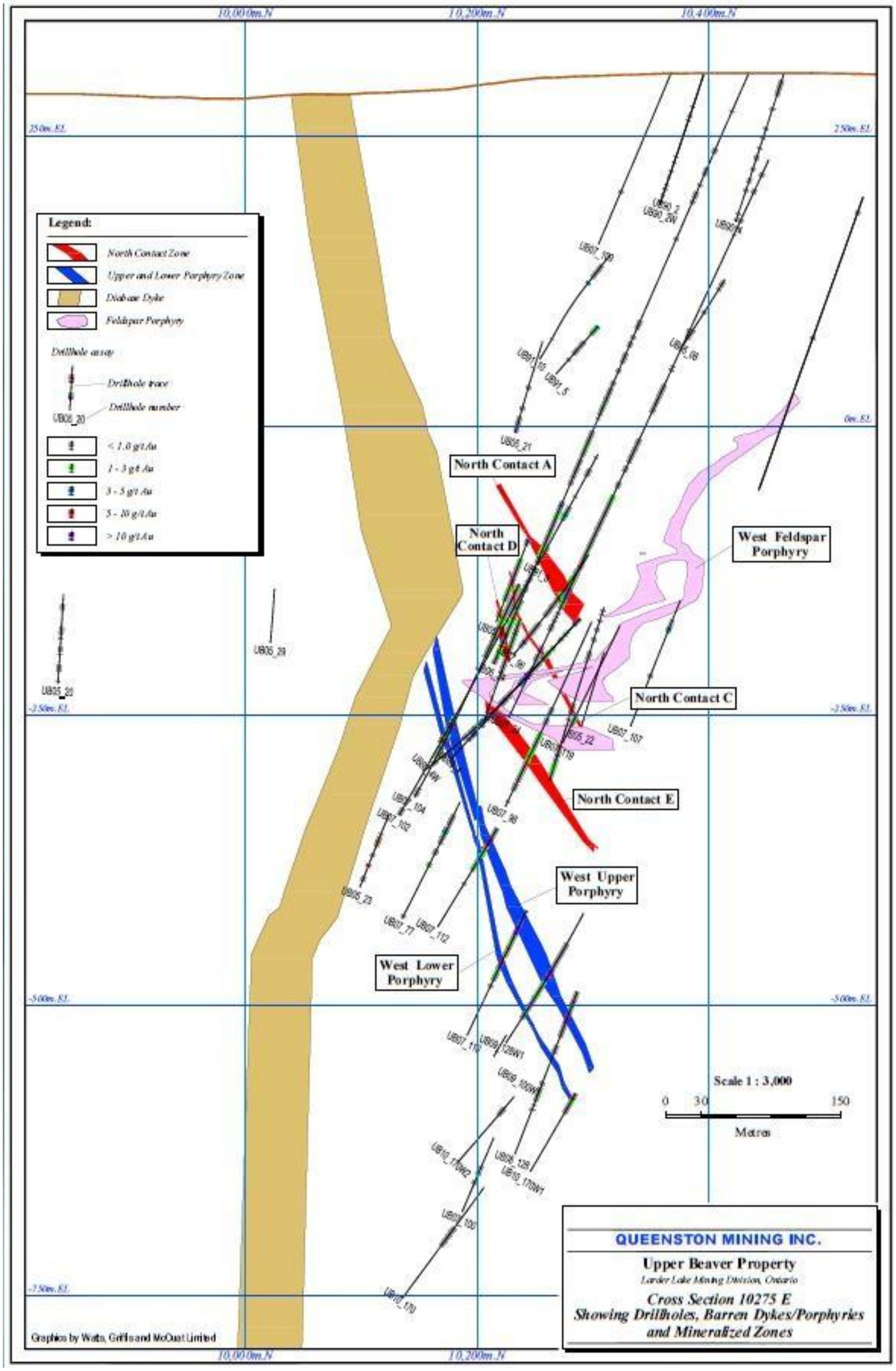
14.4 GEOLOGICAL MODELLING PROCEDURES

14.4.1 Cross Section Definition

Vertical sections were defined for the Upper Beaver Property to mimic those defined by Queenston staff for its cross sectional interpretation. The drilling for zone definition was conducted on cross sections that had a spacing that varied from 25 m to 50 m, but most drilling was conducted on the 25 m spaced sections.

In total, 37 west-looking vertical (cross) sections at 25 m spacing were defined for the mineralized zones. Figure 14.1 shows the geology and mineralized zone on cross section 10275E.

Figure 14.1 Cross Section 10275E Illustrating Geology and Mineralized Zones



14.4.2 Geological Interpretation

WGM imported Queenston's internal 3-D interpretations from the cross sections which were used as the basis to define the boundaries of the mineralized zones. Many of these same interpretations were originated from WGM's 2008 interpretation which have since been updated based on information from the newer drilling.

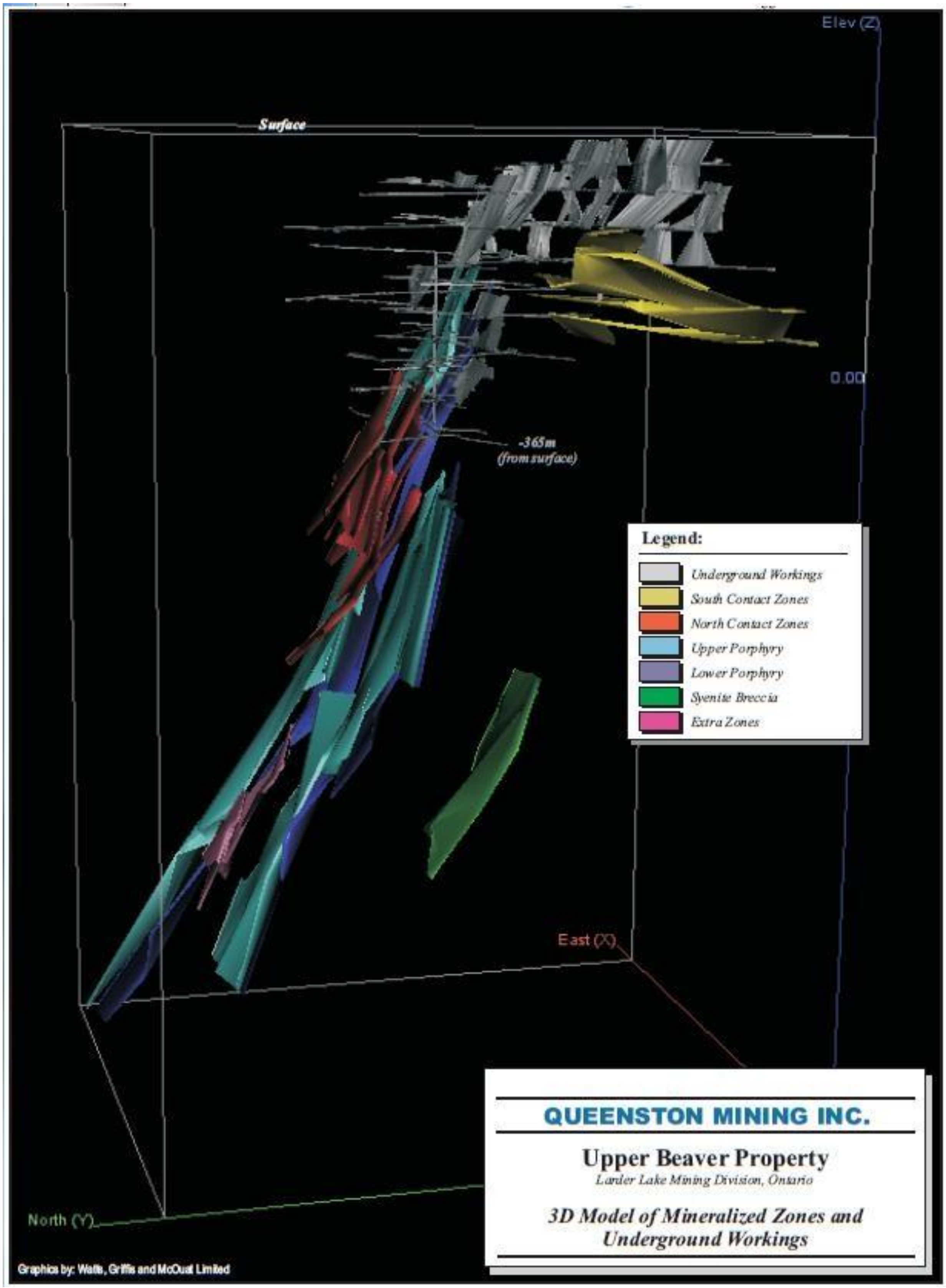
The wireframed zone interpretations and the corresponding polylines used in their generation were imported into GEMS and each was assigned an appropriate rock code. WGM verified that the digitized lines were 'snapped' to drillhole intervals to anchor the line which allows for the creation of a true 3-D wireframe that honours the 3-D position of the drillhole interval. Any discrepancies or interpretation differences between Queenston's original interpretation and those used by WGM were discussed with Queenston technical personnel and agreed upon before finalizing the interpretation to be used for the Mineral Resource estimate. The majority of the discussions centred around minimum horizontal widths in the definition of some of the zones at depth. As with the 2008 estimate, a minimum horizontal width of 2 m was used for defining the zones.

Zone boundaries were digitized from drillhole to drillhole that showed continuity of strike, dip and grade, generally from 50 to 100 m in extent, and 25 to 50 m maximum on the ends of the zones where there was no drillhole information (most extensions were limited to 25 m, unless supported by drillhole information on adjacent cross sections). Internally, the continuity of the zones was observed to be very good, and in some cases, with supporting data from adjacent sections, the interpretation was extended beyond 100 m internally. In general, extensions of the boundaries were made consistent with the trends defined by joining known boundaries and with information used from adjacent cross sections. Figure 14.2 shows a typical cross section through the Upper Beaver mineralized zones.

The Upper Beaver mineralized zones are for the most part discrete and can be identified relatively easily, however, there can also be multiple intercepts within the same general area of a mineralized section of the drillhole. Queenston used a nominal 1.0 g Au/t cut-off to determine the zone outlines for continuity purposes, but this general rule was applied on a case by case basis and was a fairly manual effort. Most bounding assay intervals used to define the zones were much higher grade than 1.0 g Au/t, however, some lower grade intercepts were used internally as internal dilution to ensure zone continuity.

WGM also used the updated 3-D interpretation of the Diabase Dyke and East and West Feldspar Porphyry Units, as supplied by Queenston, to "overprint" the WGM defined zones as the final step in order to subtract this barren material from the Mineral Resources. Queenston also supplied 3-D models of the underground workings for WGM's use. Figure 14.2 illustrates the 3-D models of the defined zones used for the Mineral Resource estimate and the Upper Beaver underground workings.

Figure 14.2 3-D Model of Mineralized Zones and Underground Workings



14.5 TOPOGRAPHIC SURFACE CREATION

A topographic surface or triangulated irregular network ("TIN") was supplied by Queenston, which was generated using collar elevations of the holes drilled from surface for the entire Upper Beaver Property area. This was not seen as being crucial for this stage of the Mineral Resource estimate, as the zones are going to be mined by underground methods.

14.5.1 Statistical Analysis, Compositing, Capping and Specific Gravity

14.5.1.1 Back-Coding of Rock Code Field

The 3-D solids that represented the interpreted mineralized zones were used to back-code a rock code field into the drillhole workspace. Each interval in the assay table was assigned a new rock code value based on the rock type solid that the interval midpoint fell within.

14.5.2 Statistical Analysis and Compositing

In order to carry out the Mineral Resource grade interpolation, a set of equal length composites of 1.0 m was generated from the raw drillhole intervals, as the original assay intervals were different lengths and required normalization to a consistent length. A total of 2,453 equal length composites were generated, of which 126 were discarded as they were comprised of less than 0.75 m of the original assay intervals. The average capped grade of the 126 omitted samples was 3.43 g Au/t. Table 14.2 summarizes the statistics of the remaining 2,327 1 metre composites inside the defined mineralized envelopes for Au and Cu, which were used for the Mineral Resource estimate. For our analysis, WGM examined each of the zones separately. The results of this study are illustrated in Figure 14.3 to Figure 14.8.

TABLE 14.2
BASIC STATISTICS OF 1 M COMPOSITES

Zone	Number	Mean Uncapped Au (g/t)	Mean Capped Au (g/t)	Mean Cu (%)	C.O.V.* Capped Au
Lower Porphyry	454	6.77	5.44	0.50	1.70
Upper Porphyry	843	7.71	5.46	0.41	1.63
North Contact	600	2.66	2.07	0.28	1.76
South Contact	205	4.98	4.08	0.40	1.91
Syenite Breccia	126	3.66	2.85	0.05	1.43
Extra Zones	99	2.75	2.75	0.17	1.47

**Co-efficient of Variation*

Figure 14.3 LOG Normal Histogram, Au Composites within Upper Porphyry Zone

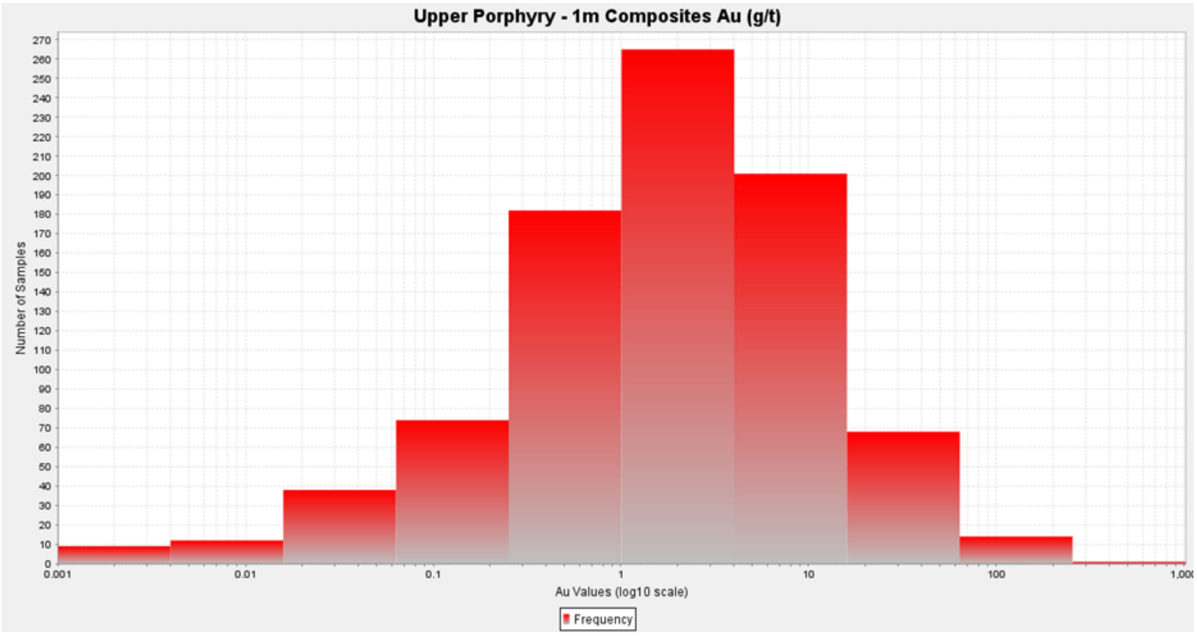


Figure 14.4 LOG Normal Histogram, Au Composites within Lower Porphyry Zone

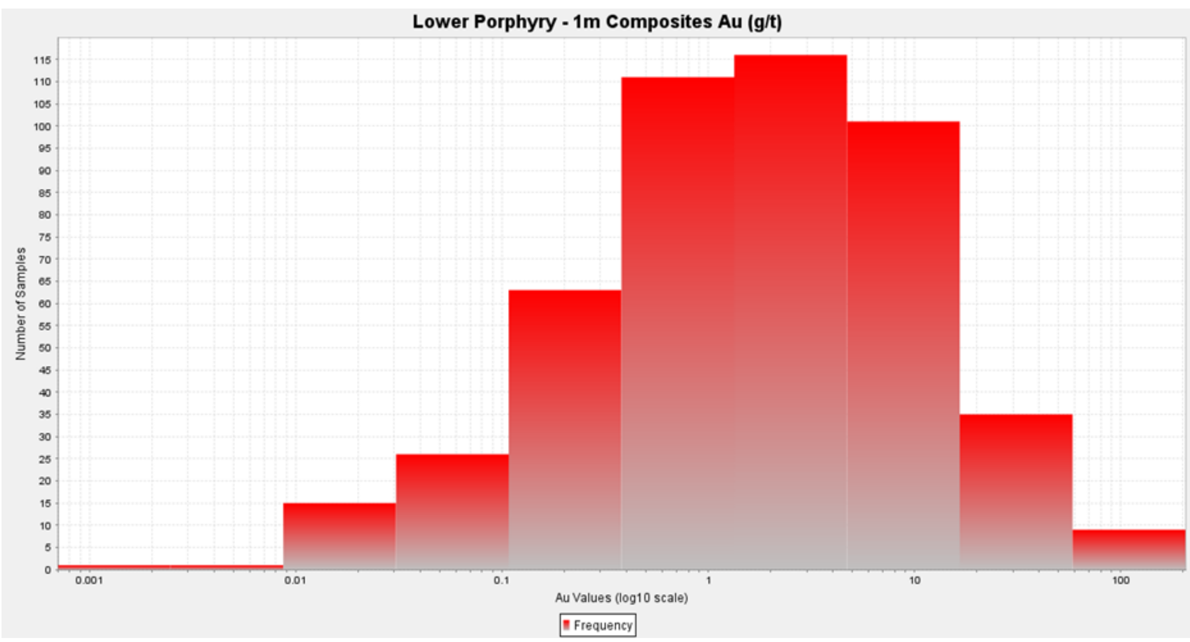


Figure 14.5 LOG Normal Histogram, Au Composites within North Contact Zone

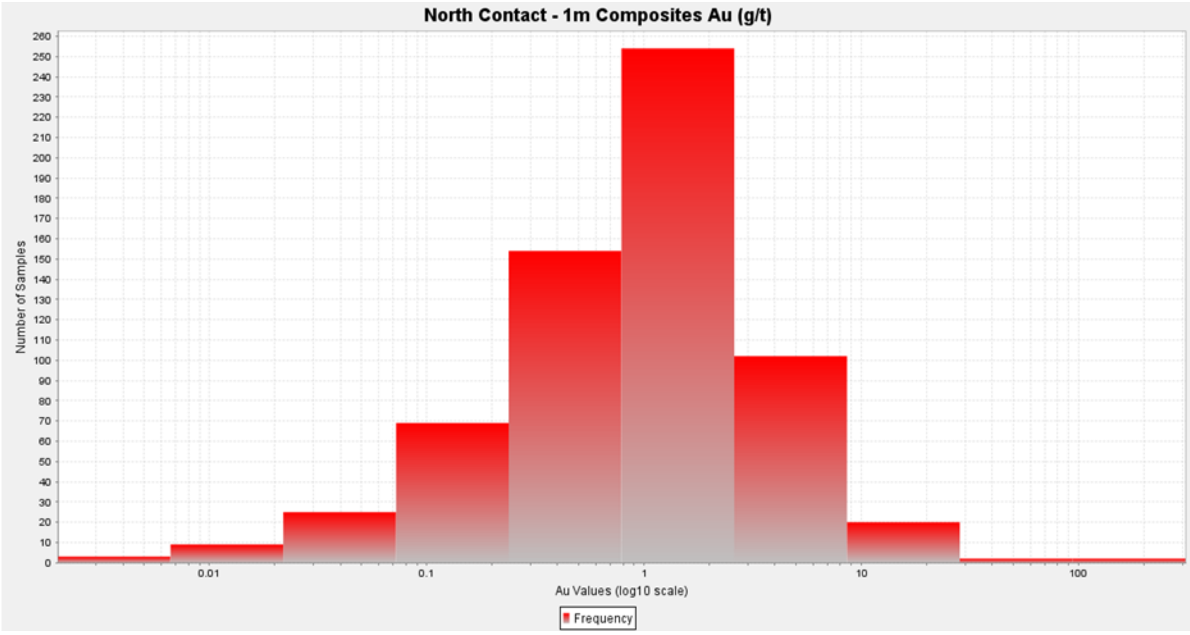


Figure 14.6 LOG Normal Histogram, Au Composites within South Contact Zone

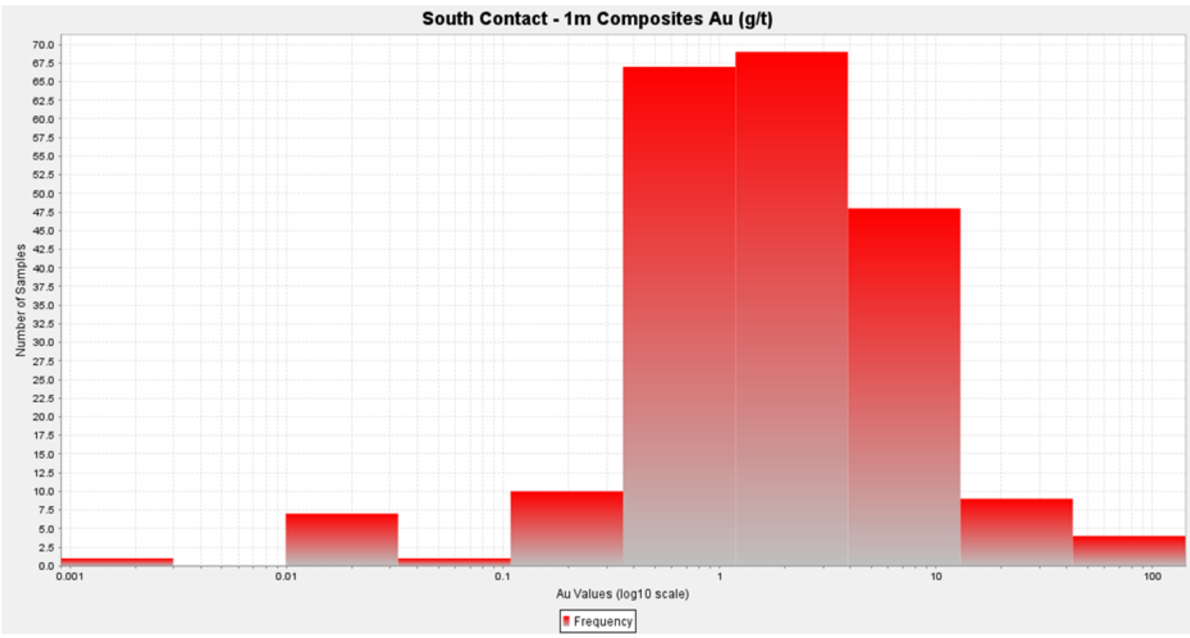


Figure 14.7 LOG Normal Histogram, Au Composites within Syenite Breccia Zone

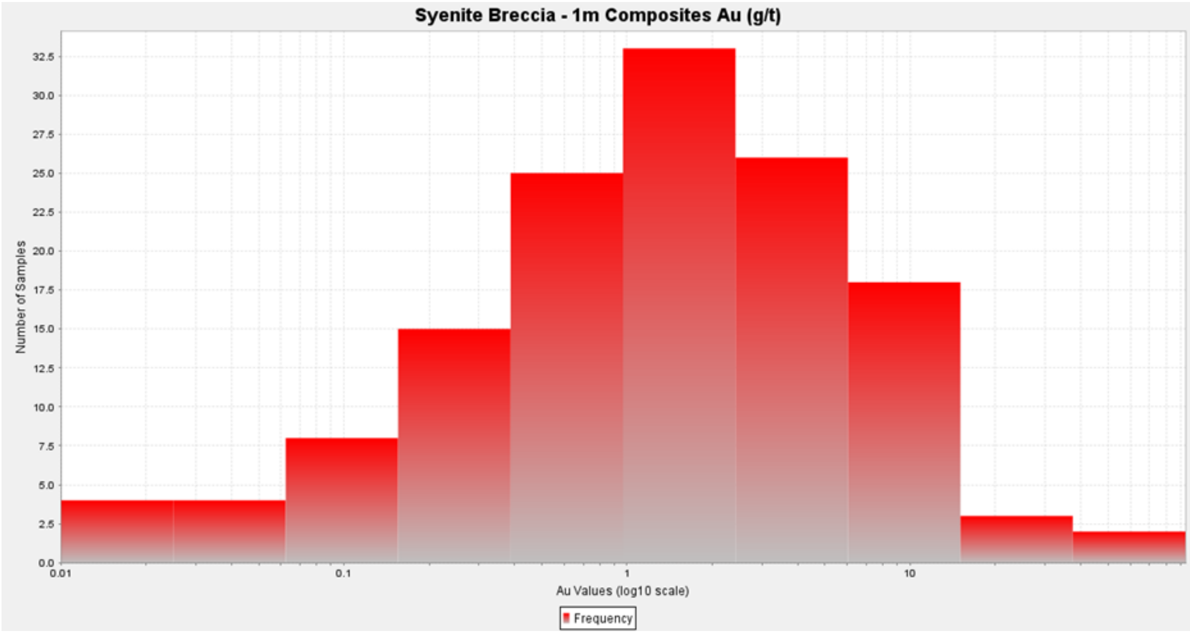
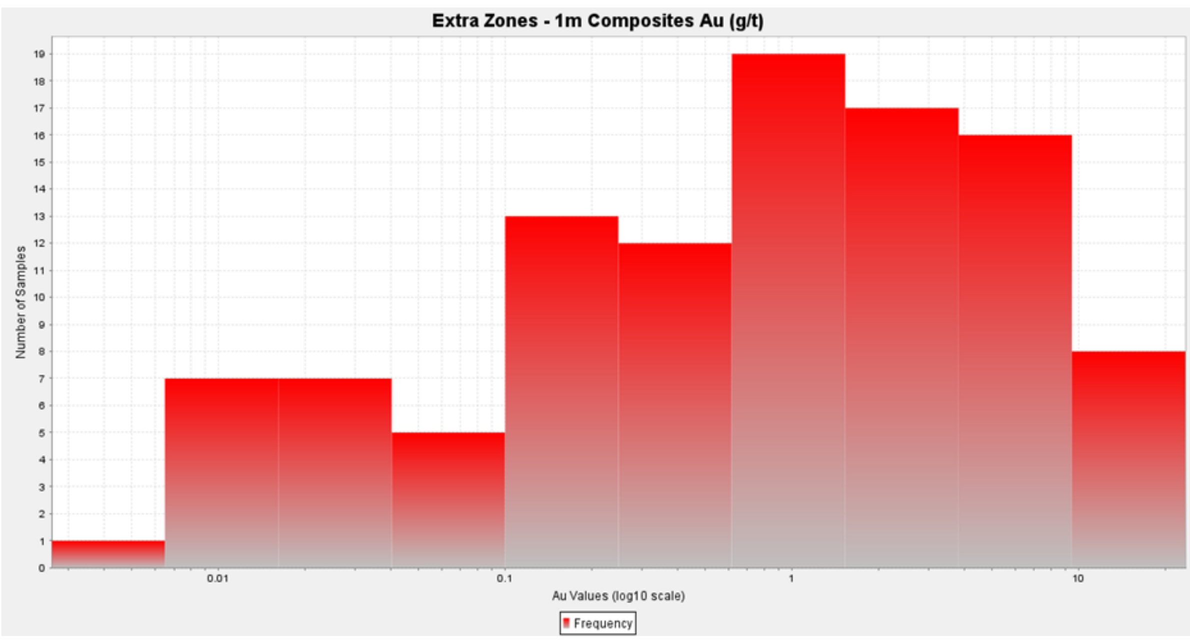


Figure 14.8 LOG Normal Histogram, Au Composites within Extra Zones



14.6 GRADE CAPPING

The statistical distributions of both Au and Cu show good lognormal distributions and most of the defined zones exhibit similar behaviour of grade distributions. Considering the nature of the mineralization and the continuity of the zones, WGM studied various capping levels for both Au and Cu. Grade capping, also sometimes referred to as top cutting, assay grades is commonly used in the Mineral Resource estimation process to limit the effect (risk) associated with extremely high assay values since high-grade outliers can contribute excessively to the total metal content of the deposit. Philosophies or approaches to establishing and using a grade cap is variable across the industry and includes, for example, not using grade caps at all, arbitrarily setting all assay grades greater than 1 oz./ton to 1 oz./ton, choosing the grade cap value to correspond to the 95 percentile in a cumulative distribution, evaluation of Mean Grades + multiple levels of Standard Deviations and the evaluation of the shape and values of histograms and/or probability plots to identify an outlier population. Another rule of thumb is to set the capping level to lower the top 10% of the metal content in the deposit.

A combination of decile analysis and a review of probability plots were used to determine the potential risk of grade distortion from higher-grade assays. A decile is any of the nine values that divide the sorted data into ten equal parts so that each part represents one tenth of the sample or population.

Typically, in a decile analysis, capping is warranted if the:

1) Last decile has >40% of metal.
2) Last decile contains >2.3 times the metal quantity contained in the one before last.
3) Last centile contains >10% of metal.
4) Last centile contains >1.75 times the metal quantity contained in the one before last.

As expected, the decile analysis results indicated that grade capping was warranted for Au, which was set to 50 g Au/t for all of the domains. Although the 2008 estimate did use a 2% Cu high grade cap to account for high grade outliers, subsequent decile analysis confirms that this was unnecessary. As such, a total of 216 Cu assays (out of 49,369 assays for which Cu grade existed) which were greater than 2% were included in the current resource estimate. The net result of Au capping for the Mineral Resource estimate at a 2.5 g Au/t cut-off grade was to reduce the Indicated Resource Au grade and contained metal by 21%, and to reduce the Inferred Resource Au grade and contained metal by 13%.

14.7 DENSITY/SPECIFIC GRAVITY

In 2008 and 2010, Queenston determined specific gravity ("SG") measurements on half core, as well as on rejects of assayed samples. The samples tested were almost entirely from its mineralized zones; no measurements were completed on host rocks (waste), although in a few cases wall rocks to mineralized zones were tested.

Half core samples were generally 6 inches long and the sample selected for SG determination represented a segment of core from assay samples 0.3 to 1.4 m long. The measurements on half core segments were completed by JVX Ltd. ("JVX") and Swastika for the 2008 samples and by Swastika only for the 2010 samples, using the weighing in water/weighing in air method. A total of 31 determinations were completed on half core in 2008 and 11 determinations were completed on half core in 2010

SG determinations on 151 rejects were completed in 2008 and one determination was completed from the composite sample used for the 2011 metallurgical testwork by pycnometer using water. A constant SG value of 2.9 was used for the Mineral Resource estimate based on the results obtained in the 2008 and 2010 determinations.

WGM recommends that the SG results, like all assays, should also be stored in an assay database table for ease of use and comparison purposes.

14.8 BLOCK MODEL PARAMETERS, GRADE INTERPOLATION AND CATEGORIZATION OF MINERAL RESOURCES

The Mineral Resources have been estimated using the Inverse Distance Cubed ("ID^3") estimation technique for Au and ID^2 for Cu. ID belongs to a distance-weighted interpolation class of methods, similar to Kriging, where the grade of a block is interpolated from several composites within a defined distance range of that block. ID uses the inverse of the distance (to the selected power) between a composite and the block as the weighting factor.

For comparison and cross checking purposes, the ID^2 (for Au) and ID^{10} method has also been used which closely resembles a Nearest Neighbour ("NN") technique. In this method, the grade of a block is estimated by assigning only the grade of the nearest composite to the block. All interpolation methods gave similar results, as the grades were very well constrained within the wireframes, and the results of the interpolation approximated the average grade of the all the composites used for the estimate.

14.8.1 Block Model Setup / Parameters

The block model was created using the GEMS v.6.2.4 software package to create a grid of regular blocks to estimate tonnes and grades. The deposit specific parameters used for the block modelling are summarized below.

The block sizes used were:

- Width of columns = 5.0 m
- Width of rows = 2.0 m
- Height of blocks = 5.0 m

The specific parameters for each block model are as follows:

- Easting coordinate of model bottom left hand corner: 9850.00
- Northing coordinate of model bottom left hand corner: 9690.00
- Datum elevation of top of model: 210.00 m
- Model rotation: 0.00
- Number of columns in model: 175
- Number of rows in model: 465
- Number of levels: 250

14.9 GRADE INTERPOLATION

Variograms were generated in an attempt to characterize the spatial continuity of the mineralization in the defined zones, however, due to the lack of data for most of the zones, meaningful variograms could not be computed. The geology and geometry is fairly well understood, so the search ellipse sizes and orientation were based on this geological knowledge, as opposed to variograms. The following lists the Au grade interpolation parameters:

ID^3 Search Ellipsoid:

- 100 m in the East-West direction
- 100 m in the North-South direction
- 25 m in the Vertical direction
- Minimum / Maximum number of composites used to estimate a block: 2 / 10
- Maximum number of composites coming from a single hole: 5
- Ellipsoidal search strategy was used with rotation about Z, X , Z: -10^o, -70^o, 0^o. For the South Contact zones, the orientation was adjusted to a rotation about Z, Y, Z as follows: -20^o, 30^o, 0^o.

As in the 2008 estimate, Cu grades were interpolated using the same parameters, except for using an ID^2 method.

GEMS does not use the sub-blocking method for determining the proportion and spatial location of a block that falls partially within a wireframed object. Instead, the system makes use of a percent model (if it is important to track the different rock type's proportions in the block – usually if there is more than one important type) or uses a "needling technology" that is similar in concept, but offers greater flexibility and granularity for accurate volumetric calculations. In this technique, all the blocks that are inside the wireframe (the user specifies the % threshold) are coded and thus are assigned the appropriate rock code and the interpolated grade. During the volumetric calculation, GEMS' needling process reports only the volume / tonnage of the block actually within the wireframe itself, but applies the interpolated grade to that portion of the block within the wireframe / solid.

14.10 MINERAL RESOURCE CATEGORIZATION

To categorize the Mineral Resources, WGM generated a distance model (distance from actual data point to the block centroid) and reported the estimated resources by distances which represented the category or classification. WGM chose to use the blocks that had a distance of 25 m or less to be Indicated category and +25 m to be Inferred category. The average distances and categories for the most of the zones were similar (especially for the Indicated) and are shown in Table 14.3.

TABLE 14.3.
AVERAGE INTERPOLATION DISTANCE FOR RESOURCE CATEGORIZATION

Zones	Average Distance for Indicated	Average Distance for Inferred
Lower Porphyry	15.1 m	44.7 m
Upper Porphyry	15.3 m	42.2 m
North Contact	13.7 m	29.7 m
South Contact	15.7 m	36.8 m
Syenite Breccia	14.8 m	40.4 m
Extra Zones	12.4 m	35.6 m

Figure 14.9 and 33 show the interpolated capped gold grade blocks and categorization on Cross Section 10375E.

Figure 14.9 Cross Section 10375E - Gold grade block model

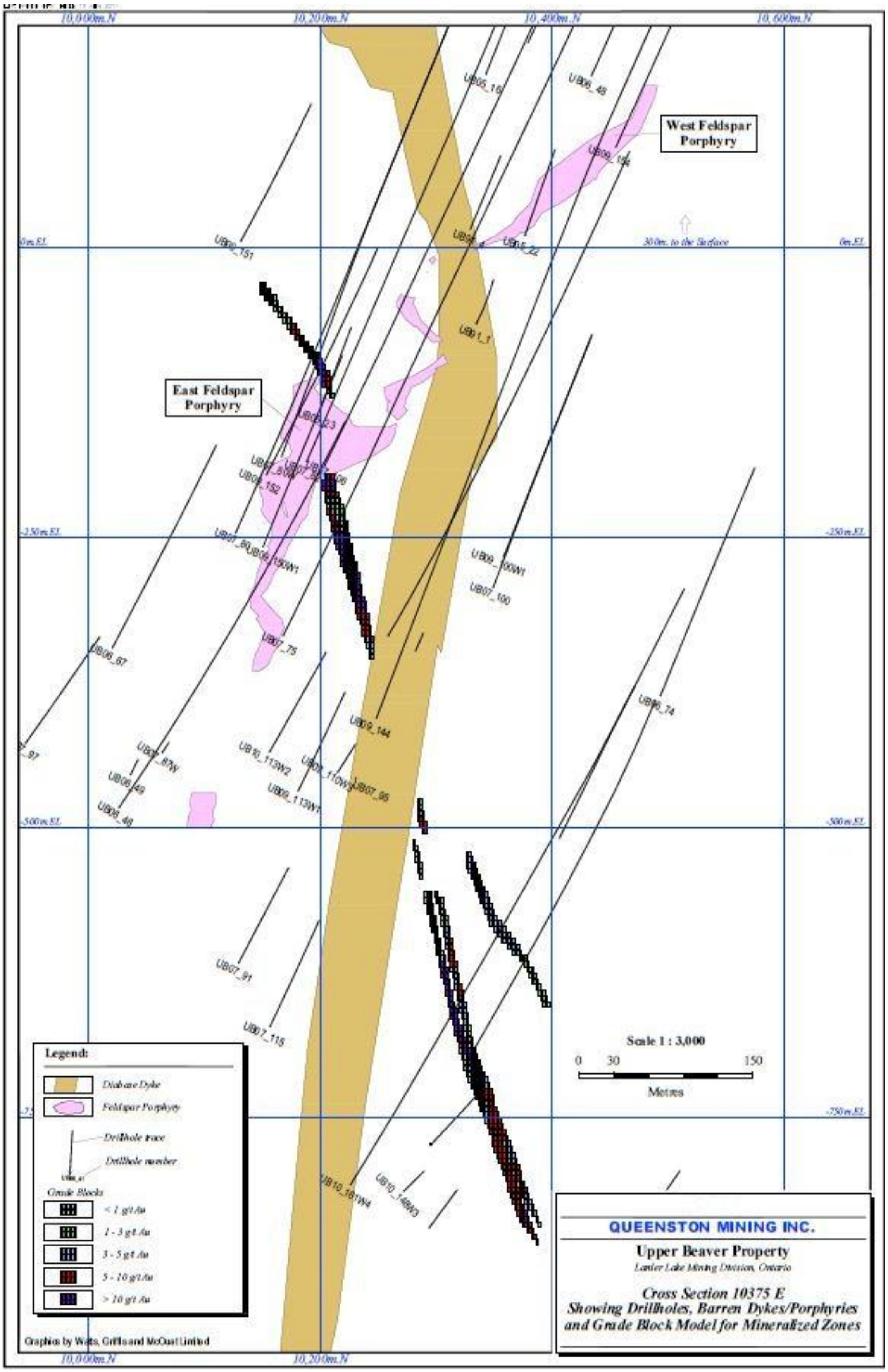
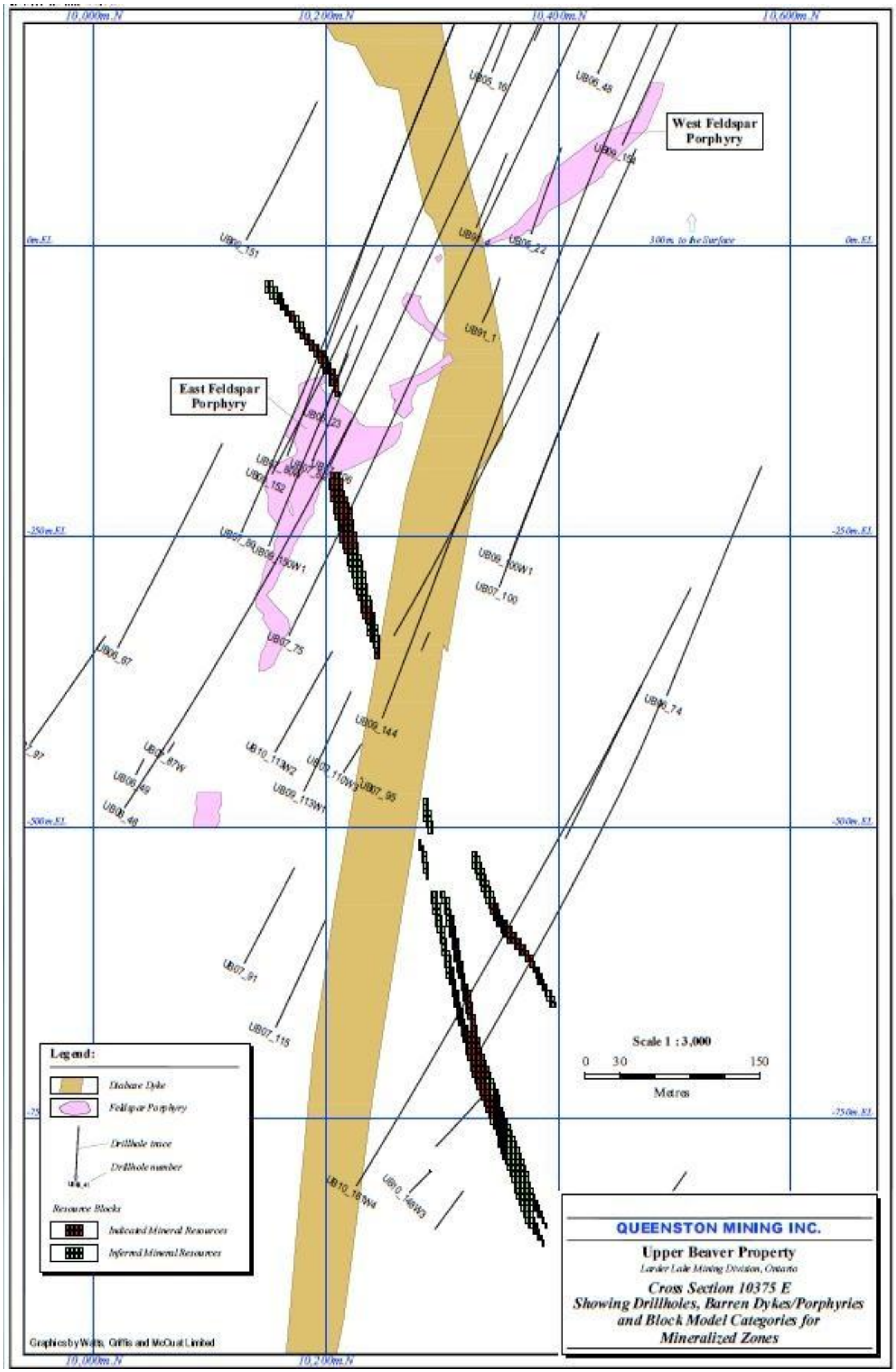


Figure 14.10 Cross Section 10375E - Categorization block model



For the Mineral Resource estimate, the minimum horizontal width of 2 m and a 2.5 g Au/t cut-off was determined to be appropriate at this stage of the project, and based on the relative increase in metal prices since the 2008 estimate (Table 14.4). These parameters were chosen based on a preliminary review of the parameters that would likely determine the economic viability of an underground mining operation and comparison to similar projects in the area that are currently being mined or are at an advanced stage of study / development.

TABLE 14.4
CATEGORIZED MINERAL RESOURCE ESTIMATE FOR MAIN UPPER BEAVER ZONES (CUT-OFF OF 2.5 G AU/T)

Category	Zone	Tonnage	Cu	Au (uncapped)	Contained Au (uncapped)	Au (capped)	Contained Au (capped)
		Tonnes	**(%)**	**(g/t)**	**(oz.)**	**(g/t)**	**(oz.)**
Indicated	Lower Porphyry	771,000	0.68	10.04	249,000	8.05	200,000
	Upper Porphyry	1,448,000	0.55	9.72	452,000	7.66	357,000
	North Contact	278,000	0.48	5.72	51,000	4.27	38,000
	South Contact	349,000	0.47	7.04	79,000	5.74	64,000
	Syenite Breccia	157,000	0.08	6.55	33,000	4.37	22,000
	Extra Zones	71,000	0.17	4.11	9,000	4.11	9,000
	Total	**3,074,000**	**0.54**	**8.84**	**874,000**	**6.98**	**690,000**
Inferred	Lower Porphyry	1,048,000	0.48	7.03	237,000	6.51	219,000
	Upper Porphyry	1,479,000	0.35	7.9	376,000	6.64	316,000
	North Contact	66,000	0.43	3.68	8,000	3.26	7,000
	South Contact	386,000	0.53	5.82	72,000	4.81	60,000
	Syenite Breccia	92,000	0.05	5.11	15,000	3.95	12,000
	Extra Zones	22,000	0.15	4.13	3,000	4.13	3,000
	Total	**3,093,000**	**0.41**	**7.15**	**711,000**	**6.19**	**616,000**

(1) *Interpretations of the mineralized zones were created as 3D wireframes/solids based on a 1.0 g Au/t outline and a minimum horizontal thickness of 2 m.*
(2) *Mineral Resources were estimated using a block model with a block size of 5m by 5m by 2m and a specific gravity of 2.9 t/m³*
(3) *Individual assays were capped at 50 g Au/t.*
(4) *Mineral Resources were estimated using a three-year rolling average of US$1,050/ounce, an exchange rate of US$0.95=CDN$1.00 and assumed metallurgical recoveries of 95%.*

The majority of the mineral resource occurs in the Porphyry Zones that contain approximately 80% of the mineral resource with 556,000 oz. (2,219,000 t grading 7.8 g Au/t) in the Indicated category and 535,000 oz. (2,528,000 t grading 6.6 g Au/t) in the Inferred category.

The following sensitivity analysis from the base case resource estimate using various cut-off grades ranging between 0.0 g Au/t and 5.0 g Au/t indicates the potential of higher grades employing higher cut-off grades.

Table 14.5 Mineral Resource Cut-off Sensitivity							
Cut-off	**Category**	**Tonnage**	**Cu**	**Au (uncapped)**	**Contained Au (uncapped)**	**Au (capped)**	**Contained Au (capped)**
(g Au/t)		**(Tonnes)**	**(%)**	**(g/t)**	**(oz.)**	**(g/t)**	**(oz.)**
0	Indicated	5,428,000	0.38	5.57	972,000	4.51	787,000
	Inferred	4,502,000	0.32	5.37	777,000	4.71	682,000
0.5	Indicated	5,061,000	0.4	5.95	969,000	4.82	784,000
	Inferred	4,327,000	0.34	5.58	776,000	4.89	681,000
1	Indicated	4,561,000	0.43	6.52	956,000	5.26	772,000
	Inferred	4,103,000	0.35	5.84	771,000	5.12	675,000
1.5	Indicated	3,986,000	0.47	7.28	933,000	5.84	748,000
	Inferred	3,844,000	0.36	6.15	760,000	5.38	665,000
2	Indicated	3,520,000	0.5	8.01	907,000	6.38	722,000
	Inferred	3,495,000	0.38	6.59	740,000	5.74	645,000
2.5	**Indicated**	**3,074,000**	**0.54**	**8.84**	**874,000**	**6.98**	**690,000**
	Inferred	**3,093,000**	**0.41**	**7.15**	**711,000**	**6.19**	**616,000**
3	Indicated	2,606,000	0.58	9.93	832,000	7.74	649,000
	Inferred	2,631,000	0.44	7.92	670,000	6.8	575,000
3.5	Indicated	2,232,000	0.62	11.05	792,000	8.5	610,000
	Inferred	2,295,000	0.47	8.59	634,000	7.32	540,000
4	Indicated	1,934,000	0.66	12.15	756,000	9.23	574,000
	Inferred	2,005,000	0.49	9.27	598,000	7.84	505,000
4.5	Indicated	1,680,000	0.7	13.31	719,000	9.98	539,000
	Inferred	1,669,000	0.53	10.26	550,000	8.56	459,000
5	Indicated	1,470,000	0.76	14.49	685,000	10.74	507,000
	Inferred	1,394,000	0.59	11.33	508,000	9.32	418,000

14.10.1 Visual Comparison

The visual comparison of block model grades with composite grades shows a reasonable correlation between the values. No significant discrepancies were apparent from the sections and plans reviewed. The orientation of the estimated grades on sections follows more or less the projection angles defined by the search ellipsoid. It is doubtful that refining the search ellipsoid orientation by adding a few additional sub-domains or using an unfolding technique would significantly improve the interpolation.

14.10.2 Global Comparisons

The grade statistics for the raw assays, composites, nearest neighbour and inverse distance models, were tabulated in Table 14.6. Statistics for the composite mean grade when compared to the raw assay grade shows a slight reduction in value partly due to the addition of zero grade assigned to the un-sampled intervals during the compositing process and also due to smoothing related to volume variance introduced with the 1 m composite size. Composite grade statistics only improved by 0.07 g Au/t (from 4.12 g Au/t to 4.19 g Au/t) when the zero grade composites were removed from the statistics. On a global basis, regardless of the methodology employed for the interpolation, the composite grade average is very close to the interpolated grade. More importantly, the grade of the nearest neighbour and inverse distance model at 0.00 g Au/t cut-off are very close to each other, showing that no global bias was introduced from the interpolation method used.

TABLE 14.6 GLOBAL GRADE COMPARISON AT 0.00 G AU/T CUT-OFF	
Method	**Average Grade (g Au/t)**
All Assays (within resource wireframes)	4.43
All Composites (>= 0.000 g Au/t)	4.12
All Composites (>=0.001 g Au/t)	4.19
Nearest Neighbour	4.50
Inverse Distance Cubed	4.51

14.11 P&E'S POTENTIALLY ECONOMIC PORTION OF THE MINERAL RESOURCE ESTIMATE

A Potentially Economic portion of the Mineral Resources was estimated by P&E, with dilution and mining losses incorporated, as a basis for a PEA of the Deposit. The results of this determination are provided in Table 14.8. P&E cautions that these Potentially Economic Portions of the Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Stope mining would commence at the -375 m and -900 m loading pocket levels and proceed upwards. The selected primary mining method is 'longhole longitudinal retreat mining', with sublevels established at 35 metres intervals. Based on this mining method, P&E has estimated minimum mining width of 2.0 m. A secondary mining method of 'up-dip pilot and slash' mining was selected for approximately 5% of the total Potentially Economic portion of the Mineral Resources, calculated before dilution and mine extraction ("recovery") are included. A summary of the Mineral Resources that are considered to be potentially economic is presented in Table 14.7.

TABLE 14.7
P&E'S POTENTIALLY ECONOMIC PORTION OF THE MINERAL RESOURCES (BEFORE DILUTION AND RECOVERY)(1-8)

Vein	Indicated Resources			Inferred Resources		
	Tonnes	Cu %	Au (g/t) (Capped)	Tonnes	Cu %	Au (g/t) (Capped)
LP EAST	302,705	0.427	6.596	656,536	0.314	5.464
LP WEST	504,651	0.792	8.291	335,059	0.763	8.067
UP EAST	736,568	0.194	6.072	873,962	0.144	6.355
UP WEST	973,617	0.765	7.337	538,452	0.652	6.276
SB HW	186,953	0.063	3.676	92,895	0.045	3.545
SB FW	21,519	0.039	2.483	624	0.049	1.926
EZ D	25,887	0.213	3.474	1,506	0.296	4.075
EZ E	7,394	0.118	3.953	836	0.086	3.287
EZ F	38,846	0.153	4.000	17,373	0.105	3.645
NC A	123,736	0.695	4.104	11,001	0.876	3.035
NC B	12,319	0.021	4.630	2,667	0.171	3.274
NC C	34,370	0.788	3.861	9,709	0.644	3.258
NC D	48,437	0.285	2.745	9,174	0.293	2.726
NC E	52,209	0.155	3.972	20,393	0.161	2.723

TABLE 14.7 P&E'S POTENTIALLY ECONOMIC PORTION OF THE MINERAL RESOURCES (BEFORE DILUTION AND RECOVERY)(1-8)						
Vein	**Indicated Resources**			**Inferred Resources**		
	Tonnes	**Cu %**	**Au (g/t) (Capped)**	**Tonnes**	**Cu %**	**Au (g/t) (Capped)**
SC M	87,892	0.137	3.539	77,169	0.205	3.472
SC L	32,774	0.243	3.613	36,591	0.253	3.049
SC U	105,093	0.170	2.671	160,953	0.639	3.911
Total	**3,294,970**	**0.490**	**6.240**	**2,844,899**	**0.385**	**5.902**

(1) The Mineral Resources in Table 14.7were 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.

(2) Mineral resources are estimated at a cut-off grade of 2.0 g/t Au and a minimum true width of 2.0 m

(3) Mineral resources are estimated using an average gold price of US$1,275/oz., an exchange rate of US$0.96=CDN$1.00 and metallurgical recoveries of 98%.

(4) Individual assays were capped at 50 g/t Au.

(5) A bulk density of 2.9 t/m³ was used.

(6) Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. Environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues may materially affect the estimate of mineral resources. 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

(7) The Potentially Economic portion of the Mineral Resource estimate was prepared by Eugene Puritch, P. Eng and James L. Pearson P.Eng. of P&E Mining Consultants Inc. Mineral resource estimates reported in Table 14.7 were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions.

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

The longhole longitudinal retreat mining method is expected to experience mining dilution in the order of 20% (from waste rock from the walls and top of the stope and backfill), containing zero metal grades. Mine recovery is estimated to be 95%. The up-dip pilot and slash mining method is estimated to experience mining dilution in the order of 10% at zero metal grade. Mine recovery for this secondary method is expected to be 75%. A summary of the Potentially Economic Portion of the Mineral Resources, including dilution and recovery, is presented in Table 14.8.

Table 14.8 P&E's Potentially Economic Portion of the Mineral Resources (1)(2)(3)						
Vein	**Indicated Resources**			**Inferred Resources**		
	Tonnes	**Cu %**	**Au (g/t) (Capped)**	**Tonnes**	**Cu %**	**Au (g/t) (Capped)**
Total Longhole	3,157,102	0.503	6.386	2,647,356	0.371	6.062
Diluted @ 20%	3,788,523	0.419	5.321	3,176,827	0.310	5.052
Extracted @ 95%	3,599,097	0.419	5.321	3,017,985	0.310	5.052
Total Up-dip Pilot & Slash	137,867	0.187	2.895	197,544	0.567	3.751
Diluted @ 10%	151,654	0.170	2.632	217,298	0.516	3.410
Extracted @ 75%	113,740	0.170	2.632	162,974	0.516	3.410
Total after Dilution & Extraction	**3,712,837**	**0.412**	**5.239**	**3,180,959**	**0.320**	**4.968**

(1) Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. Environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues may materially affect the estimate of mineral resources. 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

(2) The Potentially Economic portion of the Mineral Resource estimate was prepared by Eugene Puritch, P.Eng. and James L. Pearson P.Eng. of P&E Mining Consultants Inc. Mineral resource estimates reported in this press release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions

(3) Mine recovery and dilution are included in these quantities and average metal grades

15.0 MINERAL RESERVE ESTIMATES

The inferred mineral resources presented herein have not been sufficiently drilled to confidently demonstrate economic viability. In addition, the work undertaken on the Upper Beaver Project to date is considered to be at conceptual levels of study only. As such, and according to the NI 43-101 Regulations, it is not possible to declare a mineral reserve of any kind as of the effective date of this report.

16.0 MINING METHODS

The Upper Beaver Potentially Economic Portion of the Mineral Resource extends from the +155 metre elevation to the -1,005 metre elevation, a vertical distance of 1,160 metres. A conceptualized mining plan has been developed to extract the Deposit using mechanized trackless mining equipment. This conceptual plan is presented in this section.

Access to the mineral deposits would be via a 6.5 metre diameter, concrete lined fresh air shaft. This vertical shaft would be sunk conventionally to the -1005 metre level (the shaft collar would be located at the 302 metre elevation), for a total depth of 1,307 metre. Lateral development would start at the -375 metre level once the shaft reaches that elevation. The shaft excavation contractor would then continue sinking the shaft to the -1005 metre level as the -375 metre level lateral development is underway. Once shaft sinking is complete and the shaft has been commissioned two hoists would transport men and materials between surface and the underground levels. A series of three internal declines or ramps located in the vicinity of the stoping operations would eventually connect all working levels in the mine. Once the shaft has been commissioned stope mining would start on the -375 metre level and development on the -900 metre level would commence.

This PEA envisages the development of an underground mine with a steady state production rate of 2,000 tpd of mill feed (on a schedule of 350 days per year). The primary mining method is envisaged to be conventional longitudinal longhole retreat with paste backfill in the completed stopes. Sub-levels would be at 35 metre vertical intervals. The average thickness of the Potentially Economic Portion of the Mineral Resource is 5.6 m.

16.1 LONGHOLE LONGITUDINAL RETREAT MINING METHOD

The mining method selected for the extraction of the mineralized rock is Longhole Longitudinal Retreat mining. Sublevels would be driven in the mineralization every 35 vertical metres to allow access for drilling, blasting and mucking operations. Access to the sublevels would be provided by drifts driven either from the shaft access drifts on every third sublevel or from the internal ramps.

Stope dimensions would nominally be 65 metres long by 35 metres high by 5.6 metres wide. A slot/ventilation/backfill raise would be driven at the extremity of each stope. Successive rows of drillholes would be blasted into the slot raise and the resulting open stope. Cemented paste backfill and development waste would be placed in the stopes as they retreat from the slot raise to the entrance to the stope. The Life-of-Mine (“LOM”) schedule includes 217 stopes which would produce an average of approximately 1,750 tpd of mill feed. Typically, this corresponds to mining four sublevels concurrently (i.e. approximately 440 tpd / sublevel).

16.2 MINE AND STOPE DEVELOPMENT

All excavations in waste rock are classified as mine development. All development in mineralization that produces mill feed is classified as stope development. The LOM schedule includes a total of 24,107 metres of mine development (see Table 16.1). In additional there would be 1,307 vertical metres of shaft development and 10,650 cubic metres of shaft station and loading pocket development.

TABLE 16.1 SUMMARY OF MINE AND STOPE DEVELOPMENT									
Level	**Stope (ore) Development (m)**		**Mine Development in Waste (m)**						
	Drift	**Slot Raise**	**Shaft Access Drift**	**Sump**	**Ramps**	**Access X-cut & Drift**	**Orepass X-cut**	**Orepass**	**Vent Raise**
80 - 155	139	60			1,628	899			217
70 - 80	649					335			
45	189	420	352		271	44			
30	100					180			
10	229	120			271	44	10		
-25	298	150			271	197	10	32	
-60	297	150	294		271	201	10	32	
-95	439	270			271	246	10	34	
-130	525	270			271	243	10	34	
-165	440	270	260		271	270	10	34	
-200	753	360			271	291	10	0	
-235	530	240			843	256	10	32	
-270	585	210	476		271	122	20	32	
-305	463	180			271	155	10	32	
-340	607	240			271	204	20	32	
-375	560	270	437	40	271	218	25	32	
-410	595	240			271	282	22	32	
-445	633	270			271	409	25	34	
-480	554	330	402		271	206	25	34	
-515	548	240			271	253	25	34	
-550	608	330			271	260	30	34	
-585	407	210	330		271	298	20	34	
-620	415	210			271	308	35	34	
-655	467	240			271	223	35	34	
-690	541	360	341		271	280	22	32	
-725	492	270			271	269	25	32	
-760	281	90			271	190	35	32	
-795	435	150	318		271	197	30	32	
-830	486	180			271	201	30	32	
-865	407	240			271	244	25	32	
-900	248	150	303	40	271	283	35	32	
-935	122	60			271	47	10	35	
-970	112	60			271	62	10	35	
-1005	91	60	170		271	72	10	35	
Total	**15,019**	**6,900**	**3,683**	**80**	**10,611**	**7,988**	**604**	**925**	**217**

There is a total of 21,919 metres of stope development required over the LOM.

16.3 STOPING

The Longhole Longitudinal Retreat mining method is initially developed with sublevel drifts developed to the full width of the Deposit every 35 vertical metres (“undercuts” and “overcuts”) from the access cross-cuts. A 1.8 metre by 1.8 metre slot / ventilation / backfill raise is then driven at the end of the sublevel drift.

Blastholes measuring 92mm ($3^{5/8}$ inches) in diameter would then be drilled from the sublevel either up or down to adjacent sublevels. These blastholes would typically be drilled on a 1.5 metre by 1.5 metre pattern, in order to break the rock into the open slot and stope. The blasting powder factor necessary to produce adequate fragmentation of the rock, using emulsion explosives, is estimated to be approximately 0.85 kg/t. An estimated 1,750 tonnes of mill feed would be excavated on a daily basis from a combination of stopes. Stope development activities would add another 250 tonnes mill feed to the total, to provide a combined 2,000 tpd of mill feed. A summary of stope drilling and blasting parameters is presented in Table 16.2.

TABLE 16.2
STOPING DRILLING AND BLASTING PARAMETERS

Total Tonnes Mill Feed per Day from Mining Activities	2,000
Mineralization Specific Gravity.	2.90
Stope Height (m)	35
Nominal Stope Width (m)	5.6
Nominal Stope Length (m)	65
Total Nominal Stope Tonnage	37,035
Slot Raise Tonnage	282
Nominal Sublevel Drift Tonnage	5,291
Nominal Longhole Tonnes	31,463
Longhole Drilling Parameters @ 3.625” Dia Holes	
Total Drilling Per Stope (metres)	3,462
Drillholes Per Stope	115
Drilling Time Per Shift (minutes)	10
Metres Drilled per Shift	76
Total Metres Drilled Per Day	152
Required Metres per Day for Production Schedule	192
Blasting Parameters	
Loading Time Per Shift	10
Stemming Length Per Blasted Hole Length (m)	1.0
Load Length per Hole, (m)	29.0
Length of Holes Loaded Per Ring (metres)	110

Stope mining would initially start at the -375 metre level, followed by the -900 metre level approximately six months later. Stope mining would progress upwards from those levels, on a retreat basis, working an average four stopes at any given time. Paste backfill would be placed in the mined out area of the stope, from the level above through piping and boreholes, as stope drill/blast/mucking progresses.

The stope mining cycle would include longhole drilling, blasting, mucking and backfilling. The overall stope mining productivity is estimated to be approximately 450 tpd per stope. At any given time, a minimum of four levels should be available for stope mining, each with at least one stope available for mining. On average this would provide for an average production rate of 450 tpd per level and 1,800 tpd overall. When no development ore is being produced a minimum fifth stope would be available for stope mining

A summary of stoping productivities is presented in Table 16.3.

TABLE 16.3
STOPING PRODUCTIVITIES

Operation	Productivity
Drilling (tpd)	1,385
Blasting (tpd)	2,770
Mucking (tpd)	1,385
Backfill (tpd)	2,770
Average Stope Productivity (tpd)	462
Minimum tpd / level	437
Maximum Number of Working Levels	4

16.4 SCHEDULE

16.4.1 Shaft

Site clearing for the shaft collar will start on ‘day-one’ of the schedule. P&E estimates it will take 15 months to collar the shaft and install the headframe, hoist room and hoists and commission these installations. Shaft sinking will begin at that time and scheduled to be complete 39.6 months from the start of collaring the shaft. Details of the shaft sinking schedule are presented in Table 16.4.

TABLE 16.4
SHAFT SINKING SCHEDULE

Description	Month	
	Start	Finish
Collar / Headframe / Hoistroom	0.0	15.0
Collar to 45L Station	15.0	17.3
45L Station	17.3	17.8
45L Station to -80L Station	17.8	19.3
-80L Station	19.3	19.8
-80L Station to -165L Station	19.8	20.8
-165L Station	20.8	21.3
-165L Station to -270L Station	21.3	22.5
-270L Station	22.5	23.0
-270L Station to -375L Station	23.0	24.2
-375L Station	24.2	24.7
-375L Station Mechanical Lip Pocket	24.7	24.9
-375L Station to Loading Pocket No1	24.9	25.4
Loading Pocket No1	25.4	25.6
Install Loading Pocket	25.6	26.1
Loading Pocket Raise	26.1	26.5
Loading Pocket No1 to Spill Pocket No1	26.1	26.2
Spill Pocket No1	26.2	26.3

TABLE 16.4 SHAFT SINKING SCHEDULE		
Description	Month	
	Start	Finish
Spill Pocket No1 to -480 Station	26.3	26.8
-480L Station	26.8	27.3
-480L Station to -585L Station	27.3	28.6
-585L Station	28.6	29.1
-585L Station to -690L Station	29.1	30.3
-690L Station	30.3	30.8
-690L Station to -795L Station	30.8	32.0
-795L Station	32.0	32.5
-795L Station to -900L Station	32.5	33.7
-900L Station	33.7	34.2
-900L Station to Loading Pocket No2	34.2	34.7
Loading Pocket No2	34.7	34.9
Install Loading Pocket	34.9	35.4
Loading Pocket Raise	35.4	35.8
Loading Pocket No2 to -1005L Station	35.4	36.1
-1005L Station	36.1	36.6
Remove Sinking Geer & Commission Shaft	36.6	39.6

16.5 INTERNAL RAMP DEVELOPMENT

Internal ramping will be required to allow underground mobile equipment and personnel to travel between levels. Internal ramping will start from the -375L developing up to the -340L and down to the –410L simultaneously, double heading during the 37th month from start. This ramp system will be west of the main dyke. Internal ramping will start at the -900L during the 53rd month, from start, developing down to the -935L. This ramp system will be east of the main dyke. The balance of internal ramps will be driven as required. Details of the internal ramp development schedule are presented in Table 16.5.

TABLE 16.5 INTERNAL RAMP DEVELOPMENT SCHEDULE		
Level Interval	**Month**	
	Start	**Finish**
130L to 80L E	71.8	74.3
80L to 45L E	70.1	71.8
45L to 10L E	68.4	70.1
10L to -25L E	66.7	68.4
-25L to -60L E	65.0	66.7
-60L to -95L E	63.3	65.0
-95L to -130L E	61.5	63.3
-130L to -165L E	59.8	61.5
-165L to -200L E	58.1	59.8
-200L to -235L E	56.4	58.1
East - West Connection	54.5	56.4
-200L to -235L W	52.8	54.5
-235L to -270L W	51.1	52.8
-270L to -305L W	49.4	51.1
-305L to -340L W	45.9	47.6
-340L to -375L W	36.6	40.0
-375L to -410L W	36.6	40.0
-410L to -445L W	47.0	48.7
-445L to -480L W	77.7	79.4
-480L to -515L W	76.0	77.7
-515L to -550L W	74.3	76.0
-550L to -585L W	72.5	74.3
-585L to -620L W	70.8	72.5
-620L to -655L W	69.1	70.8
-655L to -690L W	67.4	69.1
-690L to -725L W	65.7	67.4
-725L to -760L W	64.0	65.7
-760L to -795L W	62.3	64.0
-795L to -830L W	60.6	62.3
-830L to -865L W	57.2	58.9
-865L to -900L W	53.7	55.4
-900L to -935L E	52.0	53.7
-935L to -970L E	55.4	57.2
-970L to -1005L E	58.9	60.6

16.6 LEVEL DEVELOPMENT

Level development will start on the -375L during the 26th month once shaft sinking has reached that level and the lip pocket has been installed. At that point both shaft sinking and level development will proceed, simultaneously. Once the shaft sinking has been completed and the shaft has been commissioned level development on the -900L will begin during the 40th month. Level develop will be completed, as required. All level development will be completed during the 122nd month (on the -480L). Details of the level development schedule are presented in Table 16.6.

TABLE 16.6 LEVEL DEVELOPMENT SCHEDULE		
Level	**Month**	
	Start	**Finish**
115L-150L	97.0	110.8
45L	93.3	97.0
10L	91.5	93.3
-25L	88.2	91.5
-60L	83.1	88.2
-95L	78.7	83.1
-130L	73.8	78.7
-165L	69.6	73.8
-200L	63.6	70.3
-235L	58.9	63.6
-270L	51.8	58.9
-305L	47.6	51.8
-340L	40.0	45.9
-375L	25.0	39.6
-410L	40.0	47.0
-445L	48.7	54.5
-480EL	114.4	121.9
-515EL	109.0	114.4
-550EL	103.3	109.0
-585EL	96.4	103.3
-620EL	91.6	96.4
-655EL	87.0	91.6
-690EL	79.9	87.0
-725EL	75.0	79.9
-760EL	71.6	75.0
-795EL	65.4	71.6
-830EL	59.9	65.4
-865EL	57.0	61.4
-900EL	39.6	57.0
-935EL	60.6	61.8
-970EL	61.8	63.0
-1005EL	63.0	65.3

16.7 STOPING

Commercial stoping (production) will start on the -375L during the 40th month and end during the 154th month (12 years and 10months) on the -410L, from the start.

17.0 RECOVERY METHODS

A summary of available metallurgical testwork is presented in Section 13. Based on these data, a conventional process flowsheet is selected, including crushing and grinding to a 70 micron grind at an average rate of 2,000 tpd, followed by flotation recovery of copper to a rougher concentrate. The rougher concentrate is reground and re-floated in a two stage cleaner flotation circuit to yield a final concentrate containing copper at a marketable grade. The concentrate is filtered to an assumed 8% moisture content for shipment. Flotation tailings are leached for recovery of gold in a conventional cyanidation circuit.

18.0 PROJECT INFRASTRUCTURE

The Upper Beaver Project has minimal infrastructure requirements due to its location close to the Highway 66 and Kirkland Lake, Ontario, and due to the infrastructure established during its previous operating history.

18.1 SITE SURFACE INFRASTRUCTURE

Site surface infrastructure requirements for operation would include buildings, buildings furnishings and surface mobile equipment. The site facilities would include a shaft headframe and hoist room/compressor building; a process plant; a paste backfill plant and distribution system; the tailings / waste rock co-disposal basin and dam; site roads; surface parking areas; fuel, lubricates 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. Major surface facilities to support the Upper Beaver mine would include an administration/engineering building, a dry, a warehouse and maintenance shop. Furnishings would include the surface mine shop equipment and tools; the office furniture, computers, etc.; environmental equipment; dry equipment; site communications and medical center equipment. Surface mobile equipment would include a road grader; a service truck; a garbage truck; a personnel bus; an ambulance; a fire/ rescue truck and pickup trucks.

18.2 POWER SUPPLY

Power to the project would be supplied by extending the existing 115-kV line 2 km to a substation then through a new 7 km long 44-kV transmission and communications line to Upper Beaver Mine and Mill Complex. Site overall power consumption is estimated to be approximately 15 MW.

18.3 TAILINGS MANAGEMENT

The conceptual plan for the design of the Tailings Management Facility ("TMF") is to take advantage of the historical tailings facility located approximately four kilometers from the process plant. Separate engineering and environmental studies are currently underway on this facility. The TMF design would incorporate features to manage the chemical and physical stability of the deposited tailings in accordance with existing and new practices. Approximately, 35%-55% of the tailings would be converted to paste backfill and deposited underground. The remaining tailings would be deposited in the TMF.

18.4 WASTE MANAGEMENT

The waste rock dump(s) would be designed, built and closed out so as to minimize long-term impact on the environment. Other waste materials would be recycled (e.g. spent lubricants) or disposed of in accordance with provincial and federal regulations.

18.5 HAZARDOUS MATERIAL STORAGE

Storage facilities for materials such as fuel, explosives and process chemicals have not been detailed at this scoping study level. As the project proceeds, such facilities would be designed to meet all relevant codes and regulations in order to protect employees, the public and the environment.

18.6 REGIONAL RESOURCES

The regional labour force includes experienced equipment operators, mine workers and material and equipment suppliers.

19.0 MARKET STUDIES AND CONTRACTS

There were no market studies completed or contracts in place in support of this Technical Report.

However, the commercial products produced by this project will be gold bullion for shipment to any of several available refineries and a saleable copper concentrate. Prices for these products will be based on the then-current copper and gold prices less respective smelting and refining charges.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project as described in this PEA would be implemented in stages. The first stage (underground exploration and bulk sampling) would involve exploration shaft sinking; lateral development and bulk sampling; the dewatering of the proximate historic Upper Beaver mine workings; and infrastructure construction. Queenston has commenced relevant environmental and social baseline information collection and has initiated and continued discussions with local communities, First Nations, Métis community and other interested people. The second stage (underground mining and on-site processing) would require additional permitting.

P&E has assessed the nature and scope of the Project; available environmental and social base line information including public and aboriginal consultation information; relevant regulatory requirements with an emphasis on environmental assessment and permitting requirements; the potential for significant impacts; and the proposed approach to Project development, operation and closure.

Queenston has identified advanced exploration permitting, environmental assessment and operations permitting and closure planning requirements for the Project in consultation with regulators, and commenced baseline environmental studies and consultations with local communities, aboriginal peoples and other people that could potentially be impacted by the proposed Project.

20.1 PROJECT DESCRIPTION

The Project is situated approximately 22 km east of Kirkland Lake in an area that includes the historic Upper Beaver underground mine that operated between 1914 and 1971, and a historic tailings facility. The historic workings include a 200 m deep main shaft, a winze and about ten levels. The project site is accessible from the Village of Dobie using Beaverhouse Lake Road which is also used by the Beaverhouse First Nation, anglers, hunters, cottagers and loggers. The scope of the Project for the purposes of the present PEA includes:

- An exploration shaft sinking, lateral development and bulk sampling program to confirm the mineability / continuity of the deposit. The historic Upper Beaver Mine workings would be dewatered for safety reasons. The circular concrete lined exploration shaft would be sized to accommodate projected future production hoisting and mine servicing requirements;
- Access road improvements;
- The development of a 2 km long, 115 kV line power distribution line from an existing transmission line to a new substation and a 7 km long, 44kV line transmission line from the substation to the proposed mine and mill site;
- The construction of a 2,000 tpd capacity crushing, floatation and CIL (carbon in leach) mineral processing plant. The projected gold-rich copper concentrate would be shipped to a smelter off-site;
- The construction of a tailings management facility (TMF) at an historic tailings facility site. A cyanide destruction process would be used to treat the tailings slurry prior to disposal. The amount of tailings requiring disposal in the TMF would be reduced by utilizing approximately 35% to 55% of the mill tailings as underground paste fill. The TMF and effluent treatment facility would be sized to accommodate at a minimum 1 in 100 year rain on snow precipitation event;

- The construction of ancillary facilities such as an administration and technical services building, security, weigh station, maintenance shop, warehouse, fuel storage, explosive storage, fire protection, and effluent treatment facilities;
- Pre-production underground mine development including shaft deepening;
- Underground mining (primarily long hole stoping with paste backfill); and
- Closure and rehabilitation works. The project site would be left in a physically and chemically stable condition offering a return to other productive land use. Waste rock would be disposed at the McBean open pit waste rock stockpile located approximately 5 km away from the Project site.

The above referenced scope encompasses a number of environmental best management practices and demonstrated environmental controls such as the use of a cyanide destruction process.

20.2 ENVIRONMENTAL BASE LINE

Environmental base line studies to support the permitting process have commenced. The key results of environmental and social baseline assessment as reported by Story (2011) are summarized below:

- Approximately 90% of the Project area is situated within the Township of Gauthier which has a population of 133 based on 2006 census information. There are 53 private residences and 12 cottages in the Township. The balance of the Project area is situated in a largely uninhabited area of area of McVittie Township.
- Landforms in the Project area are predominately bedrock-dominated, glaciofluvial deposits or glaciolacustrine deposits. The local climate is humid continental with warm summers. The forest is dominated by black spruce, poplar, tamarack and balsam fir. Forest harvesting is carried out near and within the Project Area.
- The Project area is a low relief area with sections of undulating to rolling terrain with occasional plateaus, and creeks, rivers and lakes. The major lakes in the Project area are Beaverhouse Lake and Little Larder Lake. The major water courses are the Misema River and Victoria Creek. No provincially significant wetlands have been identified in or adjacent to the Project area. There is no commercial fishing within the Project area. People recreationally and traditionally angle for warm/cool water game fish including pickerel (walleye), northern pike, smallmouth bass and yellow perch in the Project area.
- Wildlife habitats in the Project area include waterfowl brood rearing habitat, moose calving sites, moose aquatic feeding areas, and late winter moose habitat. Whip-poor-will, one of the species classified as at risk in the Kirkland Lake District and that may reside within the Project Area, are observed at the Project site. No known migration routes or important waterfowl nesting areas are known to exist in the Project area.
- The historic Upper Beaver mine, Upper Canada mine, Anoki and McBean mine operated in the project area. The historic tailings area which is predominately associated with the historic Upper Canada and Anoki McBean mines includes a confined tailings impoundment area and older unconfined tailings that extend into Little Larder Lake. Story (2011) reports that the impact of the discharge from these tailings areas on the quality of water in the downstream receiver (Victoria Creek) is minimal as only the concentration of total iron exceeded the Provincial Water Quality Objectives at water sampling station V3 in one of four sampling

events. The extent to which iron concentrations at Station V3 are influenced by elevated background iron concentrations is to be assessed as part of planned follow-up studies. Groundwater quality sampling in the Project area has commenced.

- The results of acid:base accounting (ABA) tests completed to date indicate that the mill tailings would be non-acid generating. Samples of waste rock to be collected from the exploration shaft pilot hole and from underground headings will be used to characterize the acid generation potential of the mine waste rock. The results of ABA tests of two samples of waste rock obtained from the historic McBean mine waste rock stockpile indicate that the waste rock is net acid consuming and not acid generating.

20.2.1 Public and Aboriginal consultation

Queenston has started to build / renew its relationships with several First Nations in the region including the Wahgoshig, Beaverhouse, Matachewan and the Temiskaming First Nations, and has been actively engaging the Métis community through the Métis Region 3 Consultation Committee representing regional Métis councils on resource consultation. Queenston's consultation program also extends to associated governing and service support organizations (e.g. Tribal Councils) other interested people and local governments (Story, 2011).

20.2.2 Permitting

Queenston is working to complete the permitting procedure for an Advanced Exploration Permit for its planned exploration shaft sinking and bulk sampling program, and has submitted a Notice of Project Status for its advanced exploration program to the Ministry of Northern Development and Mines (MNDM). Queenston is currently preparing its closure plan for submittal to the MNDM.

Queenston is consulting with regulators with regard to the environmental permitting requirements for the envisaged underground mining and on-site milling operations. The environmental permitting of the proposed mining and processing operations would require an environmental assessment with terms of reference established under a cooperative agreement for projects requiring environmental assessment and approvals under Federal and Provincial environmental assessment legislation. While the terms of reference for the environmental assessment process have yet to be established:

- Queenston has commenced base line studies and plans to continue them in support of an environmental assessment of the Project.
- Public consultation is a required component of the environmental assessment and closure planning processes. Queenston has already commenced public and aboriginal consultations, and plans to continue to work cooperatively with aboriginal communities as the Project scope, impacts and benefits become better understood both at the Advanced Exploration and Production phases.

It is expected that following the environmental assessment process for the proposed producing mine and mill and regulatory approval to proceed, Queenston would need to apply for required permits including but not limited to certificates of approvals for discharges to air and water including treated process water and treated domestic sewage.

20.2.3 Mine Closure

The present PEA is based on the Project being progressively decommissioned and closed out at the end of the mine life. The envisaged closure works include: the removal of mine equipment and recoverable services to surface and their sale or proper disposal; the capping of the mine opening at surface; the dismantling and sale of the mill; the demolition of surface infrastructure components that are not salvaged and sold; and the proper disposal of unused fuel, lubricants and chemicals, and non-hazardous and hazardous solid and liquid wastes. The project site would be rehabilitated and left in a physically and chemically stable environment. The closure approach is based on using passive environmental controls to eliminate the need for active post-closure controls. Environmental monitoring results would be used to assess and demonstrate the effectiveness of the closure and rehabilitation works.

Before the proposed mining and on-site mineral processing operations can begin, Queenston would be required to have its closure plan and financial assurance approved by the Ministry of Northern Development and Mines. The present PEA includes a $3.6M financial assurance cost allowance and closure and rehabilitation costs amounting to approximately $6M.

21.0 CAPITAL AND OPERATING COSTS

All capital and operating costs are in Canadian dollars, unless otherwise stipulated.

21.1 CAPITAL COST ESTIMATES

21.1.1 Pre-production Capital Cost Estimates

The pre-production period starts with site clearing and collaring of the shaft and ends when the shaft is commissioned and stope mining starts. Pre-production capital costs include the cost of all surface building, structures and related facilities; mine and stope development on the -340, -375 and -410 Levels; shaft equipment and related facilities; underground mining equipment; surface mobile equipment; electrical power supply infrastructure; underground infrastructure related to the shaft and -340, -375 and -410 Levels; most of the project closure bond and a 15% contingency. The total estimated pre-production capital cost is estimated to be $240.1 M. Details of the Capital Cost Estimate and schedule for the pre-production period are provided in Table 21.1.

TABLE 21.1
SUMMARY OF PRE-PRODUCTION CAPITAL COST ESTIMATES

Description	Year				Total
	1	2	3	4	
Mine & Stope Development			16.1	6.5	22.6
Shaft Development		24.3	24.5	1.2	50.0
Shaft Headframe, Hoist & Hoist Room, LP	1.0	11.4	2.0		14.5
Mine Equipment		11.3	7.1		18.3
U/G Infrastructure			2.7	0.1	2.8
Surface Infrastructure	11.4	17.3			28.7
Process Plant	45.7	22.8			68.5
Closure Bond	3.0	0.3	0.3	0.1	3.6
Contingency (15%)	9.2	13.1	7.9	1.0	31.2
Total	**70.3**	**100.4**	**60.5**	**8.9**	**240.1**

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

21.1.2 Sustaining Capital Cost Estimates

The commercial production period starts in the second quarter of the fourth year, from the start, and continues until the third quarter of the thirteenth year. Sustaining capital costs during this period include mine and stope development for the rest of the mine; underground infrastructure related to the rest of the mine; some of the project closure bond; a salvage value in Year 13 and a 15% contingency. The total estimated sustaining capital cost is estimated to be $Cdn178.0 M. Details of the Capital Cost Estimate and schedule for the commercial production period are provided in Table 21.2.

TABLE 21.2 SUMMARY OF SUSTAINING CAPITAL COST ESTIMATES											
Description	**Year**										**Total**
	4	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	
Mine & Stope Development	19.4	26.1	24.9	23.2	21.7	22.6	17.0	2.1	0.4	0.3	157.7
U/G Infrastructure	0.4	0.5	0.5	0.5	0.5						2.5
Closure Bond & Salvage	0.2	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	-7.8	-5.6
Contingency (15%)	3.1	4.0	3.9	3.6	3.4	3.4	2.6	0.4	0.1	-1.1	23.3
Total	**23.1**	**30.9**	**29.5**	**27.6**	**25.9**	**26.3**	**19.8**	**2.7**	**0.8**	**-8.6**	**178.0**

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

Details of these estimates are provided in the following subsections.

21.1.3 Mine and Stope Development Capital Costs

Mine and stope development costs include the cost of all underground development in both waste rock and ore, excluding all slot raises and shaft and shaft related excavations. This includes: the cost of all internal access ramps; drifting in ore; all crosscuts to the stoping areas; ore passes and ventilation raises. A summary of mine and stope development capital costs is presented in Table 21.3.

TABLE 21.3 SUMMARY OF MINE AND STOPE DEVELOPMENT CAPITAL COSTS ESTIMATES			
Item	**Unit Cost ($/m)**	**Units (m)**	**Total Cost**
Ramp	5,000	10,611	53.1
Drift in Ore	4,500	15,019	67.6
Cross-cuts	4,500	12,655	56.9
Orepass	2,000	925	1.9
Ventilation Raise	4,000	217	0.9
Total		**39,426**	**180.3**

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

21.1.4 Shaft Development Capital Costs

Once the shaft collar has been excavated to approximately 60 metres below surface and the headframe and hoist room are installed and commissioned, shaft sinking can begin, 15 months from the start of construction. The vertical 6.5m diameter concrete lined shaft would be sunk conventionally sunk from the bottom of the collar, at approximate elevation +242m, to the -1005m elevation, a vertical distance of 1,247m. There would be two loading pockets installed, one below the -375L Station and the other below the -900L Station. A temporary mechanical lip pocket would be installed at the -375L Station to facilitate hoisting development waste while the shaft is being sunk from the -375L Station to shaft bottom at the -1005m elevation. It would take approximately 750 days to sink and commission the shaft. A summary the shaft development capital cost and schedule is presented in Table 21.4.

TABLE 21.4 SUMMARY OF SHAFT DEVELOPMENT CAPITAL COSTS ESTIMATES						
Description	**Quantity (m or m^3)**	**Year (M$)**				**Total ($M)**
		1	**2**	**3**	**4**	
Collar to 45L Station	197		7.4			7.4
45L Station	900		0.5			0.5
45L Station to -80L Station	125		4.7			4.7
-80L Station	900		0.5			0.5
-80L Station to -165L Station	85		3.2			3.2
-165L Station	900		0.5			0.5
-165L Station to -270L Station	105		3.9			3.9
-270L Station	900		0.5			0.5
-270L Station to -375L Station	105		3.2	0.7		3.9
-375L Station	900			0.5		0.5
-375L Station Mechanical Lip Pocket				0.3		0.3
-375L Station to Loading Pocket No1	45			1.4		1.4
Loading Pocket No1	300			0.2		0.2
Install Loading Pocket				0.5		0.5
Loading Pocket Raise	250			0.1		0.1
Loading Pocket No1 to Spill Pocket No1	15			0.5		0.5
Spill Pocket No1	150			0.1		0.1
Spill Pocket No1 to -480 Station	45			1.4		1.4
-480L Station	900			0.5		0.5
-480L Station to -585L Station	105			3.3		3.3
-585L Station	900			0.5		0.5
-585L Station to -690L Station	105			3.3		3.3
-690L Station	900			0.5		0.5
-690L Station to -795L Station	105			3.3		3.3
-795L Station	900			0.5		0.5
-795L Station to -900L Station	105			3.3		3.3
-900L Station	900			0.5		0.5
-900L Station to Loading Pocket No2	45			1.4		1.4
Loading Pocket No2	300			0.2		0.2
Install Loading Pocket				0.5		0.5
Loading Pocket Raise	250			0.1		0.1
Loading Pocket No2 to -1005L Station	60			1.6	0.3	1.9
-1005L Station	900			0	0.5	0.5
Remove Sinking Geer & Commission Shaft				0	0.5	0.5
Total			**24.3**	**24.5**	**1.2**	**50.0**

****Note:*** *Some values have been rounded. The totals are accurate summations of the columns of data.*

21.1.5 Shaft Headframe, Loading Pockets, Hoists and Hoistroom Capital Costs

A summary the capital costs of the shaft headframe, two loading pocket with rockbreakers and grizzlies, two hoists and hoist room, and schedule of purchases, is presented in Table 21.5.

TABLE 21.5
SUMMARY OF SHAFT HEADFRAME, LOADING POCKETS, HOISTS AND HOISTROOM CAPITAL COSTS ESTIMATES

Description	Unit Cost (M$)	Units	Total Cost (M$)	Year		
				1	2	3
Headframe, Hoistroom, Hoists(2)	12.5	1	12.5	1.0	11.4	
Loading Pocket	0.5	2	1.0			1.0
Grizzly / Rockbreaker	0.3	4	1.0			1.0
Total			**14.5**	**1.0**	**11.4**	**2.0**

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

21.1.6 Mine Equipment Capital Costs

The mine equipment capital costs include: all underground mobile and stationary equipment and all related mine surface equipment. A summary the underground mine equipment capital costs, and schedule of purchases, is presented in Table 21.6.

TABLE 21.6
SUMMARY OF MINE EQUIPMENT CAPITAL COSTS ESTIMATES

Description	Unit Cost ($)	Units	Total Cost (M$)	Year	
				2	3
Sandvik Axera 7-260 Development Jumbo - 2 Boom	900,000	2	1.8	0.9	0.9
Cubex ITH Drill	1,000,000	2	2.0	1.0	1.0
Getman Scissor Lift	370,000	1	0.4	0.4	0.0
Sandvik T9 - 6.1 cu.m. LHD	1,225,000	3	3.7	1.2	2.5
EJC30SX	700,000	3	2.1	0.7	1.4
U/G Blasting Tractor	550,000	1	0.6		0.6
Getman ANFO Loader	440,000	1	0.4	0.4	
Cable Bolter	750,000	1	0.8	0.0	0.8
Getman Lube Service Vehicle	340,000	1	0.3	0.3	
M40 Fuel truck	375,000	1	0.4	0.4	
Mechanics Vehicle	55,000	1	0.1	0.1	
Electrician Vehicle	55,000	1	0.1	0.1	
Getman Boom Truck	325,000	1	0.3	0.3	
Grader	370,000	1	0.4	0.4	
Toyotas	55,000	3	0.2	0.2	
Alimak	300,000	1	0.3	0.3	
Shotcrete Machine	100,000	1	0.1	0.1	
Getman Personnel Carrier	300,000	1	0.3	0.3	
Misc. Underground Equipment	Lot		2.1	2.1	
Misc. Surface Equipment	Lot		2.2	2.2	
Mine Equipment Total			18.3	11.3	7.1

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

21.1.7 Processing Plant Capital Costs

The capital costs of the process plant include direct costs such as site preparation, all concrete work, all structural work, process plant equipment and installation, piping, and all electrical equipment and instrumentation. Indirect process plant capital costs include field supervision and expenses, construction equipment, engineering design and layouts, spare parts and commission costs. A summary of the process plant direct and indirect capital costs is presented in Table 21.7. The estimated capital cost of a gravity circuit has been included in these costs.

TABLE 21.7
PROCESS PLANT CAPITAL COST SUMMARY

Description	Total Cost
Direct Cost	48.3
Indirect Costs	
Field Supervision	1.7
Field Expense	2.1
Temporary Facilities	1.4
Construction Equipment	1.2
Craft Benefits	3.0
Subtotal Construction Indirect	9.4
Engineering	6.4
Freight	1.0
Spare Parts	0.7
Startup	0.2
Subtotal Project Indirect	17.8
Direct + Indirect	66.0
Gravity Circuit	2.5
Subtotal	68.5

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

The Process Plant construction expenditures are expected to occur in year 1 (2/3 of cost) and year 2 (1/3 of the cost).

21.1.8 Surface Infrastructure Capital Costs

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

The capital cost of site facilities includes the cost of: the electric power line, substation, switchgear; the paste backfill plant and distribution system; the tailings / waste rock co-disposal basin and dam; site roads; surface parking areas; the fuel storage; 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 building 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 grader; a front-end loader, a service truck; a garbage truck; a personnel bus; an ambulance; a fire/ rescue truck and pickup trucks. The surface infrastructure capital cost summary is presented in Table 21.8.

TABLE 21.8
SURFACE INFRASTRUCTURE CAPITAL COST SUMMARY*

Description	Estimated Cost (M $)
Site Facilities	23.3
Buildings	2.4
Buildings Furnishings	1.8
Surface Mobile Equipment	1.2
Total	**28.7**

**Some values have been rounded. The totals are accurate summations of the columns of data.*

21.1.9 Mine Closure and Salvage Capital Costs

A closure bond will be required to remove the process plant, for final tailings construction and seeding; the tailings spillway, final water treatment and remove surface infrastructure and final clean up. It is estimated it will cost $6.0 M to complete this work. Details of this cost estimate are presented in Table 21.9.

TABLE 21.9
MINE CLOSURE CAPITAL COST SUMMARY

Description	Total ($)
Remove headframe, collar house, hoists(2) and hoisthouse; Secure Surface Openings	In Salvage
Remove process plant	4,000,000
Final tailings dam work - 10ha @ $80k/ha plus $50k for design work	850,000
Spillway	30,000
Final water treatment (batch)	50,000
Remove surface infrastructure / clean-up	1,000,000
Total	**5,930,000**
Say	**6,000,000**

Most of this closure bond will be required during the pre-production period. The balance is spread out, on a yearly basis, over the life of the mine.

The capital cost of removing the shaft headframe, collar house, hoists and hoist room and securing the surface underground mine openings is estimated to be somewhat offset by the salvage value of these facilities. The salvage value, after removal, of the shaft headframe, collar house, hoists, process plant equipment, mine equipment and surface infrastructure is summarized in Table 21.10.

TABLE 21.10 SALVAGE VALUE SUMMARY*	
Item	**Salvage Value (M $)**
Hoists(2)	1.7
Headframe and Collar House	1.1
Plant Equipment	1.4
Mine Equipment	1.5
Surface Infrastructure	2.3
Total	**8.0**

**Some values have been rounded. The totals are accurate summations of the columns of data.*

21.1.10 Contingency Capital Costs

A 15% contingency capital cost on all capital cost items has been estimate separately.

21.2 OPERATING COST ESTIMATES

Operating costs include the cost of operating labour, maintenance labour, electrical power, operating materials and supplies, reagents and fuel. The yearly operating cost varies from $70.17 to $77.05 per tonne milled. A summary of the average operating cost estimates for the Upper Beaver Project is provided in Table 21.11.

TABLE 21.11 SUMMARY OF AVERAGE OPERATING COST PER TONNE MILLED	
Description	**Total ($/t)**
Stope Mining	20.70
Paste Backfill	7.00
Tailings to Tailings Dam	1.00
Tailings Pond Water Treatment	0.32
Process Plant	17.81
U/G Haulage	3.50
U/ G Hoisting Services Costs	1.50
Mine Air Heating	3.05
G&A COSTS	6.00
Contingency (20%)	12.18
Total Operating	**73.06**

Details of these estimates are provided in the subsections that follow.

21.2.1 Mining

On average 1,749 tpd of mill feed would be mined by stoping. The balance of 251 tpd would be extracted by stope development, for a total of 2,000 tpd.

Stope mining operating costs include the cost of material, consumables and labour for stope drilling, blasting, mucking, pipe and accessories, and stope ventilation. The total mining cost also includes the paste backfill cost, haulage cost, hoisting services cost and mine air heating cost. The estimated operating cost, per tonne of stope ore mined, is summarized in Table 21.12. The stope development costs have been included in the capital costs for the mine.

TABLE 21.12
SUMMARY OF MINE OPERATING COST

Description	Per Stope Tonne ($)	Per Tonne Milled ($)
Drilling & Blasting	4.62	
Slot Raise	0.20	
Ground Support	0.52	
Mucking	1.63	
Pipe & Accessories	0.06	
Stope Fan	0.07	
Total Stoping Consumables	**7.10**	
Services and Power	7.20	
Staff Labour	3.73	
Hourly Labour	5.65	
Total Stoping	**23.68**	**20.70**
Paste Backfill		7.00
U/G Haulage		3.50
U/G Hoisting Services Costs		1.50
Mine Air Heating		3.05
Total Mining (Expensed)		**35.75**

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

21.2.2 Mineral Processing

On average 2,000 tpd mill feed will be processed. The mineral processing operating cost includes the cost of all material, consumables and labour required to process 7,000,000 tonnes per year. This includes all operating and maintenance labour, electrical power requirements, reagents, operating and maintenance supplies. A summary of process plant operating costs, per tonne milled and total cost per year, is presented in Table 21.13.

TABLE 21.13 SUMMARY OF MINERAL PROCESSING OPERATING COST		
Item	**$/t**	**$/Year**
Operating Labour	4.67	3,270,400
Power	3.64	2,546,100
Reagents	4.98	3,485,300
Operating Supplies	1.08	758,100
Maintenance Labour	2.02	1,416,600
Maintenance Supplies	1.41	990,300
Total	**17.81**	**12,466,800**

Note: *Some values have been rounded. The totals are accurate summations of the columns of data.*

21.2.3 Other Operating Costs

In addition to the mining and processing operating costs, above, there is a 'Tailing-to-Tailings-Dam' cost, a tailings pond water treatment cost and a general and administration cost. The 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. In addition to these operating costs a 20% contingency on the total operating costs has been added. A summary of these costs per tonne milled is presented in Table 21.14.

TABLE 21.14 SUMMARY OF OTHER OPERATING COSTS	
Description	**Per Tonne Milled ($)**
Tailings to Tailings Dam	1.00
Tailings Pond Water Treatment	0.32
General and Administration	6.00
Contingency (20%)	12.18
Total Other	**19.50**

22.0 ECONOMIC ANALYSIS

This Report is considered by P&E Mining Consultants Inc. to meet the requirements of a Technical Report as defined in Canadian NI 43-101 regulations. This PEA is preliminary in nature and includes Inferred 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 PEA will be realized. There is no guarantee that Queenston will be successful in obtaining any or all of the requisite consents, permits or approvals, regulatory or otherwise for the Deposit to be placed into production.

22.1 ECONOMIC CRITERIA

22.1.1 Physicals

Mine life:	
Pre-production	39 months
Production Mining/Milling	Year 4 to 13 for a total of 9.5 years
Decommissioning	6 months in Year 13
Production rate	2,000 t per day
Total production:	
Total ore production	6,839,800 t at 0.37 % Cu & 5.1 g/t Au
Total concentrate production	123,100 t
Metallurgical parameters:	
Process recovery	98% Au and 90% Cu
Concentration ratio	56
Concentrate grade	18.6% Cu
Concentrate moisture content	8%
Total payable metal:	
Gold	1,055,200 oz. of Au
Gold	1,162,300 oz. of AuEq
Copper	22,600 t of Cu

22.1.2 Revenue

The commercial products produced by the project are copper concentrate and doré. Queenston will be paid once the copper concentrate and doré has been delivered to the smelter and refinery, off-site. The gold and copper prices used in this PEA are US$1,275/oz. Au and US$3.00/lb Cu. Revenues were calculated as Net Smelter Returns (NSR's). The NSR payables were based on the following parameters.

Smelter treatment charge	CDN$/DMT:$125.00/t
Concentrate shipping charge	CDN$/WMT:10.00/t
Smelter payable	95% Au and 90% Cu
Refining charges	CDN$/DMT:$10.00/oz. Au, 0.07/lb Cu

The US$/CDN$ exchange rate used in the PEA is 0.96.

Net revenue:

Gold / Copper:	$1,513.0 million

22.1.3 Costs

Operating costs:

Total average cost:	$73.06 per t ore milled
Cash Cost	US$415.99/oz. AuEq

Capital costs:

Preproduction	$240.0 million
Sustaining	$178.0 million
Total capital costs	$418.1 million

These capital costs include the cost of; mine and stope development; the shaft headframe, hoists, hoist room, shaft stations and loading pockets; the surface power line; mine equipment; surface infrastructure; underground infrastructure; the process plant, a closure bond, salvage value and a 15% contingency.

22.2 CASH FLOW

An after-tax cash flow (CF) model has been developed for the Upper Beaver Project. The model does not take into account the following components:

- Financing cost, other than interest included in capital lease rates
- Insurance
- Overhead cost for a corporate office

Taxes are estimated to be 30% of pre-tax cash flow. A cash flow summary is presented in Table 22.1. All costs are in 1st quarter 2012 Canadian dollars with no allowance for inflation.

TABLE 22.1
SUMMARY OF OTHER OPERATING COSTS

Description	Units / Year	1	2	3	4	5	6	7	8	9	10	11	12	13	Total
Waste	t(000's)			140	200	200	200	200	200	200	200	22	0	0	1,562
Development Ore	t(000's)			77	148	151	135	112	93	105	29	6	6	4	865
Cu	%			0.44	0.35	0.47	0.40	0.29	0.17	0.20	0.13	0.12	0.18	0.25	0.33
Au	g/t			5.66	5.14	5.00	5.27	5.23	4.64	4.11	3.56	3.86	4.15	4.05	4.94
Stope Ore	t(000's)			0	501	549	565	588	607	595	671	694	694	564	6,028
Cu	%			0.00	0.37	0.44	0.56	0.56	0.44	0.31	0.18	0.21	0.33	0.40	0.37
Au	g/t			0.00	5.68	6.16	6.02	5.39	4.80	5.71	3.80	4.36	5.06	4.94	5.14
Total	**t(000's)**			**77**	**649**	**700**	**700**	**700**	**700**	**700**	**700**	**700**	**700**	**567**	**6,894**
Cu	%			0.44	0.37	0.45	0.53	0.52	0.40	0.30	0.18	0.21	0.33	0.40	0.37
Au	g/t			5.66	5.56	5.91	5.87	5.36	4.78	5.47	3.79	4.35	5.05	4.93	5.11
NSR	$/t			244.95	236.83	255.13	258.20	237.29	208.10	229.45	157.08	180.94	215.08	214.08	219.47
Revenue	$M			18.8	153.8	178.6	180.7	166.1	145.7	160.6	110.0	126.7	150.6	121.5	1,513.0
Stope Mining	$M			0.0	11.9	13.0	13.4	13.9	14.4	14.1	15.9	16.4	16.4	13.3	142.7
Paste Backfill	$M			0.5	4.5	4.9	4.9	4.9	4.9	4.9	4.9	4.9	4.9	4.0	48.3
Tailings to Tailings Dam	$M			0.1	0.6	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.6	6.9
Tailings Pond Water Treatment	$M			0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	2.2
Process Plant	$M			1.4	11.6	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	10.1	122.8
U/G Haulage	$M			0.3	2.3	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.0	24.1
U/ G Hoisting Services Costs	$M			0.1	1.0	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.9	10.3
Mine Air Heating	$M			1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	21.0
G&A COSTS	$M			0.5	3.9	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	3.4	41.4
Contingency (20%)	$M			0.8	7.6	8.2	8.3	8.4	8.5	8.4	8.8	8.9	8.9	7.3	83.9
Total Operating	**$M**			**4.8**	**45.6**	**49.2**	**49.6**	**50.3**	**50.8**	**50.5**	**52.6**	**53.3**	**53.3**	**43.7**	**503.6**
Mine & Stope Development	$M			16.1	25.9	26.1	24.9	23.2	21.7	22.6	17.0	2.1	0.4	0.3	180.3
Shaft Development	$M		24.3	24.5	1.2										50.0
Shaft Headframe, Hoist & Hoist Room, LP	$M	1.0	11.4	2.0											14.5
Mine Equipment	$M		11.3	7.1											18.3
U/G Infrastructure	$M			2.7	0.5	0.5	0.5	0.5	0.5						5.3
Surface Infrastructure	$M	11.4	17.3												28.7
Process Plant	$M	45.7	22.8												68.5
Closure Bond &	$M	3.0	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	-7.8	-2.0

TABLE 22.1 SUMMARY OF OTHER OPERATING COSTS															
Description	**Units / Year**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**Total**
Salvage															
Contingency (15%)	$M	9.2	13.1	7.9	4.2	4.0	3.9	3.6	3.4	3.4	2.6	0.4	0.1	-1.1	54.5
Total Capital	**$M**	**70.3**	**100.4**	**60.5**	**32.1**	**30.9**	**29.5**	**27.6**	**25.9**	**26.3**	**19.8**	**2.7**	**0.8**	**-8.6**	**418.1**
Pre-tax Cash Flow	$M	-70.3	-100.4	-46.4	76.2	98.5	101.6	88.3	69.0	83.8	37.5	70.7	96.5	86.4	591.3
Cumulative Pre-tax Cash Flow	$M	-70.3	-170.7	-217.2	-141.0	-42.5	59.1	147.4	216.4	300.2	337.7	408.4	504.9	591.3	
After Tax Cash Flow	$M	-70.3	-100.4	-46.4	76.2	98.5	83.9	61.8	48.3	58.7	26.3	49.5	67.5	60.5	413.9
Cumulative After Tax Cash Flow	$M	-70.3	-170.7	-217.2	-141.0	-42.5	41.4	103.2	151.5	210.1	236.4	285.9	353.4	413.9	
After Tax IRR	%														22.1%
After Tax NPV @ 5%	$M														233.4

22.3 BASE CASE CASH FLOW ANALYSIS

The following after tax cash flow analysis was completed:

- Net Present Value NPV (at 0%, 5% 7% and 10% discount rate)
- Internal Rate of Return IRR
- Payback period

The summary of the results of the cash flow analysis is presented in Table 22.2.

TABLE 22.2
BASE CASE CASH FLOW ANALYSIS

Description	Discount Rate	Units	Value
Non Discounted After Tax CF		(M$)	413.9
Internal Rate of Return		%	22.1%
NPV at	0%	(M$)	413.9
	5%	(M$)	233.4
	7%	(M$)	183.3
	10%	(M$)	124.4
Project Payback Period in Years		Years	2.26

The project was evaluated on an after-tax cash flow basis and generates a net cash flow of $413.9 million. This results in an after tax Internal Rate of Return (IRR) of 22.1% and an after-tax Net Present Value (NPV) of $233.4 million when us

ing a 5% discount rate. In the base case scenario, the project has a payback period of 2.3 years from start of commercial production. The average life-of-mine cash cost is US$416/oz. gold, net of copper credits, at an average operating cost of $73.06 per ore tonne ore processed.

22.4 SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities to:

- Gold metal price
- Operating costs
- Capital costs, and
- Discount Rate

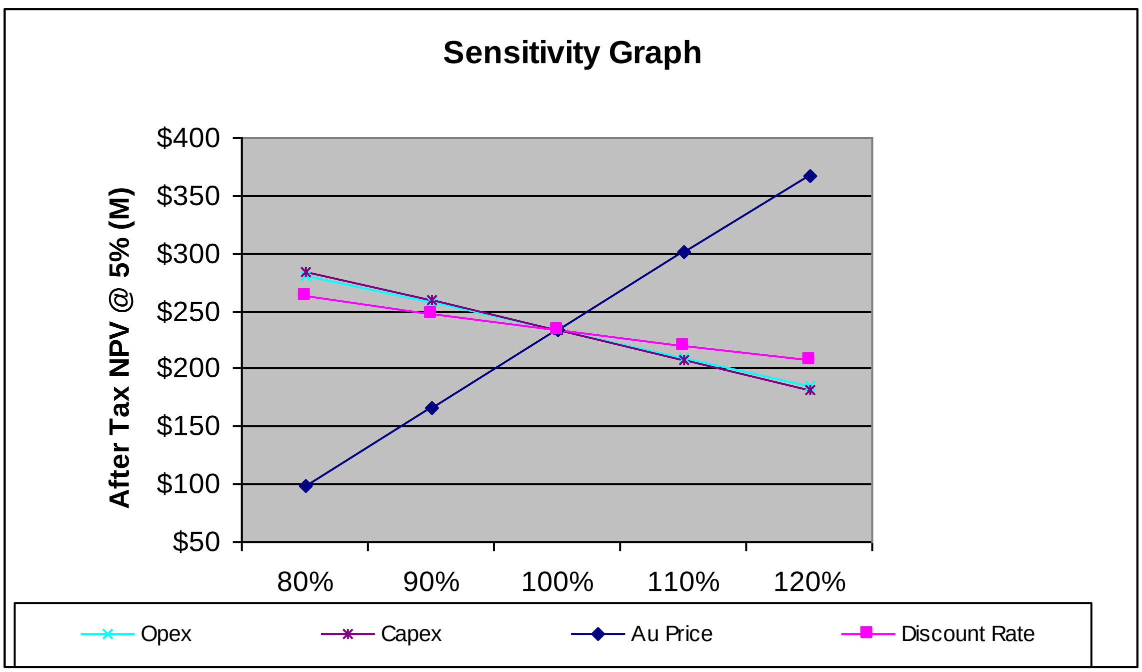
To determine what this project is most sensitive to, each of the sensitivity items were adjusted up and down by 10% and 20% to see what effect it would have on the NPV at a 5% discount rate. The value of each sensitivity item, at 80%, 90%, base, 110% and 120%, is presented in Table 22.3.

TABLE 22.3 SENSITIVITY ITEM VALUES					
Item	**80%**	**90%**	**100%**	**110%**	**120%**
Opex ($/t)	$58.44	$65.75	$73.06	$80.36	$87.67
Capex (M$)	$334.5	$376.3	$418.1	$459.9	$501.7
Au Price (US$/oz)	$1,020	$1,148	$1,275	$1,403	$1,530
Discount Rate (%)	4.0%	4.5%	5.0%	5.5%	6.0%

The resultant after-tax NPV @ 5% value of each of the sensitivity items at 80% to 120% is presented in Table 22.4 and Figure 22.1. This after-tax base case NPV is most sensitive to gold metal price followed by capital costs, operating costs and discount rate

TABLE 22.4 SUMMARY OF SENSITIVITY ANALYSIS					
NPV @ 5%					
At The Sensitivity Item Values (Table 22.3) (M$)					
Item	**80%**	**90%**	**100%**	**110%**	**120%**
Opex	$280.8	$257.1	$233.4	$209.7	$186.0
Capex	$284.2	$258.9	$233.4	$207.9	$182.4
Au Price	$98.8	$166.3	$233.4	$300.6	$367.4
Discount Rate	$262.5	$247.6	$233.4	$220.0	$207.1

Figure 22.1 Sensitivity Graph



23.0 ADJACENT PROPERTIES

Note: *The authors of this section have drawn heavily upon selected portions or excerpts from material contained in the WGM 2011 Report.*

The Kirkland Lake area has been an active exploration area for more than 100 years and has produced more than 30 million ounces of gold from multiple operations. Queenston's Kirkland Lake Project, which includes the Upper Beaver Property, consists of a large block of claims within the Kirkland Lake gold camp. The camp extends for some 50 km and encompasses five townships from the Town of Kirkland Lake in Teck Township, to the Quebec border. The Kirkland Lake Project itself is large, covering almost 1,200 mining claims in the historic gold camp. Geologically, the Kirkland Lake gold camp is defined by a 5 km corridor around the Cadillac–Larder Lake Break from Kirkland Lake to the Quebec border. The properties that are truly adjacent to the Upper Beaver Property are Lac McVittie, Upper Canada and Victoria Creek (Figure 23.1).

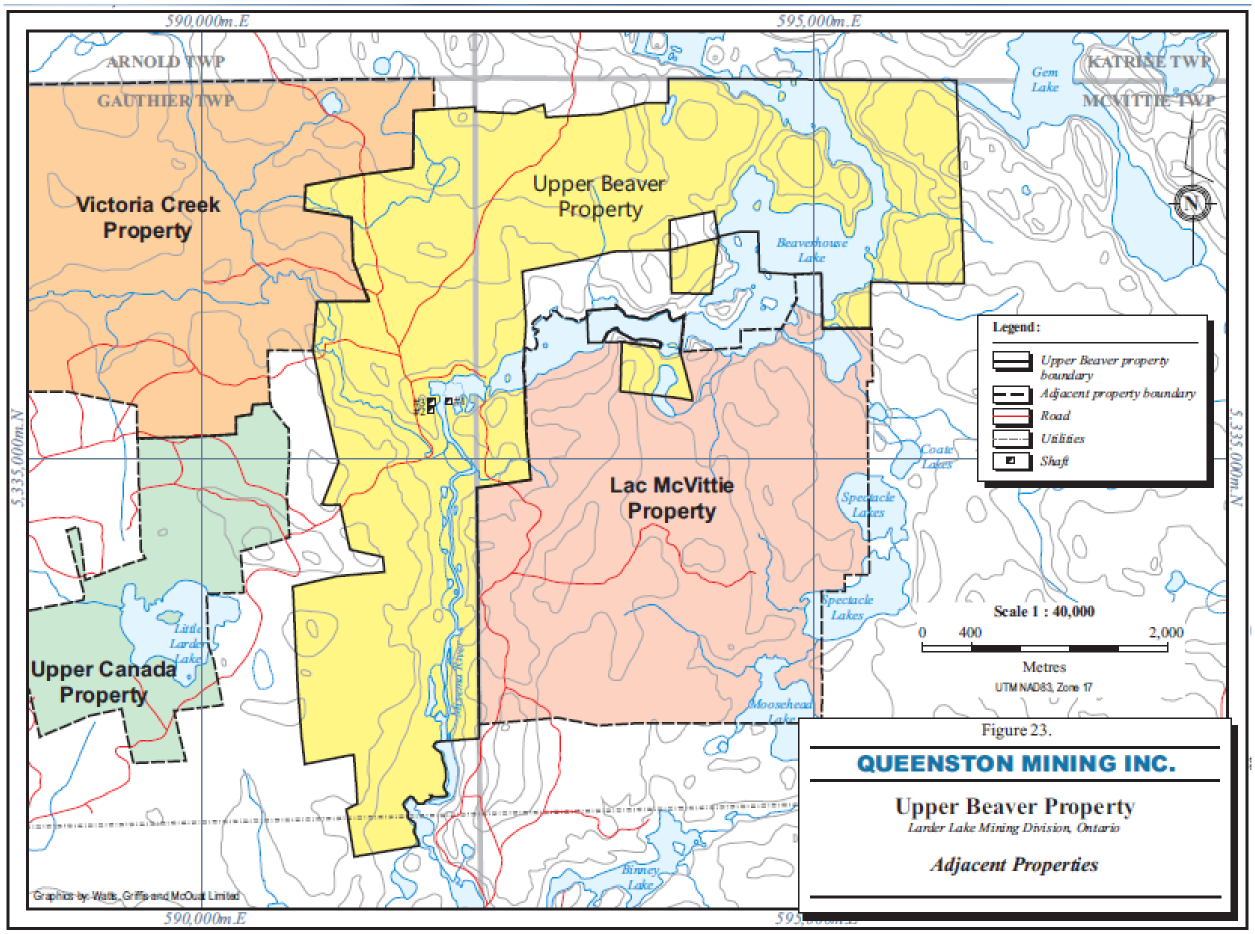
23.1 LAC MCVITTIE

The following is a description of the Lac McVittie Property from Queenston (Queeston, AIF, 2010).

The Lac-McVittie Property comprises 59 unpatented mineral claims (955 ha) and lies east and adjacent to the Upper Beaver Property. Prior to the 2009 exploration program on the Property, ownership in the JV was Barrick Gold Corporation ("Barrick") 49%, Queenston 41% and Sudbury Contact Mines Limited ("Contact") 10%. Following the 2009 program, wherein Queenston was the only participant the partners' ownership changed to Queenston 70%, Barrick 30% and Contact 0%.

The first reported work on the Property was in the late 1930s and 1940s when Spectacle Larder Lake Mines and Mary Ann Gold Mines completed trenching and a limited amount of diamond drilling. In the early 1980s Queenston completed 8 shallow drill holes and in 1985-88 Lac Minerals drilled an additional 8 holes. In 1989 the Property was optioned from Lac Minerals and from 1989 to 1996 a joint venture between Royal Oak Mines, Queenston and Contact completed a variety of exploration activities that included 16 diamond drill holes. In 2005 Queenston purchased the Royal Oak interest in the joint venture. Barrick's current interest in the joint venture came as a result of its merger with Lac Minerals in 1994. There is no past production recorded on the Property and no mineral resources.

Figure 23.1 Adjacent properties



The Property is located approximately 8 km north of the Cadillac-Larder Lake Break and is underlain by volcanic and sedimentary rocks of the Upper Tisdale and Lower Blake River assemblages intruded by small syenite plugs and stocks. The Victoria Creek Deformation Zone is traced across the central portion of the Property and the south branch of the Upper Canada Break is interpreted to trend through the northern portion of the claim group. Two main areas of alteration and mineralization occur on the Property. In the north western portion exploration drilling has intersected altered mafic volcanic fragmental rocks in the vicinity of a syenite plug with a best intersection of 3.36 g Au/t over 0.61 m in drill hole 94-6. Another drill hole in this area (94-7) intersected a broad zone of alteration assaying 0.18 g Au/t over a core length of 85.4 m. In the central portion of the Property drill hole 01-03 also intersected anomalous gold values near the Victoria Creek Deformation Zone including a composite section assaying 0.96 g Au/t over 5.64 m.

No exploration work was conducted in 2008. In December of that year, Queenston proposed an exploration program on the Property targeting previous mineralization and the potential for repetition of the gold-copper system on the adjoining Upper Beaver Property. Both Barrick and Contact declined to participate in the program.

In 2009, Queenston completed a $324,000 program on the Property that consisted of 5 drill holes for a total of 3,742 m targeting weakly anomalous gold-copper Upper Beaver-type mineralization in the northwest part of the Property. The best assay results were encountered in hole LM09-3 where a quartz carbonate vein zone associated with a north shallow dipping fault zone averaged 0.31 g Au/t and 0.1% Cu over 13.9 m.

In 2010 one hole begun in 2009 was completed for a total of 112m. Two of the drill holes completed in 2009 were the subject of down-hole Pulse EM surveys in 2010. Results of this survey were inconclusive.

23.2 UPPER CANADA

The Upper Canada Property is owned 100% by Queenston and is subject to a 2% NSR to Franco-Nevada Mining Corporation, and comprises 63 claim units (955 ha) located in the central portion of Gauthier Township, southwest of the Upper Beaver Property. The Property is underlain by Timiskaming assemblage flows, tuffs, sediments with syntectonic dykes, sill and plugs of syenite and porphyry. The deposit sits within a 300-400 m thick deformation corridor framed by the north and south branches of the Upper Canada Break, a structural splay feature emerging from the Larder Lake Break. The Property hosts two gold deposits (Upper Canada and Brock) with past production of approximately 1.5 million oz. of gold.

The initial discovery of gold at Upper Canada was in 1920 and in 1928 a shaft was sunk to 40 m. In 1929, Upper Canada Mines acquired the Property, deepened the shaft to 150 m and established 4 levels. At the Brock deposit, gold was discovered in the 1930s and between 1938-41, Brock Mines sank a shaft to 192 m with four levels. No production was reported and the Property was acquired by Upper Canada Resources in 1946. The Upper Canada deposit commenced production in 1938 and produced gold continuously to 1971. The assets of Upper Canada Resources were acquired by Queenston in 1977.

Past production amounted to 1.52 million oz. of gold from 4,294,873 tonnes averaging 11.01 g Au/t, with the primary production shaft and winze to a depth of 1,930 m. With a substantial resource remaining, the mine was closed in 1971 due to a major capital infusion required for expanding the operation; including a power change over from 25 to 60 cycle. The mill continued to operate until 1972 processing material from the Upper Beaver mine and in 1984 the mill was used to process ore from the McBean mine until 1986. In 2001, Queenston dismantled the mill and ancillary buildings as part of the Closure Plan filed with the Ministry of Northern Development and Mines. Since 1990, no exploration has been undertaken on the Property.

Production was principally recorded from the H, M, Q, B, Upper and Lower L zones. The L Zone is the largest ore bearing vein system occurring along the east side of a spotted porphyry body. It is represented by bluish quartz veins in a siliceous tuff and accounts for approximately 75% of the past production and 46% of the remaining historic non-compliant mineral resources (Table 23.1).

TABLE 23.1 HISTORIC RESOURCES – UPPER CANADA (NI 43-101 NON-COMPLIANT)		
Zone	**Measured + Indicated Resources**	**Oz. of Gold**
C Zone	720,508 t @ 7.4 g Au/t	170,700
Upper L	109,216 t @ 4.3 g Au/t	15,000
Lower L	773,475 t @ 7.7 g Au/t	191,250
M&Q	296,774 t @ 4.5 g Au/t	42,750
Total	**1,899,973 t @ 6.9 g Au/t**	**419,700**

On May 4, 2011, Queenston announced the completion of an independent NI 43-101 compliant Mineral Resource estimate for the Upper Canada deposit. Indicated and Inferred mineral resources were determined for both near surface mineralization to an average depth from surface of 125m within an optimized pit shell, with additional resources possibly amenable to underground mining methods (Table 23.2).

TABLE 23.2 MINERAL RESOURCES – UPPER CANADA						
Capped Resource	**Indicated**			**Inferred**		
Cut-off (g Au/t)	**Tonnes**	**Au (g/t)**	**Au (oz.)**	**Tonnes**	**Au (g/t)**	**Au (oz.)**
Pit (0.44 g Au/t)	1,721,000	1.88	104,000	1,273,000	1.86	76,000
UG Below Pit (2.4 g Au/t)	238,000	4.25	33,000	3,622,000	4.78	557,000
Total	**1,959,000**	**2.17**	**137,000**	**4,895,000**	**4.02**	**633,000**

23.3 GAUTHIER

The following is a description of the Gauthier Property from Queenston (Queenston, AIF, 2010).

The Property, comprising 88 staked mining claims (1,400 ha), is situated in northern Gauthier Township about 3 km north of the past-producing Upper Canada gold mine. The claim group covers 5.3 km of strike extent of the west-northwest trending Victoria Creek Deformation Zone ("VCDZ"), a major structural unconformity and that represents a high-priority gold exploration target. In October 2008, Vault acquired a 100% interest in the claim group from Stornoway Diamond Corporation, in consideration of the issuance of 100,000 common shares of Vault and the grant of a 2% NSR. The Property is one of the assets Queenston acquired through the merger with Vault, completed on April 20, 2010.

The VCDZ hosts the Victoria Creek gold deposit, 700m west of the Company's Gauthier Property, on which Sudbury Contact Mines Limited expended in excess of $20 million in development during the late 1990s. The VCDZ is thought to have exerted some structural control over mineralization at the Upper Beaver deposit, located 500 m east of the Gauthier Property. A preliminary compilation by Vault of previous exploration work on the Property reveals untested geophysical anomalies along interpreted splays off of the VCDZ. The anomalies comprise highly magnetic areas with co-incident IP and VLF conductors surrounding circular magnetic lows (at least one of which contains an identified felsic intrusive outcrop). These geophysical signatures are highly analogous to that of the Upper Beaver deposit, where mineralization is associated with magnetite bordering a circular felsic intrusive having a magnetic-low expression.

No material work projects were completed on the Property in the period from 2008 to 2009.

In 2010 a deep IP survey was completed over 6.2 kilometres of line by Insight Geophysics and along with a 3.6 km survey completed in February 2010 these surveys have identified coincident IP east-northeast trending anomalies to that of the Titan 24 survey. Of interest, these anomalies are situated to the immediate west of the projected strike of the Upper Beaver North Basalt Zone. Following the receipt of the Insight Interpretive Report and field investigation, a proposed diamond drill program will be designed to test these IP targets.

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

The majority of the mineral resources in the Upper Beaver deposit occur in a series of breccia zones that dip steeply north (75°) below the old mine workings. These zones contain chalcopyrite, magnetite, pyrite and visible gold within a mineralized corridor that extends over a horizontal length of approximately 500 metres and a dip length of approximately 1,300 m. A conceptualized mining plan has been developed to extract the Deposit using mechanized trackless mining equipment. Similarly, a processing plant and related facilities have been planned in concept that would accept the mine product and reduce it to saleable gold bullion and copper concentrate. This PEA indicates that a profitable mining and processing operation could be constructed at the Upper Beaver site.

Note: This PEA is preliminary in nature and its Mineral 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.

Exploration drilling can likely extend the known pay-shoots at depth and infill drilling may convert Inferred Resources to Indicated Resources.

P&E Mining Consultants Inc. offers the following interpretation and conclusions:

- P&E concludes that the Deposit has economic potential as an underground mining and milling operation producing copper concentrates and gold doré;
- This Report is considered by P&E Mining Consultants Inc. to meet the requirements of a Technical Report as defined in Canadian NI 43-101 regulations. The economic analysis contained in this Report is based on indicated and inferred resources. The mineral resources in this PEA 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 CIM Council, December 11, 2005;
- There is no guarantee that Queenston will be successful in obtaining any or all of the requisite consents, permits or approvals, regulatory or otherwise for the Upper Beaver Property development or that the Property will be placed into production.
- The envisaged Longhole Longitudinal Retreat mining method is estimated to experience mining dilution in the order of 20% at zero grade. Mine recovery (extraction) is estimated to be 95%;
- The project was evaluated on an after-tax cash flow basis and it is estimated that it could generate a net cash flow of $413.9 million, after tax. This results in an after tax Internal Rate of Return (IRR) of 22.1% and an after-tax Net Present Value (NPV) of $233.4 million when using a 5% discount rate. In the base case scenario, the project has a payback period of 2.5 years from start of commercial production. The average life-of-mine cash cost is US$416/oz. gold, net of copper credits, at an average operating cost of $73.06 per ore tonne ore processed.
- The after-tax base case NPV is most sensitive to realized copper and gold metal prices and currency exchange rates, followed by capital and operating costs.

26.0 RECOMMENDATIONS

P&E recommends that the Company 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 Queenston take the following actions to develop the project to a Pre-Feasibility Study level:

- Complete detailed engineering and develop an exploration shaft which will provide access for bulk sampling and confirm the mineability/continuity of the deposit. This will include shaft sinking contractor selection and hoists procurement;
- Update current mineral resource by incorporating all new drilling that was not included in the 2011 WGM mineral resource;
- Complete the permitting procedure to procure an Advanced Exploration Permit for shaft sinking.
- Continue with baseline studies to support the environmental permitting process;
- Continue to engage the community and aboriginal groups in the project development. It is expected that Queenston will continue to work cooperatively with aboriginal communities to communicate the project's scope, impacts and benefits during the Advanced Exploration and Production stages;
- Carry out additional metallurgical testwork to improve metallurgical recoveries and process optimization. It is also recommended that tests on direct cyanidation of the mineralization be carried out.

Queenston should also continue with infill and step-out drilling for further exploration and mineral resource definition, as well as permitting and community matters. A proposed budget for this work in 2012 is provided in Table 26.1.

TABLE 26.1
PROPOSED BUDGET

Description	Cost
Drilling	$11,000,000
Environmental Work	$250,000
First Nation Consultation	$500,000
Metallurgical Testwork	$100,000
Resource Estimation	$100,000
Hydrogeology Study	$100,000
Archaeological	$75,000
Advance Exploration Closure Report	$250,000
Geotechnical and Condemnation Drilling	$1,000,000
Housing and Accommodation	$500,000
Site Preparation	$500,000
Total	**$14,375,000**

P&E also recommends that the Mineral Resource estimates be updated to incorporate any additional information that has become available since the WGM (2011) Report, including any material results from exploration and diamond drilling work that has been underway during and prior to this period.

27.0 REFERENCES

Cementation Canada Inc. Upper Beaver Project Conceptual Engineering for Advanced Exploration Program prepared for Queenston Mining Inc., dated June 6, 2011.

SGS Lakefield Research Limited, Lakefield, Ontario. An Investigation into the Recovery of Gold and Copper from the Upper Beaver Deposit (Phase I) – Project 11981 001 Final Report, prepared for Queenston Mining Inc., dated April 22, 2009.

SGS Canada Inc, Lakefield , Ontario. An Investigation into Scoping Level Testwork on Samples from Queenston's Kirkland Lake Properties (Project 12599-001 Final Report), prepared for Queenston Mining Inc., dated July 20, 2011.

Story Environmental Inc., Project Definition for Advanced Exploration Upper Creek Mine Site Improvement District of Gauthier, Ontario.(Volume 1 of 2, Revision 1). Report prepared for Queenston Mining Inc., dated November 2, 2011.

Story Environmental Inc. Project Definition for Advanced Exploration Upper Creek Mine Site Improvement District of Gauthier, Ontario (Volume 2 of 2, Revision 1), prepared for Queenston Mining Inc., dated November 2, 2011.

SENES Consultants Limited., A Practitioner's Guide to planning for and permitting a mineral development project in Ontario prepared for Ministry of Northern Development and Mines (MNDM), dated March 2008. Available at www.mndm.gov.on.ca.March 2008.

Watts, Griffis and McOuat Limited, Technical Report and Mineral Resource Estimate for the Upper Beaver Property, Ontario. Report prepared for Queenston Mining Inc., dated June 2011

28.0 CERTIFICATES
CERTIFICATE OF QUALIFIED PERSON

EUGENE J. PURITCH, P. ENG.

I, Eugene J. Puritch, P. Eng., residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P & E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment of the Upper Beaver Gold-Copper Deposit, Kirkland Lake, Ontario, Canada" (the "Technical Report") with an effective date of February 16, 2012.
3. 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

4. I have visited the Property that is the subject of this report on October 5, 2011.
5. I am responsible for contributing to portions of Sections 15 and 16 of the Technical Report.
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 project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This 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: February 16, 2012
Signed Date: March 30, 2012

{SIGNED AND SEALED}
[Eugene Puritch]

Eugene J. Puritch, P. Eng

KIRK RODGERS, P.ENG.

CERTIFICATE OF AUTHOR

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 of the Upper Beaver Gold-Copper Deposit, Kirkland Lake, Ontario, Canada" (the "Technical Report") with an effective date of February 16, 2012.
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 15 and 19 co-authoring the Sections 1, 25 and 26 of this Technical Report.
5. I have not visited the Property that is the subject of this 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: February 16, 2012
Signed Date: March 30, 2012

{SIGNED AND SEALED}
{Kirk Rodgers}

____
Kirk Rodgers, P. Eng.

JAMES L. PEARSON, P.ENG.

CERTIFICATE OF AUTHOR

I, James L. Pearson, P.Eng., residing at 5 Clubhouse Court, Bolton, Ontario, Canada, L7E 0B3, do hereby certify that::

1. I am an independent Mining Engineering Consultant, contracted by P& E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment of the Upper Beaver Gold-Copper Deposit, Kirkland Lake, Ontario, Canada" (the "Technical Report") with an effective date of February 16, 2012.
3. I am a graduate of Queen's University, Kingston, Ontario, Canada, in 1973 with a Bachelor of Science degree in Mining Engineering. I am registered as a Professional Engineer in the Province of Ontario (Reg. No. 36043016). I have worked as a mining engineer for a total of 37 years since my graduation.

   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:

   - Review and report as a consultant on numerous exploration and mining projects around the world for due diligence and regulatory requirements;
   - Project Manager and Superintendent of Engineering and Projects at several underground operations in South America;
   - Senior Mining Engineer with a large Canadian mining company responsible for development of engineering concepts, mine design and maintenance;
   - Mining analyst at several Canadian brokerage firms

4. I have visited the Property that is the subject of this report on October 5, 2011.
5. I am responsible for authoring Sections 22 and 24 as well as co-authoring Sections 1, 3, 14.11, 18, 21 and 25 through 27 of the Technical Report;
6. I am 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 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 with that Instrument and Form.
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: February 16, 2012
Signing Date: March 30, 2012

{SIGNED AND SEALED}
[James L. Pearson]

James L. Pearson, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

DAVID BURGA, P. GEO.

I, David Burga, P. Geo., residing at 3884 Freeman Terrace, Mississauga, Ontario, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment of the Upper Beaver Gold-Copper Deposit, Kirkland Lake, Ontario, Canada" (the "Technical Report") with an effective date of February 16, 2012.
3. I am a graduate of the University of Toronto with a Bachelor of Science degree in Geological Sciences (1997). I have worked as a geologist for a total of 12 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No 1836).

   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:

   - Exploration Geologist, Cameco Gold 1997-1998
   - Field Geophysicist, Quantec Geoscience 1998-1999
   - Geological Consultant, Andeburg Consulting Ltd. 1999-2003
   - Geologist, Aeon Egmond Ltd. 2003-2005
   - Project Manager, Jacques Whitford 2005-2008
   - Exploration Manager – Chile, Red Metal Resources 2008-2009
   - Consulting Geologist 2009-Present

4. I have not visited the Property that is the subject of this report.
5. I am responsible for authoring Sections 2, 4 through 10 and 23 as well as co-authoring Sections 1, 3 and of the Technical Report.
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 this 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: February 16, 2012
Signed Date: March 30, 2012

{SIGNED AND SEALED}
[David Burga]

____
David Burga, P. Geo.

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 of the Upper Beaver Gold-Copper Deposit, Kirkland Lake, Ontario, Canada" (the "Technical Report") with an effective date of February 16, 2012.
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 of the Technical Report.
6. I am an independent of the issuer applying all of the tests in Section 1.5 of NI 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 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: February 16, 2012
Signed Date: March 30, 2012

{SIGNED AND SEALED}
[David Orava]

____

David Orava, M. Eng., 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 of the Upper Beaver Gold-Copper Deposit, Kirkland Lake, Ontario, Canada" (the "Technical Report") with an effective date of February 16, 2012.
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 42 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 report.
5. I am responsible for authoring of Section 13 and 17 of the Technical Report
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: February 16, 2012
Signing Date: March 30, 2012

{SIGNED AND SEALED}
[Alfred Hayden]

____
Alfred S. Hayden, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

KURT BREEDE, P. ENG

CERTIFICATE

To Accompany the Report Entitled
"Technical Report and Preliminary Economic Assessment
of the Upper Beaver Gold-Copper Deposit
Kirkland Lake, Ontario, Canada" March 30, 2012

I, Kurt Breede, do hereby certify that:

1. I reside at 76 Woodrow Avenue, Toronto, Ontario, M4C 1G7.
2. I am a Senior Resource Engineer and Vice-President, Marketing with Watts, Griffis and McOuat Limited, a firm of consulting geologists and engineers, which has been authorized to practice professional engineering by Professional Engineers Ontario since 1969, and professional geoscience by the Association of Professional Geoscientists of Ontario.
3. This certificate accompany the report titled "*Technical Report and Preliminary Economic Assessment of the Upper Beaver Gold-Copper Deposit Kirkland Lake, Ontario, Canada for Queenston Mining Inc."* dated March 30, 2012.
4. I am a graduate from the University of Toronto, Toronto, Ontario with a B.A.Sc. Degree in Geological and Mineral Engineering (1996), and I have practised my profession continuously since that time.
5. I am a Professional Engineer licensed by Professional Engineers Ontario (Registration Number 90501859) and the Association of Professional Engineers and Geoscientists of Saskatchewan (Registration Number 17014).
6. I have read the definition of "qualified person" set out in the 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.
7. I am the co-author of the June 15, 2011 Technical Report titled "*Technical Report and Mineral Resource Estimate Update for the Upper Beaver Property, Ontario for Queenston Mining Inc.*". I visited the Upper Beaver Property on March 30, 2011.
8. I am responsible for Sections 12, and 14.1 to 14.10, of the report.
9. I am independent of the issuer as described in Section 1.5 of NI 43-101.
10. I am an independent Qualified Person for the purposes of NI 43-101 and have extensive experience with gold deposits, a variety of other deposit types, Mineral Resource estimation techniques and the preparation of technical reports.
11. I have read NI 43-101, Form 43-101F1 and the technical report and have prepared the technical report in compliance with NI 43-101, Form 43-101F1 and generally accepted Canadian mining industry practice.
12. As of the date of the technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information as at June 15, 2011, that is required to be disclosed to make the technical report not misleading.

Effective Date: February 16, 2012
Signing Date: March 30, 2012

{SIGNED AND SEALED}
[Kurt Breede]

____
Kurt Breede, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

RICHARD W. RISTO, M.Sc., P.GEO.

CERTIFICATE

To Accompany the Report Entitled
"Technical Report and Preliminary Economic Assessment
of the Upper Beaver Gold-Copper Deposit
Kirkland Lake, Ontario, Canada" March 30, 2012

I, Richard W. Risto, do hereby certify that:

1. I reside at 22 Northridge Ave, Toronto, Ontario, Canada, M4J 4P2.
2. I am a Senior Associate Geologist with Watts, Griffis and McOuat Limited, a firm of consulting engineers and geologists, which has been authorized to practice professional engineering by Professional Engineers Ontario since 1969, and professional geoscience by the Association of Professional Geoscientists of Ontario.
3. This certificate accompany the report titled "*Technical Report and Preliminary Economic Assessment of the Upper Beaver Gold-Copper Deposit Kirkland Lake, Ontario, Canada for Queenston Mining Inc*." dated March 30, 2012.
4. I am a graduate from the Brock University, St. Catherines, Ontario with an Honours B.Sc. Degree in Geology (1977), Queens University, Kingston, Ontario with a M.Sc. Degree in Mineral Exploration (1983), and I have practised my profession for over 26 years.
5. I am a member of the Association of Professional Geoscientists of Ontario (Membership Number 276).
6. I have read the definition of “qualified person” set out in the 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.
7. I am the co-author of the June 15, 2011 Technical Report titled "*Technical Report and Mineral Resource Estimate Update for the Upper Beaver Property, Ontario for Queenston Mining Inc*.". I did not visit the Upper Beaver Property.
8. I am responsible for Section 11 of the report.
9. I am independent of the issuer as described in Section 1.5 of NI 43-101.I am an independent Qualified Person for the purposes of NI 43-101 and have extensive experience with gold deposits, a variety of other deposit types, and the preparation of technical reports.
10. I have read NI 43-101, Form 43-101F1 and the technical report and have prepared the technical report in compliance with NI 43-101, Form 43-101F1 and generally accepted Canadian mining industry practice.
11. As of the date of the technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information as at June 15, 2011, that is required to be disclosed to make the technical report not misleading.

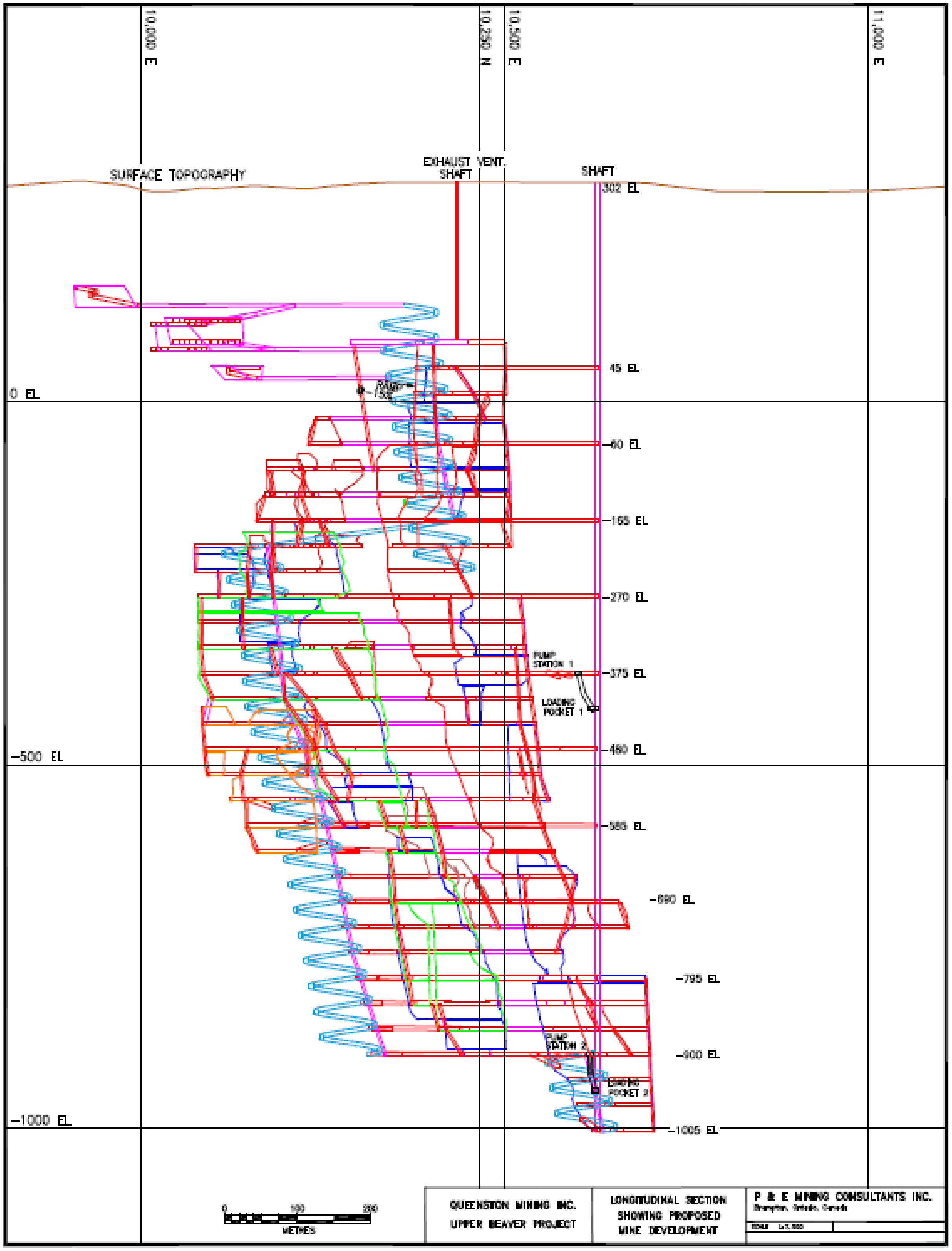
Effective Date: February 16, 2012
Signing Date: March 30, 2012

{SIGNED AND SEALED}
[Richard W. Risto]

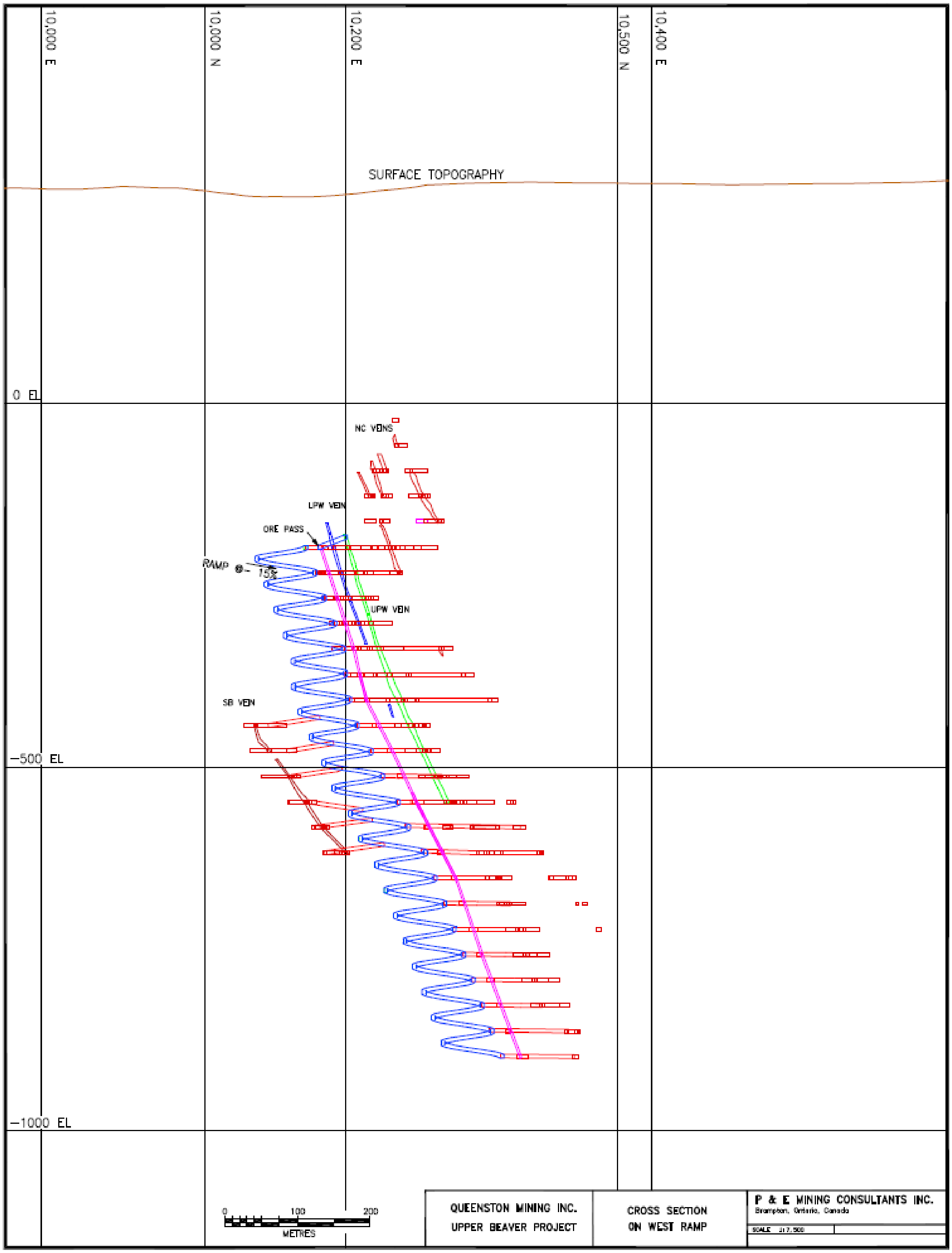
____
Richard W. Risto, B.Sc., M.Sc., P.Geo.

APPENDIX I. MINE PLAN DRAWINGS

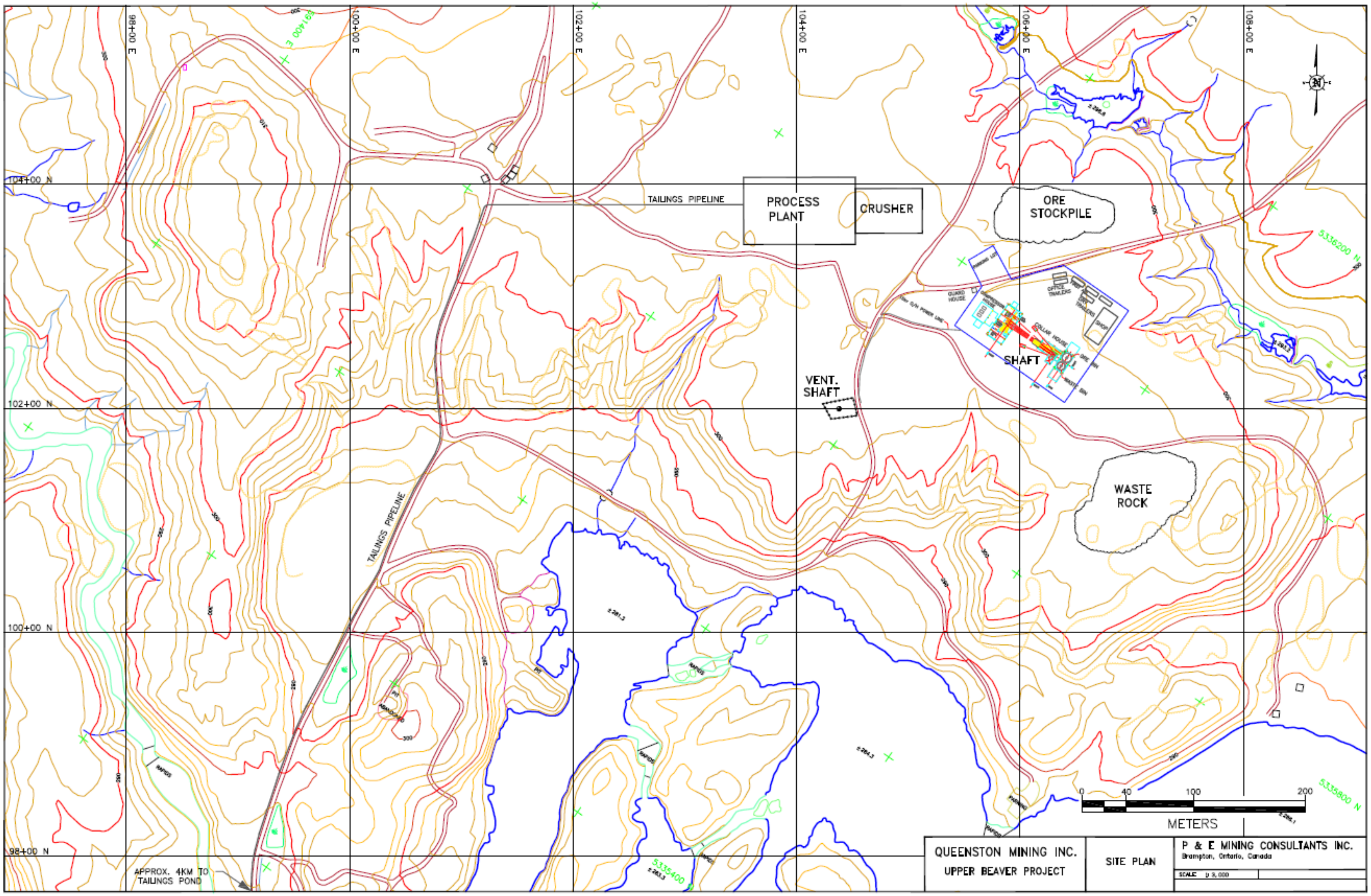
A longitudinal section of the proposed mine layout.



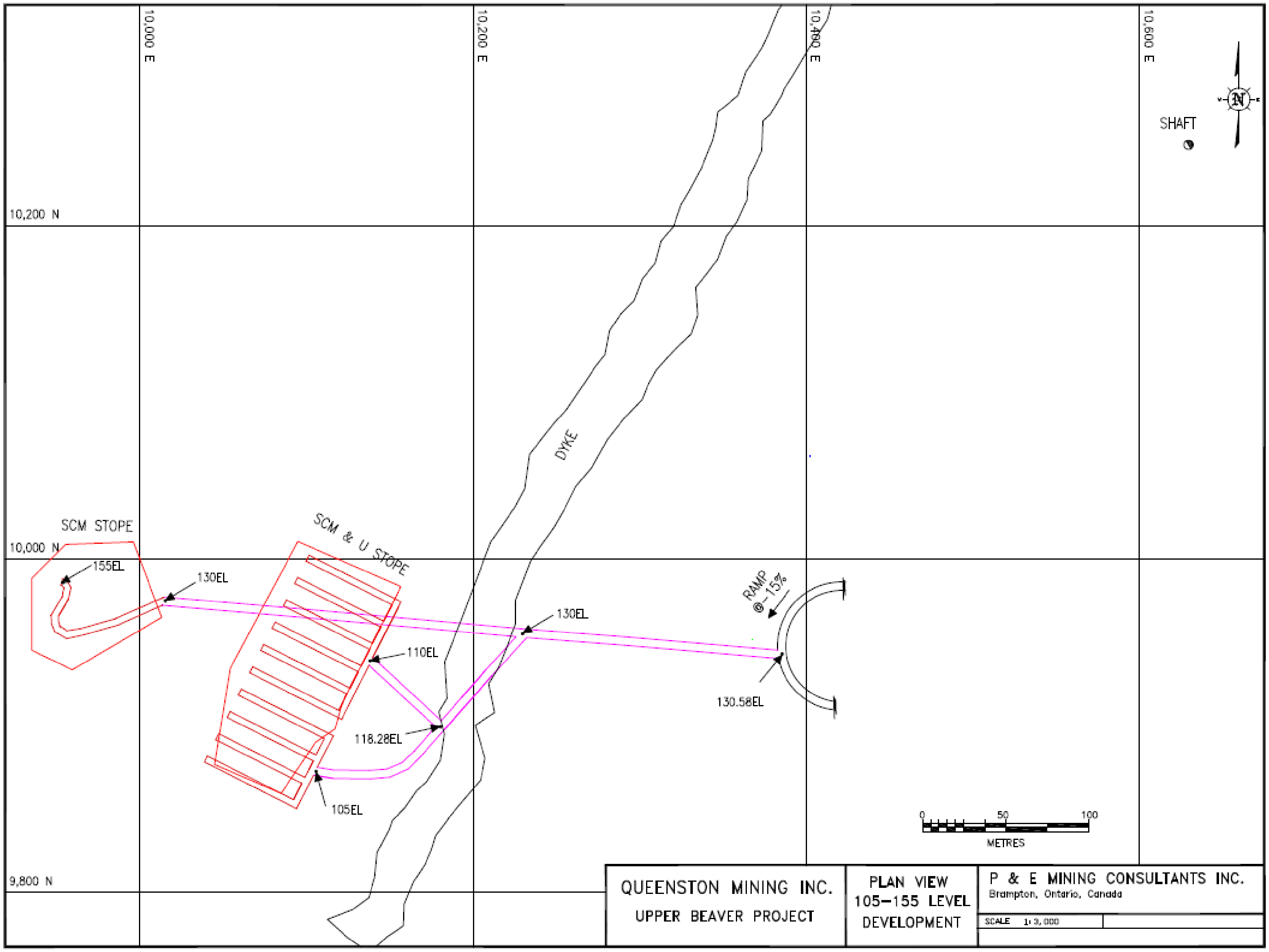
Cross Section of West Ramp

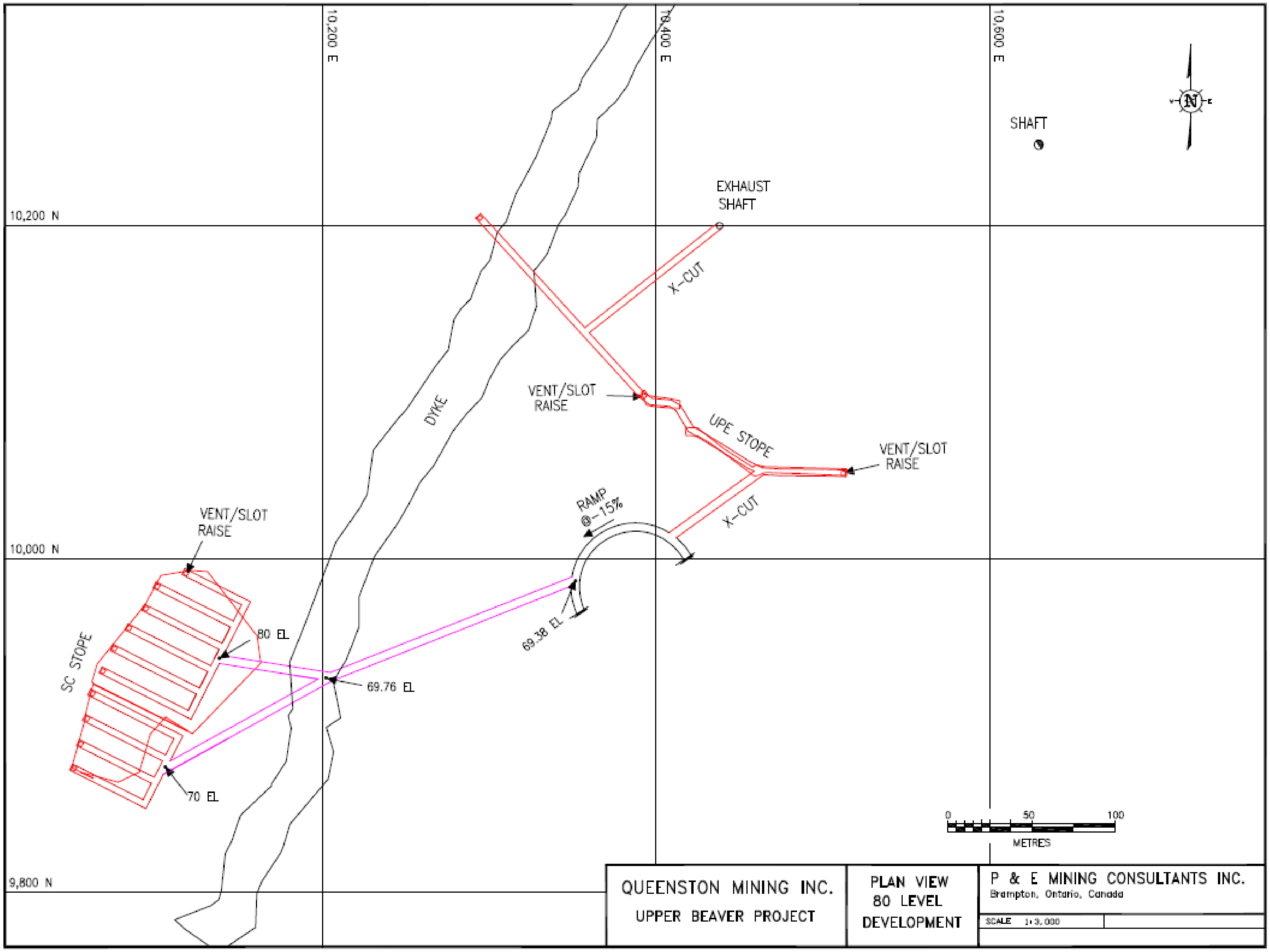


Upper Beaver Project Site Plan



Typical plans of proposed mine development are presented on the following pages.

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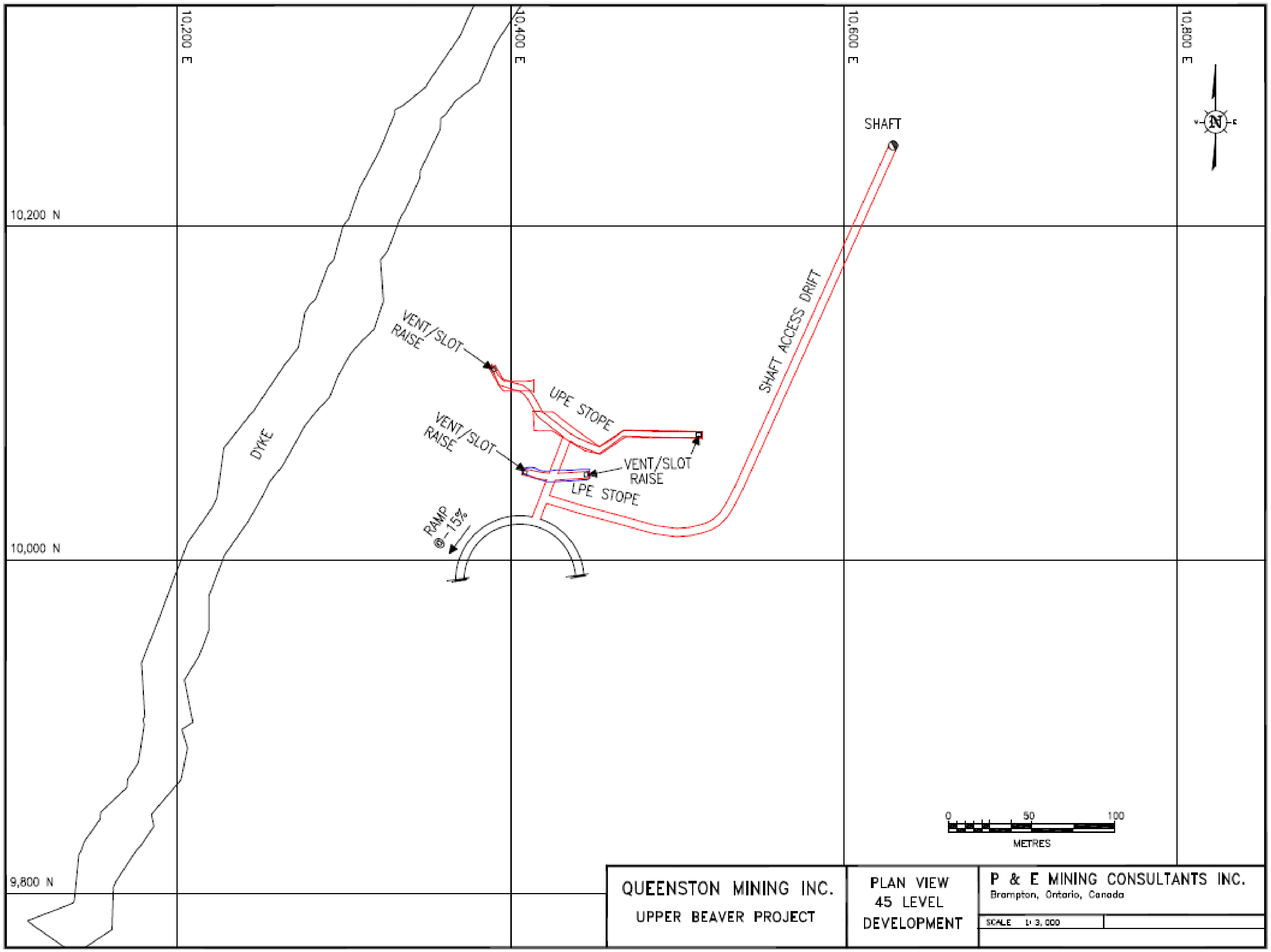
EXHAUST SHAFT
10,200 N
X-CUT
DYKE
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
RAMP @-15%
X-CUT
VENT/SLOT RAISE
10,000 N
80 EL
SC STOPE
69.38 EL
69.76 EL
70 EL
0
50
100
METRES
9,800 N
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
80 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1:3,000



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10,800 E
N
SHAFT
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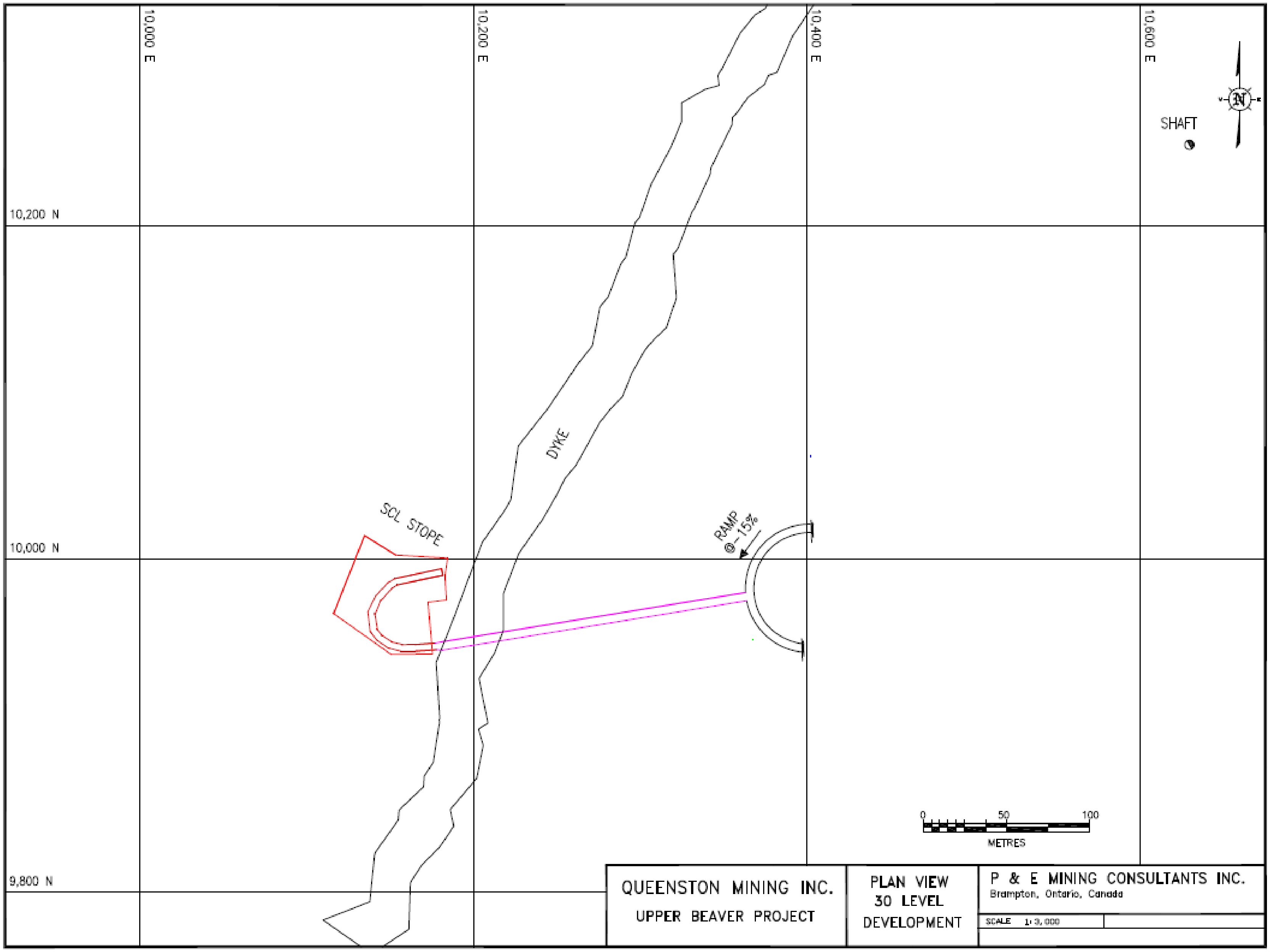
10,000 N
9,800 N
DYKE
VENT/SLOT RAISE
UPE STOPE
SHAFT ACCESS DRIFT
VENT/SLOT RAISE
VENT/SLOT RAISE
LPE STOPE
RAMP @-15%
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
45 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3, 000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10,600 E
SHAFT
N
10,200 N
DYKE
SCL STOPE
RAMP
@-15%
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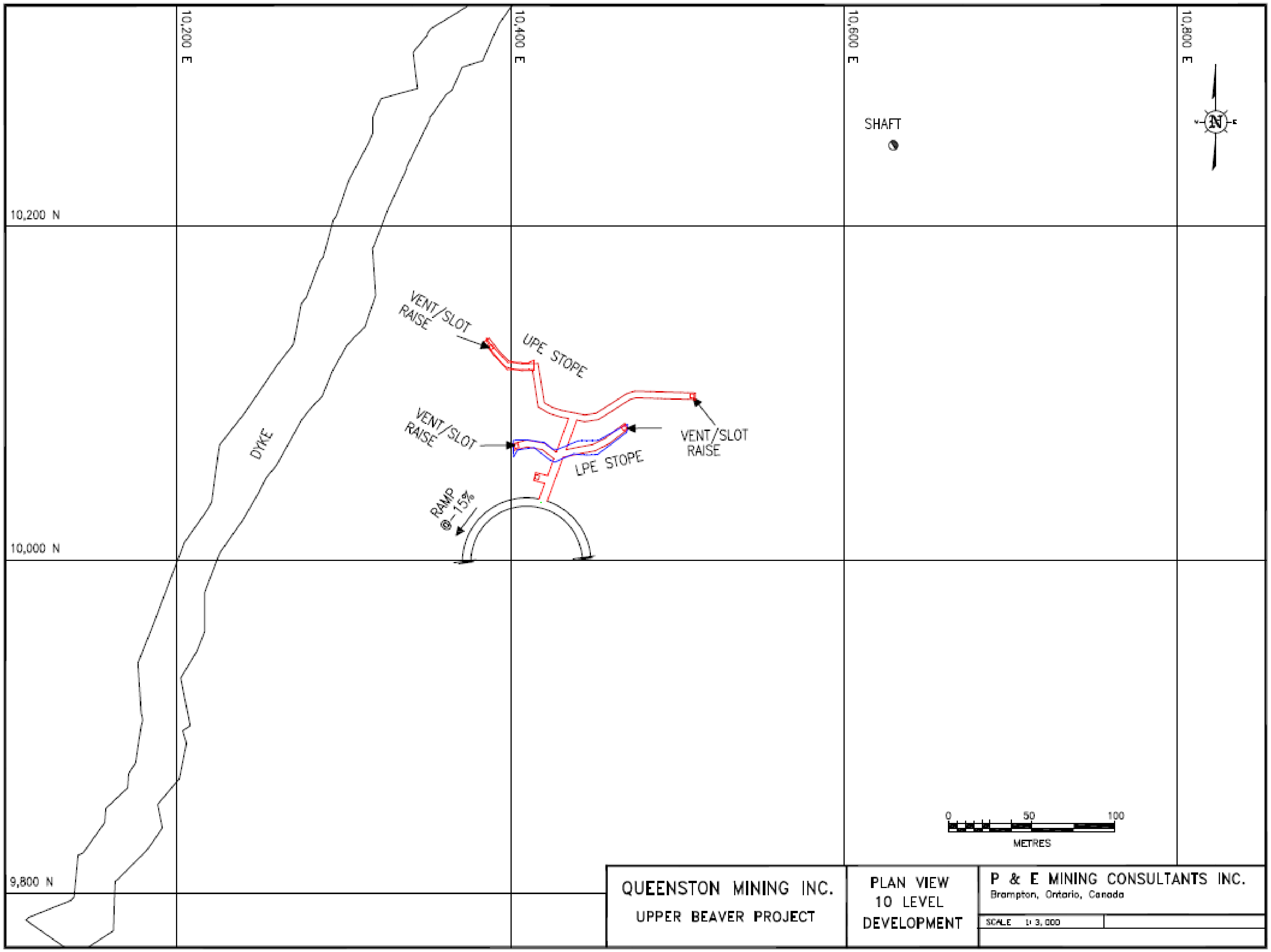
0
50
100
METRES
9,800 N
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
30 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1:3,000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10,600 E
10,800 E
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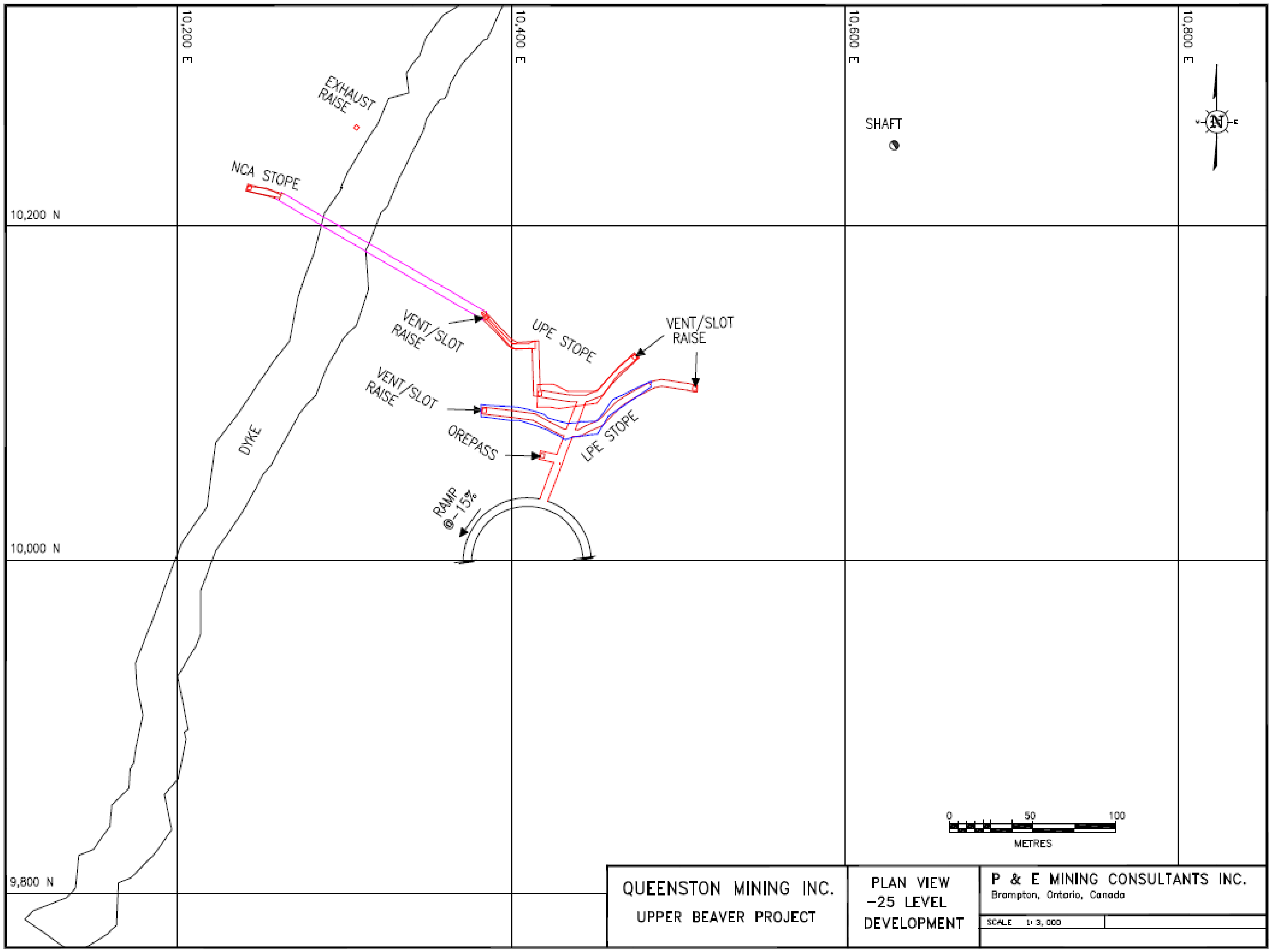
10,000 N
9,800 N
SHAFT
DYKE
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
VENT/SLOT RAISE
LPE STOPE
RAMP @-15%
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
10 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000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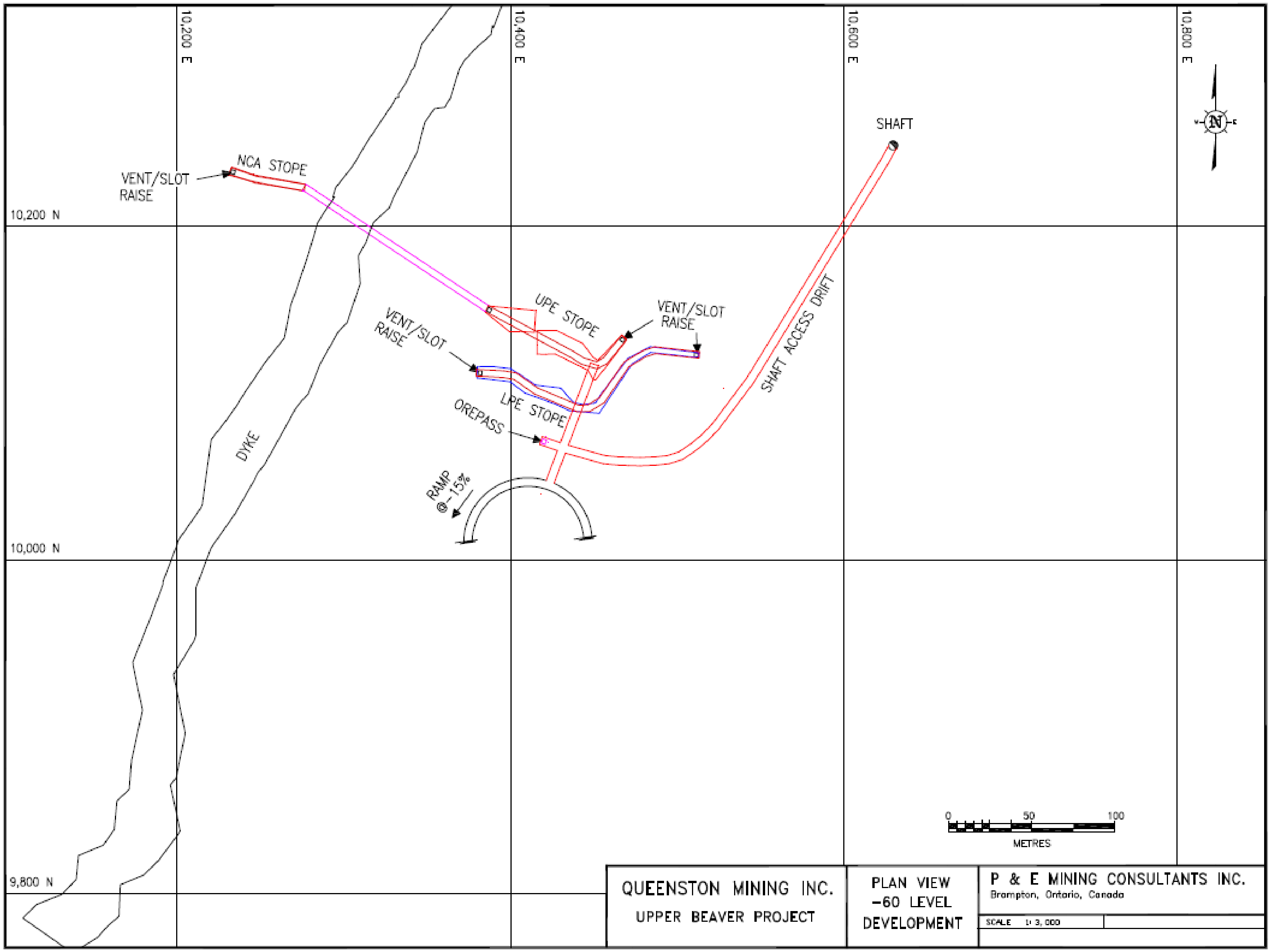
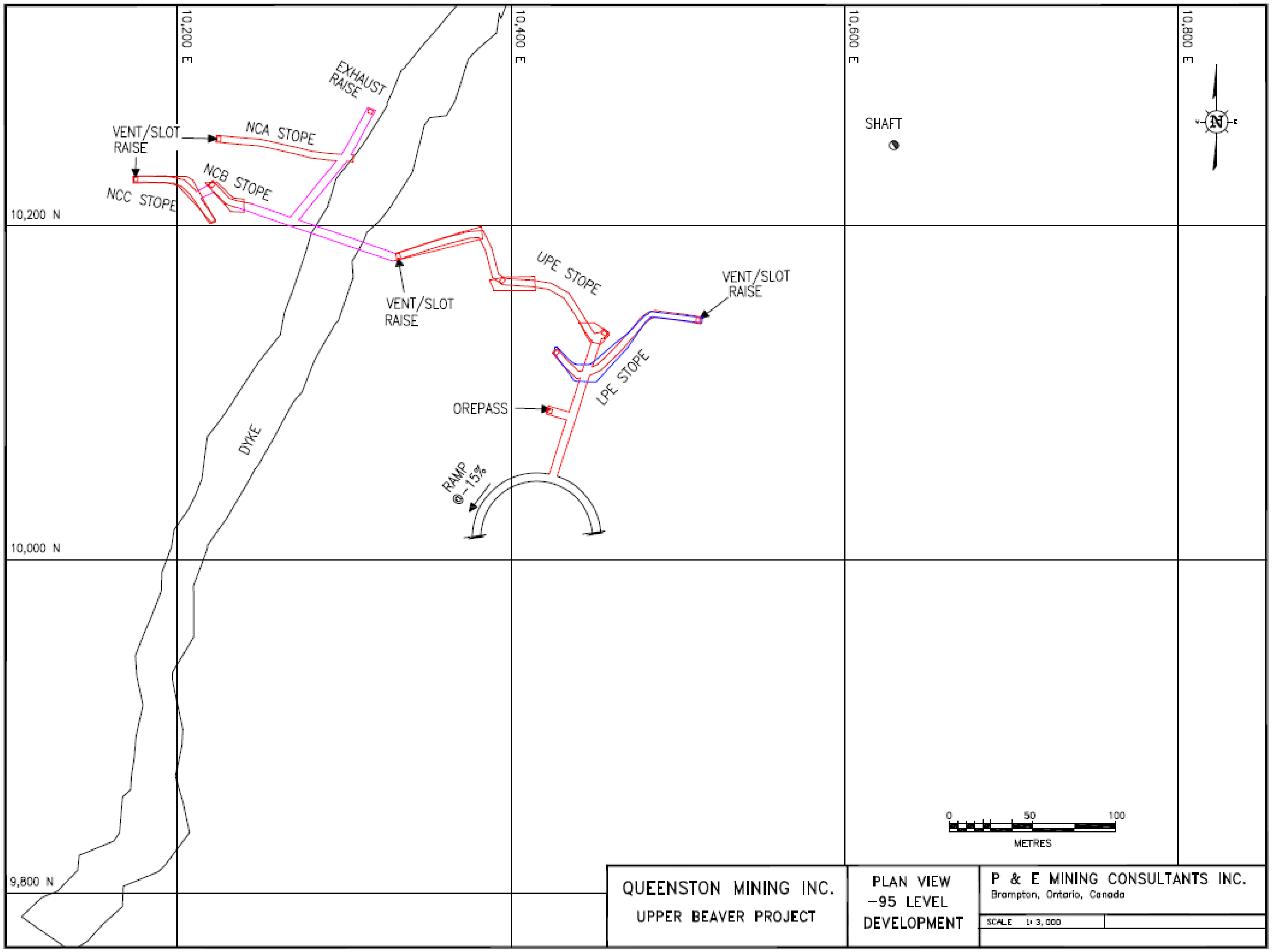
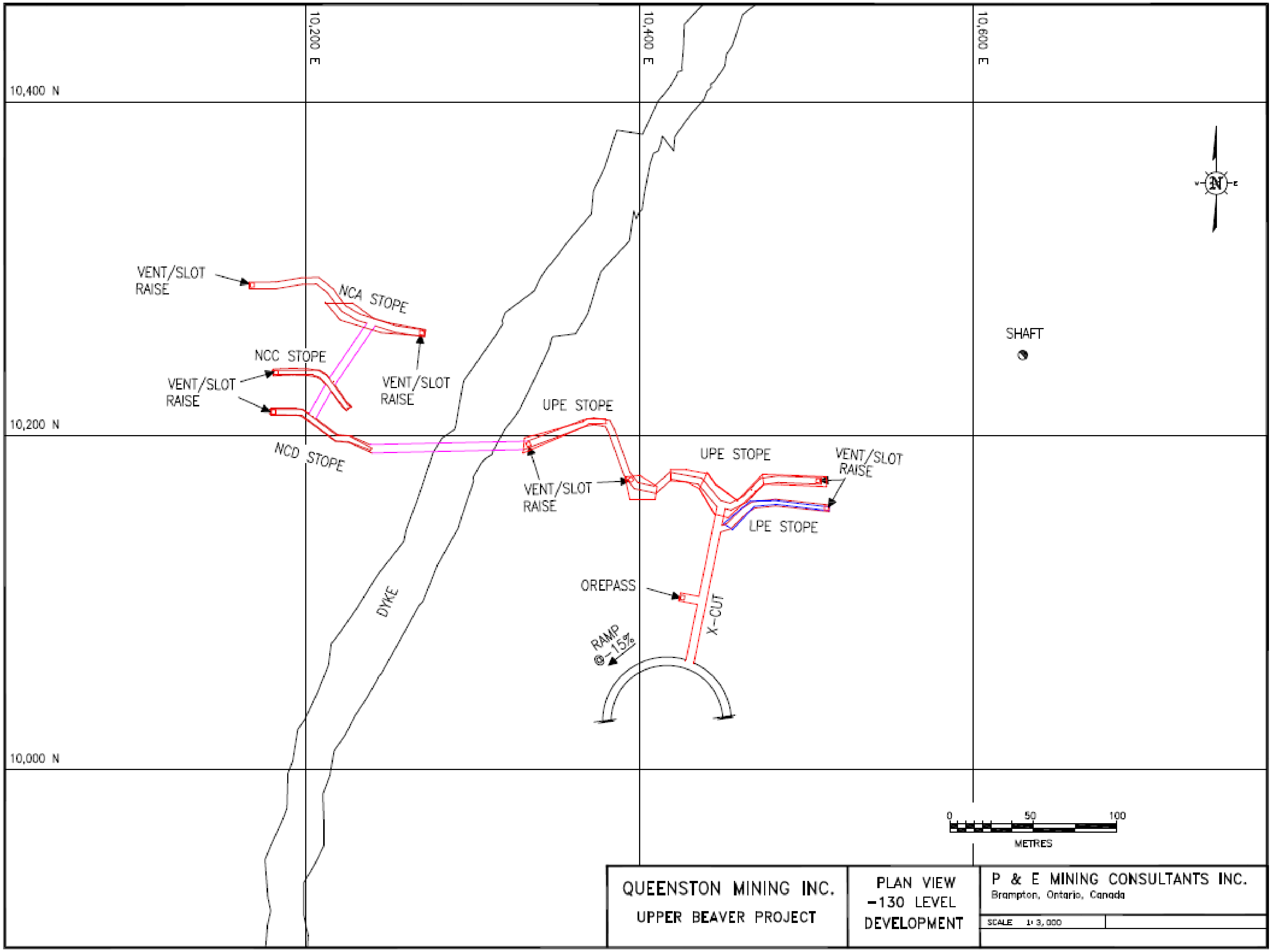
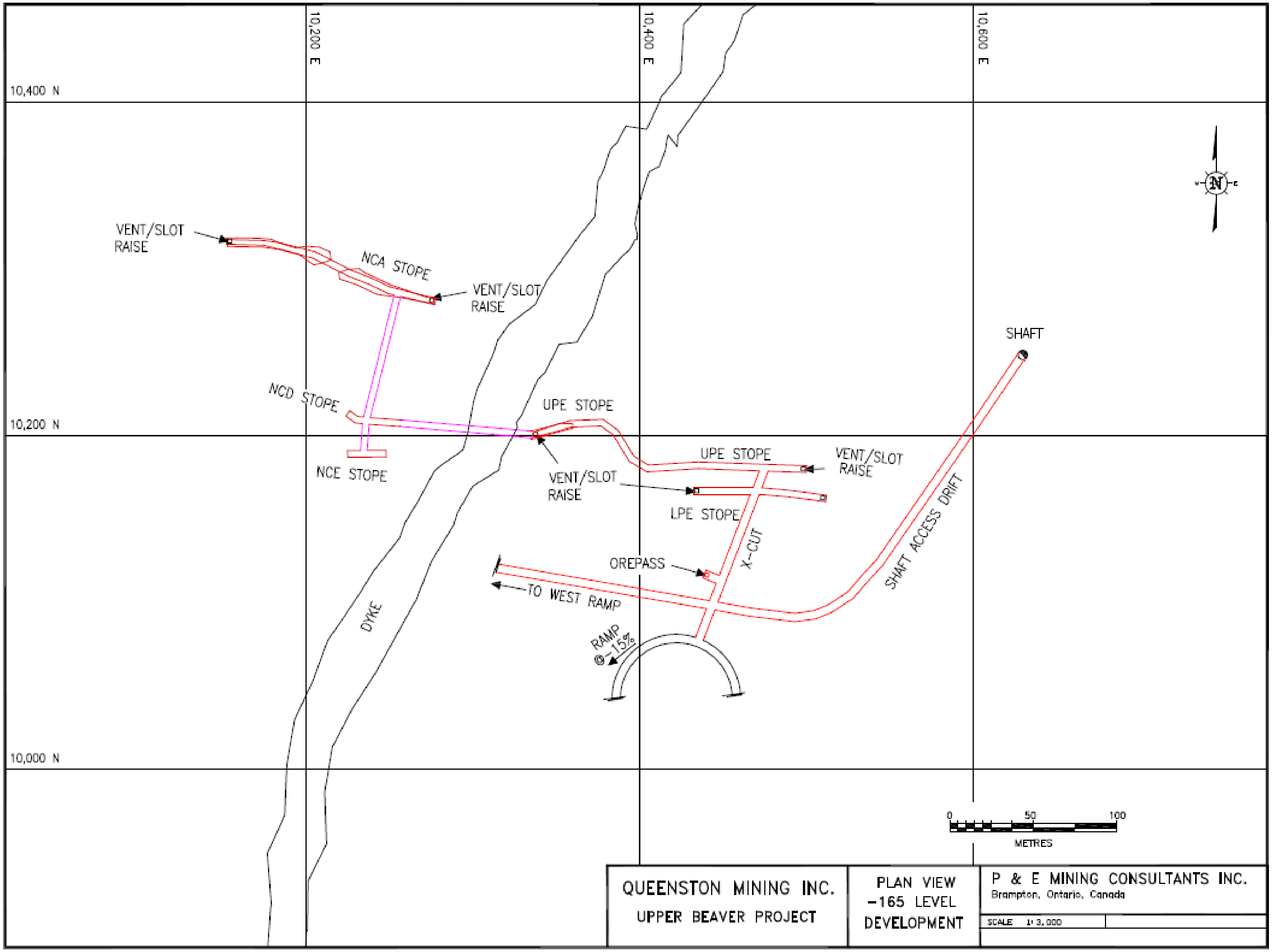
10,400 E
10,600 E
10,800 E
10,200 N
10,000 N
9,800 N
EXHAUST RAISE
NCA STOPE
SHAFT
N
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
VENT/SLOT RAISE
OREPASS
LPE STOPE
DYKE
RAMP @-15%
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-25 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

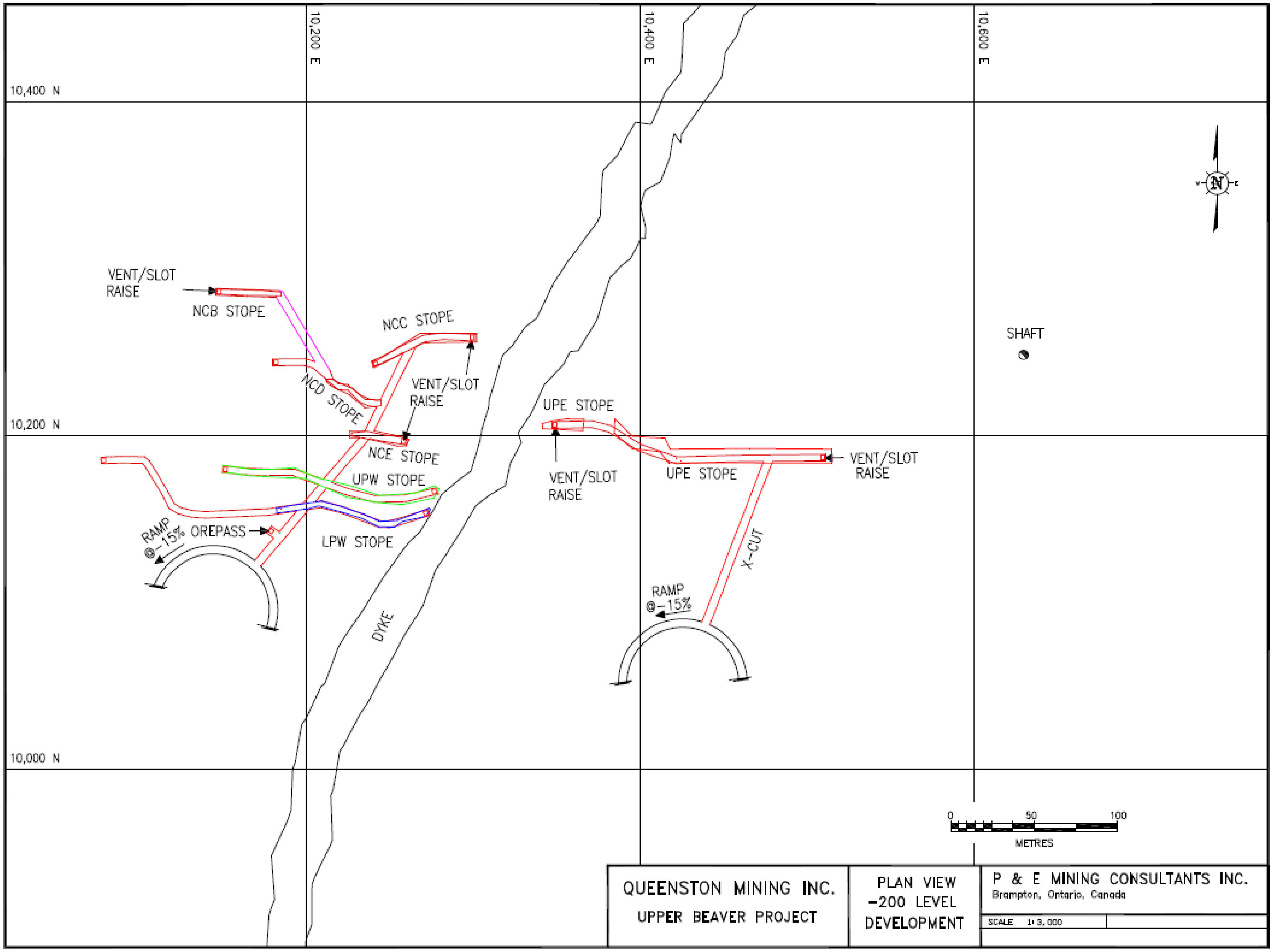

10,200 E
10,400 E
10,600 E
10,800 E
10,200 N
10,000 N
9,800 N
SHAFT
NCA STOPE
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
VENT/SLOT RAISE
LPE STOPE
OREPASS
SHAFT ACCESS DRIFT
DYKE
RAMP @-15%
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-60 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

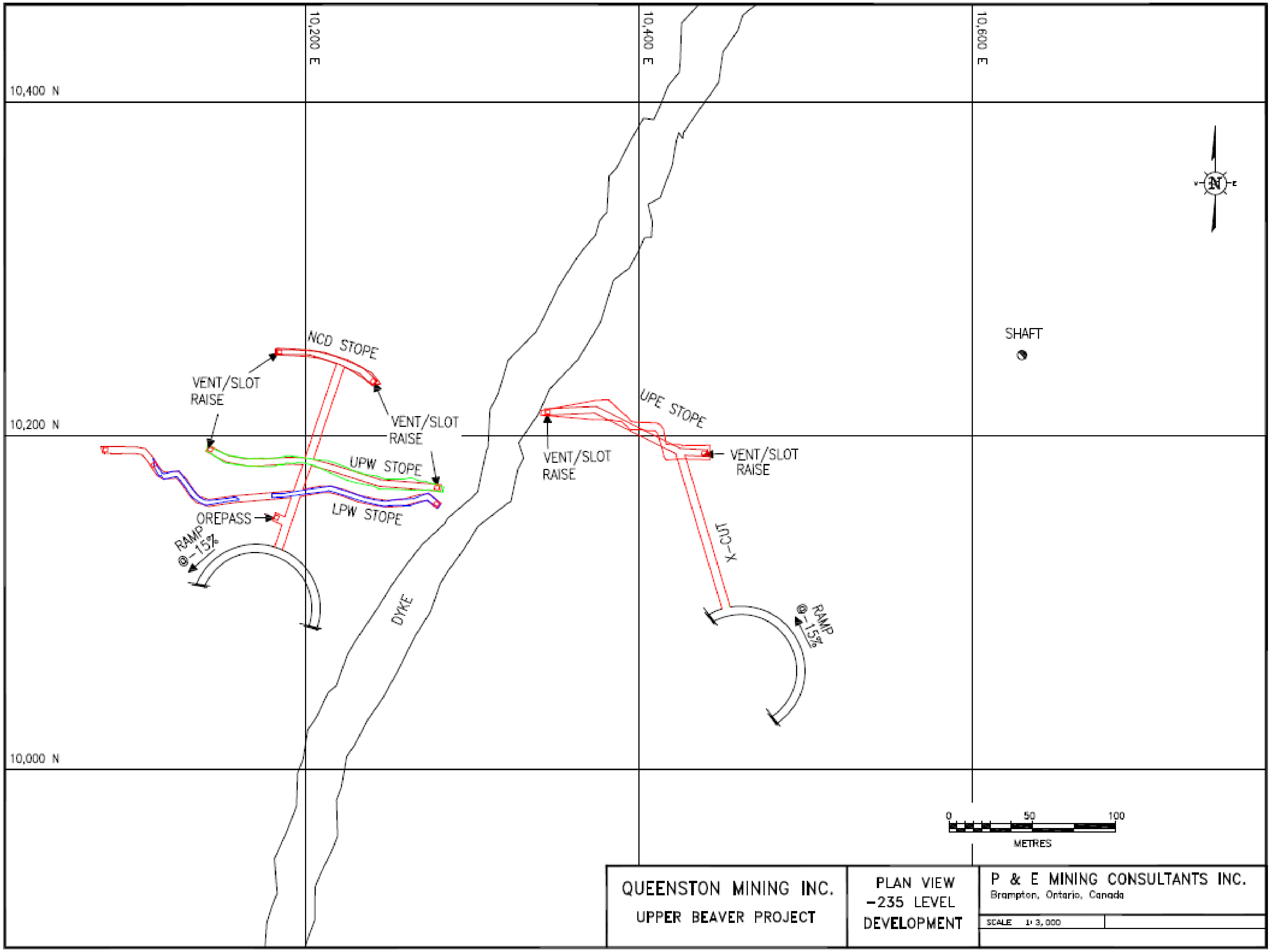

10,200 E
10,400 E
10,600 E
10,800 E
10,200 N
10,000 N
9,800 N
EXHAUST RAISE
VENT/SLOT RAISE
NCA STOPE
NCB STOPE
NCC STOPE
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
LPE STOPE
OREPASS
RAMP @-15%
DYKE
SHAFT
N
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-95 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

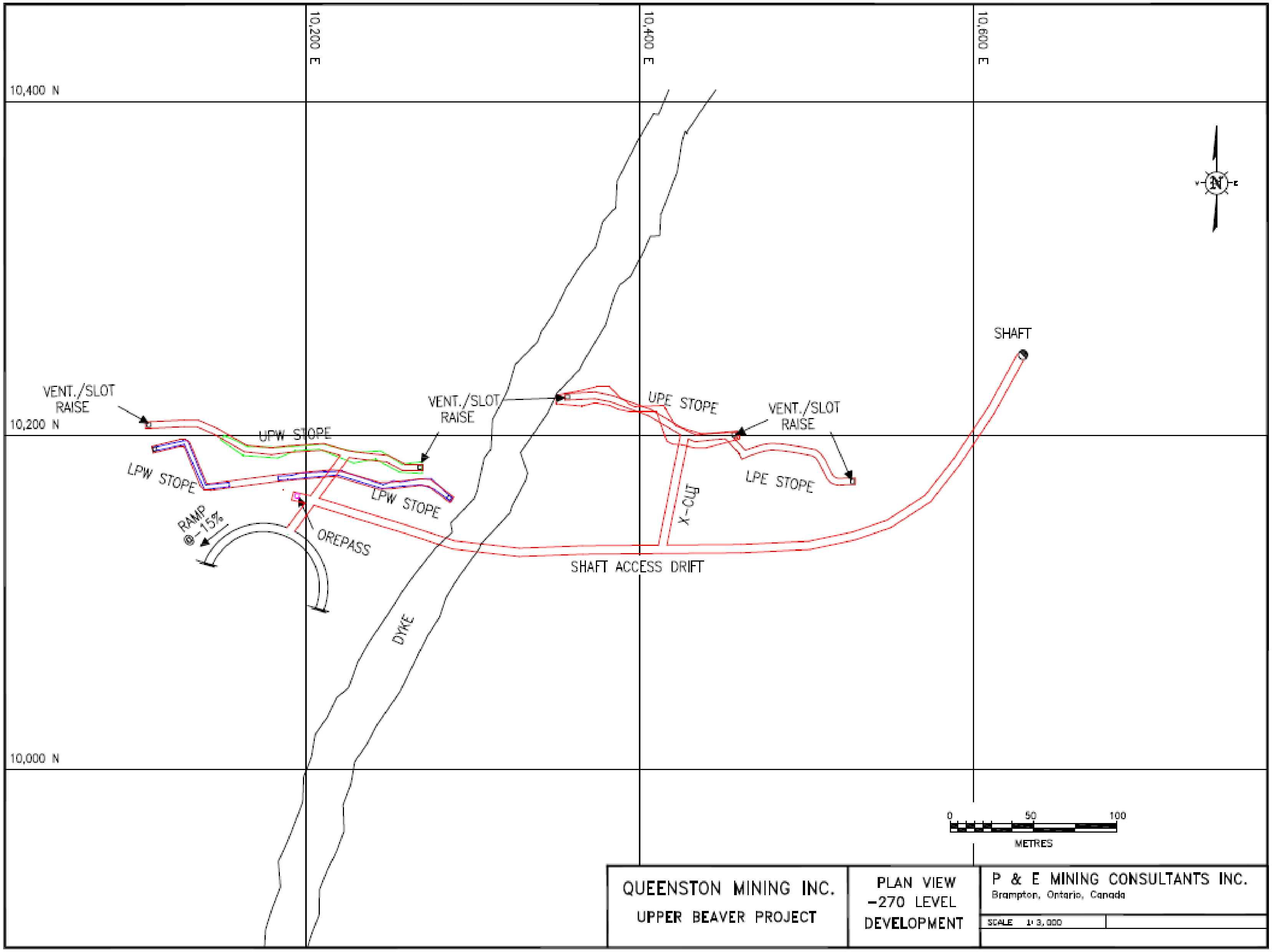

10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
VENT/SLOT RAISE
NCA STOPE
NCC STOPE
VENT/SLOT RAISE
VENT/SLOT RAISE
NCD STOPE
UPE STOPE
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
LPE STOPE
OREPASS
X-CUT
RAMP @-15%
DYKE
SHAFT
N
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-130 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

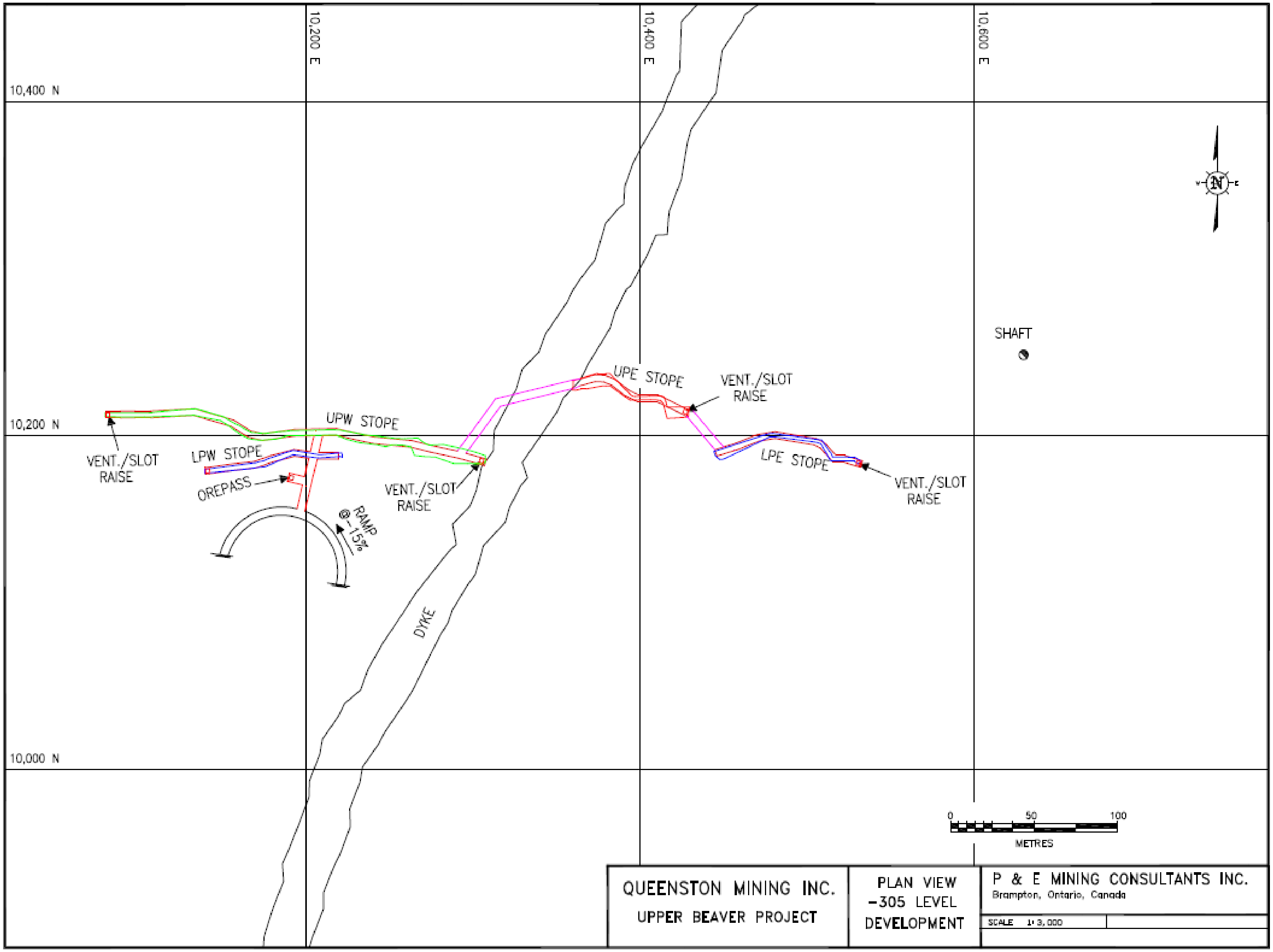


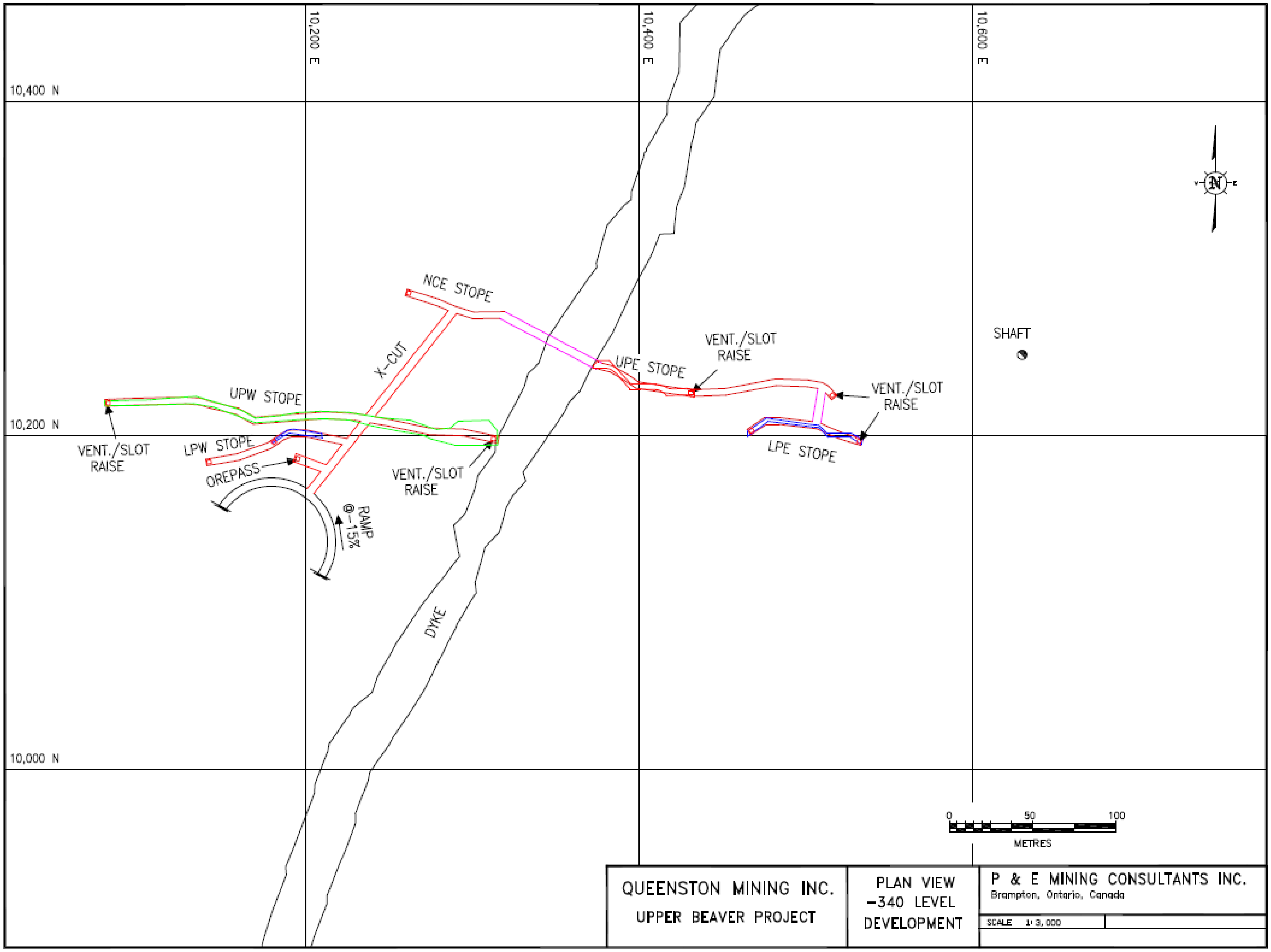
10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
VENT/SLOT RAISE
NCA STOPE
VENT/SLOT RAISE
NCD STOPE
NCE STOPE
UPE STOPE
UPE STOPE
VENT/SLOT RAISE
VENT/SLOT RAISE
VENT/SLOT RAISE
LPE STOPE
X-CUT
OREPASS
TO WEST RAMP
RAMP @-15%
DYKE
SHAFT
SHAFT ACCESS DRIFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-165 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000


10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
VENT/SLOT RAISE
NCB STOPE
NCC STOPE
NCD STOPE
VENT/SLOT RAISE
NCE STOPE
UPW STOPE
LPW STOPE
RAMP @-15%
OREPASS
DYKE
UPE STOPE
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
X-CUT
RAMP @-15%
SHAFT
N
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-200 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1:3,000

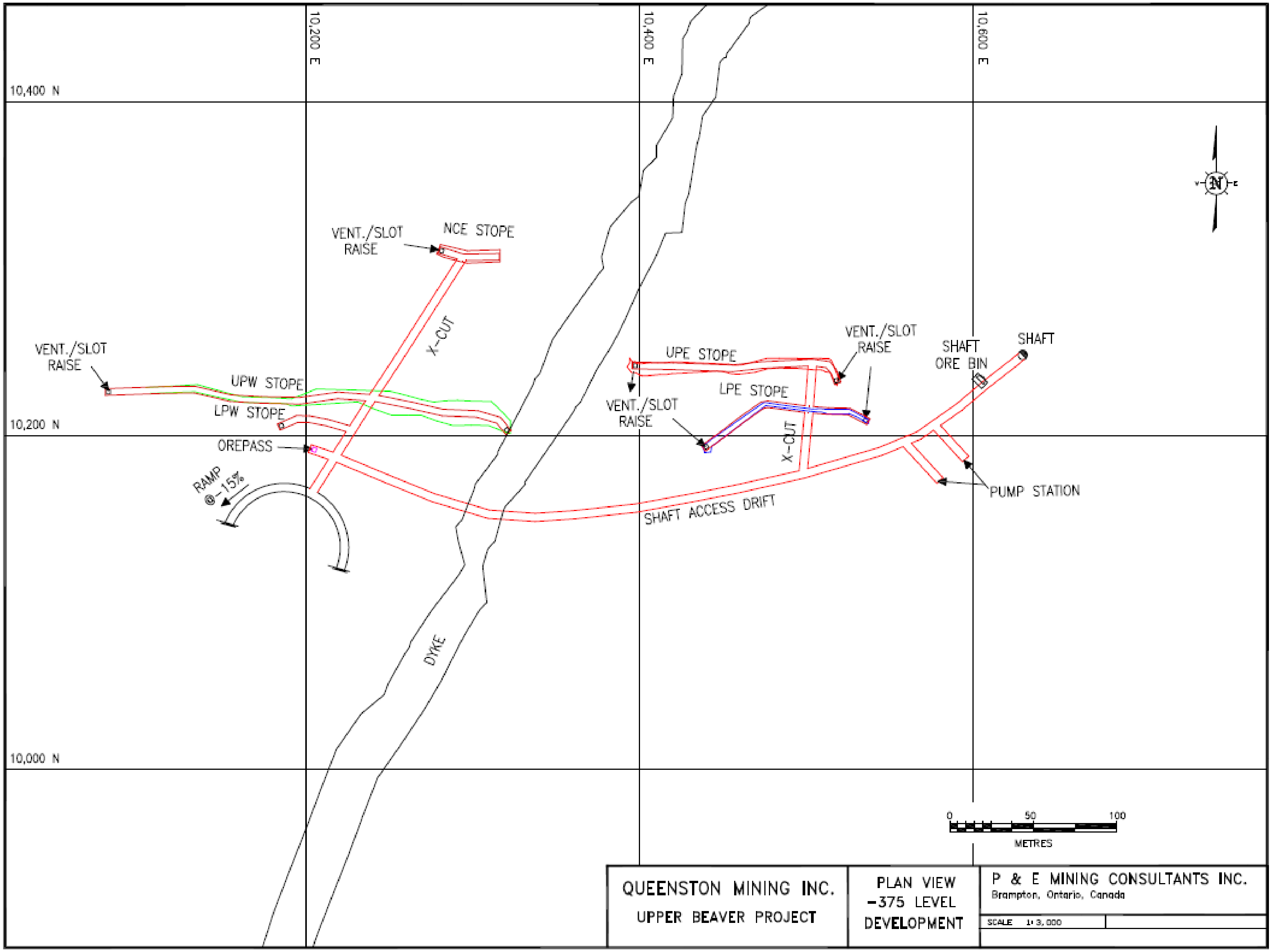

10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
NCD STOPE
VENT/SLOT RAISE
VENT/SLOT RAISE
UPW STOPE
LPW STOPE
OREPASS
RAMP @-15%
DYKE
VENT/SLOT RAISE
UPE STOPE
VENT/SLOT RAISE
X-CUT
RAMP @-15%
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-235 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000


10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
N
SHAFT
VENT./SLOT RAISE
UPW STOPE
VENT./SLOT RAISE
UPE STOPE
VENT./SLOT RAISE
LPW STOPE
LPW STOPE
LPE STOPE
X-CUT
RAMP @-15%
OREPASS
SHAFT ACCESS DRIFT
DYKE
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-270 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

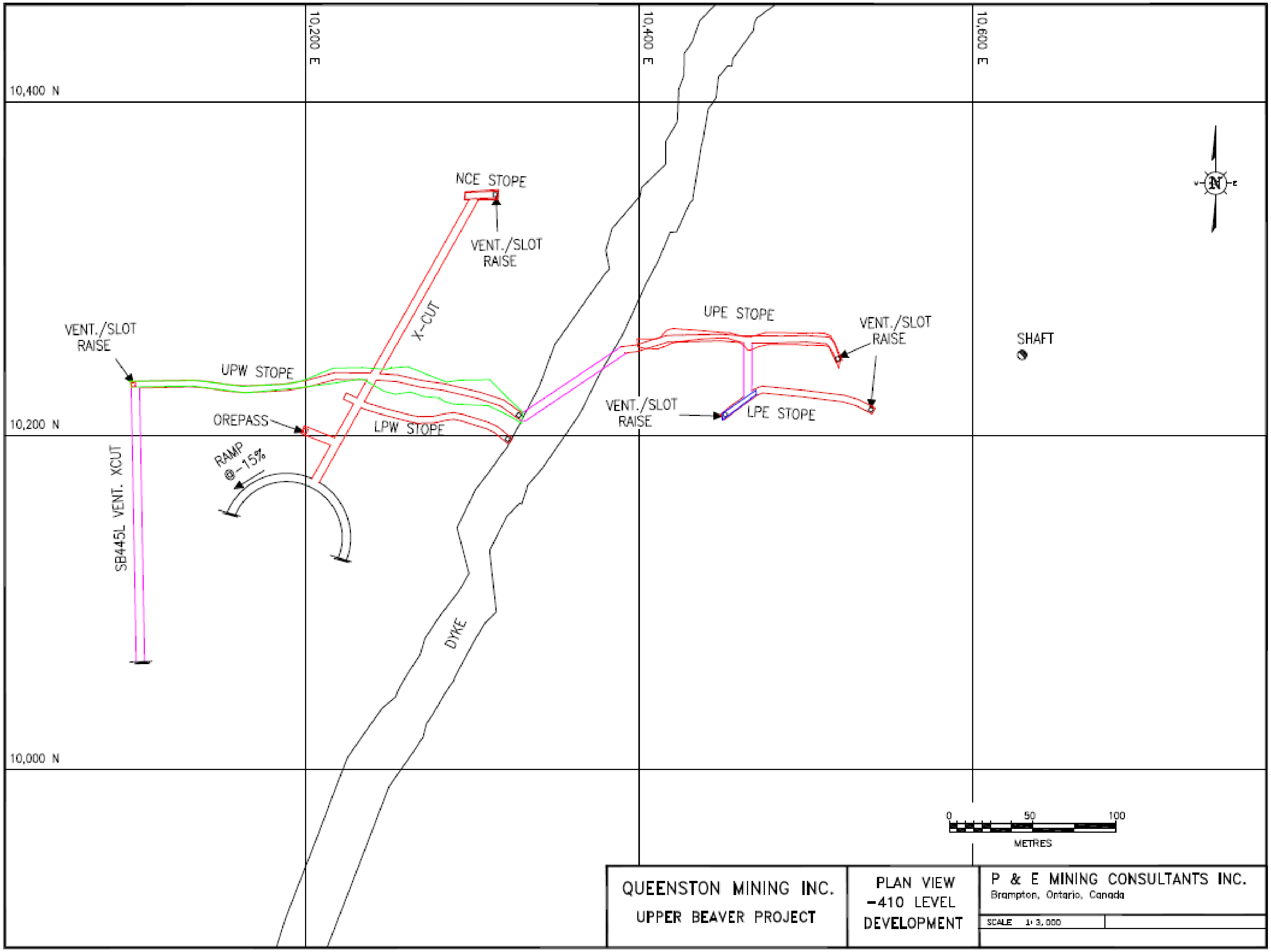

10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
SHAFT
UPE STOPE
VENT./SLOT RAISE
UPW STOPE
LPW STOPE
LPE STOPE
OREPASS
RAMP @-15%
DYKE
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-305 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

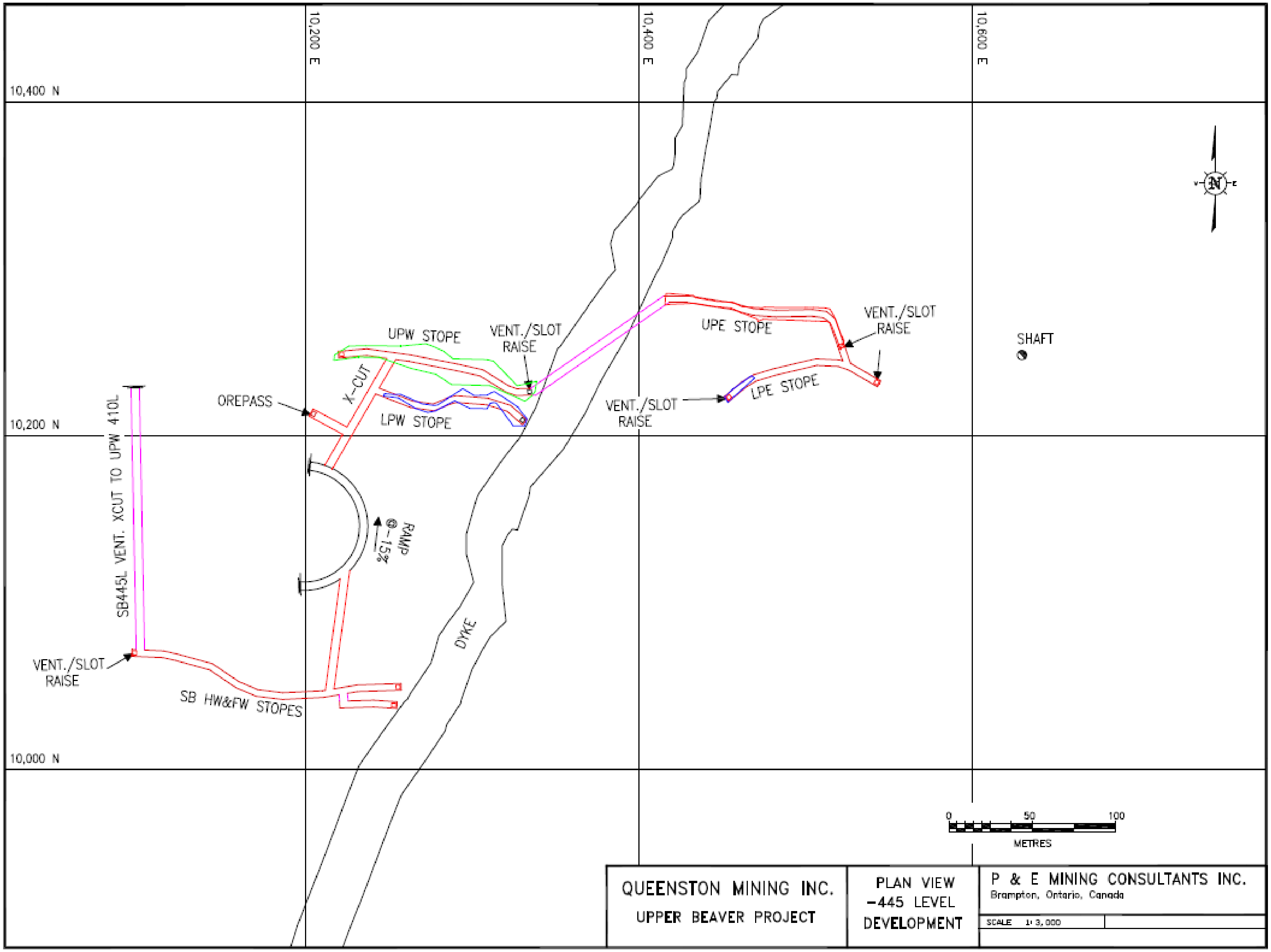


10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
NCE STOPE
X-CUT
UPW STOPE
UPE STOPE
VENT./SLOT RAISE
VENT./SLOT RAISE
VENT./SLOT RAISE
VENT./SLOT RAISE
LPW STOPE
OREPASS
LPE STOPE
RAMP @-15%
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW -340 LEVEL DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1:3,000

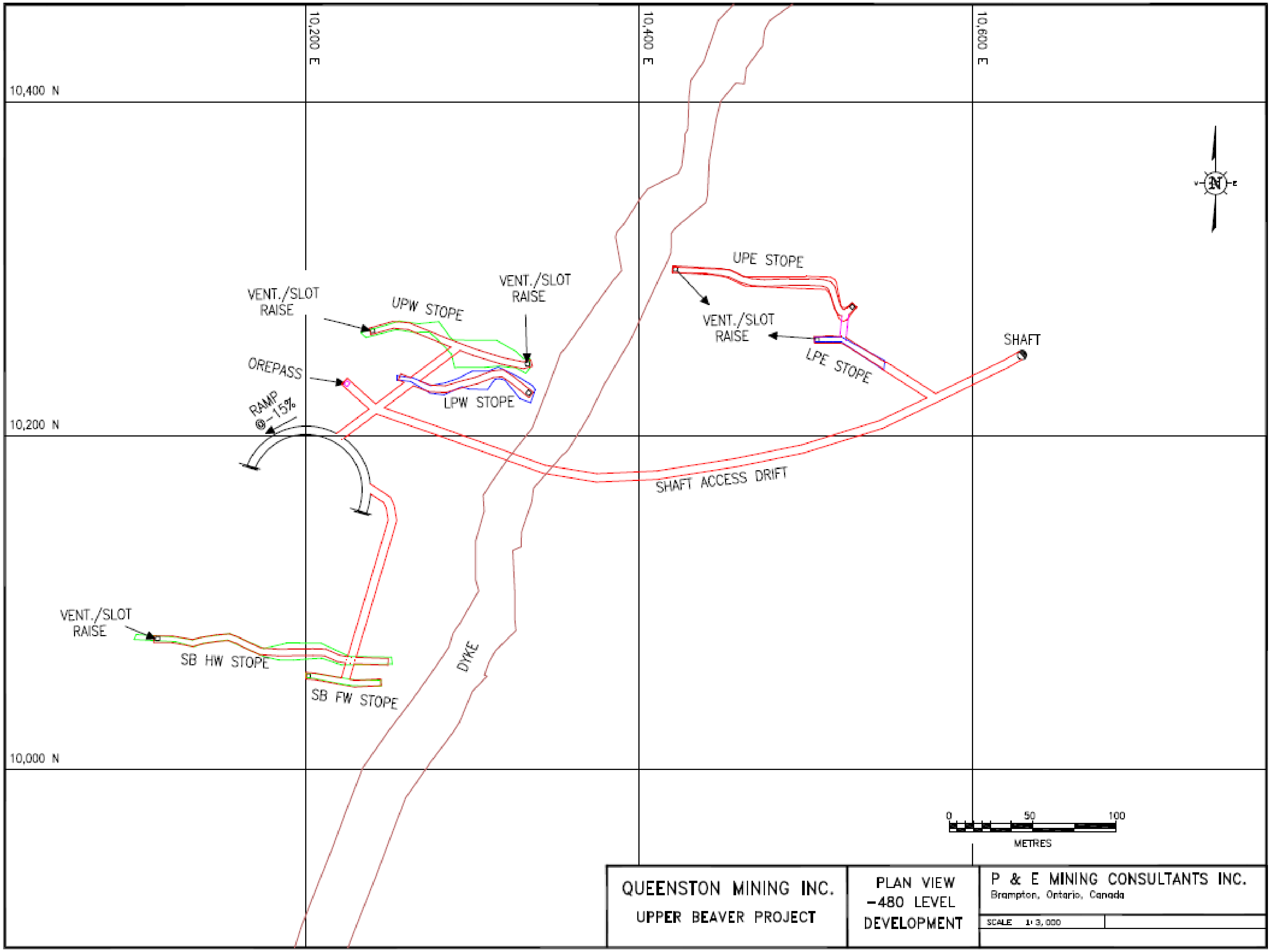


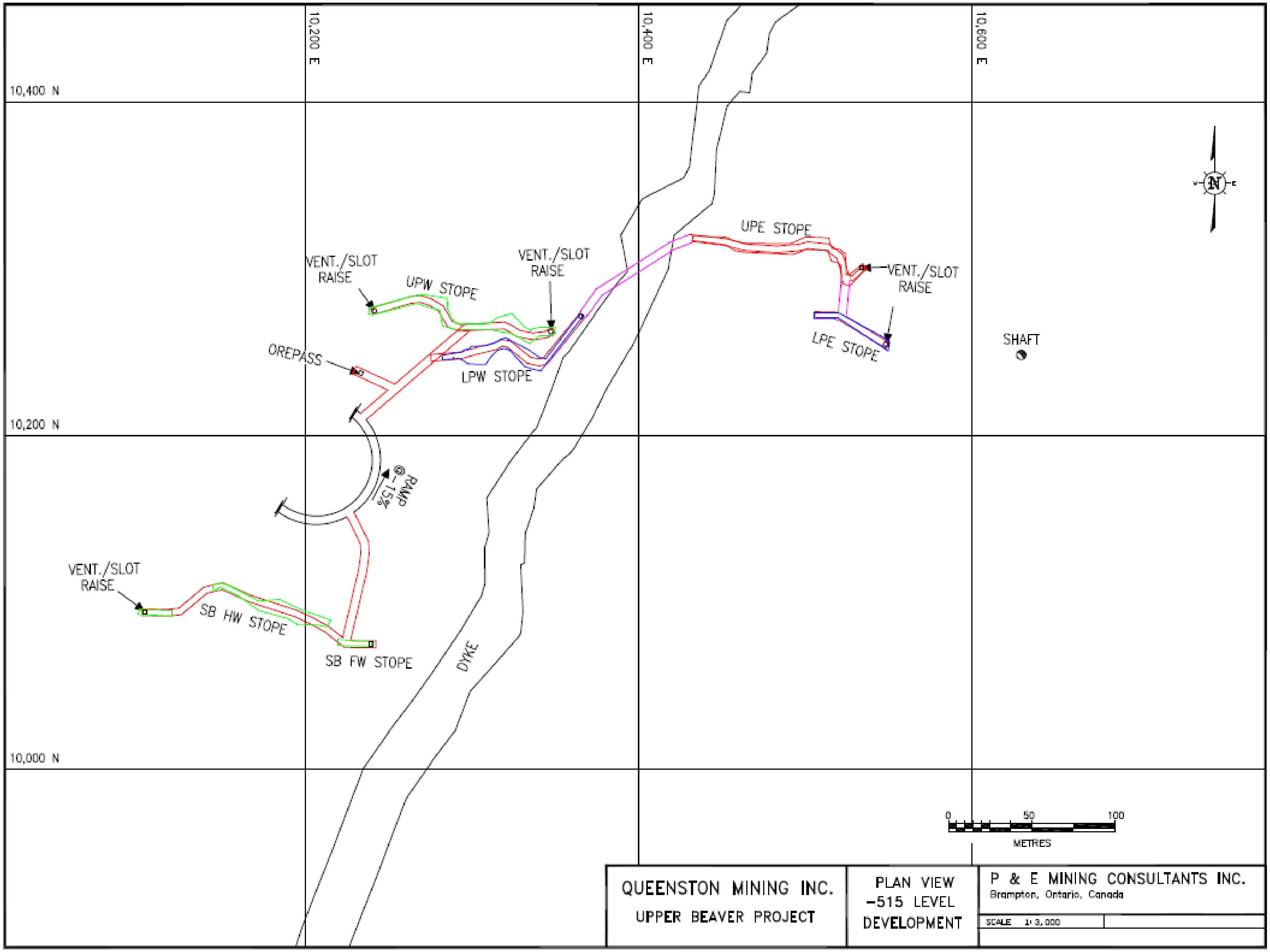
10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
N
VENT./SLOT RAISE
NCE STOPE
X-CUT
VENT./SLOT RAISE
UPW STOPE
LPW STOPE
OREPASS
RAMP @-15%
UPE STOPE
VENT./SLOT RAISE
SHAFT ORE BIN
SHAFT
LPE STOPE
VENT./SLOT RAISE
X-CUT
PUMP STATION
SHAFT ACCESS DRIFT
DYKE
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-375 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

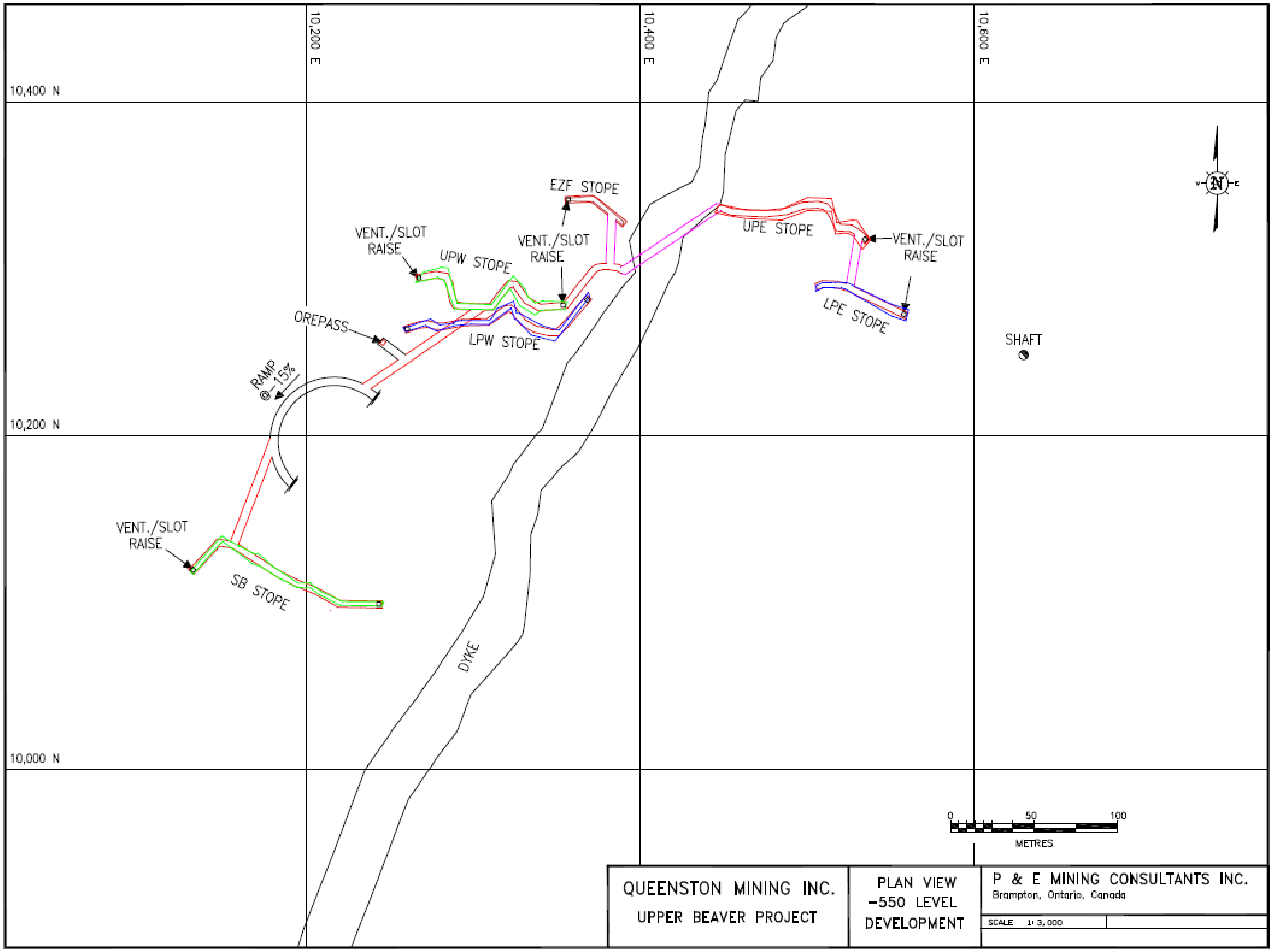

10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
NCE STOPE
VENT./SLOT
RAISE
X-CUT
VENT./SLOT
RAISE
UPW STOPE
UPE STOPE
VENT./SLOT
RAISE
SHAFT
OREPASS
LPW STOPE
VENT./SLOT
RAISE
LPE STOPE
RAMP
@-15%
SB445L VENT. XCUT
DYKE
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-410 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

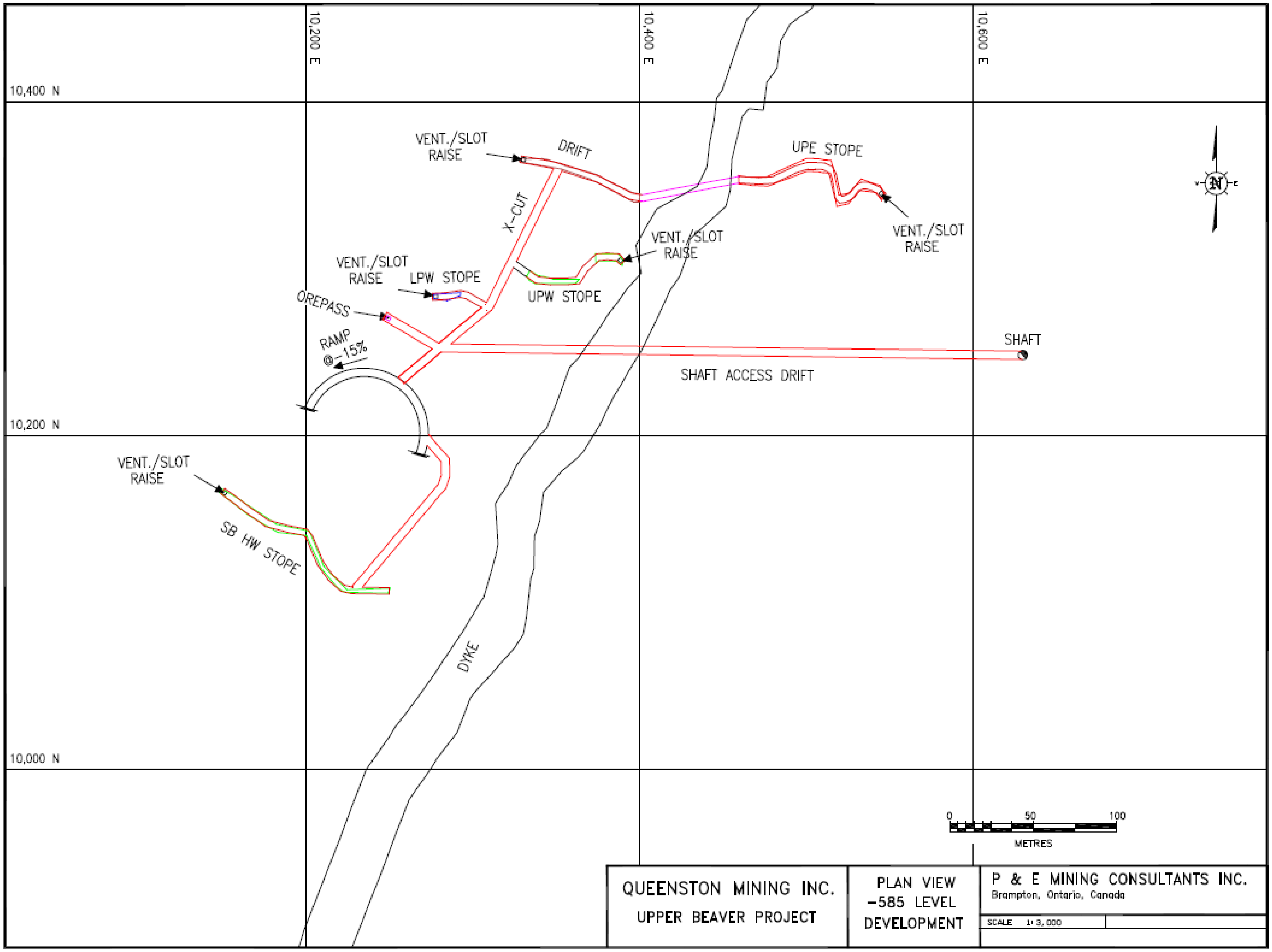


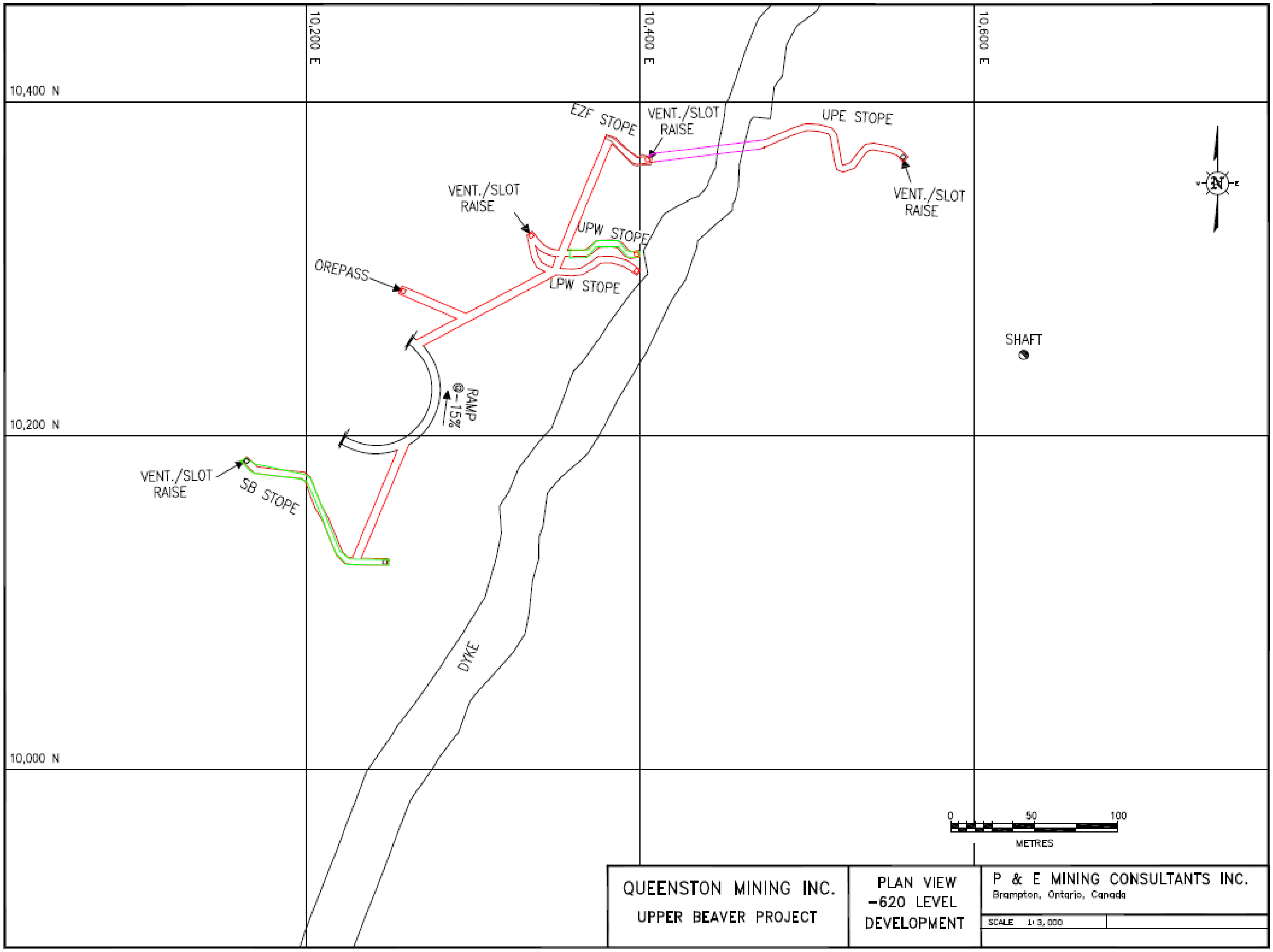
10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
UPW STOPE
VENT./SLOT RAISE
UPE STOPE
VENT./SLOT RAISE
SHAFT
OREPASS
X-CUT
LPW STOPE
VENT./SLOT RAISE
LPE STOPE
SB445L VENT. XCUT TO UPW 410L
RAMP @ -15%
DYKE
VENT./SLOT RAISE
SB HW&FW STOPES
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-445 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000


10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
N
VENT./SLOT RAISE
UPW STOPE
VENT./SLOT RAISE
OREPASS
RAMP @-15%
LPW STOPE
UPE STOPE
VENT./SLOT RAISE
LPE STOPE
SHAFT
SHAFT ACCESS DRIFT
VENT./SLOT RAISE
SB HW STOPE
SB FW STOPE
DYKE
0 50 100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-480 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

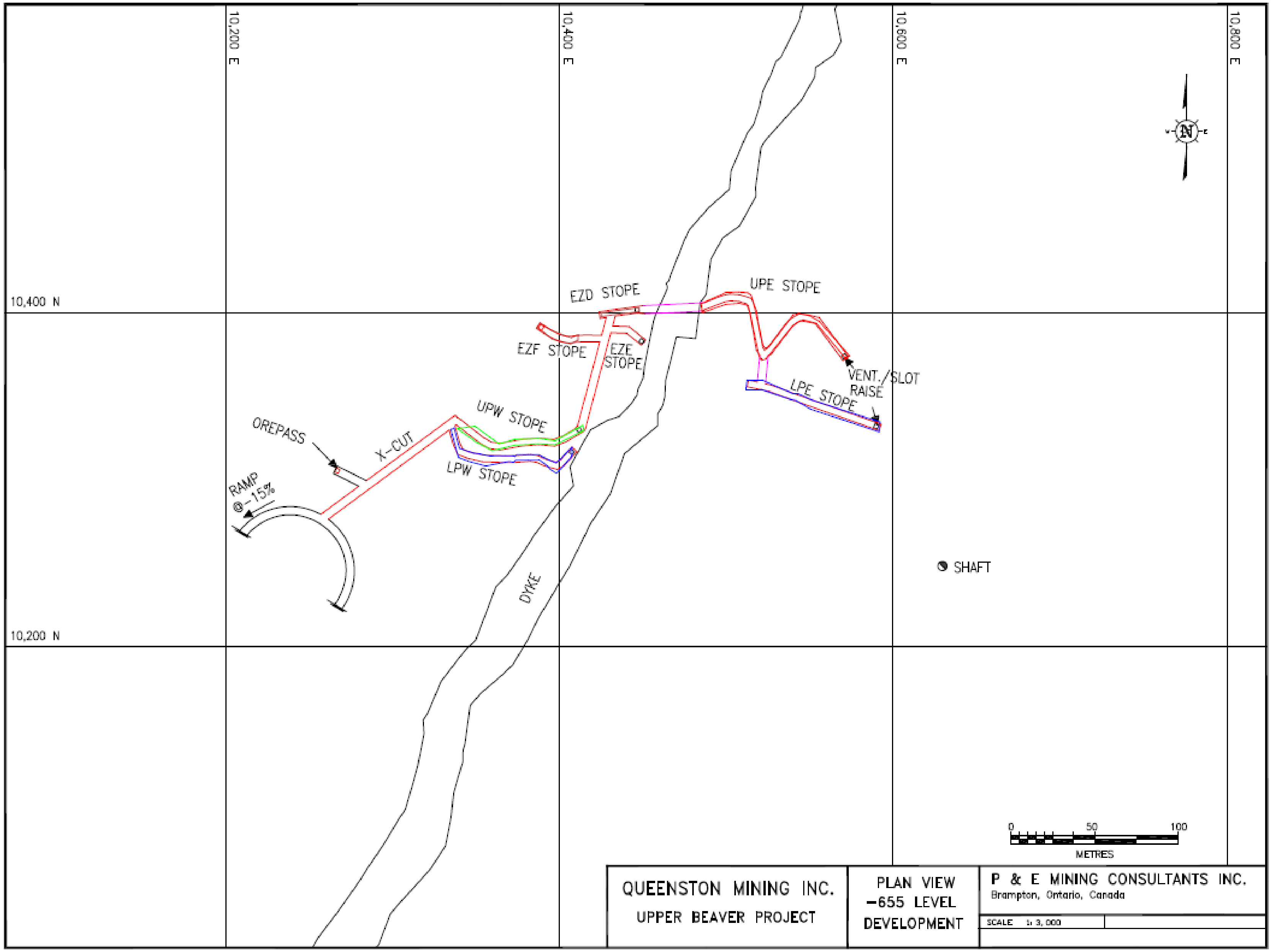

10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
VENT./SLOT RAISE
UPW STOPE
VENT./SLOT RAISE
UPE STOPE
VENT./SLOT RAISE
OREPASS
LPW STOPE
LPE STOPE
SHAFT
RAMP @ -15%
VENT./SLOT RAISE
SB HW STOPE
SB FW STOPE
DYKE
N
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-515 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

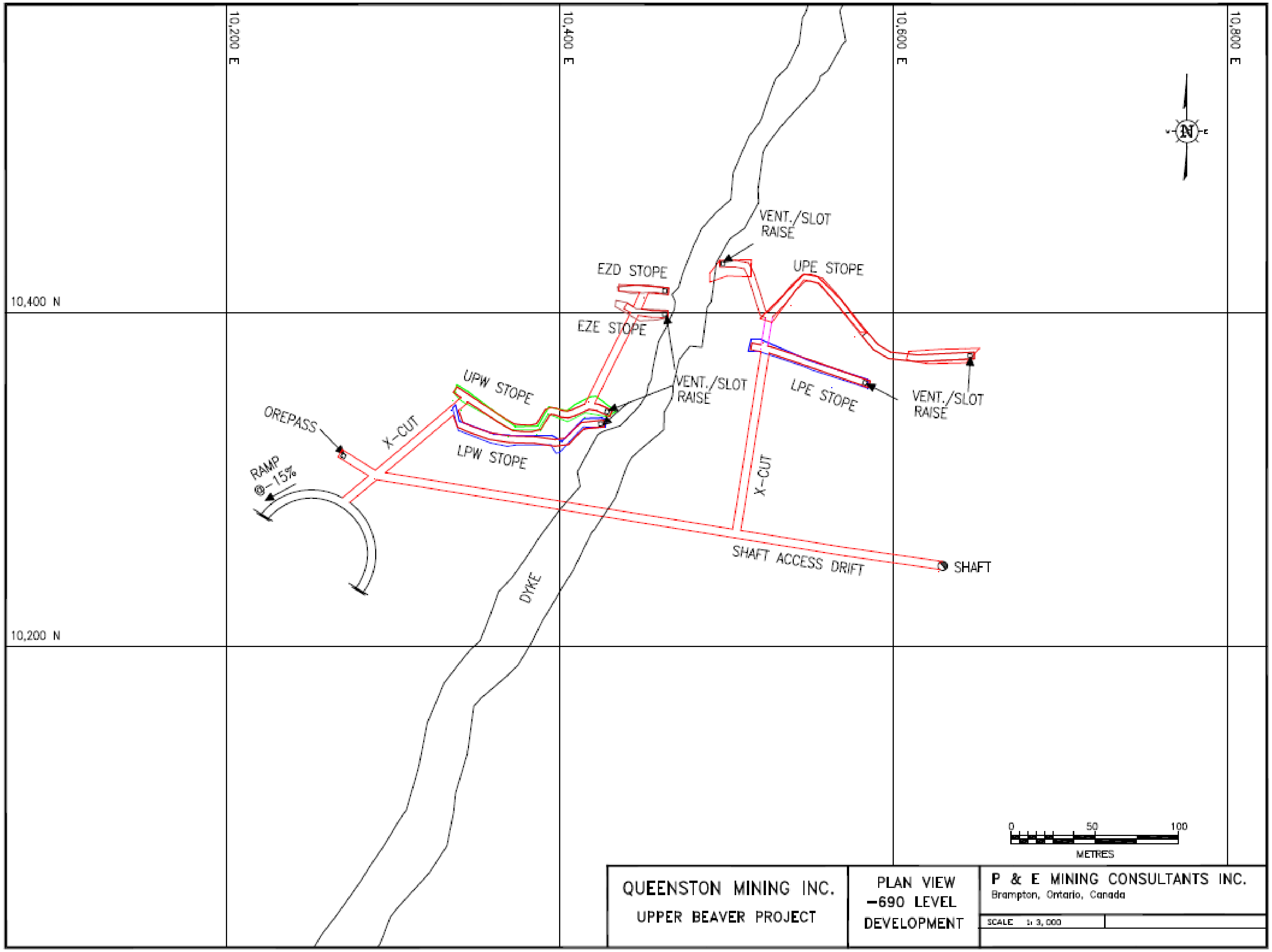

10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
EZF STOPE
VENT./SLOT RAISE
UPW STOPE
VENT./SLOT RAISE
UPE STOPE
VENT./SLOT RAISE
OREPASS
LPW STOPE
LPE STOPE
SHAFT
RAMP @-15%
VENT./SLOT RAISE
SB STOPE
DYKE
N
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-550 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

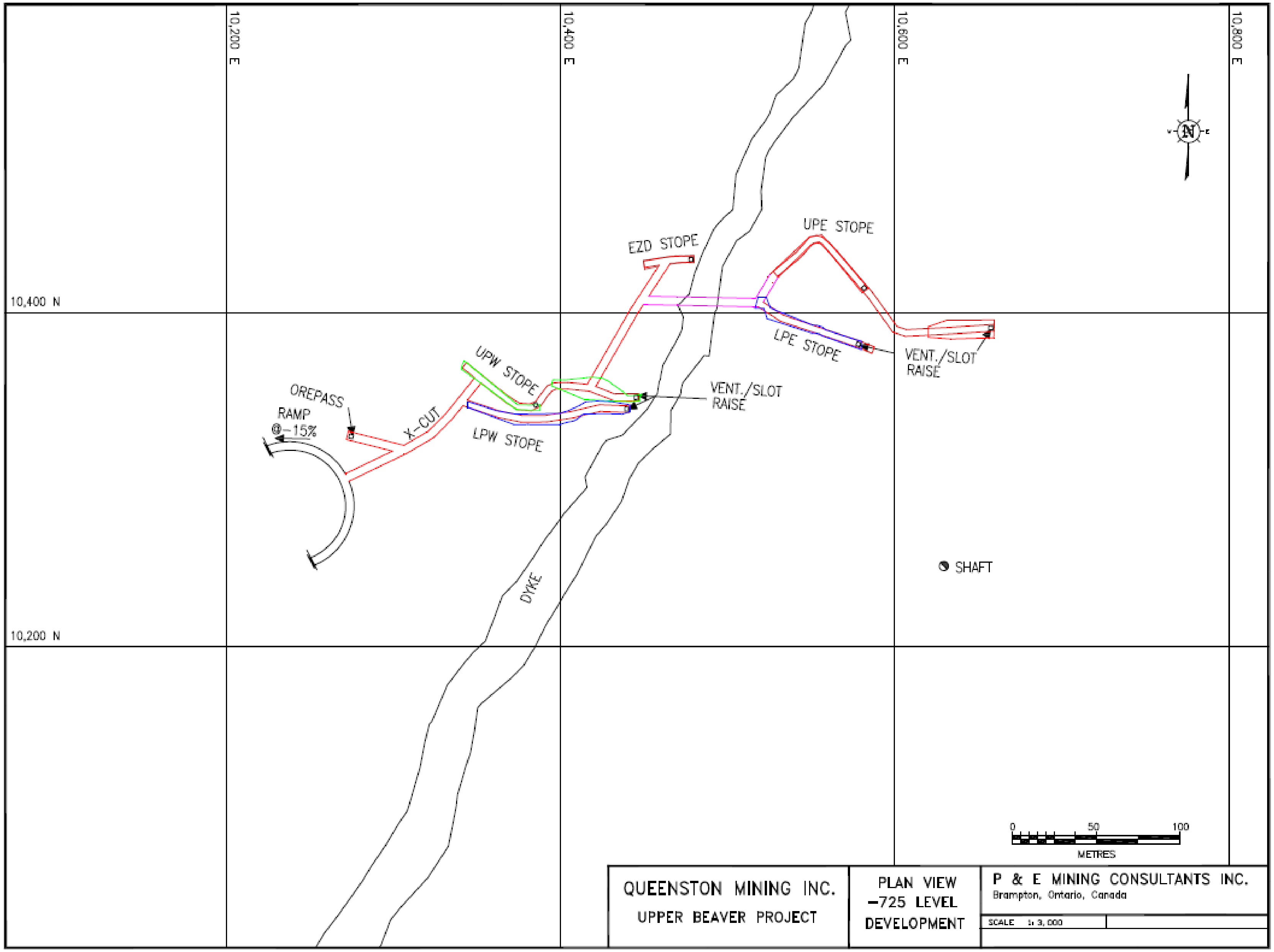

10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
VENT./SLOT RAISE
DRIFT
UPE STOPE
X-CUT
VENT./SLOT RAISE
VENT./SLOT RAISE
VENT./SLOT RAISE
LPW STOPE
UPW STOPE
OREPASS
RAMP @-15%
SHAFT
SHAFT ACCESS DRIFT
VENT./SLOT RAISE
SB HW STOPE
DYKE
N
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-585 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1:3,000



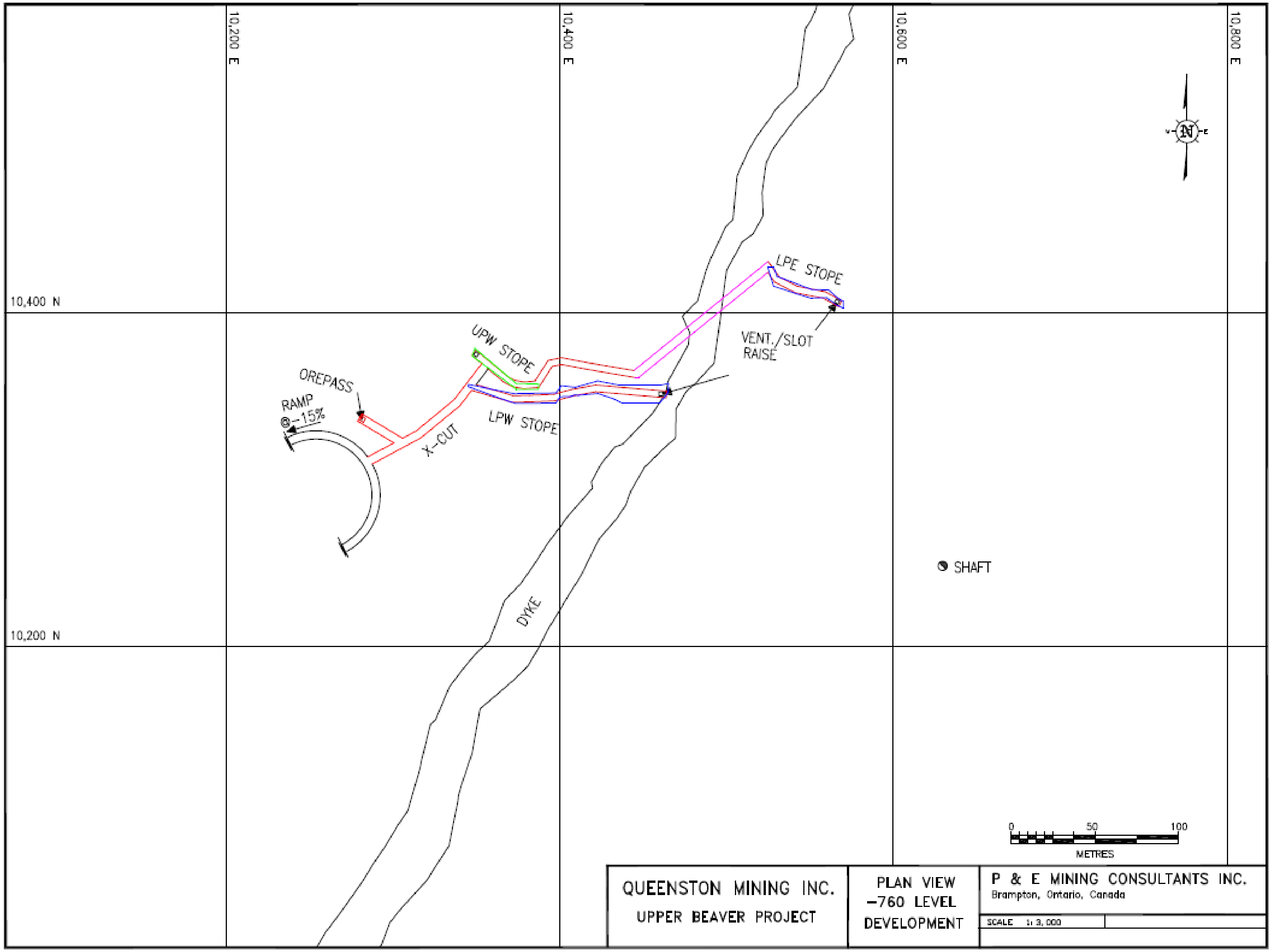
10,200 E
10,400 E
10,600 E
10,400 N
10,200 N
10,000 N
EZF STOPE
VENT./SLOT RAISE
UPE STOPE
VENT./SLOT RAISE
VENT./SLOT RAISE
UPW STOPE
OREPASS
LPW STOPE
SHAFT
RAMP @-15%
VENT./SLOT RAISE
SB STOPE
DYKE
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-620 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

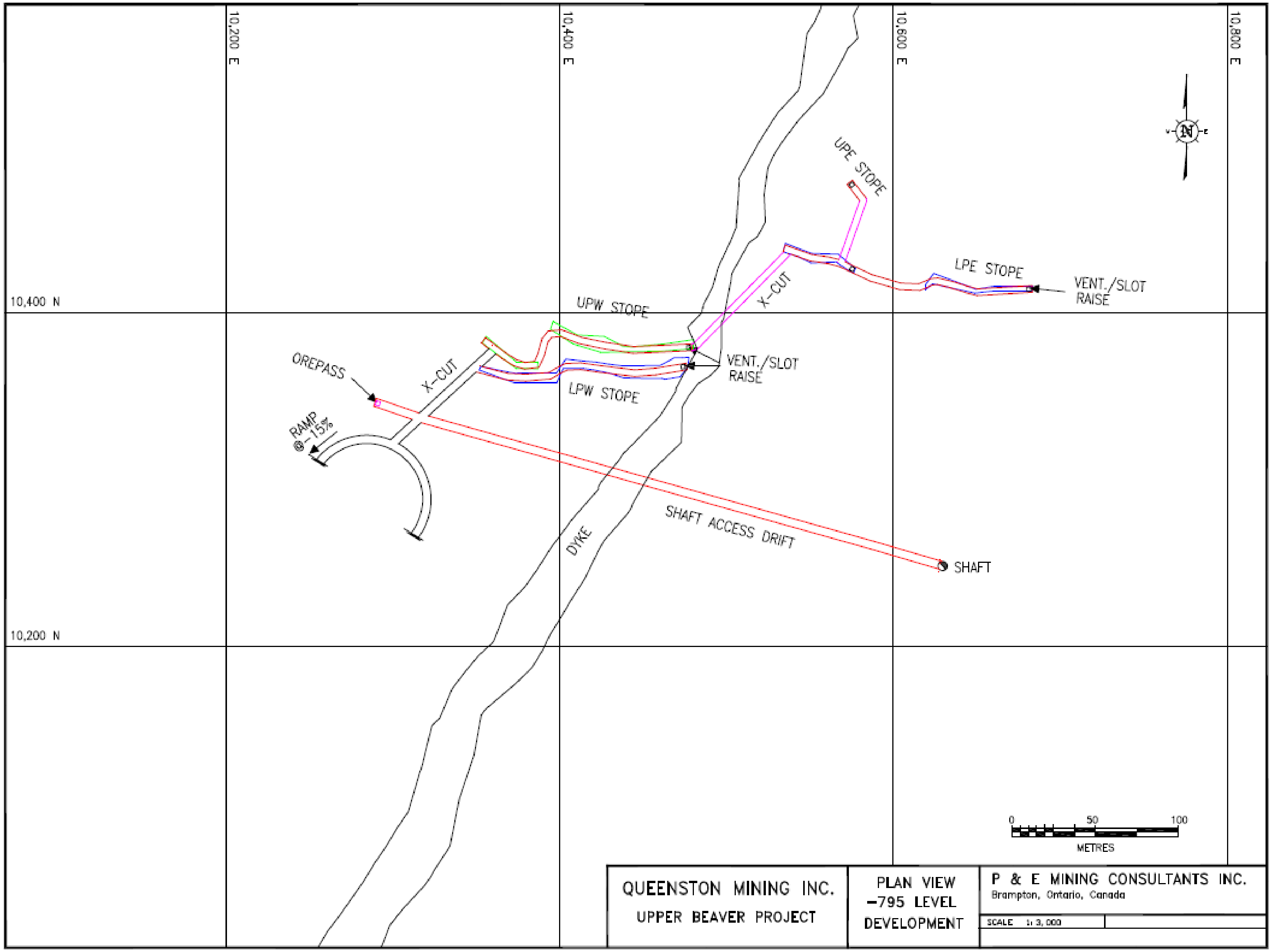

10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
EZD STOPE
UPE STOPE
EZF STOPE
EZE STOPE
VENT./SLOT RAISE
LPE STOPE
UPW STOPE
OREPASS
X-CUT
LPW STOPE
RAMP @-15%
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-655 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

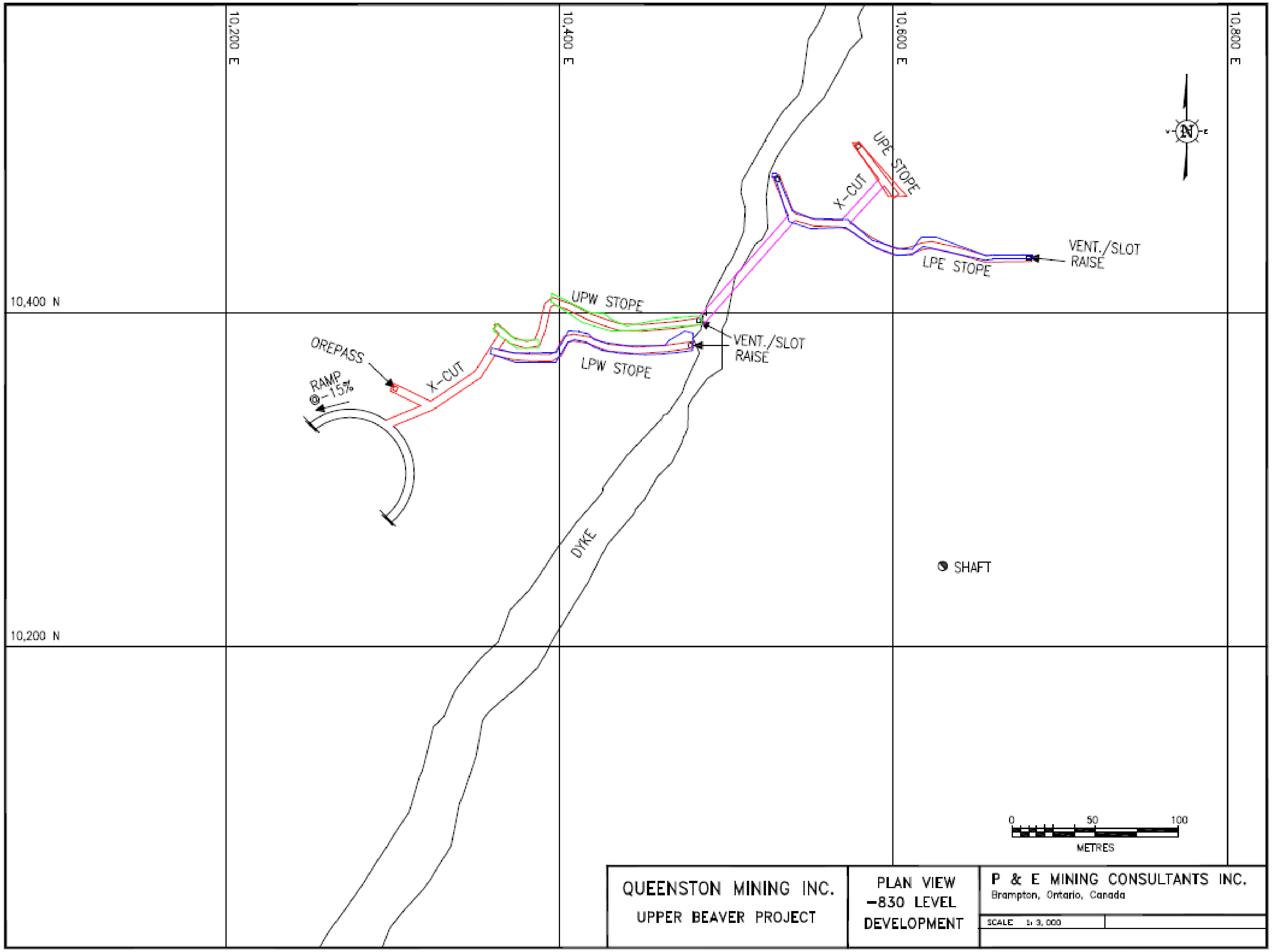

10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
VENT./SLOT RAISE
EZD STOPE
UPE STOPE
EZE STOPE
UPW STOPE
VENT./SLOT RAISE
LPE STOPE
VENT./SLOT RAISE
OREPASS
X-CUT
LPW STOPE
RAMP @-15%
X-CUT
SHAFT ACCESS DRIFT
SHAFT
DYKE
0 50 100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-690 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

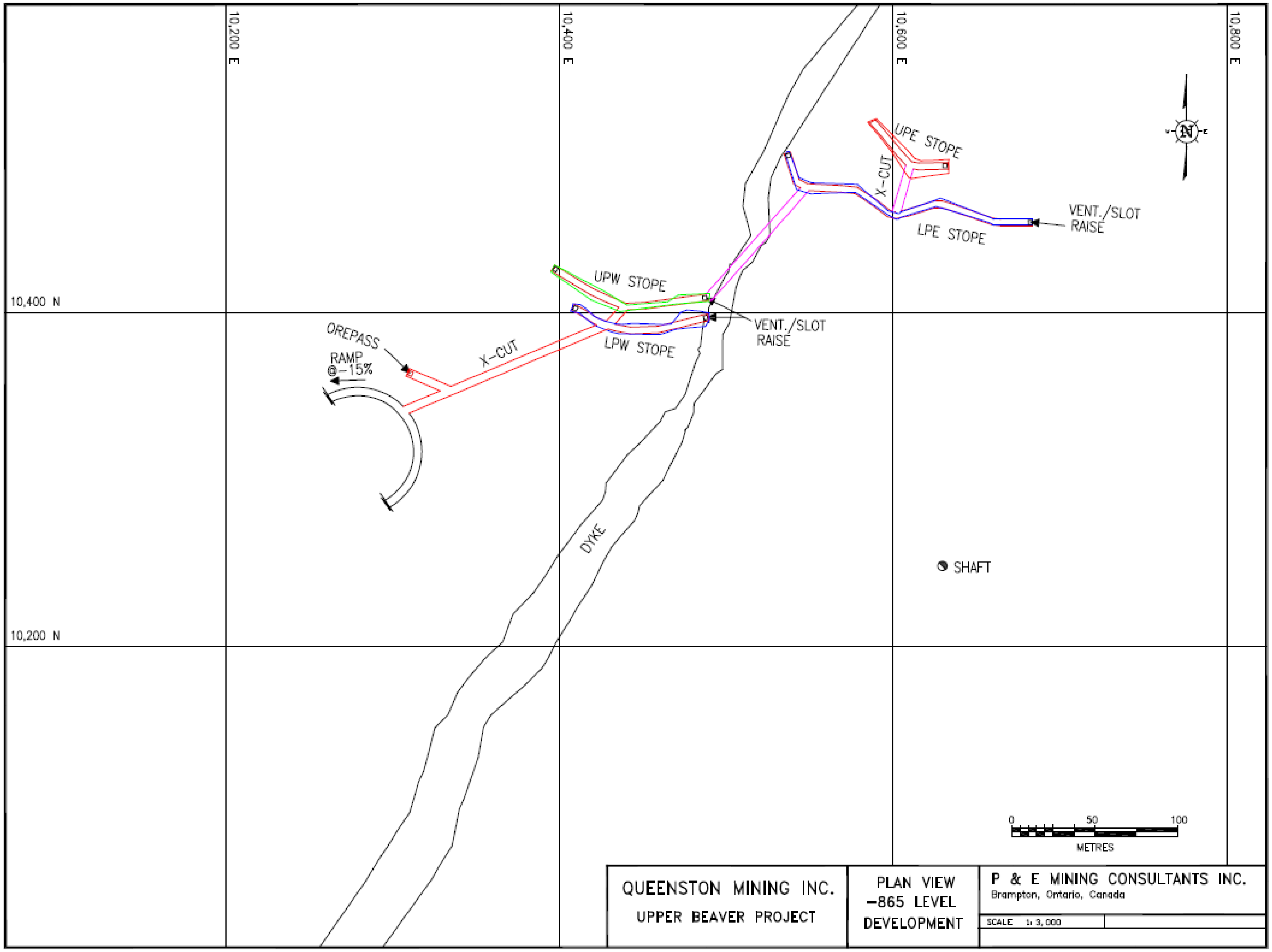


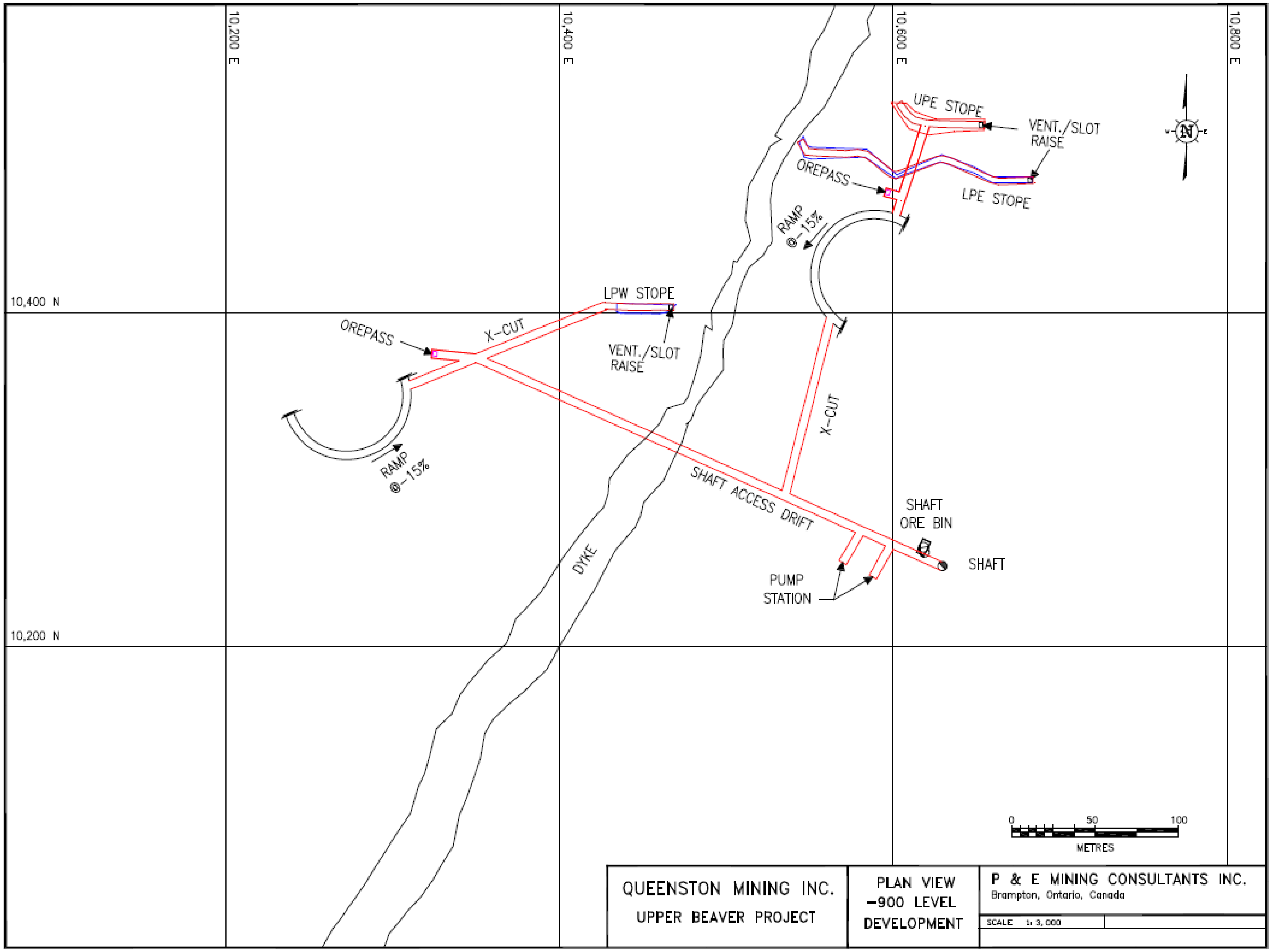
10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
EZD STOPE
UPE STOPE
LPE STOPE
VENT./SLOT RAISE
UPW STOPE
LPW STOPE
VENT./SLOT RAISE
OREPASS
RAMP @-15%
X-CUT
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-725 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

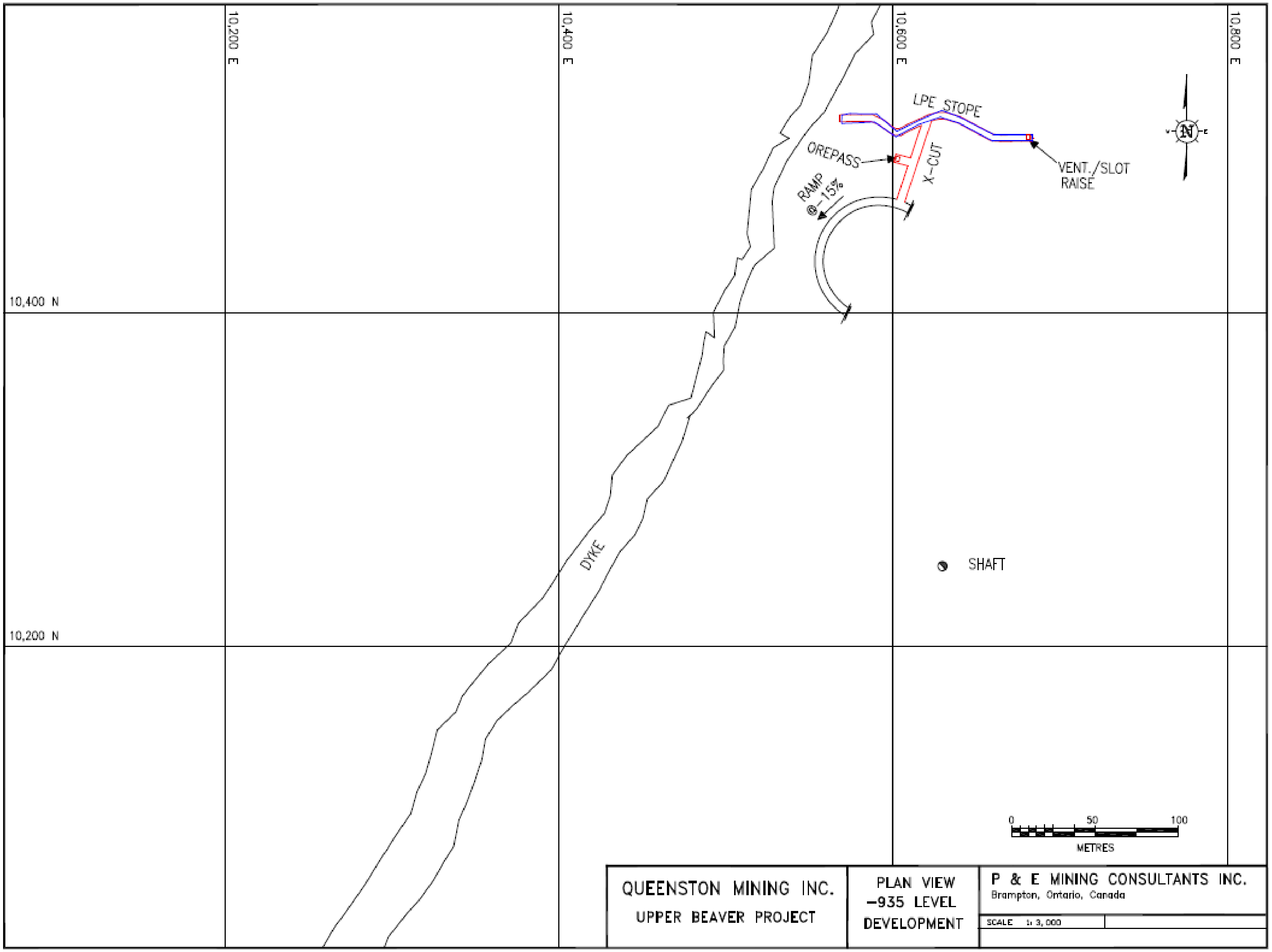

10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
N
LPE STOPE
VENT./SLOT RAISE
UPW STOPE
OREPASS
RAMP @-15%
LPW STOPE
X-CUT
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-760 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000

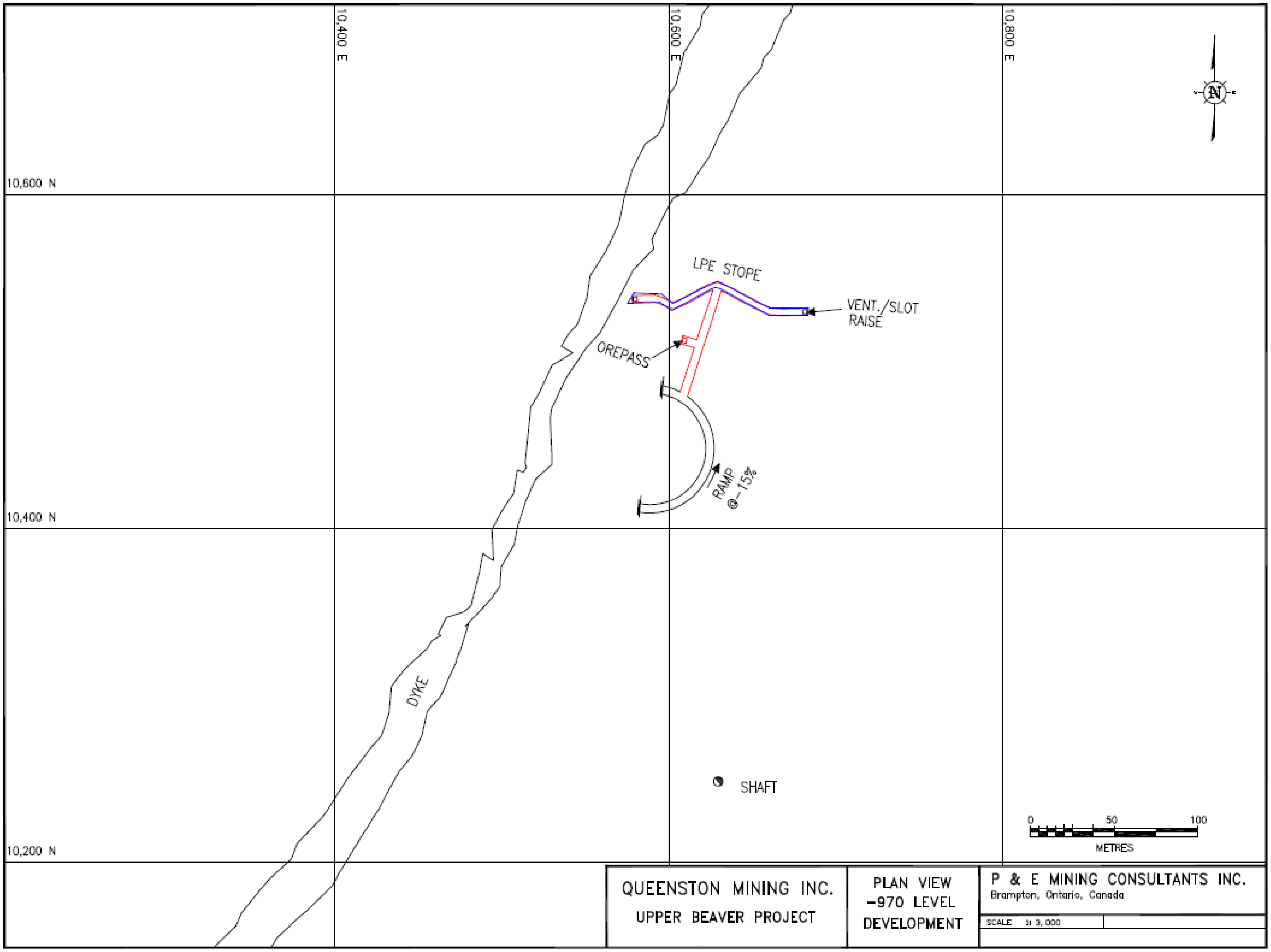

10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
UPE STOPE
LPE STOPE
VENT./SLOT RAISE
X-CUT
UPW STOPE
LPW STOPE
VENT./SLOT RAISE
OREPASS
X-CUT
RAMP @-15%
SHAFT ACCESS DRIFT
DYKE
SHAFT
N
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-795 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000


10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
UPE STOPE
X-CUT
VENT./SLOT
RAISE
LPE STOPE
UPW STOPE
LPW STOPE
OREPASS
RAMP
@-15%
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-830 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000


10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
UPE STOPE
X-CUT
VENT./SLOT
RAISE
LPE STOPE
UPW STOPE
VENT./SLOT
RAISE
LPW STOPE
OREPASS
RAMP
@-15%
X-CUT
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-865 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000


10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
UPE STOPE
VENT./SLOT
RAISE
OREPASS
LPE STOPE
RAMP
@-15%
LPW STOPE
OREPASS
X-CUT
VENT./SLOT
RAISE
RAMP
@-15%
X-CUT
SHAFT ACCESS DRIFT
SHAFT
ORE BIN
SHAFT
PUMP
STATION
DYKE
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-900 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3, 000


10,200 E
10,400 E
10,600 E
10,800 E
10,400 N
10,200 N
LPE STOPE
OREPASS
X-CUT
RAMP @-15%
VENT./SLOT RAISE
N
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-935 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000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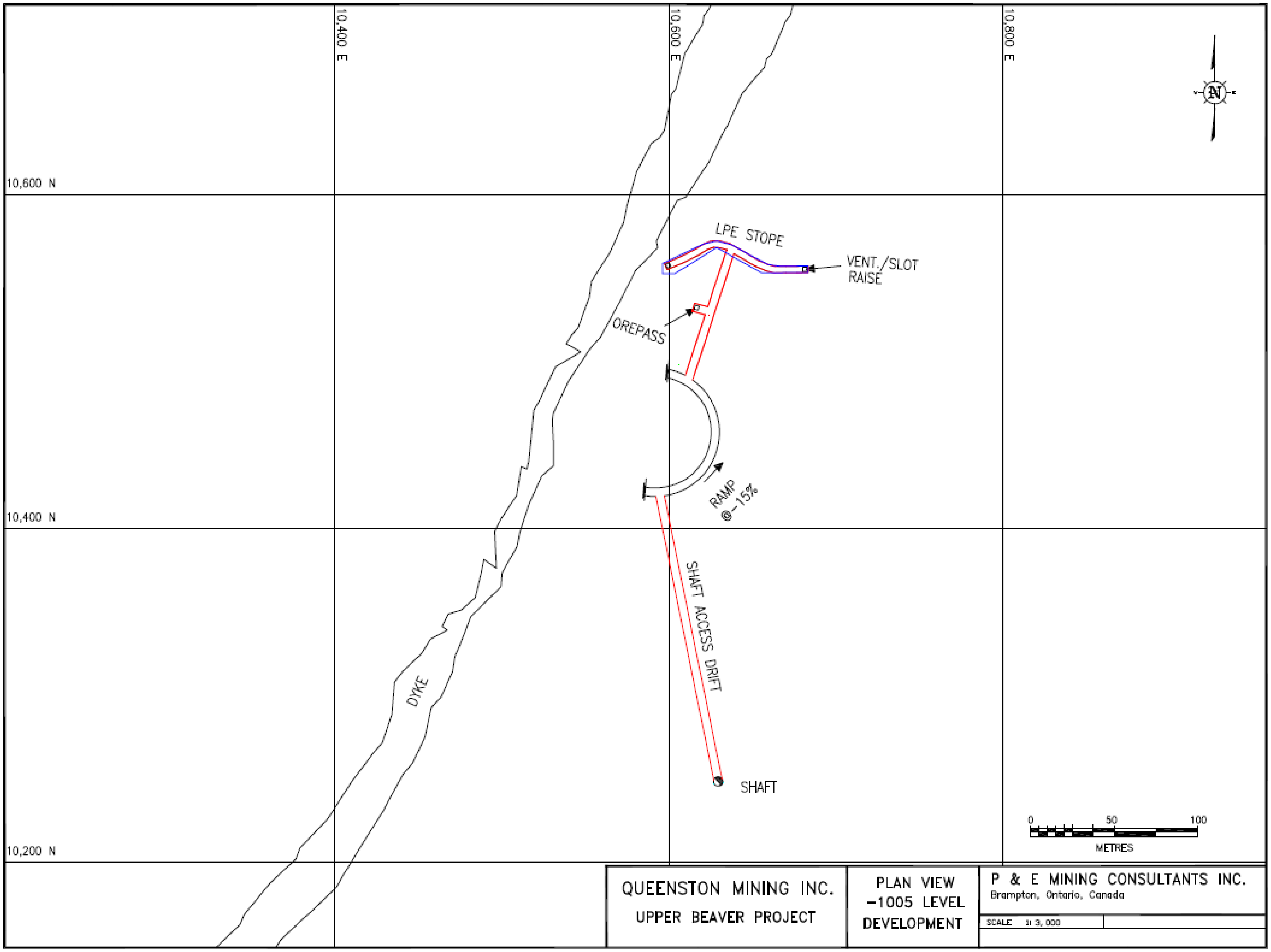
10,600 E
10,800 E
10,600 N
10,400 N
10,200 N
LPE STOPE
VENT./SLOT RAISE
OREPASS
RAMP @-15%
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-970 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000


10,400 E
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N
10,600 N
10,400 N
10,200 N
LPE STOPE
VENT./SLOT
RAISE
OREPASS
RAMP
@-15%
SHAFT ACCESS DRIFT
DYKE
SHAFT
0
50
100
METRES
QUEENSTON MINING INC.
UPPER BEAVER PROJECT
PLAN VIEW
-1005 LEVEL
DEVELOPMENT
P & E MINING CONSULTANTS INC.
Brampton, Ontario, Canada
SCALE 1: 3,000