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PRELIMINARY ECONOMIC ASSESSMENT OF THE THIERRY CU-NI-PGE DEPOSIT THIERRY PROJECT, PICKLE LAKE AREA PATRICIA MINING DISTRICT NORTH-WESTERN ONTARIO, CANADA

51°29’51.32” N 90°20’52.45” W

FOR
BRAVEHEART RESOURCES INC.

NI 43-101 & 43-101F1
TECHNICAL REPORT

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P & E Mining Consultants Inc.
Report 391

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1.0 SUMMARY

Braveheart Resources Inc. ("Braveheart") retained P&E Mining Consultants Inc. ("P&E") to prepare an independent NI 43-101 Technical Report and Preliminary Economic Assessment ("PEA") of the Thierry copper-nickel platinum group elements ("PGE") underground Deposit ("Thierry Project"), and to report the Mineral Resources of the adjacent K1-1 Deposit. Both deposits are located 12 km west of Pickle Lake and 450 km northwest of Thunder Bay, Ontario. The purpose of this report is to provide updated Mineral Resource Estimates and to evaluate the economic potential of an underground mining operation at the Thierry Deposit that would feed a processing facility to produce two concentrates containing copper, nickel, silver, gold, platinum and palladium.

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

1.1 PROPERTY LOCATION AND TENURE

The Thierry Project is located 12 km west of Pickle Lake and is accessible by all-weather road. The Thierry Project is comprised of 27 mining leases totalling 4,670 hectares located in the Dona Lake, Ponsford Lake, Tarp Lake and Kapkichi Lake areas in the Patricia Mining District, northwestern Ontario. The Thierry Project also contains 163 contiguous cell claims and 15 boundary claims totalling 3,258 hectares. The total combined Property area is 7,907 hectares.

1.2 HISTORY

Records indicate that gold was discovered at the Property in 1928. Intermittent gold mining operations were carried out over the years until 1966. Union Miniere Explorations and Mining Corporation ("UMEX") acquired the Project in 1969 and conducted exploration and metallurgical testwork. After a positive Feasibility Study, a decision was made in 1974 to proceed with development of the Deposit, and mine production. The mine was operated from 1976 to 1982. Further exploration was subsequently carried out by various firms, and in 2010 Cadillac Ventures acquired the Project. Cadillac conducted drilling exploration until 2012 and issued a PEA on the Project. Braveheart Resources acquired the Project in 2020.

1.3 SAMPLE PREPARATION, DATA VERIFICATION, QA/QC

It is P&E's opinion that sample preparation, security and analytical procedures for the Project drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate. Based upon the evaluation of the QA/QC programs undertaken by Cadillac, P&E concludes that the data are of good quality for use in the Mineral Resource Estimate. Based upon P&E's due diligence sampling and data verification, P&E concludes that the data are satisfactory for use in the Mineral Resource Estimate.

1.4 MINERAL RESOURCES

The updated Thierry Mineral Resource Estimate is presented in Table 1.1.

TABLE 1.1 **THIERRY MINERAL RESOURCE ESTIMATE AT $60/T NSR CUT-OFF (1-6)**							
Classification	**Tonnes**	**Cu (%)**	**Ni (%)**	**Au (g/t)**	**Pt (g/t)**	**Pd (g/t)**	**Ag (g/t)**
Measured	3,233,000	1.65	0.19	0.03	0.03	0.09	4.6
Indicated	5,582,000	1.66	0.19	0.05	0.05	0.14	3.8
Meas & Ind	**8,815,000**	**1.66**	**0.19**	**0.05**	**0.04**	**0.13**	**4.0**
Inferred	14,922,000	1.64	0.16	0.10	0.07	0.21	6.4

Notes:

1) *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2) *The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3) *The Mineral Resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*
4) *The December 31, 2020 two-year trailing average US metal prices used in this estimate were $3.75/lb Cu, $6.25/lb Ni, $18.5/oz Ag, $1,600/oz Au, $900/oz Pt and $1,600/oz Pd. The CDN$:US$ exchange rate was 0.75.*
5) *Overall payable metal (process recovery x smelter payable) in the NSR calculation were 86% Cu, 33% Ni and 25% for Ag, Au, Pt & Pd.*
6) *Costs used to determine the $60/t NSR cut-off value are as follows: mining $40/t, processing $15/t and G&A $5/t.*

The K1-1 updated Mineral Resource Estimate is presented in Table 1.2.

TABLE 1.2 **K1-1 PIT CONSTRAINED INFERRED MINERAL RESOURCE ESTIMATE (1-9)**							
Cut-off NSR ($/tonne)	**Tonnes (k)**	**Cu (%)**	**Ni (%)**	**Au (g/t)**	**Pt (g/t)**	**Pd (g/t)**	**Ag (g/t)**
$12	53,614	0.38	0.10	0.03	0.05	0.14	1.8

Notes:

1) *CIM Definitions (2014) and Best Practices (2019) were followed for Mineral Resources.*
2) *Mineral Resources are estimated by conventional 3-D block modelling based on wireframing at a $12/tonne NSR cut-off value and ID^2 grade interpolation.*
3) *Metal prices for the estimate are: US$3.75/lb Cu, US$6.25/lb Ni, US$900/oz Pt, US$1,600/oz Pd, US$1,600/oz Au, US$18.50/oz Ag, based on Dec 31/2020 two-year trailing averages.*
4) *A uniform bulk density of 3.12 t/m^3 has been applied for volume to tonnes conversion.*
5) *The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
6) *Classification of Inferred Resources is based on wide drill hole spacing, lack of collar and down surveys for UMEX and 2002 series drilling and the lack of Au, Ag, Pt and Pd assays for more than 50% the sample data*

in the Mineral Resource. Regression based on available assays was used to generate PGM/PM values for the Mineral Resource Estimate.

7) *The Mineral Resource Estimate was determined within a constraining pit shell with 50 degree slopes utilizing mining costs of \$2.50/tonne for mineralized material, \$2.50/tonne for waste rock, and \$2.00/tonne for overburden. The pit constrained Mineral Resource is estimated below surface to a depth of 268 m.*
8) *Costs used to determine the \$12/tonne NSR Mineral Resource cut-off value were processing at \$10/tonne and G&A at \$2/tonne.*
9) *Overall payable metal in the NSR calculation were 86% Cu, 33% Ni and 25% for Ag, Au, Pt & Pd.*

1.5 CONCEPTUAL MINING PLAN

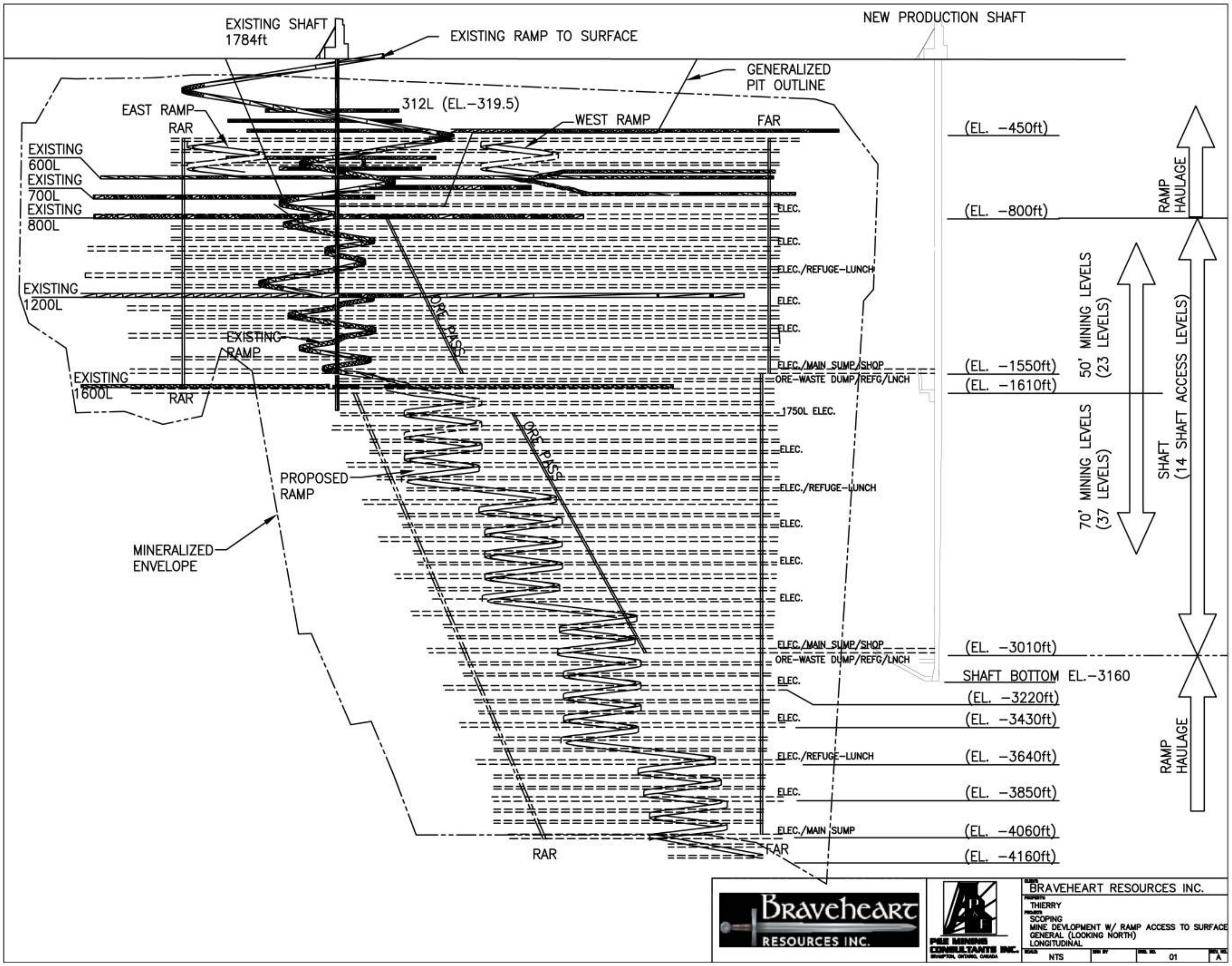
The Thierry Deposit will be mined by a combination of underground sublevel retreat long-hole methods and is envisaged to produce 4,000 tpd of process plant feed. A longitudinal projection of the proposed underground mine is shown in Figure 1.1.

Access to the Thierry Deposit would be via a 6.5 m diameter, concrete lined 965 m (3,160 ft) deep fresh air shaft and a -15% ramp from surface to a depth of 1,260 m (4,130 ft). There will be two shaft loading pocket stations, one at a depth of 475 m (1,550 ft) and a second at a depth of 920 m (3,010 ft). Two hoists would be configured to transport workers and skip mineralized rock between surface and the underground loading pocket levels. Construction and consumable materials would access or exit the underground via the ramp from surface.

The primary mining method would be conventional longitudinal long-hole retreat with paste backfill. Above the 490 m (1,610 ft) elevation, sub-levels will be developed at 15 m (50 ft) vertical intervals. Below the 490 m (1,610 ft) elevation, sub-levels will be developed at 21 m (70 ft) vertical intervals. Drifts in mineralization would be developed to the full width of the Mineral Resources. These drifts would provide access for the successive operations of slot raise development, blasthole drilling and blasting and backfill placement. Remote-operated underground load/haul/dump ("LHD") units would remove broken mineralized rock from the stopes. The stopes would be backfilled primarily with cemented paste backfill, supplemented with waste rock. Initially, mineralized rock above the 290 m (950 ft) level will be mined and hauled up the existing ramp, while the shaft is being sunk and commissioned from the start of work to the 35th month. Once the shaft is commissioned both the 475 m (1,550 ft) and 920 m (3,010 ft) Levels will be developed from the shaft. A steady state production of 4,000 tpd development and stope production will begin during the 27th month. Stope mining will proceed upwards from the 290 m (950 ft), 475 m (1,550 ft) and 920 m (3,010 ft) levels, and downwards from the 920 m (3,010 ft) level towards the end of the operation.

It is estimated that 432 stopes would be mined over the mine life. This would generate an average of 4,000 tonnes per day ("tpd") composed of 3,421 stoping tonnes and 579 tonnes from mineralized development in drifts and slot raises. Over the 14 year mine life it is estimated that 8.13 Mt of Measured and Indicated Mineral Resource, and 11.51 Mt of Inferred Mineral Resource will be mined at overall diluted grades of 1.46% Cu, 0.16% Ni, 5.07 g/t Ag, 0.07 g/t Au, 0.05 g/t Pt and 0.14 g/t Pd.

FIGURE 1.1 THIERRY LONGITUDINAL PROJECTION



Source: *P&E (2021)*

1.6 PROCESS PLANT

A conventional process plant and flowsheet has been selected, including crushing and grinding to a 100 µm grind at an annual rate of 1.4 Mtpa (4,000 tpd). This will be followed by a two-product flotation circuit; a 30% copper concentrate containing low nickel and 50% of the PGM, and a nickel-copper concentrate containing 10% metal (8% Ni, 2% Cu) and low PGM. The projected concentrates would be shipped to an off-site smelter. Estimated process recoveries are:

- Copper concentrate: 30% Cu, 0.5% Ni, with 92% Cu recovery, 15% Ni recovery, and 50% Au, Ag and PGM recoveries.

- Nickel-copper concentrate: 8% Ni, 2% Cu, with 40% Ni recovery, 1% Cu recovery and 3% Au, Ag and PGM recoveries.

1.7 SITE INFRASTRUCTURE

The Thierry Project has the advantage of being close to Pickle Lake and Thunder Bay, Ontario and has a previous operating history. The regional labour force includes experienced equipment operators, mine workers and material and equipment suppliers.

An electrical power line and electrical substation will have to be constructed and tied into the local grid, approximately 8 km away. Overall site power consumption is estimated to be approximately 16 MW. The site facilities would also include an administration/safety/mine dry building, a shaft headframe and hoistroom/compressor building; a process plant with attached concentrate handling; a paste backfill plant and distribution system; a tailings/waste rock co-disposal basin and dam; site roads; surface parking areas; fuel, lubricant and oil storage facilities; surface explosive magazines; yard piping; a fire prevention and fighting system; potable water treatment plant and storage tanks; tailings water treatment plant and pond, and a water management pond building.

Tailings would be thickened to approximately 55% solids for use in mine paste backfill. The remainder would be disposed in the waste rock and tailings co-disposal facility (“CDF”) that is proposed to be located within the historic Thierry tailings management area and its sub-watershed catchment area. The CDF would be designed to provide a physically and chemically stable environment that would be suitable for the long term storage of waste rock and tailings. The CDF would also be utilized to manage any potentially acid generating / metal leaching components.

1.8 ENVIRONMENTAL IMPACT AND REHABILITATION

Braveheart has not yet commenced with formal discussions with regulatory authorities in regard to environmental assessment and permitting requirements necessary for production. The environmental assessment and permitting processes for mines in Ontario are well-established.

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.

The terms of reference for the environmental assessment of the proposed producing mine and processing plant have yet to be established. The Project would be developed, operated and closed in accordance with environmental and health and safety regulatory requirements, and would include a cooperative agreement with the Mishkeegogamang First Nation.

1.9 CAPITAL COSTS

The total capital cost of the Project is estimated at approximately $710.5 M. This is composed of $407.0 M in pre-production capital and $303.5 M in sustaining capital. An overall allowance for contingency of 5% has been included in these totals.

The pre-production period starts with site clearing and collaring of the shaft, and ends when the shaft is commissioned 35 months later. Pre-production capital costs include the cost of all surface buildings, process plant and related facilities and structures; mine and stope development on the 137 m (450 ft) to 290 m (950 ft) Levels; initial stope mining, initial support services, initial paste backfilling, initial underground haulage and initial G&A costs, shaft development, shaft commissioning and related facilities; initial ramp development to the 917 m (3,010 ft) Level; underground mining equipment; surface mobile equipment; electrical power supply infrastructure; underground infrastructure related to the shaft and 137 m (450 ft) to 290 m (950 ft) Levels, and part of the Project closure bond.

Commercial production commences after the three year pre-production period, in the first quarter of the fourth year (Production Yr1). Sustaining capital costs during this period include mine and stope development; ramp development near the bottom of the mine in Production Yr13 and Yr14; underground mining equipment; underground infrastructure; Project closure bond contributions; a salvage value in Production Yr17, and a contingency allowance.

1.10 OPERATING COSTS

Operating costs include the cost of operating labour, maintenance labour, electrical power, operating materials and supplies, reagents and fuel. A 5.4% allowance for contingency has been included. The yearly operating cost varies from $56.73 to $60.89 per tonne processed. A summary of the average operating cost estimates for the Thierry Project is provided in Table 1.3. Underground mining is estimated to average $38.64 per tonne processed over the life of mine ("LOM").

TABLE 1.3 SUMMARY OF AVERAGE OPERATING COST PER TONNE PROCESSED			
Description	**Subtotal ($/t)**	**Contingency ($/t)**	**Total ($/t)**
U/G Stope Mining	8.35	0.00	**8.35**
U/G Support Services	13.36	0.67	**14.02**
U/G Haulage	8.76	0.44	**9.20**
Paste Backfill	6.73	0.34	**7.06**
Process Plant	13.15	1.32	**14.47**
G&A	5.05	0.25	**5.30**
Total Operating Cost	**55.40**	**3.01**	**58.41**

1.11 ECONOMIC ANALYSIS

The metal prices used in this PEA are US$3.48/lb Cu, US$8.00/lb Ni, US$21/oz Ag, US$1,600/oz Au, US$1,100/oz Pt and US$1,250/oz Pd. The Project was evaluated on an after-tax cash flow basis which generates a net undiscounted cash flow estimated at $549.1 M. This results in an after-tax IRR of 18.9% and an after-tax NPV of $240.4 M when using a 6% discount rate. In the base case scenario, the Project has a payback period of 3.2 years from start of commercial production. The average life-of-mine cash cost is US$1.08/lb copper, net of nickel and by-product credits, at an average operating cost of $58.41/t processed. The average life-of-mine all-in sustaining cost ("AISC") is estimated at US$1.98/lb copper, net of nickel and by-product credits.

A summary of the results of the cash flow analysis is presented in Table 1.4.

TABLE 1.4 BASE CASE CASH FLOW ANALYSIS			
Description	**Discount Rate**	**Units**	**Value**
After-Tax CF NPV at:	0%	$M	549.1
	5%	$M	277.5
	6%	$M	240.4
	7%	$M	207.4
	8%	$M	177.9
	9%	$M	151.6
	10%	$M	128.1
Internal Rate of Return		%	18.9
Project Payback Period in Years		**Years**	**3.2**

A sensitivity analysis on NPV at a 6% discount rate was performed by adjusting key parameters up and down by 10% and 20%. The value of each parameter, at 80%, 90%, base, 110% and 120%, is presented in Table 1.5. The same type of analysis was done on IRR in Table 1.6.

TABLE 1.5
AFTER-TAX NPV AT 6% DISCOUNT RATE ($M)

Parameter	80%	90%	100%	110%	120%
OPEX	330.5	285.5	240.4	193.5	140.8
CAPEX	316.0	278.1	240.4	199.4	152.6
Cu Price	6.6	127.1	240.4	341.7	443.4
Ni Price	223.1	231.7	240.4	249.0	257.7
Cu Head Grade	31.6	139.1	240.4	331.6	423.4
Cu Recoveries in Cu Conc.	N/A	140.1	240.4	N/A	N/A

TABLE 1.6
AFTER-TAX IRR (%)

Parameter	80%	90%	100%	110%	120%
OPEX	23.6	21.3	18.9	16.5	13.7
CAPEX	27.4	22.6	18.9	15.8	12.9
Cu Price	6.4	13.1	18.9	24.0	28.9
Ni Price	18.0	18.5	18.9	19.4	19.8
Cu Head Grade	7.8	13.7	18.9	23.5	28.0
Cu Recoveries in Cu Conc.	N/A	13.8	18.9	N/A	N/A

The after-tax NPV's and IRR's are most sensitive to copper metal price followed by copper head grade, copper recovery in the copper concentrate, OPEX, CAPEX and nickel price.

1.12 CONCLUSIONS AND RECOMMENDATIONS

1.12.1 Conclusions

P&E concludes that the Thierry Project has economic potential as an underground mining and mineralized material processing operation producing a copper and a nickel-copper concentrate. This conclusion would need to be confirmed in a subsequent and more detailed Pre-Feasibility Study supported by additional metallurgical and tailings characterization tests.

P&E notes that this PEA is preliminary in nature, and its Mineral Resources include 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.

1.12.2 Recommendations

P&E recommends that Braveheart advance the Thierry Project with: 1) programs to expand and upgrade the Mineral Resources; and 2) extended and advanced technical studies, particularly in metallurgical, geotechnical and environmental matters with the intention to advance the Project to a Pre-Feasibility Study.

P&E recommends 9,000 m of diamond drilling be carried out on the upper portion of the Thierry Deposit from surface, and 150,000 m of drilling be carried out underground on the deep portion of the "Main Zone" to increase the overall Indicated Mineral Resource tonnage in the mine area. The Mineral Resource Estimate should be updated to incorporate the new data. A representative bulk sample for metallurgical testwork should be obtained when underground access is available.

With regard to K1-1 Mineral Resources, P&E recommends 11,000 m of drilling followed by an update of the Mineral Resource Estimate.

P&E recommends the proposed work program and budget presented in Table 1.7. The program is comprised of two phases. The results of Phase 1 would be assessed before commencing, revising or curtailing Phase 2. The cost for both phases combined is estimated at $44.4 M.

TABLE 1.7
RECOMMENDED WORK PROGRAM AND BUDGET

Program	Units	Unit Cost ($)	Budget ($M)
Phase 1			
Thierry Deposit			
Mine dewatering & rehabilitation			6.00
Surface drilling at Thierry	9,000 m	289	2.60
Underground drilling at Thierry	150,000 m	165	24.75
Underground development (3 m x 3 m) for drilling	1,200 m	3,000	3.60
Underground bulk sampling			0.10
Mineral Resource Estimate update			0.80
Subtotal			**37.85**
K1-1 Deposit			
Fill-in & twin drilling	11,000 m	289	3.10
Mineral Resource Estimate update			0.08
Subtotal			**3.18**
Phase 1 Total			**41.03**

TABLE 1.7
RECOMMENDED WORK PROGRAM AND BUDGET

Program	Units	Unit Cost ($)	Budget ($M)
Phase 2			
Metallurgical testwork			0.25
Geological & mineralogical studies			0.05
Environmental study work			0.25
Hydrogeology study			0.08
Archaeological study			0.05
Advance exploration closure report			0.01
Geotechnical and condemnation drilling	3,000 m	250	0.75
Housing and accommodation	2,700 days	150	0.40
First Nation consultation			0.04
Pre-feasibility study			1.50
Phase 2 Total			**3.38**
Total			**44.41**

Note: *Subject to HST*

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 TERMS OF REFERENCE

Braveheart Resources Inc. (“Braveheart” or the “Company”) has retained P&E Mining Consultants Inc. (“P&E”) to prepare a Technical Report and Preliminary Economic Assessment (“PEA”) on the past producing Thierry Mine (the “Thierry Project”, the “Project” or the “Property”). The Thierry Project is located 12 km west of Pickle Lake and 450 km north of Thunder Bay, Ontario. Updated Mineral Resources of the adjacent K1-1 Deposit are also reported.

This Technical Report was prepared pursuant to NI 43-101 regulations and guidelines by P&E Mining Consultants Inc., at the request of Mr. Ian Berzins, President and CEO of Braveheart, an Ontario registered company trading under the symbol of “BHT” on the TSX Venture Exchange with its corporate office at:

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Calgary, Alberta
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Telephone number 403-512-8202
Fax number 403-282-2876

This Technical Report has an effective date of January 21, 2021.

P&E understands that this Technical Report will support the public disclosure requirements of Braveheart and will be filed on SEDAR as required under NI 43-101 disclosure regulations.

Mr. Eugene Puritch P.Eng., a Qualified Person under the terms of NI 43-101, conducted site visits to the Property on December 15, 2005, May 5, 2010 and again on June 2, 2011. Data verification drill core sampling programs were conducted as part of the on-site reviews. Mr. Puritch has not returned to the site since that time; however, the Property condition has remained the same, and there has been no drilling on the Thierry Deposit since his last site visit.

2.2 SOURCES OF INFORMATION

This Technical Report is based, in part, on internal Company technical reports, and maps, published government reports, Company letters and memoranda, and public information as listed in the "References" section at the conclusion of this Technical Report. P&E has not conducted detailed land status evaluations, and has relied upon existing reports, public documents, and statements by previous owners regarding the Property tenure and status, third party agreements, and legal title to the Property. Additional details of the topic can be found in the public filings of Braveheart and are available on SEDAR at www.sedar.com.

The present Technical Report is prepared in accordance with the requirements of National Instrument 43-101 (“NI 43-101”) and in compliance with Form NI 43-101F1 of the Ontario

Securities Commission ("OSC") and the Canadian Securities Administrators ("CSA"). The Mineral Resource Estimate is prepared in compliance with the CIM Definitions (2014) and Best Practices (2019) on Mineral Resources and Mineral Reserves that are in force as of the effective date of this Technical Report.

Table 2.1 presents the authors and co-authors of each section of the Technical Report, who acting as Qualified Persons as defined by NI 43-101, take responsibility for those sections of the Technical Report as outlined in Section 28 Certificates of Author.

TABLE 2.1
REPORT AUTHORS AND CO-AUTHORS

Qualified Person	Employer	Sections of Technical Report
Mr. Andrew Bradfield, P.Eng.	P&E Mining Consultants Inc.	2, 3, 15, 19, 24 and Co-author 1, 25, 26
Mr. David Burga, P.Geo.	P&E Mining Consultants Inc.	4 to 11, 23 and Co-author 1, 25, 26
Mr. D. Grant Feasby, P.Eng.	P&E Mining Consultants Inc.	13, 17, 20 and Co-author 1, 21, 25, 26
Mr. Eugene Puritch, P.Eng.	P&E Mining Consultants Inc.	12, 14 and Co-author 1, 25, 26
Mr. James Pearson, P.Eng.	P&E Mining Consultants Inc.	16, 18, 22 and Co-author 1, 21, 25, 26

2.3 UNITS AND CURRENCY

Unless otherwise stated, all units used in this Technical Report are metric. Base metal assays (Ni, Cu) are reported in percent (%) while gold, silver and platinum group precious metal assay values (Au, Ag, Pt, Pd) are reported in grams of metal per tonne ("g Au/t") unless ounces per ton ("oz Au/T") are specifically stated. Canadian dollars ($) are used throughout this report unless the United States dollars (US$) are specifically stated otherwise. At the time of this report the rate of exchange between the US$ and the Canadian dollars is 1 US$ = 0.75 Canadian dollars.

2.4 GLOSSARY OF TERMS

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

Abbreviation	**Meaning**
Activation	Activation Laboratories in Thunder Bay, Ontario
Ag	silver
AGAT	AGAT Laboratories
AISC	all-in sustaining cost
ARD	acid rock drainage
Au	gold
the Authors	the authors of this Technical Report
Braveheart	Braveheart Resources Inc.
Cadillac	Cadillac Ventures Inc.
CDF	co-disposal facility, waste rock and tailings

Cdn	Canadian
cm	centimetre(s)
company, the	Braveheart Resources Inc.
CSA	Canadian Securities Administrators
Cu	copper
EA	Environmental Assessment
ft	foot or feet
g/t	grams per tonne of rock
ha	hectare(s)
ICPOES	inductively coupled plasma – optical emission spectrometry
IRR	internal rate of return
km	kilometre(s)
Lakefield	Lakefield Research, Ontario now SGS
LHD	load/haul/dump
LOM	life of mine
m	metre(s)
M	millions
MC	master composite
MECP	Ministry of Environment, Conservation and Parks
Ni	nickel
NI 43-101	National Instrument 43-101
Noranda	Noranda ore dressing laboratory of Noranda Mines, Quebec or Noranda Mines
NPV	net present value
NSR	net smelter return
OSC	Ontario Securities Commission
P&E	P&E Mining Consultants Inc.
Pd	palladium
PEA	Preliminary Economic Assessment
PGE	platinum group elements (herein collectively to mean Pt, Pd, Au, Ag)
PGM	platinum group metals
Project, the	Thierry Mine
Property, the	Thierry Mine
Pt	platinum
QA/QC or QC	Quality Assurance/Quality Control
Richview	Richview Resources Inc.
Salman	Salman Mineral Research, Montreal
SCC	Standards Council of Canada
SEDAR	System for Electronic Document Analysis and Retrieval
SRK	SRK Consulting
T	short ton(s)
t	metric tonne (s)
tpd	tonnes per day
Thierry Project	Thierry Mine
U of L	Louvain University, Belgium
UMEX	Union Miniere Exploration and Mining Inc. or Union Miniere Explorations and Mining Corporation
yr	year

3.0 RELIANCE ON OTHER EXPERTS

P&E has assumed that all of the information and technical documents listed in the References section of this Technical Report are accurate and complete in all material aspects. While the Authors carefully reviewed all the information they were presented with, they cannot guarantee its accuracy and completeness. P&E reserves the right, but will not be obligated to revise this Technical Report and conclusions, if additional information becomes known to us subsequent to the date of this Technical Report.

Copies of the licenses, permits and work contracts were reviewed, and an independent assessment of land title and tenure was performed in February 2021 using the Ministry of Energy, Northern Development and Mine's MLAS Map Viewer website. 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 upon the efficacy of the legal due diligence process conducted by the legal counsel to Braveheart.

A draft copy of this Technical Report has been reviewed for factual errors by Braveheart. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement 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 Technical Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 DESCRIPTION

The Thierry Project is comprised of 27 mining leases, totalling 4,670 hectares (11,538 acres) located in the Dona Lake, Ponsford Lake, Tarp Lake and Kapkichi Lake areas in the Patricia Mining District, northwestern Ontario (Figure 4.1). The mining leases are subject to a royalty interest payable to both UMEX and Kapkichi Nickel Mines Limited, although this interest does not include the actual Thierry Mine site located on CLM 195.

In 2018, the Ministry of the Environment switched to an updated online system and converted to an online registry system for mining claims based on a latitudinal and longitudinal grid as opposed to a ground or map staking system. Unpatented mining claims under the old system were converted into cell claims and boundary claims. In addition to the mining leases, the Thierry Project also had 5 unpatented mining claims. The legacy claims (formerly claims 4284794, 4284795, 4284796, 4284797, 4284798, 4284799, 4247646, 4247647, 4247648, 4284836, 4284837, 4284838) were split into 163 contiguous cell claims and 15 boundary claims totalling 3,258 hectares. A cell claim includes all the land in one or more cells on the provincial grid, while a boundary claim is held by separate companies within one cell. The mining claims are summarized in Table 4.1. The mining leases and patented claims are summarized in Table 4.2. The total combined Property area is 7,907 hectares.

The cell and boundary mining claims are presently in good standing but will require that approximately $65,600 worth of assessment work be filed, per year, starting on April 20, 2021 to keep them current.

Table 4.1
Mining Claims - Thierry Project

Legacy Claim No.	Cell/ Boundary Claim	Recorded Date	Due Date	Work Required	Ownership
PA 4247646	105436	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	105437	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	137330	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	137331	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	153241	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	169846	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	183313	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	183314	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	189350	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	189351	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	189352	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	189353	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	189354	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%

Table 4.1 Mining Claims - Thierry Project					
Legacy Claim No.	**Cell/ Boundary Claim**	**Recorded Date**	**Due Date**	**Work Required**	**Ownership**
	219183	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	226627	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	226628	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	238764	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	238765	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	255997	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	255998	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	285146	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	293251	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	321878	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	344803	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	344804	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
PA 4247647	106889	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	106890	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	128684	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	140670	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	146079	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	162633	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	175336	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	221924	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	221925	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	241451	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	242062	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	242063	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	249389	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	288491	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	288492	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	308761	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	308762	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	323882	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
PA 4247648	132657	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	132658	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	132659	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	168115	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	168116	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%

Table 4.1 Mining Claims - Thierry Project					
Legacy Claim No.	**Cell/ Boundary Claim**	**Recorded Date**	**Due Date**	**Work Required**	**Ownership**
	197371	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	205398	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	210262	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	234153	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	247648	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	247649	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	247650	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	263423	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	263424	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	271404	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	274103	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	283511	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	286792	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
	301234	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	317984	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	331362	2009-Apr-21	2021-Apr-20	$400	Cadillac 100%
	331363	2009-Apr-21	2021-Apr-20	$200	Cadillac 100%
PA 4284794	138930	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	144888	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	151199	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	151200	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	154678	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	169347	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	173506	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	187802	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	187803	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	203576	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	207337	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	207338	2017-Sep-07	2021-Sep-06	$200	Cadillac 100%
	209729	2017-Sep-07	2021-Sep-06	$200	Cadillac 100%
	236417	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	236418	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	239606	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	273317	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	322689	2017-Sep-07	2021-Sep-06	$200	Cadillac 100%

Table 4.1 Mining Claims - Thierry Project					
Legacy Claim No.	**Cell/ Boundary Claim**	**Recorded Date**	**Due Date**	**Work Required**	**Ownership**
PA 4284795	127398	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	138928	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	138931	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	144889	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	173491	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	173492	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	173507	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	190902	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	191430	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	203558	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	228767	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	239607	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	239608	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	240133	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	247586	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	294754	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
PA 4284796	114109	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	160733	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	160734	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	166063	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	213372	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	225543	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	232068	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	232069	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	232070	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	250626	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	250627	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	256334	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	261349	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	274417	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	291488	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
PA 4284797	136385	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	181561	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	189002	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	201183	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%

Table 4.1 Mining Claims - Thierry Project					
Legacy Claim No.	**Cell/ Boundary Claim**	**Recorded Date**	**Due Date**	**Work Required**	**Ownership**
	237666	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	267799	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	274418	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	291489	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	304441	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	311724	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
PA 4284798	113765	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	132590	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	132591	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	196813	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	263334	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	263335	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	263353	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	263354	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	283455	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	300659	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	330782	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
PA 4284799	113758	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	113759	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	132592	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	148700	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	168052	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	168053	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	196809	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	204823	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	234086	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	234087	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	270830	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	283451	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
	330778	2017-Sep-07	2021-Sep-06	$400	Cadillac 100%
PA 4284836	107351	2017-Sep-07	2021-Apr-20	$200	Cadillac 100%
	127445	2017-Sep-07	2021-Sep-20	$400	Cadillac 100%
	139495	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	144944	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	144945	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%

Table 4.1 Mining Claims - Thierry Project					
Legacy Claim No.	**Cell/ Boundary Claim**	**Recorded Date**	**Due Date**	**Work Required**	**Ownership**
	144946	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	173545	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	191470	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	191471	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	203623	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	240166	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	247650	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	286792	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	307497	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	307498	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	314200	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	335093	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
PA 4284837	213397	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	225481	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	258184	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	295539	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	315982	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
PA 4284838	126990	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	202108	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	202110	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	210166	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	222216	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	258184	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
	276174	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	295539	2017-Sep-07	2021-Apr-06	$200	Cadillac 100%
	324841	2017-Sep-07	2021-Apr-06	$400	Cadillac 100%
Total number of claims	**178**	**3,258 ha**		**$65,600**	

4.2 PERMITS AND ENVIRONMENTAL ISSUES

Both the Thierry and K1-1 deposits are considered to be advanced exploration stage projects as no development or pre-development programs are currently being conducted. The most recent work programs conducted on the Thierry Project were the 2010-2012 drilling programs by Cadillac Ventures on both the Thierry and K1-1 deposits. The Thierry Project was permitted for underground mine dewatering.

TABLE 4.2 SUMMARY OF MINING LEASES, THIERRY PROJECT				
Claim / Disposition ID	**Area (ha)**	**Tenure Type**	**Tenure Rights**	**Expiry Date**
CLM192	449	Lease	Mining & Surface	2038-Aug-30
CLM193	285	Lease	Mining & Surface	2038-Aug-30
CLM194	374	Lease	Mining & Surface	2038-Aug-30
CLM195	374	Lease	Mining & Surface	2038-Aug-30
CLM196	486	Lease	Mining & Surface	2038-Aug-30
CLM197	192	Lease	Mining & Surface	2038-Aug-30
CLM198	292	Lease	Mining & Surface	2038-Aug-30
CLM199	202	Lease	Mining & Surface	2038-Aug-30
CLM200	267	Lease	Mining & Surface	2038-Aug-30
CLM211	266	Lease	Mining & Surface	2038-Aug-30
CLM212	342	Lease	Mining & Surface	2021-Aug-30
CLM213	243	Lease	Mining & Surface	2021-Aug-30
CLM214	226	Lease	Mining & Surface	2021-Aug-30
CLM215	199	Lease	Mining & Surface	2021-Aug-30
CLM320	263	Lease	Mining	2028-Nov-30
PA15461	13	Lease	Mining & Surface	2033-Oct-30
PA15462	15	Lease	Mining & Surface	2033-Oct-30
PA15464	13	Lease	Mining	2033-Oct-30
PA17490	17	Lease	Mining	2033-Oct-30
PA20875	23	Lease	Mining & Surface	2033-Oct-30
PA20876	20	Lease	Mining	2033-Oct-30
PA20880	19	Lease	Mining	2033-Oct-30
PA20891	15	Lease	Mining	2033-Oct-30
PA20894	14	Lease	Mining	2033-Oct-30
PA20895	22	Lease	Mining & Surface	2033-Oct-30
PA20896	20	Lease	Mining	2033-Oct-30
PA21124	19	Lease	Mining	2033-Oct-30
Total: 27 Mining Leases	**4,670**			

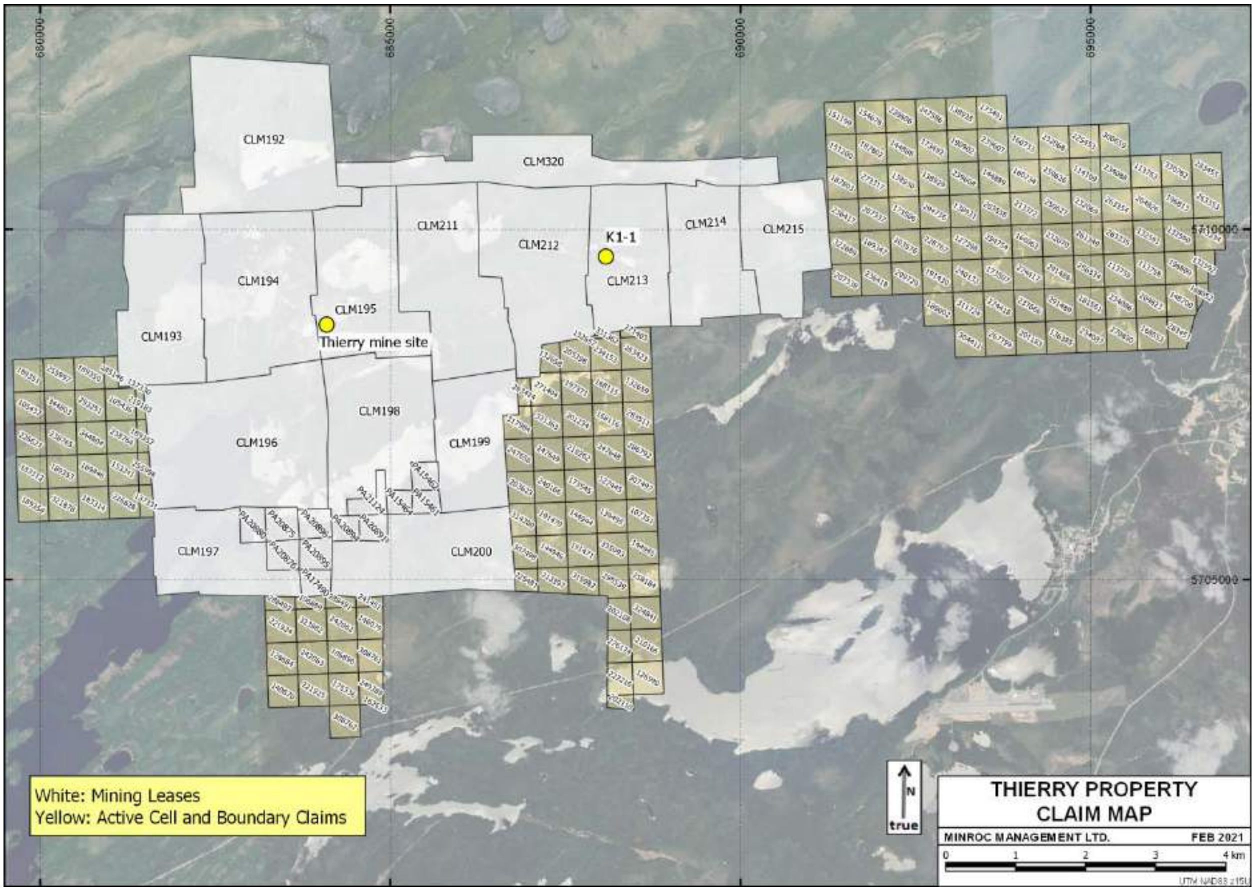
Source: *Ministry of Energy, Northern Development and Mine's MLAS Map Viewer website (February 2021)*

4.3 LOCATION

The Thierry Project is located 12 km west-northwest of the Town of Pickle Lake, which is situated 450 km northwest of Thunder Bay, Ontario, Canada (Figure 4.1).

The geographical centre of the Property lies at approximately 51°29'51.32" N Latitude and 90°20'52.45" W Longitude.

FIGURE 4.1 CLAIM MAP THIERRY PROJECT



Source: *Minroc (2021)*

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 LOCATION AND ACCESS

Pickle Lake is accessed by Provincial Highway No. 599 approximately 300 km north of the town of Ignace which is situated on the Trans-Canada Highway No. 17 (see Figure 5.1). The Canadian National Railway passes through the Town of Savant Lake, on Highway 599, approximately 170 km south-west of Pickle Lake.

The Thierry Project site is accessible by all-weather road from Pickle Lake (Figure 5.2).

5.2 CLIMATE AND PHYSIOGRAPHY

The climate is typical of northern areas within the Canadian Shield with long winters and short but warm to hot summers. Temperatures range from 30°C in the summer to -30°C in the winter. Mean annual rainfall is 48 cm and mean annual snowfall is 263 cm. Vegetation consists of black and white spruce and minor balsam poplar. Glacial overburden typically varies from 20 to 50 m thick.

The climate is suitable for exploration with diamond drilling, and other non-geological/geochemical work is able to be carried out at any time of the year without difficulty, except for limited access issues during the 4-week period of “Spring Break-up”, when most gravel roads are not suitable for driving, and load restrictions on the Highways are in place.

The Pickle Lake area is characterized by a gentle topography, with flat lying to gently rolling hills that are less than 35 m in height and numerous lakes in the intervening valleys. Elevations range from 360 m above sea level ("A.S.L.") to 390 m A.S.L. The Project is located within the Arctic Watershed and all local streams eventually drain to the Albany River.

Wildlife includes black bear, wolves, moose, rabbits, various migratory birds and various species of fish including lake trout, pike and pickerel.

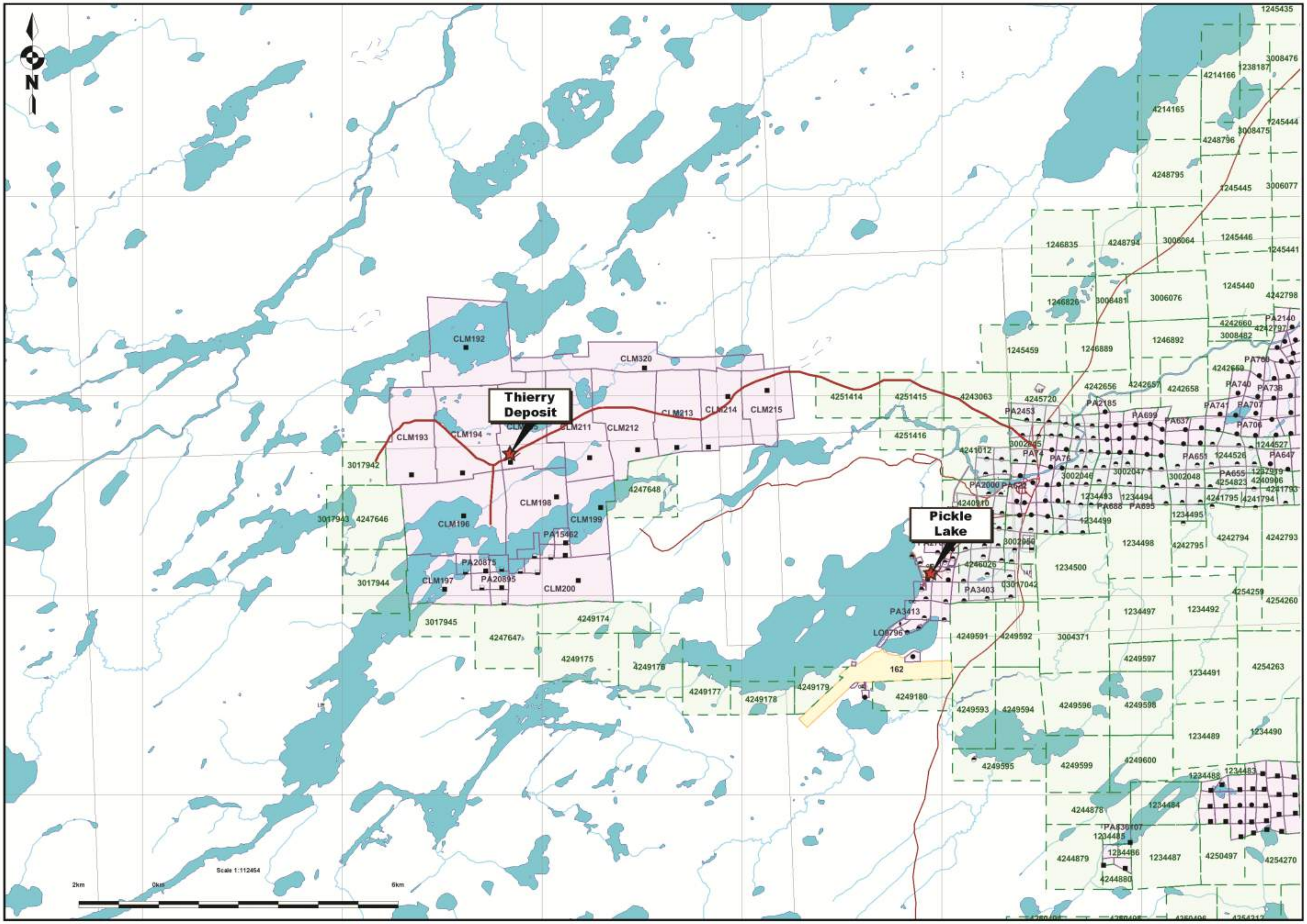
5.3 INFRASTRUCTURE

General mining related infrastructure in the Pickle Lake area is adequate to support a reasonable size mining operation, with an available workforce and amenities including power, paved roads, airport, housing, hospital and a school. Pickle Lake supported a mining and processing operation at the UMEX Thierry Deposit between 1976 and 1982, and also the Placer Dome Dona Lake mining operations between 1989 and 1993.

FIGURE 5.1 REGIONAL LOCATION MAP

Source: *Natural Resources Canada (2002)*

FIGURE 5.2 DETAILED LOCATION MAP



Source: *Ont. Gov. website (as of August 2011)*

6.0 HISTORY

This Technical Report section deals with material that is historical in nature and as such much of the terminology used may not be in keeping with modern or current usage. In particular, Mineral Resource/Reserve classifications and related terms may not be considered appropriate or acceptable under current rules and regulations. However, the context of the source material has been kept intact to ensure historical accuracy and the reader is cautioned not to rely on historical information as necessarily relevant or appropriate under current circumstances.

6.1 HISTORICAL OVERVIEW

Detailed historical accounts of the Pickle Lake region in general, and of the Thierry/K1-1 Project area specifically, are given in Curtis, L. (2001), Puritch et al., (2006) as well as in other reports as shown in the reference section 27.0 of this Technical Report. Detailed accounts of the Richview Diamond Drill Programs are given in Puritch et al., (2012). The reader is referred to these references for additional information. A summary of the pertinent historical events is provided in Table 6.1.

TABLE 6.1
THIERRY PROJECT – SUMMARY OF HISTORICAL EXPLORATION AND DEVELOPMENT ACTIVITIES

Year	Company	Exploration
1928-1929		Gold was discovered along the banks of the Kawinogans River. Technological advances, namely air transport, made the area accessible and mining began in 1929. Pickle Lake, being the closest lake to the two new gold mines, became the transportation center of the area (www.picklelake.ca).
1934-1951	Pickle Crow Gold Mines	The Pickle Crow gold mine operated from 1934 to 1951 producing 2,969,720 tonnes of ore grading 15.4 g/t Au (Klein and Day, 1994).
1935-1966	Central Patricia Gold Mines Limited	The Central Patricia gold mine, which operated from 1935 to 1966, produced 1,520,000 tonnes of ore at a grade of 12.5 g/t Au, (Klein and Day, 1994; Fyon et al., 1992).
1946-1950	Central Patricia Gold Mines Limited	Central Patricia Gold Mines Limited carried out drilling from 1946 to 1950 on several gabbro hosted copper-nickel prospects in the Kapkichi Lake area.
1946-1947	Albany River, Crowshore Patricia, and Norpic Gold Mines	Albany River Gold Mines was one of the mining companies active in the area at the time. Albany sunk a shaft and mined mineralization but did not go into production. In 1946, Pickle Crow took over the assets and liabilities of this company. Crowshore Patricia Gold Mines was situated approximately 3 miles east of Pickle Crow. This company sunk a shaft to 550 feet. It closed down in 1947. Norpic Gold Mines, situated north of Pickle Crow, did extensive drilling on their property. Dona Lake Gold Mines took an option on this

TABLE 6.1 THIERRY PROJECT – SUMMARY OF HISTORICAL EXPLORATION AND DEVELOPMENT ACTIVITIES		
Year	**Company**	**Exploration**
		property in 1979 and did more diamond drilling.
1956-1966	Kapkichi Nickel Mines Limited	Kapkichi Nickel Mines Limited continued work in this area with geophysical surveys and diamond drilling, between 1956 and 1958. Gold mining activity in the Pickle Lake Area ceased by 1966.
1969	UMEX Inc.	On January 1, 1969, UMEX signed a joint-venture agreement with Kapkichi Nickel Mines regarding 12 claims and a one mile surrounding zone (the "Kapkichi Property"). McPhar Geophysics of Toronto conducted ground geophysical (magnetometer and EM) surveys on the agreement area. The actual claims covering the Thierry Deposit were optioned by Union Miniere Explorations and Mining Corporation ("UMEX") from Kapkichi Nickel Mines in 1969. In 1969, UMEX conducted ground electromagnetic, magnetometer and geologic surveys on the Kapkichi property. Follow-up drilling led to the discovery of low-grade copper and nickel mineralization in mafic and ultramafic rocks underlying Kapkichi Lake. Additional drilling in the immediate area by UMEX outlined 4 principal areas with copper-nickel mineralization: the K1-1, K2-1, G and J anomalies.
1970	UMEX Inc	Preliminary metallurgical testwork on the Thierry mineralization indicated a much more favourable metallurgical response than the nearby K1-1, K1-2, K2-1, J and G deposits. In September 1970, the first hole drilled outside the Kapkichi Property area, intersected 20 feet of sulphides in biotite and chlorite schist containing 1.24% copper and 0.14% nickel. This was the discovery hole of the Thierry Deposit. Following the discovery drill hole, the Thierry Deposit was drilled off on a grid of cross sections 200 feet apart. 77 holes totalling 45,000 feet were drilled. The mineralization is now known to cover 4,000 feet in length and to have a vertical depth of at least 2,500 feet. The Deposit was still open at

Table 6.1 Thierry Project – Summary of Historical Exploration and Development Activities		
Year	**Company**	**Exploration**
		depth.
1971-1976	UMEX Inc.	UMEX awarded Kilborn Engineering a contract to prepare a preliminary Feasibility Study of the Thierry Deposit and to assume the project engineering. The decision to proceed with development of the Deposit was made in 1974.
1976-1982	UMEX Inc.	The Thierry Deposit initially produced from two open pits followed by underground operations. A total of 52,000 ft (15,850 m) of underground diamond drilling was completed to delineate mineralization. Historical UMEX records indicate production of approximately 5.8 million tons of ore with an average grade of 1.13% copper and 0.14% nickel, between October 1976 and April 1982 (Novak and Mlot, 2004). Initially only a copper concentrate was produced; by 1981 limited amount of nickel concentrate was produced. Late in the mine life precious metals and PGE's were also recovered: platinum 17,500 troy ounces; palladium 47,000 troy ounces; gold 17,000 troy ounces and silver 900,000 troy ounces. The average grades of PGE's reported by UMEX were 0.0046 oz/t gold, 0.004 oz/t platinum and 0.020 oz/t silver (Gurgurewicz-Luck, 1988). In 1981, UMEX began test mining of a large low-grade zone of disseminated copper-nickel mineralization at the K1-1 anomaly.
1987-1989	UMEX Inc.	UMEX staff geologist, D. Unger, implemented re-sampling and assaying of selected diamond drill holes. The PGE studies undertaken between 1987 and 1988, revealed that higher grade nickel-copper zones were coincident with anomalous PGE's. An airborne geophysical survey (EM/Resistivity / Magnetometer/ VLF) was flown by DIGHEM in 1988 over the Kibler Lake Stock.
1990-1995	Etruscan Resources Inc.	Etruscan purchased the Property in 1990 with a view to placing it into production.

TABLE 6.1
THIERRY PROJECT – SUMMARY OF HISTORICAL EXPLORATION AND DEVELOPMENT ACTIVITIES

Year	Company	Exploration
		In 1991, Watts, Griffis and McOuat Limited ("WGM") prepared an economic analysis for the reactivation of the Thierry operation.
2000-2003	PGM Ventures Inc.	In 2002, PGM Ventures completed 25 drill holes totalling 8,952 m to test mineralization at the Thierry Deposit (11 of 25 holes) and at other targets on the Property. JVX completed a Time-Domain EM and Mag Survey over the Property.
2004-2005	Richview Resources Inc.	Richview conducted a multi-phased drill program to explore the Thierry Deposit and other target areas of the Thierry Project during the period Oct 2004 to March 2005.
2006	Richview Resources Inc.	An NI 43-101 Mineral Resource Estimate with an effective date of February 1, 2006, was undertaken by P&E Mining Consultants Inc., and Billiken Management Services Inc. The Mineral Resource consisted of 4,623,000 tonnes of Measured & Indicated Mineral Resource at a grade of 1.81% Cu, 0.20% Ni, with 4,366,000 tonnes of Inferred Mineral Resource at a grade of 1.71% Cu and 0.18% Ni.
2007	Richview Resources Inc.	Richview commenced its summer validation and exploration program on May 9, 2007. A 45,900 ft (14,000 m) drilling program was competed. Surface drilling around the K1-1 open pit area to confirm and validate the historic drilling was completed. A compilation of all mine data was conducted. A 3 km corridor of unexplored ground between the Thierry Mine and the K1-1 Deposit was cleared of overburden.
2008	Richview Resources Inc.	Richview committed to an ongoing relationship with the First Nations with respect to the company's exploration activities and the Thierry Deposit. A summer work program including excavation, geological mapping, prospecting and geochemical sampling was completed by October 2008. Richview completed its 45,900 ft (14,000 m) deep drill hole program. A Mobile Metal Ion ("MMI") geochemical survey of the Thierry Project was conducted.
2010	Cadillac Ventures Inc.	The amalgamation of Cadillac Ventures Inc. and Richview Resources Inc. ("Richview"), pursuant to a three-cornered agreement became effective on Jan 15, 2010. Cadillac

TABLE 6.1 THIERRY PROJECT – SUMMARY OF HISTORICAL EXPLORATION AND DEVELOPMENT ACTIVITIES		
Year	**Company**	**Exploration**
		assumed 100% control of the Thierry Project and conducted drill exploration programs during 2010 to 2012.
2020	Braveheart Resources Inc.	Braveheart acquired the Thierry Project from Cadillac Ventures Inc. A 2% net smelter return ("NSR") royalty was retained by Cadillac.
2021	Braveheart Resources Inc.	Braveheart purchases the 2% NSR royalty from Cadillac for 2.5 million common shares of Braveheart.

6.2 HISTORICAL RESOURCE ESTIMATES

A Qualified Person has not done sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves, and Braveheart is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

Table 6.2 summarizes the historical Mineral Resource estimates prepared for the Thierry Deposit (Novak and Mlot 2004):

TABLE 6.2 HISTORICAL MINERAL RESOURCE/RESERVE ESTIMATES – THIERRY DEPOSIT					
Company	**Date**	**Mineral Reserves (t)**	**Cu (%)**	**Ni (%)**	**Category**
UMEX	1974	13,500,000	1.62	0.18	Mining start-up in-situ Mineral Reserve estimate
UMEX	1989	7,000,000	1.88	0.23	Drill indicated in-situ Mineral Reserve to 2,500 ft
WGM	1991	2,700,000	1.65	--	Diluted Measured Mineral Resource to 1,800 ft
WGM	1991	3,000,000	1.78	0.25	Probable Mineral Reserves to 1,800 ft

In addition to the K2-1 (Thierry) Deposit, UMEX in its exploration program on the Property identified a number of other mineralized zones: G, J, and K1-1. Limited exploration drilling was conducted allowing UMEX to reported in-situ Mineral Resources for these deposits as shown in Table 6.3.

TABLE 6.3 HISTORICAL MINERAL RESOURCES FOR K1-1 AND J & G ZONES					
Drill Indicated, Undiluted, In-Situ Mineral Reserves					
Deposit	**Historical Mineral Resource Parameters**	**Tons**	**Cu (%)**	**Ni (%)**	**Remarks**
K1-1	Surface to level 1,000 ft	75,000,000	0.38	0.11	UMEX 1973, 1981
J&G Zones	Surface to level 600 ft	44,700,000	0.40	0.11	UMEX 1974, 1981
J&G Zones	Surface to level 1,000 ft	55,000,000	0.40	0.11	UMEX 1974, 1981

Drill hole data from the 2004-2005 drill programs, along with results from all previous drilling programs, were incorporated into an initial NI 43-101 Mineral Resource Estimate completed by P&E in 2006, as shown in Table 6.4.

TABLE 6.4 2006 P&E MINERAL RESOURCE ESTIMATE @ 1.3% CU CUT-OFF GRADE (1-2)					
Classification	**Tons**	**Cu (%)**	**Ni (%)**	**Cu (Mlb)**	**Ni (Mlb)**
Measured	17,000	1.71	0.25	0.6	0.1
Indicated	4,606,000	1.81	0.20	166.7	18.4
Measured & Indicated	**4,623,000**	**1.81**	**0.20**	**167.3**	**18.5**
Inferred	4,366,000	1.71	0.18	149.3	15.7

Notes:

1) *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2) *The quantity and grade reported in this Inferred Mineral Resource estimation are conceptual in nature and there has been insufficient exploration to define an Indicated Mineral Resource on the Property and it is uncertain if further exploration will result in discovery of an Indicated or Measured Mineral Resource on the Property.*

A 2010 updated P&E Mineral Resource Estimate was prepared using additional data generated by drilling 21 holes in 2007 and 2008 by Richview, as well as the drilling used in previous NI 43-101 Mineral Resource Estimates.

The 2010 P&E updated Mineral Resource Estimate for the Thierry Deposit, as shown in Table 6.5, consists of an Indicated Mineral Resource of 6,228,000 tonnes containing 1.92% Cu and 0.2% Ni and an Inferred Mineral Resource of 8,379,000 tonnes containing 1.79% Cu and 0.16% Ni using an NSR cut-off value of $46/t.

TABLE 6.5 2010 P&E UPDATED MINERAL RESOURCE ESTIMATE @ $46/TONNE NSR CUT-OFF (1-3)							
Classification	**Tonnes**	**Cu (%)**	**Ni (%)**	**Au (g/t)**	**Pt (g/t)**	**Pd (g/t)**	**Ag (g/t)**
Measured	2,221,000	1.90	0.21	0.13	0.13	0.41	7.7
Indicated	4,007,000	1.93	0.20	0.14	0.14	0.41	7.1
Measured & Indicated	**6,228,000**	**1.92**	**0.20**	**0.14**	**0.14**	**0.41**	**7.3**
Inferred	8,379,000	1.79	0.16	0.18	0.12	0.35	9.6

Notes:

1) *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2) *The quantity and grade of reported Inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral 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 classification.*
3) *The 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 and adopted by CIM Council December 11, 2005.*

In 2011, P&E prepared an initial Mineral Resource Estimate on the K1-1 and updated the Mineral Resource Estimate on the past-producing Thierry Mine. This information is presented in Table 6.6.

TABLE 6.6 OCTOBER 2011 P&E UPDATED MINERAL RESOURCE ESTIMATE FOR K1-1 @ $15/TONNE NSR CUT-OFF (1-8)							
NSR Cut-off	**Tonnes (M)**	**Cu (%)**	**Ni (%)**	**Au (g/t)**	**Pt (g/t)**	**Pd (g/t)**	**Ag (g/t)**
$15/tonne	19.897	0.42	0.10	0.03	0.05	0.15	2.0

Notes:

1) *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2) *The quantity and grade of reported Inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral 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 classification.*
3) *The 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 and adopted by CIM Council.*
4) *The July 31, 2011 two-year trailing average US metal prices used in this estimate were $3.57/lb Cu, $9.98/lb Ni, $23.87/oz Ag, $1,258/oz Au, $1,605/oz Pt and $557/oz Pd. The US Exchange rate was 0.98.*
5) *Overall payable metal (process recovery x smelter payable) in the NSR calculation were 84% Cu, 13% Ni and 37% for Ag, Au, Pt & Pd.*
6) *Mineral Resources were determined within a Whittle pit shell with 50 degree slopes utilizing mining costs of $2.00/t for mineralized material and waste rock, and $1.50/t for overburden.*

7) *Costs used to determine the C$15/t NSR Mineral Resource cut-off value were processing at $12/t and G&A $3.00/t.*
8) *The K1-1 Mineral Resource Estimate was undertaken by Antoine Yassa, P.Geo., and Eugene Puritch, P.Eng., of P&E Mining Consultants Inc.*

P&E prepared a PEA on the Thierry Project that included Mineral Resource Estimates on the Thierry and K1-1 deposits, with an effective date of May 15, 2012. This information is presented in Table 6.7 and 6.8.

TABLE 6.7
MAY 2012 P&E THIERRY MINERAL RESOURCE ESTIMATE @ $41/TONNE NSR CUT-OFF (1-6)

Classification	Tonnes	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)
Measured	3,233,000	1.65	0.19	0.03	0.03	0.09	4.6
Indicated	5,582,000	1.66	0.19	0.05	0.05	0.14	3.8
Measured & Indicated	**8,815,000**	**1.66**	**0.19**	**0.05**	**0.04**	**0.13**	**4.0**
Inferred	14,922,000	1.64	0.16	0.10	0.07	0.21	6.4

Notes:
1) *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2) *The quantity and grade of reported Inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral 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 classification.*
3) *The 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 and adopted by CIM Council.*
4) *The January 31, 2012 two-year trailing average US metal prices used in this estimate were $3.72/lb Cu, $10.15/lb Ni, $28.18/oz Ag, $1,419/oz Au, $1,663/oz Pt and $639/oz Pd. The US Exchange rate was 0.99.*
5) *Overall payable metal (process recovery x smelter payable) in the NSR calculation were 84% Cu, 13% Ni and 37% for Ag, Au, Pt & Pd.*
6) *Costs used to determine the $41/tonne NSR cut-off value are as follows: mining $30/tonne, processing $9.50/tonne and G&A $1.50/tonne.*

TABLE 6.8
MAY 2012 P&E UPDATED K1-1 INFERRED MINERAL RESOURCE ESTIMATE @ $11/TONNE NSR CUT-OFF (1-8)

NSR Cut-off	Tonnes (M)	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)
$11/tonne	53.614	0.38	0.10	0.03	0.05	0.14	1.83

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

2) *The quantity and grade of reported Inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral 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 classification.*
3) *The 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 and adopted by CIM Council.*
4) *The January 31, 2012 two year trailing average US metal prices used in this estimate were $3.72/lb Cu, $10.15/lb Ni, $28.18/oz Ag, $1,419/oz Au, $1,663/oz Pt and $639/oz Pd. The Canadian$/US$ Exchange rate was 0.99.*
5) *Overall payable metal (process recovery x smelter payable) in the NSR calculation were 84% Cu, 13% Ni and 37% for Ag, Au, Pt & Pd.*
6) *Mineral Resources were determined within a Whittle pit shell with 50 degree slopes utilizing mining costs of C$1.85/tonne for mineralized material and waste rock, and $1.65/tonne for overburden.*
7) *Costs used to determine the $11/tonne NSR Mineral Resource cut-off value were processing at $9.50/tonne and G&A $1.50/tonne.*
8) *The K1-1 Mineral Resource Estimate was prepared by Eugene Puritch, P.Eng. of P&E Mining Consultants Inc.*

The Mineral Resource Estimates noted in this section are superseded by the Updated Mineral Resource Estimates presented in Section 14 of this Technical Report.

6.3 HISTORICAL METALLURGICAL TESTING

6.3.1 Lakefield Research Metallurgical Testing

In April 1973, Lakefield Research undertook metallurgical testing on the Thierry mineralization which involved several pilot plant test runs to produce a bulk Copper-Nickel concentrate. The following results were obtained:

- Copper in concentrate: from 15% to 24% Cu at recoveries from 83% to 94%.
- Nickel in concentrate: from 1.4% to 2.5% Ni at recoveries from 29% to 68%.

The conclusions from the Lakefield metallurgical tests and from market conditions enabled UMEX to design a process plant for Thierry that produced a simple Copper concentrate fulfilling the conditions required by the Noranda toll smelter.

In 1980, Lakefield produced a copper concentrate from a Thierry Deposit and K1-1 sample (in a 1:1 ratio) and produced a copper concentrate grading 24.1% Cu at 86.7% recovery. High nickel in copper concentrate was observed (1.4% Ni).

Also in 1980, UMEX conducted tests on copper rougher concentrates with the objective of producing a separate, marketable nickel concentrate. A nickel concentrate, grading 6 to 14%, was achieved with a very low copper and some cobalt content. Based on this data, the possibility of producing a separate nickel concentrate from copper rougher concentrates is possible, however, it will be at very low overall nickel metallurgical recoveries (Xstrata, 2008).

In late 2005, Lakefield undertook metallurgical testing on three composites of drill core samples from the 2004 drill program. Initial rougher tests were conducted to analyze grinding time and size. Additional testwork with cleaner and scavenger stages will be required to arrive at a conclusive result for study purposes (Xstrata, 2008).

6.3.2 Salman Mineral Research Metallurgy

Salman Mineral Research Ltd. ("Salman") started metallurgical tests on Thierry mineralized samples in 1973. The initial work was performed in order to optimize the copper concentrate grade and recovery. Salman's metallurgical tests were quite conclusive and confirmed Lakefield's results regarding production of a copper concentrate (high grade, high recovery), however, similar to Lakefield his tests were deficient regarding the production of a separate nickel concentrate.

Historical testwork for the J and G deposits conducted at the University of Louvain, ("U of L") SGS Lakefield and McGill University showed that it was practically impossible to produce high grade, high recovery concentrates from the J and G deposits. Low grade – high recovery and low recovery – high-grade concentrates were produced; however, the J and G deposits can be considered consistently poor.

6.3.3 Noranda Mineralogical Processing Studies

The Noranda mineral dressing laboratory of Noranda Mines ("Noranda"), Quebec, carried out flotation tests and a mineralogical study on a 500 lb Thierry mineralized rock sample in February 1974. Results comparable to those obtained by SGS Lakefield were produced from the flotation of a copper concentrate.

6.3.4 Xstrata Process Support Metallurgical Review 2008

Xstrata Process Support conducted a review of existing mineral processing and metallurgical testwork on the Thierry Deposit. The objective of their report was to provide an assessment of the quality of work done to date, and mineralogical and metallurgical flags that may exist. Recommendations concerning protocols for future work were also included and are outlined in this subsection.

The documents reviewed covered both the Thierry Project and Kapkichi Lake area for the period between 1970 and 2007. In total, ten reports on the Thierry Project and surrounding area as noted below were reviewed.

- Anderson, S., 2007: Observations Pertaining to the Structural Geology of the Thierry Cu-Ni (PGE) Deposit.
- Lascelles, D., Fleming, C., 2006: The Recovery of Copper, Nickel and PGM from the Thierry Deposit, SGS Lakefield Research Limited.
- Puritch, E., Ewert, W., Armstrong, T., 2006: Technical Report and Resource Estimate on the Thierry Cu-Ni –PGE Mine Property, Pickle Lake Area, Patricia Mining District North-western Ontario, Canada., NI 43-101, P&E Mining Consultants Inc.

- Goodman, S., 2004: Structural Architecture of the Thierry Mine Cu-Ni-PGM Deposit, Pickle Lake, Ontario, SRK Consulting.

- Curtis, L., 2001: Thierry Mine Property Thunder Bay Ontario, Curtis & Associates Inc.

- Patterson, G., Watkinson, D., 1984: The Geology of the Thierry Cu-Ni Mine, North-western Ontario., Canadian Mineralogist, Vol. 22 pp. 3-11.

- UMEX Inc., 1982: The Kapkichi Deposits, Internal report dated Apr. 1982.

- MacLellan, J., 1980: UMEX – Internal Thierry Mine Report re: Nickel Circuit, dated Dec. 1980.

- Patterson, G. C., 1980: The Geology of the Kapkichi Lake Ultramafic-Mafic Bodies and Related Cu-Ni Mineralization Pickle Lake, Ontario, Ph.D. thesis, Carlton University.

- UMEX Inc., 1970: Petrography of the Kapkichi Lake Copper-Nickel Deposits and Associated Rocks, Pickle Lake Area, North-West Ontario, Internal UMEX report dated Dec. 1970.

Based on the conclusions resulting from their review Xstrata Process Support recommended the following:

- The Thierry Deposit should be considered primarily as a copper deposit with credits obtained for minor Pt, Pd, Ag and Au content in copper concentrate. Testwork has shown that nickel concentrate production can be challenging, with both grades and recoveries being poor.

- It is recommended that an economic evaluation and Mineral Resource Estimate on the Property is completed including payable metals Cu, Pt, Pd, Au and Ag but without nickel. A decision to proceed with more testwork should be made once it is clear whether the economics of a Cu-Ag-Au-PGE deposit are sufficient to support the project moving forward.

- If the economics are still favourable for a copper PGM deposit, only then would XPS recommend a full mineralized rock characterization study using spatially representative and fresh drill core samples. This could involve QEMSCAN and microprobe analysis to quantify the minerals present, payable metal deportments and association of the PGMs. This would be valuable information that would assist the mineral processing team to develop a sound flowsheet that maximizes the profitability of the Thierry Deposit.

- For all future testwork, it is recommended that a more rigorous sampling procedure be implemented, where the composite to be tested is representative of the population to be investigated in terms of average grade, grade distribution, lithology and space.

- For all future testwork, effort should be made to prevent oxidation of drill core. The amount of oxidation in old versus new drill core should be assessed to appropriately design a protocol which can limit oxidation in drill core from the current program. Drill core which oxidizes easily may require special handling protocols (e.g. frozen or nitrogen purge).

6.4 HISTORICAL PEA

P&E prepared a PEA on the Thierry and K1-1 Cu-Ni-PGE deposits in June 2012. The Mineral Resource Estimates had an effective date of May 15, 2012 and are noted in Table 6.7 and 6.8 above. The PEA was based on underground mining the Thierry Deposit and open pit mining the K1-1 Deposit. The PEA concluded that on a pre-tax cash flow basis, a net undiscounted cash flow of $881.1 M was estimated. This resulted in a pre-tax Internal Rate of Return (“IRR”) of 19.0% and a pre-tax Net Present Value (“NPV”) of $379.9 M when using a 6% discount rate. The Project had a payback period of 4 years from start of commercial production. The average life-of-mine cash cost was estimated at CDN$1.76/lb copper, net of nickel and by-product credits, at an average operating cost of $27.48 per tonne processed.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Thierry and K1-1 deposits occur along the northwest margin of the Pickle Lake Metavolcanic-Metasedimentary Belt that forms part of the Uchi Lake Greenstone Belt (Figure 7.1). The Thierry Property is underlain by a 1.5 km wide belt of metavolcanics that widens to the southwest. This sequence is intruded by the Pickle Lake and the Tarp Lake granitic plutons. The rocks have been structurally modified by four distinct tectonic events, the most significant being a late cataclastic episode that produced a major shear zone (mylonite) in the vicinity of the Thierry Deposit (Figure 7.2).

7.2 DEPOSIT GEOLOGY

The mine sequence is interpreted as consisting of metamorphosed gabbro and ultrabasic rocks hosted by sequences of massive to pillowed mafic volcanic rocks (Patterson, 1980). The intrusions have been described by various other authors as amphibolite, peridotite and metagabbro. The temperature of metamorphism was determined from garnet-biotite, calcite dolomite and magnetite-ilmenite geothermometers as being approximately 600°C.

The pillowed flows around the Thierry Deposit open pits have been highly deformed and flattened. Relatively undeformed flat lying amphibolitic pillows are found along the southeast shore of Kapkichi Lake near the Kapkichi Lake gold showing. The metavolcanics are moderately to strongly foliated and epidotized.

Interlayered with the mafic rocks of the Thierry Deposit sequence is a chert magnetite iron formation of variable thickness that can be traced for at least a kilometre west and southwest of the Thierry Mine where it appears to become truncated by a northwest trending sinistral fault. Mullen (1988) first observed chert-magnetite zones in drill core while re-logging old mine holes for UMEX's platinum program in 1988 and Gurgurewicz-Luck (1988) noted chert-magnetite iron formation while re-logging core for a 1987 study. According to Mullen (1988), the iron formation horizon may have acted as a focus for the main shearing event that preferentially allowed the intrusion of the mineralized mafic-ultramafic bodies at Thierry.

According to Mullen (1988), the siliceous metasediments and cherty iron formation observed in drill holes west of the mine are probably not the strike extension of the main iron formation horizon west of the West Pit, but represent another sedimentary horizon. Iron formations cored under Kapkichi Lake and further south in drill hole K-92 are probably the on strike extension of the "Mine Iron Formation". Magnetite-rich mafic intrusions similar to mafic-ultramafic bodies at Thierry underlie Kapkichi Lake.

FIGURE 7.1 REGIONAL GEOLOGY

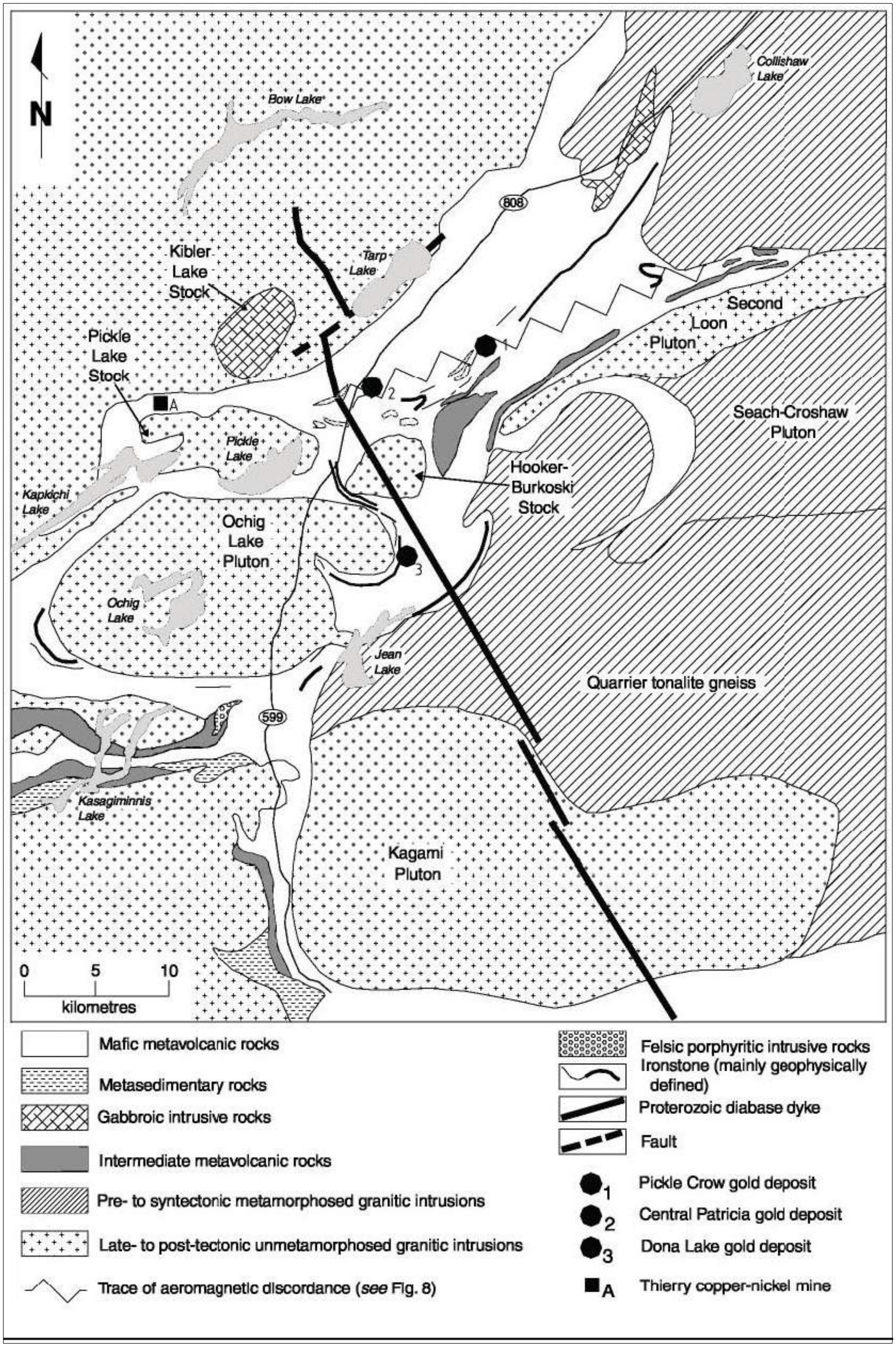
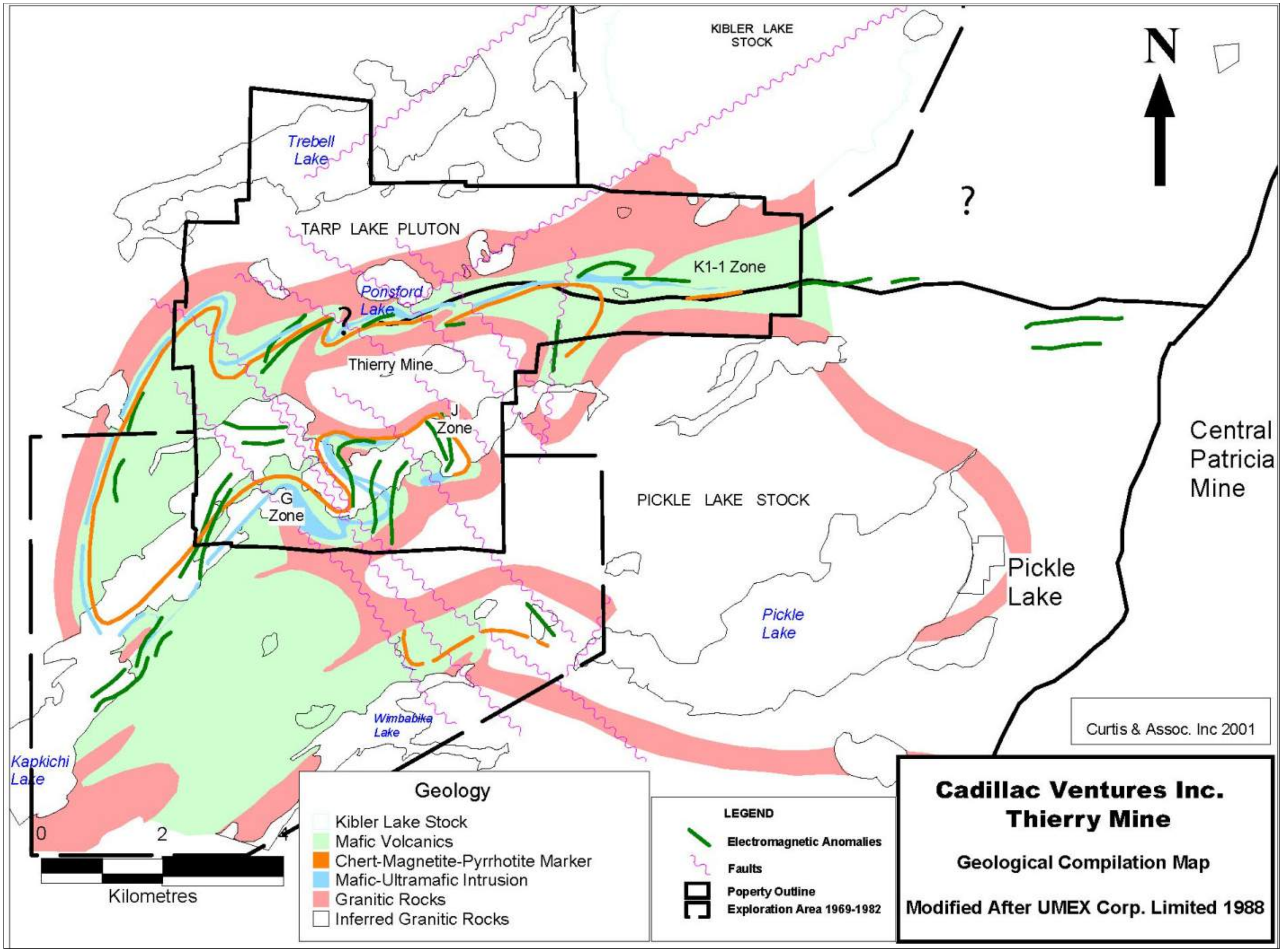


FIGURE 7.2 DETAILED PROPERTY GEOLOGY



Source: *Cadillac (2012)*

7.3 MINERALIZATION

Mineralization at the main Thierry and adjacent K1-1 deposits, is more or less coincident with what is best characterized as a chlorite-biotite-hornblende altered mylonitic shear zone (the "CBS shear zone"). The shear zone extends across the ultramafic intrusive along a strike length of approximately one kilometre and a width up to 50 m. Within the shear zone mineralization is hosted by highly schistose rocks containing stringer sulphides to less schistose ultramafic rocks containing massive stringers or veins and disseminated sulphides. Primary sulphides, listed in approximate order of decreasing abundance are pyrrhotite, chalcopyrite, pyrite and pentlandite. Cubanite, bornite, magnetite and minor ilmenite have also been identified. Violarite and mackinawite have developed from alteration of pentlandite.

Outside of the main mineralized zone, chalcopyrite and bornite occur as stringers as well as finely dissemination sulphides. Bornite is commonly associated with carbonate and quartz veins. Oxidized mineralizations are reported to contain violarite, millerite and bornite.

Copper-nickel-PGE mineralization at the Thierry and K1-1 deposits is hosted within a highly deformed and altered ultramafic sequence. Copper-nickel-PGE mineralization consists of:

- Sulphide matrix breccia;
- Blebs and small stringers, occasionally net textured sulphides; and
- Disseminated sulphides.

The sulphide mineral assemblage consists of chalcopyrite, pyrrhotite, pentlandite and pyrite.

7.4 COPPER-NICKEL MINERALIZATION

Four principal types of sulphide mineralization are recognized at the Thierry Deposit (Patterson and Watkinson, 1984b) with Patterson (1980) noting a fifth:

- Breccia Mineralization: 40% of all mineralized rock and composed of 20-30% sulphide, consisting of rounded to angular fragments of gangue in a matrix of chalcopyrite, pyrrhotite, pyrite and pentlandite. Breccia mineralized rock grades into CBS mineralized rock.

- Chlorite-Biotite Schist Mineralization (mylonitic mineralization): 56% of all mineralized rock (CBS), containing 5-20% sulphide as stringers of chalcopyrite, pyrrhotite, pentlandite and pyrite; the stringers parallel foliation and where gradational with breccia mineralized rock, the breccia fragments are flattened and elongated.

- Bornite Mineralization: 2% of all mineralized rock, containing 1-5% sulphide as stringers and disseminations of chalcopyrite and bornite in carbonate veins associated with blocks of amphibolite schist in the main shear zone.

- Primary Disseminated Sulphide Mineralization: 1% of all mineralized rock, occurring as blocks of chalcopyrite (with exsolution of bornite or cubanite) plus pyrrhotite and pentlandite between remnants of olivine.

- Oxidized Mineralization: 1% of all mineralization comprised of several varieties, characterized by violarite, millerite, bornite etc.

The mylonite and breccia mineralization has a copper-to-nickel ratio of 8:1, compared to a 2:1 ratio in the disseminated sulphides. In addition, the chalcopyrite-pyrrhotite ratio is approximately 1:1 in the mylonite mineralization and 1:10 in the disseminated sulphides (Patterson, 1980).

7.5 PLATINUM-GROUP ELEMENTS AND SILVER MINERALIZATION

Precious metal minerals have been found in the Thierry Deposit in two distinct associations:

- In the breccia mineralization, the precious metal minerals merenskyite, moncheite, stutzite and an unnamed mineral Ag_3BiTe_3 occur with chalcopyrite, pyrrhotite. pentlandite, pyrite and violarite.

- In the bornite mineralized rock, the precious metal minerals, native silver, acanthite, stutzite and merenskyite are associated with chalcopyrite, bornite and copper bismuth sulphosalt (wittichenite and emplectite).

The strongest positive correlation of metals is between silver and copper. There is a corresponding negative correlation between silver and nickel at values of nickel greater than 0.5%.

A plot of Pt/(Pt + Pd) versus Cu/(Cu + Ni) shows average head grades of the Thierry Deposit to be enriched in copper and somewhat in platinum relative to other similar deposits (Naldrett and Cabri, 1976). From a PGE perspective the Thierry mineralization falls into two groups, both of which fall well off a characteristic trend line defined by Naldrett and Cabri, (1976) for typical PGE mineralization. The first group is pyrrhotite-rich and correspondingly has a high Ni content. This group is platinum poor compared to the second Cu-rich, chalcopyrite rich fraction which has a high platinum content.

Mineralization at the Thierry Deposit underwent intense modification after their initial deposition as magmatic sulphides. Dynamic metamorphism has mobilized much of the breccia and mylonite mineralization. The occurrence of mylonite fragments in the breccia mineralization along with the localization of breccia mineralization along faults related to the main shear emphasizes this relationship. It is important to note the occurrence of merenskyite in carbonate veins (bornite mineralization) which cut across metamorphic foliation in these amphibolite blocks is evidence that PGE minerals were mobile during dynamic metamorphism.

SRK examinations of UMEX plans (Goodman, 2004) and sections indicated that there is strong structural control on the geometry of the zones of mineralization. As a result, mineralization is believed to occur as pinching and swelling structures. Thicker and higher grade structures are expected to be associated with steeply dipping and/or right stepping portions of shear zone segment.

The ultramafic-mafic rocks that host the mineralization are highly deformed along shear zones and are best defined texturally as blastomylonite rocks. A structural corridor, referred to as the "CBS Shear Zone", is defined by zones of mylonite host rocks that enclose virtually all of the defined sulphide mineralization. The CBS Shear Zone has been traced over 6 km, and extends from east of the K1-1 Pit to approximately 1 km west of the West Pit. The CBS Shear Zone is truncated by a NW-SE sinistral fault, 1 km west of the West Pit. The mineralized rock horizon may continue to the west of this fault as evidenced by the presence of a chert-sulphide iron formation under Kapkichi Lake. The CBS Shear Zone is narrow (2-30 m wide) and is occasionally offset by east-west and north-northeast to northeast-trending fault zones. The shear structure dips 48°-55° to the north-northwest. The angle of dip increases to 70° in the eastern part of the mineralized rock. Felsic intrusives and gabbro are reported to occur as lens shaped and narrow dykes from 5 cm to 3 m thick.

8.0 DEPOSIT TYPES

Early investigations of the Thierry Deposit by workers such as Bowdidge (1970), Patterson (1980), and Patterson and Watkinson (1983, 1984) concluded that the mineralization at the Thierry Deposit had undergone intense modification after their initial deposition as magmatic sulphides. This observation also applies to the K1-1 area.

Bowdidge (1970) suggested that the textural evidence and the occurrence of sulphide inclusions in olivine supported the argument that the mineralization was originally an intercumulus sulphide phase. The excess Cu to Ni (3 to 1 ratio) was said by Bowdidge (1970) to occur as a result of depletion of Ni due to removal of olivine and pyroxene. There is a strong suggestion, especially at the main Thierry Deposit, that re-mobilization of the original sulphide material, is responsible for the observation that chalcopyrite increases in late-stage veins relative to pentlandite.

Patterson and Watkinson (1984) noted that during regional metamorphism the primary disseminated sulphides were modified into veins and veinlets by the recrystallization of the surrounding silicates. Strong dynamic metamorphism mobilized the sulphides into fractures and pressure shadows. It also significantly changed the copper-to-nickel ratio of the mylonite and breccia hosted sulphides compared to the present disseminated material. Further, the occurrence of fragments of mylonite in the breccia mineralization suggests that the breccia hosted sulphides were formed during dynamic metamorphism (Patterson and Watkinson, 1984).

Curtis (2001) observed that the Thierry Deposit contains significant concentrations of platinum and palladium with unusual characteristics. Unlike many Ni-Cu-PGE deposits, the Thierry Deposit is not obviously of primary magmatic derivation.

Within the spectrum of magmatic Ni-Cu-PGE deposits, the two dominant examples are those from the Norilsk and Sudbury Districts. Naldrett (1999) has indicated that the two key factors which discriminate major deposits of this kind include:

- The efficient segregation and subsequent concentration of sulphides from a large volume of magma; and

- Sufficient time and element mobility to allow the sulphides to interact with enough magma to concentrate Ni, Cu and PGEs.

It is evident that despite the fact that Ni, Cu, PGE mineralization in the various zones on the Thierry Project is associated with a metamorphosed ultramafic mafic complex, within which Ni and Cu are enriched, there is little textural evidence to suggest that primary magmatic concentration of sulphides played a major role in elevating the PGE content of the mineralization.

According to Curtis (2001), what is more evident with Thierry, and common to several other PGE enriched deposits is the following:

- The PGE enriched mineralization is structurally confined.

- The PGE's occur in association with higher concentrations of Ni and/or Cu, but the relationship is not exclusive, i.e. high concentration of PGE's are recognized also in zones that have low concentrations of Ni and Cu.

- Host rocks to the mineralization have been subjected to upper greenschist-lower amphibolite grade.

- There is evidence for late stage remobilization of sulphides with PGE's.

- There is evidence for involvement of hydrothermal fluids synchronous with metamorphism and remobilization.

Evidence is emerging from studies of similar deposits (the New Rambler Mine in Wyoming (McCallum et al, 1976), the Rathburn Lake occurrence in north-eastern Ontario (Rowell and Edgar, 1986), the Salt Chuck Intrusion in Alaska (Watkinson and Melling, 1992) and parts of the Lac des Isles complex (Pyle, 1968)) that platinum and palladium (in particular when associated with bismuth and tellurium) can be mobilized and concentrated by hydrothermal fluids. Aqueous solutions are also known to remobilize and concentrate PGE's in laterites and placers. The additional association of hydrous silicates, in particular chlorite, biotite, sericite, and actinolite-talc that are atypical of magmatic environments strongly suggest that PGE's are remobilized and re-concentrated by hydrothermal fluids in the metamorphic regime.

A structural study by SRK (Goodman, 2004) contends that this primary relationship between host rocks and mineralization has been obscured by deformation, metamorphism, and remobilization of mineralized rock. The present form of the Deposit is believed to result from extensive remobilization by hydrothermal fluids, into a ductile shear zone setting.

SRK (Keller, 2005) concluded that the Thierry Deposit is a shear-zone hosted deposit. As such, it shares the characteristic of any fault or shear-zone system that there are predictable areas of dilation and compression where the shear-zone bends or splays. Mineralization commonly accumulates in areas of dilation, as these areas are local low pressure zones, physically favouring sulphide precipitation, and allowing fluid mixing, which can provide a chemical trigger for precipitation.

Any model of mineralized rock genesis at the Thierry Project must take into account the unusual Cu/Ni, Pt/Pd and chalcopyrite / pyrrhotite ratios in the rocks. According to Naldrett and Cabri (1976), intrusive complexes similar to those at Thierry Deposit contain sulphides with a copper-nickel ratio of 2:1, a platinum-palladium ratio of 1:4, and a chalcopyrite/pyrrhotite ratio of 1:10. These ratios at the Thierry Deposit are approximately: copper-nickel 8:1, platinum-palladium 1:4 and chalcopyrite/pyrrhotite 1:1.

9.0 EXPLORATION

Braveheart has not conducted any exploration work since acquiring the Thierry Project. Previous exploration work is summarized in Section 6 of this Technical Report.

10.0 DRILLING

Braveheart has not conducted any drilling on the Thierry Project. A summary of the most recent drilling by Cadillac is provided below.

Cadillac completed three (3) drill holes in 2010 and 12 drill holes in 2011 on the Thierry Deposit, and 42 drill holes on the K1-1 Deposit in 2011-2012. Previous drilling conducted by Richview is summarized in Puritch, et al (2012).

10.1 THIERRY DEPOSIT DRILL PROGRAM

In 2010, Cadillac completed three (3) drill holes totalling 3,330 m (10,926 ft) of drilling, on the Thierry Deposit with the intention of infilling missing information from the 2010 P&E Thierry Deposit block model at depth. All three holes intersected mineralization and aided in closing a void in the model where there was no drilling. A table of significant intersections is presented in Table 10.1.

TABLE 10.1
2010 THIERRY DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS

Drill Hole ID	Azimuth (°)	Dip (°)	From (ft)	To (ft)	Length (ft)	Cu (%)	Ni (%)
CV-10-06B	180	-80	3,486.48	3,503.75	17.27	1.043	0.047
including	180	-80	3,491.48	3,503.75	12.27	1.26	0.065
including	180	-80	3,496.48	3,500.72	4.24	2.21	0.147
CV-10-06B	180	-80	3,654.92	3,657.82	2.90	1.82	0.056
CV-10-06B	170	-80	3,710.63	3,715.68	5.05	1.24	0.047
CV-10-01	170	-80	3,290.40	3,309.17	18.77	1.064	0.184
including	170	-80	3,290.40	3,305.40	15.00	1.191	0.196
CV-10-01	170	-80	3,338.17	3,368.64	30.47	0.336	0.081
including	170	-80	3,338.17	3,343.17	5.00	0.558	0.128
including	170	-80	3,366.14	3,368.64	2.50	1.310	0.079
CV-10-01	170	-80	3,378.64	3,388.13	9.49	0.574	0.285
including	170	-80	3,384.66	3,388.13	3.47	0.617	0.592
CV-10-04	172.5	-82	3,811.21	3,820.87	9.66	0.80	0.140
including	172.5	-82	3,815.14	3,817.94	2.80	1.52	0.233
CV-10-04	172.5	-82	3,835.57	3,857.55	21.98	2.02	0.120
including	172.5	-82	3,835.57	3,840.57	5	3.38	0.119
CV-10-04	172.5	-82	3,996.06	4,001.29	5.23	1.49	0.09

In 2011, Cadillac advanced 12 drill holes on the Thierry Deposit. Three drill holes, CV-11-02, CV-11-03 and CV-11-04, were the last deep holes drilled into the same void at depth as 2010 drill holes. The six (6) deep drill holes, from 2010 and 2011, totalled 6,817 m (22,367 ft) of drilling.

Three (3) drill holes, CV-11-08 to CV-11-10 were used to extend the mineralization to the west of the known Deposit along strike as part of the 2011 shallow drilling program. In addition, 6 drill holes, CV-11-11 to CV-11-16 were used to extend the eastern strike of the Thierry Deposit, also as part of the 2011 shallow drilling program. A list of significant intersections is presented in Table 10.2.

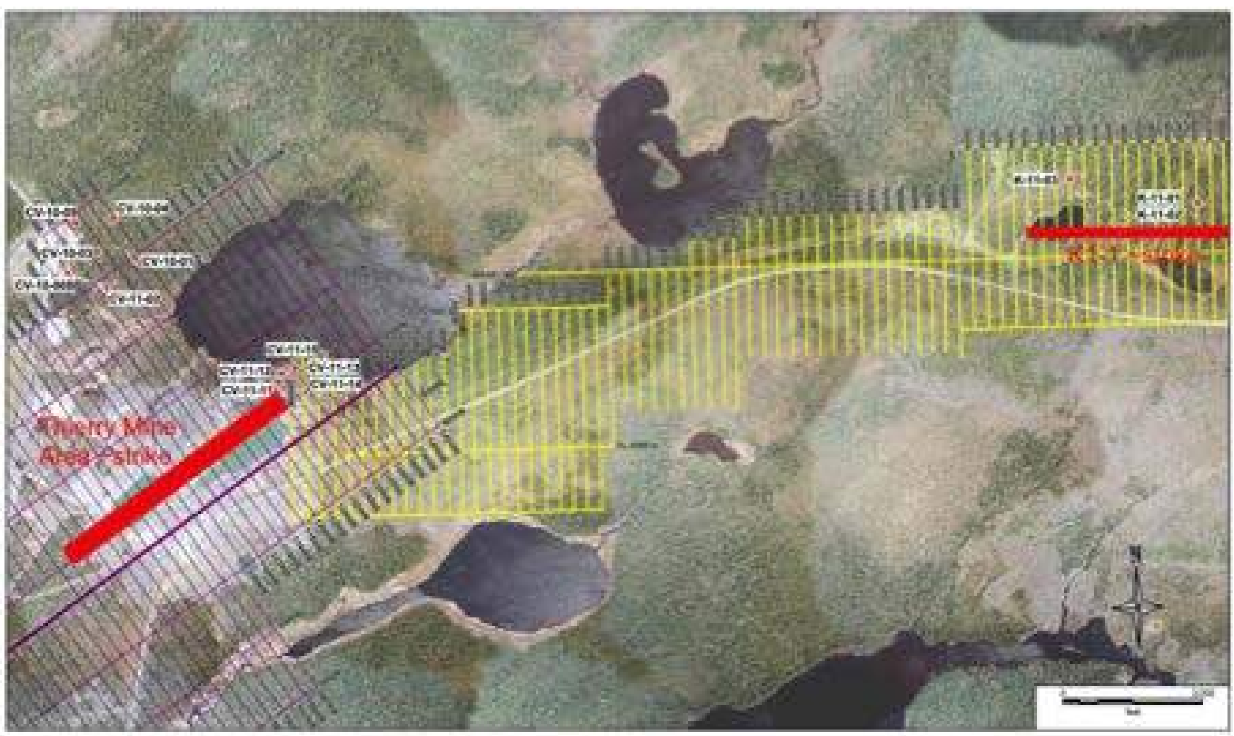
TABLE 10.2
2011 THIERRY DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS

Drill Hole ID	Azimuth (°)	Dip (°)	From (ft)	To (ft)	Length (ft)	Cu (%)	Ni (%)
CV-11-02	170	-80	3,025.10	3,038.10	13.00	0.98	
including	170	-80	3,028.40	3,038.10	9.70	1.11	
CV-11-03	170.8	-80	3,405.20	3,436.60	33.40	1.42	
including	170.8	-80	3,411.20	3,426.30	15.10	1.70	
including	170.8	-80	3,418.10	3,426.30	8.20	2.12	
CV-11-03	170.8	-80	3,418.10	3,426.30	8.20	2.12	
CV-11-05	170	-80	3,878.3	3,883.5	5.2	1.76	0.158
CV-11-05	170	-80	3,883.5	3,887.4	3.9	3.38	0.156
CV-11-05	170	-80	3,887.4	3,890.4	3	1.72	0.120
CV-11-05	170	-80	3,890.4	3,892.4	2	0.02	0.004
CV-11-05	170	-80	3,892.4	3,897.7	5.3	0.98	0.148
CV-11-05	170	-80	3,897.7	3,899.7	2	0.85	0.59
CV-11-08	172	-85	1,159.00	1,176.00	17.00	0.74	
including	172	-85	1,170.00	1,173.80	3.80	1.08	
CV11-09	172	-85	726.90	742.50	15.60	1.25	
including	172	-85	726.90	735.15	8.25	1.30	
including	172	-85	740.00	742.50	2.50	1.71	
CV-11-10	172	-85	705.00	710.40	5.40	0.64	
CV-11-11	172	-70	573.20	595.60	22.40	0.77	
including	172	-70	580.00	590.00	10	1.17	
CV-11-12	136	-70	633.50	654.50	16	0.59	
including	136	-70	633.50	645.00	6.5	0.88	
CV-11-13	136	-70	630.30	636.10	5.8	1.23	
CV-11-14	136	-70	563	575	12	0.29	
CV-11-15	136	-70	625	631.5	6.5	0.59	
including	136	-70	629	631.5	2.5	0.81	
CV-11-16	144	-50	557	570	13	0.63	
including	144	-50	565.5	570	4.5	0.81	

10.2 K1-1 PROPERTY DRILL PROGRAM

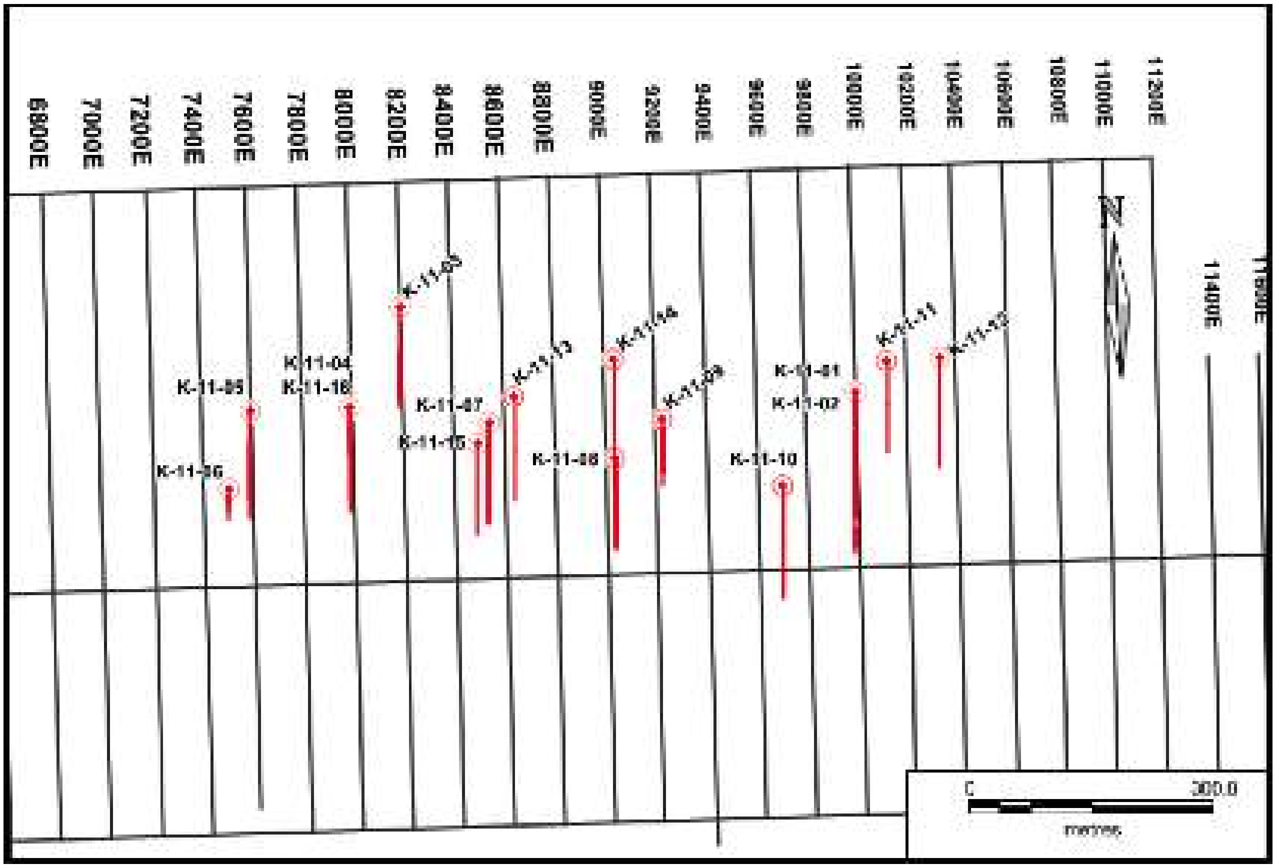
The K1-1 Deposit area is located approximately 3 km east of the Thierry Deposit as shown in Figure 10.1 which also shows an interpretation of the strike for both deposits. In February and March of 2011 Cadillac completed three shallow drill holes on the K1-1 Deposit (Boreholes K-11-01 to K-11-03) designed to confirm results obtained previously by UMEX and other previous operators. Drill hole K-11-01 was drilled at eastern limit of the K1-1 mineralization, drill hole K-11-02 was drilled to undercut K-11-01 at the same location, and drill hole K-11-03 was drilled at the western end of the K1-1 mineralization. Based on the positive results of the initial 3 drill holes, an additional 13 drill holes were advanced on the K1-1 Deposit and completed in June 2011. The March-June drilling totalled 3,802 m (12,475 ft) in 16 holes. A summary of the significant intersections is presented in Table 10.3 and the borehole locations are presented in Figure 10.2. A second phase of drilling, consisting of 6,406 m (21,018 ft) drilled in 26 holes, was completed in 2012 and targeted the K1-1 open pit area. The program was designed to address gaps within the pit shell area and to test for extensions along strike and at depth. A list of significant intersections is presented in Table 10.4 and the borehole locations are presented in Figure 10.3.

FIGURE 10.1 DRILL HOLE LOCATIONS FOR 2011 K1-1 DRILL PROGRAM



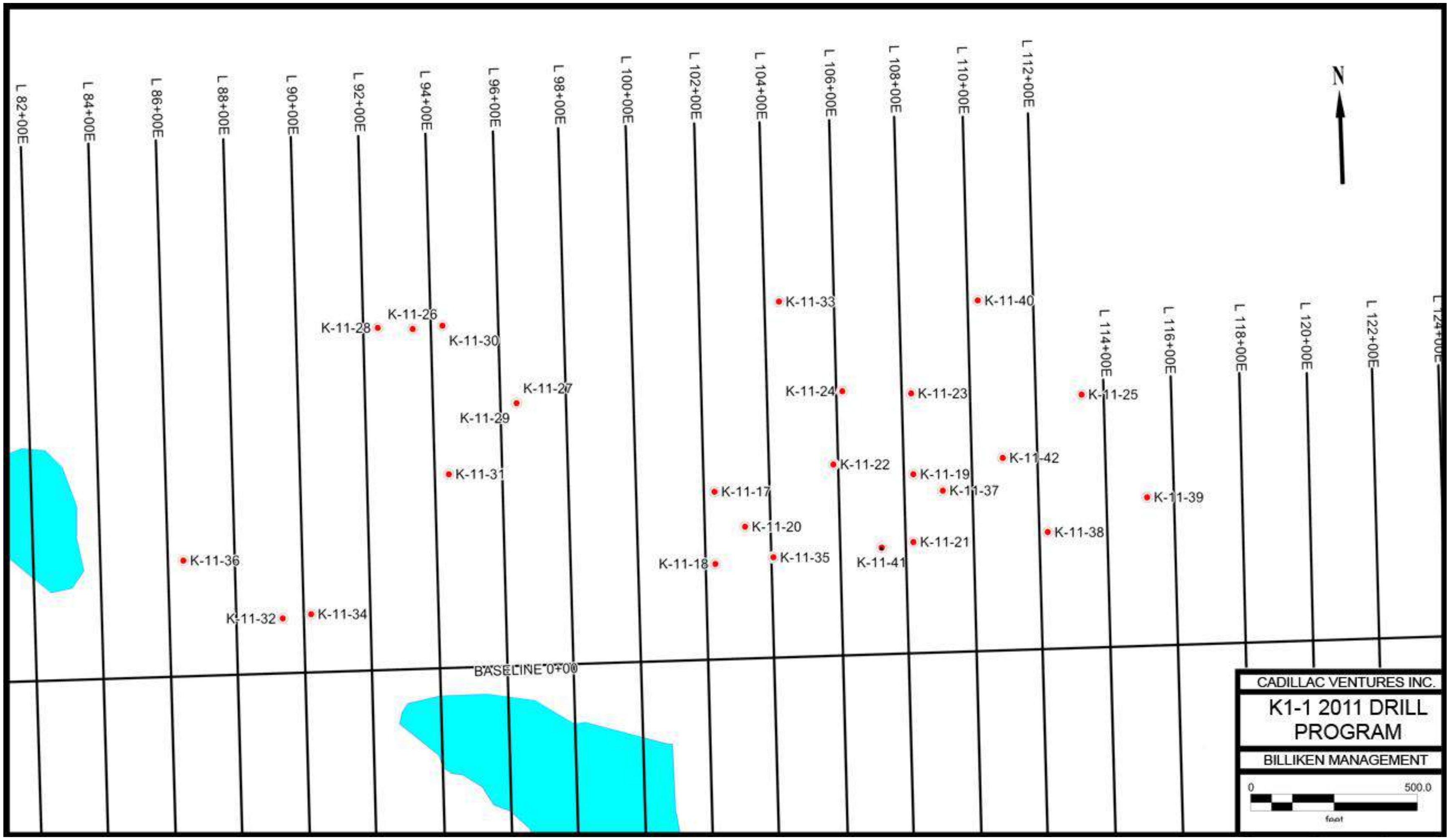
Source: *www.cadillacventures.com (2012)*

FIGURE 10.2 PLAN VIEW OF BOREHOLE LOCATIONS FOR 2011 K1-1 DRILL PROGRAM



Source: *www.cadillacventures.com (2012)*

FIGURE 10.3 2011 BOREHOLE LOCATIONS



Source: *Cadillac (2012)*

TABLE 10.3
2011 K1-1 DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS

Drill Hole ID	Azimuth (°)	Dip (°)	From (ft)	To (ft)	Length (ft)	Cu (%)	Ni (%)
K-11-01	180	-50	348	381.50	33.50	0.41	0.10
K-11-01	180	-50	400	475	75	0.30	0.06
K-11-01	180	-50	500	510	10	0.65	0.09
K-11-01	180	-50	525	615	90	0.58	0.08
including	180	-50	555	585	30	0.92	0.12
K-11-01	180	-50	620	793.50	173.50	0.27	0.05
K-11-01	180	-50	925	970	45	0.31	0.09
K-11-02	180	-80	485	615	130	0.35	0.08
K-11-02	180	-80	639.75	648.75	9	0.46	0.04
K-11-03	180	-80	530	1112	582	0.39	0.11
K-11-04	180	-50	150	580	430	0.36	0.10
including	180	-50	207	316	108.6	0.55	0.09
including	180	-50	235	270	35	0.84	0.09
K-11-05	180	-50	215	560	345	0.18	0.06
including	180	-50	290	310	20	0.27	0.06
including	180	-50	340	355	15	0.26	0.10
including	180	-50	365	440	75	0.26	0.08
including	180	-50	480	545	65	0.26	0.08
K-11-06	180	-70	35.5	280	138.55	0.28	0.06
including	180	-70	115	120	5	1.04	0.17
K-11-07	180	-50	45	270	225	0.16	0.08
including	180	-50	55	120	65	0.26	0.09
K-11-07	180	-50	274.5	635	360.5	0.27	0.13
including	180	-50	520	615	95	0.50	0.26
including	180	-50	546.5	550	3.5	1.12	0.405
including	180	-50	550	555	5	0.778	0.23
including	180	-50	555	560	5	0.649	0.63
including	180	-50	560	565	5	0.427	0.636
including	180	-50	565	570	5	0.748	0.251
including	180	-50	570	575	5	0.264	0.139
including	180	-50	575	580	5	0.68	0.322
including	180	-50	580	585	5	0.488	0.158
including	180	-50	585	590	5	0.494	0.476
including	180	-50	590	595	5	0.549	0.459
including	180	-50	595	598.5	4.5	0.712	0.417
K-11-08	180	-60	110.0	720.0	610.0	0.265	0.105

TABLE 10.3 2011 K1-1 DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS							
Drill Hole ID	**Azimuth (°)**	**Dip (°)**	**From (ft)**	**To (ft)**	**Length (ft)**	**Cu (%)**	**Ni (%)**
including	180	-60	110.0	205.0	95.0	0.392	0.110
including	180	-60	155.0	200.0	45.0	0.456	0.116
including	180	-60	340.0	405.0	65.0	0.339	0.103
including	180	-60	435.0	535.0	100.0	0.317	0.098
including	180	-60	571.1	660.0	88.9	0.298	0.086
including	180	-60	675.0	710.0	35.0	0.339	0.088
K-11-08	180	-60	150.0	335.0	185.0	0.238	0.135
including	180	-60	195.25	280.0	84.75	0.197	0.149
including	180	-60	215.0	245.0	30.0	0.186	0.166
K-11-09	180	-60	120.0	464.0	344.0	0.353	0.080
including	180	-60	120.0	180.0	60.0	0.438	0.080
including	180	-60	135.0	175.0	40.0	0.534	0.083
including	180	-60	230.0	340.0	110.0	0.383	0.091
including	180	-60	385.0	464.0	79.0	0.372	0.082
including	180	-60	280.0	340.0	60.0	0.403	0.100
including	180	-60	300.0	320.0	20.0	0.549	0.108
K-11-10	180	-50	18.6	535.0	516.4	0.329	0.064
including	180	-50	130.0	190.0	60.0	0.41	0.07
including	180	-50	210.0	265.0	55.0	0.52	0.08
including	180	-50	320.0	350.0	30	0.42	0.07
including	180	-50	385.0	470.0	85.0	0.39	0.09
including	180	-50	495.0	535.0	40.0	0.51	0.10
including	180	-50	550.0	570.0	20.0	0.43	0.11
including	180	-50	643.5	685.0	41.5	0.34	0.1
K-11-11	180	-70	590	673.9	83.9	0.26	0.06
including	180	-70	645.0	673.9	28.9	0.33	0.1
K-11-11	180	-70	820.0	845.0	25.0	0.42	0.13
K-11-11	180	-70	900.0	965.0	65.0	0.35	0.09
including	180	-70	915.0	965.0	50.0	0.39	0.1
K-11-11	180	-70	992.3	1,055	62.7	0.31	0.07
including	180	-70	992.3	1,030	37.7	0.35	0.08
K-11-12	180	-65	570	955	385.0	0.24	0.07
including	180	-65	696.2	765	68.8	0.35	0.08
including	180	-65	790	800	10	0.39	0.07
including	180	-65	860	875	15	0.31	0.11
K-11-13	180	-60	139	150	11	0.37	0.09

TABLE 10.3 2011 K1-1 DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS							
Drill Hole ID	**Azimuth (°)**	**Dip (°)**	**From (ft)**	**To (ft)**	**Length (ft)**	**Cu (%)**	**Ni (%)**
K-11-13	180	-60	227	245	18	0.31	0.11
K-11-13	180	-60	415	440	25	0.30	0.13
K-11-13	180	-60	565	610.9	45.9	0.36	0.13
including	180	-60	595	610.9	15.9	0.52	0.18
K-11-14	180	-60	260	502.5	242.5	0.25	0.10
including	180	-60	300	465	165	0.30	0.11
including	180	-60	395	425	30	0.37	0.11
K-11-15	180	-50	190	305	115	0.28	0.11
including	180	-50	285	305	20	0.36	0.11
K-11-15	180	-50	340	490	150	0.24	0.09
K-11-16	180	-70	210	570	360	0.27	0.11
including	180	-70	230	275	45	0.39	0.17
including	180	-70	310	405	95	0.34	0.12
including	180	-70	355	405	50	0.40	0.07
including	180	-70	450	490	40	0.37	0.09
including	180	-70	545	555	10	0.39	0.13
Total 16 holes							

TABLE 10.4 2012 K1-1 DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS							
Drill Hole ID	**Azimuth (°)**	**Dip (°)**	**From (ft)**	**To (ft)**	**Length (ft)**	**Cu (%)**	**Ni (%)**
K-11-17	180	-50	335	345	10	0.41	0.1
	180	-50	418.6	440	21.4	0.31	0.1
	180	-50	480	605	125	0.35	0.1
Including	180	-50	480	510	30	0.34	0.10
Including	180	-50	545	570	25	0.42	0.11
	180	-50	645	660	15	0.42	0.07
K-11-18	180	-50	60	205	145	0.32	0.08
Including	180	-50	125	205	80	0.42	0.12
Including	180	-50	125	140	15	0.56	0.12
K-11-19	180	-45	341.8	429.3	87.5	0.55	0.01
Including	180	-45	341.8	410	68.2	0.60	0.1
Including	180	-45	341.8	374.5	32.7	0.72	0.12
K-11-20	180	-45	290	320	30	0.32	0.6

TABLE 10.4 2012 K1-1 DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS							
Drill Hole ID	**Azimuth (°)**	**Dip (°)**	**From (ft)**	**To (ft)**	**Length (ft)**	**Cu (%)**	**Ni (%)**
	180	-45	340	345	5	0.72	0.10
	180	-45	530	585	55	0.32	0.07
K-11-21	180	-45	190	320	110	0.63	0.09
Including	180	-45	190	275	85	0.71	0.10
Including	180	-45	200	225	25	0.97	0.15
	180	-45	377.5	485	107.5	0.33	0.07
Including	180	-45	377.5	465	87.5	0.36	0.08
Including	180	-45	410	465	55	0.37	0.09
K-11-22	180	-50	330	400	70	0.48	0.08
Including	180	-50	330	375	45	0.537	0.087
	180	-50	420	430	10	0.42	0.10
	180	-50	450	465	15	0.53	0.12
	180	-50	490	560	70	0.34	0.08
Including	180	-50	540	550	20	0.46	0.10
	180	-50	580	600	20	0.34	0.08
	180	-50	640	650	10	0.5	0.7
K-11-23	180	-50	475	535	60	0.51	0.06
Including	180	-50	475	495	20	0.74	0.07
Including	180	-50	515	535	20	0.60	0.07
K-11-24	180	-55	550	790	240	0.31	0.08
Including	180	-55	550	570	20	0.46	0.10
Including	180	-55	590	610	20	0.49	0.11
K-11-25	180	-60	520	770	250	0.34	0.08
Including	180	-60	520	555	35	0.42	0.07
Including	180	-60	690	770	80	0.42	0.08
Including	180	-60	735	760	25	0.62	0.09
K-11-26	180	-50	518.75	1,325	806.25	0.37	0.09
Including	180	-50	1,010	1,220	210	0.42	0.11
Including	180	-50	630	660	30	0.31	0.07
K-11-27	180	-60	415	1,005	585	0.34	0.06
Including	180	-60	415	535	120	0.65	0.08
Including	180	-60	995	1,005	10	0.48	0.07
K-11-28	180	-50	495	1,320	825	0.4	0.10
Including	180	-50	720	925	205	0.52	0.10
Including	180	-50	1,010	1,015	5	1.33	0.10
K-11-29	180	-55	395	520	125	0.31	0.05

TABLE 10.4 2012 K1-1 DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS							
Drill Hole ID	**Azimuth (°)**	**Dip (°)**	**From (ft)**	**To (ft)**	**Length (ft)**	**Cu (%)**	**Ni (%)**
Including	180	-55	395	420	25	0.42	0.06
Including	180	-55	440	470	30	0.35	0.05
	180	-55	845	885	40	0.44	0.02
	180	-55	1,010	1,020	10	0.55	0.12
K-11-30	180	-50	515	550	35	0.33	0.05
	180	-50	675	967.7	292.7	0.36	0.08
Including	180	-50	715	755	40	0.42	0.08
Including	180	-50	835	935	100	0.41	0.10
	180	-50	1,015	1,045	30	0.46	0.11
	180	-50	1,140	1,170	30	0.36	0.09
K-11-31	180	-55	165	180	15	0.58	0.1
	180	-55	285	655	370	0.37	0.10
Including	180	-55	325	420	95	0.48	0.09
Including	180	-55	625	645	20	0.35	0.12
	180	-55	690	780	90	0.41	0.11
	180	-55	825	850	25	0.38	0.11
K-11-32	180	-50	278	325	47	0.33	0.12
	180	-50	390	410	20	0.38	0.11
K-11-33	180	-50	240	380	140	0.29	0.07
Including	180	-50	265	295	30	0.42	0.07
Including	180	-50	340	380	40	0.33	0.07
	180	-50	555	575	20	0.38	0.09
K-11-34	180	-50	140	195	55	0.34	0.11
	180	-50	345	370	20	0.38	0.09
K-11-35	180	-50	44	45.33	1.33	0.92	2.13
	180	-50	95	130	35	0.34	0.06
	180	-50	255	260	5	0.99	0.12
	180	-50	280	300	20	0.43	0.09
	180	-50	365	375	10	0.40	0.10
	180	-50	425	440	15	0.39	0.14
K-11-36	180	-65	35	40	5	0.97	0.15
	180	-65	70	135	65	0.36	0.1
Including	180	-65	110	135	25	0.41	0.13
	180	-65	160	180	20	0.31	0.08
	180	-65	243.3	245.3	2	0.12	2
	180	-65	445	455	10	0.34	0.16

TABLE 10.4 2012 K1-1 DRILLING PROGRAM - SIGNIFICANT INTERCEPTS AND ASSAY RESULTS							
Drill Hole ID	**Azimuth (°)**	**Dip (°)**	**From (ft)**	**To (ft)**	**Length (ft)**	**Cu (%)**	**Ni (%)**
K-11-37	180	-50	270	285	15	0.38	0.05
	180	-50	310	390	80	0.33	0.06
Including	180	-50	335	350	15	0.64	0.10
	180	-50	515	565	50	0.36	0.07
Including	180	-50	515	535	20	0.43	0.08
K-11-38	180	-50	105	125	20	0.34	0.05
	180	-50	140	150	10	0.40	0.07
	180	-50	245	325	80	0.46	0.11
Including	180	-50	245	295	50	0.50	0.11
K-11-39	180	-50	165	185	20	0.35	0.05
K-11-40	180	-50	425	435	10	0.35	0.05
	180	-50	460	470	10	0.57	0.07
	180	-50	525	565	40	0.58	0.14
	180	-50	615	620	5	0.66	0.09
	180	-50	640	650	10	0.65	0.08
K-11-41	180	-45	105	115	10	0.38	0.06
	180	-45	175	185	10	0.50	0.11
	180	-45	210	235	25	0.41	0.11
	180	-45	285	320	35	0.53	0.08
	180	-45	340	510	150	0.36	0.08
Including	180	-45	425	450	25	0.45	0.11
K-11-42	180	-50	245	250	5	0.70	0.06
	180	-50	295	315	20	0.43	0.05
	180	-50	550	575	25	0.36	0.07
Total 26 holes							

10.3 CORE RECOVERY AND SAMPLING

Drill core recovery was greater than 99% in all sections sampled allowing all samples to be truly representative of the encountered mineralization. No factors that could materially impact the accuracy and reliability of the samples were identified. Rock types and geological controls were described in detail in the drill logs, as were samples and true widths, where known.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

This Technical Report section pertains to the drill program conducted by Cadillac during the period 2010 to 2012.

All sulphidic zones deemed to have potential of hosting precious or base metals were sampled. 1.5 m was sampled on either side of every mineralized zone. Drill core was cut in half, with one half stored in core boxes on site and the other half cut in half again. This quarter drill core was sampled (other ¼ for duplicate).

Drill core sample lengths ranged from 0.3 m to 1.5 m. The drill core was cut on site by contract labourers under the supervision of the Brian H. Newton, P.Geo who was directly responsible for all aspects of sample collection, on-site sample preparation and subsequent shipping to the assay laboratory. Once cut, the remaining drill core was stored on-site in clearly labelled wooden core boxes placed in metal core racks.

Each individual sample was packaged in a labelled plastic bag with matching sample tags, placed in rice bags and secured with duct tape and flagged. Samples were transported by bonded carrier to Activation Laboratories in Thunder Bay, Ontario. Samples were prepared and assayed using Fire Assay ICPOES. Samples which assayed over 1% Cu were reprocessed using total digestion ICP (Total).

Activation Laboratories is an independent, internationally recognized minerals testing laboratory operating in 10 countries. The laboratory in Thunder Bay has also been accredited to ISO 17025 standards for specific laboratory procedures by the Standards Council of Canada ("SCC").

It is the author's opinion that there are no drilling, sampling, security or recovery factors that could materially impact the accuracy and reliability of the results, and the procedures were adequate for the purposes of this Mineral Resource Estimate.

Sample pulps from the 2011 summer drill program are stored at Activation Laboratories' Thunder Bay storage facility. Sample pulps from the 2012 program are stored at AGAT labs in Sudbury.

12.0 DATA VERIFICATION

12.1 SITE VISIT AND INDEPENDENT SAMPLING

Mr. Eugene Puritch, P.Eng., visited the Thierry and K1-1 deposits on December 15, 2005, May 5, 2010 and June 2, 2011 for the purpose of site visits and completion of an independent verification sampling program. Mr. Puritch has not returned to the site since 2011. However, the Property condition has remained the same, and there has been no drilling on the Thierry Deposit since his site visit.

Six samples were collected from six diamond drill holes by taking a quarter split of the half drill core remaining in the core box. An effort was made to sample a range of grades. At no time were any employees of Cadillac advised as to the identification of the samples to be chosen during the visit.

The samples were selected by Mr. Puritch and placed into sample bags which were sealed with tape and placed in a larger bag. The samples were brought by Mr. Puritch to the P&E office in Brampton and from there they were sent by courier to AGAT Laboratories ("AGAT") in Mississauga for analysis.

At each of its locations, AGAT has developed and implemented a Quality Management System ("QMS") designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards.

AGAT maintains ISO registrations and accreditations, which provide independent verification that a QMS is in operation at the location in question. Most AGAT laboratories are registered or are pending registration to ISO 9001:2000.

Samples were analyzed for copper, nickel and silver using a multi-acid-digest (HCl/HNO3/HClO4/HF), with an ICP finish.

Gold, palladium and platinum were determined using lead collection fire assay, with an ICP finish.

A comparison of the results is presented in Figure 12.1 through Figure 12.5.

FIGURE 12.1 K1-1 AND THIERRY DEPOSITS SITE VISIT SAMPLE RESULTS FOR COPPER

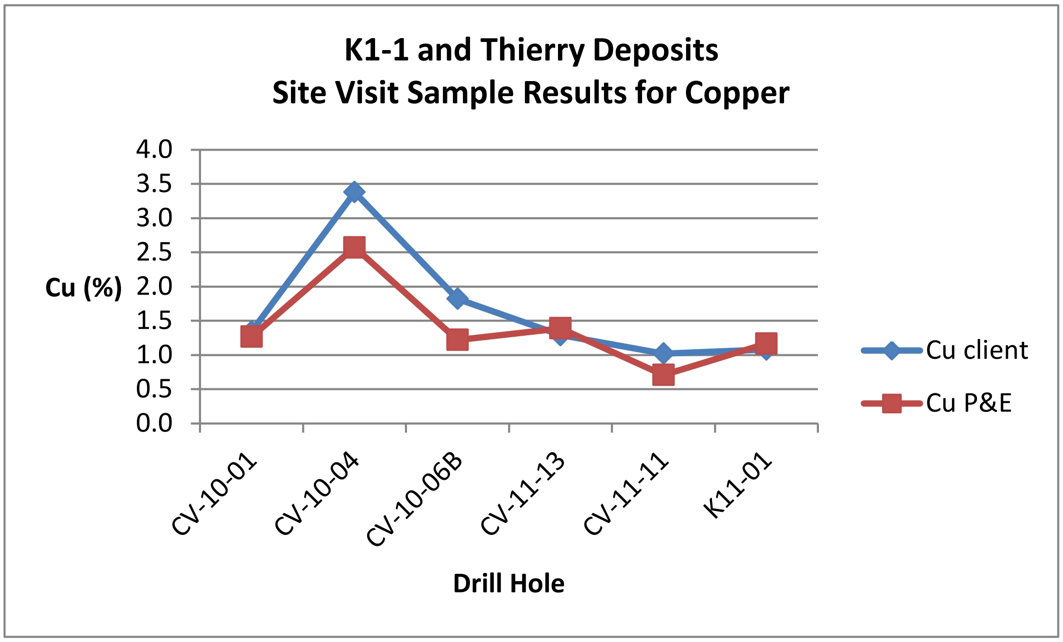


FIGURE 12.2 K1-1 AND THIERRY DEPOSITS SITE VISIT SAMPLE RESULTS FOR NICKEL

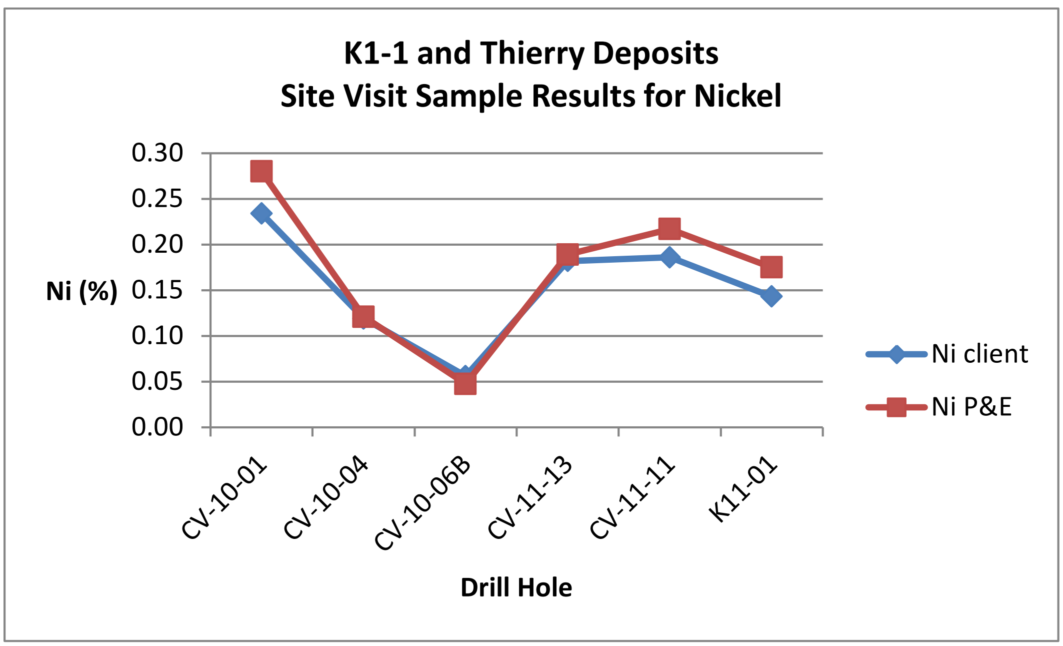


FIGURE 12.3 K1-1 AND THIERRY DEPOSITS SITE VISIT SAMPLE RESULTS FOR GOLD

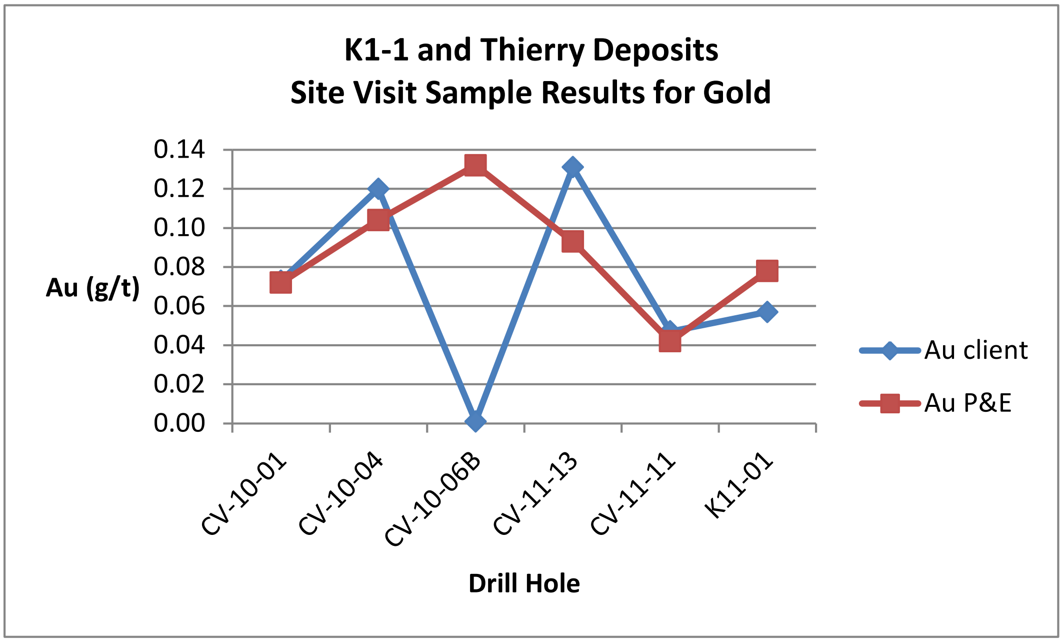


FIGURE 12.4 K1-1 AND THIERRY DEPOSITS SITE VISIT SAMPLE RESULTS FOR PALLADIUM

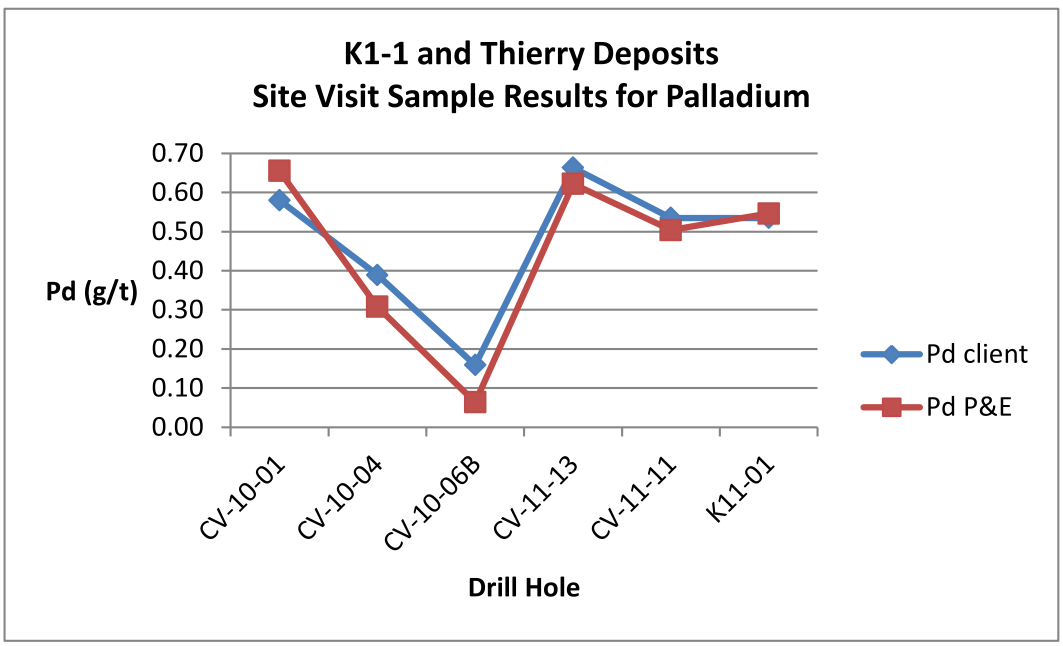
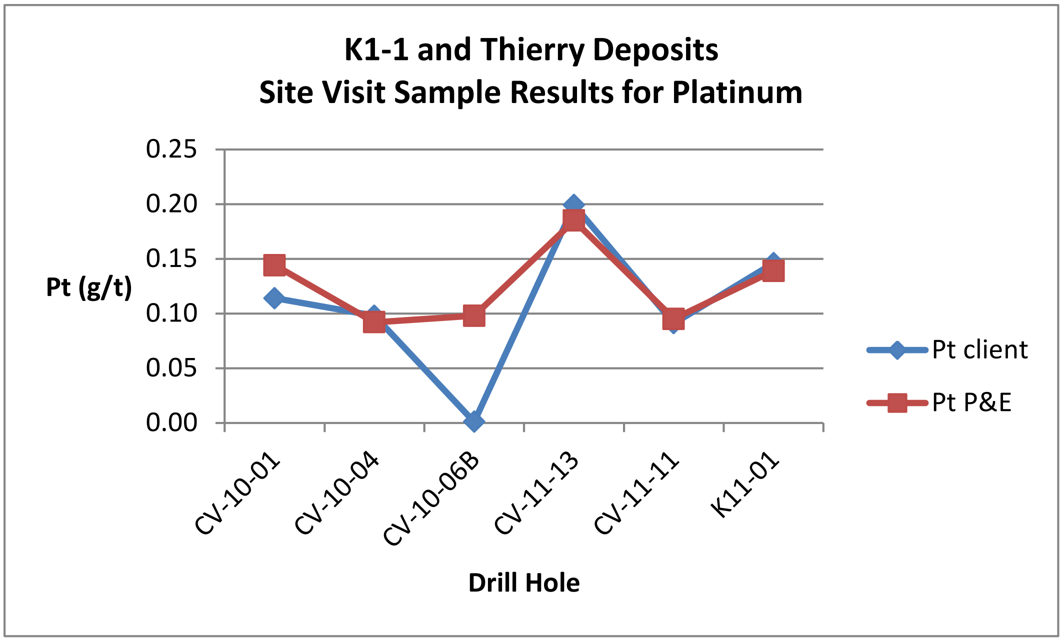


FIGURE 12.5 K1-1 AND THIERRY DEPOSITS SITE VISIT SAMPLE RESULTS FOR PLATINUM



12.2 QUALITY ASSURANCE/QUALITY CONTROL REVIEW

For the winter 2011 diamond drill program that was comprised of 26 holes, Cadillac essentially maintained the same Quality Assurance/Quality Control ("QA/QC" or "QC") program as had been initiated for the previous drilling with only a few minor changes. One certified reference material ("CRM" or "standard") was purchased from CDN Resource Labs in Langley, BC, and one from Analytical Solutions Ltd. in Toronto, ON, who are distributors for the OREAS certified reference materials from Australia.

Cadillac's QC program included the insertion of one blank, one standard and (provision for) one pulp duplicate approximately every 20 to 24 samples.

The certified reference materials monitored Cu, Ni, Au, Pd, and Pt.

12.3 PERFORMANCE OF CERTIFIED REFERENCE MATERIALS

There was a total of 126 certified reference materials inserted with the 26 drill holes. All values were graphed and compared to the warning limit of +/- 2 standard deviations from the mean of the between lab round robin characterization values. Assay values were also compared to a tolerance limit of +/- 3 standard deviations. For copper, there was an unacceptable level of failures, all exceeding +3 standard deviations from the mean. Drill hole samples bracketing the estimated Cu cut-off grade of 0.25%, (from 0.20% to 0.30%) that were associated with failed reference materials were re-run for Cu. All values re-run were lower in value that the original values, and the re-run values were imported into the master database. Cadillac inserted eight certified reference materials with these batches, and all were within the warning limits.

Accuracy for the other metals was acceptable.

12.3.1 Performance of Pulp Duplicates

A total of 68 pulp duplicate pairs were analyzed as part of the QC program.

Simple scatter graphs were prepared for each element for each duplicate type. At the pulp duplicate level, all metals apart from Au demonstrated excellent precision. There was imprecision demonstrated on the Au pulp duplicates, with a scatter higher than desired. An investigation will be conducted to try to determine the reasons for the imprecision.

12.3.2 Performance of Blank Material

Cadillac used a blank material obtained from sterile historical drill core for a total of 72 data points for each of the elements. 100% of the Ag values were < 3 times detection limit. For Pt, 100%, apart from one value, were less than 3 times detection limit. For Au, approximately 50% of the values exceeded 3 times detection limit, however, the average was 0.003 g/t Au. For Pd, the values were approximately 3 times detection limit with an average of 0.006 g/t Pd and one rogue high value of 0.15 g/t Pd. The rogue high value was examined and found to be the result of sample misallocation.

The copper and nickel data points were 100% above the set threshold of three times detection limit. Copper had an average value of 0.01% and two rogue high values of 0.05% and 0.06%. Nickel had an average value of 0.004% and a high value of 0.02%. The values as reported in the blanks are low and have no impact on Mineral Resource estimation. It is recommended however, that a completely sterile blank material be sourced for future analysis.

12.4 CONCLUSION

Based upon the evaluation of the QA/QC programs undertaken by Cadillac, P&E concludes that the data are of good quality for use in the Mineral Resource Estimate. Based upon P&E’s due diligence sampling and data verification, P&E concludes that the data are satisfactory for use in the Mineral Resource Estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 GENERAL

The Thierry Deposit was exploited by UMEX from 1976 to 1982. Approximately 5.3 Mt (5.8 M tons) of mineralization containing 1.13% copper and 0.14% nickel were processed. Initially only a copper concentrate was produced, however, by 1981 a limited amount of nickel concentrate was also produced. No records of plant metallurgical results from the six years of production are available.

Mineral processing and metallurgical testwork of mineralized material from the Thierry Deposit and its satellite deposits were conducted in late 1970s and in the early 1980s. More recently (2005 – 2006), metallurgical testwork was conducted on three composites from a 2004 drill program.

The following sections summarize the historical flotation testwork undertaken on material from the Thierry Deposits.

13.2 PRE-2005 METALLURGICAL TESTWORK

Testing on the Thierry Deposit included various metallurgical tests and mineralogical investigations. The following laboratories were involved:

- Louvain University, Belgium (U of L).
- Lakefield Research, Ontario (Lakefield – now SGS).
- Salman Mineral Research, Montreal (Salman).
- CANMET, Energy Mines and Resources, Ottawa.
- Noranda laboratory, Noranda Quebec (Noranda).
- UMEX Thierry plant as well consultants for hydrometallurgical treatment of bulk copper-nickel concentrates.

Lakefield conducted metallurgical tests (bench scale rougher-cleaner, locked cycle and pilot plant) needed to develop the Thierry flow sheet.

The initial tests performed by Lakefield were designed to evaluate the feasibility of producing separate smelter-acceptable copper and nickel concentrates. However, it was quickly concluded in March 1972 that a separate marketable nickel concentrate would be challenging. This was due to the low nickel content in mineralized rock, only about half present as recoverable pentlandite, with the remainder being present as extremely fine-grained pentlandite, nickeliferous pyrrhotite, and as nickeliferous silicates. Also, due to smelter restrictions, the focus was directed in producing a high quality copper concentrate.

Subsequent tests suggested that a substantial proportion of the nickel reported to the copper concentrate. This was remediated by Lakefield by applying a fine grind to rougher copper-nickel concentrates. Follow-up tests to produce a marketable bulk copper-nickel concentrate in addition to a copper concentrate were unsuccessful.

In April 1973, Lakefield conducted a pilot scale test on a bulk sample with the intention to produce a copper and a bulk copper-nickel concentrate from mineralized rock coming from crosscuts at levels 600 m and 1,600 m of the Thierry Mine which was under development.

The following results were obtained:

- Copper in concentrate: from 15% to 24% Cu at recoveries from 83% to 94%.
- Nickel in concentrate: from 1.4% to 2.5% Ni at recoveries from 29% to 68%.

It was concluded that the Thierry concentrator would be designed to produce a simple copper concentrate that met smelter criteria (Noranda, QC).

Salman initiated metallurgical tests on Thierry mineralized rock samples in 1973, with the targets of maximizing copper grade and recovery. Salman's metallurgical tests conclusively confirmed Lakefield's results of high grade, high recovery of copper, but similar to Lakefield, Salman was unable to produce a smelter-acceptable nickel concentrate.

The metallurgical laboratory at U of L carried out limited testing on the Thierry mineralized rock and concluded that the production of a high-grade copper concentrate with high metallurgical recovery was achievable. U of L did not attempt to produce a separate nickel concentrate. Detailed assays and mineralogical studies showed that (in the samples received) more than 50% of the nickel was present as violarite ($Fe^{2+}Ni_2{}^{3+}S_4$), the balance being present as pentlandite and in solid solution with pyrrhotite.

Noranda carried out flotation tests and a mineralogical study on a 0.23 t (500 lb) Thierry mineralized rock sample in February 1974. Good copper grade and recoveries were achieved; nickel recoveries were low:

- Copper in concentrate: 28.9% grade at 90. 6% recovery.
- Nickel in concentrate: 0.54% grade at 8.6% recovery.

When copper recovery was pushed up to 95%-96%, nickel reporting to the copper concentrate significantly increased. This was interpreted as being the result of a substantial amount of nickel in a small fraction of the chalcopyrite. Noranda suggested that much of the nickel was associated with copper in the form of nickeliferous chalcopyrite. The report concluded that a copper-nickel concentrate assaying between 3% and 5% nickel might be achieved by flotation, even in combination with magnetic separation. An 8% combined copper-nickel concentrate was suggested, that could be obtained by recycling the copper cleaner tailings.

In 1980, Lakefield performed several flotation tests on samples from the Thierry Deposit, and a blend (1:1) of both Thierry and the nearby K1-1 (Kapkichi) Deposit samples. The Thierry samples responded to the flotation tests as anticipated with good concentrate grade and recovery rates: 27.1% copper and 0.25% nickel grade with recoveries of 86.4% and 6.6%, respectively. As in the previous tests, nickel recoveries of up to about 65% were achieved in the rougher concentrates, however, only 5% to 30% in the copper cleaner concentrates.

Testwork by UMEX in 1980-81 in the Thierry process plant laboratory and in the process plant showed that a nickel concentrate could be floated from first or second copper cleaner tails. Two test results from the UMEX laboratory testwork reported a 5.9% and a 12.0% Ni grade in concentrate derived from copper first cleaner tails, with respective nickel recoveries of 70.0% and 61.5%. The laboratory work also indicated a 7.6% to 18.4% nickel grade in concentrate derived from second cleaner tails was achievable, with a nickel recovery ranging from 46.8% to 82.5%.

Process plant testwork based on treatment of copper first cleaner tails for Thierry indicated:

- 1.94% Ni to 10.61% Ni concentrate grade at recoveries of 9.9% to 42.0%.

The lower recoveries in both cases are isolated instances and may be anomalous.

There was some in-plant data where the relationship between Ni recovery and grade was not clear due to possible circuit instability. However, based on the available data, the metallurgical recovery versus metal grade relationship is summarized in Table 13.1.

TABLE 13.1
METALLURGICAL RECOVERY VS GRADE

Feed (%)		Copper Concentrate (%)				Nickel Concentrate (%)			
Cu	Ni	Cu	Ni	Cu Recovery	Ni Recovery	Cu	Ni	Cu Recovery	Ni Recovery
1.20	0.10	26.0	0.5	91.6	15.0	2.0	6.0	1.4	50.0

13.3 METALLURGICAL TESTWORK 2005-2006

13.3.1 Summary

A limited program of metallurgical testwork by SGS Lakefield in late 2005 and early 2006 was undertaken on crushed assay reject samples and on drill core. The study consisted of a grindability (Bond) index measurement, and bench scale flotation testwork including a single locked cycle test. The flotation behaviour of the composites tested was generally comparable to earlier work in that a good copper concentrate could be obtained, however, a nickel or copper-nickel concentrate containing nickel at saleable concentrations was not achieved.

The flowsheet evaluated by locked cycle testing involved flotation of a bulk copper-nickel rougher concentrate followed by regrinding and flotation of a copper concentrate containing minor nickel. The cleaner tailings from copper cleaner flotation was designated as a "copper-nickel cleaner feed" and contained about 3% each of copper and nickel. Cleaning of such a product to produce a copper-nickel concentrate was reported to be difficult and a single attempt on the cycle test material was unsuccessful. The small amount of cleaner tailings sample possibly contributed to the lack of success.

The locked cycle test conducted on the drill core ("DC") composite returned a copper concentrate containing 30.9% copper representing a copper recovery of 90.6%, with 10.48 g/t combined PGM plus gold.

13.3.2 Samples

Four composites from the Thierry Deposit were created from crushed reject borehole samples to provide a Master Composite and three sub-composites distinguished by copper grade. A single drill core composite sample was made and freezer-stored to minimize oxidation prior to testing. Table 13.2 summarizes the analyses on the various composites.

TABLE 13.2
ANALYSIS RESULTS ON COMPOSITE SAMPLES

Element		Low	Medium	High	Master	DC
XRF						
Cu	%	0.69	1.12	2.00	1.33	1.90
Ni	%	0.12	0.22	0.23	0.18	0.26
Fire Assay						
Pt	g/t	0.13	0.21	0.20	0.17	0.26
Pd	g/t	0.56	0.59	0.88	0.69	0.83
Au	g/t	0.07	0.12	0.19	0.24	0.18
Ag	g/t	4.7	8.0	13.0	8.5	
ICP-Scan						
Al	g/t	56000	43000	44000	50000	
As	g/t	<30	<30	<30	<30	
Ba	g/t	170	190	160	160	
Be	g/t	0.96	1.2	1.8	1.2	
Bi	g/t	<20	<20	<20	<20	
Ca	g/t	48000	32000	35000	40000	
Cd	g/t	<3	<3	<3	<3	
Co	g/t	130	210	220	180	
Cr	g/t	140	95	72	100	
Cu	g/t	6900	11000	21000	12000	
Fe	g/t	140000	160000	170000	150000	
K	g/t	8000	10000	8800	10000	
Li	g/t	13	14	14	14	
Mg	g/t	60000	64000	64000	64000	
Mn	g/t	1800	2100	1900	1900	
Mo	g/t	<5	<5	<5	<5	
Na	g/t	19000	14000	16000	16000	
Ni	g/t	1300	2200	2400	1800	
P	g/t	273	368	279	298	
Pb	g/t	<30	<30	<30	<30	
Sb	g/t	<10	<10	<10	<10	
Se	g/t	<30	<30	<30	<30	
Sn	g/t	<20	<20	<20	<20	
Sr	g/t	300	250	230	270	
Ti	g/t	3200	3000	2400	3000	
Tl	g/t	<30	<30	<30	<30	
V	g/t	120	90	77	94	
Y	g/t	8.0	6.5	5.6	6.3	
Zn	g/t	140	170	170	150	
Leco						
S	%	2.24	3.55	4.62	3.31	3.93
S=	%	1.81	3.01	3.88	2.74	
SO4	%	<0.4	0.4	0.4	0.4	
S°	%	<0.5	<0.5	<0.5	<0.5	
Bromine -Methanol						
Ni Sulfide	as Ni %	0.075	0.13	0.14	0.11	
%Ni as NiS		62.5	59.1	60.9	61.1	

13.3.3 Grinding

A single Bond ball mill work index test was performed on the DC sample. The grinding index was determined to be 15.9 kWh/tonne (at 150 mesh), indicating a moderately hard material.

13.3.4 Flotation

Preliminary work was conducted on the Master Composite ("MC"), followed by variability testing on the sub-composites and confirmatory work on the DC composite.

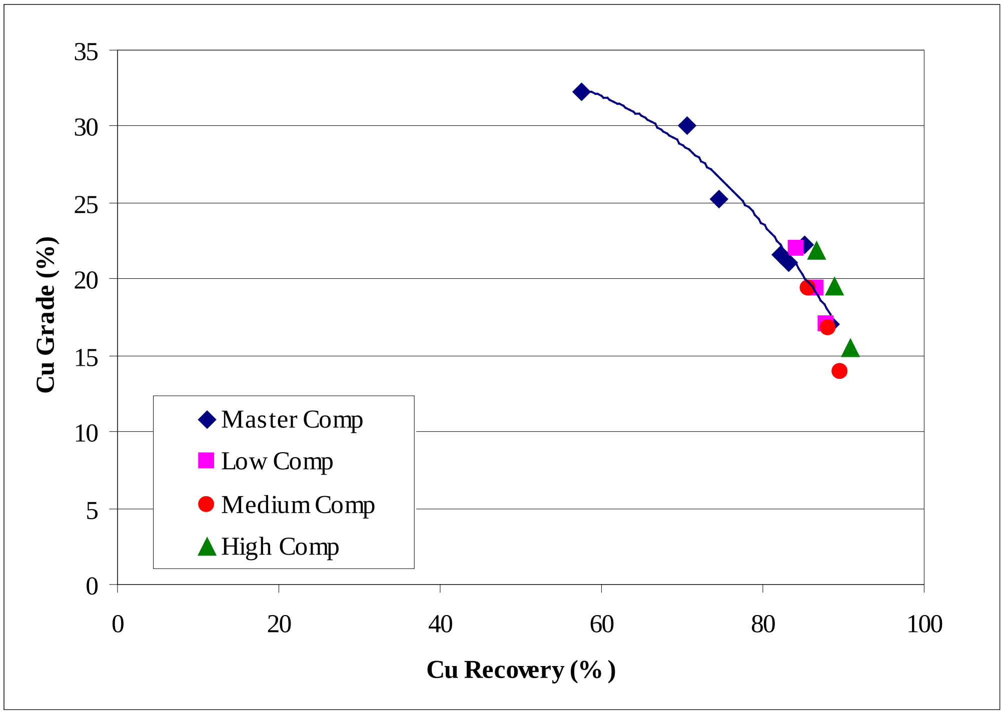
Bulk rougher flotation tests on the MC indicated that a simple reagent scheme was appropriate for the copper flotation. Testing of the effect of grind indicated improved copper flotation as the grind was increased from a K_{80} of 125 µm to 67 µm. A grind of 90 µm was selected for selected testwork to minimize the effect of overgrinding of nickel mineralization.

Grind had little effect on the slow flotation rate of copper that continued after 20 minutes.

A number of cleaner flotation tests were conducted on the MC sample, using conditions selected from the rougher tests; principally a grind of 90 microns. Bulk rougher concentrate cleaning was followed by copper-nickel separation at an elevated pH and testing of the use of CMC as a gangue depressant. Regrinding of the rougher concentrate was found to be necessary to obtain saleable copper grades and a grind of approximately 27 µm was used. A copper grade of 30% at a recovery of 70% was achieved in one test.

Variability testing showed little apparent difference in performance among the sub-composites as shown in Figure 13.1. The copper grade-recovery relationship was indicated to be independent of head grade.

FIGURE 13.1 COPPER RECOVERY VERSUS HEAD GRADE

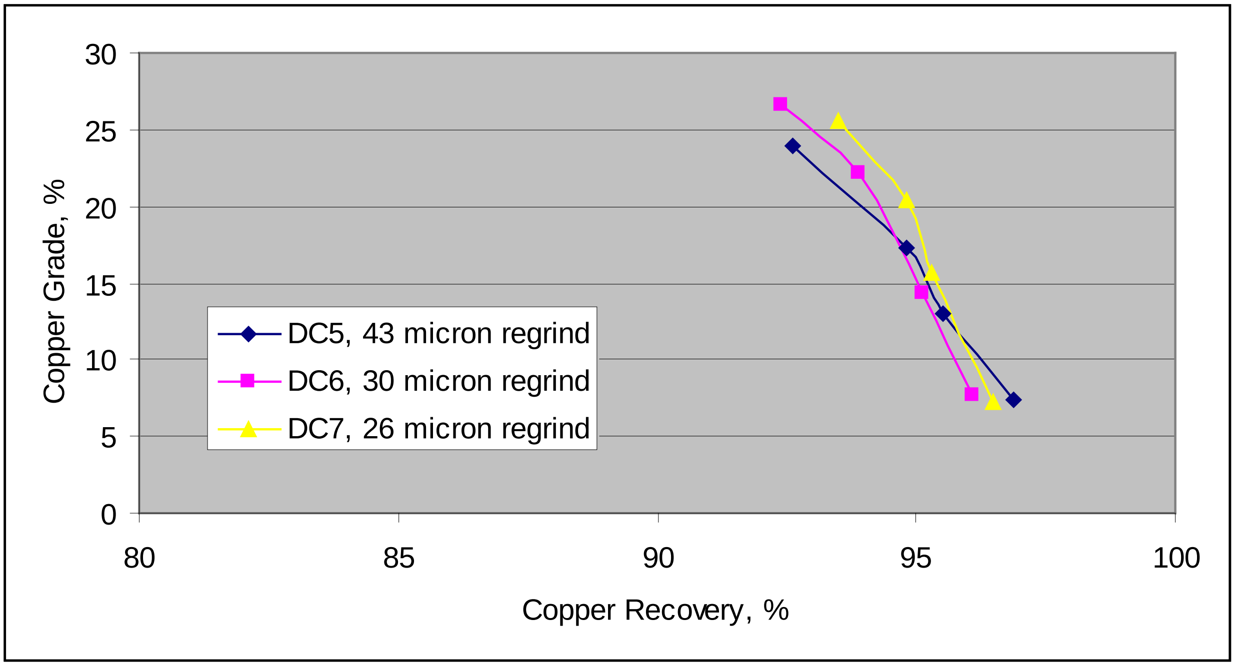


Source: *SGS (2006)*

Rougher flotation test results on the DC sample were similar to those obtained on the MC sample, although the grade-recovery relationship was improved indicating that the DC may have been less oxidized in storage. Variations in grinds of 105 and 149 µm had minimal effect on grade-recovery.

Cleaner tests were conducted rougher grind size of 105 µm and regrind sizes ranging from 26 to 43 µm. As shown in Figure 13.2 the results show a significantly positive effect of grind of the rougher concentrate.

FIGURE 13.2 CLEANER GRADE-RECOVERY RELATION TO GRIND SIZE OF ROUGHER CONCENTRATE



***Source:** SGS (2006)*

A single locked cycle test was conducted on the DC. The circuit design allowed for the production of a Cu-Ni concentrate, however, a low mass of material recovered as Cu-Ni feed allowed for only one attempt to produce a Ni concentrate. This test was not successful. The cycle test produced a high grade copper concentrate (30.9% Cu) at a copper recovery of 90.1%. The cycle test results are generally consistent with the cleaner test results and with earlier metallurgical work on the Thierry mineralization.

The locked cycle test yielded PGM metallurgical recoveries of 44.5% for Pt, 56.0% for Pd and 47.1% for Au, at concentrations of 1.71, 7.51, and 1.26 g/t, respectively, in the copper concentrate. Nickel was maintained at less than 0.5%.

13.4 METALLURGICAL PERFORMANCE ESTIMATES

The following assumptions may be used to estimate metallurgical performance in a revived Thierry process plant:

- A conventional crushing-grinding-flotation process is assumed;
- Two mineral concentrates are produced: high grade copper, and based on success in former Thierry process plant tests, a moderate grade nickel-copper;
- The copper concentrate will be marketed to a conventional copper smelter; the nickel-copper to a pyrometallurgical (smelter) facility or to a hydrometallurgical processor;

- New metallurgical tests will be performed on fresh drill core using best up-to-date grinding and flotation technology to maximize concentration performance and copper-nickel separation. Improved metallurgical results will be confirmed; and

- Hydrometallurgical testing of a bulk Cu-Ni-PGM concentrate (e.g. Platsol, Polymet type process) could be considered later.

The anticipated metallurgical performance is:

- Copper concentrate: 30% Cu, <1% Ni, @ 92% Cu and 50% PGM recoveries.
- Nickel concentrate: 8% Ni and 2% Cu, 40% Ni recovery.

Concentrate tonnage and concentration ratios will depend on head grade.

Table 13.3 summarizes the anticipated concentrate production.

TABLE 13.3
ESTIMATED CONCENTRATION PERFORMANCE

Average Heads		Copper Concentrate		Nickel Concentrate	
Element / Tonnes	**Grade**	**Grade**	**Recovery %**	**Grade**	**Recovery %**
Cu %	1.462	30.0	92	2.0	1
Ni %	0.160	0.54	15	8.0	40
Au g/t	0.069	0.77	50	0.26	3
Pt g/t	0.052	0.58	50	0.19	3
Pd g/t	0.144	1.61	50	0.54	3
Ag g/t	5.07	57.0	50	19.0	3
tpy	1.4 M		62,700		11,200

In the example shown in Table 13.3, the PGM's may be non-payable. It is noted that the current (2021) Mineral Resource grade PGM concentrations are significantly lower than in the SGS test composites (Table 13.2).

14.0 MINERAL RESOURCE ESTIMATE

There are two Mineral Resources described in this section which are the Thierry Mineral Resource and the K1-1 Mineral Resource.

14.1 P&E 2021 THIERRY MINERAL RESOURCE ESTIMATE

The purpose of this section is to delineate the Thierry Deposit Mineral Resources in compliance with NI 43-101 and CIM (2014) Standards and Best Practices (2019). The Mineral Resource Estimate was undertaken by Eugene Puritch, P.Eng., FEC, CET of P&E Mining Consultants Inc. of Brampton, Ontario. The effective date of this Mineral Resource Estimate is January 21, 2021.

14.1.1 Database

All drilling data was provided by Cadillac Ventures in the form of Microsoft Excel files, drill logs and assay certificates. Eighty-nine (89) drill cross sections were developed on a local grid looking east on an azimuth of 90° on a 50 ft (15 m) spacing named 8,350-E to 12,750-E. A GEOVIA GEMS™ database was provided by the client containing 1,455 diamond drill holes of which 324 were drilled from surface and 1,131 were drilled from underground. Of these drill holes, 202 surface and 993 underground drill holes were utilized in the Mineral Resource Estimate. The remaining data were not in the area that was modeled for this Mineral Resource Estimate. Drill hole plans are shown in Appendix A.

The database was validated in GEMS™ with only minor corrections required. The assay table of the database contains 22,520 values for Cu, 21,702 for Ni, 2,832 for Au, Pt, Pd and 5,856 for Ag. All drill hole collar, downhole survey and interval data are expressed in imperial units and grid coordinates are in a local system. Assays are expressed as percent (%) for Cu and Ni while in g/t for Au, Pt, Pd and Ag.

14.1.2 Data Verification

Verification of assay data entry was performed on 5,490 assay intervals. A few data entry errors were observed and corrected. The 5,490 verified intervals were checked against assay lab certificates from ALS Chemex of Vancouver, B.C., ACME Analytical Laboratories Ltd. of Vancouver, B.C., Bondar Clegg & Company Ltd. of Vancouver, B.C. and XRAL Laboratories of Don Mills, Ont. The checked assays represented 60% of the data to be used for the Mineral Resource Estimate and approximately 25% of the entire database.

14.1.3 Domain Interpretation

Domain boundaries were determined on drill hole sections from lithology, structure and NSR values. Three domains were developed named Main, Hanging Wall and Footwall. These domains were created with computer screen digitizing on drill hole sections in GEMS™ by the authors of this report. The outlines were influenced by the selection of mineralized material above an NSR value of $CDN 60/tonne that demonstrated good zonal continuity along strike and down dip. In a very few cases mineralization below an NSR value of $CDN 60/tonne was included for the purpose of maintaining zonal continuity. Smoothing was utilized to remove obvious jogs and dips in the domains and incorporated a minor addition of Inferred mineralization. This exercise allowed for easier domain creation without triangulation errors from solids validation.

On each section, polyline interpretations were digitized from drill hole to drill hole but not extended more than 250 ft (75 m) into untested territory. Minimum constrained true width for interpretation was 6 ft (1.8 m). The interpreted polylines from each section were "wireframed" in GEMS™ into 3-dimensional (3-D) domains. The resulting solids (domains) were used for statistical analysis, grade interpolation, rock coding and Mineral Resource reporting purposes. See Appendix B.

14.1.4 Rock Code Determination

The rock codes used for the Mineral Resource model were derived from the mineralized domain solids. The list of rock codes used follows:

Rock Code	Description
0	Air
10	Hanging Wall Zone
20	Main Zone
30	Footwall Zone
40	East Zone (Subset of Main Zone)
50	Deep Plug Zone (Subset of Main Zone)
99	Waste Rock

14.1.5 Composites

Length weighted composites were generated for the drill hole data that fell within the constraints of the above-mentioned domains. These composites were calculated for Cu, Ni, Au, Pt, Pd and Ag over 5.0 ft (1.5 m) lengths starting at the first point of intersection between assay data hole and hanging wall of the 3-D zonal constraint. The compositing process was halted upon exit from the footwall of the aforementioned constraint. Un-assayed intervals were assigned a ½ assay detection limit value. Any composites calculated that were less than 1.0 ft (0.3 m) in length, were discarded so as to not introduce a short sample bias in the interpolation process. The composite data were transferred to GEMS™ extraction files for the grade interpolation as X, Y, Z, Cu, Ni, Au, Pt, Pd and Ag files.

14.1.6 Grade Capping

Grade capping was investigated on the raw assay values in the combined domains to ensure that the possible influence of erratic high values did not bias the database. Extraction files were created for constrained Cu and Ni data within each mineralized domain. From these extraction files, log-normal histograms were generated. The grade capping values are provided in Table 14.1. Refer to Appendix C for histogram graphs.

14.1.7 Variography

Variography was carried out on the constrained domain composites within the three domains in the deposit model. All mineralized domains exhibited good directional variography for Cu, however, due to lower population densities, Ni, Au, Pt, Pd and Ag yielded only omnivariograms. Refer to Appendix D for variograms.

TABLE 14.1
THIERRY GRADE CAPPING VALUES

Element	Capping Value	Number of Assays Capped	Cumulative Percent for Capping	Raw Coefficient of Variation	Capped Coefficient of Variation
Main Zone					
Cu	15%	13	99.8	0.99	0.93
Ni	1.5%	14	99.8	1.00	0.86
Au	1 g/t	3	98.8	1.37	1.24
Pt	0.9 g/t	1	99.6	2.82	1.20
Pd	2 g/t	3	98.8	1.29	1.18
Ag	60 g/t	12	99.1	1.60	1.21
Hanging Wall Zone					
Cu	No Cap	0	100	1.23	1.23
Ni	2%	2	99.8	1.19	1.14
Au	1.5 g/t	1	97.1	2.78	1.33
Pt	1.5 g/t	1	97.1	1.48	1.14
Pd	No Cap	0	100	0.98	0.98
Ag	80 g/t	1	98.9	1.45	1.28
Footwall Zone					
Cu	15%	1	99.8	1.12	1.10
Ni	1%	5	99.2	1.08	0.84
Au	No Cap	0	100	1.50	1.50
Pt	No Cap	0	100	1.50	1.50
Pd	0.8 g/t	3	87.5	1.55	1.21
Ag	No Cap	0	100	2.07	2.07

TABLE 14.1 THIERRY GRADE CAPPING VALUES					
Element	**Capping Value**	**Number of Assays Capped**	**Cumulative Percent for Capping**	**Raw Coefficient of Variation**	**Capped Coefficient of Variation**
East Zone					
Cu	No Cap	0	100	0.68	0.68
Ni	No Cap	0	100	0.55	0.55
Au	No Cap	0	100	0.77	0.77
Pt	No Cap	0	100	0.81	0.81
Pd	No Cap	0	100	0.95	0.95
Ag	No Cap	0	100	2.46	2.46
Deep Plug Zone					
Cu	No Cap	0	100	0.47	0.47
Ni	No Cap	0	100	0.37	0.37
Au	No Cap	0	100	0.51	0.51
Pt	No Cap	0	100	0.62	0.62
Pd	No Cap	0	100	0.52	0.52
Ag	No Cap	0	100	1.07	1.07

14.1.8 Bulk Density

The bulk density used for the Mineral Resource model was derived from measurements of test work performed by ALS Chemex of Don Mills, Ontario and AGAT Laboratories of Mississauga Ontario. Representative samples obtained by P&E of the mineralized zones of the deposit were utilized. The average bulk density from the 27 samples collected was calculated to be 10.4 ft^3/ton or 3.07 t/m^3.

14.1.9 Block Modeling

The Mineral Resource model was divided into a 3-D block model framework. The block model has 21,830,040 blocks that are 15 ft (4.57 m) in the X direction, 15 ft (4.57 m) in the Y direction and 15 ft (4.57 m) in the Z direction. There were 306 columns (X), 246 rows (Y) and 290 levels. The block model was not rotated. Separate block models were created for rock type, density, volume percent, class, Cu, Ni, Au, Pt, Pd and Ag. Previously mined blocks were removed from the block model.

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

The Cu, Ni, Au, Pt, Pd and Ag composites were extracted from the Microsoft Access database composite table into separate files for each Mineralized Zone. Inverse distance squared ("ID^2") grade interpolation was utilized. There were three interpolation passes performed on each domain for each element for the Measured, Indicated and Inferred classifications. The resulting Cu and NSR blocks can be seen on the block model cross-sections and plans in Appendix E and F. The grade blocks within all domains were interpolated using the parameters listed in Table 14.2 and Table 14.3.

TABLE 14.2
THIERRY CU BLOCK MODEL INTERPOLATION PARAMETERS

Profile*	Dip Dir. (°)	Strike (°)	Dip (°)	Dip Range ft (m)	Strike Range ft (m)	Across Dip Range ft (m)	Max No. per Hole	Min No. Samples	Max No. Samples
Main Measured	0	90	-52	90 (27)	90 (27)	25 (8)	2	5	20
Main Indicated	0	90	-52	140 (43)	140 (43)	40 (12)	2	3	20
Main Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20
HW Measured	0	90	-52	80 (24)	65 (20)	25 (8)	2	5	20
HW Indicated	0	90	-52	120 (37)	100 (30)	40 (12)	2	3	20
HW Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20
FW Measured	0	90	-52	80 (24)	80 24)	20 (6)	2	5	20
FW Indicated	0	90	-52	125 (38)	125 (38)	30 (9)	2	3	20
FW Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20

***Note:* *** *HW = hanging wall, FW = footwall*

TABLE 14.3
THIERRY NI, AU, PT, PD & AG BLOCK MODEL INTERPOLATION PARAMETERS

Profile*	Dip Dir. (°)	Strike (°)	Dip (°)	Dip Range ft (m)	Strike Range ft (m)	Across Dip Range ft (m)	Max No. per Hole	Min No. Samples	Max No. Samples
Ni All Indicated	0	90	-52	50 (15)	50 (15)	25 (8)	2	3	20
Ni All Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20
Au Indicated	0	90	-52	200 (61)	200 (61)	50 (15)	2	3	20
Au Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20
Pt Indicated	0	90	-52	175 (53)	175 (53)	50 (15)	2	3	20
Pt Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20
Pd Indicated	0	90	-52	165 (50)	165 (50)	50 (15)	2	3	20
Pd Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20
Ag Indicated	0	90	-52	75 (23)	75 (23)	35 (11)	2	3	20
Ag Inferred	0	90	-52	1,000 (305)	1,000 (305)	500 (152)	2	1	20

14.1.10 Mineral Resource Classification

For the purpose of this Mineral Resource Estimate, classifications of all interpolated grade blocks were determined from the Cu interpolations for Measured, Indicated and Inferred due to Cu being the dominant revenue producing element. See block model classification cross-sections and plans in Appendix G.

14.1.11 Thierry Mineral Resource Estimate

The Mineral Resource Estimate was derived from applying an NSR cut-off value to the block model and reporting the resulting tonnes and grade for Potentially Economic Portions of the Mineral Resources. The following calculations demonstrate the rationale supporting the NSR cut-off value that determines the potentially economic portion of the mineralized domains.

<u>NSR Cut-off Value Calculation Components (All currency in Canadian dollars unless stated otherwise)</u>

\$CDN/\$US (Exchange Rate)	US\$0.75 = CDN\$1.00
Cu Price	US \$3.75/lb (approx Dec 31/20 two-yr trailing average)
Ni Price	US \$6.25/lb (approx Dec 31/20 two-yr trailing average)
Au Price	US \$1,600/oz (approx Dec 31/20 two-yr trailing average)
Pt Price	US \$900/oz (approx Dec 31/20 two-yr trailing average)
Pd Price	US \$1,600/oz (approx Dec 31/20 two-yr trailing average)
Ag Price	US \$18.5/oz (approx Dec 31/20 two-yr trailing average)
U/G Mining Cost (4,000 tpd)	\$40/tonne mined
Process Cost (4,000 tpd)	\$15/tonne processed
General/Administration	\$5/tonne processed
Cu Flotation Recovery	90%
Ni Flotation Recovery	50%
Au Flotation Recovery	50%
Pt Flotation Recovery	50%
Pd Flotation Recovery	50%
Ag Flotation Recovery	50%
Concentration Ratio	22:1
Cu Smelter Payable	95%
Ni Smelter Payable	65%
Au Smelter Payable	50%
Pt Smelter Payable	50%
Pd Smelter Payable	50%
Ag Smelter Payable	50%
Cu Refining Charges	US \$0.08/lb
Ni Refining Charges	US \$0.50/lb
Au Refining Charges	US \$15/oz
Pt Refining Charges	US \$15/oz
Pd Refining Charges	US \$15/oz
Ag Refining Charges	US \$0.50/oz

Smelter Treatment Charges	US $85/dry tonne ($85/22/0.75 = CDN$5.15/tonne processed)
Concentrate Shipping	$90/tonne ($90/22x1.08 = CDN$4.42/tonne processed)
Moisture Content	8%

This data was derived from metallurgical reports on Thierry and other similar mining operations.

In the anticipated underground mining operation, Mining, Processing and G&A costs combine for a total of ($40 + $15 + $5) = $60/tonne processed which became the NSR/tonne value cut-off. Recovered contributions by Cu, Ni, Au, Pt, Pd and Ag were as follows:

Cu	= [(90% Rec. x 95% Payable x 22.05 lb/t x ($2.75/lb -$0.08/lb)] /0.75	= $67.10/%/tonne.
Ni	= [(50% Rec. x 65% Payable x 22.05 lb/t x ($6.25/lb -$0.50/lb)] /0.75	= $54.93/%/tonne.
Au	= [(50% Rec. x 50% x ($1,600/oz -$10/oz)] /31.1035/0.75	= $17.04/g/tonne.
Pt	= [(50% Rec. x 50% x ($900/oz -$10/oz)] /31.1035/0.75	= $9.54/g/tonne.
Pd	= [(50% Rec. x 50% x ($1,800/oz -$10/oz)] /31.1035/0.75	= $19.18/g/tonne.
Ag	= [(50% Rec. x 90% x ($18.50/oz -$0.50/oz)] /31.1035/0.75	= $0.35/g/tonne.

The resulting Thierry Mineral Resource Estimate can be seen in Table 14.4.

TABLE 14.4
THIERRY MINERAL RESOURCE ESTIMATE AT CDN$60/T NSR CUT-OFF (1-3)

Classification	Tonnes (k)	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)
Measured	3,233	1.65	0.19	0.03	0.03	0.09	4.6
Indicated	5,582	1.66	0.19	0.05	0.05	0.14	3.8
Measured & Indicated	**8,815**	**1.66**	**0.19**	**0.05**	**0.04**	**0.13**	**4.0**
Inferred	14,922	1.64	0.16	0.10	0.07	0.21	6.4

Notes:
1) *Mineral Resources, which are not Mineral Reserves, and do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2) *The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3) *The Mineral Resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*

The sensitivity of the Thierry Mineral Resource to NSR cut-off is demonstrated in Table 14.5.

TABLE 14.5
SENSITIVITY TO THIERRY MINERAL RESOURCE ESTIMATE

NSR Cut-off ($/tonne)	Measured and Indicated							Inferred						
	Tonnes	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)	Tonnes	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)
$100	5,311,883	2.05	0.21	0.06	0.05	0.16	5.1	11,799,332	1.82	0.17	0.12	0.08	0.23	7.2
$95	5,849,090	1.98	0.21	0.05	0.05	0.15	4.9	12,526,894	1.79	0.16	0.11	0.08	0.23	7.1
$90	6,360,964	1.93	0.21	0.05	0.05	0.15	4.8	13,146,085	1.77	0.16	0.11	0.08	0.23	6.9
$85	6,829,824	1.88	0.20	0.05	0.05	0.14	4.7	13,647,414	1.75	0.16	0.11	0.08	0.22	6.8
$80	7,311,543	1.83	0.20	0.05	0.05	0.14	4.5	14,003,035	1.73	0.16	0.11	0.07	0.22	6.7
$75	7,751,521	1.78	0.20	0.05	0.05	0.14	4.4	14,319,409	1.72	0.16	0.11	0.07	0.22	6.6
$70	8,181,889	1.74	0.20	0.05	0.04	0.13	4.3	14,575,168	1.70	0.16	0.11	0.07	0.21	6.6
$65	8,576,163	1.70	0.19	0.05	0.04	0.13	4.3	14,790,246	1.69	0.16	0.10	0.07	0.21	6.5
$60	**8,815,315**	**1.66**	**0.19**	**0.05**	**0.04**	**0.13**	**4.0**	**14,922,094**	**1.64**	**0.16**	**0.10**	**0.07**	**0.21**	**6.4**
$55	9,218,477	1.64	0.19	0.05	0.04	0.12	4.1	15,211,522	1.67	0.16	0.10	0.07	0.21	6.4
$50	9,498,088	1.61	0.19	0.04	0.04	0.12	4.1	15,431,417	1.65	0.16	0.10	0.07	0.20	6.3
$45	9,732,884	1.59	0.19	0.04	0.04	0.12	4.0	15,574,616	1.64	0.16	0.10	0.07	0.20	6.3

14.1.12 Confirmation of Mineral Resource Estimate

As a test of the reasonableness of the Thierry Mineral Resource Estimate, the block model was queried at a 0.01% Cu cut-off grade with blocks in all classifications summed and their grades weight averaged. This average is the average grade of all blocks within the mineralized domains. The values of the interpolated grades for the block model were compared to the length weighted capped average grades and average grade of composites of all samples from within the domain. The results are presented in Table 14.6.

TABLE 14.6
COMPARISON OF WEIGHTED AVERAGE GRADE OF CAPPED ASSAYS AND COMPOSITES WITH THIERRY TOTAL BLOCK MODEL AVERAGE GRADE

Category	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)
Capped Assays	1.60	0.18	0.11	0.10	0.31	6.1
Composites	1.48	0.17	0.11	0.10	0.21	4.8
Block Model	1.52	0.16	0.08	0.06	0.16	5.2

This comparison shows the average grade of all of the blocks in all domains to be similar to the weighted average of all capped assays and composites used for grade estimation.

In addition, a volumetric comparison was performed with the block volume of the model versus the geometric calculated volume of the domain solids.

Block Model Volume = 303,732,184 ft^3
Geometric Domain Volume = 305,252,543 ft^3
Difference = 0.50%

14.2 P&E 2021 K1-1 MINERAL RESOURCE ESTIMATE

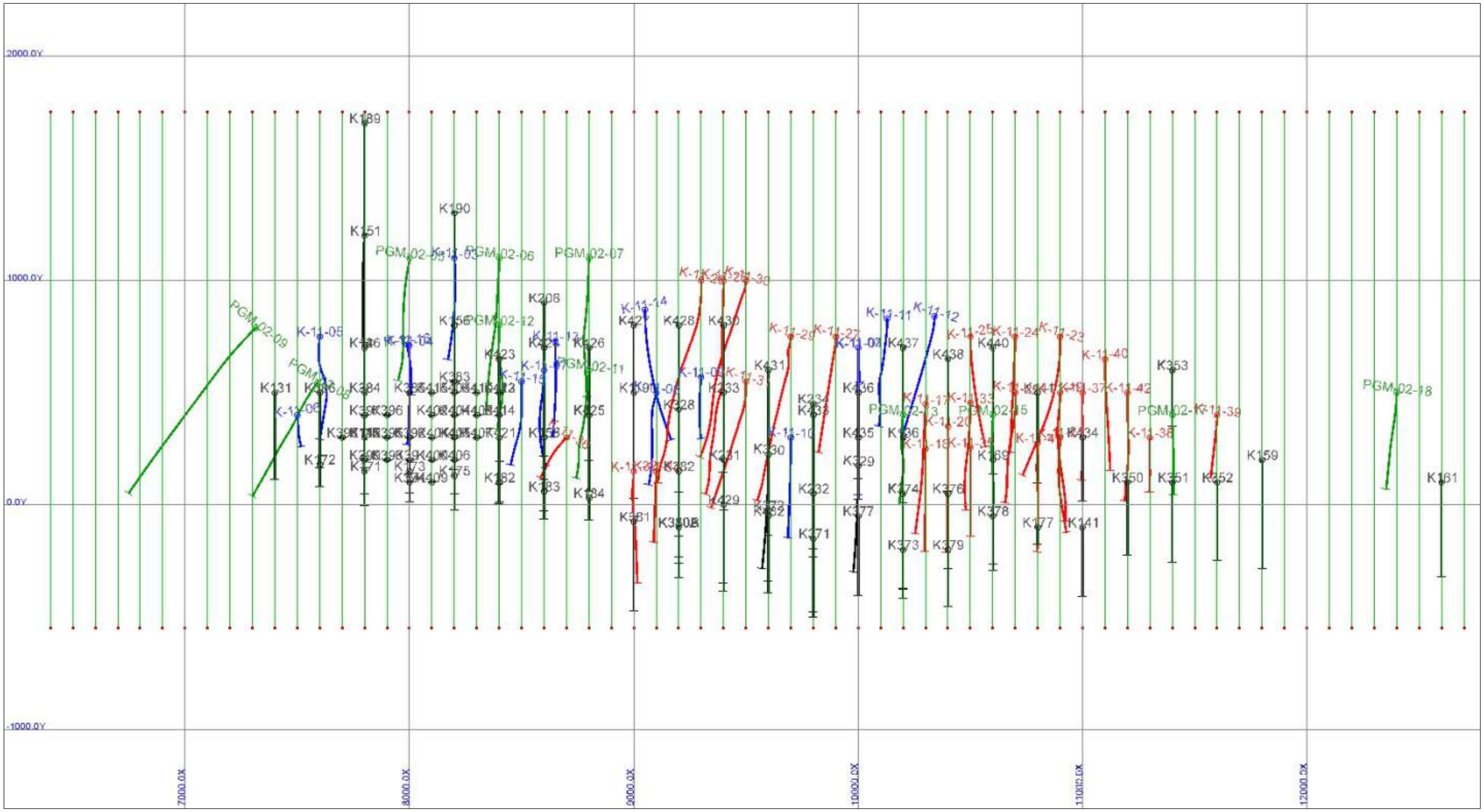
14.2.1 Summary

The K1-1 Mineral Resource Estimate is based entirely on diamond drilling, core sampling and assaying. The Mineral Resource database is based on Imperial measure, consistent with the UMEX drill hole database that originated in 1969, with assay grades in SI. Short tons were converted to metric tonnes for Mineral Resource reporting purposes. The K1-1 drill hole database contains 158 diamond drill holes totaling 96,593.82 ft (29,441.80 m). Drill holes in the K1-1 Mineral Resource area, showing the fill-in drilling, are located in plan in Figure 14.1.

The Mineral Resources for K1-1 were estimated by conventional 3-D computer block modelling using GEMS™ 6.3 modelling software. Mineral Resources have been estimated for copper, nickel, platinum, palladium, gold and silver, with reporting done by net smelter return (“NSR”) cut-off as appropriate for a polymetallic deposit. A preliminary pit shell, with 50° slopes, was created from the Mineral Resource block model and a pit constrained Inferred Mineral Resource reported. P&E’s estimate of open pit operating costs for the K1-1 is $12/tonne. The pit constrained Inferred Mineral Resources for a $12/tonne NSR cut-off value are estimated at

53.6 million tonnes averaging 0.38% Cu, 0.10% Ni, 0.05 g/t Pt, 0.14 g/t Pd, 0.02 g/t Au and 1.8 g/t Ag.

FIGURE 14.1 K1-1 MINERAL RESOURCE AREA DRILL HOLE LOCATION PLAN



Note: *Scale in feet*

Legend

	Drilling Campaign		
	UMEX Series		2011 Series
	2002 Series		2011 Fill in Series

Source: *P&E (2012)*

14.2.2 Interpretation, Wireframes and Cut-off

The K1-1 Deposit has been subdivided geologically into seven mineralized zones, denoted from "A" to "G", for the purpose of Mineral Resource estimation (Figure 14.2). The seven mineral wireframes were constructed at an NSR cut-off value of $12/tonne. The NSR values were based on approximate 24 month trailing average metal prices as of December 31, 2020 and calculated from P&E's assessment of open pit mining costs and processing costs and metal recoveries from previous processing at the Thierry Deposit. These NSR values were applied to individual assays and wireframed on screen. The NSR calculation included historical recoveries for the Thierry process plant, smelter recoveries and payables with smelting costs based on the Thierry Deposit concentration ratio.

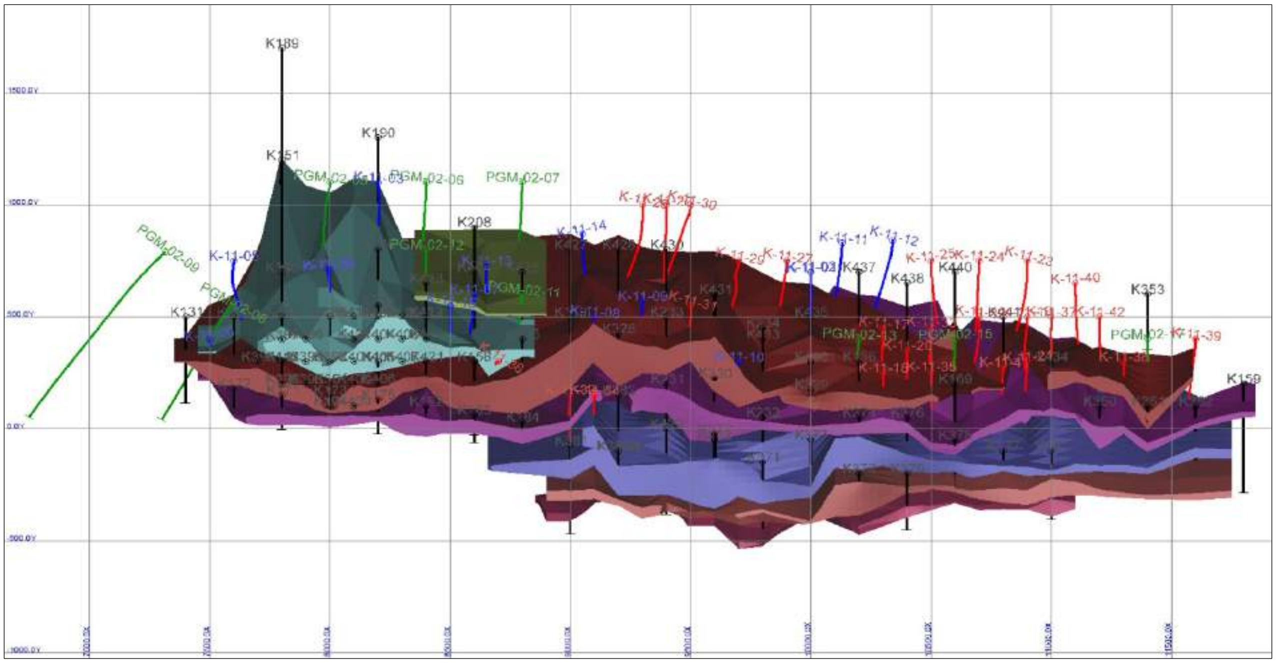
NSR ($/tonne) = (Cu% x $67.10) + (Ni% x $54.93) + (Ag g/t x $0.35) + (Au g/t x $17.04) + (Pt g/t x $9.54) + (Pd g/t x $19.18) - $9.57/tonne (Smelter related treatment/shipping/penalty costs)

North-south vertical cross sections were generated at 100 ft (30 m) intervals in GEMS™ consistent with the 200 ft (61 m) drill hole section spacing and areas filled in to 50 ft (15 m) to 100 ft (30 m). The wireframe was developed on these sections from polylines enclosing drill hole core samples with assay NSR values ≥$12/tonne. Where Au, Ag, Pt, and Pd assays were lacking the NSR was based on Cu and Ni only. This may somewhat underestimate the NSR values, however, P&E notes that Au, Ag, Pt and Pd account for only approximately 12% of the expected revenue therefore the lack of these assays has a minor impact on the wireframe boundary.

The wireframes extend on EW strike up to 4,530 ft (1,380 m) in length and in NS surface projection for up to 1,745 ft (532 m). The wireframes extend from bedrock surface variously to a depth up to 1,465 ft (446 m). Total volume of the wireframes is 904 million ft^3 (26 million m^3) or approximately 88 million tons (80 million tonnes) at a bulk density of 0.09721 t/ft^3 (3.12 t/m^3). Figure 14.2 shows a plan view of the Mineral Resource domain wireframes.

The Mineral Resource domains are reasonably continuous at the $12/tonne NSR cut-off value, however, to preserve zone continuity locally some low grade material and non-assayed intervals (at zero grade) were incorporated as internal dilution. This dilution accounts for approximately 19% of the global wireframe material or approximately 18% of the wireframe Mineral Resource within the constraining pit shell.

FIGURE 14.2 3-D PERSPECTIVE VIEW OF THE K1-1 MINERAL RESOURCE WIREFRAMES



Note: *Scale in feet*
Source: *P&E (2012)*

14.2.3 Drill Hole Database

The K1-1 Mineral Resource drill hole database consists of 158 diamond drill holes totalling 96,593.82 ft (29,441.80 m) of which 25 are vertical short holes for 4,921.00 ft (1,499.92 m). Drilling is generally on 200 ft (60 m) cross-sections and at ±95 ft (29 m) to ±275 ft (84 m) pierce points along dip with intercepts wider at depth in part due to fanned drilling. At the west end of the deposit, collars for vertical holes are on a 100 ft (30 m) by 100 ft (30 m) grid. Drill hole density is somewhat wide for the grade continuity as indicated by variography.

The drill core sampling interval, nominally at 5 ft (1.5 m), is appropriate to the deposit scale and mineralization continuity.

P&E's site visit confirmed only a small range in elevation difference, for the 16 holes surveyed for elevation, indicates that the terrain at the K1-1 Deposit is relatively flat. Since drill hole collar elevation data is generally lacking, the topographic surface was set at 0 ft elevation i.e. 0 RL and generated as a flat surface. All drill hole collars were set at this elevation for the purpose of Mineral Resource estimation. In the authors' of this Technical Report section opinion, the assumption of a flat topographic surface will not introduce substantial error in the Mineral Resource Estimate. Down hole surveys were performed for 55 of the 158 drill holes (47% by length), acid dip tests (no azimuth deviation) for 65 drill holes (44%) and no surveys for 38 drill holes (9%). As such azimuth deviation is unknown for 103 drill holes (53%). The deviation for surveyed drill holes is significant indicating that there is uncertainty in the drill core sample locations for more than half of the drilling.

14.2.4 Assay Grade Distributions

P&E notes that copper and nickel assays are available for all of the drill core samples captured within the mineral wireframes, however, only 68% of these intervals have Au, Ag, Pt and Pd assays. In order to provide for Au, Ag, Pt and Pd grade interpolation and allow for the economic evaluation of the Mineral Resources, the missing Au, Ag, Pt and Pd assays were generated in the database by regression analysis from the available 2002 and 2011 Au, Ag, Pt and Pd assay data. Regression was only applied to intervals with existing Cu assays. Various metal correlation relationships were examined and P&E used the regression formulae with the highest correlation coefficients that also made geologic sense (Table 14.7). The Au, Ag, Pt and Pd "assay" data so generated were reviewed for implausible values and adjusted where necessary. Inherent in this process is the assignment of a low base level of Au, Ag, Pt and Pd for very low copper values; however, the impact on Mineral Resource estimation is negligible.

TABLE 14.7
SUMMARY OF METAL ASSAY REGRESSIONS UTILIZED

Regression	R2	Polynomial Formula
Pd versus Cu	0.3439	Pd = -0.0228 Cu_2 + 0.3017 Cu + 0.0335
Au versus Cu	0.1487	Au = -0.00009 Cu_2 + 0.0672 Cu + 0.0023
Ag versus Cu	0.8039	Ag = 0.1583 Cu_2 + 4.2293 Cu + 0.1522
Pt versus Pd	0.5588	Pt = -0.1525 Pd_2 + 0.328 Pd + 0.0066

P&E understands that for a period of time, UMEX had a policy of assaying every second interval that resulted in incomplete assaying for a number of drill holes and intercepts of mineralization within the Mineral Resource. P&E reviewed the UMEX intercepts and for missing intervals, the adjacent assays were averaged to provide a value for the unassayed interval. This affected 71 intervals in 11 UMEX holes. For later drilling, the explicit missing and implicit missing assay intervals were assigned zero grade for Mineral Resource estimation under the assumption that any visible base metal mineralization would have been sampled and assayed. Where large intervals were not assayed or the entire hole not assayed, the wireframe was modeled to exclude the gaps or the hole was ignored for Mineral Resource estimation.

14.2.5 Grade Capping

Assay grade distributions are somewhat positively skewed. P&E notes that the coefficient of variation is low and grade distributions not strongly skewed but there are some apparent high grade outliers indicating that grade capping is warranted.

Histograms were prepared to examine grade distributions of assays for each metal in each zone and grade capping set to eliminate the outliers evident in the graphs. See Appendix H. Log-probability plots for all assays in the zones were also prepared to support grade capping conclusions from the copper and nickel histograms. Table 14.8 lists the grade capping levels, number of values capped, and percent capped. The capping levels of Table 14.8 were applied to all zones.

TABLE 14.8 **GRADE CAPPING LEVELS**			
Metal	**Capping Level**	**Number Capped**	**Percent Capped**
Cu	2.00%	7	0.13
Ni	0.60%	38	0.68
Pt	0.15 g/t	24	0.43
Pd	0.50 g/t	23	0.41
Au	0.15 g/t	12	0.22
Ag	7.00 g/t	7	0.14

14.2.6 Bulk Density

To convert volume to tonnes, a bulk density of 0.09721 tons/ft^3, equivalent to a bulk density of 3.12 t/m^3, was applied uniformly throughout the deposit based on limited bulk density testing by former property operators and P&E.

14.2.7 Assay Compositing

Assay composites at five foot lengths were generated down hole by length weighting the assays captured by GEMS™ in the domain wireframes. The 5 ft (1.5 m) length is at the 95.2 percentile of the sample length distribution. Table 14.9 presents summary statistics for wireframe capped assays and composites.

The Mineral Resource block model is oriented with X axis at 090° azimuth, i.e. non-rotated, and has block dimensions at 50 ft (15 m) EW x 10 ft (3 m) NS x 20 ft (6 m) vertical. Block dimensions take into account the drill hole spacing on 200 ft (60 m) sections, zone widths, and bench heights based on a minimum of 20 ft (6 m). Since the boundaries of the zones are locally within several metres and within the Mineral Resource block dimensions of 10 ft (3 m), three partial–percent models were created in GEMS™ so that varied percentages of the zones and waste could be coded into the parent blocks.

TABLE 14.9 **MINERAL RESOURCE CAPPED ASSAYS AND COMPOSITE STATISTICS**							
Zone Assays Statistics	**Length**	**Cu (%)**	**Ni (%)**	**Au (g/t)**	**Pt (g/t)**	**Pd (g/t)**	**Ag (g/t)**
Count	5,630	5,630	5,630	5,630	5,630	5,630	5,630
Sum (m)	8,592.9	-	-	-	-	-	-
Minimum (m)	0.15	0.001	0.001	0.001	0.003	0.001	0.01
Maximum (m)	7.62	2.000	0.600	0.150	0.150	0.500	7.00
Average (m)	1.52	0.376	0.103	0.025	0.050	0.140	1.81
Coefficient of Variation	0.18	0.570	0.634	0.672	0.422	0.508	0.56

TABLE 14.9 MINERAL RESOURCE CAPPED ASSAYS AND COMPOSITE STATISTICS							
Zone Assays Statistics	**Length**	**Cu (%)**	**Ni (%)**	**Au (g/t)**	**Pt (g/t)**	**Pd (g/t)**	**Ag (g/t)**
Composite Statistics	Length	Cu%	Ni%	Au g/t	Pt g/t	Pd g/t	Ag g/t
Count	5,815	5,815	5,815	5,815	5,815	5,815	5,815
Sum (m)	8,797.4	-	-	-	-	-	-
Minimum (m)	0.38	0.000	0.000	0.000	0.000	0.000	0.00
Maximum (m)	1.52	2.000	0.600	0.150	0.150	0.500	7.00
Average (m)	1.51	0.366	0.100	0.024	0.049	0.136	1.76
Coefficient of Variation	0.06	0.565	0.622	0.655	0.437	0.511	0.56

Note: *Composites total length exceeds assays total length due to incorporation of implicit missing assay intervals (dilution) not accounted for in the assay table and the generation of extra composites from long assay intervals.*

14.2.8 Variography

Variography, carried out for the Mineral Resource Estimate, guided the interpolation search strategy. A linear semi-variogram (variogram) of the 5 ft (1.5 m) Mineral Resource composites was prepared down hole to assess the nugget effect which was found to be relatively high at 46%. Three-dimensional variography, using spherical and nested spherical models, was carried out on strike at~090°/0°, on dip 360°/-55° and transverse to the dip at 360°/+35°. The strike and dip variograms are not robust and show ranges less than the 200 ft (60 m) drill hole spacing. Table 14.10 shows results of the variography. See Appendix I.

TABLE 14.10 VARIOGRAPHY RESULTS						
Variogram	**Nugget**	**%**	**C1**	**Range ft (m)**	**C2**	**Range ft (m)**
Down Hole	0.019752	46	0.008859	18 (5.5)	0.043068	85 (26)
090°/0°	0.003432	10	0.033816	111 (34)	-	-
360°/-55°	0.020110	43	0.024184	43 (13)	0.003285	142 (43)
360°/+35°	0.018233	44	0.011659	23 (7)	0.011958	101 (31)

14.2.9 Block Model Grade Interpolation

Search Strategy and Grade Interpolation

Variography results guided the interpolation search strategy. The search ellipse main axis orientation is 085°/0° along the strike of the zones. Intermediate axis is 355° and minor axis vertical. Since the mineralization appears to dip~20° north in the west portion of the Deposit, is flat in the centre and dips ~35° south in the east, the search ellipse was similarly oriented in these sections of the block model. The ID^2 interpolation was carried out in three passes (Table 14.11).

The third pass was designed to fill the wireframe. The number of blocks interpolated varied in each pass with 60% populated in the first pass and 99% populated after two passes.

The ID^2 method is reasonable since the nugget effect is moderate and some smoothing is desirable. In addition, the variography is not particularly robust due to few drill holes per zone and the generally low number of samples per zone that would impact on the use of kriging as an alternative method. Short composites of 5 ft (1.5 m), composite sample minimums and maximums, and expanded passes were adopted to avoid over-smoothing and preserve local grade variability. P&E examined, by means of cell de-clustering, whether sample clustering in this area could affect ID interpolation and found no significant impact.

TABLE 14.11
INTERPOLATION PARAMETERS AND SEARCH DISTANCES

Parameter	Pass 1	Pass 2	Pass 3
Minimum Composites	4	2	1
Maximum Composites	12	12	12
Maximum Composites from One Hole	3	N/A	N/A
Ellipse Search Distance X ft (m) 90°/0°	200 (60)	400 (120)	600 (180)
Ellipse Search Distance Y ft (m) 360°/-55°	200 (60)	400 (120)	600 (180)
Ellipse Search Distance Z ft (m) 360°/+35°	25 (8)	30 (9)	200 (60)

14.2.10 Mineral Resource Classification

In P&E's opinion, the level of drilling, assaying and exploration work completed is sufficient to show that the K1-1 copper-nickel deposit has the size and grade to indicate reasonable potential for economic open pit extraction and thus qualify it as a Mineral Resource under CIM definition standards. Mineral Resources were classified as Inferred based on the wide drill hole spacing and data limitations inherent in the older UMEX drilling including lack of collar elevation and down hole surveys and lack of assaying for Au, Ag, Pt and Pd for 32% of the sampling in the Mineral Resource.

14.2.11 Block Model Inventory

The K1-1 Deposit is a near surface, low grade polymetallic deposit for which the economic mining potential rests with open pit mining. A NSR block model was created from the interpolated grades in the block model. The NSR value calculation is as follows:

<u>NSR Cut-off Value Calculation Components (All currency in Canadian dollars unless stated otherwise)</u>

\$CDN/\$US (Exchange Rate)	US\$0.75 = CDN\$1.00
Cu Price	US \$3.75/lb (approx Dec 31/20 two-yr trailing average)
Ni Price	US \$6.25/lb (approx Dec 31/20 two-yr trailing average)
Au Price	US \$1,600/oz (approx Dec 31/20 two-yr trailing average)
Pt Price	US \$900/oz (approx Dec 31/20 two-yr trailing average)
Pd Price	US \$1,600/oz (approx Dec 31/20 two-yr trailing average)

Ag Price	US $18.5/oz (approx Dec 31/20 two-yr trailing average)
Pit Mining Cost	$2.50/tonne mined
Process Cost (15,000 tpd)	$10/ tonne processed
General/Administration	$2/tonne processed
Cu Flotation Recovery	90%
Ni Flotation Recovery	50%
Au Flotation Recovery	50%
Pt Flotation Recovery	50%
Pd Flotation Recovery	50%
Ag Flotation Recovery	50%
Concentration Ratio	22:1
Cu Smelter Payable	95%
Ni Smelter Payable	65%
Au Smelter Payable	50%
Pt Smelter Payable	50%
Pd Smelter Payable	50%
Ag Smelter Payable	50%
Cu Refining Charges	US $0.08/lb
Ni Refining Charges	US $0.50/lb
Au Refining Charges	US $15/oz
Pt Refining Charges	US $15/oz
Pd Refining Charges	US $15/oz
Ag Refining Charges	US $0.50/oz
Smelter Treatment Charges	US $85/dry tonne ($85/22/0.75 = CDN$5.15/tonne processed)
Concentrate Shipping	$90/tonne ($90/22x1.08 = CDN$4.42/tonne processed)
Moisture Content	8%

This data was derived from metallurgical reports on K1-1 and other similar mining operations.

In the anticipated K1-1 open pit mining operation, Processing and G&A costs combine for a total of ($10 + $2) = $12/tonne processed which became the NSR/tonne value cut-off. Recovered contributions by the Cu, Ni, Au, Pt, Pd and Ag were as follows:

Cu	= [(90% Rec. x 95% Payable x 22.05 lb/t x ($2.75/lb -$0.08/lb)] /0.75	= $67.10/%/tonne
Ni	= [(50% Rec. x 65% Payable x 22.05 lb/t x ($6.25/lb -$0.50/lb)] /0.75	= $54.93/%/tonne
Au	= [(50% Rec. x 50% x ($1,600/oz -$10/oz)] /31.1035/0.75	= $17.04/g/tonne
Pt	= [(50% Rec. x 50% x ($900/oz -$10/oz)] /31.1035/0.75	= $9.54/g/tonne
Pd	= [(50% Rec. x 50% x ($1,800/oz -$10/oz)] /31.1035/0.75	= $19.18/g/tonne
Ag	= [(50% Rec. x 90% x ($18.50/oz -$0.50/oz)] /31.1035/0.75	= $0.35/g/tonne.

14.2.12 Mineral Resource Reporting

The NSR block model was exported to open pit optimization software to define a constraining pit shell. The constraining pit shell was based on parameters listed in Table 14.12 and the metal prices from Section 14.2.11.

Table 14.12 Pit Optimization Parameters	
Pit Slopes	50°
Overburden Stripping	\$2.00/tonne
Mining Cost (mineralized rock)	\$2.50/tonne
Waste Rock Stripping	\$2.50/tonne
Process Cost	\$10.00/tonne
G&A	\$2.00/tonne
Breakeven Cut-off Value	\$12.00/tonne

Ramp design and pit floor modifications were not done to finalize the pit at this Mineral Resource estimation stage. Such work has an impact on the stripping ratio and on the pit constrained Mineral Resources. In addition, there is no geotechnical information available to confirm the pit slopes and their modification will also impact on the stripping ratio and the pit constrained Mineral Resources. Material in the block modeled mineral wireframes lying outside the pit shells does not show open pit economic potential and the grade is not high enough to support underground mining and thus is not considered to be Mineral Resources under CIM definitions.

The resulting constraining pit shell was used to report Inferred Mineral Resources as shown in Table 14.13 and Figure 14.3 and Figure 14.4.

Table 14.13
K1-1 Pit Constrained Inferred Mineral Resource Estimate (1-9)

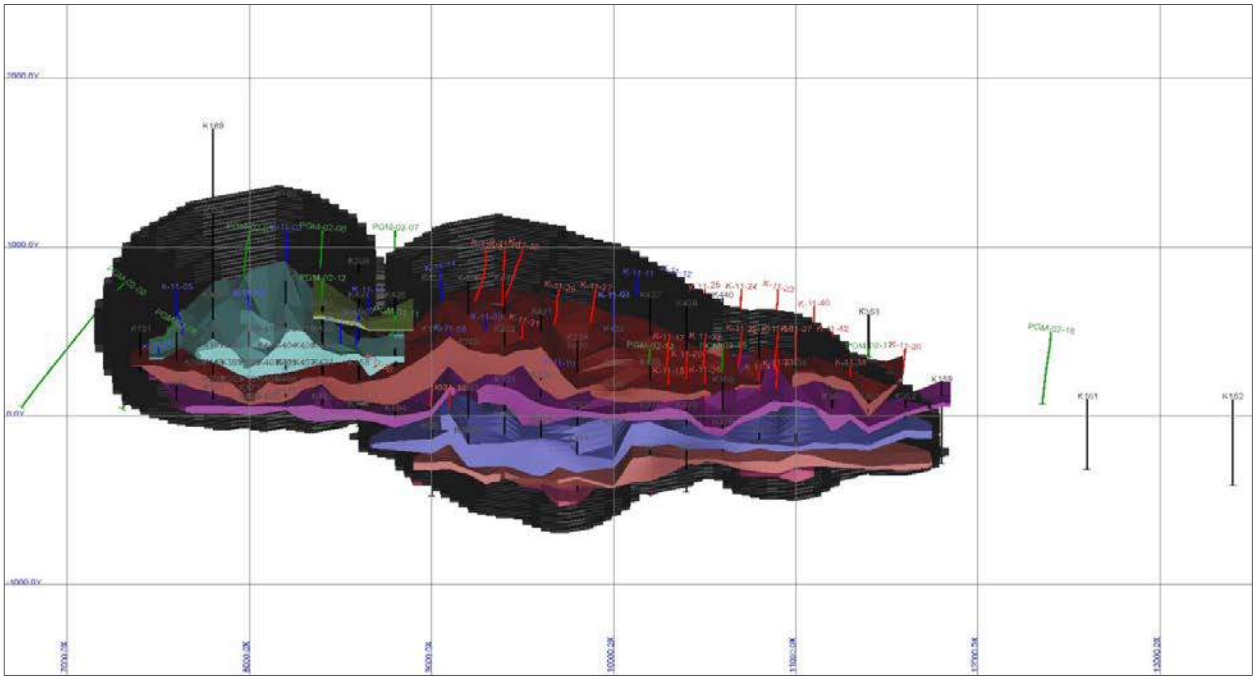
Cut-off NSR (\$/tonne)	Tonnes (k)	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)
\$12	53,614	0.38	0.10	0.03	0.05	0.14	1.8

Notes:

1) *CIM definitions (2014) and Best Practices (2019) were followed for Mineral Resources.*
2) *Mineral Resources are estimated by conventional 3-D block modelling based on wireframing at a \$12/tonne NSR cut-off value and ID^2 grade interpolation.*
3) *Metal prices for the estimate are: US\$3.75/lb Cu, US\$6.25/lb Ni, US\$900/oz Pt, US\$1,600/oz Pd, US\$1,600/oz Au, US\$18.50/oz Ag, based on Dec 31/2020 two-year trailing averages.*
4) *A uniform bulk density of 3.12 t/m^3 has been applied for volume to tonnes conversion.*
5) *The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
6) *Classification of Inferred Resources is based on wide drill hole spacing, lack of collar and down surveys for UMEX and 2002 series drilling and the lack of Au, Ag, Pt and Pd assays for more than 50% the sample data in the Mineral Resource. Regression based on available assays was used to generate PGM/PM values for the Mineral Resource Estimate.*
7) *The Mineral Resource Estimate was determined within a constraining pit shell with 50 degree slopes utilizing mining costs of \$2.50/tonne for mineralized material, \$2.50/tonne for waste rock, and \$2.00/tonne for overburden. The pit constrained Mineral Resource is estimated below surface to a depth of 268 m.*
8) *Costs used to determine the \$12/tonne NSR Mineral Resource cut-off value were processing at \$10/tonne and G&A at \$2/tonne.*

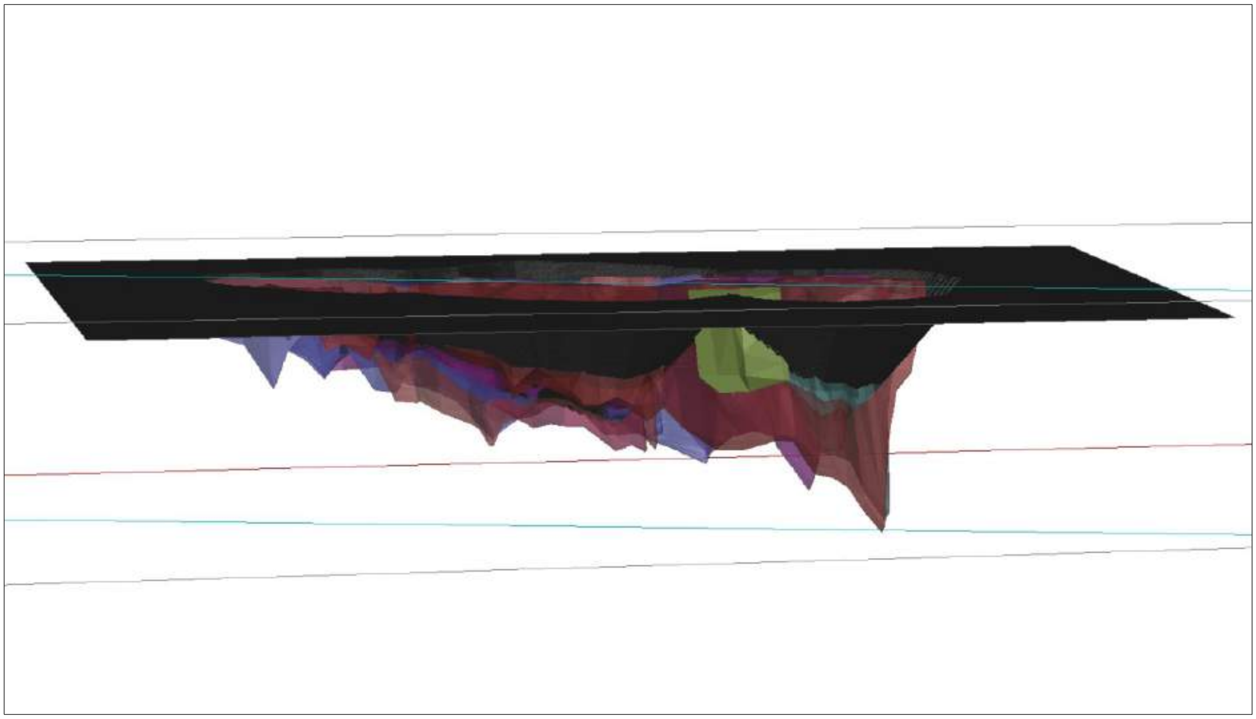
9) *Overall payable metal in the NSR calculation were 86% Cu, 33% Ni and 25% for Ag, Au, Pt & Pd.*

FIGURE 14.3 PLAN VIEW OF OPTIMIZED PIT SHELL AND MINERAL ZONE WIREFRAMES



Note: *Scale in Feet*
Source: *P&E (2012)*

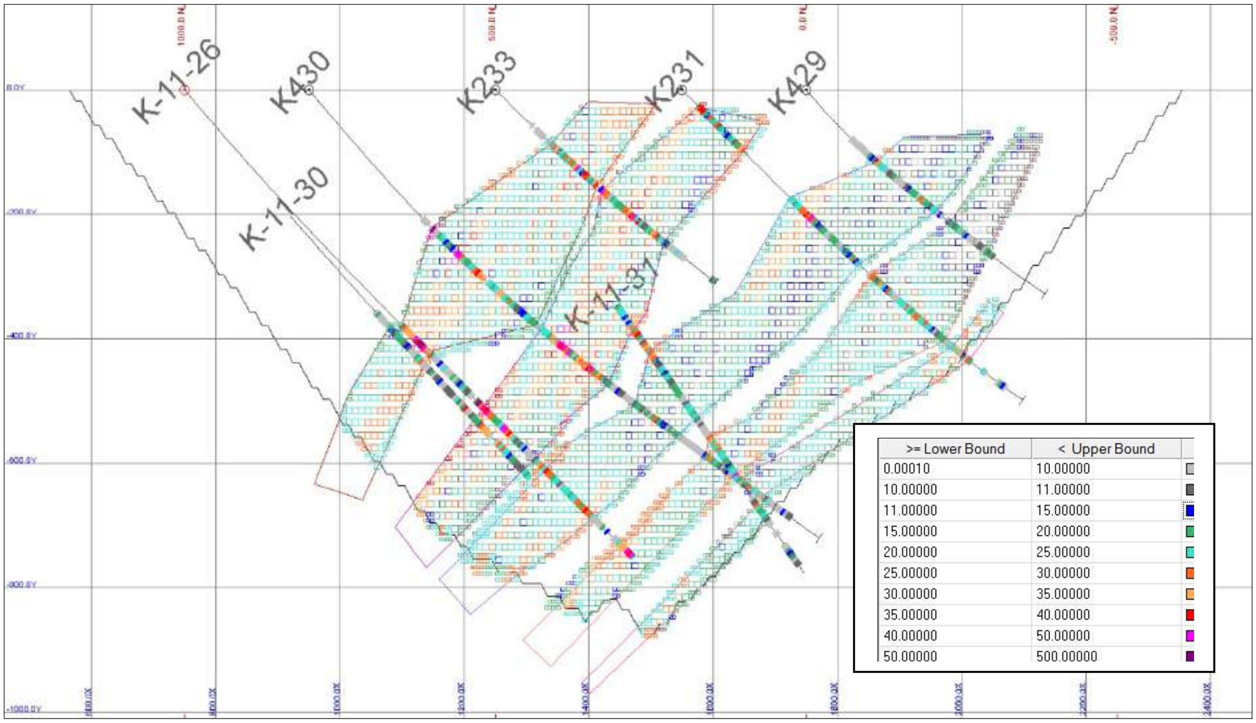
FIGURE 14.4 3-D VIEW OF CONSTRAINING PIT SHELL AND MINERAL WIREFRAMES-LOOKING SOUTH



Source: *P&E (2012)*

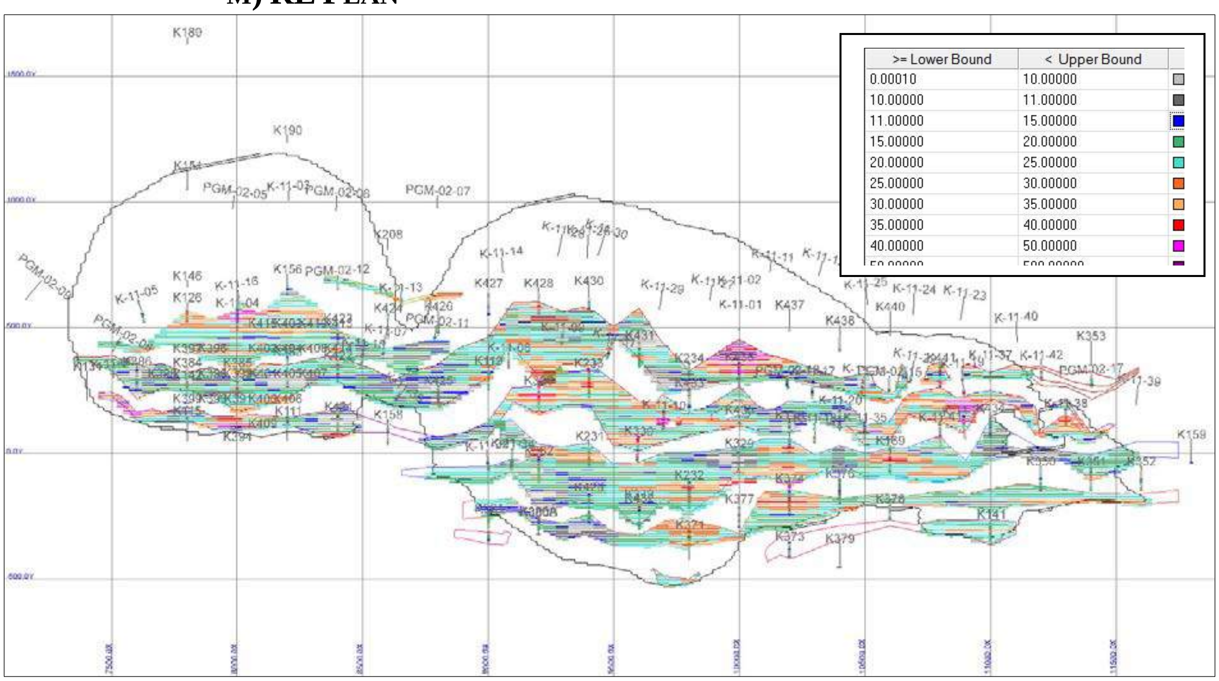
Pit constrained Mineral Resource blocks by NSR cut-off value are shown in cross section 9,400E in Figure 14.5 and in plan at -200 ft (-60 m) RL in Figure 14.6.

FIGURE 14.5 PIT CONSTRAINED MINERAL RESOURCE BLOCKS BY NSR, CROSS SECTION 9,400E - LOOKING EAST



Note: *Scale in Feet*
Source: *P&E (2012)*

FIGURE 14.6 PIT CONSTRAINED MINERAL RESOURCE BLOCKS BY NSR, -200 FT (-60 M) RL PLAN



Note: *Scale in Feet*
Source: *P&E (2012)*

Table 14.14 summarizes the pit constrained Inferred Mineral Resources for incremental NSR cut-off values. P&E recommends that the pit constrained Inferred Mineral Resource at the $12/tonne NSR cut-off value be used for public disclosure and further economic evaluation.

TABLE 14.14
K1-1 PIT CONSTRAINED INFERRED MINERAL RESOURCES SENSITIVITY AT VARIOUS NSR CUT-OFFS

Cut-off NSR ($/tonne)	Tonnes (k)	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Ag (g/t)
$35	4,616	0.62	0.14	0.04	0.07	0.21	2.9
$30	9,982	0.55	0.12	0.03	0.06	0.19	2.6
$25	20,589	0.49	0.11	0.03	0.06	0.17	2.3
$20	36,227	0.43	0.11	0.03	0.05	0.16	2.1
$15	49,021	0.40	0.10	0.03	0.05	0.15	1.9
$12	**53,614**	**0.38**	**0.10**	**0.03**	**0.05**	**0.14**	**1.8**
$10	54,287	0.38	0.10	0.03	0.05	0.14	1.8
$5	55,718	0.37	0.10	0.02	0.05	0.14	1.8

14.2.13 Block Model Validation

There has been no development or previous mining on the K1-1 Deposit, therefore no reconciliation studies or data are available for validation of the Mineral Resource Estimate. As such, estimated tonnages, grades, and contained metal cannot be compared to actual production, or gauge the sensitivity of the grade estimate to drill hole density. Validation of the grade interpolation and the updated model was carried out by on-screen review of grades and other block model estimation parameters versus drill hole composites, by comparison of assay, composites, zone intercepts and block grades, comparison to global results for alternate ID^3 and nearest neighbour (“NN”) interpolations, and review of the volumetrics of wireframes versus reported Mineral Resources.

15.0 MINERAL RESERVE ESTIMATE

No National Instrument 43-101 Mineral Reserve currently exists for the Thierry Project. This section is not applicable to this Technical Report.

16.0 MINING METHODS

The Thierry underground Deposit will be mined by underground long-hole retreat method. Mine plan drawings are shown in Appendix J. A longitudinal projection of the proposed underground mine is shown in Figure 16.1.

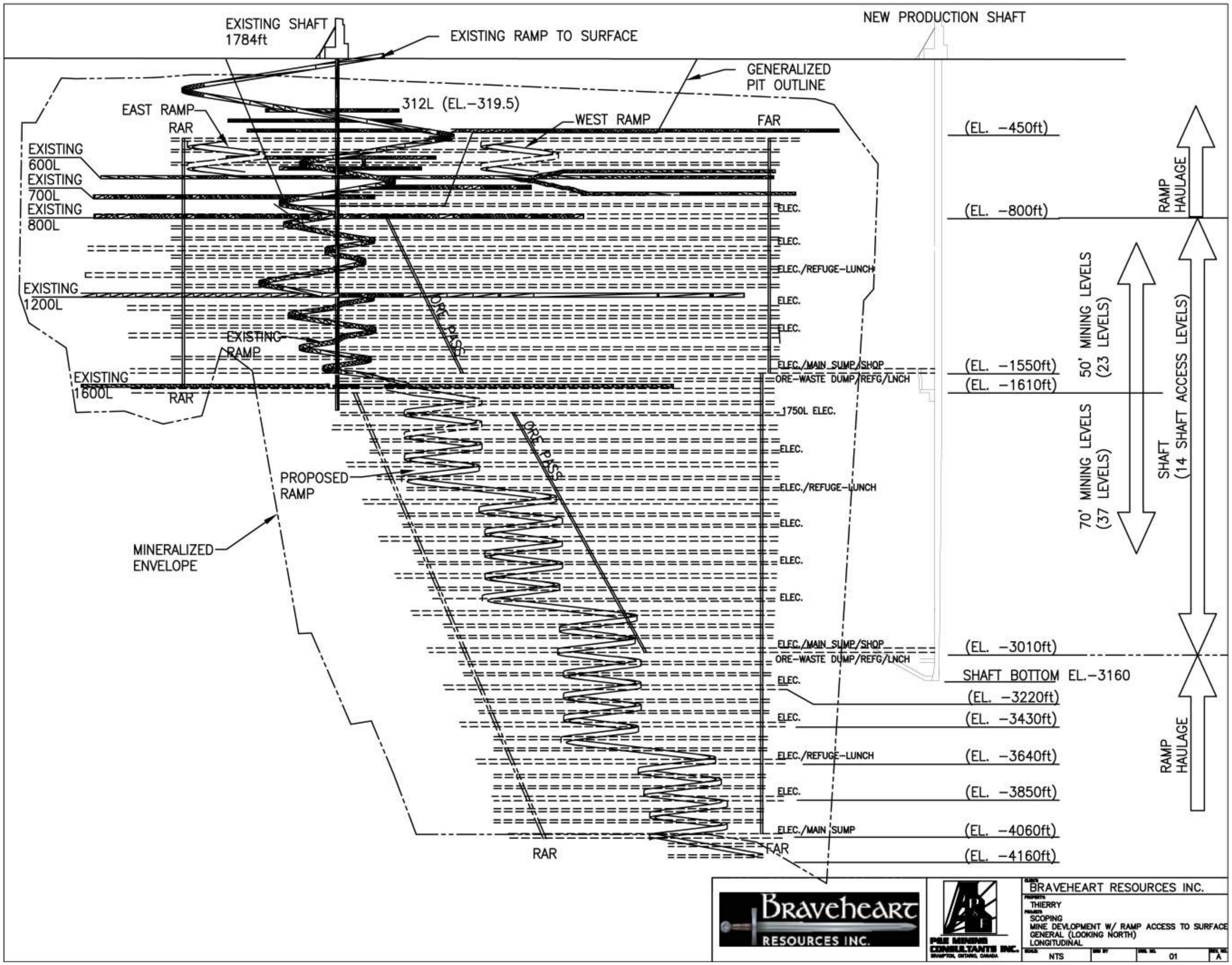
Access to the Thierry Deposit would be via a 6.5 m diameter, concrete lined 965 m (3,160 ft) deep fresh air shaft and a -15% ramp from surface to a depth of 1,260 m (4,130 ft). There will be two shaft loading pocket stations, one at a depth of 475 m (1,550 ft) and a second at a depth of 920 m (3,010 ft). Two hoists would be configured to transport workers and skip mineralized rock between surface and the underground loading pocket levels. Operating materials would be transported to and from the mine via the ramp from surface.

A conceptualized mining plan has been developed using mechanized trackless mining equipment. The primary mining method would be conventional longitudinal long-hole retreat with paste backfill. Above the 490 m (1,610 ft) elevation sub-levels will be developed at 15 m (50 ft) vertical intervals. Below the 490 m (1,610 ft) elevation sub-levels will be developed at 21 m (70 ft) vertical intervals. Drifts in mineralization would be developed to the full length of the Thierry Deposit. These drifts would provide access for the successive operations of slot raise development, blasthole drilling and blasting, and backfill placement. The average thickness of the mineralization is 6.7 m (22.1 ft). Remote-operated underground load/haul/dump ("LHD") units would remove broken mineralization from the stope and from the excavated drifts in mineralized rock. The stopes would be backfilled primarily with cemented paste backfill, supplemented with waste rock. Initially, mineralization above the 290 m (950 ft) level will be mined and hauled up the existing ramp during the pre-production period, while the shaft is being sunk and commissioned from the start of work to the 35th month. Once the shaft is commissioned both the 475 m (1,550 ft) and 920 m (3,010 ft) Levels will be developed from the shaft.

A steady state production of 4,000 tpd of development and stope production will begin during the 27th month, on a schedule of 350 days per year. Stope mining will proceed upwards from the 290 m (950 ft), 475 m (1,550 ft) and 920 m (3,010 ft) Levels and downwards from the 920 m (3,010 ft) Level towards the end of the mine life, on a retreat basis.

It is estimated that 432 stopes would be mined over the mine life. This would generate an average of 4,000 tpd composed of an average of 3,421 stoping tonnes and 579 tonnes from developing the drifts in mineralization and slot raises. The envisaged underground long-hole mining method for the Thierry Mineral Resource is estimated to experience mining dilution in the order of 20% at diluting grades of 0.59% Cu, 0.10% Ni and 2.20 g/t Ag with negligible Au, Pt and Pd. Mining recovery (extraction) is estimated at 90%.

FIGURE 16.1 THIERRY LONGITUDINAL PROJECTION



Source: *P&E (2021)*

16.1 LONG-HOLE LONGITUDINAL RETREAT STOPING METHOD

The Long-hole Longitudinal Retreat mining method is initially developed with sublevel drifts developed to the full width of the Thierry Deposit mineralization every 15 m (50 ft) or 21 m (70 ft) vertical intervals ("undercuts" and "overcuts") from the access cross-cuts. A 1.8 m by 1.8 m slot / ventilation / backfill raise is then driven at the end of the sublevel drift.

Blastholes measuring 92 mm ($3^{5/8}$ inches) in diameter would be drilled from the sublevel either up or down to adjacent sublevels. These blastholes would typically be drilled on a 2 m by 2 m 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 0.60 kg/t. An average estimated 3,421 tonnes of process plant feed would be excavated on a daily basis from a combination of stopes. Stope development activities would add another 579 tonnes of feed to the total, to provide a combined 4,000 tpd of process plant feed. A summary of stope drilling and blasting parameters is presented in Table 16.1.

TABLE 16.1
STOPING DRILLING AND BLASTING PARAMETERS

Parameter	Amount
Total Tonnes Process Plant Feed per Day from Mining Activities	4,000
Mineralization Bulk Density	3.1
Stope Height (m)	15
Nominal Stope Width (m)	6.7
Nominal Stope Length (m)	133
Total Nominal Stope Tonnage	45,458
Slot Raise Tonnage	133
Nominal Sublevel Drift Tonnage	6,449
Nominal Longhole Tonnes	38,876
Longhole Drilling Parameters @ 92 mm Dia Holes	
Total Drilling Per Stope (metres)	3,102
Drill holes Per Stope	220
Drilling Time Per Shift (minutes)	10
Metres Drilled per Shift	76
Total Metres Drilled Per Day	152
Required Metres per Day for Production Schedule	273
Blasting Parameters	
Loading Time Per Shift (hr)	10
Stemming Length Per Blasted Hole Length (m)	0.3
Load Length per Hole, (m)	14.7

TABLE 16.1 STOPING DRILLING AND BLASTING PARAMETERS	
Parameter	**Amount**
Length of Holes Loaded Per Ring (m)	46.1

Paste backfill and development waste would be placed in the mined-out stopes, from the level above through piping and/or boreholes, as stope drill/blast/mucking cycles progress.

The stope mining cycle would include long-hole drilling, blasting, loading and backfilling. The overall stope mining productivity is estimated to be a minimum 428 tpd per level. At any given time, a maximum of eight sublevels (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 428 tpd per level, 3,421 tpd overall. When no development mineralized rock is being produced a minimum fifth stope would be available for stope mining.

A summary of stoping productivities is presented in Table 16.2.

TABLE 16.2 STOPING PRODUCTIVITIES	
Operation	**Productivity**
Drilling (tpd)	1,910
Blasting (tpd)	1,910
Mucking (tpd)	1,910
Backfill (tpd)	1,910
Average Stope Productivity (tpd)	477
Minimum tpd / level	428
Maximum Number of Working Levels	8

16.2 MINE AND STOPE DEVELOPMENT

All excavations in waste rock are classified as mine development. All development in mineralization that produces process plant feed is classified as stope development. The life of mine ("LOM") schedule includes a total of 43,984 m of mine development (Table 16.3). In additional there would be 963 vertical m of shaft development and 3,794 m^3 of shaft station and loading pocket development.

TABLE 16.3 LOM SUMMARY OF UNDERGROUND MINE DEVELOPMENT		
Description	**Size (W x H) (m)**	**m**
Ramp	4.9x4.9	5,702
Safety Bays	0.9x1.8	155
X-Cuts/Miscellaneous.	4.9x4.9	35,473
Garage	10.1x4.9	58
Garage	7.0x7.9	76
Garage	9.1x4.9	3
Garage	8.2x4.9	98
Garage	5.5x4.9	13
Ventilation Raises	3.0x3.0	2,406
Total		**43,984**

There is a total of 62,615 m of stope development required over the LOM. This includes 57,574 m of drifting in mineralized rock and 5,041 m of slot raising. A summary of stope development is presented in Table 16.4.

TABLE 16.4 LOM SUMMARY OF UNDERGROUND STOPE DEVELOPMENT		
Description	**Size (W x H) (m)**	**m**
Drifts	4.0 x 4.0	57,574
Slot Raises	1.8 x 1.8	5,041
Total		**62,615**

In summary there is a total 106,599 m of mine and stope development completed over the LOM.

16.3 SHAFT SINKING AND CONSTRUCTION SCHEDULE

P&E estimates it will take 18 months to clear the site of the shaft collar, collar the shaft and install the headframe, hoist room and hoists and commission these installations. Shaft sinking will begin when this is complete. It is anticipated that the shaft will be commissioned 35 months from the start of collaring. Details of the shaft sinking schedule are presented in Table 16.5.

TABLE 16.5 SHAFT SINKING SCHEDULE		
Description	**Month**	
	Start	**Finish**
Collar / Headframe / Hoistroom	0	18.0
Collar to -1550L Station	18.0	23.5
-1550L Station	23.5	24.0
-1550L Station to Loading Pocket No.1	24.0	24.7
Loading Pocket No.1	24.7	24.8
Install Loading Pocket	24.8	25.3
Loading Pocket Raise	25.3	25.7
Loading Pocket No.1 to Spill Pocket No.1	25.3	25.5
Spill Pocket No.1	25.5	25.6
Spill Pocket No.1 to -3010 Station	25.6	30.8
-3010L Station	30.8	31.3
-3010L Station to Loading Pocket No.2	31.3	31.7
Loading Pocket No.2	31.7	31.8
Install Loading Pocket	31.8	32.3
Loading Pocket Raise	32.3	32.7
Loading Pocket No.2 to Shaft Bottom	32.3	32.5
Remove Sinking Gear & Commission Shaft	32.5	34.5

16.4 ACCESS RAMP FROM SURFACE

The existing ramp from surface extends down to the 490 m (1,610 ft) Level. From there a new ramp will ultimately be driven at a -15% gradient to the 1,260 m (4,130 ft) Level by a contractor. This access ramp will allow underground mobile equipment, personnel and supplies to travel between levels, as well as to and from surface.

Initially the exiting ramp and mine will be dewatered and rehabilitated down to the 290 m (950 ft) level, during the first six months of pre-production work. At that point mine development will start on the 290 m (950 ft) Level. Mine dewatering and rehabilitation will continue down to the 490 m (1,610 ft) Level, scheduled to be completed by the 19th month. The development of the extension of the existing ramp system will start during the 20th month, to be completed to the 920 m (3,010 ft) level in the 35th month, as the shaft is being commissioned. Development of the ramp below the 920 m (3,010 ft) Level will be ‘as-required’ near the end of the underground mining operation.

Details of the ramp development schedule are presented in Table 16.6.

TABLE 16.6 **ACCESS RAMP DEVELOPMENT SCHEDULE**			
Description / Level Interval	**Quantity (m)**	**Month**	
		Start	**Finish**
Dewater and Rehab Existing Ramp to 950		0	5.7
Dewater and Rehab Existing Ramp 950 to 1610		5.7	19.0
1610 - 1890	651	19.0	22.1
1960 - 2170	651	22.1	25.2
2240 - 2450	651	25.2	28.3
2520 - 2730	651	28.3	31.4
2800 - 3010	651	31.4	34.5
3080 - 3290	651	147.8	150.9
3360 - 3570	651	150.9	154.0
3640 - 3850	651	154.0	157.1
3920 - 4130	651	157.1	160.2

16.5 LEVEL DEVELOPMENT

Pre-production level development will start on the 290 m (950 ft) Level once mine dewatering and rehabilitation has reached that elevation. Once the 244 m to 950 m (800 ft to 950 ft) development work is complete, development crews will proceed to develop all levels and sublevels up to the 137 m (450 ft) Level, from the bottom up.

Once the shaft has been commissioned, development crews will proceed to develop the 475 m (1,550 ft) and 920 m (3,010 ft) Levels simultaneously. Mine development work will proceed upwards to the 290 m (950 ft) and the 475 m (1,550 ft) Levels, respectively, as required.

Mine development on the 1,003 m (3,290 ft) Level, below the shaft bottom 920 m (3,010 ft) loading pocket, will start during the 121st month. From this point all mine development will be developed from the bottom down in group of four sublevels, as required. There is a total of 38,126 m of level mine development over the LOM. A summary of the mine development schedule is presented in Table 16.7.

TABLE 16.7 LOM MINE DEVELOPMENT SCHEDULE				
Level	**Type / Size (m)**	**Quantity (m)**	**Month**	
			Start	**Finish**
450 - 600	H4.9 x 4.9	1,435	30.1	35.6
	V3.0 x 3.0	90	34.6	35.6
650 - 750	H4.9 x 4.9	3,294	18.3	30.1
	V3.0 x 3.0	68	29.3	30.1
800 - 950	H4.9 x 4.9	3,514	5.7	18.3
	V3.0 x 3.0	90	17.4	18.3
1000 - 1150	H4.9 x 4.9	4,519	75.8	100.2
	V3.0 x 3.0	90	99.2	100.2
1200 - 1350	H4.9 x 4.9	3,023	62.2	70.3
	V3.0 x 3.0	90	69.3	70.3
1400 - 1550	H4.9 x 4.9	4,997	34.5	48.0
	V3.0 x 3.0	330	44.4	48.0
	H10.1 x 4.9	29	48.0	48.1
	H7.0 x 7.9	38	48.1	48.3
	H9.1 x 4.9	2	48.3	48.3
	H8.2 x 4.9	37	48.3	48.4
	H5.5 x 4.9	6	48.4	48.5
1610 - 1890	H4.9 x 4.9	3,259	120.9	129.7
	V3.0 x 3.0	174	127.8	129.7
1960 - 2170	H4.9 x 4.9	1,370	98.1	105.4
	V3.0 x 3.0	139	104.0	105.4
2240 - 2450	H4.9 x 4.9	1,330	75.4	82.6
	V3.0 x 3.0	139	81.1	82.6
2520 - 2730	H4.9 x 4.9	1,454	51.9	59.8
	V3.0 x 3.0	139	58.3	59.8
2800 - 3010	H4.9 x 4.9	2,796	34.5	42.1
	V3.0 x 3.0	553	36.1	42.1
	H10.1 x 4.9	29	42.1	42.2
	H7.0 x 7.9	38	42.2	42.4
	H9.1 x 4.9	2	42.4	42.4
	H8.2 x 4.9	61	42.4	42.6
	H5.5 x 4.9	6	42.6	42.6
3080 - 3290	H4.9 x 4.9	1,230	120.7	127.3
	V3.0 x 3.0	139	125.8	127.3
3360 - 3570	H4.9 x 4.9	1,244	139.7	146.4

TABLE 16.7 LOM MINE DEVELOPMENT SCHEDULE				
Level	**Type / Size (m)**	**Quantity (m)**	**Month**	
			Start	**Finish**
	V3.0 x 3.0	139	144.9	146.4
3640 - 3850	H4.9 x 4.9	1,277	152.7	159.6
	V3.0 x 3.0	139	158.1	159.6
3920 - 4130	H4.9 x 4.9	730	160.2	164.1
	V3.0 x 3.0	87	163.2	164.1
Total		**38,126**	**5.7**	**164.1**

16.6 STOPE DEVELOPMENT

Stope development includes both drifting and slot raising in mineralized rock. Stope development will start on the 290 m (950 ft) Level during the 16th pre-production month. Once the 244 m to 950 m (800 to 950 ft) development work is complete, development crews will proceed to develop all levels and sublevels up to the 137 m (450 ft) Level, upwards.

Stope development on the 920 m (3,010 ft) Levels will start during the 4th month from the start of full production and six months after the start of production on the 475 m (1,550 ft) Level.

Stope development on the 1,003 m (3,290 ft) Level, below the bottom 920 m (3,010 ft) loading pocket, will start during the 92nd month of full production. From this point all stope development will be developed from the bottom down in groups of four sublevels. There is a total of 57,574 m of level stope development over the LOM. A summary of the stope development schedule is presented in Table 16.8.

TABLE 16.8 LOM STOPE DEVELOPMENT SCHEDULE			
Level	**Quantity (m)**	**Month**	
		Start	**Finish**
450 - 600	2,060	33.1	38.6
650 - 750	1,739	28.4	33.1
800 - 950	4,838	15.3	28.4
1,000 – 1,150	5,472	100.2	129.7
1,200 – 1,350	5,544	70.3	100.2
1,400 – 1,550	5,254	42.0	70.3
1,610 – 1,890	5,545	129.7	159.6
1,960 – 2,170	4,059	105.4	127.3
2,240 – 2,450	4,248	82.6	105.4
2,520 – 2,730	4,230	59.8	82.6

TABLE 16.8 **LOM STOPE DEVELOPMENT SCHEDULE**			
Level	**Quantity (m)**	**Month**	
		Start	**Finish**
2,800 – 3,010	3,837	39.1	59.8
3,080 – 3,290	3,541	127.3	146.4
3,360 – 3,570	3,284	146.4	164.1
3,640 – 3,850	2,904	159.6	175.2
3,920 – 4,130	1,018	164.1	169.6
Total	**57,574**	**15.3**	**175.2**

16.7 STOPING

Stope production will start on the 290 m (950 ft) Level during the 26th month of pre-production development. A summary of the LOM stoping schedule is presented in Table 16.9.

TABLE 16.9 **LOM STOPING SCHEDULE**			
Level	**Quantity (t)**	**Month**	
		Start	**Finish**
450 - 600	595,512	40.8	47.7
650 - 750	406,654	37.0	40.8
800 - 950	1,151,469	25.4	37.0
1,000 – 1,150	1,490,947	106.7	137.1
1,200 – 1,350	1,474,588	76.7	106.7
1,400 – 1,550	1,421,341	48.0	76.7
1,610 – 1,890	2,318,069	137.1	182.2
1,960 – 2,170	1,847,995	135.6	172.9
2,240 – 2,450	1,887,099	97.3	135.6
2,520 – 2,730	1,491,419	66.9	97.3
2,800 – 3,010	1,038,401	45.1	66.9
3,080 – 3,290	712,373	172.9	185.4
3,360 – 3,570	539,359	182.2	191.5
3,640 – 3,850	324,767	185.4	190.9
3,920 – 4,130	94,617	190.9	192.5
Total Stoping Tonnes	**16,794,610**	**25.4**	**192.5**

17.0 RECOVERY METHODS

A summary of available metallurgical testwork has been presented in Section 13. While the flowsheet and process data from the historical operations are not available and the post-operation laboratory testwork data is limited, it is assumed that a new process plant will be a conventional facility with crushing, grinding, flotation, concentrate dewatering and tailings thickening for paste backfill preparation as well as for surface disposal. The process plant will be sized for a nominal capacity of 4,000 tpd with a surge capability of 4,500 tpd.

17.1 MINERALIZED PLANT FEED HANDLING

Mineralized material, sized to less than 300 mm, will be hoisted from underground and crushed to -100 mm in a jaw crusher. No crusher will be installed underground, and oversize will be managed underground with rock breakers and grizzlies. The single primary crusher size on surface could be as large as 110 mm by 120 mm (42 in by 48 in) and powered by a 180 kW drive. The crushed feed will be drawn from a mine skip surge bin by an apron feeder discharging to a conveyor equipped with metallic scrap removal magnets. Crusher discharge would be transferred to a 10,000 t capacity covered stockpile, from which material would be drawn by at least three feeders. The stockpile would be manipulated with a front-end loader to reduce stockpile segregation by size and to compensate for freezing.

17.2 GRINDING

Conventional SAG and ball mill grinding is proposed. SAG feed will be automatically weighed and grab-sampled for moisture content. With a target grind size P_{80} of 100 µm, a SAG size of 5.5 m diameter by 4.4 m long and a ball mill size of 5.5 m by 8.4 m long should be suitable. Steel ball consumption could be in the order of 3-4 kg/t with energy draw approximately 25-30 kWh/t.

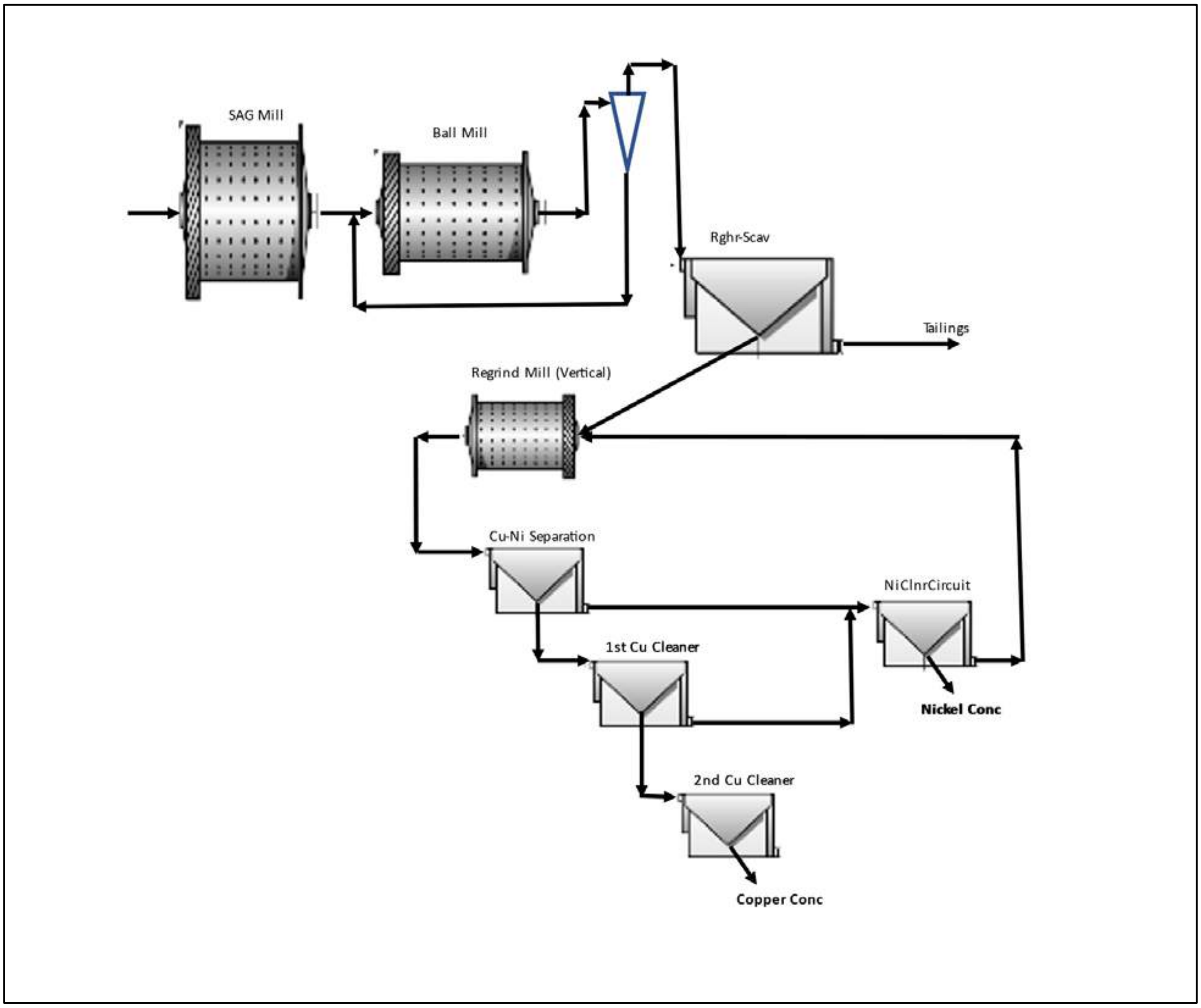
The SAG mill is equipped with a pebble circuit where pebbles are recycled to the SAG feed. Pebble return is expected to be low, at less than 5% of feed. At this low rate, a pebble crusher (gyratory) would not be required but could be installed later. The ball mill will be in closed circuit with two banks of cyclones, with cyclone overflow sent to a flotation conditioner following automatic two-stage slurry sampling.

17.3 FLOTATION

The conceptual concentration circuit is shown in Figure 17.1. A low-grade copper-nickel concentrate is obtained in a rougher-scavenger circuit which will have a retention time of 20-25 minutes. The rougher scavenger tailings will be automatically sampled with a two-stage Vezin-type sampler. The rougher-scavenger concentrate is finely ground to be approximately P_{80} 20-25 µm. This smaller grinding unit could be a rubber-lined ball mill, but a vertical attrition-grinding mill using ceramic grinding media may be preferred. Less than 5% (200 tpd, 8.3 tph) of the plant feed will report to the regrind mill and to the subsequent flotation circuits.

The finely ground rougher-scavenger concentrate would be subject to precise flotation reagent conditioning and directed to a copper-nickel separation flotation step, with tailings reporting to a nickel concentration/cleaner circuit. The copper concentrate from the copper-nickel separation step would be subject to at least two copper cleaner stages.

FIGURE 17.1 CONCEPTUAL THIERRY GRINDING AND FLOTATION CIRCUIT



Source: *P&E (2021)*

Some caveats to the definition of a new Thierry processing plant flowsheet include:

- Available test results indicate that most of the copper content floats well, however, a portion floats slowly, resulting in the need for long rougher flotation retention time (>20 minutes). However, a marketable grade concentrate can be obtained with minimal influence of head grade on recovery;

- Success was limited in attempts to produce a saleable nickel concentrate from copper cleaner tailings. One reason for this was the absence of significant quantities of cleaner tailings required to feed a laboratory nickel cleaner circuit. However, the reports indicate that testwork completed at the Thierry process plant before closure in 1982 indicated that a saleable nickel concentrate could be obtained with a 1.7% Cu, 6.9% Ni concentrate grade at 50% Ni recovery. Copper cleaner tailings had been fed to a test circuit aimed at producing a saleable nickel concentrate;

- Nickel in the copper concentrate is expected to be below payable limits;

- Gold and PGM's largely follow the copper; laboratory testwork on higher PGM grade samples (than in the current Mineral Resource Estimate) indicated 50% recovery of each metal was achieved. This value is selected for current estimates;

- An optimum Thierry grinding and flotation circuit remains to be developed. Additional bench-scale testing on fresh drill core supported by mineralogical examination is required. This could be followed by pilot-scale confirmation; and

- Supplementary metallurgical testwork could be considered, such as potential upgrading of rougher concentrate by removal of pyrrhotite by magnetic separation, as well as consideration of hydrometallurgical test extraction of all valuable metals from a bulk concentrate or a nickel-rich concentrate.

17.4 CONCENTRATE HANDLING

The two flotation concentrates will be separately thickened in conventional type thickeners and filtered using plate and frame pressure filters. The filtered concentrate moisture content is expected to be 10% or greater. A higher than desirable moisture content will be caused by the fine particle size in each of the concentrates.

The moist concentrate is expected to be trucked to smelters in Sudbury, ON and Rouyn-Noranda, QC. Subject to confirmation of no liquefaction potential in transport, the shipment will be as a bulk concentrate in warm weather and in 1 tonne tote bags in colder weather. No on-site concentrate drying is proposed.

Concentrate will be slurry automatic-sampled as thickener feed, and manually batch pipe-sampled for each shipment. Copper concentrate would be expected to be approximately 180 tpd, while a copper-nickel concentrate may be as high as 30 tpd.

17.5 TAILINGS AND WATER MANAGMENT

Tailings will be thickened to approximately 55% solids using a conventional high-rate thickener located in the process plant for proportional delivery to the paste backfill plant and to the tailings facility 1.5 km away. The paste backfill plant may be a separate structure from the process plant. It is anticipated that no fines scalping of the paste plant feed will be required to achieve desirable paste consistencies.

Subject to confirmation that tailings thickener overflow water quality is not detrimental to flotation performance, process water will be a combination of tailings thickener reclaim water and tailings facility reclaim water. Underground mine water is an additional potential process water source.

18.0 PROJECT INFRASTRUCTURE

The Thierry Project has minimal infrastructure requirements due to its location close to Pickle Lake and Thunder Bay, Ontario and due to the infrastructure established during its previous operating history.

18.1 SITE SURFACE INFRASTRUCTURE

The site facilities would include a shaft headframe and hoistroom/compressor building; a process plant; a paste backfill plant and distribution system; a tailings/waste rock co-disposal basin and dam; site roads; surface parking areas; fuel, lubricant and oil storage facilities; surface explosive magazines; yard piping; a fire prevention and fighting system; potable water treatment plant and storage tanks; tailings water treatment plant and pond, and a water management pond building.

Major surface facilities to support the Thierry Project 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, safety 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

An electrical power line and electrical substation will have to be constructed on site and connected to the existing nearby grid. Overall site power consumption is estimated to be approximately 16 MW.

18.3 TAILINGS AND WASTE ROCK MANAGEMENT

It is planned that the process plant tailings would be thickened for use in mine backfilling or otherwise disposed in the waste rock and tailings co-disposal facility (“CDF”) that is proposed to be located within the historical Thierry tailings management area and its sub-watershed catchment area. The CDF would be designed to provide a physically and chemically stable environment that would be suitable for the long term storage of waste rock and tailings, as well as providing suitable management of any potentially acid generating / metal leaching components.

It is expected that the CDF waste disposal cells would be sequentially filled and capped over the operating life of the mine and that the final closure works would include the capping of the final cells. This would be followed by a five year batch water pond treatment and closure performance monitoring program and a subsequent five year post-closure environmental monitoring program.

A summary of the tailings disposal schedule is presented in Table 18.1.

TABLE 18.1
SUMMARY OF TOTAL LOM TAILINGS DISPOSAL SCHEDULE

Description / Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	Total
	Preproduction			Production														
	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	
Tailings to Dam (kt)	0.0	151.1	505.2	639.3	629.8	631.0	630.1	629.3	633.0	638.6	638.5	635.3	633.5	634.0	633.9	636.5	0.2	8,899.3

Note: *Yr = year*

18.4 WASTE MANAGEMENT

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

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

18.5 REGIONAL RESOURCES

The regional labour force includes a reasonable number of experienced equipment operators, mine workers and material and equipment suppliers.

19.0 MARKET STUDIES AND CONTRACTS

Metal prices and the CDN:US dollar exchange rate are based on a December 2020 long-term consensus forecast,t by approximately 30 financial and brokerage institutions, and are presented in Table 19.1. Both the metal prices and exchange rate are potentially subject to spot market conditions. There are no metals streaming or hedging agreements in place.

TABLE 19.1
METAL PRICES AND EXCHANGE RATE

Item	Price
Copper (US$/lb)	3.48
Nickel (US$/lb)	8.00
Silver (US$/oz)	21.00
Palladium (US$/oz)	1,250
Platinum (US$/oz)	1,100
Gold (US$/oz)	1,600
Exchange Rate (CDN:US$)	0.75

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

The commercial products produced by this Project are estimated to be a 30% copper concentrate containing 0.5% nickel, with accompanying platinum, palladium, gold and silver, and an 8% nickel concentrate containing 2% Cu, with accompanying platinum, palladium, gold and silver. These concentrates will be shipped to any of several available smelters. Concentrate revenue will be based on future metal prices, less respective transportation, smelting and refining charges. The likely destination for copper concentrates is the Horne Smelter in Rouyn-Noranda, Quebec, and for nickel concentrates it is Sudbury, Ontario.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 PROJECT LOCATION DESCRIPTION

The Thierry Project is located 12 km WNW of the Town of Pickle Lake at the site of the historical underground Thierry Mine, where there were also two open pits, a process plant, a tailings facility and several waste rock-piles. The Project area is located in flat and sloping terrain interspersed with coniferous and deciduous forest areas, rivers, creeks, lakes, ponds and swampy areas. The proposed underground mine and process plant site is accessible by a 19 km gravel road from Pickle Lake.

The idle Thierry mine complex (buildings now removed) is shown in a 1988 aerial view (looking west) in Figure 20.1.

FIGURE 20.1 IDLE THIERRY MINE COMPLEX - 1988

Source: *UMEX (1988)*

The historical mine ceased operation 39 years ago and all surface infrastructure was later removed. The tailings facility, a combination of tailings and waste rock deposited in a small lake is evident approximately 1.3 km west of the mine and plant site (blue arrow) as shown in Figure 20.2. The two pits, shown empty in Figure 20.1 are flooded in Figure 20.2 (red arrow). This tends to confirm that the pits are hydraulically connected to the underground workings.

FIGURE 20.2 HISTORICAL THIERRY MINE SITE

Source: *Google Earth (2020)*
blue arrow points to historic tailings facility, red arrow points to two historic pits.

A close up view of the mine site is shown in Figure 20.3, and this suggests that some features, e.g. waste rock piles, may not have been reclaimed to current Ontario standards. Also, natural re-vegetation of disturbed areas appears to have been minimal, suggesting some metal and/or acid toxicity.

FIGURE 20.3 CLOSE-UP VIEW OF THIERRY SITE

Source: *Google Earth (2020)*

20.2 SCOPE OF THE THIERRY MINE PROJECT

The scope of the Project as currently envisaged includes:

- The dewatering of the historical underground Thierry Mine. The adjoining east and west open pits (Figure 20.3) will also be dewatered due to likely hydraulic connections to the underground workings as well as mine safety concerns;

- A Certificate of Approval (No. 0764-6S2HFF) had been obtained for an underground mine dewatering program in 2006 and an approved closure plan and bonding were established. The mine dewatering program had been delayed in favour of additional drilling from surface;

- Environmental studies will be initiated with a Baseline Study. A Project Environmental Assessment ("EA") – either or both Federal and Provincial EA processes are possible. A Federal EA could be triggered by impacts to fish habitat and potential impacts to First Nations rights and traditions. A provincial EA could be triggered by tailings management and rebuilding of the power line;

- Many new permits and certificates of approval would be required to re-open the Thierry Mine and re-establish surface infrastructure;

- The dewatered underground mine would be rehabilitated and re-developed as a 4,000 tpd mine with both shaft and ramp access;

- A new 4,000 tpd process plant would be constructed, with a rebuild of the power line and substations;

- The development and operation of a new tailings facility within the sub-watershed of the historical tailings management area;

- The development of the site infrastructure;

- 14 years of mine operations and concentrate shipment; and

- Progressive and final closure works including post-closure monitoring.

20.3 ENVIRONMENTAL ASSESSMENT AND PERMITTING

Braveheart and previous owners have focused on mineral exploration activities and assessing the potential economic viability and practicality of developing the Project. Additional water sampling had been conducted as part of a study to assess alternate approaches to dewatering the historical mine. Mine water quality data is not available to P&E; treatment of mine water is considered likely. In consideration of the present PEA status of the Project, no formal discussions have been initiated with Provincial regulatory authorities in regard to environmental assessment and permitting requirements for the proposed mining and processing Project.

20.3.1 Federal Environmental Assessment Process

In 2012, the 1992 Canadian Environmental Assessment Act was updated to CEAA 2012 and this was recently updated under Federal Legislation C-69. Triggers for the Federal EA process include:

- Disturbance of fish habitat and that of other aquatic species;
- Impact on migratory birds;
- Federal lands and effects of crossing interprovincial boundaries;
- Effects on Aboriginal peoples such as their use of traditional lands and resources; and
- A physical activity that is designated by the Federal Minister of Environment that can cause adverse environmental effects or result in public concerns.

Only the potential effects on Aboriginal peoples could be a federal EA trigger, however, this is anticipated to be mitigated following consultation and negotiation with the Mishkeegogamang First Nation.

20.3.2 Provincial Environmental Assessment Process

The Ontario EA process is administered by the recently renamed Ministry – the Ministry of Environment, Conservation and Parks ("MECP"). In addition to promoting responsible environmental management, interested third parties, e.g. members of the public, can comment on a mining project and request that the MECP minister call for an EA.

Ontario mining projects are not often subject to the provincial EA Act ("OEA") since many mine development activities are not specified in the relevant Act. However, specifications that do include the need to conduct and EA are:

- Transfer of Crown resources including land;
- Building electric power generation facilities or transmission lines;
- Constructing new roads and transport facilities; and
- Establishing a tailings management facility.

No Crown resources are affected by the Thierry Project. The former transmission corridor to the south remains cleared. The proposed tailings management strategy includes maximizing mine backfill and expansion of the historical tailings facility that employed underwater disposal in combination with a waste rock covering.

20.3.3 Environmental Approval Requirements

A significant number of approvals, permits, and authorizations will be needed following the EA process, and in advance of construction and operations. Federal items are:

- If fish occupy the historical tailings facility, authorization may be required for alteration to fish habitat; and
- Acquiring an Explosives Handling License.

Provincial approvals, permits and authorizations are numerous and may include:

- Approvals for emissions, discharges and waste management;
- Permit to take water;
- Work permit for construction of mine facilities;
- Building and land use permits;
- Endangered species permit consideration (if relevant);
- Bulk fuel handling, domestic waste water treatment permits;
- Forest license – allowance for any additional clearances;
- Approval of health and safety procedures and management, as well as emergency provisions; and
- Approval of a financed Closure Plan.

In addition, several municipal permits are anticipated to be required – e.g. accommodation and catering facilities as well as modifications to roads and management of domestic water, solid waste and sewage.

20.3.4 Environmental Management Strategies

The Thierry Project will be designed and operated with strategies that will minimize environmental impact on the local environment and practice transparent accountability. Important aspects include:

- Minimization of Project footprint – little to no expansion beyond the areas modified by the historical operations;

- Curtailing the need to draw fresh water – process water will be sourced from mine water, the tailings facility and site surface run-off;

- Design for closure – stockpile soils, progressively closed out during operations;

- Design and implement measures to prevent acid rock drainage (“ARD”) from mine waste rock and tailings; and

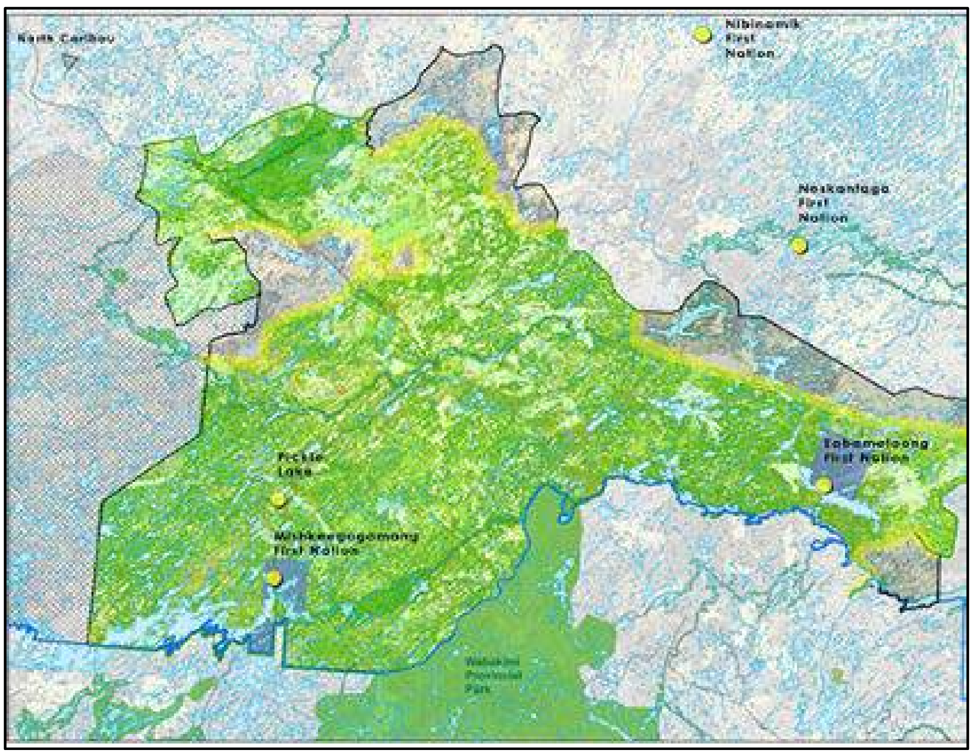
- Include the interested public in Project environmental management, monitoring and reporting.

20.4 SOCIAL AND COMMUNITY REQUIREMENTS

20.4.1 First Nations Community Consultation

A cooperative agreement will be developed with the Mishkeegogamang First Nation, an Ojibway First Nation, based on a Treaty 9 Reserve formerly known as Osnaburgh House. This Reserve is located on Highway 599 approximately 30 km south of the Town of Pickle Lake. The Thierry Project is located in the lower third of the traditional Mishkeegogamang lands as shown in Figure 20.4. A cooperative and mutually respectful relationship which would include ongoing consultations, as well as business and employment opportunities, is anticipated. The recently-developed Musselwhite-Mishkeegogamang Partnership Agreement could be a guide. Newmont's Musselwhite gold mine is 200 km (by road) north of Pickle Lake.

FIGURE 20.4 MISHKEEGOGAMANG FIRST NATION TRADITIONAL LAND USE [1]



Source:*https://www.ontario.ca/page/eabametoong-and-mishkeegogamang-first-nations-far-north-community-based-land-use-planning-terms (2020)*

20.4.2 Local Community Consultation

Pickle Lake (population ~400) is the nearest main community in the area with tourism, recreation and forestry the principle economic activities. The Thierry Project would have a major economic impact on the community since mine access by the 19 km road would be Pickle Lake-

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[1] Eabametoong First Nation and Mishkeegogamang First Nation Land Use Plan, Approved by Band Councils and Ontario Ministry of Natural Resources, July 2013

based. The community could also be a housing location for off-shift Thierry workers who may commute to Pickle Lake for a work schedule by air or road from larger communities such as Thunder Bay. Consultation would be undertaken with the local public and various stakeholders to agree on necessary community infrastructure upgrades and opportunities.

20.5 ENVIRONMENTAL ASPECTS

The historical Thierry Deposit was exploited by mining and processing operations between 1976 and 1982. The works included a tailings pond (~235 ha), a fresh water pond (~40 ha) and associated containment structures; and waste rock and low-grade rock stockpiles (Figures 20.1-20.3). The de-energized power transmission line extended from the Ear Falls-Central Patricia power line and crosses the Kawinigans River at Kapkichi Lake near the location of the proposed mine and plant sites.

The Project scope will include measures to reduce or mitigate environmental impacts and maximize social benefits. It is assumed that the Project would make use of the historical process plant site, mine, administration, warehouse, switchyard and lay-down areas that are currently rehabilitated.

It is expected that a key potential environmental aspect of the proposed Project are potential impacts due to acid rock drainage and metal leaching from tailings, waste rock and mine openings. It is proposed that tailings management focuses on maximizing use of underground paste backfill and management of the residual amount of tailings by underwater disposal. The paste backfill involves the addition of alkaline cements which inhibits acid generation. Underwater disposal, which was historically practiced at Thierry, is a recognized technology for the prevention of acid rock drainage and the inhibition of metal leaching.

The dewatering of the underground mine and the associated pits is anticipated to result in the potential discharge of contaminated mine water. This water will be used in the process plant and any excess will be treated to meet acceptable national and provincial effluent standards before discharge to the environment.

20.6 MINE CLOSURE

It is assumed that closure works would be progressively carried out as soon as possible over the operating life of the mine and completed following the end of operations. Major aspects would involve:

- The removal of mobile equipment, mine services and unused fuel, lubricants and explosives from the underground mine. The mine openings to surface would be appropriately sealed, and the underground and the connected pits would be allowed to naturally flood. The mine flooding is expected to take several years, and batch treatment of open pit water will be considered.

- The salvaging and sale of mine plant facilities including the headframe and hoist room, shop building, and process plant equipment. Other buildings and Project infrastructure would be demolished and properly disposed. The mine and process plant sites and on-site roads would be rehabilitated.

- The capping of the tailings facility cells with clean rock and soil. The associated pond would be lime-batch treated for as many years as required; discharge would be monitored for metals, pH and biotoxicity and treated as required. Post-closure environmental performance monitoring would be targeted to last an additional five years.

An approved closure plan and financial assurance was in place for a previously proposed underground mine dewatering program. Braveheart would be required to provide an updated Closure Plan with accompanying financial assurance for the Project to be approved by the Ministry of Energy, Northern Development and Mines prior to developing the Project.

21.0 CAPITAL AND OPERATING COSTS

The estimated capital and operating costs related to the construction and operation of the mining and processing facilities are provided in this section.

All capital and operating costs are shown in Canadian dollars as at Q4 2020, unless otherwise stipulated.

21.1 CAPITAL COSTS

The total capital cost of the Theirry Project is estimated at approximately $710.5M. This is composed of $407.0M in pre-production capital and $303.5M in sustaining capital. An allowance for contingency of 5% has been included in these totals. A breakdown of this estimate is provided in Table 21.1 and includes contingencies.

TABLE 21.1
SUMMARY OF TOTAL LIFE-OF-MINE CAPITAL COST

Description / Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	
	<< Pre-Production			**Production >>**														
	Yr 1	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**	**Yr 16**	**Yr 17**	**Total ($M)**
Stope Mining			10.2															**10.2**
Support Services	3.2	5.2	11.1															**19.5**
Paste Backfill			10.2															**10.2**
Process Plant Commissioning		2.3	18.5															**20.8**
Haulage		1.9	14.3															**16.1**
G&A Costs	4.5	5.6	7.2															**17.3**
Mine & Stope Development	5.7	20.0	26.4	35.5	17.9	22.3	21.8	19.4	19.9	12.1	27.8	14.7	17.8	15.6	4.5			**281.4**
Shaft Development		25.1	32.7															**57.9**
Ramp Development		5.5	11.7										9.2	4.7				**31.1**
Headframe, Hoist & Hoist Room, LP	12.6	5.5	4.4															**22.5**
Mine Equipment		4.9	27.5	8.5			1.5	3.5	3.2	22.2	4.9	3.2	4.9	1.7	3.2	1.7		**91.1**
U/G Infrastructure		2.8	0.2	4.5	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0	0	0		**8.3**
Surface Infrastructure	8.3	28.2								1.6								**38.1**
Process Plant Equip and Construct	57.3	28.7																**86.0**
Closure Bond & Salvage	4.6	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	-8.9	**0.0**
Total CAPEX ($M)	**96.3**	**136.0**	**174.8**	**48.8**	**18.2**	**22.6**	**23.7**	**23.3**	**23.4**	**36.3**	**33.1**	**18.3**	**32.3**	**22.3**	**8.0**	**2.1**	**-8.9**	**710.5**

Note: *Yr = year.*

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 36 months later. Pre-production capital costs include the cost of all surface buildings, process plant and related facilities and structures; mine and stope development on the 137 m (450 ft) to 290 m (950 ft) Levels; initial stope mining, initial support services, initial paste backfilling, initial underground haulage and initial G&A costs, shaft development, shaft commissioning and related facilities; initial ramp development to the 917 m (3,010 ft) Level; underground mining equipment; surface mobile equipment; electrical power supply infrastructure; underground infrastructure related to the shaft and 137 m (450 ft) to 290 m (950 ft) Levels, and part of the Project closure bond. The total estimated pre-production capital cost is estimated to be $407.0 M.

21.1.1.1 Pre-production Underground Mine and Stope Development Capital Cost

A summary of pre-production underground mine and stope capital cost estimates are presented in Table 21.2. This work will be completed by Company crews.

TABLE 21.2
PRE-PRODUCTION UNDERGROUND MINE & STOPE DEVELOPMENT CAPITAL COSTS

Heading Type	Unit Cost ($/m)	Year ($M)			Subtotal ($M)	Contingency ($M)	Total ($M)
		-3	-2	-1			
X-cut 4.9 x 4.9 m	3,268	5.7	11.0	13.8	30.5	0.0	**30.5**
V3.0 x 3.0 m	3,086	0.0	0.3	0.5	0.8	0.0	**0.8**
H4.0 x 4.0 m	2,723	0.0	8.7	12.1	20.9	0.0	**20.9**
Total Capital ($M)		**5.7**	**20.0**	**26.4**	**52.1**	**0.0**	**52.1**

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

21.1.1.2 Pre-production Shaft Development Capital Costs

Once the shaft collar has been excavated to approximately 61 m below surface and the headframe and hoistroom are installed and commissioned, shaft sinking can begin. A vertical 6 m to 6.5 m diameter concrete lined shaft would be sunk conventionally from the bottom of the collar to the 963 m (3,160 ft) elevation, a vertical distance of approximately 902 m (2,960 ft). There will be two loading pockets installed, one below the 472 m (1,550 ft) Level station and the other below the 917 m (3,010 ft) Level station. It will take approximately 500 days to sink and commission the shaft. A summary of the shaft development capital cost and schedule is presented in Table 21.3.

TABLE 21.3 PRE-PRODUCTION SHAFT DEVELOPMENT CAPITAL COSTS						
Description	**Year ($M)**			**Subtotal ($M)**	**Contingency ($M)**	**Total ($M)**
	-3	**-2**	**-1**			
Shaft Sinking Setup	0.0	1.5	0.0	1.5	0.2	**1.7**
Collar to -1550L Station	0.0	20.8	0.0	20.8	2.1	**22.9**
-1550L Station	0.0	0.5	0.0	0.6	0.1	**0.6**
-1550L Station to Loading Pocket No.1	0.0	0.0	2.3	2.3	0.2	**2.5**
Loading Pocket No.1	0.0	0.0	0.2	0.2	0.0	**0.2**
Install Loading Pocket	0.0	0.0	1.5	1.5	0.2	**1.7**
Loading Pocket Raise	0.0	0.0	0.4	0.4	0.0	**0.5**
Loading Pocket No.1 to Spill Pocket No.1	0.0	0.0	0.8	0.8	0.1	**0.8**
Spill Pocket No.1	0.0	0.0	0.1	0.1	0.0	**0.1**
Spill Pocket No.1 to -3010 Station	0.0	0.0	19.4	19.4	1.9	**21.3**
-3010L Station	0.0	0.0	0.6	0.6	0.1	**0.6**
-3010L Station to Loading Pocket No.2	0.0	0.0	1.5	1.5	0.2	**1.7**
Loading Pocket No.2	0.0	0.0	0.2	0.2	0.0	**0.2**
Install Loading Pocket	0.0	0.0	1.0	1.0	0.1	**1.1**
Loading Pocket Raise	0.0	0.0	0.4	0.4	0.0	**0.5**
Loading Pocket No.2 to Shaft Bottom	0.0	0.0	0.8	0.8	0.1	**0.8**
Remove Sinking Gear & Commission Shaft	0.0	0.0	0.5	0.5	0.1	**0.6**
Total ($M)	**0.0**	**22.8**	**29.8**	**52.6**	**5.3**	**57.9**

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

21.1.1.3 Pre-production Ramp Capital Costs

A total of 3,254 m of main ramp development will be completed during the pre-production period by a contractor. A summary of pre-production ramp capital costs, and schedule, is presented in Table 21.4.

TABLE 21.4 PRE-PRODUCTION RAMP CAPITAL COST ESTIMATES							
Heading Type	**Unit Cost ($/m)**	**Year ($M)**			**Subtotal ($M)**	**Contingency ($M)**	**Total ($M)**
		-3	**-2**	**-1**			
Ramp 4.9 x 4.9 m	5,312	0.0	5.5	11.7	17.3	0.0	**17.3**

21.1.1.4 Pre-production Shaft Facility Capital Costs

A summary of pre-production capital costs for the shaft headframe, two loading pockets with rockbreakers and grizzlies, two hoists and hoistroom is presented in Table 21.5.

TABLE 21.5 PRE-PRODUCTION SHAFT FACILITY CAPITAL COST						
Description	**Year ($M)**			**Subtotal ($M)**	**Contingency ($M)**	**Total ($M)**
	-3	**-2**	**-1**			
Headframe, hoistroom, hoists (2)	8.3	4.2	0.0	12.5	1.2	**13.7**
Bins, collar house, batch plant, substation	1.7	0.8	0.0	2.5	0.3	**2.8**
Mine and shaft dewatering/pumping system	1.5	0.0	0.0	1.5	0.2	**1.7**
Purchase of skips, cage and hoist ropes, install	0.0	0.0	1.0	1.0	0.1	**1.1**
Loading pockets (2)	0.0	0.0	2.0	2.0	0.2	**2.2**
Grizzly / Rockbreaker (4)	0.0	0.0	1.0	1.0	0.1	**1.1**
Total ($M)	**11.5**	**5.0**	**4.0**	**20.5**	**2.0**	**22.5**

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

21.1.1.5 Pre-production Underground Mine Equipment Capital Costs

All of the underground mining equipment required to complete the pre-production ramp, mine and stope development will be purchased in the pre-production period. A summary of the capital cost of this pre-production underground equipment is presented in Table 21.6.

TABLE 21.6 PRE-PRODUCTION UNDERGROUND MINE EQUIPMENT CAPITAL COST							
Description	**Unit Cost ($)**	**Year ($M)**			**Sub-total ($M)**	**Contin-gency ($M)**	**Total ($M)**
		-3	**-2**	**-1**			
Sandvik DD421 Devel Jumbo - 2 Boom	1,342,200	0.0	0.0	2.7	2.7	0.1	**2.8**
Cubex ITH Drill (DU311)	1,241,200	0.0	0.0	2.5	2.5	0.1	**2.6**
Getman Scissor Lift	311,500	0.0	0.0	0.3	0.3	0.0	**0.3**
Sandvik LH514 6.1 m^3 LHD – Devel.	1,394,000	0.0	0.0	2.8	2.8	0.1	**2.9**
Sandvik LH514 6.1 m^3 LHD - Haulage	1,394,000	0.0	1.4	1.4	2.8	0.1	**2.9**
Sandvik TH551 50 t Haul Truck	1,654,200	0.0	3.3	8.3	11.6	0.6	**12.2**
MCU 2700 UG Blasting Tractor	463,100	0.0	0.0	0.5	0.5	0.0	**0.5**
Getman ANFO Loader	370,500	0.0	0.0	0.4	0.4	0.0	**0.4**
Sandvik DS420 Cable Bolter	1,080,800	0.0	0.0	1.1	1.1	0.1	**1.1**
Getman Lube Service Vehicle	286,300	0.0	0.0	0.3	0.3	0.0	**0.3**
M40 Fuel truck	315,700	0.0	0.0	0.3	0.3	0.0	**0.3**
Mechanics Vehicle	46,300	0.0	0.0	0.0	0.0	0.0	**0.0**
Electrician Vehicle	46,300	0.0	0.0	0.0	0.0	0.0	**0.0**
Getman Boom Truck	273,600	0.0	0.0	0.3	0.3	0.0	**0.3**
Grader	311,500	0.0	0.0	0.3	0.3	0.0	**0.3**
Toyotas	46,300	0.0	0.0	0.3	0.3	0.0	**0.3**
Alimak	252,600	0.0	0.0	0.3	0.3	0.0	**0.3**
Shotcrete Machine	84,200	0.0	0.0	0.1	0.1	0.0	**0.1**
Getman Personnel Carrier	252,600	0.0	0.0	0.3	0.3	0.0	**0.3**
Misc. Underground equipment	Lot	0.0	0.0	1.8	1.8	0.1	**1.9**
Misc. Surface Equipment	Lot	0.0	0.0	2.4	2.4	0.1	**2.5**
Total ($M)		**0.0**	**4.7**	**26.2**	**30.9**	**1.5**	**32.4**

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

21.1.1.6 Pre-production Underground Infrastructure Capital Costs

Dewatering and rehabilitating the existing underground workings will be completed in Year -2, using the existing ramp to access the workings. A summary of this capital cost is presented in Table 21.7.

TABLE 21.7 PRE-PRODUCTION UNDERGROUND INFRASTRUCTURE CAPITAL COST							
Description	**Unit Cost ($)**	**Year ($M)**			**Subtotal ($M)**	**Contin-gency ($M)**	**Total ($M)**
		-3	**-2**	**-1**			
Dewater/Rehab Existing Devel.	2,500,000	0.0	2.5	0.0	2.5	0.3	**2.8**
Install Grizzly / Rockbreaker	50,000	0.0	0.0	0.2	0.2	0.0	**0.2**
Total ($M)		**0.0**	**2.5**	**0.2**	**2.7**	**0.3**	**3.0**

21.1.1.7 Pre-production Capitalized OPEX

During the pre-production period (first three years) all operating costs are capitalized. A summary of these capitalized operating costs is presented in Table 21.8.

TABLE 21.8 PRE-PRODUCTION CAPITALIZED OPERATING COSTS						
Description	**Year ($M)**			**Subtotal ($M)**	**Contin-gency ($M)**	**Total ($M)**
	-3	**-2**	**-1**			
Stope Mining	0.0	0.0	10.2	10.2	0.0	**10.2**
Support Services	3.1	4.9	10.6	18.6	0.9	**19.5**
Paste Backfill	0.0	0.0	9.7	9.7	0.5	**10.2**
Process Plant Commissioning	0.0	2.1	16.8	18.9	1.9	**20.8**
Haulage	0.0	1.8	13.6	15.4	0.8	**16.1**
G&A Costs	4.3	5.4	6.9	16.5	0.8	**17.3**
Total ($M)	**7.4**	**14.2**	**67.7**	**89.3**	**4.9**	**94.2**

21.1.1.8 Summary of Pre-production Underground Capital Costs

A summary of all pre-production underground capital costs is presented in Table 21.9.

TABLE 21.9 SUMMARY OF PRE-PRODUCTION UNDERGROUND CAPITAL COST						
Description	**Year ($M)**			**Subtotal ($M)**	**Contin-gency ($M)**	**Total ($M)**
	-3	**-2**	**-1**			
Capitalized OPEX	7.4	14.2	67.7	89.3	4.9	**94.2**
Mine & Stope Development	5.7	20.0	26.4	52.1	0.0	**52.1**
Shaft Development	0.0	22.8	29.8	52.6	5.3	**57.9**
Ramp Development	0.0	5.5	11.7	17.3	0.0	**17.3**
Shaft Headframe, Hoist & Hoist Room, LP	11.5	5.0	4.0	20.5	2.0	**22.5**
Mine Equipment	0.0	4.7	26.2	30.9	1.5	**32.4**
U/G Infrastructure	0.0	2.5	0.2	2.7	0.3	**3.0**
Total ($M)	**24.6**	**74.7**	**166.0**	**265.3**	**14.0**	**279.4**

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

21.1.1.9 Pre-production Surface Infrastructure Capital Cost Estimates

Pre-production 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.

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, safety 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 pre-production surface infrastructure capital cost summary is presented in Table 21.10.

TABLE 21.10 **SUMMARY OF PRE-PRODUCTION SURFACE INFRASTRUCTURE CAPITAL COST**						
Description	**Year ($M)**			**Subtotal ($M)**	**Contin-gency ($M)**	**Total ($M)**
	-3	**-2**	**-1**			
Site Facilities	6.3	21.2	0.0	27.5	2.7	**30.2**
Buildings	0.7	1.7	0.0	2.5	0.2	**2.7**
Buildings Furnishings	0.5	1.3	0.0	1.8	0.2	**2.0**
Surface Mobile Equipment	0.0	1.5	0.0	1.5	0.1	**1.6**
Total ($M)	**7.5**	**25.7**	**0.0**	**33.2**	**3.3**	**36.5**

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

21.1.1.10 Pre-production Process Plant Capital Cost

The pre-production 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 estimated process plant direct and indirect capital costs is presented in Table 21.11.

TABLE 21.11 PRE-PRODUCTION PROCESS PLANT CAPITAL COST ESTIMATES						
Description	**Year ($M)**			**Subtotal ($M)**	**Contin-gency ($M)**	**Total ($M)**
	-3	**-2**	**-1**			
Direct Costs	**35.0**	**17.5**	**0.0**	**52.5**	**7.9**	**60.4**
Indirect Costs						
Field Supervision	2.1	1.0	0.0	3.1	0.5	**3.6**
Field Expense	1.5	0.8	0.0	2.3	0.3	**2.6**
Temporary Facilities	0.7	0.3	0.0	1.0	0.2	**1.2**
Construction Equipment	0.9	0.4	0.0	1.3	0.2	**1.5**
Worker Trade Benefits	2.2	1.1	0.0	3.3	0.5	**3.8**
Engineering	5.4	2.7	0.0	8.1	1.2	**9.3**
Freight	1.4	0.7	0.0	2.1	0.3	**2.4**
Spare Parts	0.5	0.3	0.0	0.8	0.1	**0.9**
Start-up	0.2	0.1	0.0	0.3	0.0	**0.3**
Total Indirects	**14.8**	**7.4**	**0.0**	**22.3**	**3.3**	**25.6**
Total ($M)	**49.8**	**24.9**	**0.0**	**74.8**	**11.2**	**86.0**

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

21.1.1.11 Pre-production Closure Bond Capital Cost Estimates

A pre-production closure bond will be required to: remove the headframe, hoist and process plant; for final tailings construction and seeding; provide for the permanent spillway; water treatment; removal of surface infrastructure; final clean up, and secure mine openings. It is estimated it will cost $5.2M to complete these closure works.

21.1.2 Sustaining Capital Cost Estimates

Commercial production commences after the three year pre-production period, in the first quarter of the fourth year (Production Yr1). Sustaining capital costs during this period include mine and stope development; ramp development near the bottom of the mine in Production Yr10 and Yr11; underground mining equipment; underground infrastructure; Project closure bond contributions; a salvage value in Production Yr14, and a contingency allowance. The total sustaining capital cost is estimated to be $303.5 M. A summary of the sustaining capital cost estimate and schedule for the commercial production period are provided in Table 21.12.

TABLE 21.12 SUMMARY OF SUSTAINING CAPITAL COSTS																	
Description	**Production Year ($M)**														**Sub-total ($M)**	**Contin-gency ($M)**	**Total ($M)**
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**			
Mine & Stope Development	35.5	17.9	22.3	21.8	19.4	19.9	12.1	27.8	14.7	17.8	15.6	4.5	0.0	0.0	229.3	0.0	**229.3**
Ramp Development	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	9.2	4.7	0.0	0.0	0.0	13.8	0.0	**13.8**
Mine Equipment - U/G	8.1	0.0	0.0	1.4	3.3	3.0	21.2	4.7	3.0	4.7	1.7	3.0	1.7	0.0	55.9	2.8	**58.7**
U/G Infrastructure	4.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0	0.0	0.0	4.8	0.5	**5.3**
Surface Infrastructure	0.0	0.0	0.0	0.0	0.0	0.0	1.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.5	0.1	**1.6**
Closure Bond & Salvage	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	-7.8	-4.5	-0.7	**-5.2**
Total Sustaining Capital ($M)	**48.8**	**18.2**	**22.6**	**23.6**	**23.1**	**23.2**	**35.1**	**32.8**	**18.1**	**32.0**	**22.2**	**7.8**	**1.9**	**-7.8**	**300.7**	**2.7**	**303.5**

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.

21.1.2.1 Mine and Stope Development Sustaining Capital Costs

Mine and stope development sustaining capital costs include the cost of all underground development in both waste rock and mineralized rock, excluding all slot raises and shaft and shaft related excavations. This includes: the cost of all cross-cuts; drifting in mineralization; sumps, electrical rooms, lunchrooms, re-muck bays, garages and ventilation raises. A summary of mine and stope development capital costs is presented in Table 21.13.

TABLE 21.13 MINE AND STOPE DEVELOPMENT SUSTAINING CAPITAL COSTS																	
Heading Type	**Unit Cost ($/m)**	**Production Year ($M)**													**Sub-total ($M)**	**Contin-gency ($M)**	**Total ($M)**
		1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**			
X-cut 4.9 x 4.9 m	3,268	21.9	4.8	9.9	9.3	7.3	7.0	0.0	14.7	2.6	3.5	4.6	0.0	0.0	0.0	85.4	0.0	**85.4**
V3.0 x 3.0 m	3,086	2.7	0.4	0.3	0.4	0.0	0.7	0.0	1.0	0.0	0.4	0.7	0.0	0.0	0.0	6.7	0.0	**6.7**
H10.0 x 4.9 m	5,351	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	**0.3**
H7.0 x 7.9 m	5,887	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.0	**0.4**
H9.1 x 4.9 m	4,983	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
H8.2 x 4.9 m	4,615	0.3	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.5	0.0	**0.5**
H5.5 x 4.9 m	3,513	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
H4.0 x 4.0 m	2,723	10.2	12.1	12.1	12.1	12.1	12.1	12.1	12.1	12.1	13.9	10.3	4.5	0.0	0.0	135.9	0.0	**135.9**
Total ($M)		**35.5**	**17.9**	**22.3**	**21.8**	**19.4**	**19.9**	**12.1**	**27.8**	**14.7**	**17.8**	**15.6**	**4.5**	**0.0**	**0.0**	**229.3**	**0.0**	**229.3**

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

21.1.3 Ramp Development Sustaining Capital Cost

The main -15% ramp will be developed as required, from a depth of 917 m (3,010 ft) to a depth of 1,259 m (4,130 ft). The total sustaining capital cost of ramp development is $13.8 M. A scheduled summary of this estimate is presented in Table 21.14.

TABLE 21.14 RAMP DEVELOPMENT SUSTAINING CAPITAL COSTS																		
Heading Type	**Unit Cost ($/m)**	**Production Year ($M)**														**Sub-total ($M)**	**Contin-gency ($M)**	**Total ($M)**
		1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**			
Ramp 4.9 x 4.9 m	5,312	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	9.2	4.6	0.0	0.0	0.0	13.8	0.0	**13.8**

21.1.4 Sustaining Underground Equipment Capital Cost

Approximately $58.7 M of sustaining capital will be spent for underground equipment during the production period, mainly for stope drilling, blasting, mucking and haulage. These costs include: all underground mobile and stationary equipment and all related mine surface equipment. In Year 7 most of the equipment will have to be replaced. A schedule of sustaining capital expenditures for underground equipment is presented in Table 21.15.

Table 21.15 Underground Equipment Sustaining Capital Costs																	
Equipment Description	**Production Year ($M)**														**Sub-total ($M)**	**Contin-gency ($M)**	**Total ($M)**
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**			
Sandvik DD421 Devel Jumbo - 2 Boom	1.3	0.0	0.0	0.0	0.0	0.0	4.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5.4	0.3	**5.6**
Cubex ITH Drill (DU311)	1.2	0.0	0.0	0.0	0.0	0.0	3.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5.0	0.2	**5.2**
Getman Scissor Lift	0.3	0.0	0.0	0.0	0.0	0.0	0.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.9	0.0	**1.0**
Sandvik LH514 6.1 m^3 LHD - Development	0.0	0.0	0.0	0.0	0.0	0.0	2.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.8	0.1	**2.9**
Sandvik LH514 6.1 m^3 LHD - Haulage	2.8	0.0	0.0	1.4	0.0	1.4	1.4	1.4	1.4	1.4	0.0	1.4	0.0	0.0	12.5	0.6	**13.2**
Sandvik TH551 50 t Haul Truck	0.0	0.0	0.0	0.0	3.3	1.7	1.7	3.3	1.7	3.3	1.7	1.7	1.7	0.0	19.9	1.0	**20.8**
MCU 2700 UG Blasting Tractor	0.5	0.0	0.0	0.0	0.0	0.0	0.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.4	0.1	**1.5**
Getman ANFO Loader	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.0	**0.4**
Sandvik DS420 Cable Bolter	0.0	0.0	0.0	0.0	0.0	0.0	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.1	0.1	**1.1**
Getman Lube Service Vehicle	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	**0.3**
M40 Fuel truck	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	**0.3**
Mechanics Vehicle	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
Electrician Vehicle	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
Getman Boom Truck	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	**0.3**
Grader	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	**0.3**
Toyotas	0.2	0.0	0.0	0.0	0.0	0.0	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.6	0.0	**0.7**
Alimak	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	**0.3**
Shotcrete Machine	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	**0.1**
Getman Personnel Carrier	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.0	**0.3**
Misc. Underground equipment	1.7	0.0	0.0	0.0	0.0	0.0	1.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	3.6	0.2	**3.8**
Misc. Surface Equipment	0.1	0.0	0.0	0.0	0.0	0.0	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.5	0.0	**0.5**
Total ($M)	**8.1**	**0.0**	**0.0**	**1.4**	**3.3**	**3.0**	**21.2**	**4.7**	**3.0**	**4.7**	**1.7**	**3.0**	**1.7**	**0.0**	**55.9**	**2.8**	**58.7**

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

21.1.5 Underground Infrastructure Sustaining Capital Costs

Underground infrastructure sustaining capitals costs include expenditures for: two underground shops, one on the 472 m (1,550 ft) Level and a second on the 917 m (3,010 ft) Level; two main sumps; a mine air heating system; four lunchroom / refuge stations; four powder magazines, and 30 ventilation bulkhead / regulators. A summary underground infrastructure sustaining capital costs is presented in Table 21.16.

TABLE 21.16 UNDERGROUND INFRASTRUCTURE SUSTAINING CAPITAL COSTS																	
Description	**Unit Cost ($)**	**Production Year ($M)**														**Sub-total ($M)**	**Contin-gency ($M)**	**Total ($M)**
		1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**			
Underground Shop	1,000,000	2.0														2.0	0.2	2.2
Sump	200,000	0.4														0.4	0.0	0.4
Mine Air Heaters	500,000	0.5														0.5	0.1	0.6
Refuge Station	150,000	0.6														0.6	0.1	0.7
Latrine	40,000	0.2														0.2	0.0	0.2
Powder Magazine	50,000	0.2														0.2	0.0	0.2
Detonator Magazine	20,000	0.1														0.1	0.0	0.1
Ventilation Walls and Regulators	30,000	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.9	0.1	1.0
Total ($M)		**4.1**	**0.1**	**0.1**	**0.1**	**0.1**	**0.1**	**0.1**	**0.1**	**0.1**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**4.8**	**0.5**	**5.3**

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

21.1.5.1 Surface Infrastructure Sustaining Capital Costs

Surface infrastructure sustaining capitals costs includes the cost of replacing surface mobile equipment in production year 7, as summarized in Table 21.17.

TABLE 21.17
SURFACE INFRASTRUCTURE SUSTAINING CAPITAL COSTS

Item	Quantity	Unit Cost ($)	Subtotal ($M)	Contingency ($M)	Total ($M)
Motor Grader	1	250,000	0.25	0.025	0.28
FEL - Cat 930H	1	220,000	0.22	0.022	0.24
Flatbed Truck	1	25,000	0.03	0.003	0.03
Garbage Truck w/dumpsters (used)	1	50,000	0.05	0.005	0.06
Buses - 30 Person	2	100,000	0.20	0.020	0.22
Ambulance	1	150,000	0.15	0.015	0.17
Fire/Rescue Truck	1	197,500	0.20	0.020	0.22
Pickup Trucks	6	50,000	0.30	0.030	0.33
SUV	1	60,000	0.06	0.006	0.07
Total Surface Mobile Equipment ($M)			**1.45**	**0.145**	**1.60**

21.1.5.2 Mine Closure and Salvage Sustaining Capital Costs

A closure bond, and/or other form of financial assurance acceptable to the Ontario Ministry of Energy, Northern Development and Mines, will be required for the cost to remove the process plant, for final tailings construction and seeding; the tailings permanent spillway, final water treatment, remove surface infrastructure and final clean up. The sustaining capital closure costs are estimated to be offset by the salvage value of these facilities, LOM. The sustaining mine closure and salvage capital cost is summarized in Table 21.18.

TABLE 21.18 MINE CLOSURE AND SALVAGE SUSTAINING CAPITAL COSTS																	
Description	**Year ($M)**														**Sub-total ($M)**	**Contin-gency ($M)**	**Total ($M)**
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**			
Closure:																	
Tailing management area	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	1.8	0.3	**2.0**
Remove process plant	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	**0.1**
Final tailings dam work	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.1	**0.4**
Spillway	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
Final water treatment (batch)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
Treatment / monitoring	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.9	0.1	**1.0**
Miscellaneous	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.1	**0.5**
Closure Subtotal	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**3.5**	**0.5**	**4.0**
Salvage:																	
Hoists(2)														-1.7	-1.7	-0.3	**-2.0**
Headframe														-0.1	-0.1	0.0	**-0.1**
Mine Equipment														-2.1	-2.1	-0.3	**-2.4**
Infrastructure														-4.1	-4.1	-0.6	**-4.7**
Salvage Subtotal														**-8.0**	**-8.0**	**-1.2**	**-9.2**
Total closure/ salvage ($M)	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**0.3**	**-7.8**	**-4.5**	**-0.7**	**-5.2**

21.2 OPERATING COST ESTIMATES

Operating costs include the cost of operating labour, maintenance labour, electrical power, operating materials and supplies, reagents and fuel. A $3.01 per tonne (5.4%) contingency has been included. The yearly operating cost varies from $56.73 to $60.89 per tonne processed. A summary of the average operating cost estimates for the Thierry Project is provided in Table 21.19. Underground mining is estimated to average $38.64 per tonne processed over the LOM.

TABLE 21.19
SUMMARY OF AVERAGE OPERATING COST PER TONNE PROCESSED

Description	Subtotal ($/t)	Contingency ($/t)	Total ($/t)
U/G Stope Mining	8.35	0.00	**8.35**
U/G Support Services	13.36	0.67	**14.02**
U/G Haulage	8.76	0.44	**9.20**
Paste Backfill	6.73	0.34	**7.06**
Process Plant	13.15	1.32	**14.47**
G&A	5.05	0.25	**5.30**
Total Operating Cost	**55.40**	**3.01**	**58.41**

Details of these estimates are provided in the following subsections.

21.2.1 Underground Stope Mining

On average 3,421 tpd of process plant feed would be mined by stoping in the underground mine. An additional 579 tpd would be extracted by stope development, for a total of 4,000 tpd of process plant feed. Stope mining operating costs include the cost of material, consumables and direct labour for stope drilling, blasting, slot raise, ground support, pipe and accessories, stope ventilation and services. The estimated operating cost of stope mineralized rock mined is summarized in Table 21.20. Note that the mine and stope development costs have been included in the capital cost summaries.

TABLE 21.20 STOPE MINING OPERATING COST		
Description	**Stoping ($/t)**	**Total Mine ($/t)**
LOM Tonnes	15,737,385	18,200,383
Consumables		
Drilling & Blasting	3.75	3.24
Slot Raise	0.11	0.09
Ground Support	0.44	0.38
Pipe & Accessories	0.09	0.08
Stope Fan	0.06	0.05
Consumables Subtotal	**4.46**	**3.85**
Services	0.16	0.14
Direct Mine Labour	5.04	4.36
Total	**9.66**	**8.35**

Notes:

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

2) The column labelled 'Stoping' refers to process plant feed produced by the stoping process and the costs associated with this. The column labelled 'Total Mine' includes feed from stoping plus feed produced directly from lateral development work. The cost of lateral development work is included in the capital estimates for the mine.

21.2.2 Underground Support Services

Underground support services include the cost of underground supervision and technical staff, support labour including: backfill labour, hoist support labour, cagetenders, deckmen, grizzly operators, service leaders, grader operators, pump/construction operators, service truck operators and mine labourers. It also includes the cost of mine air heating, mine surface vehicle operation and maintenance, underground support vehicle operation and maintenance and the cost of all electric power to service the underground. A summary of these operating costs per tonne processed on a yearly basis, is presented in Table 21.21.

TABLE 21.21
UNDERGROUND SUPPORT SERVICES OPERATING COSTS

Description	Production Year ($/t)														Average	Contin-gency	Total
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	($/t)	($/t)	($/t)
Mine Staff Labour	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	2.76	0.14	**2.90**
Mine Direct Labour	4.43	4.39	4.39	4.39	4.39	4.39	4.39	4.39	4.39	4.39	4.39	4.39	4.39	4.39	4.39	0.22	**4.61**
Mine Air Heating	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	4.94	1.70	0.09	**1.79**
Surface Equipment & Support Vehicles	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.02	**0.33**
Underground Support Vehicles	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	0.07	**1.41**
Power	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	2.84	0.14	**2.98**
Total ($/t)	**13.39**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**13.35**	**16.59**	**13.36**	**0.67**	**14.02**

21.2.3 Underground Haulage

All development and stope mineralized rock will either be hauled from the base of the orepasses on the 472 m (1,550 ft) and of 917 m (3,010 ft) levels, to the grizzly dumps on those levels, or from the 137 m (450 ft) to 290 m (950 ft) levels up the ramp to surface. Thus, only mineralized rock below the 290 m (950 ft) level will be hoisted up the shaft. The shaft will only be used for mineralized rock and waste rock hoisting and transporting workers to-and-from the 472 m (1,550 ft) and the 917 m (3,010 ft) levels. A summary of the estimated cost for underground haulage is presented in Table 21.22.

TABLE 21.22
SUMMARY OF UNDERGROUND HAULAGE MINING OPERATING COSTS

Cost Item	Production Year ($M)														Sub-total ($M)	Contin-gency ($M)	Total ($M)
	1	2	3	4	5	6	7	8	9	10	11	12	13	14			
Truck Operating Costs	5.09	5.37	5.53	6.00	6.18	6.39	6.51	6.45	6.33	6.30	6.25	5.93	5.30	5.34	5.97	0.30	**6.27**
LHD Operating Costs	2.00	2.92	2.93	3.07	3.13	3.08	2.98	2.97	2.87	2.80	2.70	2.71	2.06	1.30	2.79	0.14	**2.93**
Haulage Total Cost / Tonne	**7.09**	**8.29**	**8.46**	**9.08**	**9.31**	**9.48**	**9.49**	**9.43**	**9.20**	**9.10**	**8.95**	**8.64**	**7.36**	**6.64**	**8.76**	**0.44**	**9.20**
Haulage Total Cost ($M)	**9.9**	**11.6**	**11.8**	**12.7**	**13.0**	**13.3**	**13.3**	**13.2**	**12.9**	**12.7**	**12.5**	**12.1**	**10.3**	**0.0**	**159.4**	**8.0**	**167.4**

21.2.4 Paste Backfill

Paste backfill will be required to replace all mined mineralized material. Operating costs include binder, plant operator, testing, plant maintenance, reticulation holes, pipe runs to the stopes, plant consumables, and barricades. Underground backfill labour is included in support services operating costs. A summary paste backfill operating costs is presented in Table 21.23.

TABLE 21.23
PASTE BACKFILL OPERATING COST DETAILS

Description	Cost $/m^3$
Binder	16.36
Plant operator	2.44
Testing allowance	0.33
Plant maintenance	1.90
Reticulation vertical holes	1.19
Supply & install temp reticulation pipe runs to stopes	1.76
Plant consumables	0.33
Barricades	7.18
Backfill Cost $/m^3	**31.48**
Backfill cost per tonne insitu process plant feed	6.73
Contingency $/t processed	0.34
Total Paste Backfill Cost $/t processed	**7.06**
Annual Paste Backfill Cost	**$9.9 M**

21.2.5 Process Plant

The average processing rate is 4,000 tpd, or 1,400,000 tpy, with a nominal process plant design capacity of approximately 4,500 tpd, corresponding to a plant availability of 89%. Costs include all electrical power requirements, reagents, operating and maintenance supplies and labour. A summary of process plant operating costs, per tonne processed and total cost per year, is presented in Table 21.24.

TABLE 21.24 PROCESS PLANT OPERATING COSTS			
Item	**Annual ($M)**	**Processing ($/t)**	**Percent of Total (%)**
Labour	6.9	4.96	34
Power and Fuel	4.3	3.07	21
Consumables and Maintenance Supplies	6.1	4.36	30
Mobile Equipment	0.0	0.02	0
Tailings	1.1	0.75	5
Contingency	1.8	1.32	9
Total Cost	**20.3**	**14.47**	**100**

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

21.2.6 General and Administration (G&A)

The general and administration ("G&A") cost items include site administrative staff, surface support vehicles, office expenses, environmental/permitting, safety equipment and personal protective equipment, insurance, community support and professional services cost. A 5% contingency has been added to these costs. A summary of G&A costs per tonne processed is presented in Table 21.25.

Table 21.25 Summary of Site General & Administrative Operating Costs																	
Cost Item	**Production Year ($M)**														**Sub-Total ($M)**	**Contin-gency ($M)**	**Total ($M)**
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**			
Site Administration Staff	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	5.1	0.0	66.6	3.3	**69.9**
Surface Equipment & Support Vehicles	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.0	5.2	0.3	**5.4**
Office Expenses	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	2.6	0.1	**2.7**
Environmental / Permitting	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.0	6.5	0.3	**6.8**
Software/Safety	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.0	3.3	0.2	**3.4**
Insurance	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.0	6.5	0.3	**6.8**
Community	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	1.3	0.1	**1.4**
Total ($M)	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**7.1**	**0.0**	**91.9**	**4.6**	**96.5**
Total ($/t)	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**5.05**	**0.25**	**5.30**

22.0 ECONOMIC ANALYSIS

This Technical Report is considered by P&E Mining Consultants Inc. to meet the requirements of a Preliminary Economic Assessment as defined in Canadian NI 43-101 Standards of Disclosure for Mineral Projects. This PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized. There is no guarantee that Braveheart will be successful in obtaining any or all of the requisite consents, permits or approvals, regulatory or otherwise for the Project to be placed into production.

A financial model was developed to estimate the LOM plan comprised of mining the Measured, Indicated and Inferred Mineral Resources of the Thierry Project. The LOM plan covers a 17-year period. Currency is in Q4 2020 Canadian dollars unless otherwise stated. Inflation has not been considered in the financial analysis. Millions of dollars are stated as $ M.

22.1 ECONOMIC CRITERIA

22.1.1 Physical Parameters

Mine life:	
Pre-production	Year 1 to 3 (36 months)
Production Mining/Processing	Year 4 to 17 for a total of 13.0 years
Decommissioning	6 months in Year 17
Production rate:	4,000 tpd @ 350 days/year, 1.4 Mtpa
Total production:	
Total mineralized rock production	19.6 Mt at 1.46% Cu and 0.16% Ni
Total concentrate production	Cu Conc. 880,300 t; Ni-Cu Conc. 157,100 t
Metallurgical parameters:	
Process recovery	93% Cu and 55% Ni
Concentration ratio	Cu Conc. 22.3; Ni-Cu Conc. 125
Concentrate grade	Cu Conc. 30.0% Cu; Ni-Cu Conc. 8.0% Ni
Concentrate moisture content	10%
Total payable metal:	
Copper	257,200 t Cu (567 M lb)
Nickel	9,400 t Ni (21 M lb)

22.1.2 Revenue

The commercially saleable products generated by the Project would be copper and nickel-copper concentrates. Braveheart would be paid once the concentrates have been delivered to the smelter and refinery, off-site. The metal prices used in this PEA are US$3.48/lb Cu, US$8.00/lb Ni, US$21/oz Ag, US$1,600/oz Au, US$1,100/oz Pt and US$1,250/oz Pd. Revenues were estimated as Net Smelter Returns ("NSR"). The copper concentrates were estimated to contain mainly payable Cu, with no revenue from Ni or Pt, and minor payable amounts of Ag, Au and Pd. The nickel-copper concentrates were estimated to contain mainly payable Ni, with minor payable Cu, and no payable amounts of Ag, Au, Pt or Pd. The NSR payables were based on the following parameters.

Smelter treatment charge	Cu Conc. US$/DMT: $85/t; Ni-Cu Conc. US$/DMT: $175/t
Concentrate shipping charge	$/WMT: $90/t
Smelter payable	Cu Conc. – 96.5% Cu & 0% Ni; Ni-Cu Conc. - 75% Ni & 75% Cu
Cu concentrate refining charges	US$0.085/lb Cu; US$0.50/lb Ni
Ni-Cu concentrate refining charges	US$0.10/lb Cu; US$0.50/lb Ni

The CDN$/US$ exchange rate used in the PEA is 0.75.

Net revenue:	$2,579 M

22.1.3 Costs

Operating costs:	
Total average cost:	$58.41/t processed
Cash Cost	US$1.08/lb CuEq, net of by-product credits
All-in sustaining cost ("AISC")	US$1.98/lb CuEq, net of by-product credits

Capital costs:	
Preproduction	$407.0 M
Sustaining	$303.5 M
Total capital costs	$710.5 M

Capital costs include the cost of pre-production stope mining, support services, paste backfill, process plant, mined material haulage, and G&A costs; and production mine and stope development; shaft sinking, ramp development, the shaft headframe, hoists, hoist room, shaft stations and loading pockets; surface power line; mine equipment; surface infrastructure; underground infrastructure; the process plant, a closure bond and salvage value; and a 5% contingency.

22.2 CASH FLOW

An after-tax financial model has been developed for the Thierry 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.

An after-tax cash flow summary is presented in Table 22.1. All estimated costs are in Q4 2020 Canadian dollars with no allowance for inflation.

TABLE 22.1
CASH FLOW SUMMARY

Description / Year	Unit	Year																	Total
		-3	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	
Production																			
Development Tonnes	kt		159.0	221.2	184.2	219.1	219.0	219.2	219.2	219.7	220.4	220.3	220.3	220.4	220.2	81.0			2,843.2
Copper	%		1.35	1.27	1.49	1.56	1.52	1.56	1.50	1.42	1.37	1.39	1.48	1.49	1.46	1.47			1.45
Nickel	%		0.16	0.18	0.17	0.16	0.16	0.17	0.17	0.16	0.16	0.15	0.15	0.16	0.13	0.11			0.16
Stoping Tonnes	kt			1,057.2	1,215.8	1,180.9	1,181.0	1,180.8	1,180.8	1,180.3	1,179.6	1,179.7	1,179.7	1,179.6	1,179.8	1,319.0	1,400.0	0.4	16,794.6
Copper	%			1.36	1.28	1.57	1.54	1.55	1.57	1.50	1.37	1.37	1.44	1.49	1.49	1.49	1.46	1.37	1.46
Nickel	%			0.16	0.19	0.16	0.16	0.17	0.17	0.17	0.15	0.15	0.16	0.16	0.16	0.15	0.13	0.08	0.16
Total Tonnes	**kt**		**159.0**	**1,278.4**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**1,400.0**	**0.4**	**19,637.8**
Copper	%		1.35	1.35	1.31	1.57	1.54	1.55	1.56	1.48	1.37	1.38	1.45	1.49	1.48	1.49	1.46	1.37	1.46
Nickel	%		0.16	0.16	0.18	0.16	0.16	0.17	0.17	0.17	0.15	0.15	0.16	0.16	0.15	0.15	0.13	0.08	0.16
Gold	g/t		0.02	0.02	0.04	0.10	0.09	0.07	0.07	0.06	0.05	0.06	0.06	0.07	0.07	0.09	0.12	0.13	0.07
Platinum	g/t		0.02	0.02	0.03	0.08	0.07	0.06	0.06	0.05	0.03	0.04	0.05	0.05	0.05	0.06	0.09	0.08	0.05
Palladium	g/t		0.05	0.05	0.09	0.20	0.18	0.15	0.15	0.13	0.11	0.11	0.13	0.15	0.15	0.18	0.25	0.25	0.14
Silver	g/t		2.91	2.89	3.77	6.95	6.17	5.27	5.18	4.80	4.51	4.65	4.84	4.79	4.83	5.77	6.57	6.99	5.07
NSR	$/t		121.44	120.86	119.25	140.84	137.99	138.98	140.18	133.31	123.47	123.79	129.72	133.12	132.52	133.18	131.63	118.04	131.33
Total Revenue	**M$**		19.3	154.5	166.9	197.2	193.2	194.6	196.3	186.6	172.9	173.3	181.6	186.4	185.5	186.5	184.3	0.05	2,579.0
OPEX																			
Stope Mining	M$				11.7	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	12.7	13.5	0.00	152.0
Support	M$				19.7	19.6	19.6	19.6	19.6	19.6	19.6	19.6	19.6	19.6	19.6	19.6	19.6	0.01	255.2

TABLE 22.1
CASH FLOW SUMMARY

Description / Year	**Unit**	**Year**																	**Total**
		-3	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	
Services																			
Paste Backfill	M$				9.9	9.9	9.9	9.9	9.9	9.9	9.9	9.9	9.9	9.9	9.9	9.9	9.9	0.00	128.6
Process Plant	M$				20.3	20.3	20.3	20.3	20.3	20.3	20.3	20.3	20.3	20.3	20.3	20.3	20.3	0.01	263.3
Haulage	M$				10.4	12.2	12.4	13.3	13.7	13.9	13.9	13.9	13.5	13.4	13.2	12.7	10.8	0.00	167.4
G&A Costs	M$				7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	0.00	96.5
Total OPEX	**M$**				**79.4**	**80.8**	**81.0**	**81.9**	**82.3**	**82.5**	**82.5**	**82.4**	**82.1**	**82.0**	**81.8**	**82.6**	**81.5**	**0.02**	**1,063.0**
EBITDA Cash	**M$**		19.3	154.5	87.5	116.4	112.1	112.6	114.0	104.1	90.3	90.9	99.5	104.4	103.8	103.8	102.7	0.0	1,516.0
CAPEX																			
Stope Mining	M$			10.2															10.2
Support Services	M$	3.2	5.2	11.1															19.5
Paste Backfill	M$			10.2															10.2
Process Plant Commissioning	M$		2.3	18.5															20.8
Haulage	M$		1.9	14.3															16.1
G&A Costs	M$	4.5	5.6	7.2															17.3
Mine & Stope Development	M$	5.7	20.0	26.4	35.5	17.9	22.3	21.8	19.4	19.9	12.1	27.8	14.7	17.8	15.6	4.5			281.4
Shaft Development	M$		25.1	32.7															57.9
Ramp Development	M$		5.5	11.7										9.2	4.7				31.1
Headframe, Hoist & Hoist Room, LP	M$	12.6	5.5	4.4															22.5

TABLE 22.1 CASH FLOW SUMMARY																			
Description / Year	**Unit**	**Year**																	**Total**
		-3	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	
Mine Equipment	M$		4.9	27.5	8.5			1.5	3.5	3.2	22.2	4.9	3.2	4.9	1.7	3.2	1.7		91.1
U/G Infrastructure	M$		2.8	0.2	4.5	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1					8.3
Surface Infrastructure	M$	8.3	28.2								1.6								38.1
Process Plant	M$	57.3	28.7																86.0
Closure Bond & Salvage	M$	4.6	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	-8.9	0.0
Total CAPEX	**M$**	**96.3**	**136.0**	**174.8**	**48.8**	**18.2**	**22.6**	**23.7**	**23.3**	**23.4**	**36.3**	**33.1**	**18.3**	**32.3**	**22.3**	**8.0**	**2.1**	**-8.9**	**710.5**
Taxes	**M$**					4.4	6.1	15.8	27.4	25.3	20.0	20.3	24.9	26.2	26.8	28.8	30.5	0.0	256.4
After-Tax CF	**M$**	-96.3	-116.7	-20.3	38.7	93.8	83.4	73.2	63.3	55.4	33.9	37.5	56.3	45.9	54.7	67.0	70.2	8.9	549.1
After-Tax CCF	M$	-96.3	-213.0	-233.2	-194.5	-100.7	-17.4	55.8	119.2	174.6	208.5	246.1	302.3	348.3	403.0	470.0	540.2	549.1	
After-tax IRR			**%**	**18.9**															
After-tax NPV @ 6%			**M$**	**240.4**															

22.3 BASE CASE CASH FLOW ANALYSIS

The following after-tax cash flow analysis was completed:

- Net Present Value ("NPV") (at 5%, 6%, 7%, 8%, 9% and 10% discount rates).
- 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
After-Tax CF NPV at:	0%	(M$)	549.1
	5%	(M$)	277.5
	6%	(M$)	240.4
	7%	(M$)	207.4
	8%	(M$)	177.9
	9%	(M$)	151.6
	10%	(M$)	128.1
Internal Rate of Return		%	18.9
Project Payback Period in Years		Years	3.2

The Project was evaluated on an after-tax cash flow basis which generates a net undiscounted cash flow estimated at $549.1 M. This results in an after-tax IRR of 18.9% and an after-tax NPV of $240.4 M when using a 6% discount rate. In the base case scenario, the Project has a payback period of 3.2 years from the start of commercial production. The average life-of-mine cash cost is US$1.08/lb copper, net of nickel and by-product credits, at an average operating cost of $58.41/t processed. The average life-of-mine all-in sustaining cost ("AISC") is estimated at US$1.98/lb copper, net of nickel and by-product credits.

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:

- Copper metal price;
- Nickel metal price;
- Operating costs;
- Capital costs;
- Copper head grade; and
- Copper recoveries in copper concentrate.

Each of the sensitivity items were varied up and down by 10% and 20% to assess the effect it would have on the NPV at a 6% discount rate. The value of each parameter, at 80%, 90%, base, 110% and 120%, is presented in Table 22.3.

TABLE 22.3
SENSITIVITY PARAMETER VALUES

Parameter	80%	90%	100%	110%	120%
OPEX (M$)	850	957	1,063	1,169	1,276
CAPEX (M$)	568	639	710	782	853
Cu Price (US$/lb)	2.78	3.13	3.48	3.83	4.18
Ni Price (US$/lb)	6.40	7.20	8.00	8.80	9.60
Cu Head Grade (g/t)	1.17	1.32	1.46	1.61	1.75
Cu Recoveries in Cu Conc. (%)	N/A	82.8	92.0	N/A	N/A

The resultant after-tax NPV @ 6%, and IRR values of each of the sensitivity parameters at 80% to 120% is presented in Tables 22.4 and 22.5, and Figures 22.1 and 22.2. The after-tax IRR at various copper and nickel metal prices is presented in Table 22.6.

TABLE 22.4
AFTER-TAX NPV SENSITIVITY AT 6% DISCOUNT RATE (M$)

Parameter	80%	90%	100%	110%	120%
OPEX	330.5	285.5	240.4	193.5	140.8
CAPEX	316.0	278.1	240.4	199.4	152.6
Cu Price	6.6	127.1	240.4	341.7	443.4
Ni Price	223.1	231.7	240.4	249.0	257.7
Cu Head Grade	31.6	139.1	240.4	331.6	423.4
Cu Recoveries in Cu Conc.	N/A	140.1	240.4	N/A	N/A

TABLE 22.5
AFTER-TAX IRR (%) SENSITIVITY

Parameter	80%	90%	100%	110%	120%
OPEX	23.6	21.3	18.9	16.5	13.7
CAPEX	27.4	22.6	18.9	15.8	12.9
Cu Price	6.4	13.1	18.9	24.0	28.9
Ni Price	18.0	18.5	18.9	19.4	19.8
Cu Head Grade	7.8	13.7	18.9	23.5	28.0
Cu Recoveries in Cu Conc.	N/A	13.8	18.9	N/A	N/A

TABLE 22.6 **AFTER-TAX IRR (%) AT VARIOUS METAL PRICES**					
Cu Price (US$/lb)	**Ni Price (US$/lb)**				
	7.50	**7.75**	**8.00**	**8.25**	**8.50**
3.00	10.3	10.5	10.7	10.8	11.0
3.25	14.9	15.0	15.2	15.4	15.6
3.50	18.9	19.1	19.2	19.4	19.5
3.75	22.6	22.7	22.8	23.0	23.1
4.00	26.1	26.3	26.4	26.5	26.7

FIGURE 22.1 AFTER-TAX NPV SENSITIVITY GRAPH

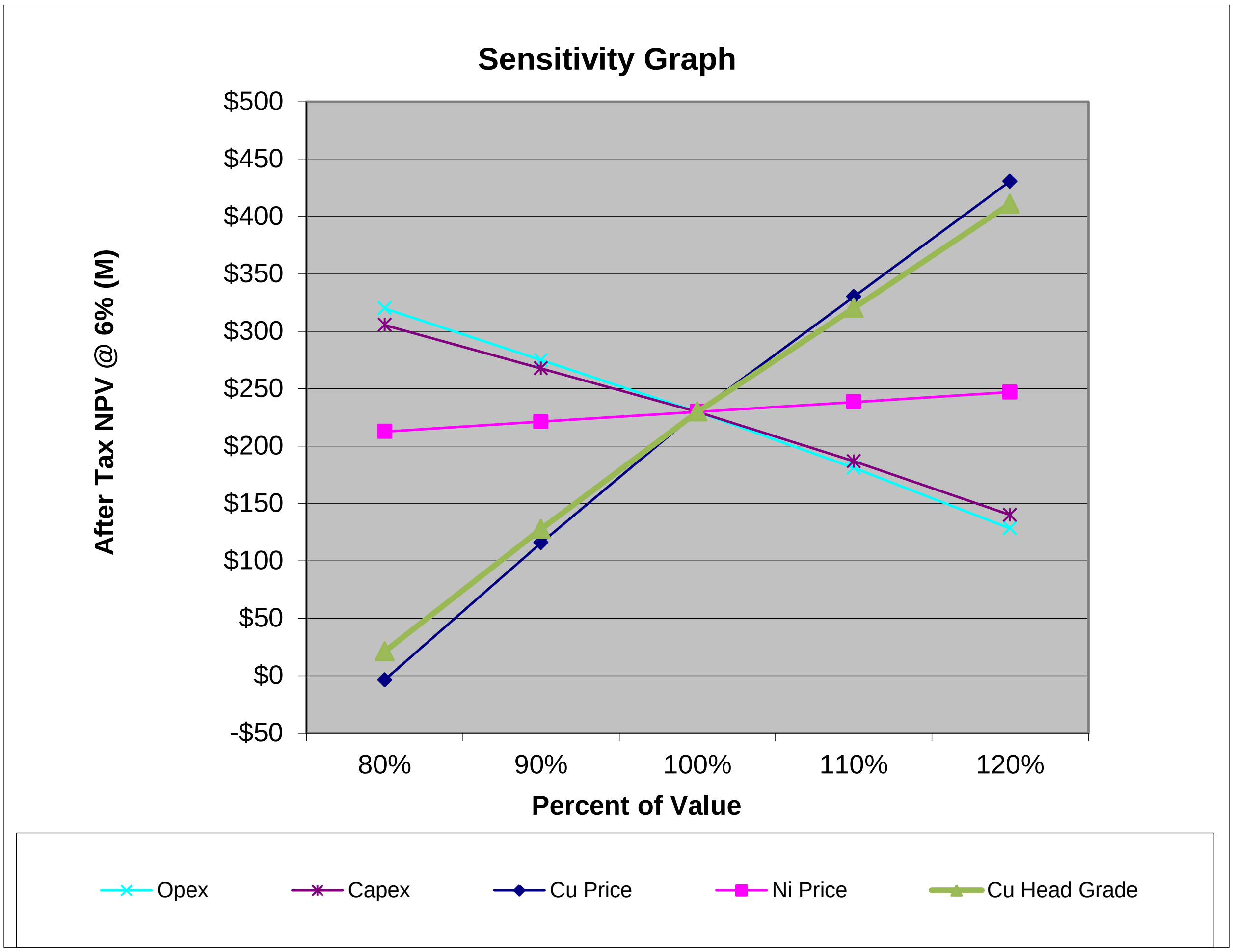
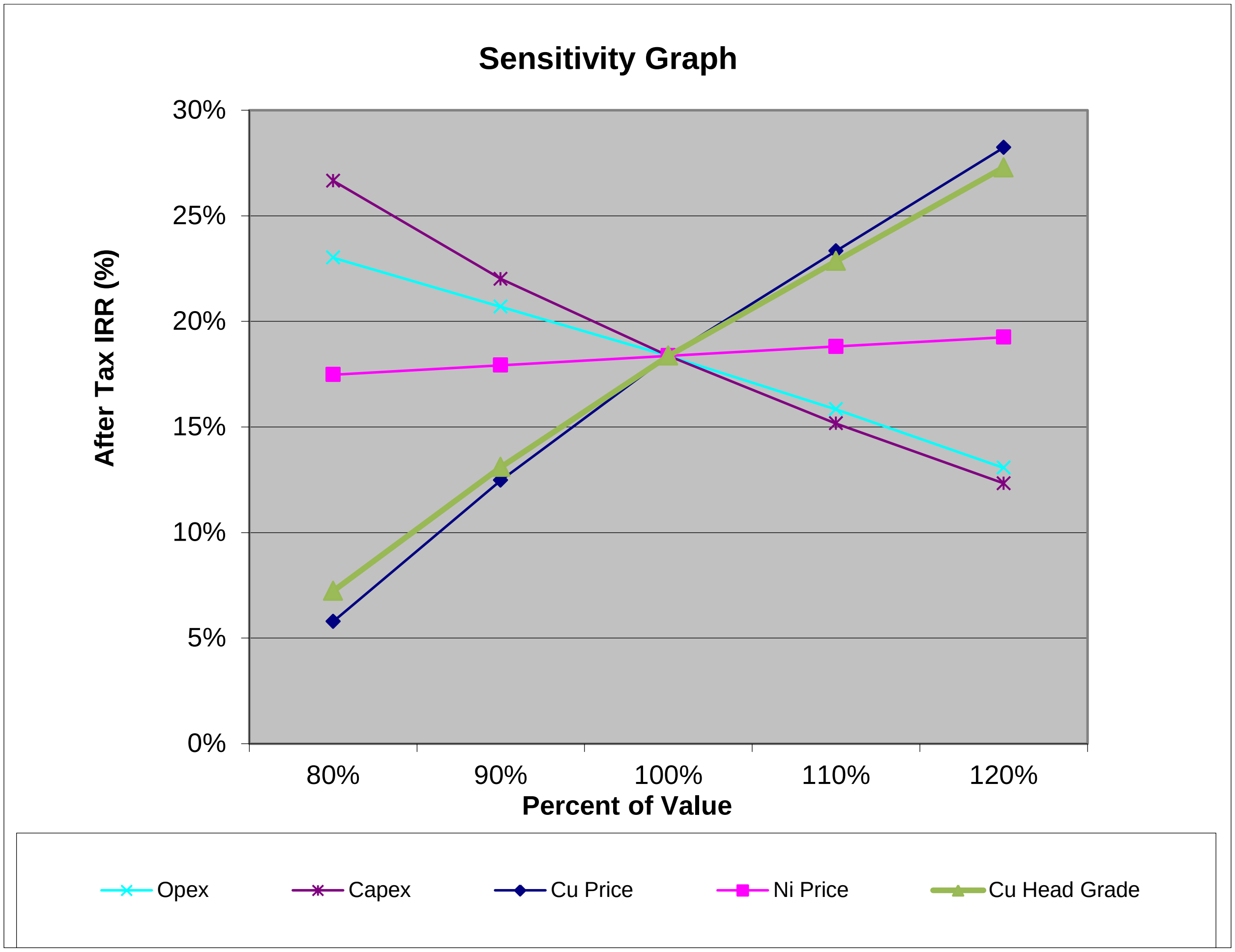


FIGURE 22.2 AFTER-TAX IRR SENSITIVITY GRAPH



The after-tax base case NPV's and IRR's are most sensitive to copper metal price followed by copper head grade, copper recovery in the copper concentrate, OPEX, CAPEX and nickel price.

23.0 ADJACENT PROPERTIES

There are no similar Cu-Ni-PGE properties in the immediate Thierry Mine area that are being actively explored or are under development.

24.0 OTHER RELEVANT DATA AND INFORMATION

Risks and opportunities have been identified for the Project. The anticipated impact on the Project is listed in brackets after each item, using low-medium-high categories.

24.1 RISKS

24.1.1 Mineral Resource Estimate

This PEA is preliminary in nature and 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. The mine design and cost estimating work undertaken in this Thierry PEA is considered to be at conceptual levels of study only. As such, and according to the NI 43-101 Standards of Disclosure for Mineral Projects, it is not possible to declare a Mineral Reserve as of the effective date of this Technical Report. (medium)

24.1.2 Underground Mining

Further geotechnical analysis is required to classify the ground conditions. Poor or hard to manage ground conditions will affect the safety of the underground work places and will increase operating costs, and possibly increase mining dilution, thus lowering grades, and possibly lower LOM tonnes processed. (medium)

Hydrogeology is not well understood. Water re-charge rates are currently unknown. Initial dewatering assumes a minimal re-charge rate, and it may take longer to draw down the water if the re-charge rate is higher. (low)

24.1.3 Process Plant and Tailings

It may be difficult to achieve a 90% copper recovery in the process plant (low), and a 30% copper grade in the copper concentrate. (medium)

It may be difficult to consistently produce a saleable nickel-copper concentrate. (medium)

An additional thickening step for slimes removal may be required to produce paste backfill with desirable rheology. (medium)

The concentrates may experience a moisture content higher than 10%. (low)

There may be a requirement by government permitting agencies to design a new tailings facility as opposed to re-establishing the past lake disposal site. (medium)

Extensive (and costly) Environmental Assessment Studies will be required. (low to medium).

24.1.4 Financial Aspects

Lower metal prices would decrease the Project economics. Sensitivity analysis indicates that a 10% decrease in metal prices would result in an IRR of approximately 10%. (medium)

The after-tax cash flow model does not take into account financing cost and overhead cost for a corporate office. (low)

24.2 OPPORTUNITIES

24.2.1 Mineral Resource Estimate

The Thierry Deposit remains open down dip. There is an opportunity to extend the Mineral Resource with additional drilling. (medium)

It may be possible in the future to increase the process plant capacity and supplement the Thierry underground mine with near surface mineralization from the K1-1 Deposit once a drill program has been conducted at the K1-1 Deposit. (medium)

24.2.2 Underground Mining

The Project includes existing underground mine workings that can be rehabilitated and used for future production. (medium)

24.2.3 Process Plant and Tailings

An opportunity may exist to increase process plant feed grade and effective capacity by the use of mineral sorting technology. Sensing technology, e.g. x-ray transmission (”XRT”), has significantly improved in recent years allowing waste rock to be selectively identified and rejected. Subject to confirmatory testing, crushed mineralized material would be washed and screened to size ranges appropriate for efficient sorting. Assuming a 40% rejection of +20 mm material, and the crushed material being 50% -20 mm, plant feed grades and effective capacity could potentially be increased by 20%. (medium)

An additional opportunity exists to increase process plant grinding capacity by the installation a pebble crusher to crush and recycle screened-out SAG discharge. Primary grinding capacity could potentially be increased by 15% or more. (medium)

It may be possible to increase recoveries of Au and PGM's in the copper concentrates to payable levels. (medium)

There is a possibility to safely use dewatered, previously mined-out open pits for tailings disposal. (low)

24.2.4 Financial Aspects

Currently spot metals prices are trading above the prices used in the financial analysis. (medium)

25.0 INTERPRETATION AND CONCLUSIONS

P&E concludes that the Thierry Project has economic potential as an underground mining and mineralized material processing operation producing a copper and a nickel-copper concentrate. This conclusion would need to be confirmed in a subsequent Pre-Feasibility Study.

P&E notes that this PEA is preliminary in nature, and its Mineral Resources include Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary assessment will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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

- P&E concludes that the Thierry Deposit has economic potential as an underground mining and processing operation producing copper and nickel-copper concentrates.

- The economic analysis contained in this Technical Report is based on Measured, Indicated and Inferred Mineral Resources. The present Technical Report is prepared in accordance with the requirements of National Instrument 43-101 (“NI 43-101”) and is in compliance with Form 43-101F1 of the Canadian Securities Administrators (“CSA”). The Mineral Resource estimates presented herein have been prepared in conformity with Canadian Institute of Mining and Metallurgy and Petroleum (“CIM”) “CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (November 29, 2019). Mineral Resources have been classified in accordance with the “CIM Definition Standards for Mineral Resources and Mineral Reserves” (May 10, 2014).

- It is P&E’s opinion that sample preparation, security and analytical procedures for the Project drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate. Based upon the evaluation of the QA/QC programs undertaken by Cadillac, P&E concludes that the data are of good quality for use in the Mineral Resource Estimate. Based upon P&E’s due diligence sampling and data verification, P&E concludes that the data are of good quality for use in the Mineral Resource Estimate.

- The envisaged 4,000 tpd underground long-hole mining method is estimated to experience mining dilution in the order of 20% at grades of 0.59% Cu, 0.10% Ni and 2.20 g/t Ag, with negligible Au, Pt and Pd. Mining recovery (extraction) is estimated at 90%.

- The initial (pre-production) capital cost of the Project is estimated at $407.0 M and includes a new shaft and hoist, and a process plant. Sustaining capital costs over the LOM are estimated at approximately $303.5 M, for a total capital cost of $710.5 M.

- At metal prices of US$3.48/lb Cu, US$8.00/lb Ni, US$21/oz Ag, US$1,600/oz Au, US$1,100/oz Pt and US$1,250/oz Pd, the Project was evaluated on an after-tax cash flow basis to generate a net undiscounted cash flow estimated at $549.1 M. This results in an after-tax IRR of 18.9% and an after-tax NPV of $240.4 M when using a 6% discount rate. The Project has a payback period estimated at 3.2 years from start of commercial production. The average life-of-mine cash cost is US$1.08/lb copper, net of nickel and by-product credits, at an average operating cost of $58.41/t processed. The average life-of-mine all-in sustaining cost ("AISC") is estimated at US$1.98/lb copper, net of nickel and by-product credits.

- The after-tax NPV's and IRR's are most sensitive to copper metal price followed by copper head grade, copper recovery in the copper concentrate, OPEX, CAPEX and nickel price.

- There is no guarantee that Braveheart will be successful in obtaining any or all of the requisite consents, permits or approvals, regulatory or otherwise for the Thierry Project development or that the Project will be placed into production.

26.0 RECOMMENDATIONS

P&E recommends that Braveheart advance the Thierry Project with: 1) programs to expand and upgrade the Mineral Resources; and 2) extended and advanced technical studies, particularly in metallurgical, geotechnical and environmental matters with the intention to advance the Project to a Pre-Feasibility Study.

P&E recommends 9,000 m of diamond drilling be carried out on the upper portion of the Thierry Deposit from surface, and 150,000 m of drilling be carried out underground on the deep portion of the "Main Zone" to increase the overall Indicated Mineral Resource tonnage in the mine area. The Mineral Resource Estimate should be updated to incorporate the new data. A representative bulk sample for metallurgical testwork should be obtained when underground access is available.

With regard to K1-1 Mineral Resources, P&E recommends the following:

- Assaying of available drill core from previous holes where assaying was incomplete for Cu, Ni, Pt, Pd, Au and Ag.

- Complete 43 in-fill drill holes and twin 13 UMEX drill holes for which assaying was limited to Cu and Ni and was locally discontinuous, and for which drill core for re-assay is unavailable. The objective is to upgrade the Inferred Mineral Resource to Indicated. Proposed drilling totals 11,000 m.

- All assaying of core from future holes should be done for all six metals: Cu, Ni, Pt, Pd, Au and Ag. Collar surveys and down hole azimuth and dip surveys should be completed on all holes and where practical for previously un-surveyed/partially surveyed holes. P&E recommends use of Gyro-based down-hole survey instrumentation which is more reliable for surveys in magnetic rocks such as at K1-1.

- The drill hole database should be converted to SI (metric units).

- Update the Mineral Resource Estimate incorporating new data acquired from the above work.

It is recommended that Braveheart take the following actions to advance the Project to a Pre-Feasibility Study:

- Complete detailed engineering and access the underground via the existing ramp for bulk sampling and to confirm the continuity of the Thierry Deposit and the appropriateness of the long-hole mining method.

- Commence baseline studies to support the environmental permitting process.

- Continue to engage the community and First Nations in the Project development, and communicate the Project's scope, impacts and benefits.

- Carry out additional metallurgical testwork to improve metallurgical recoveries and process optimization.

- Additional metallurgical testwork should be conducted on drill core composite samples representing the first five years of expected production from the Thierry Deposit. In addition to normal head analyses and environmental characterization, this work should include:
  - Comminution testwork, including Bond and JKSimMet tests, and possibly other tests depending on the planned crushing/grinding circuits.
  - Rougher, cleaner and locked cycle testwork. Based on past work, copper flotation is relatively straightforward, but production of a Ni-Cu concentrate may be problematic. Mineralogy may be required to support the metallurgical work.
  - Liquid-solid separation and rheology testwork as appropriate.

P&E recommends the proposed work program and budget presented in Table 26.1. The program is comprised of two phases. The results of Phase 1 would be assessed before commencing, revising or curtailing Phase 2. The cost for both phases combined is estimated at $44.4 M.

TABLE 26.1
RECOMMENDED WORK PROGRAM AND BUDGET

Program	Units	Unit Cost ($)	Budget ($M)
Phase 1			
Thierry Deposit			
Mine dewatering & rehabilitation			6.00
Surface drilling at Thierry	9,000 m	289	2.60
Underground drilling at Thierry	150,000 m	165	24.75
Underground development (3 m x 3 m) for drilling	1,200 m	3,000	3.60
Underground bulk sampling			0.10
Mineral Resource Estimate update			0.80
Subtotal			**37.85**
K1-1 Deposit			
Fill-in & twin drilling	11,000 m	289	3.10
Mineral Resource Estimate update			0.08
Subtotal			**3.18**
Phase 1 Total			**41.03**
Phase 2			
Metallurgical testwork			0.25
Geological & mineralogical studies			0.05
Environmental study work			0.25

Table 26.1 Recommended Work Program and Budget			
Program	**Units**	**Unit Cost ($)**	**Budget ($M)**
Hydrogeology study			0.08
Archaeological study			0.05
Advance exploration closure report			0.01
Geotechnical and condemnation drilling	3,000 m	250	0.75
Housing and accommodation	2,700 days	150	0.40
First Nation consultation			0.04
Pre-feasibility study			1.50
Phase 2 Total			**3.38**
Total			**44.41**

Note: *Subject to HST*

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28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

EUGENE PURITCH, P. ENG., FEC, CET

I, Eugene J. Puritch, P. Eng., FEC, CET, residing at 44 Turtlecreek Blvd., Brampton, Ontario, Canada, 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 "Preliminary Economic Assessment of the Thierry Cu-Ni-PGE Deposit, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada", (The "Technical Report") with an effective date of January 21, 2021.

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 a Bachelor's degree in Engineering Equivalency. I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National 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 on December 15, 2005, May 5, 2010 and on June 2, 2011.

5. I am responsible for authoring Sections 12, 14 and co-authoring Sections 1, 25 and 26 of this Technical Report.

6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.

7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for several Technical Reports, the most recent one titled "Technical Report and Preliminary Economic Assessment of the Thierry and K1-1 Cu-Ni-PGE Deposits, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada" with an effective date of May 15, 2012.

8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: January 21, 2021

Signed Date: February 22, 2021

{SIGNED AND SEALED}

[Eugene Puritch]

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Eugene Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON

ANDREW BRADFIELD, P. ENG.

I, Andrew Bradfield, P. Eng., residing at 5 Patrick Drive, Erin, Ontario, Canada, N0B 1T0, do hereby certify that:

1. I am an independent mining engineer contracted by P&E Mining Consultants.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Thierry Cu-Ni-PGE Deposit, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada", (The "Technical Report") with an effective date of January 21, 2021.
3. I am a graduate of Queen's University, with an honours B.Sc. degree in Mining Engineering in 1982. I am a Professional Engineer of Ontario (License No.4894507). I am also a member of the National CIM.

   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 1982. My summarized career experience is as follows:
   - Various Engineering Positions – Palabora Mining Company, 1982-1986
   - Mines Project Engineer – Falconbridge Limited, 1986-1987
   - Senior Mining Engineer – William Hill Mining Consultants Limited, 1987-1990
   - Independent Mining Engineer, 1990-1991
   - GM Toronto – Bharti Engineering Associates Inc, 1991-1996
   - VP Technical Services, GM of Australian Operations – William Resources Inc, 1996-1999
   - Independent Mining Engineer, 1999-2001
   - Principal Mining Engineer – SRK Consulting, 2001-2003
   - COO – China Diamond Corp, 2003-2006
   - VP Operations – TVI Pacific Inc, 2006-2008
   - COO – Avion Gold Corporation, 2008-2012
   - Independent Mining Engineer, 2012-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 2, 3, 15, 19 and 24, and co-authoring Sections 1, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
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. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: January 21, 2021
Signing Date: February 22, 2021

{SIGNED AND SEALED}
[Andrew Bradfield]

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Andrew Bradfield, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

DAVID BURGA, P.GEO.

I, David Burga, P. Geo., residing at 3884 Freeman Terrace, Mississauga, Ontario, Canada, L5M 6P6 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 "Preliminary Economic Assessment of the Thierry Cu-Ni-PGE Deposit, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada", (The "Technical Report") with an effective date of January 21, 2021.
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 over 20 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 visited the Property that is the subject of this Technical Report on May 5, 2010.
5. I am responsible for authoring Sections 4-11, 23 and co-authoring Sections 1, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for several Technical Reports, the most recent one titled "Technical Report and Preliminary Economic Assessment of the Thierry and K1-1 Cu-Ni-PGE Deposits, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada" with an effective date of May 15, 2012.
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 effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: January 21, 2021
Signed Date: February 22, 2021

{SIGNED AND SEALED}
[David Burga]

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David Burga, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

D. GRANT FEASBY, P. ENG.

I, D. Grant Feasby, P. Eng., residing at 12,209 Hwy 38, Tichborne, Ontario, Canada, K0H 2V0, do hereby certify that:

1. I am currently the Owner and President of:
   Feasby Environmental Advantage Services
   38 Gwynne Ave, Ottawa, Ontario, Canada, K1Y1W9
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Thierry Cu-Ni-PGE Deposit, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada", (The "Technical Report") with an effective date of January 21, 2021.
3. I graduated from Queens University in Kingston Ontario, in 1964 with a Bachelor of Applied Science in Metallurgical Engineering, and a Master of Applied Science in Metallurgical Engineering in 1966. I am a Professional Engineer registered with Professional Engineers Ontario. I have worked as a metallurgical engineer for over 50 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.

   My relevant experience for the purpose of the Technical Report has been acquired by the following activities:
   - Metallurgist, Base Metal Processing Plant.
   - Research Engineer and Lab Manager, Industrial Minerals Laboratories in USA and Canada.
   - Research Engineer, Metallurgist and Plant Manager in the Canadian Uranium Industry.
   - Manager of Canadian National Programs on Uranium and Acid Generating Mine Tailings.
   - Director, Environment, Canadian Mineral Research Laboratory.
   - Senior Technical Manager, for large gold and bauxite mining operations in South America.
   - Expert Independent Consultant associated with several companies, including P&E Mining Consultants, on mineral processing, environmental management, and mineral-based radiation assessment.
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 13, 17 and 20 and co-authoring Sections 1, 21, 25 and 26 of this 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 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: January 21, 2021
Signing Date: February 22, 2021
{SIGNED AND SEALED}
[D. Grant Feasby]

____
D. Grant Feasby, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

JAMES PEARSON, P. ENG.

I, James Pearson, P.Eng., residing at 105 Stornwood Court, Brampton, Ontario. Canada, L6W 4H6, do hereby certify that:

1. I am a Mining Engineering Consultant, contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Thierry Cu-Ni-PGE Deposit, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada", (The "Technical Report") with an effective date of January 21, 2021.
3. I am a graduate of Queen's University, Kingston, Ontario, Canada, in 1973 with an honours 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 47 years since my graduation.

   I have read the definition of "Qualified Person" set out in National Instrument ("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 has been acquired by the following activities:

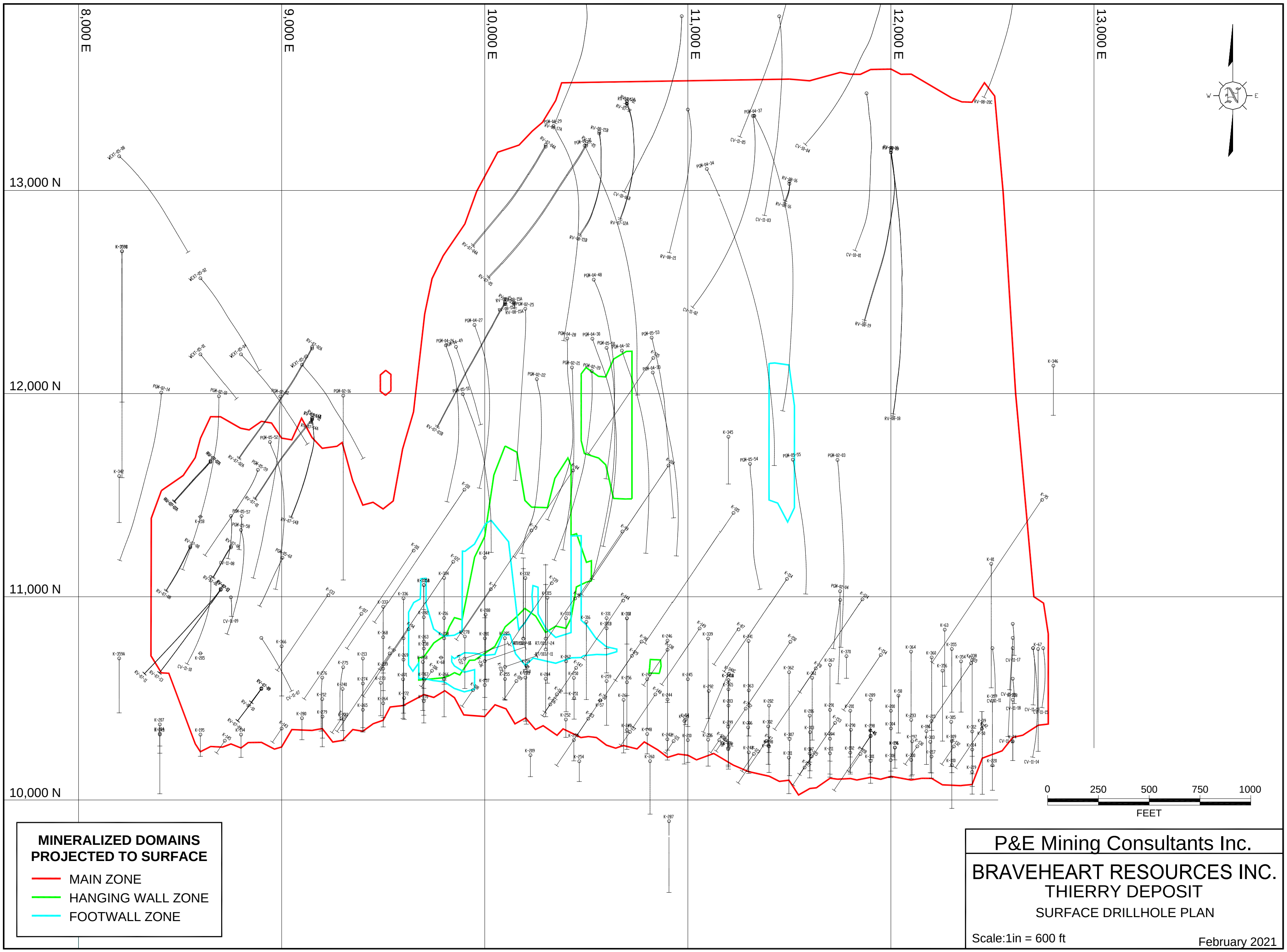
   - Review and report as a consultant on numerous exploration and mining projects around the world for due diligence and regulatory requirements;
   - Project Manager and Superintendent of Engineering and Projects at several underground operations in South America;
   - Senior Mining Engineer with a large Canadian mining company responsible for development of engineering concepts, mine design and maintenance;
   - Mining analyst at several Canadian brokerage firms.
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 16, 18, 22 and co-authoring Sections 1, 21, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Technical Report and Preliminary Economic Assessment of the Thierry and K1-1 Cu-Ni-PGE Deposits, Thierry Project, Pickle Lake Area, Patricia Mining District, North-Western Ontario, Canada" with an effective date of May 15, 2012.
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 effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: January 21, 2021
Signed Date: February 22, 2021

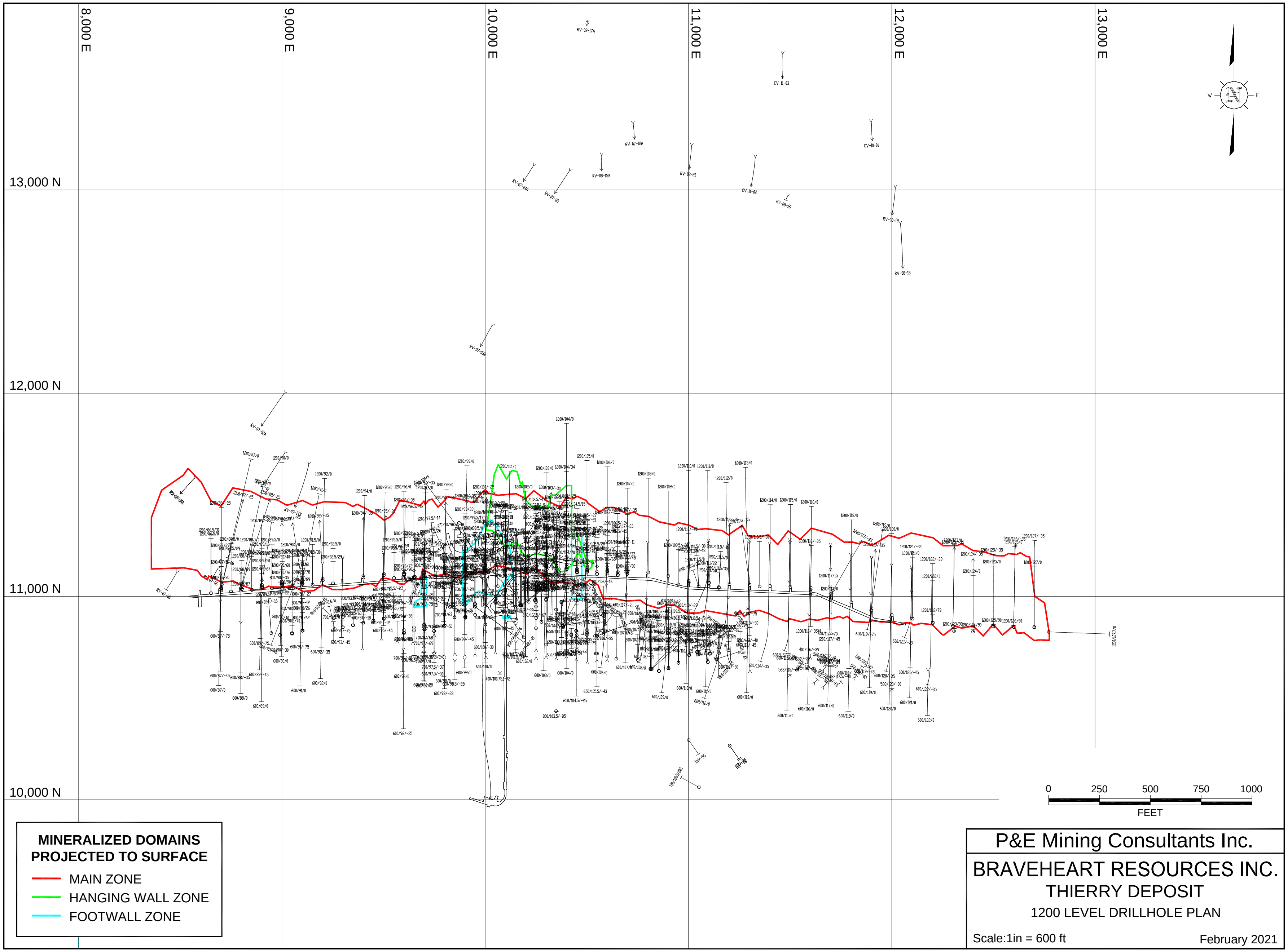
{SIGNED AND SEALED}
[James Pearson]

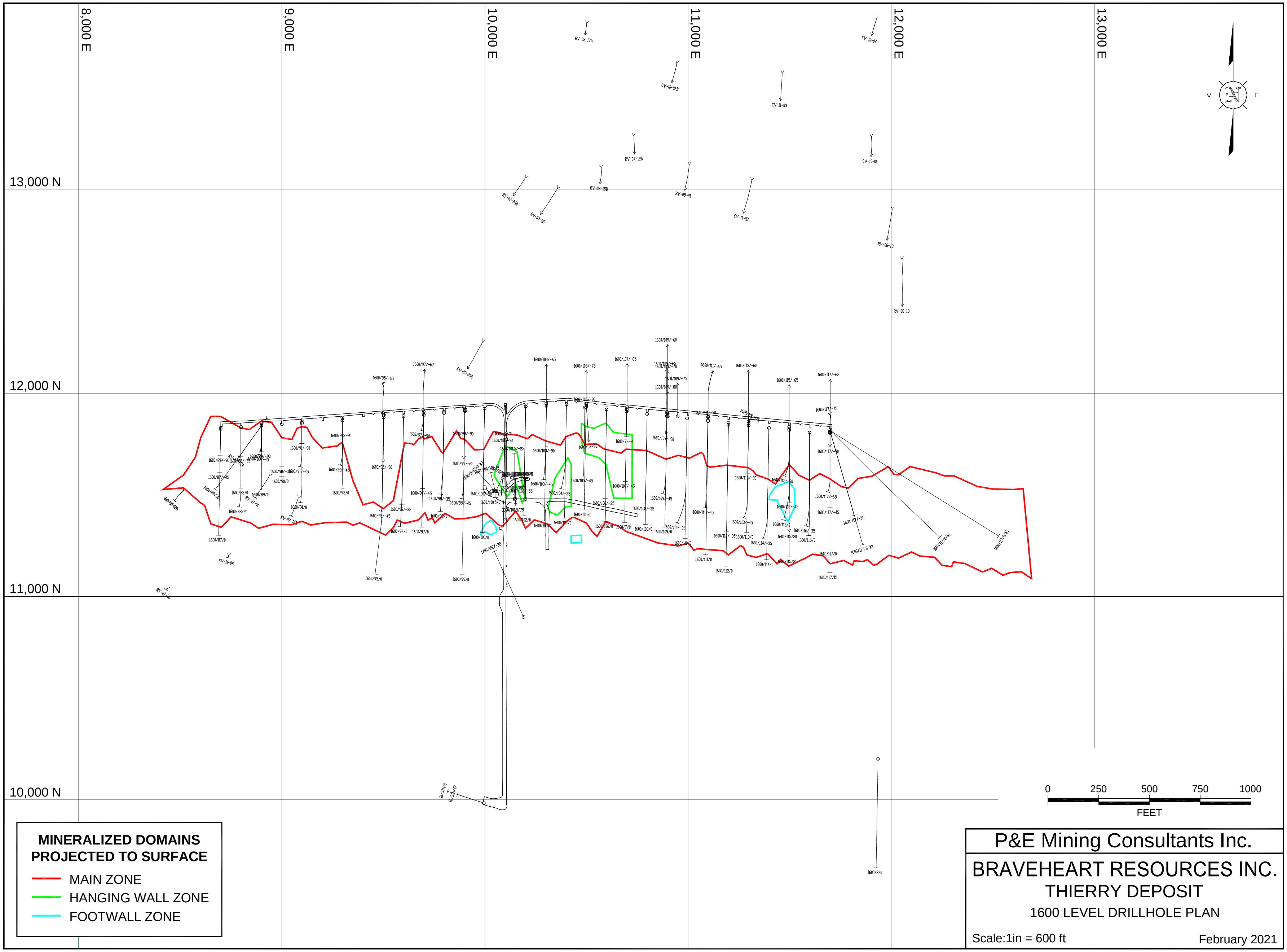
James Pearson, P.Eng.

APPENDIX A SURFACE DRILL HOLE PLANS

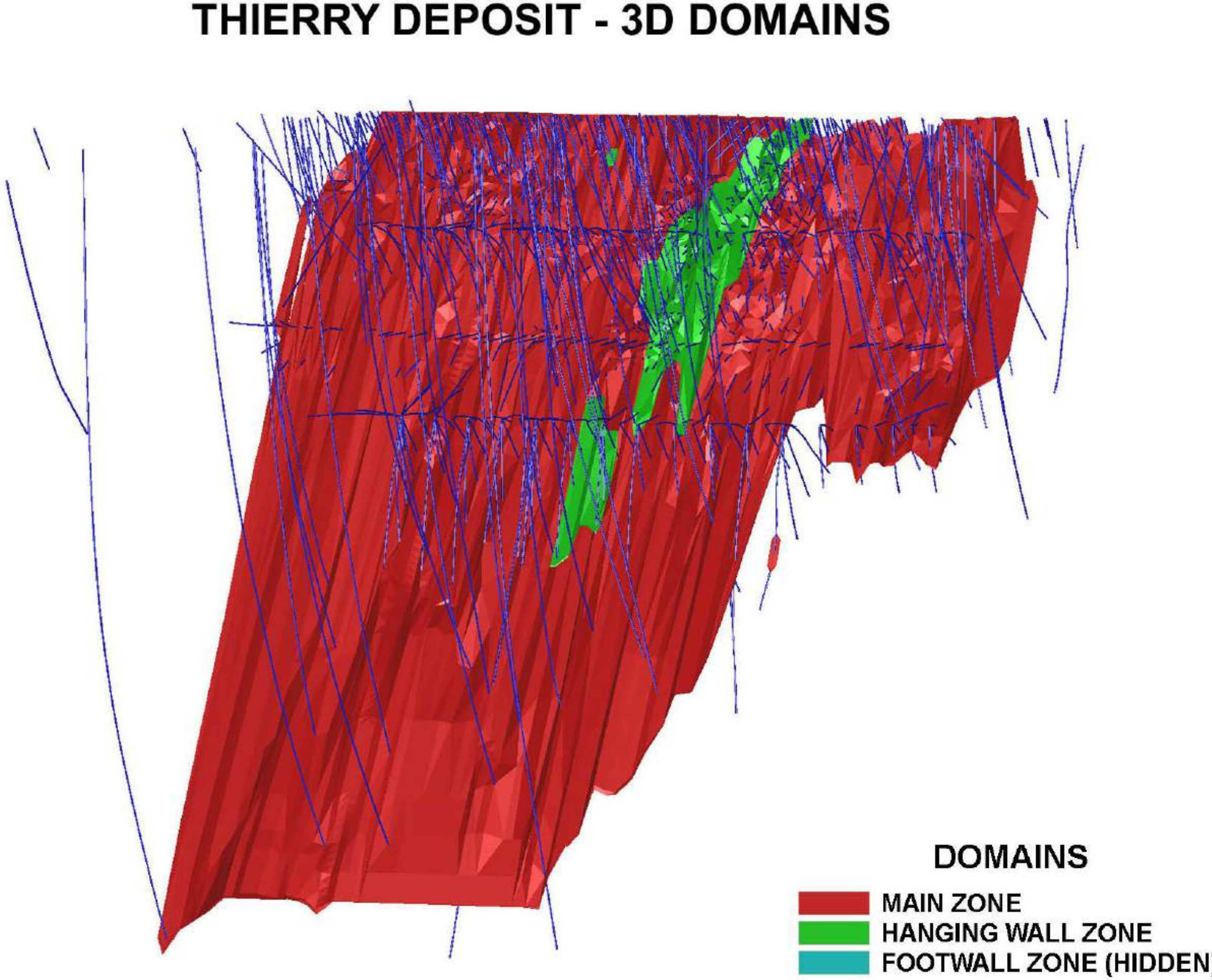


8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO SURFACE
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
SURFACE DRILLHOLE PLAN
Scale:1in = 600 ft
February 2021

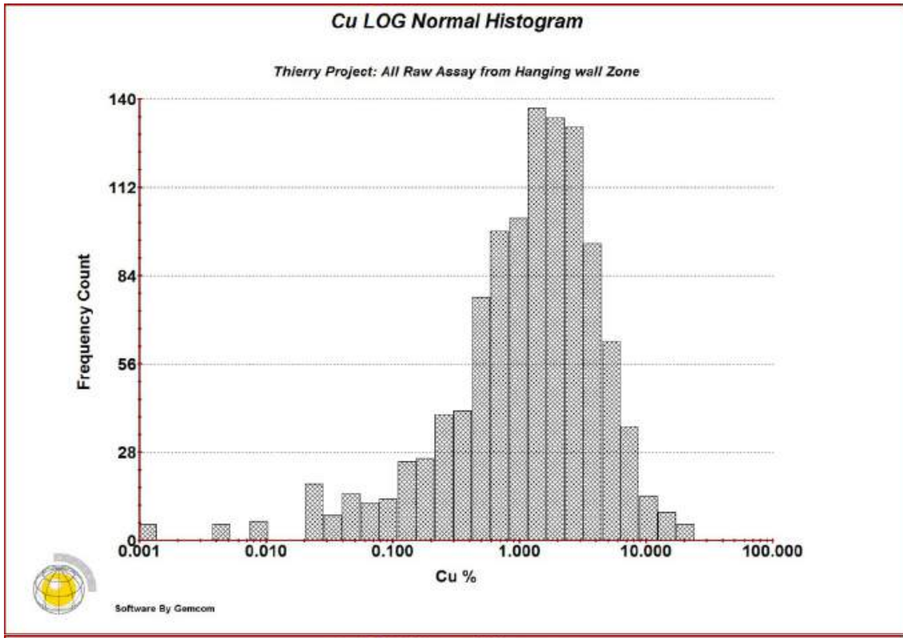

8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO SURFACE
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
1200 LEVEL DRILLHOLE PLAN
Scale:1in = 600 ft
February 2021


8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO SURFACE
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
1600 LEVEL DRILLHOLE PLAN
Scale:1in = 600 ft
February 2021

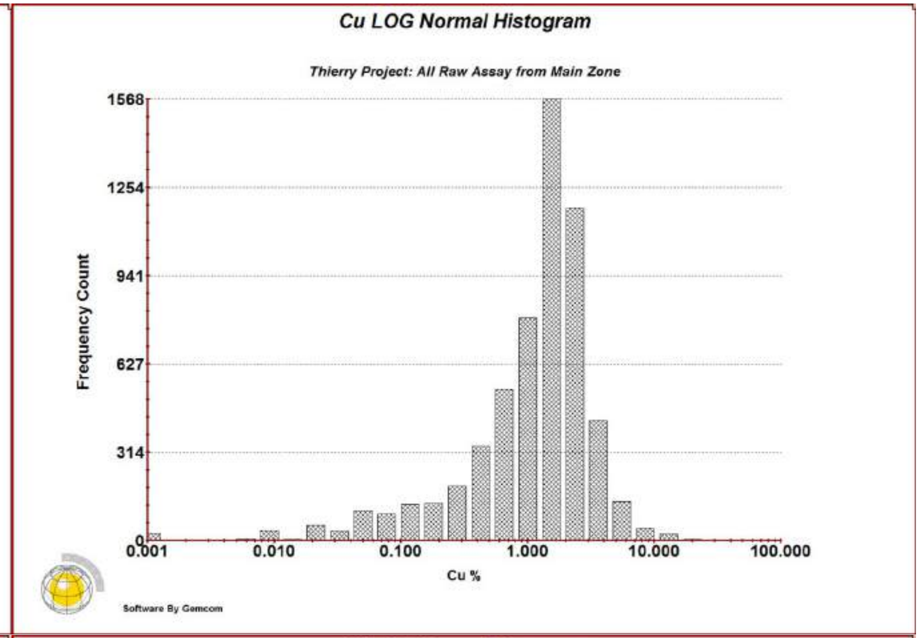
APPENDIX B 3-D DOMAINS


THIERRY DEPOSIT - 3D DOMAINS
DOMAINS
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE (HIDDEN)

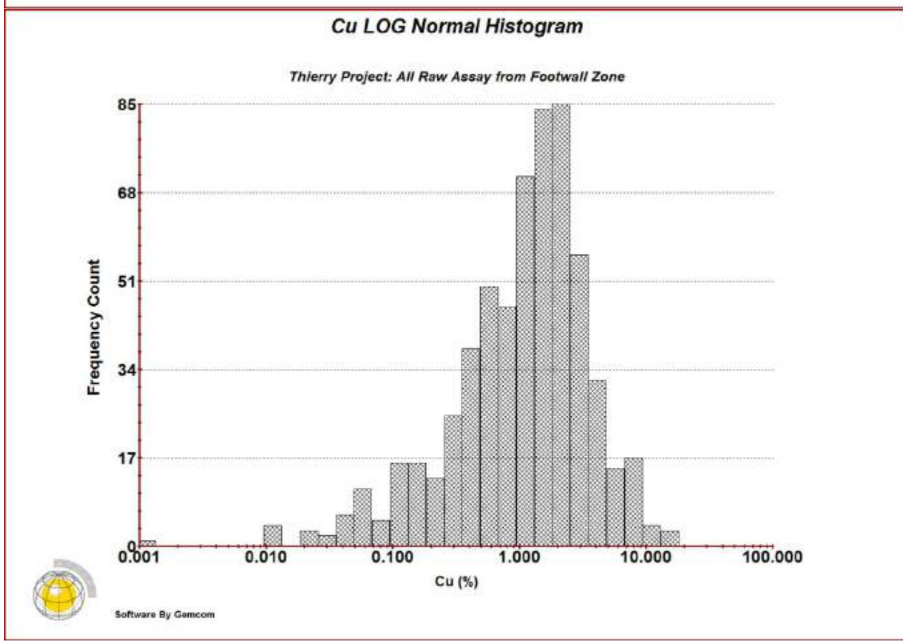
APPENDIX C LOG NORMAL HISTOGRAMS



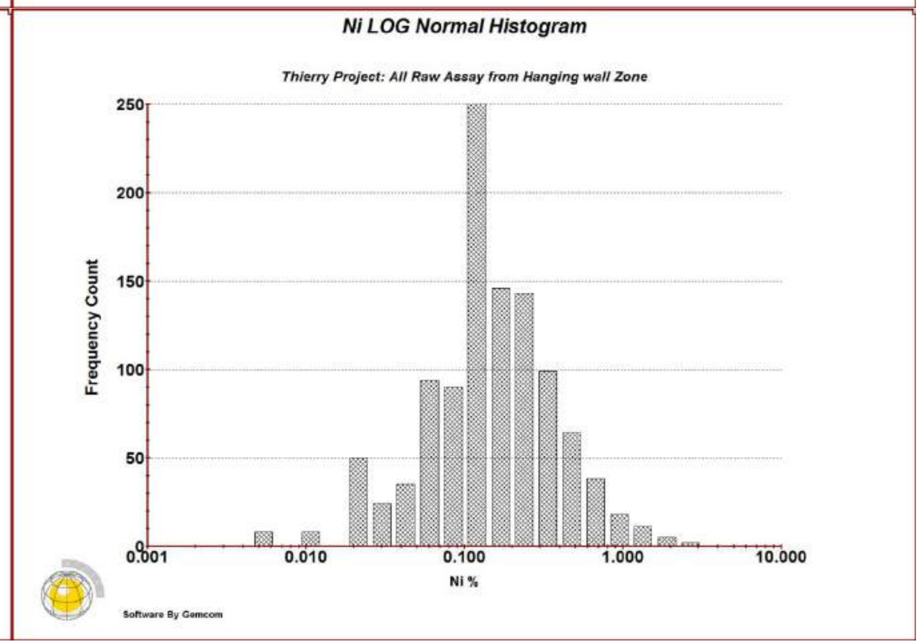
Cu LOG Normal Histogram
Thierry Project: All Raw Assay from Hanging wall Zone
Frequency Count
Cu %
Software By Gemcom



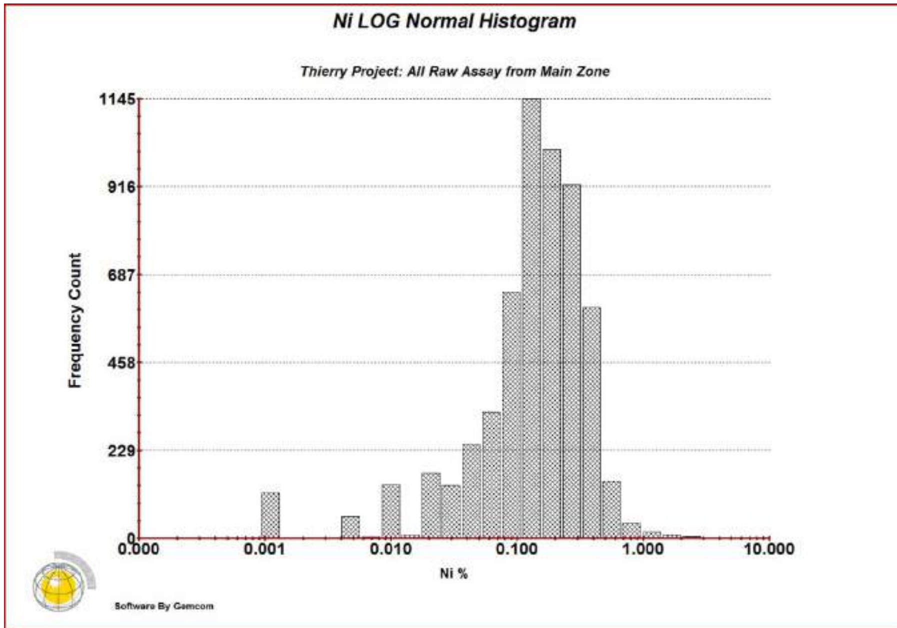
Cu LOG Normal Histogram
Thierry Project: All Raw Assay from Main Zone
Frequency Count
Cu %
Software By Gemcom

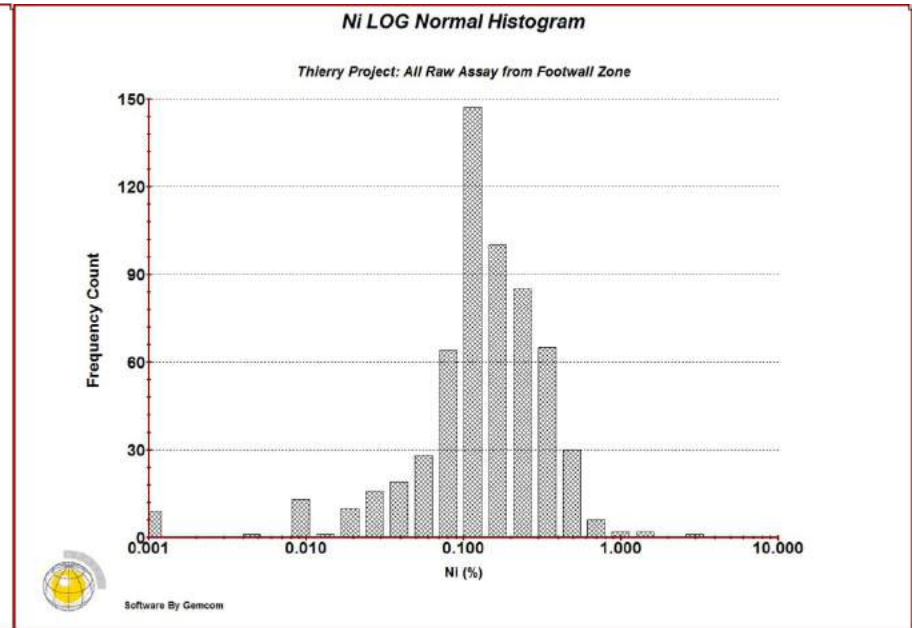


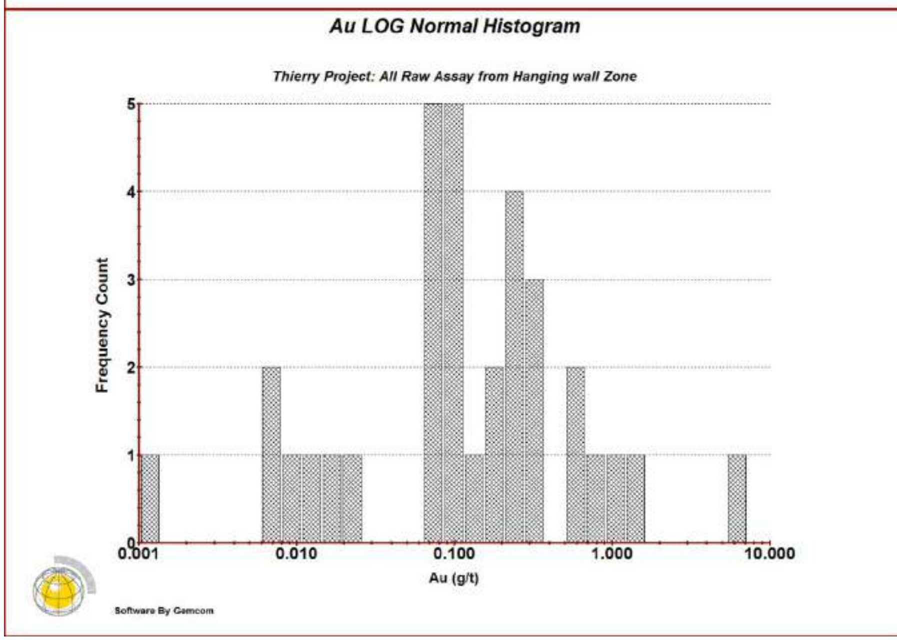
Cu LOG Normal Histogram
Thierry Project: All Raw Assay from Footwall Zone
Frequency Count
Cu (%)
Software By Gemcom

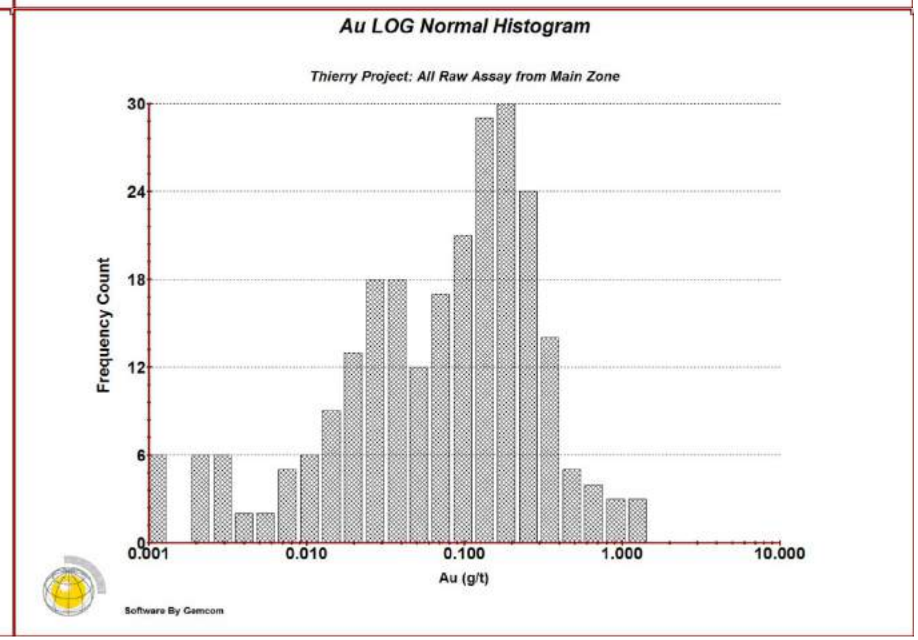


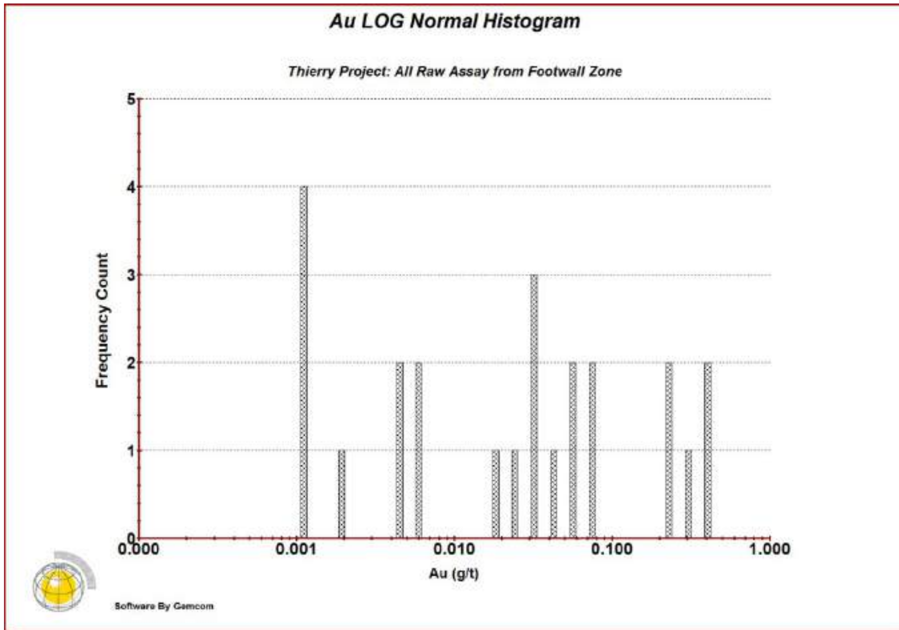
Ni LOG Normal Histogram
Thierry Project: All Raw Assay from Hanging wall Zone
Frequency Count
Ni %
Software By Gemcom

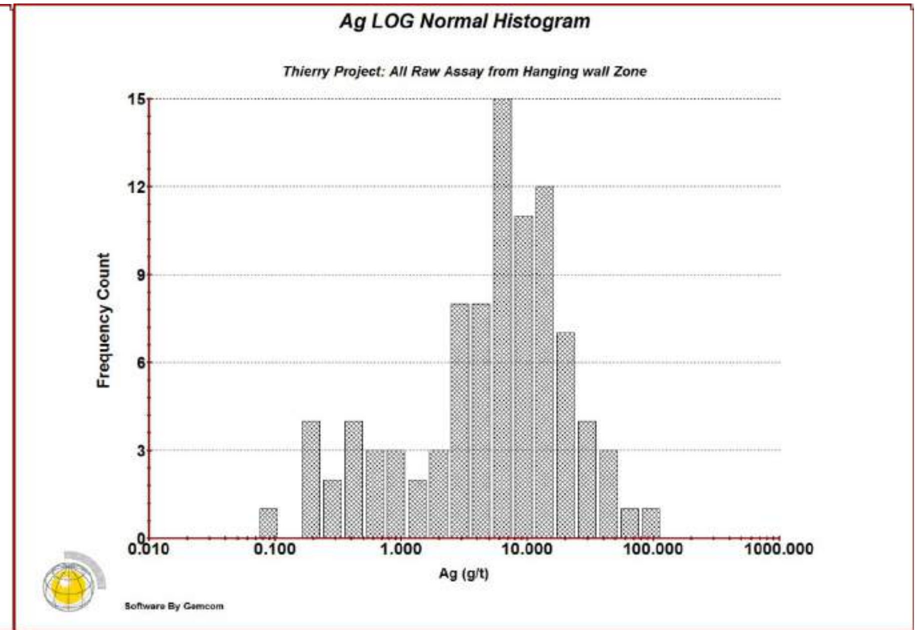

Ni LOG Normal Histogram
Thierry Project: All Raw Assay from Main Zone
Frequency Count
Ni %
Software By Gemcom

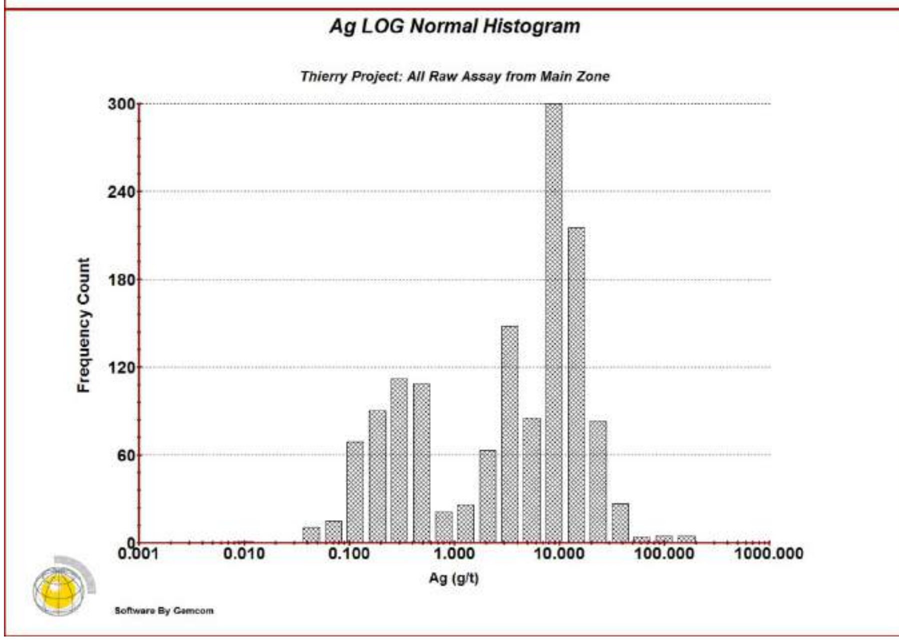

Ni LOG Normal Histogram
Thierry Project: All Raw Assay from Footwall Zone
Frequency Count
Ni (%)
Software By Gemcom

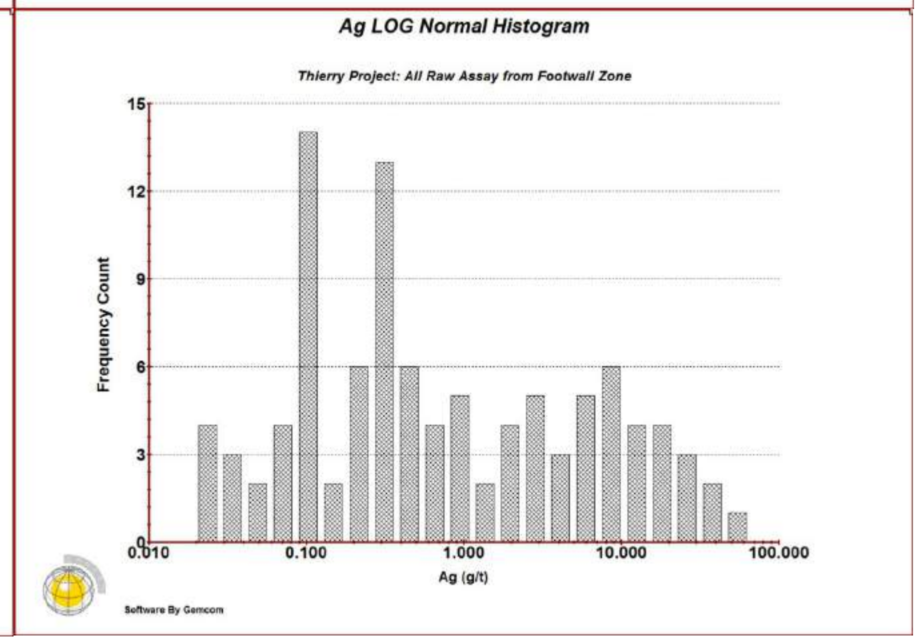

Au LOG Normal Histogram
Thierry Project: All Raw Assay from Hanging wall Zone
Frequency Count
Au (g/t)
Software By Gemcom

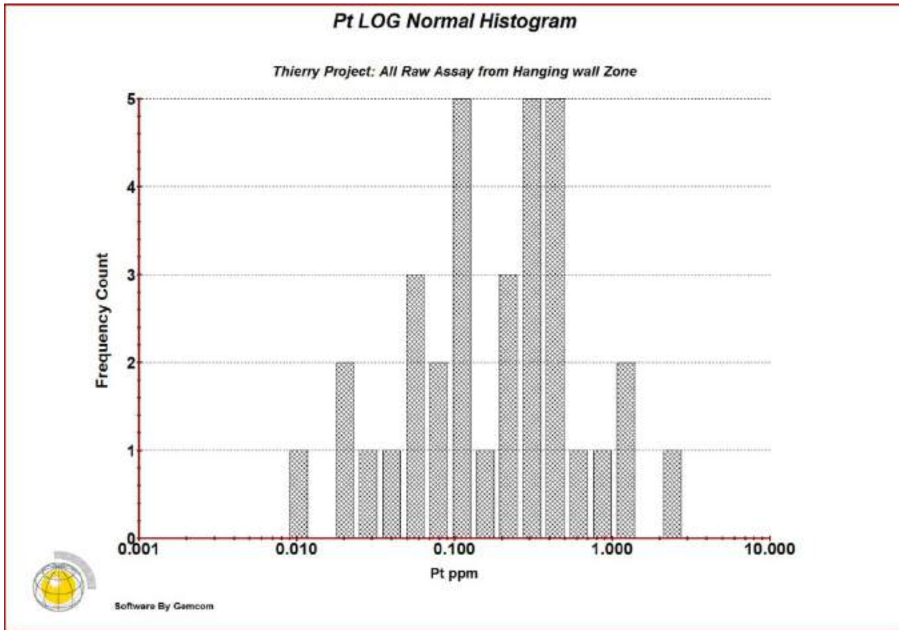

Au LOG Normal Histogram
Thierry Project: All Raw Assay from Main Zone
Frequency Count
Au (g/t)
Software By Gemcom

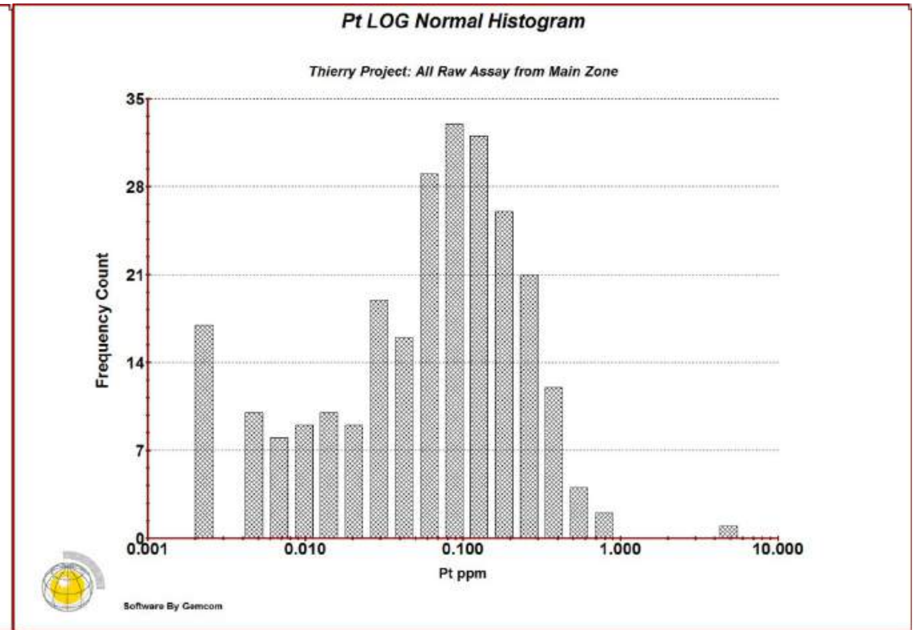

Au LOG Normal Histogram
Thierry Project: All Raw Assay from Footwall Zone
Frequency Count
Au (g/t)
Software By Gemcom

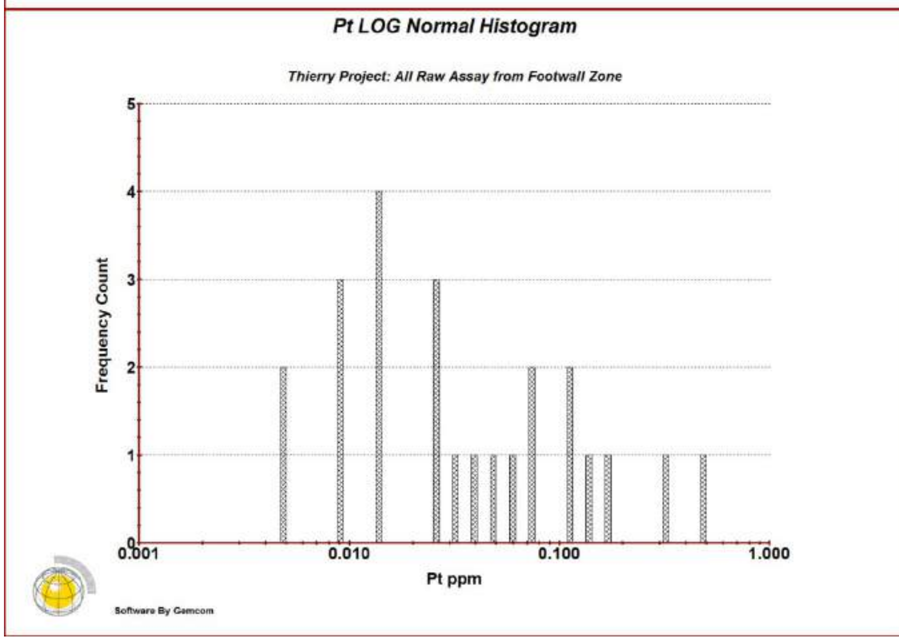

Ag LOG Normal Histogram
Thierry Project: All Raw Assay from Hanging wall Zone
Frequency Count
Ag (g/t)
Software By Gemcom

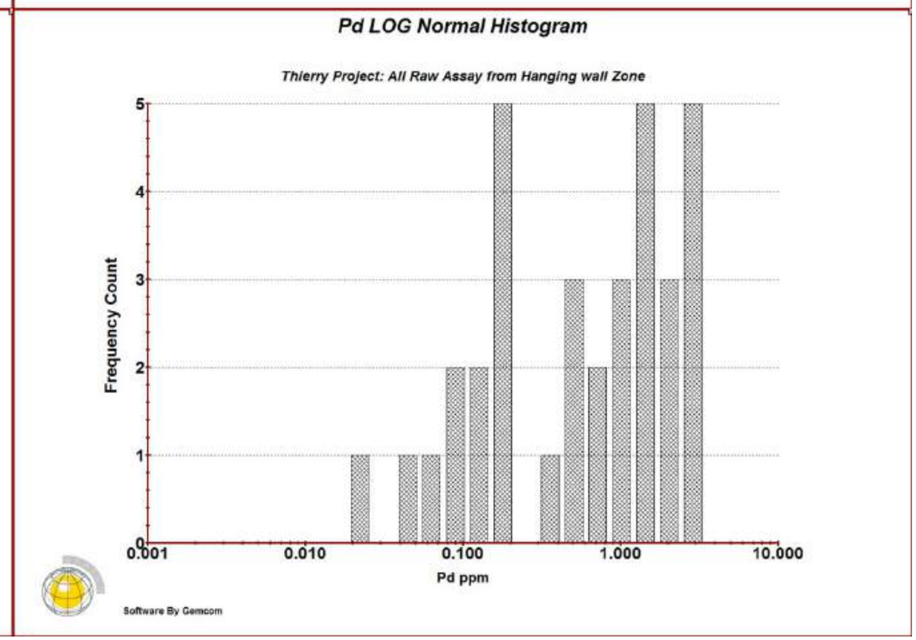

Ag LOG Normal Histogram
Thierry Project: All Raw Assay from Main Zone
Frequency Count
Ag (g/t)
Software By Gemcom

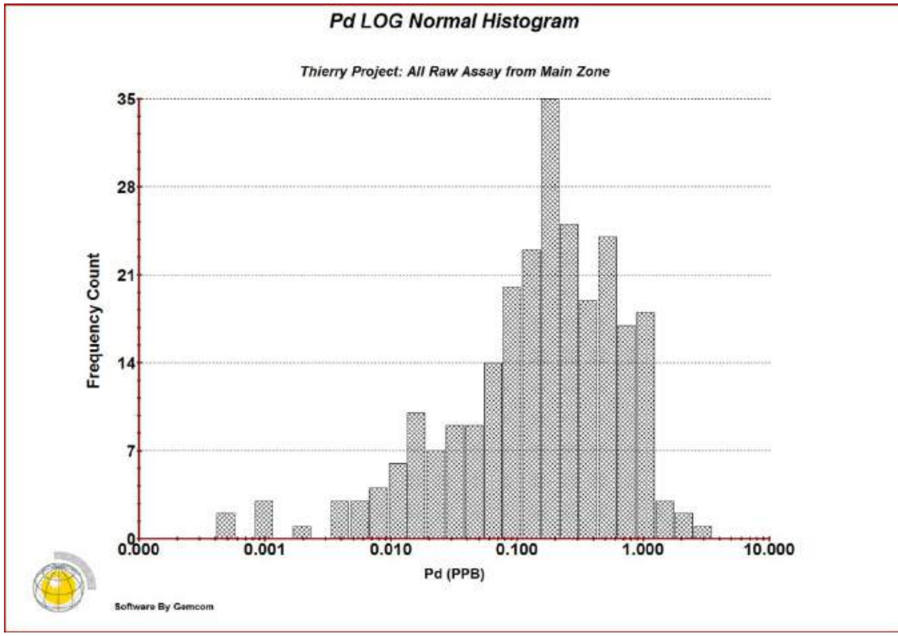

Ag LOG Normal Histogram
Thierry Project: All Raw Assay from Footwall Zone
Frequency Count
Ag (g/t)
Software By Gemcom

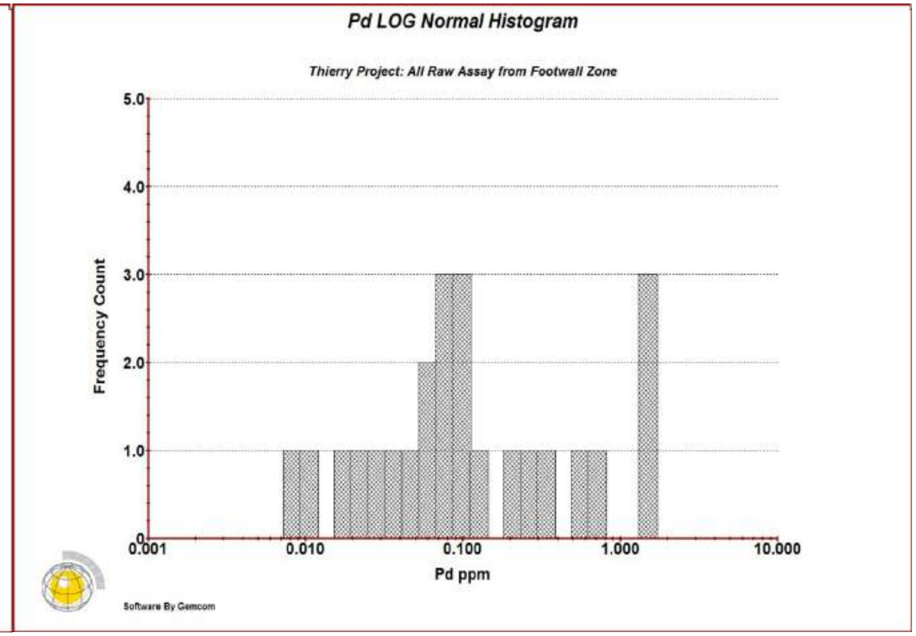

Pt LOG Normal Histogram
Thierry Project: All Raw Assay from Hanging wall Zone
Frequency Count
Pt ppm
0.001
0.010
0.100
1.000
10.000
Software By Gemcom


Pt LOG Normal Histogram
Thierry Project: All Raw Assay from Main Zone
Frequency Count
Pt ppm
0.001
0.010
0.100
1.000
10.000
Software By Gemcom

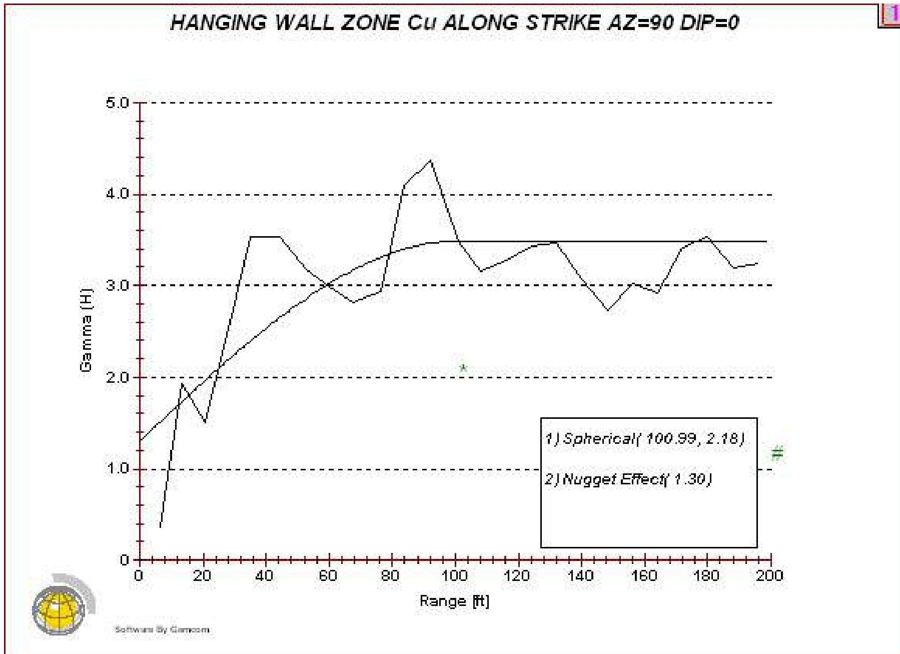

Pt LOG Normal Histogram
Thierry Project: All Raw Assay from Footwall Zone
Frequency Count
Pt ppm
0.001
0.010
0.100
1.000
Software By Gemcom

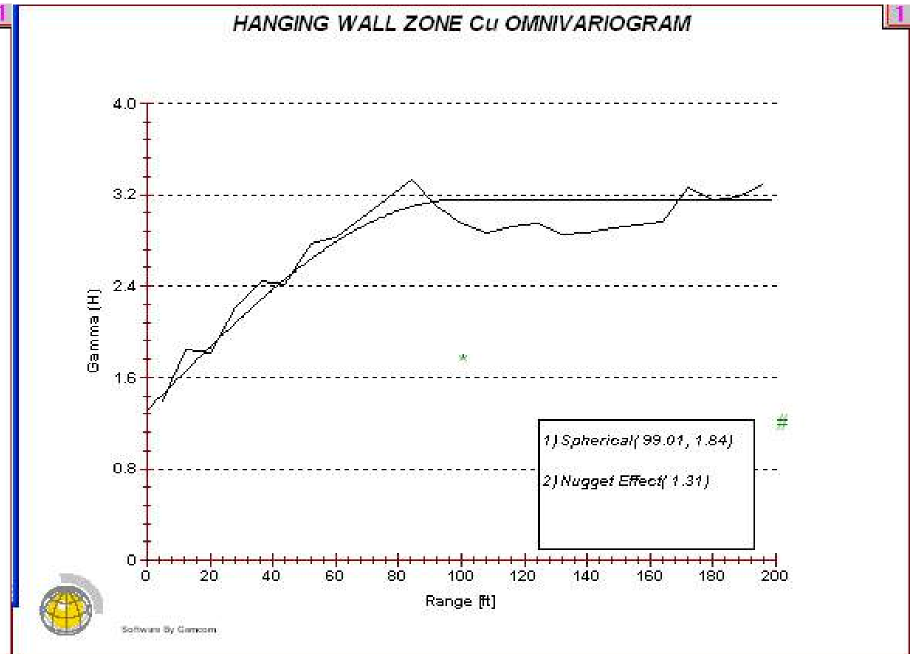

Pd LOG Normal Histogram
Thierry Project: All Raw Assay from Hanging wall Zone
Frequency Count
Pd ppm
0.001
0.010
0.100
1.000
10.000
Software By Gemcom

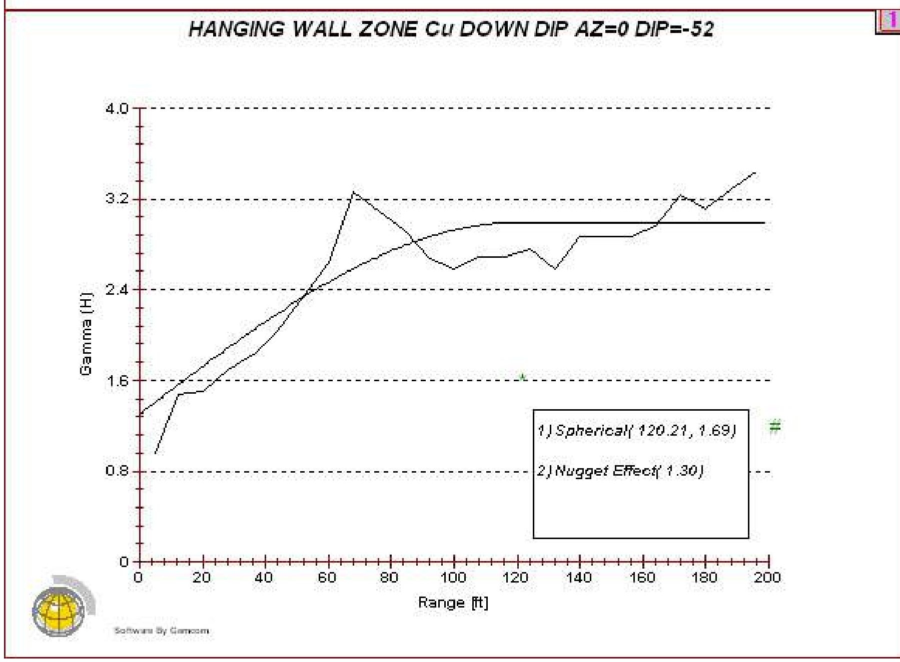

Pd LOG Normal Histogram
Thierry Project: All Raw Assay from Main Zone
Frequency Count
35
28
21
14
7
0
0.000
0.001
0.010
0.100
1.000
10.000
Pd (PPB)
Software By Gemcom

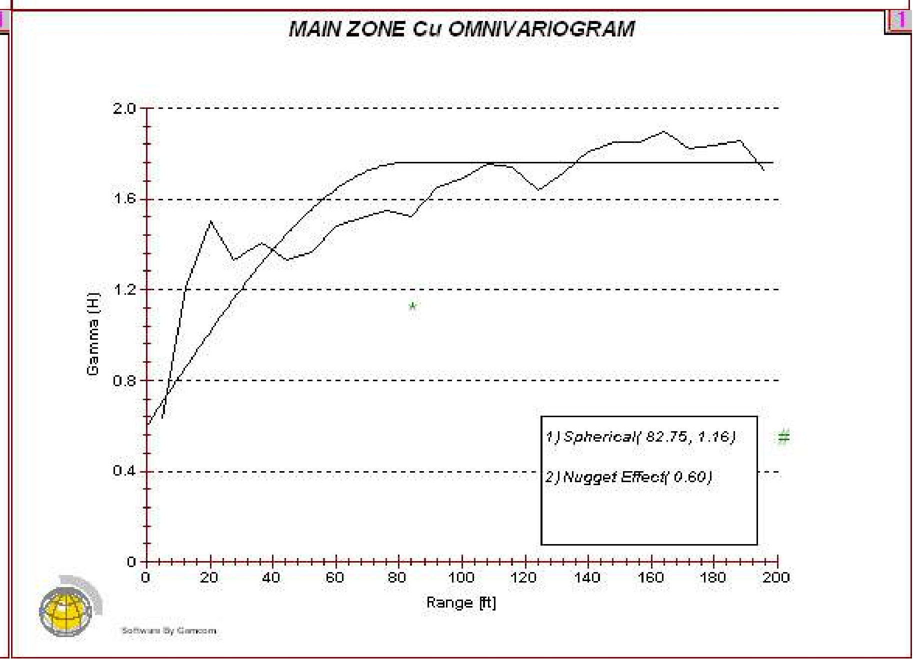

Pd LOG Normal Histogram
Thierry Project: All Raw Assay from Footwall Zone
Frequency Count
5.0
4.0
3.0
2.0
1.0
0
0.001
0.010
0.100
1.000
10.000
Pd ppm
Software By Gemcom

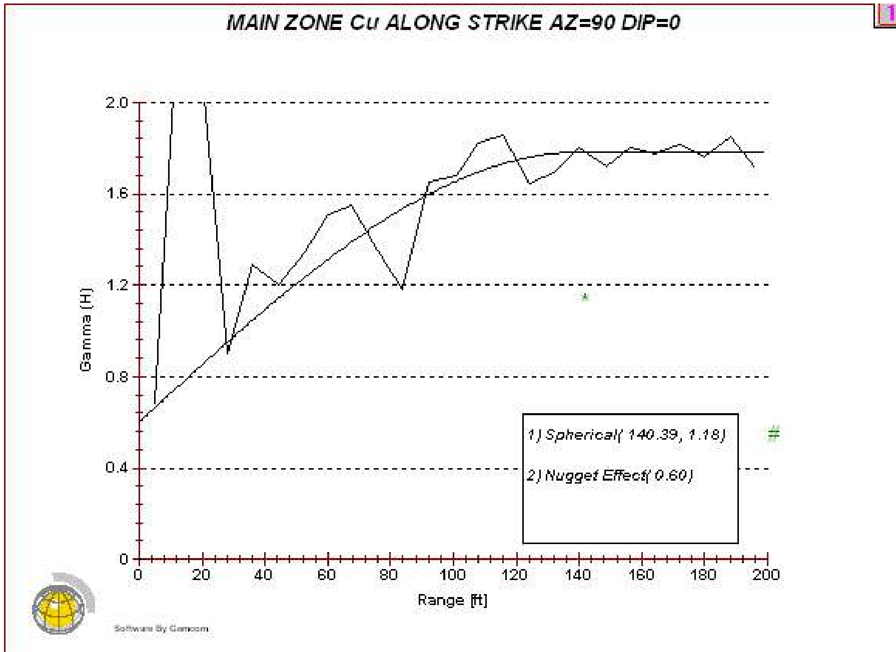
APPENDIX D VARIOGRAMS

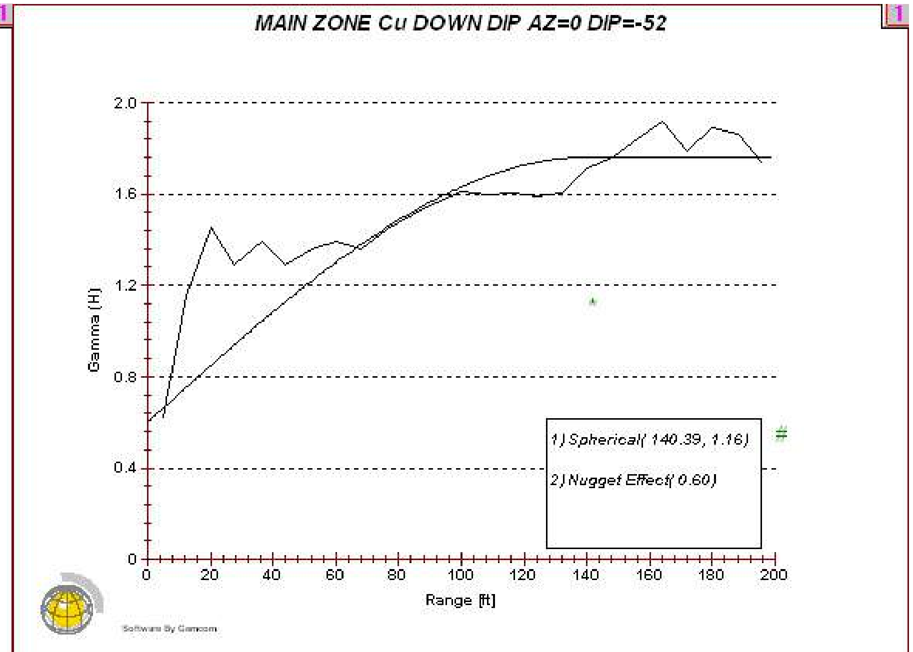

HANGING WALL ZONE Cu ALONG STRIKE AZ=90 DIP=0
Gamma (H)
Range [ft]
1) Spherical(100.99, 2.18)
2) Nugget Effect(1.30)
Software By Gemcom

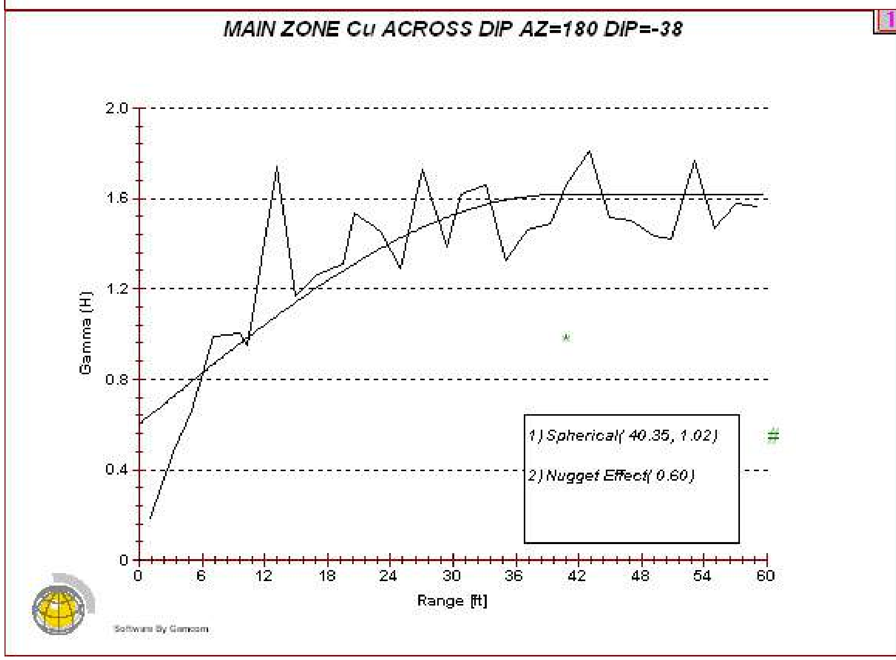

HANGING WALL ZONE Cu OMNIVARIOGRAM
Gamma (H)
Range [ft]
1) Spherical(99.01, 1.84)
2) Nugget Effect(1.31)
Software By Gemcom

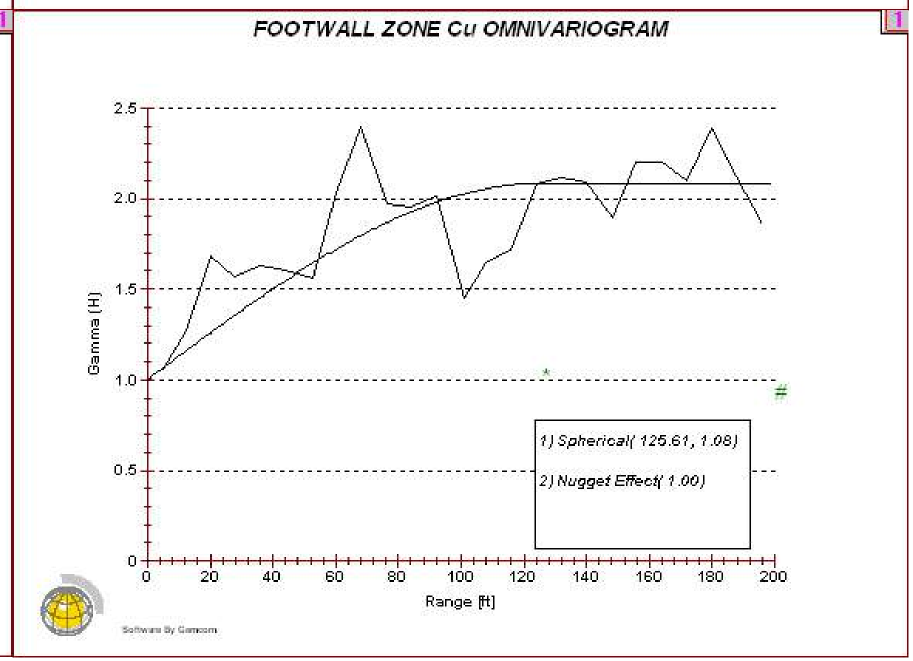

HANGING WALL ZONE Cu DOWN DIP AZ=0 DIP=-52
Gamma (H)
Range [ft]
1) Spherical(120.21, 1.69)
2) Nugget Effect(1.30)
Software By Gemcom

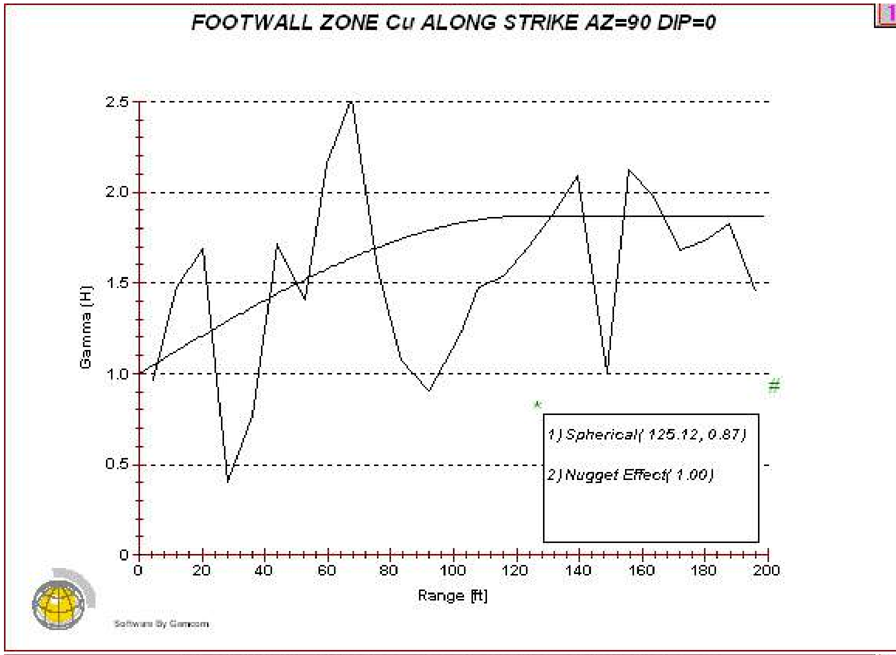

MAIN ZONE Cu OMNIVARIOGRAM
Gamma (H)
Range [ft]
1) Spherical(82.75, 1.16)
2) Nugget Effect(0.60)
Software By Gemcom

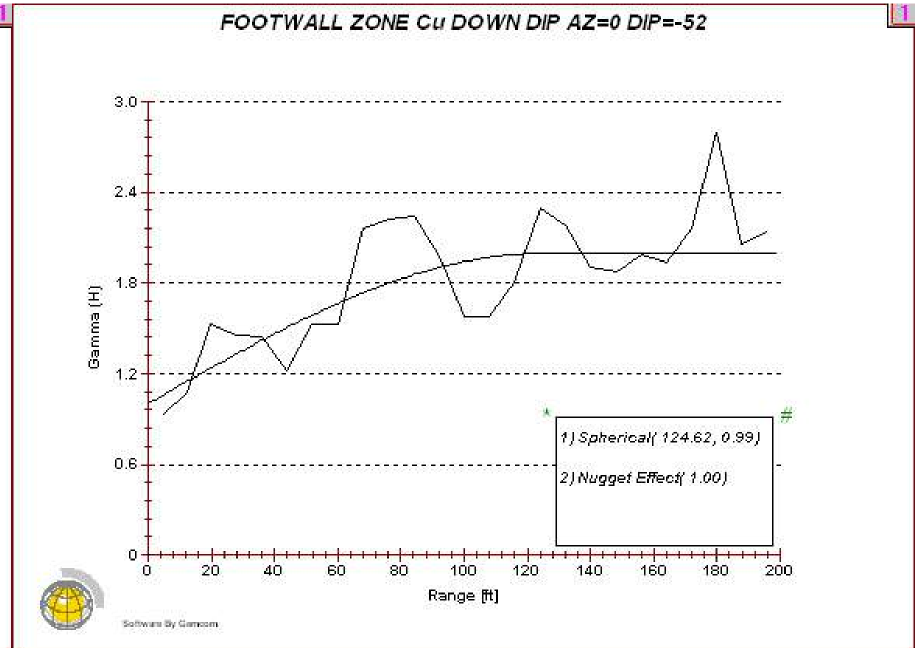

MAIN ZONE Cu ALONG STRIKE AZ=90 DIP=0
Gamma (H)
Range [ft]
1) Spherical(140.39, 1.18)
2) Nugget Effect(0.60)
Software By Gemcom

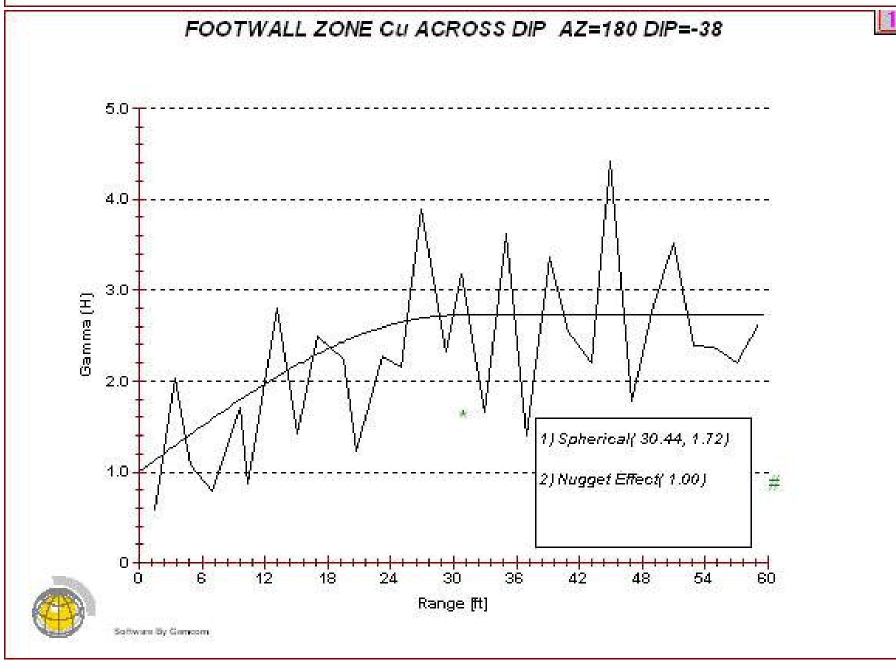

MAIN ZONE Cu DOWN DIP AZ=0 DIP=-52
Gamma (H)
Range [ft]
1) Spherical(140.39, 1.16)
2) Nugget Effect(0.60)
Software By Gemcom

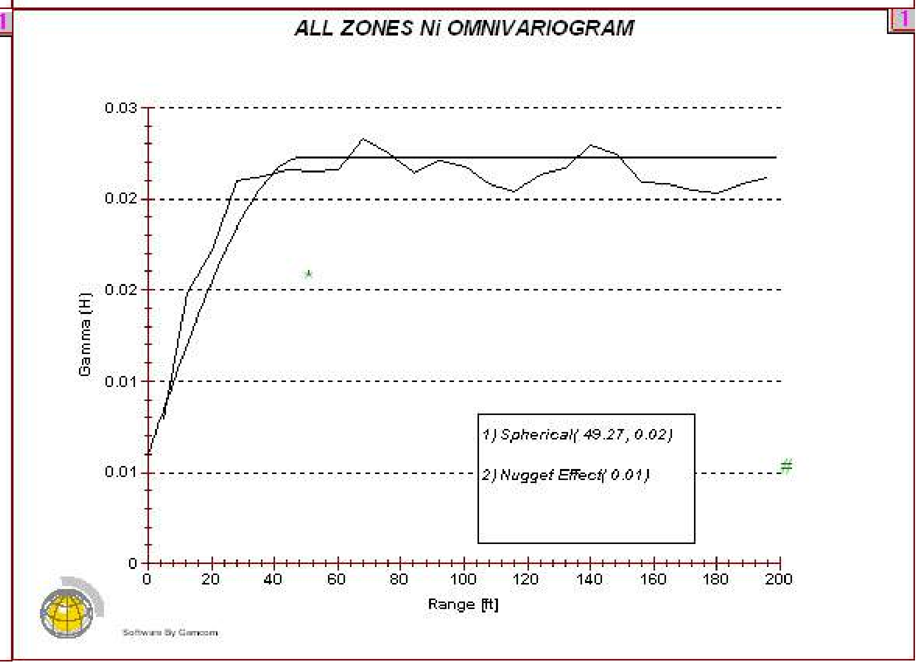

MAIN ZONE Cu ACROSS DIP AZ=180 DIP=-38
Gamma (H)
Range [ft]
1) Spherical(40.35, 1.02)
2) Nugget Effect(0.60)
Software By Gemcom

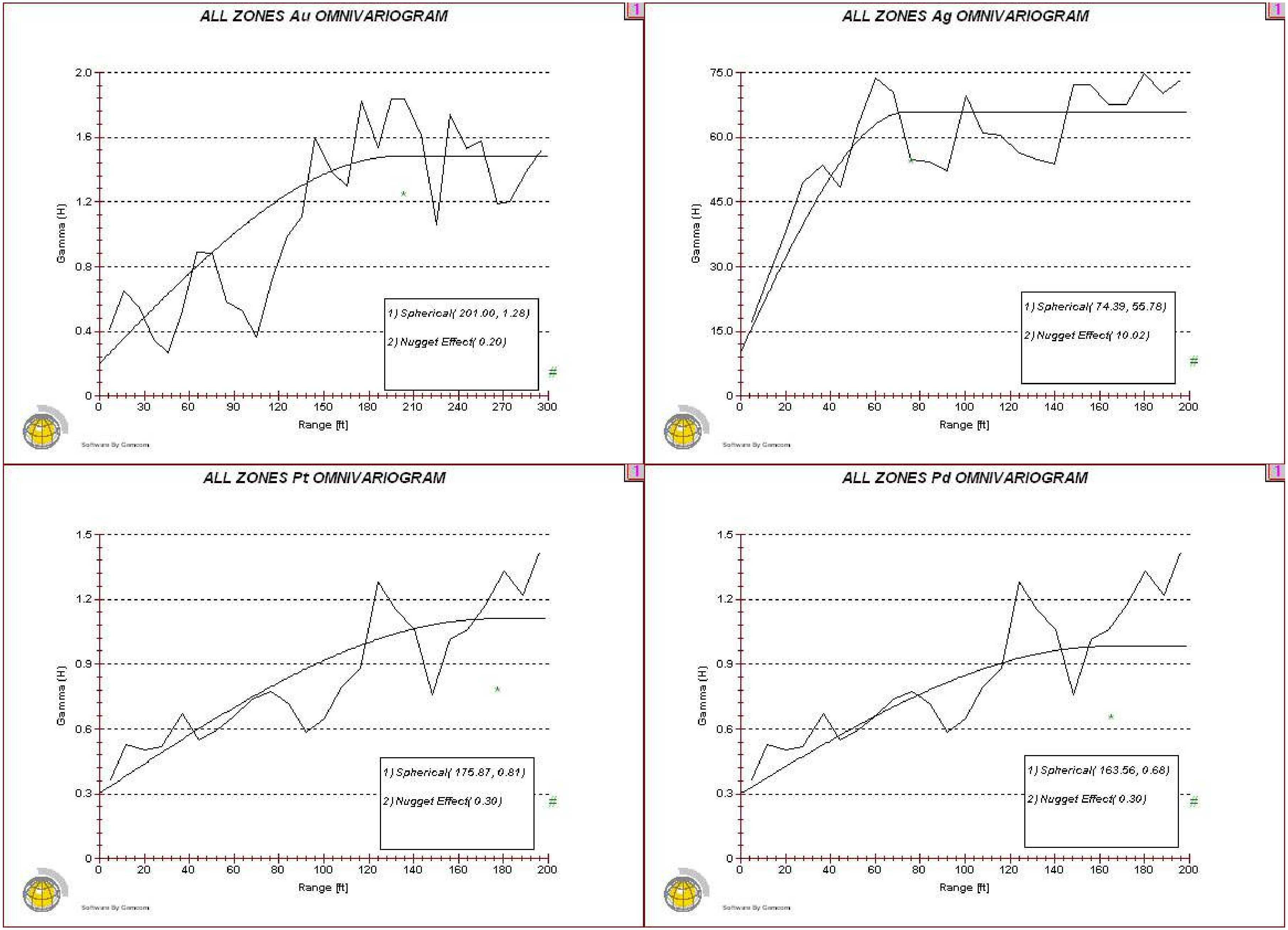

FOOTWALL ZONE Cu OMNIVARIOGRAM
Gamma (H)
Range [ft]
1) Spherical(125.61, 1.08)
2) Nugget Effect(1.00)
Software By Gemcom


FOOTWALL ZONE Cu ALONG STRIKE AZ=90 DIP=0
Gamma (H)
Range [ft]
1) Spherical(125.12, 0.87)
2) Nugget Effect(1.00)
Software By Gemcom

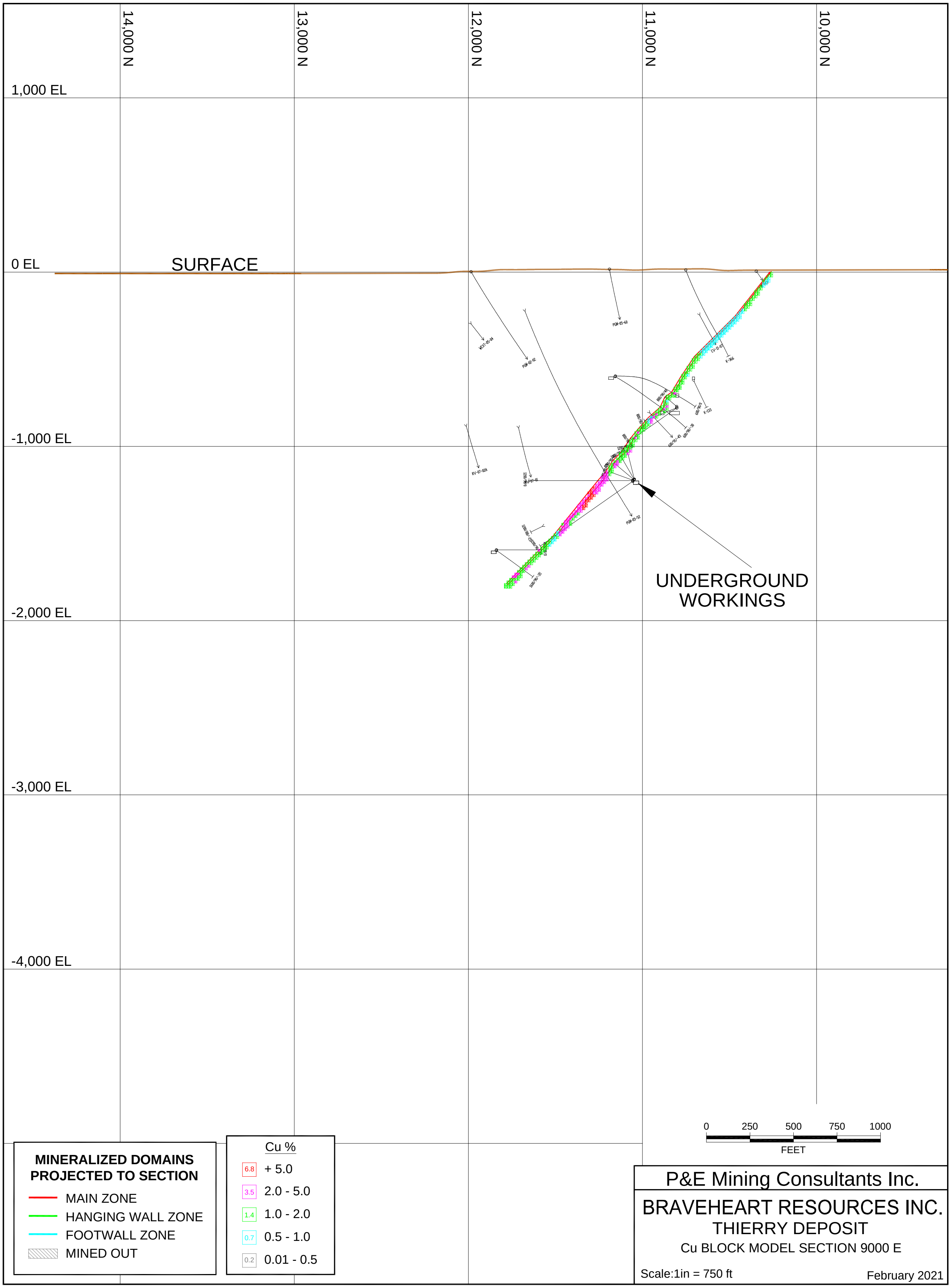

FOOTWALL ZONE Cu DOWN DIP AZ=0 DIP=-52
Gamma (H)
Range [ft]
1) Spherical(124.62, 0.99)
2) Nugget Effect(1.00)
Software By Gemcom


FOOTWALL ZONE Cu ACROSS DIP AZ=180 DIP=-38
Gamma (H)
Range [ft]
1) Spherical(30.44, 1.72)
2) Nugget Effect(1.00)
Software By Gemcom

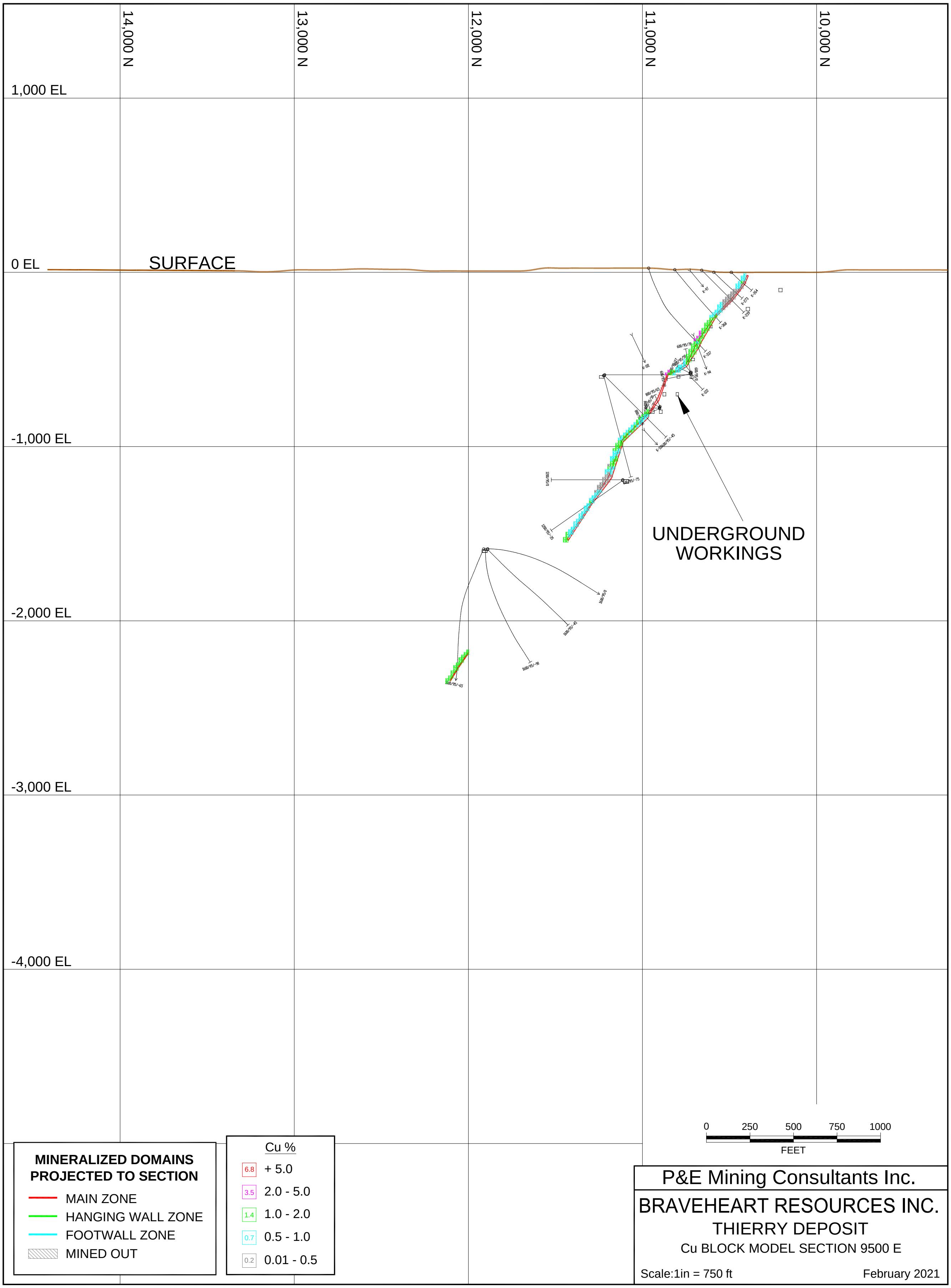

ALL ZONES Ni OMNIVARIOGRAM
Gamma (H)
Range [ft]
1) Spherical(49.27, 0.02)
2) Nugget Effect(0.01)
Software By Gemcom

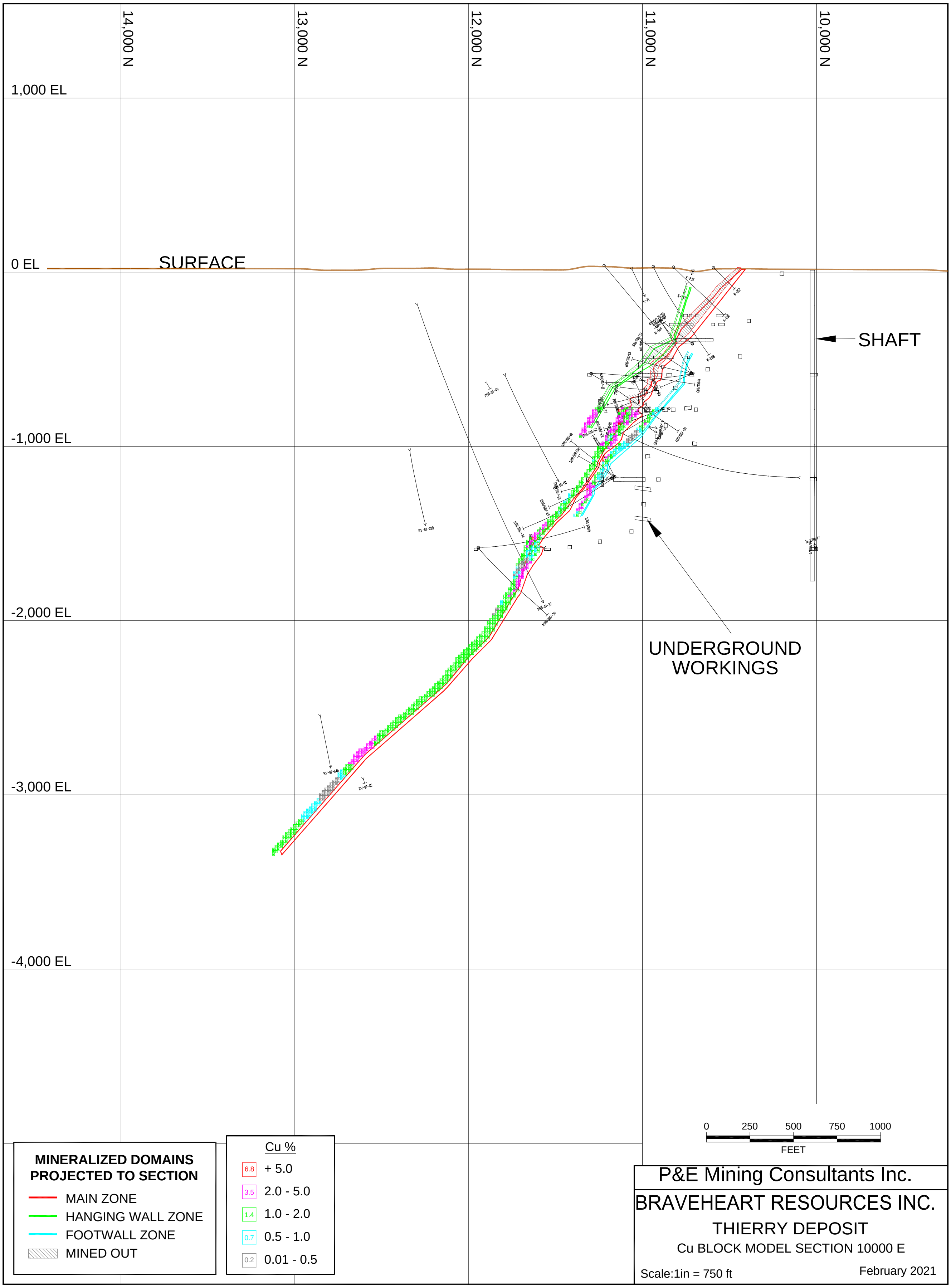

ALL ZONES Au OMNIVARIOGRAM
1) Spherical(201.00, 1.28)
2) Nugget Effect(0.20)
Gamma (H)
Range [ft]
ALL ZONES Ag OMNIVARIOGRAM
1) Spherical(74.39, 55.78)
2) Nugget Effect(10.02)
Gamma (H)
Range [ft]
ALL ZONES Pt OMNIVARIOGRAM
1) Spherical(175.87, 0.81)
2) Nugget Effect(0.30)
Gamma (H)
Range [ft]
ALL ZONES Pd OMNIVARIOGRAM
1) Spherical(163.56, 0.68)
2) Nugget Effect(0.30)
Gamma (H)
Range [ft]

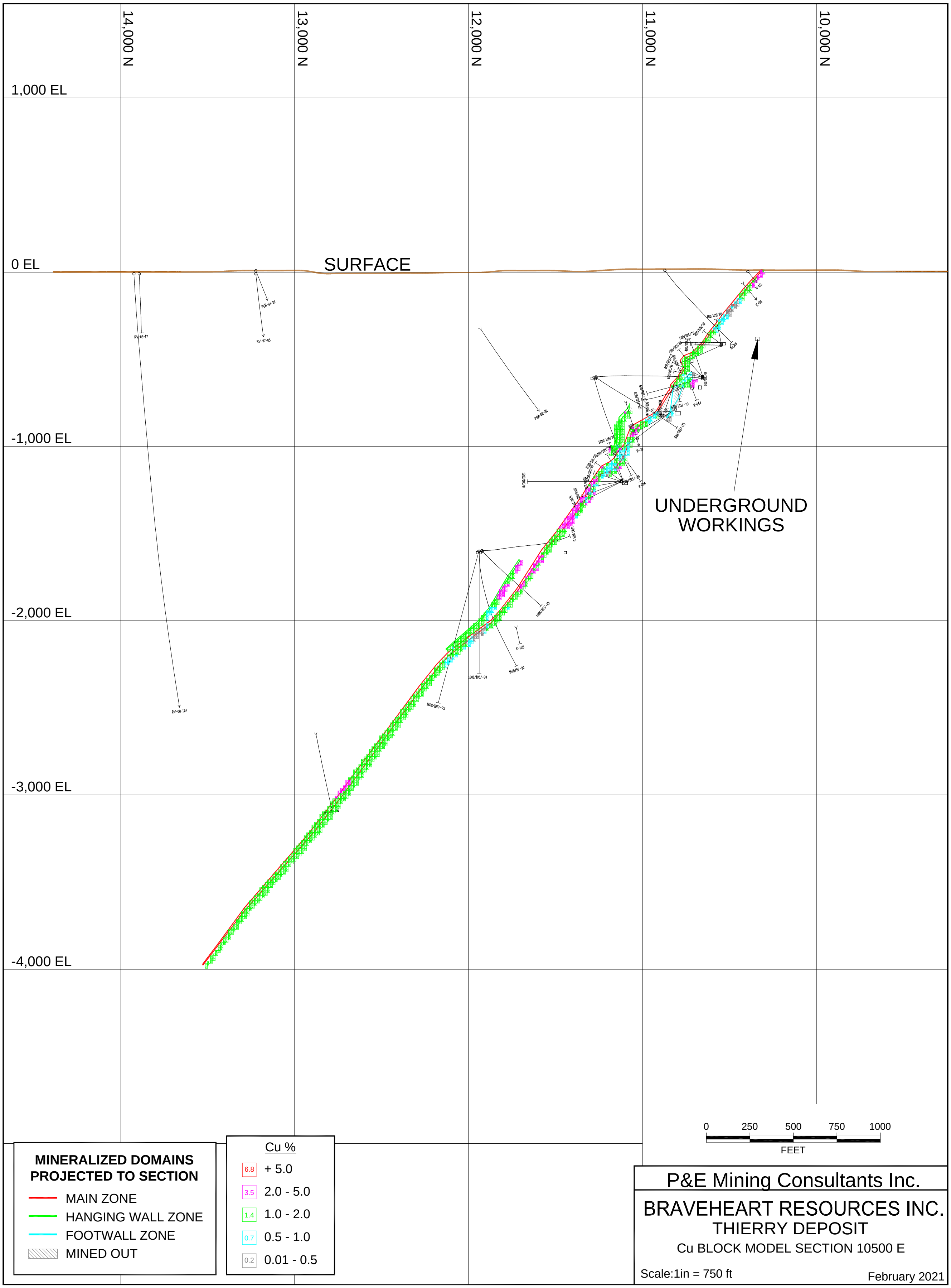
APPENDIX E CU BLOCK MODEL CROSS SECTIONS AND PLANS

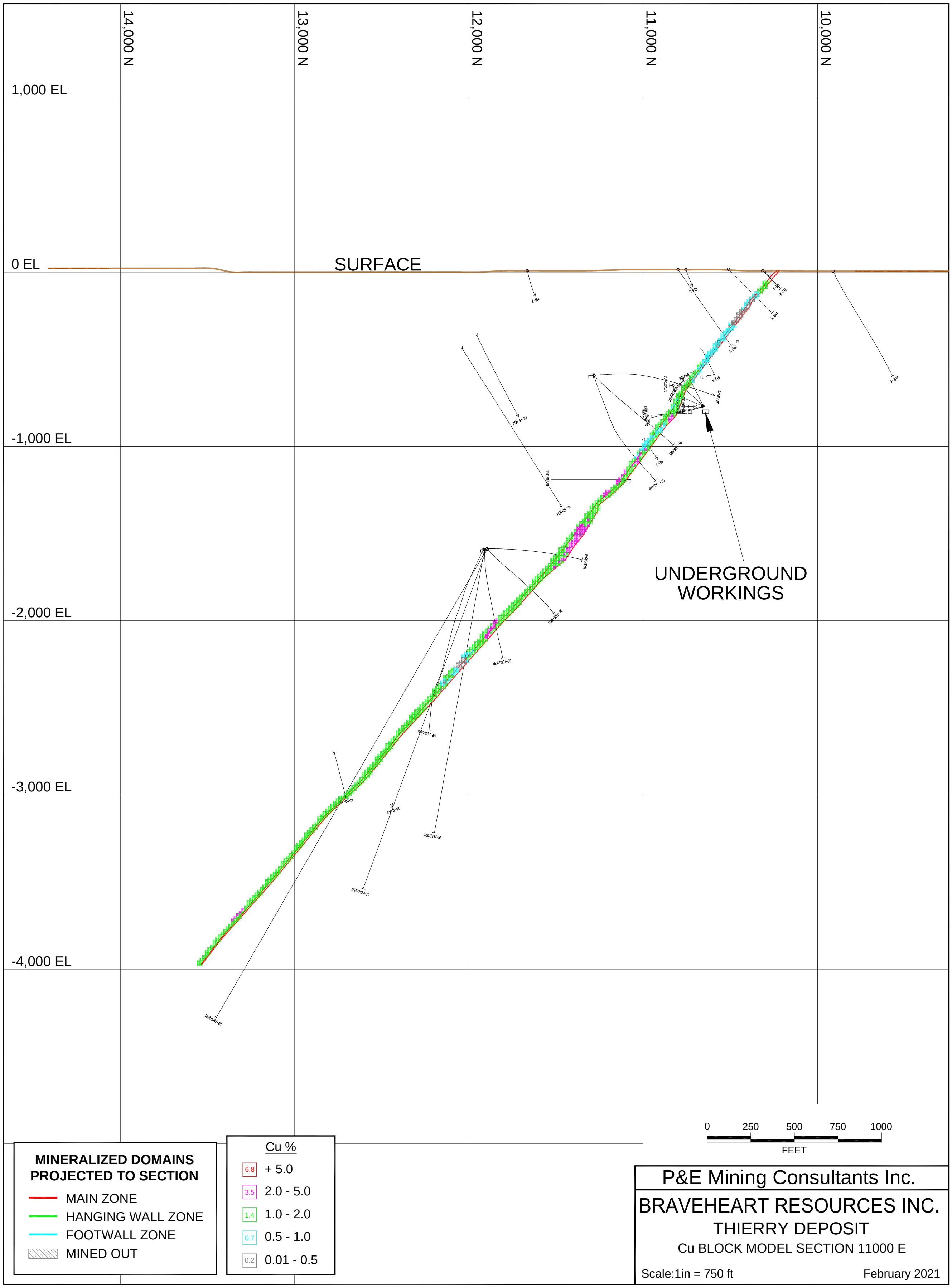


14,000 N
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8 + 5.0
3.5 2.0 - 5.0
1.4 1.0 - 2.0
0.7 0.5 - 1.0
0.2 0.01 - 0.5
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 9000 E
Scale:1in = 750 ft
February 2021


14,000 N
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND
WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS
PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8
+ 5.0
3.5
2.0 - 5.0
1.4
1.0 - 2.0
0.7
0.5 - 1.0
0.2
0.01 - 0.5
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 9500 E
Scale:1in = 750 ft
February 2021


14,000 N
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
SHAFT
-1,000 EL
-2,000 EL
UNDERGROUND WORKINGS
-3,000 EL
-4,000 EL
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
+ 5.0
2.0 - 5.0
1.0 - 2.0
0.5 - 1.0
0.01 - 0.5
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 10000 E
Scale:1in = 750 ft
February 2021


14,000 N
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8 + 5.0
3.5 2.0 - 5.0
1.4 1.0 - 2.0
0.7 0.5 - 1.0
0.2 0.01 - 0.5
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 10500 E
Scale:1in = 750 ft
February 2021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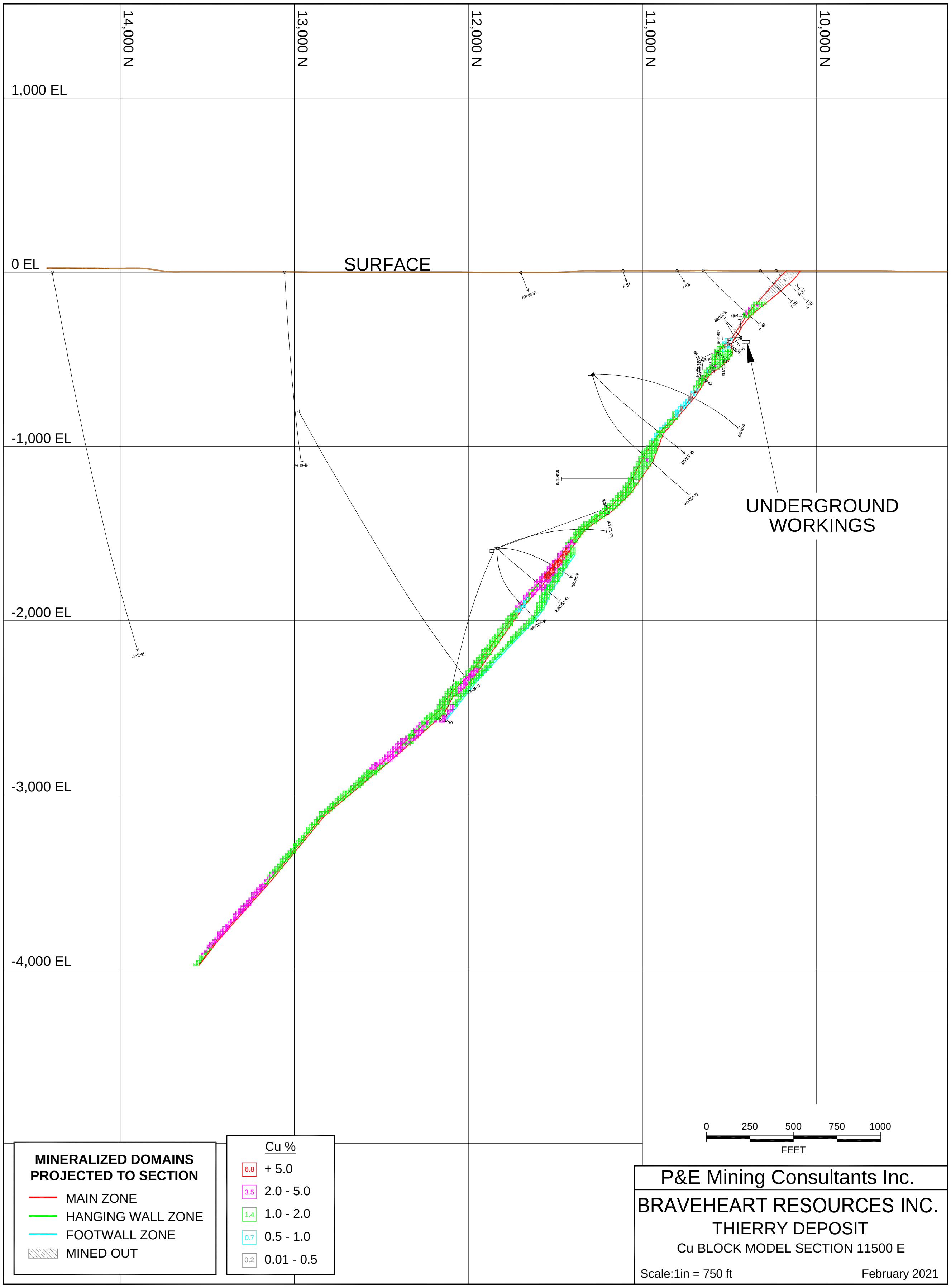
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8
+ 5.0
3.5
2.0 - 5.0
1.4
1.0 - 2.0
0.7
0.5 - 1.0
0.2
0.01 - 0.5
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 11000 E
Scale:1in = 750 ft
February 2021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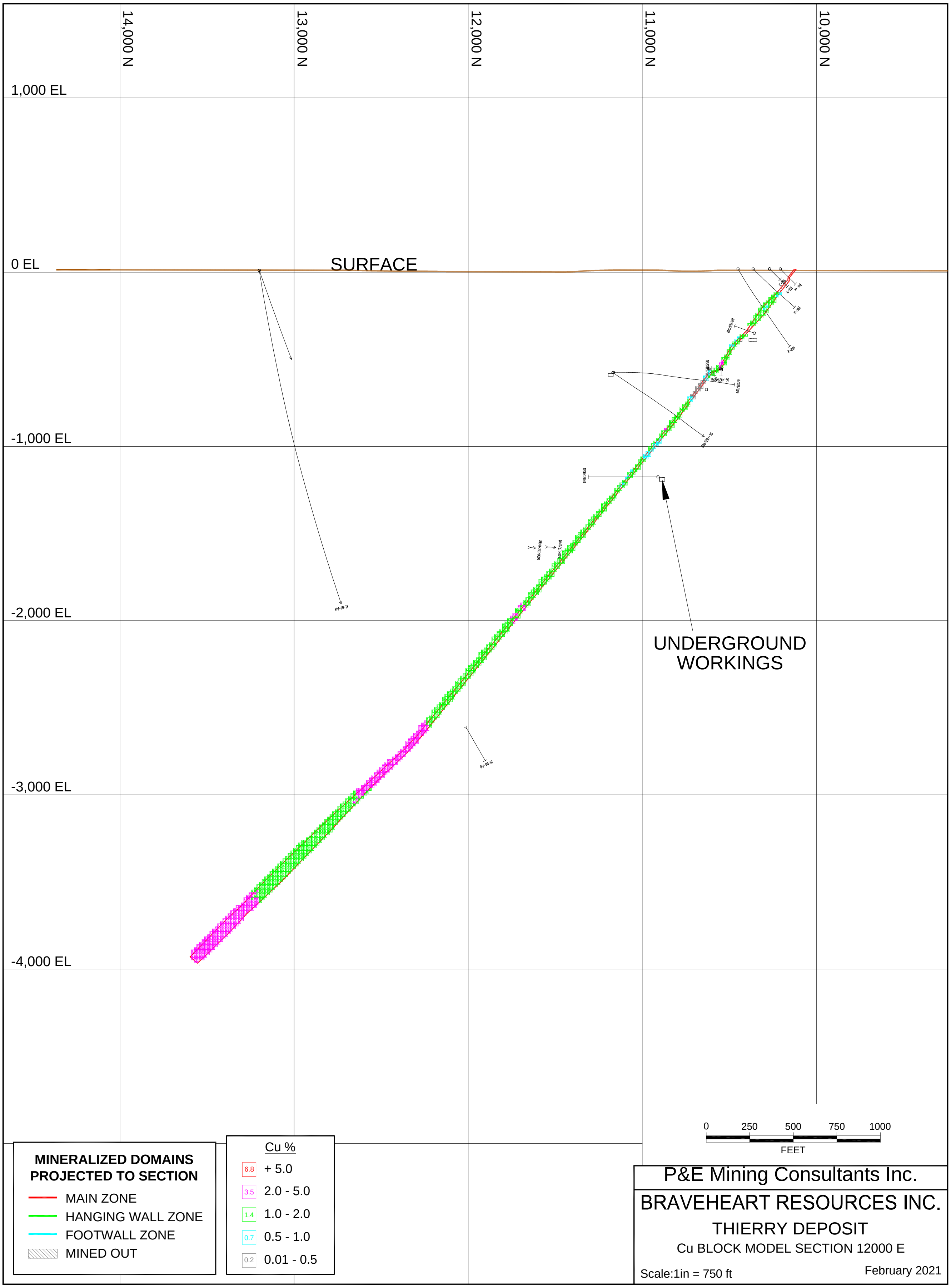
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND
WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS
PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8
+ 5.0
3.5
2.0 - 5.0
1.4
1.0 - 2.0
0.7
0.5 - 1.0
0.2
0.01 - 0.5
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 11500 E
Scale:1in = 750 ft
February 2021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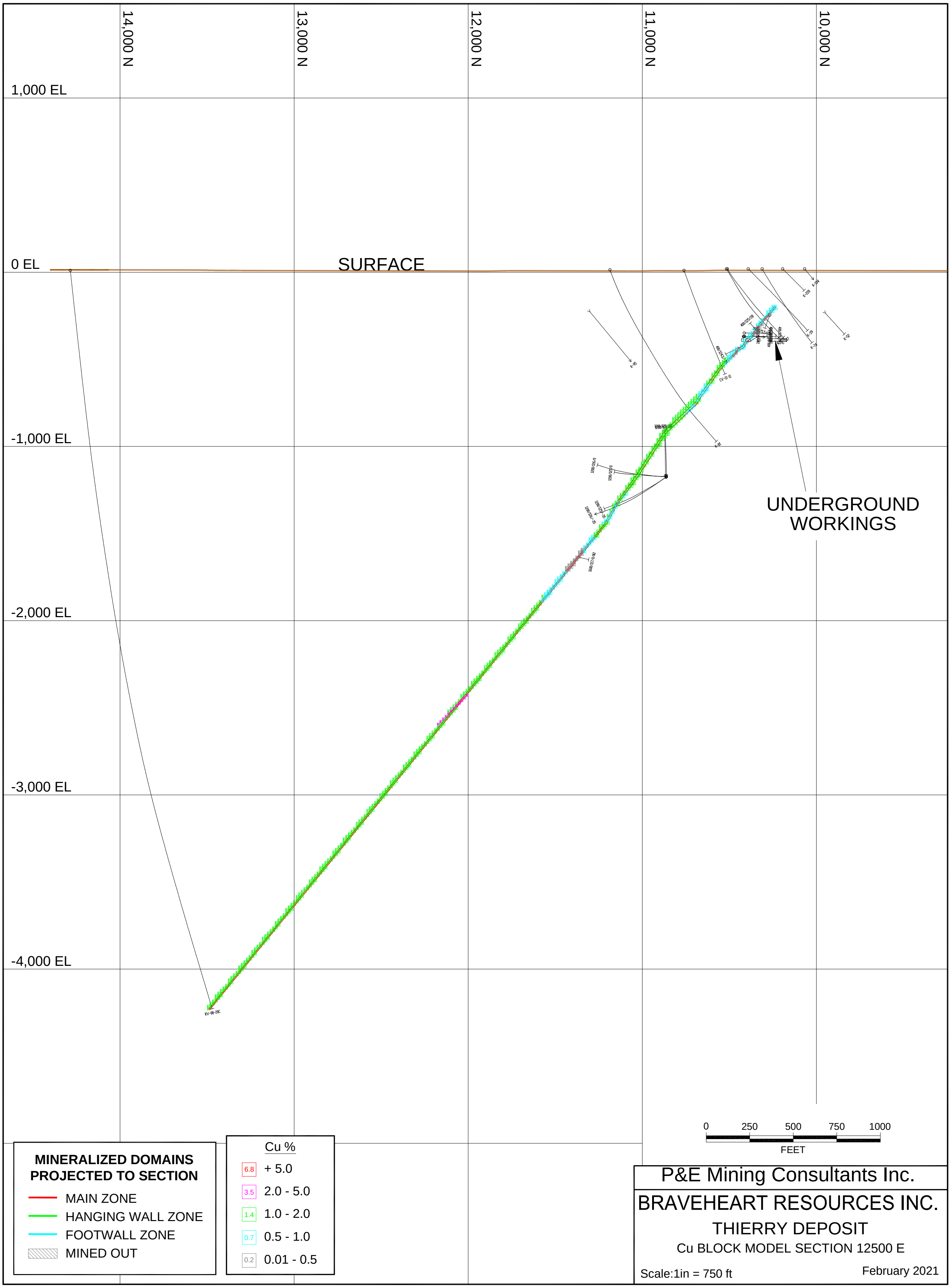
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
+ 5.0
2.0 - 5.0
1.0 - 2.0
0.5 - 1.0
0.01 - 0.5
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 12000 E
Scale:1in = 750 ft
February 2021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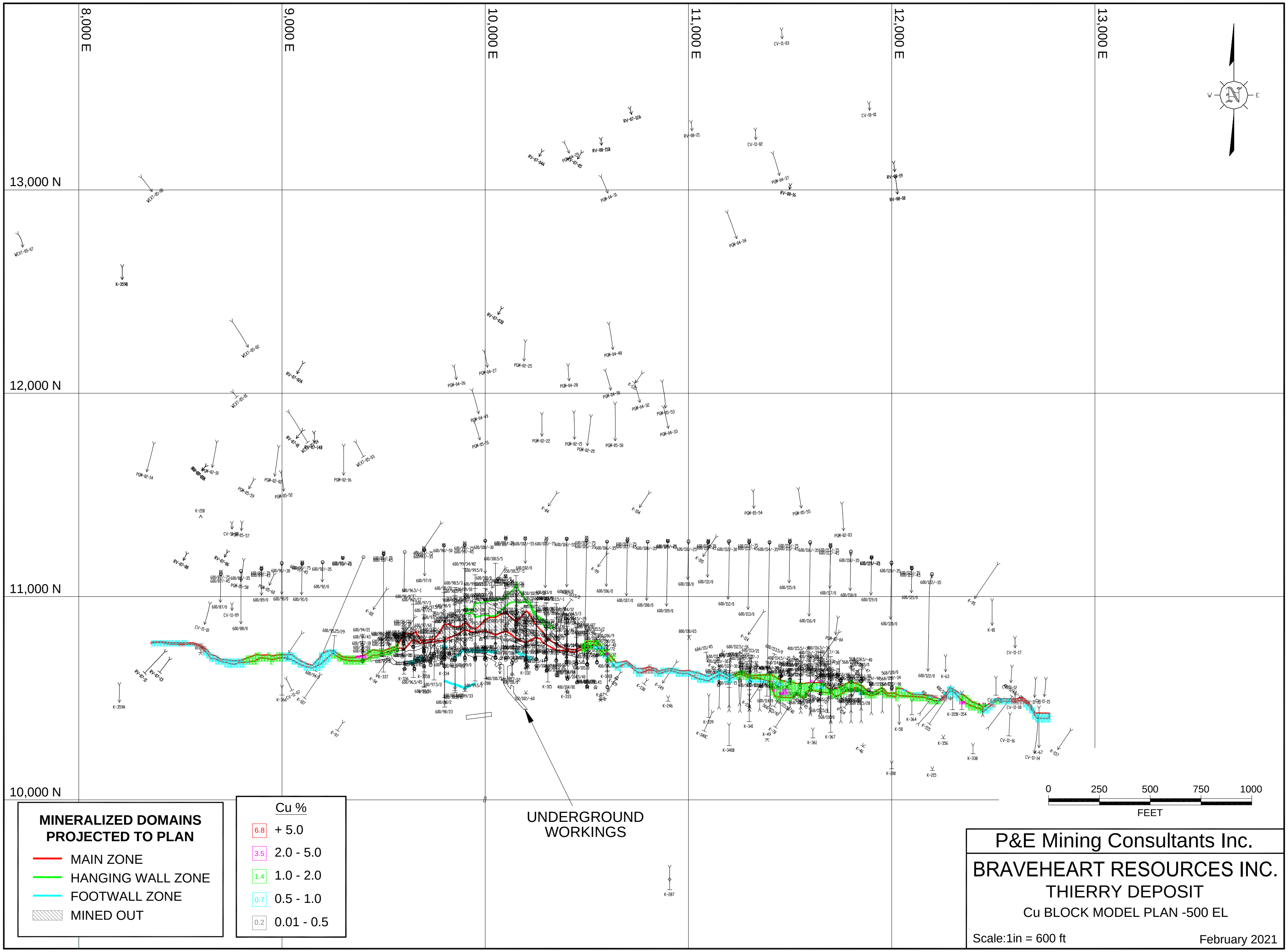
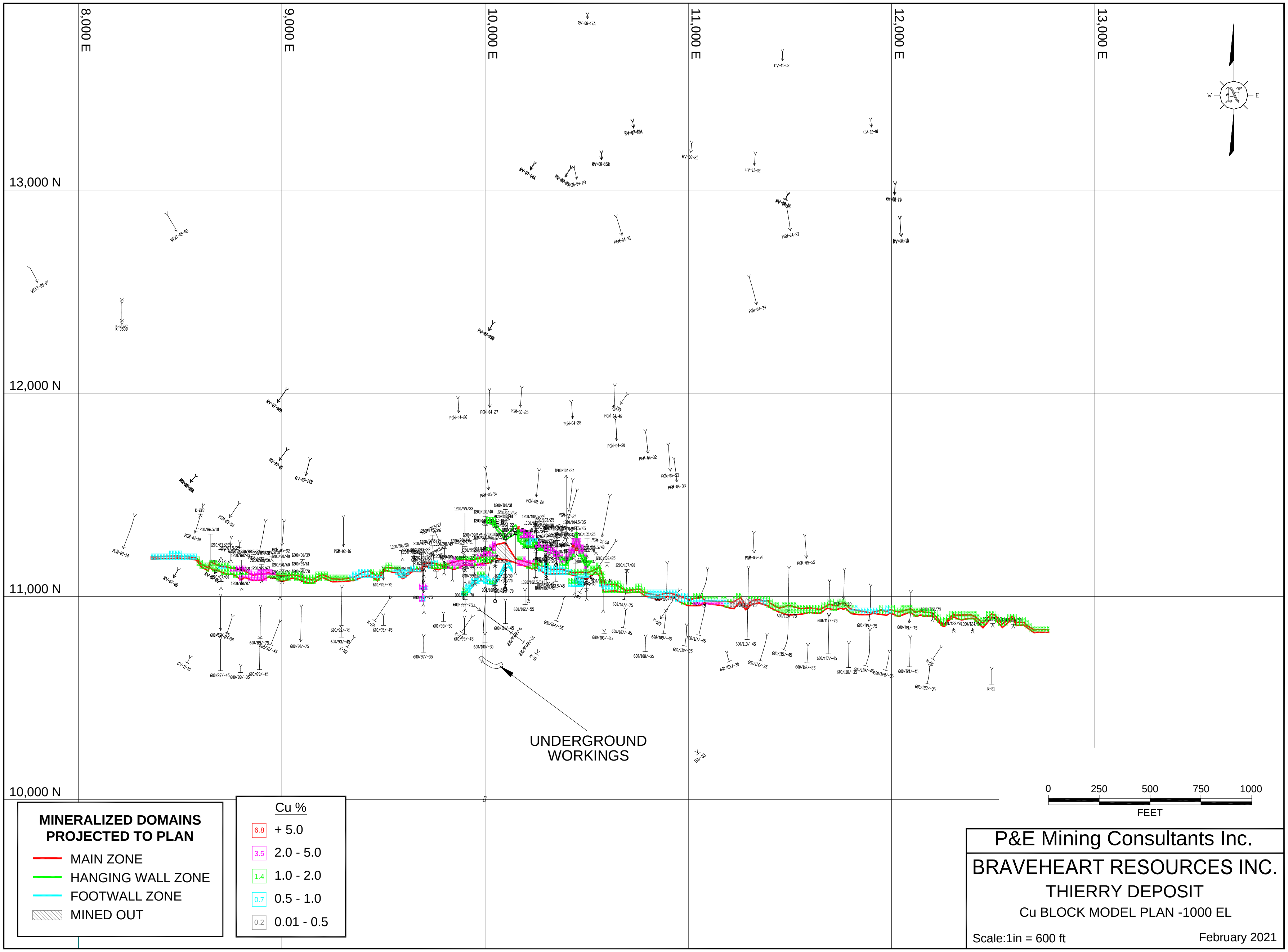
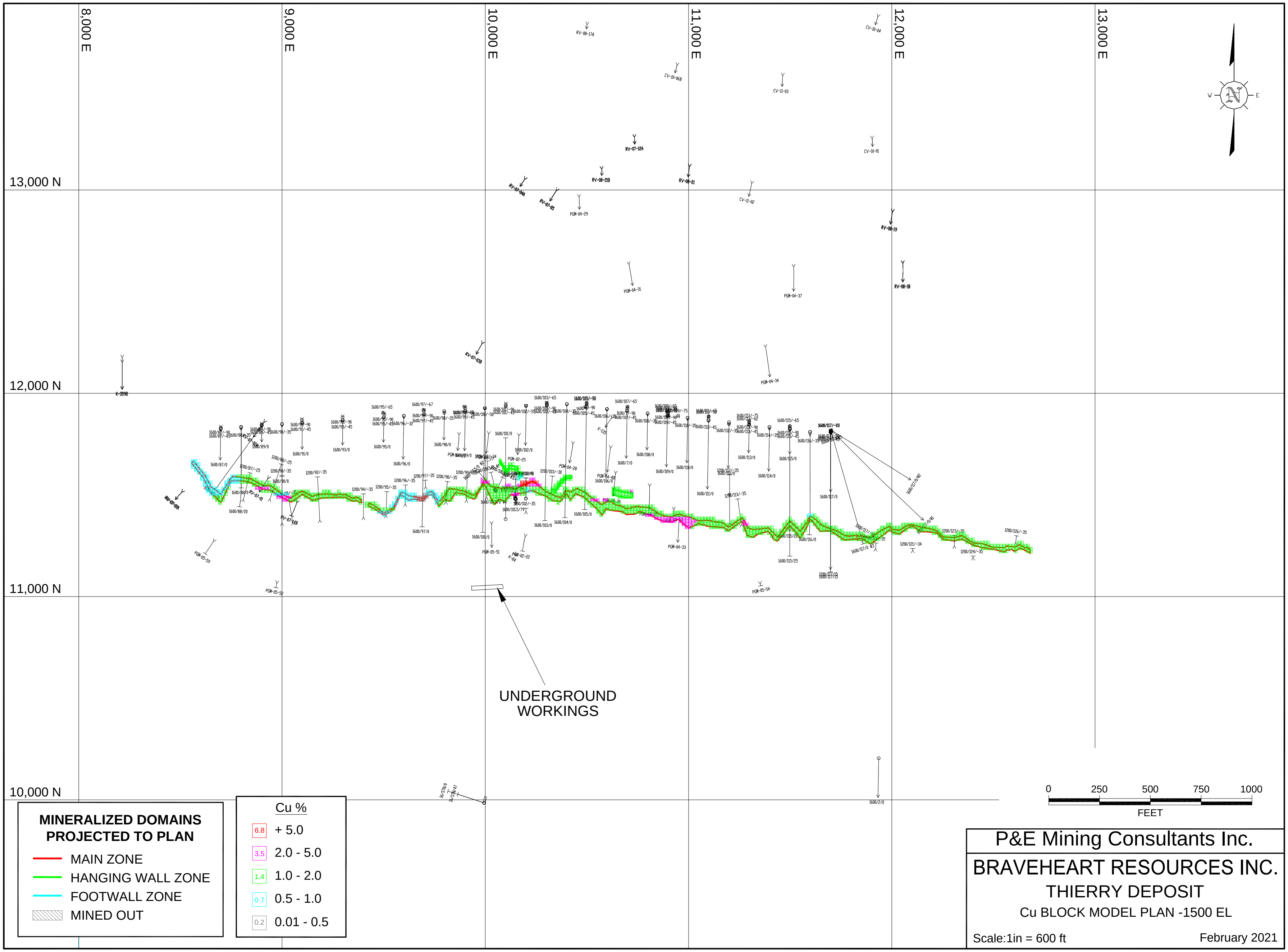
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8
+ 5.0
3.5
2.0 - 5.0
1.4
1.0 - 2.0
0.7
0.5 - 1.0
0.2
0.01 - 0.5
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL SECTION 12500 E
Scale:1in = 750 ft
February 2021


8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND
WORKINGS
MINERALIZED DOMAINS
PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
+ 5.0
2.0 - 5.0
1.0 - 2.0
0.5 - 1.0
0.01 - 0.5
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL PLAN -500 EL
Scale:1in = 600 ft
February 2021


8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8 + 5.0
3.5 2.0 - 5.0
1.4 1.0 - 2.0
0.7 0.5 - 1.0
0.2 0.01 - 0.5
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL PLAN -1000 EL
Scale:1in = 600 ft
February 2021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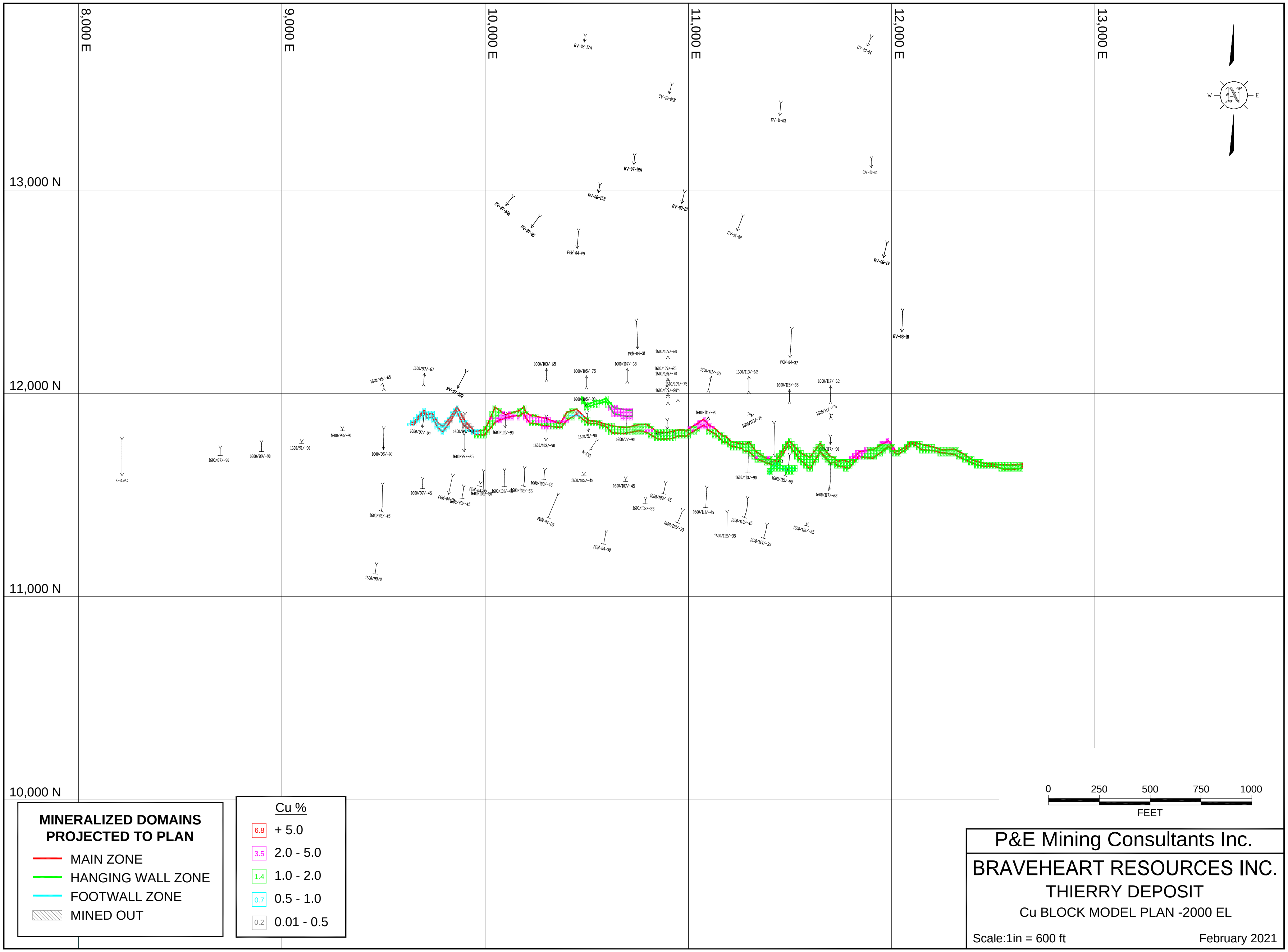
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND
WORKINGS
MINERALIZED DOMAINS
PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
+ 5.0
2.0 - 5.0
1.0 - 2.0
0.5 - 1.0
0.01 - 0.5
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL PLAN -1500 EL
Scale:1in = 600 ft
February 2021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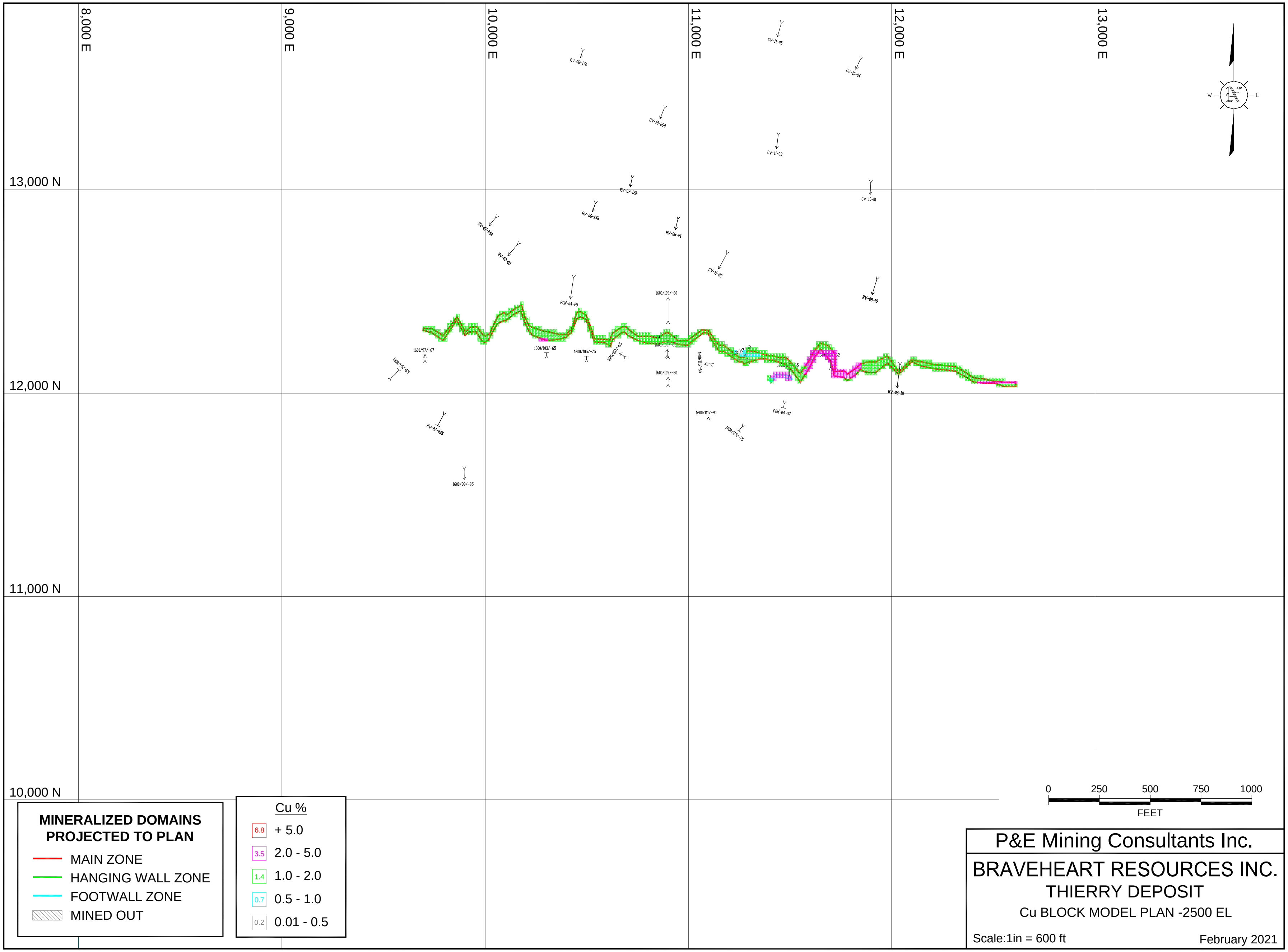
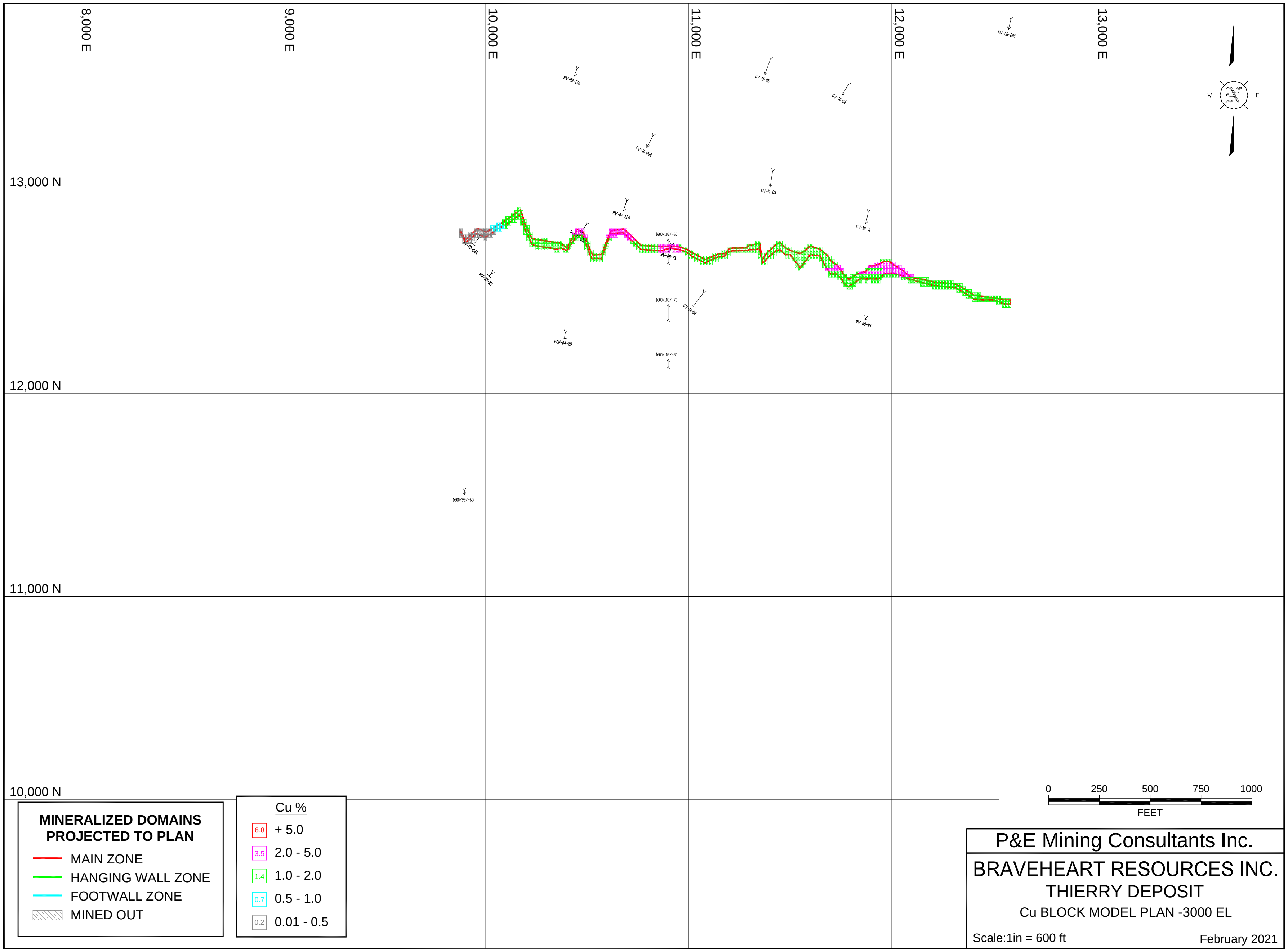
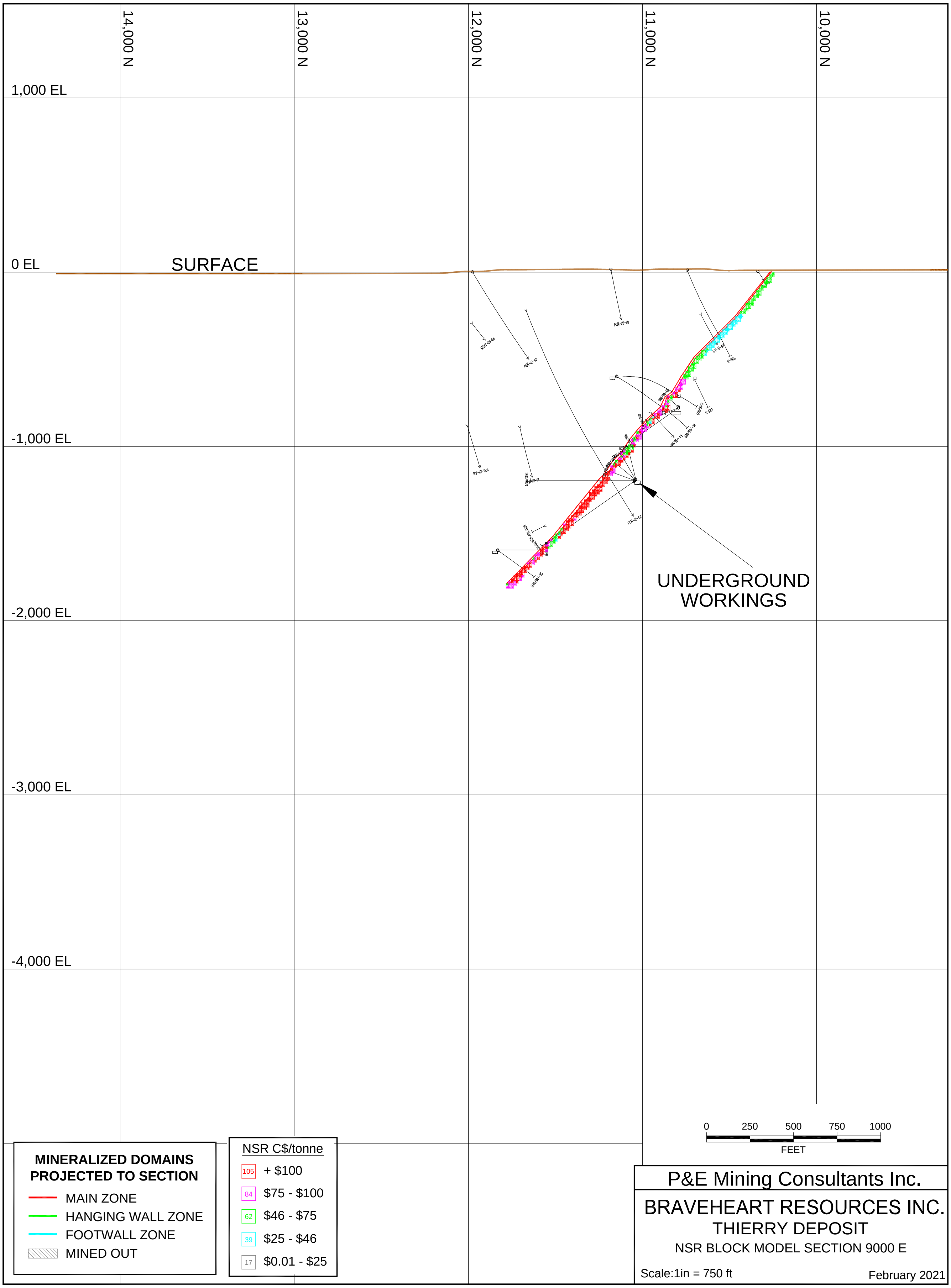
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8 + 5.0
3.5 2.0 - 5.0
1.4 1.0 - 2.0
0.7 0.5 - 1.0
0.2 0.01 - 0.5
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL PLAN -2000 EL
Scale:1in = 600 ft
February 2021


8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
+ 5.0
2.0 - 5.0
1.0 - 2.0
0.5 - 1.0
0.01 - 0.5
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL PLAN -2500 EL
Scale:1in = 600 ft
February 2021


8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
Cu %
6.8 + 5.0
3.5 2.0 - 5.0
1.4 1.0 - 2.0
0.7 0.5 - 1.0
0.2 0.01 - 0.5
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
Cu BLOCK MODEL PLAN -3000 EL
Scale:1in = 600 ft
February 2021

APPENDIX F NSR BLOCK MODEL CROSS SECTIONS AND PLANS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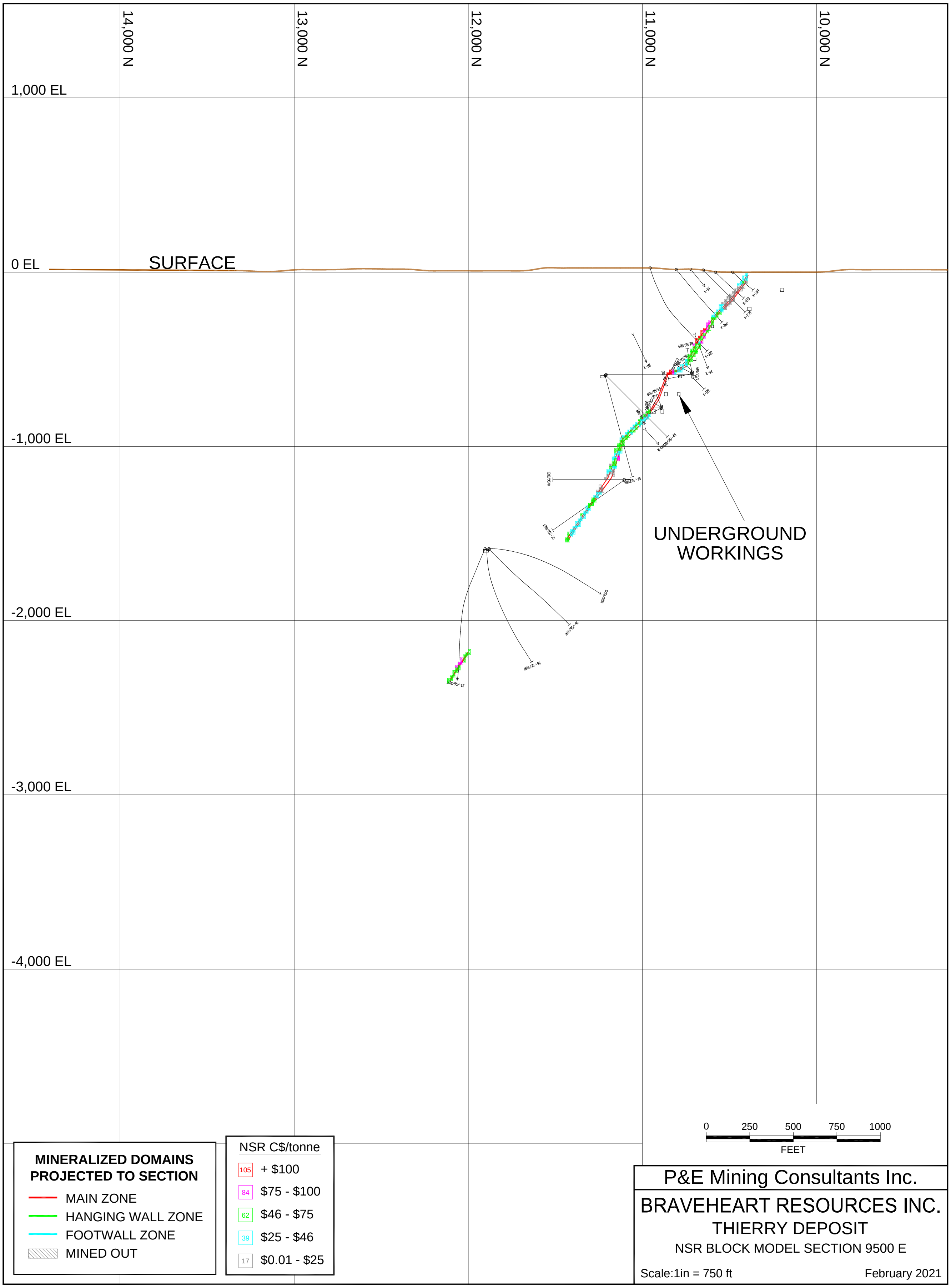
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 9000 E
Scale:1in = 750 ft
February 2021



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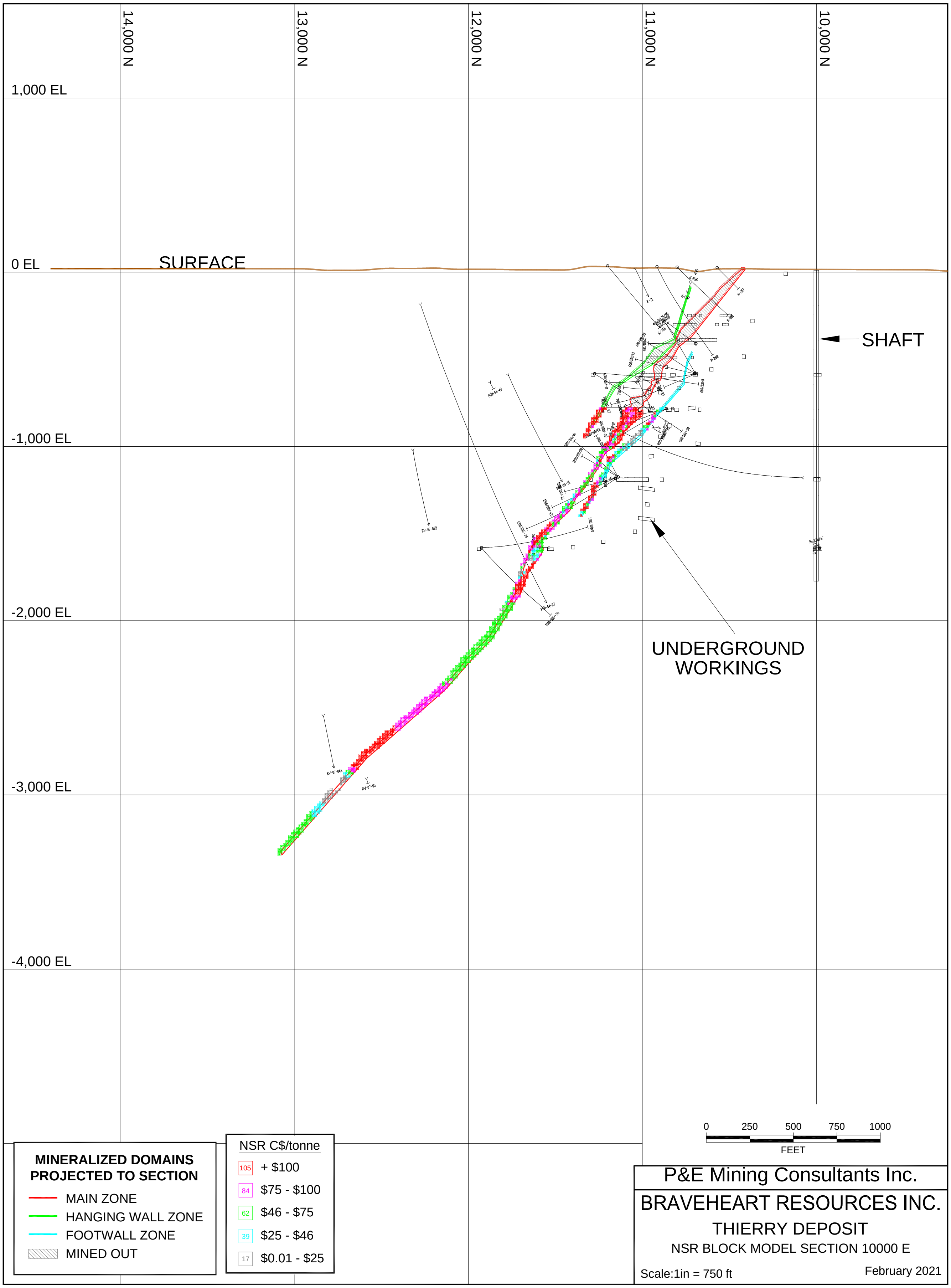
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105
+ $100
84
$75 - $100
62
$46 - $75
39
$25 - $46
17
$0.01 - $25
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 9500 E
Scale:1in = 750 ft
February 2021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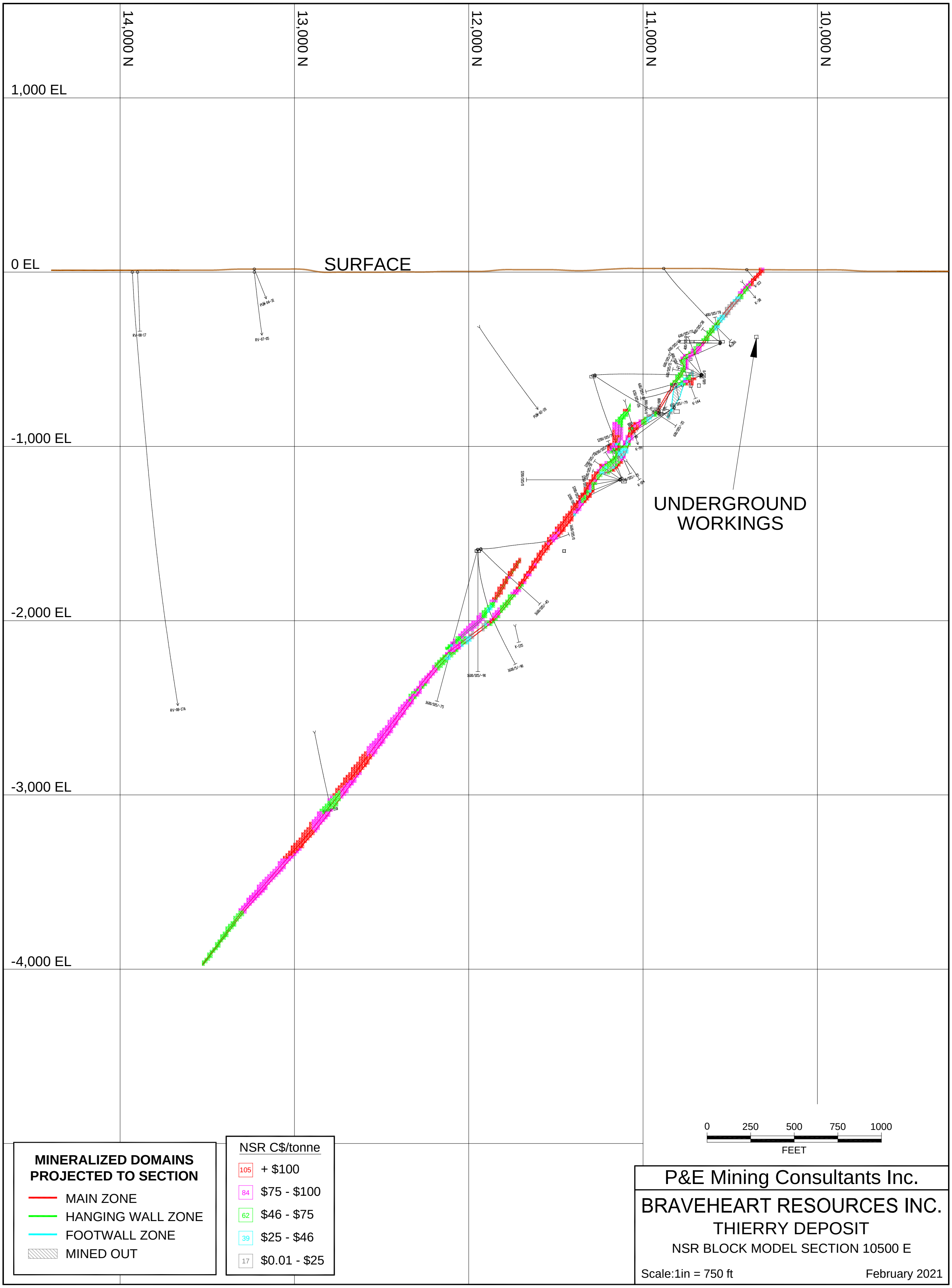
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
SHAFT
-1,000 EL
-2,000 EL
UNDERGROUND WORKINGS
-3,000 EL
-4,000 EL
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 10000 E
Scale:1in = 750 ft
February 2021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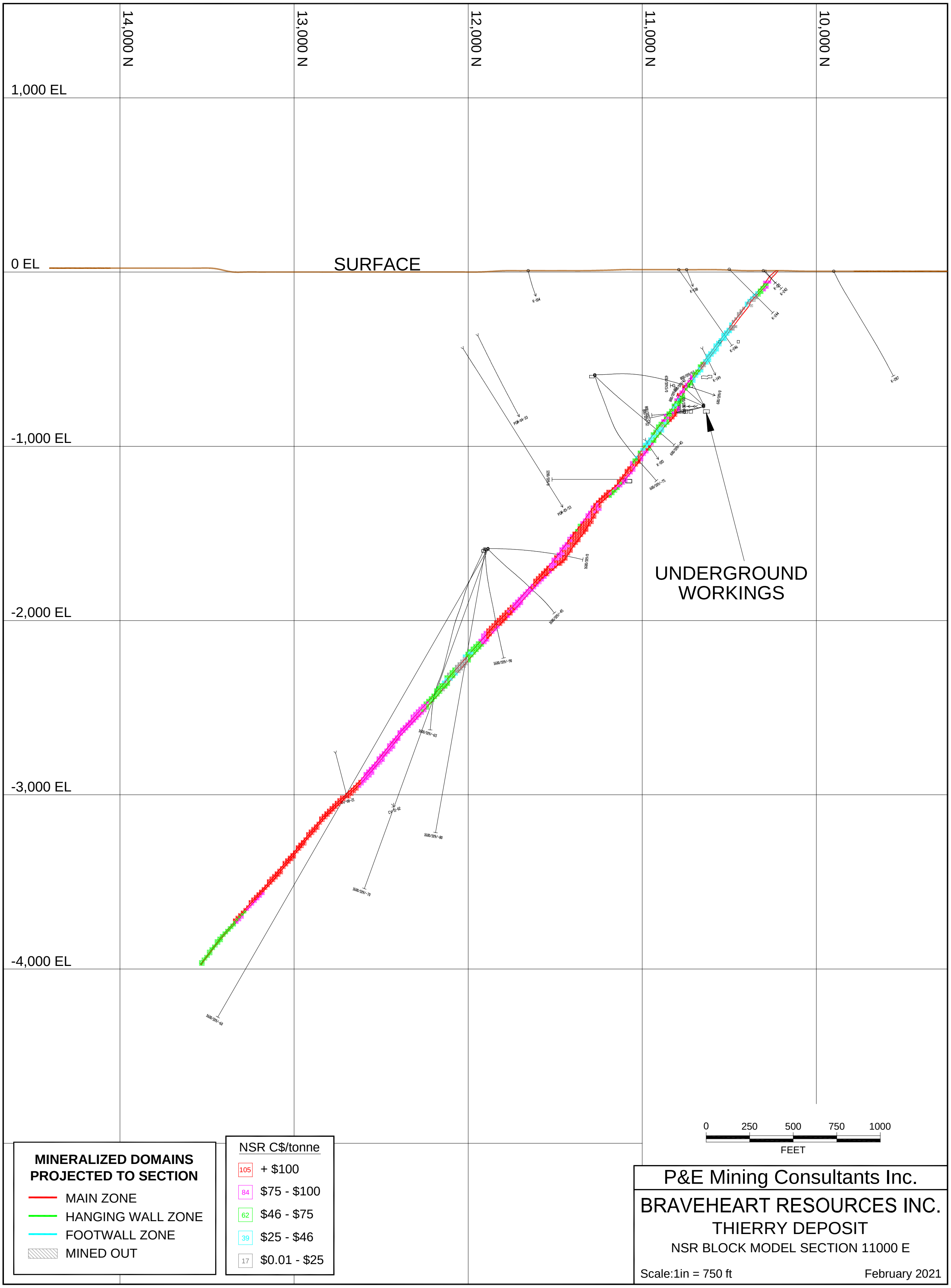
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND
WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS
PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
+ $100
$75 - $100
$46 - $75
$25 - $46
$0.01 - $25
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 10500 E
Scale:1in = 750 ft
February 2021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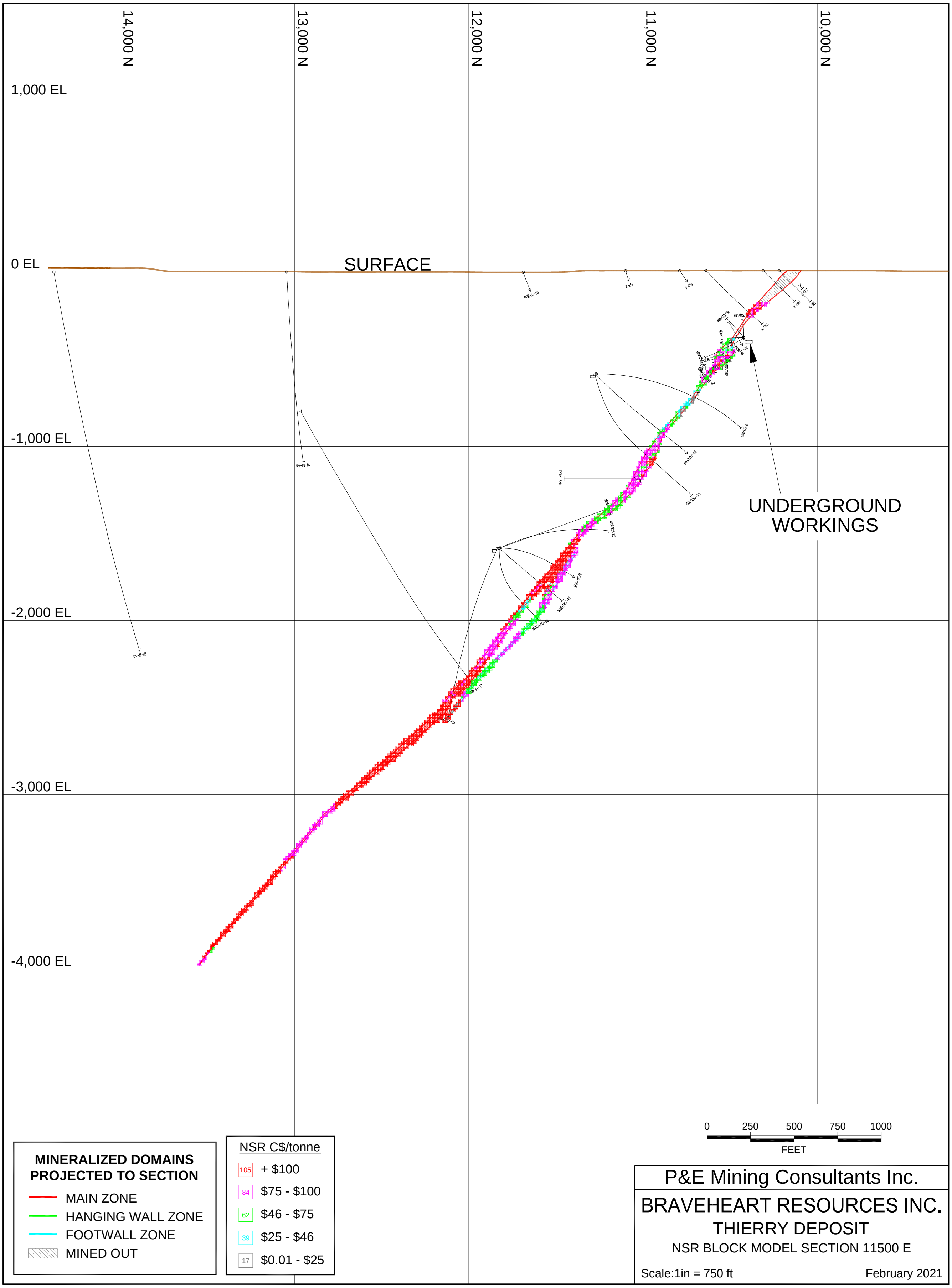
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
+ $100
$75 - $100
$46 - $75
$25 - $46
$0.01 - $25
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 11000 E
Scale:1in = 750 ft
February 2021



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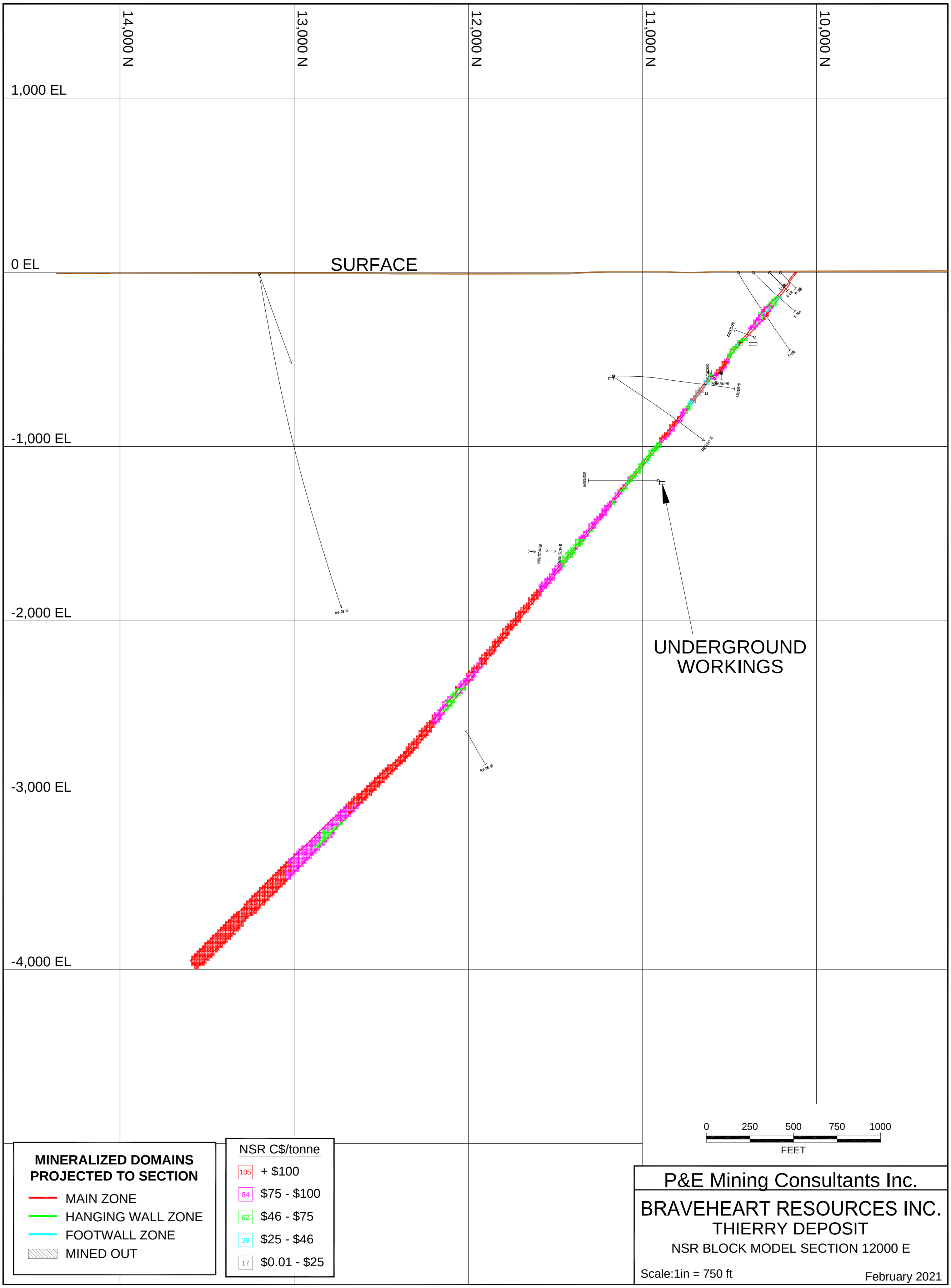
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 11500 E
Scale:1in = 750 ft
February 2021



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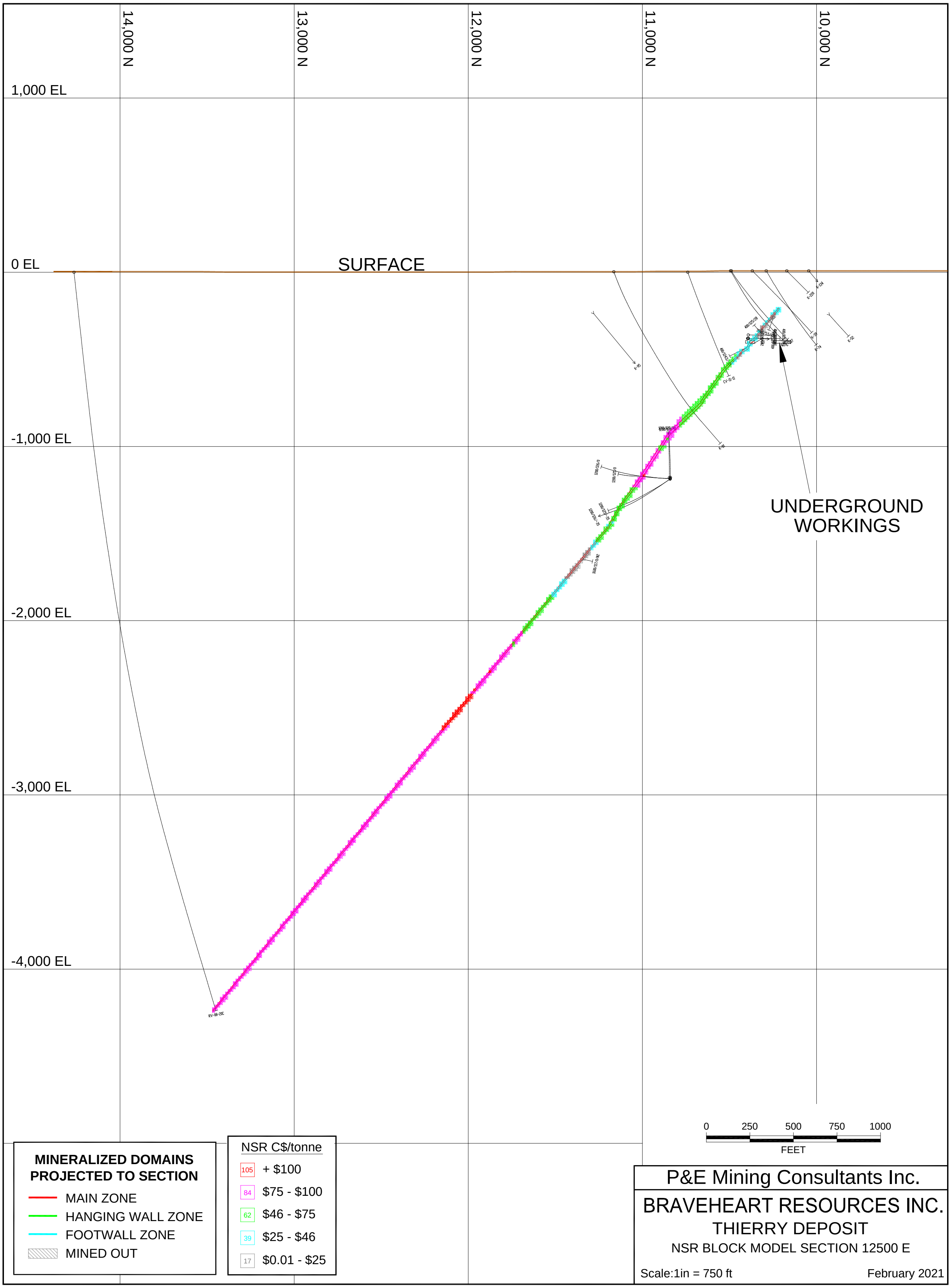
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
SURFACE
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
+ $100
$75 - $100
$46 - $75
$25 - $46
$0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 12000 E
Scale:1in = 750 ft
February 2021



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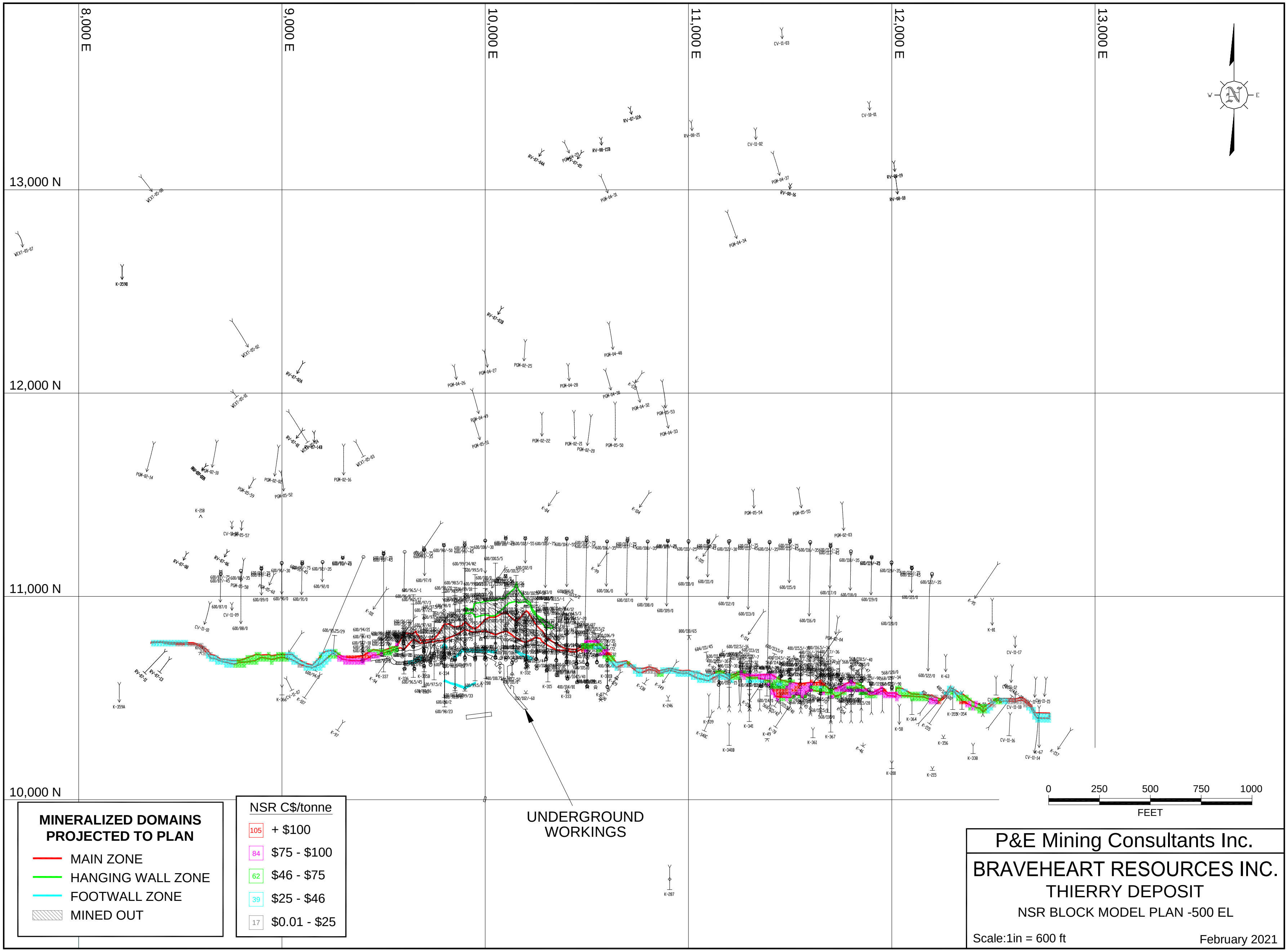
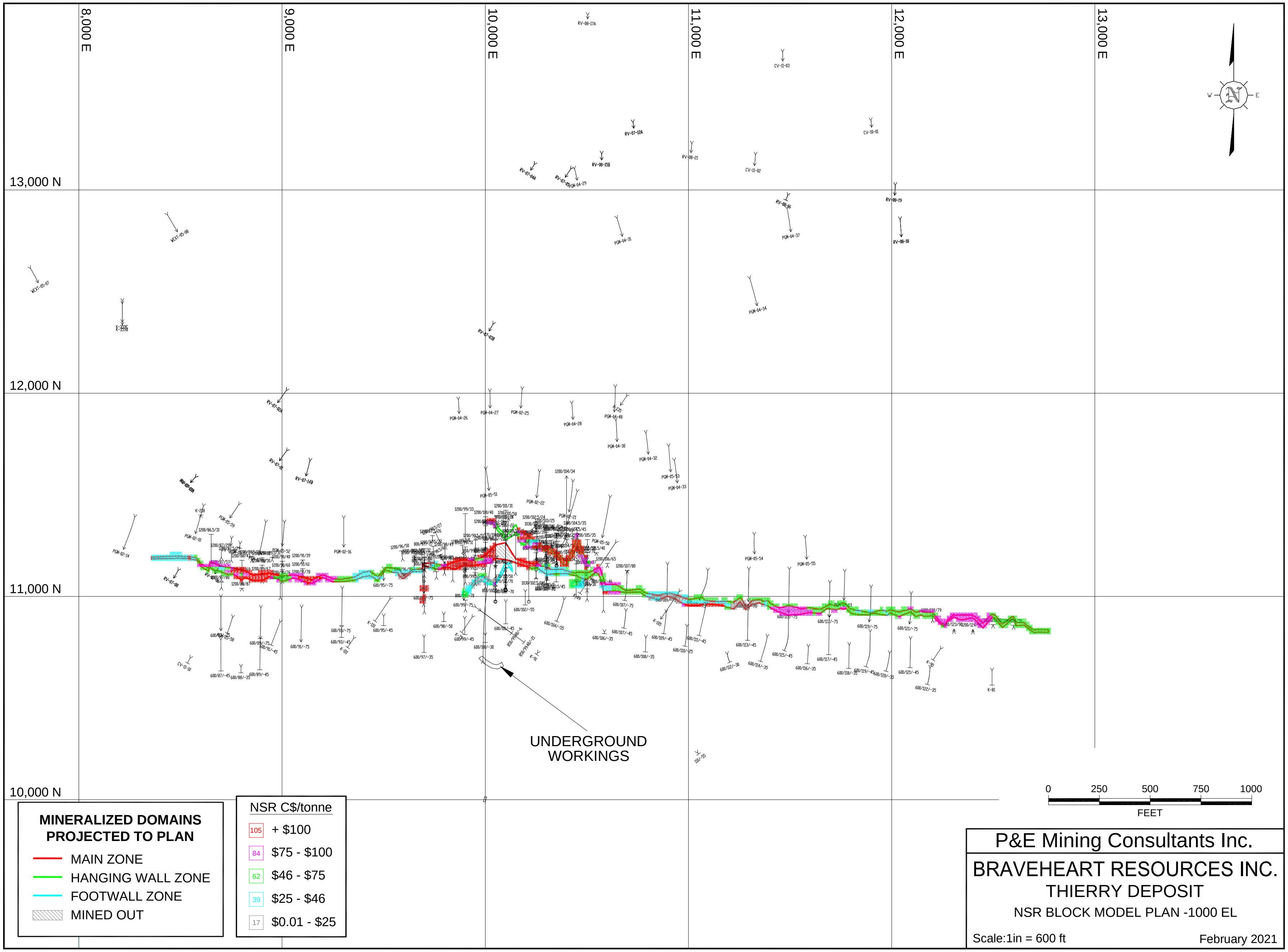
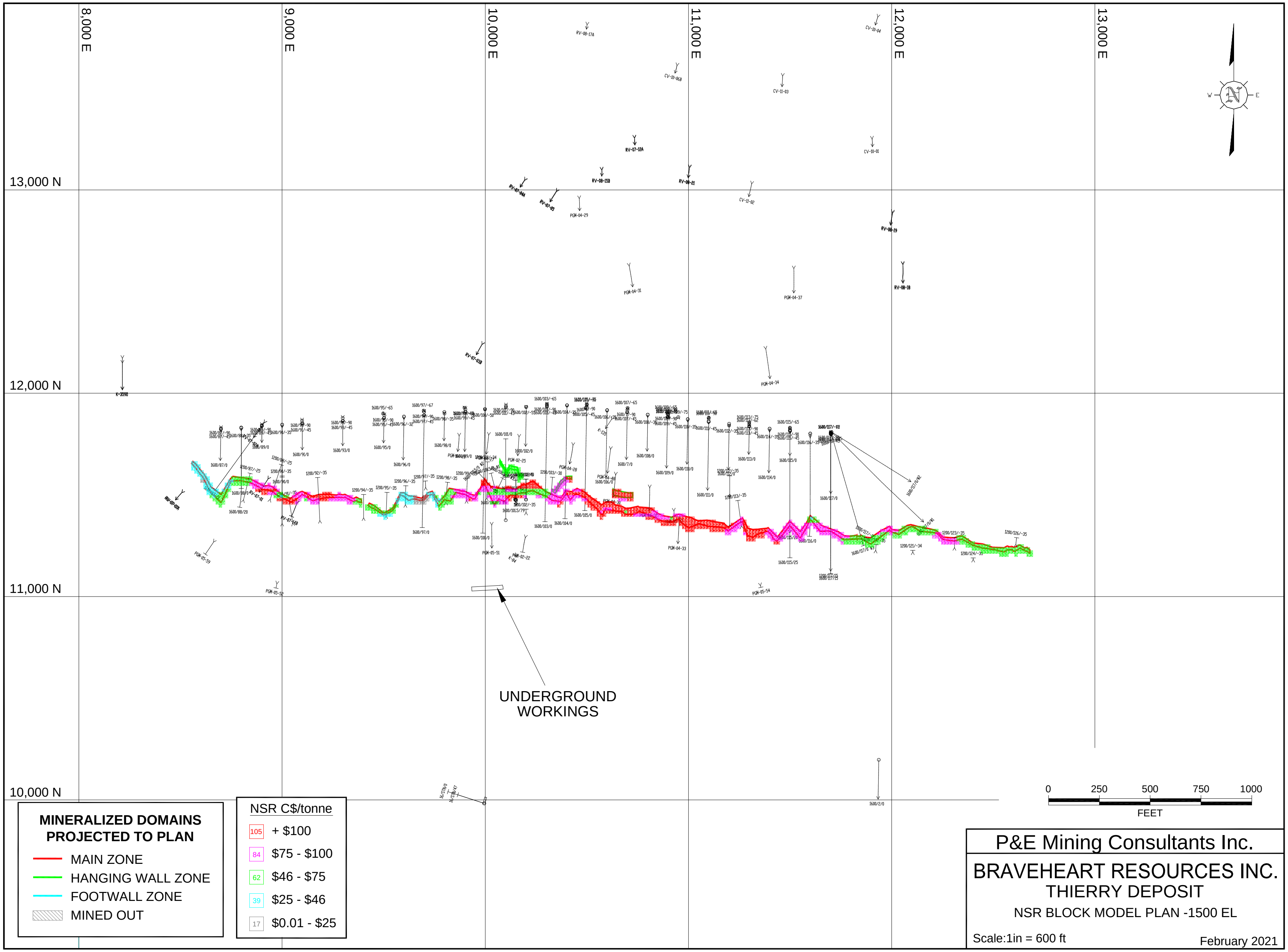
13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL SECTION 12500 E
Scale:1in = 750 ft
February 2021


8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL PLAN -500 EL
Scale:1in = 600 ft
February 2021



8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL PLAN -1000 EL
Scale:1in = 600 ft
February 2021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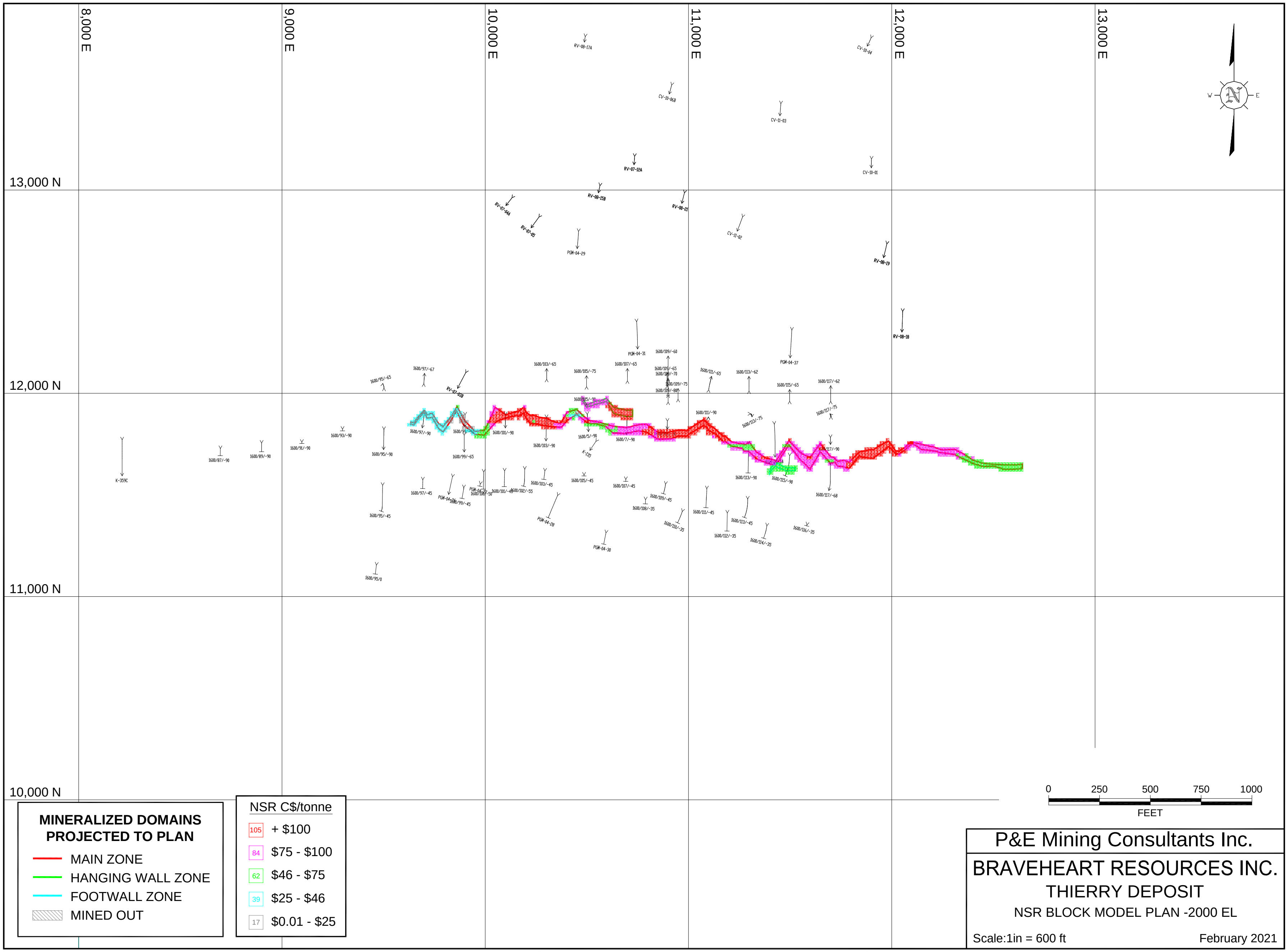
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL PLAN -1500 EL
Scale:1in = 600 ft
February 2021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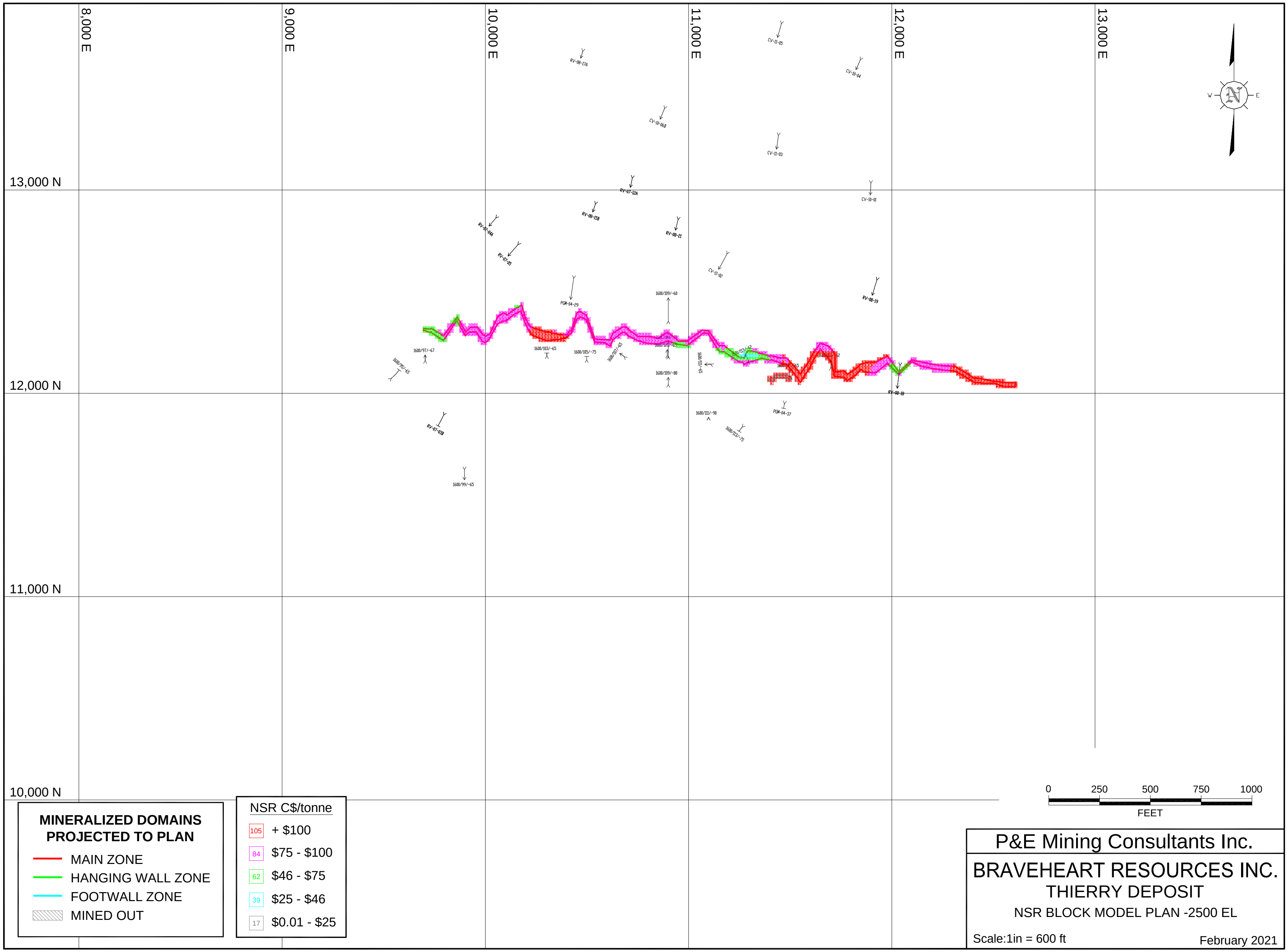
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL PLAN -2000 EL
Scale:1in = 600 ft
February 2021



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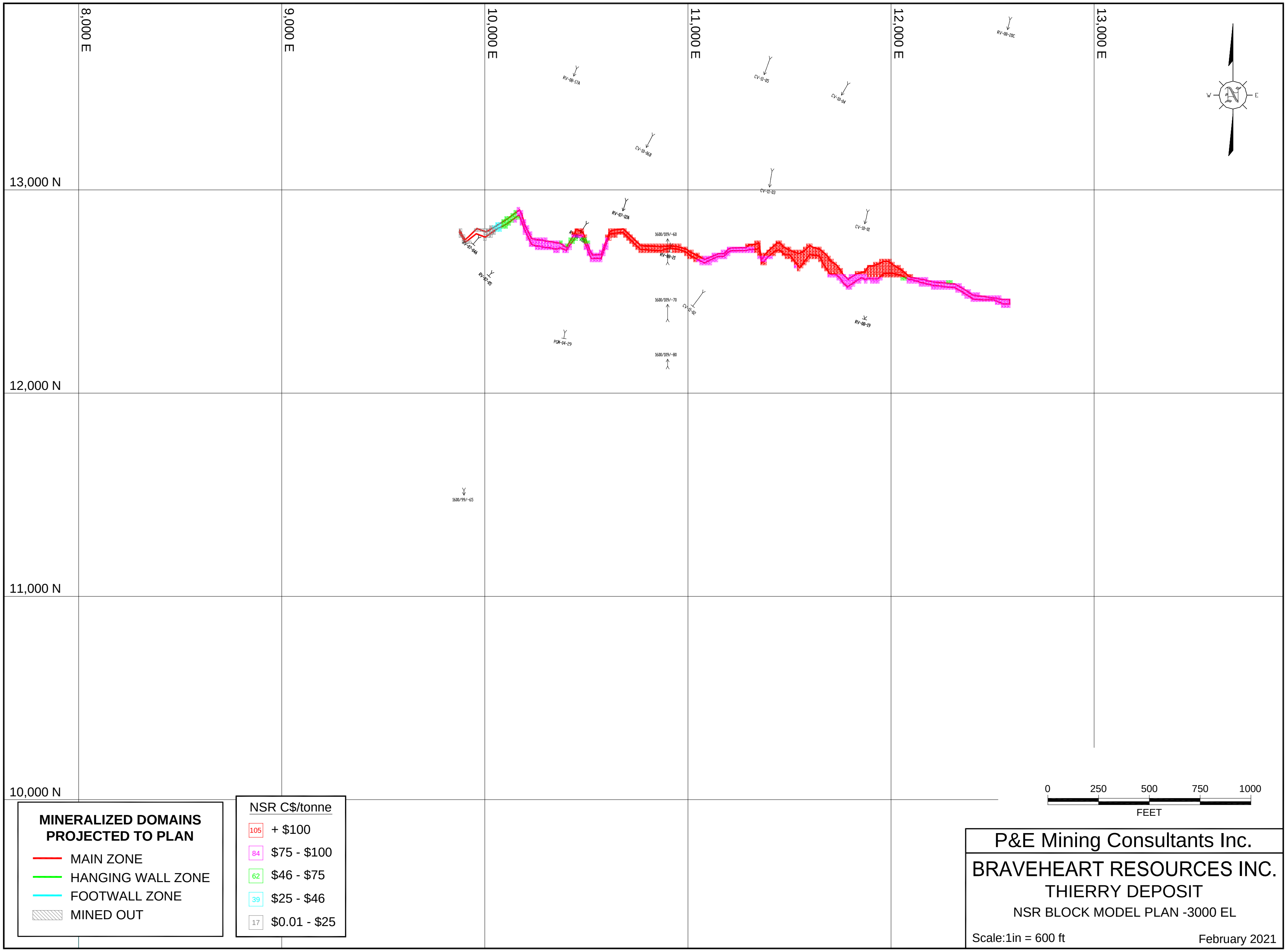
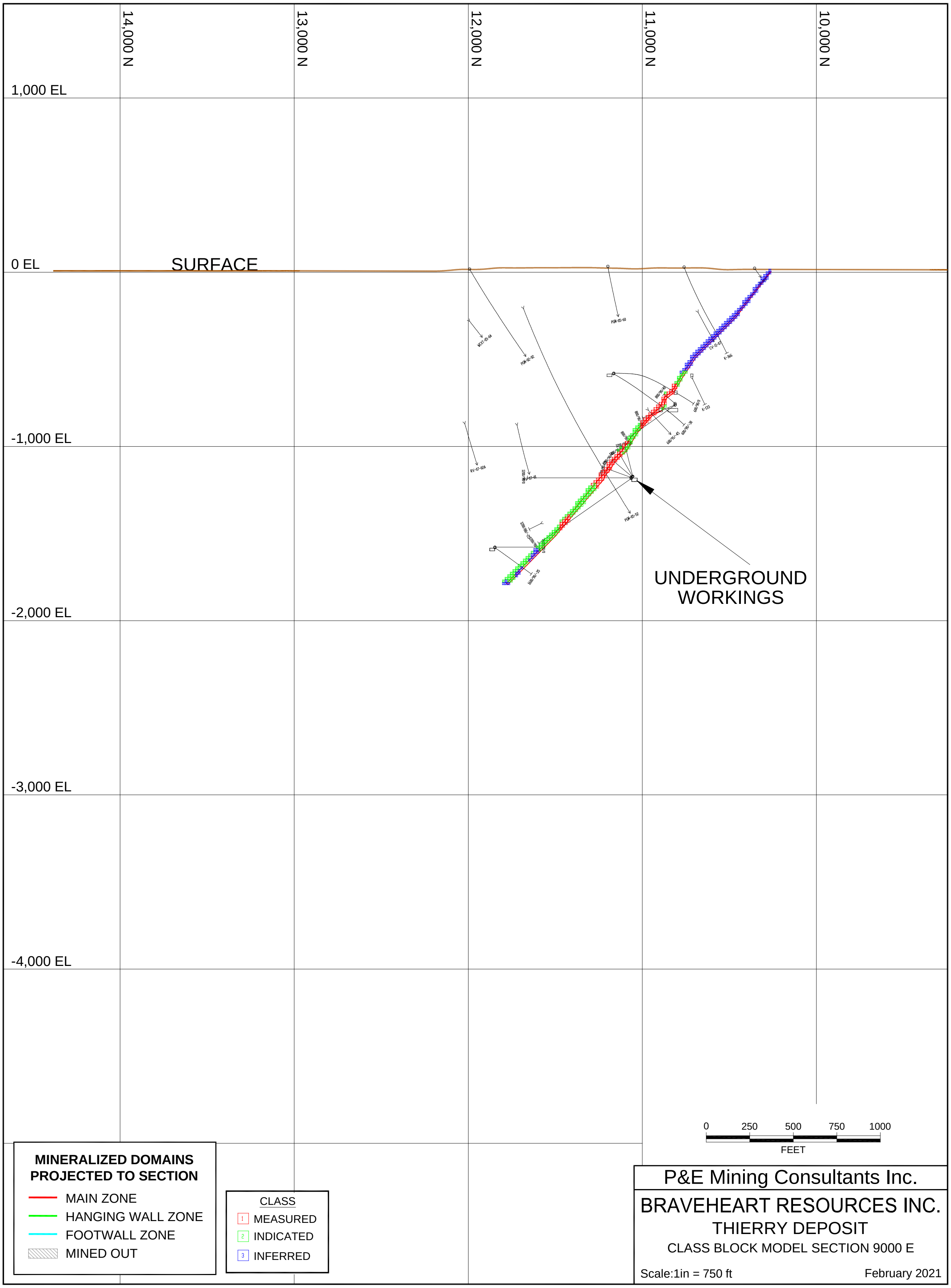
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
105 + $100
84 $75 - $100
62 $46 - $75
39 $25 - $46
17 $0.01 - $25
0 250 500 750 1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL PLAN -2500 EL
Scale:1in = 600 ft
February 2021



8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
NSR C$/tonne
+ $100
$75 - $100
$46 - $75
$25 - $46
$0.01 - $25
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
NSR BLOCK MODEL PLAN -3000 EL
Scale:1in = 600 ft
February 2021

APPENDIX G CLASSIFICATION BLOCK MODEL CROSS SECTIONS AND PLANS



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13,000 N
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
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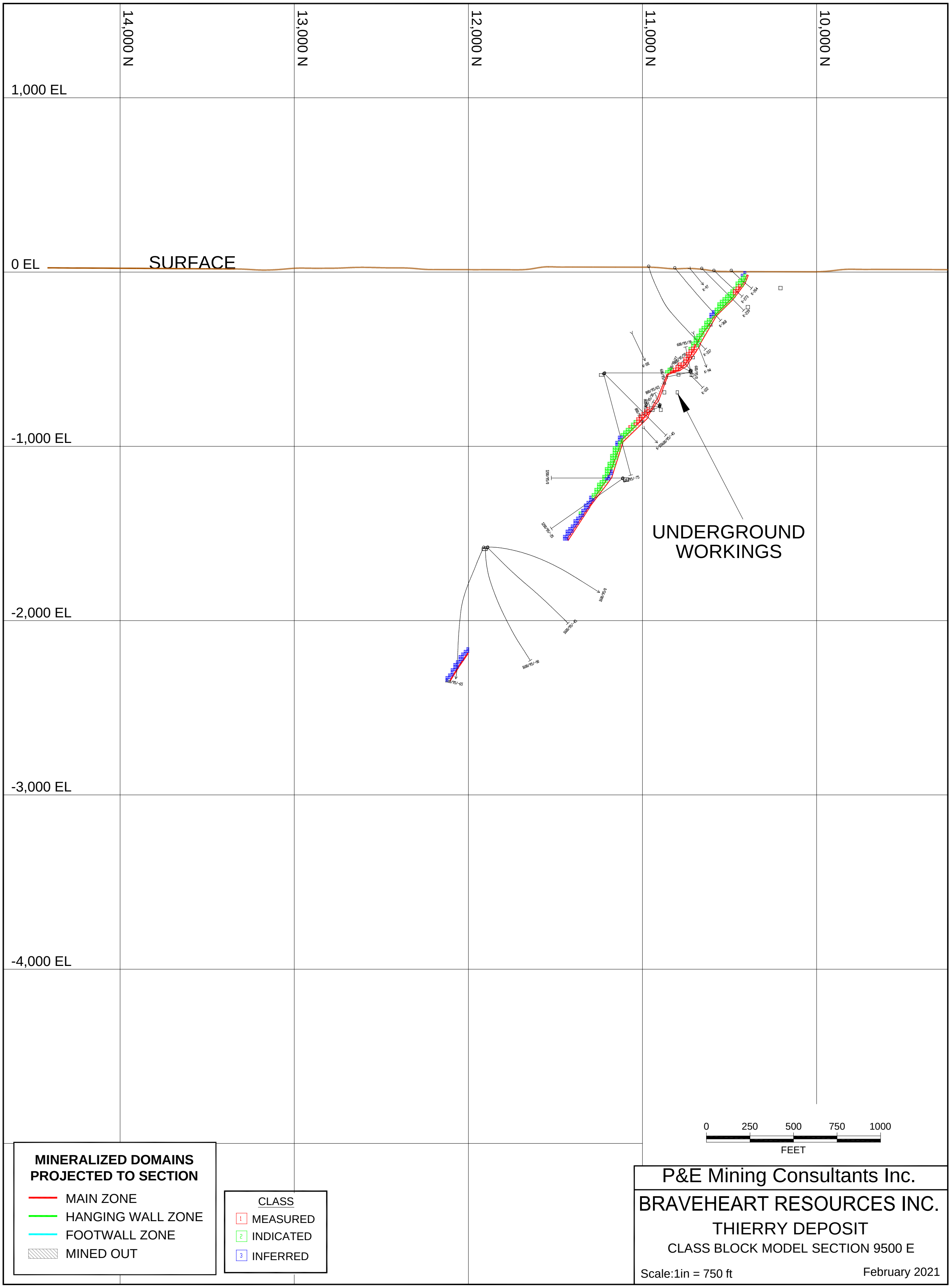
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 9000 E
Scale:1in = 750 ft
February 2021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10,000 N
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SURFACE
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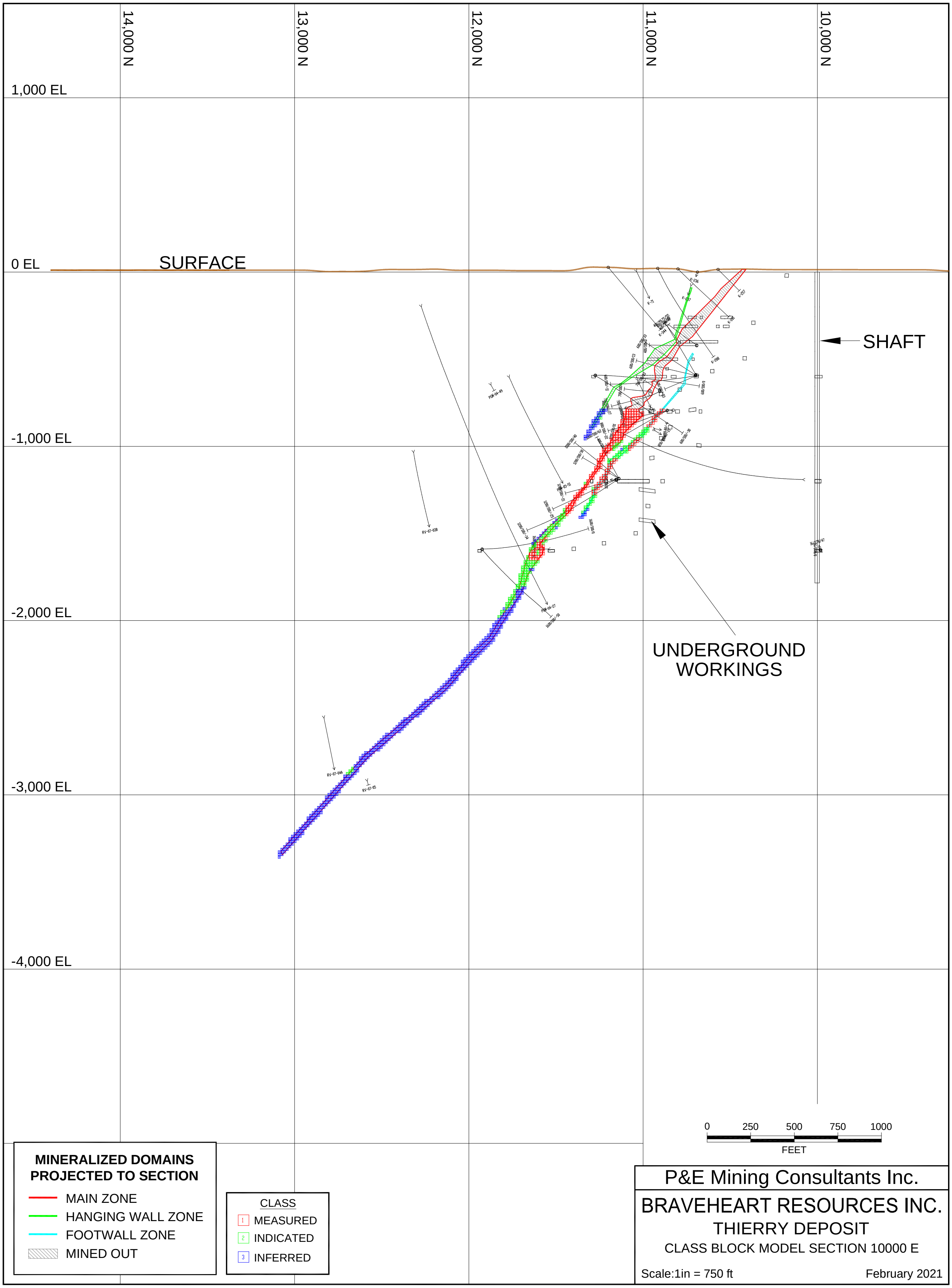
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 9500 E
Scale:1in = 750 ft
February 2021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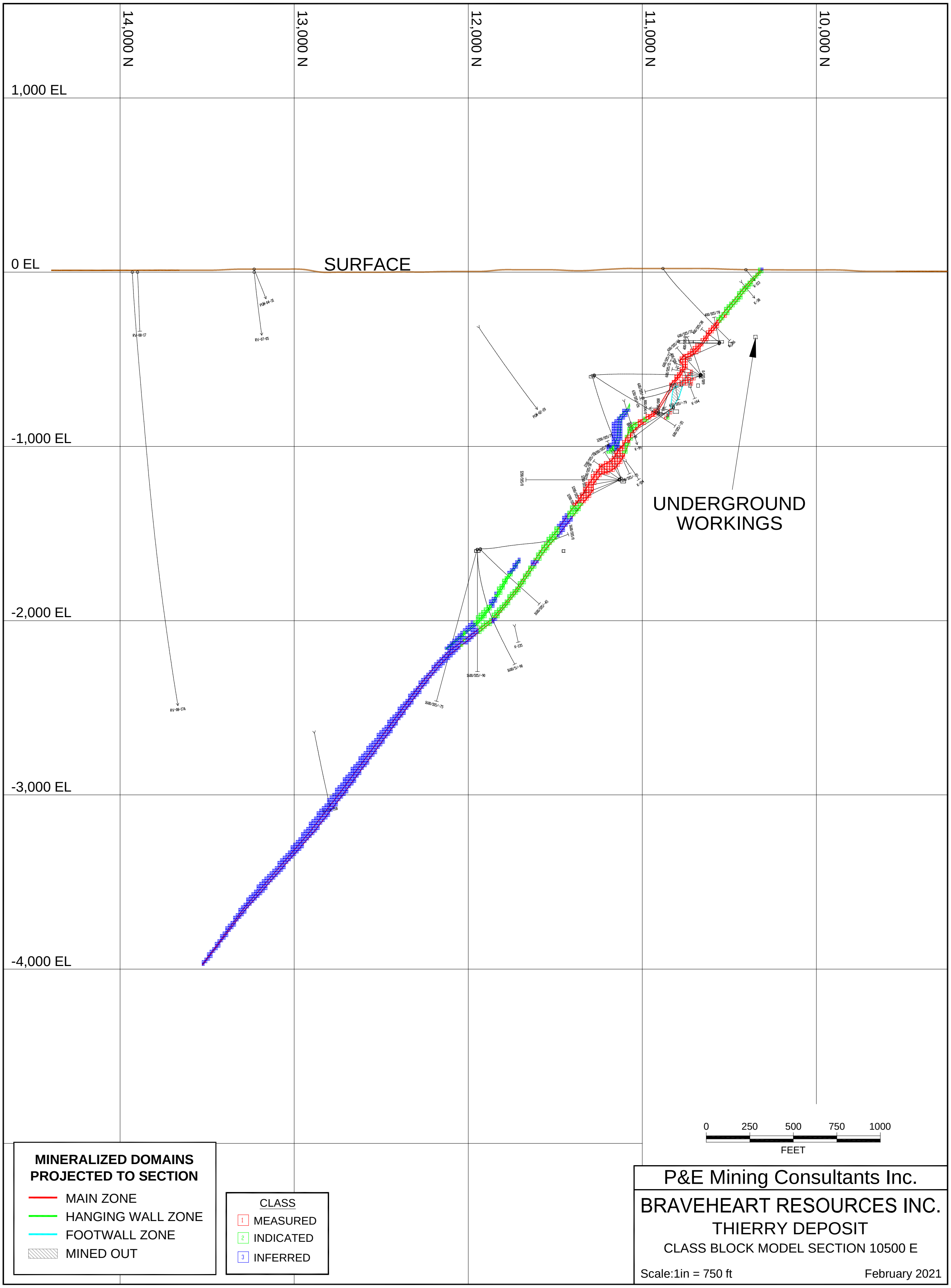
UNDERGROUND WORKINGS
-3,000 EL
-4,000 EL
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 10000 E
Scale:1in = 750 ft
February 2021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10,000 N
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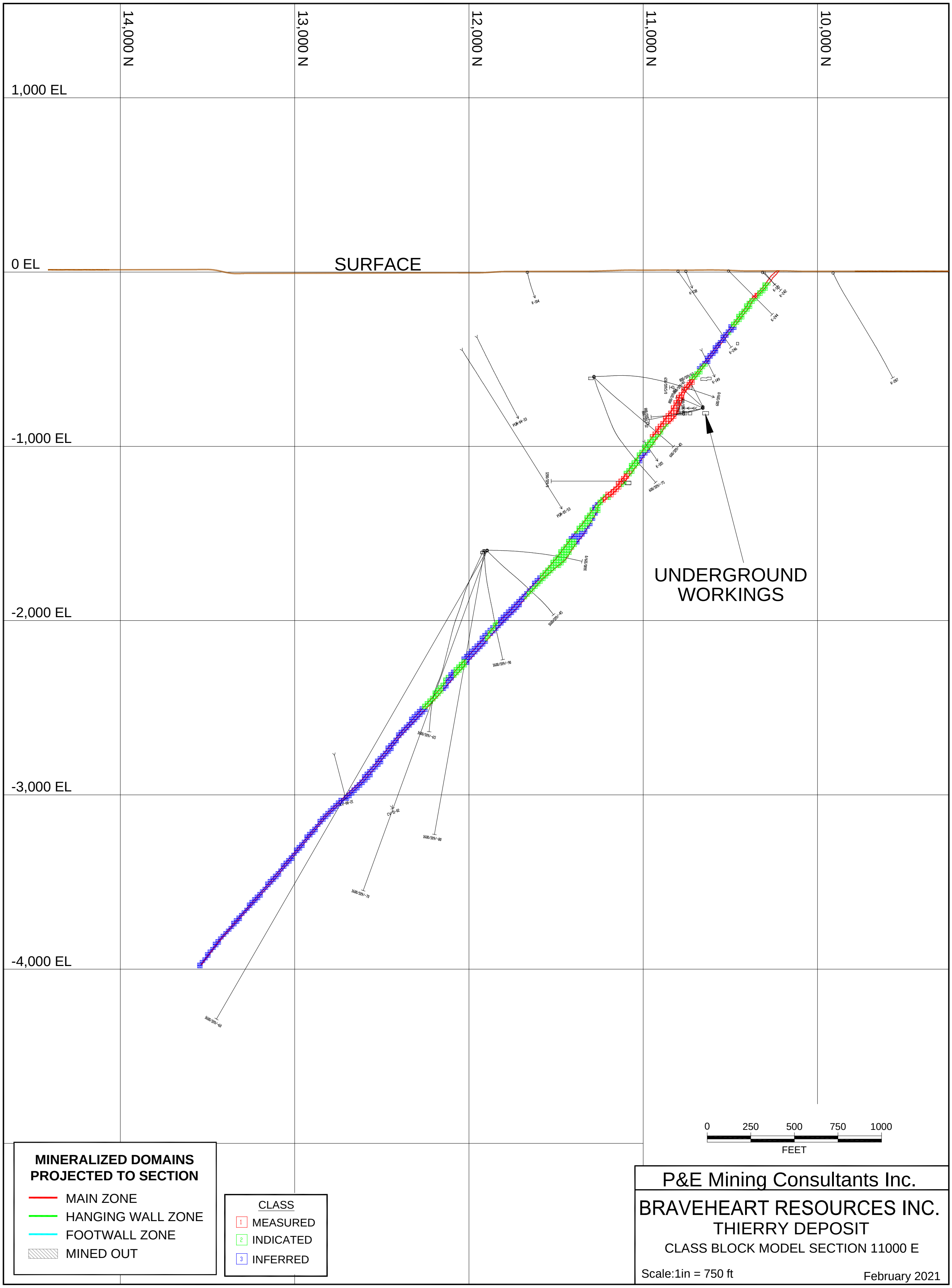
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND
WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS
PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 10500 E
Scale:1in = 750 ft
February 2021



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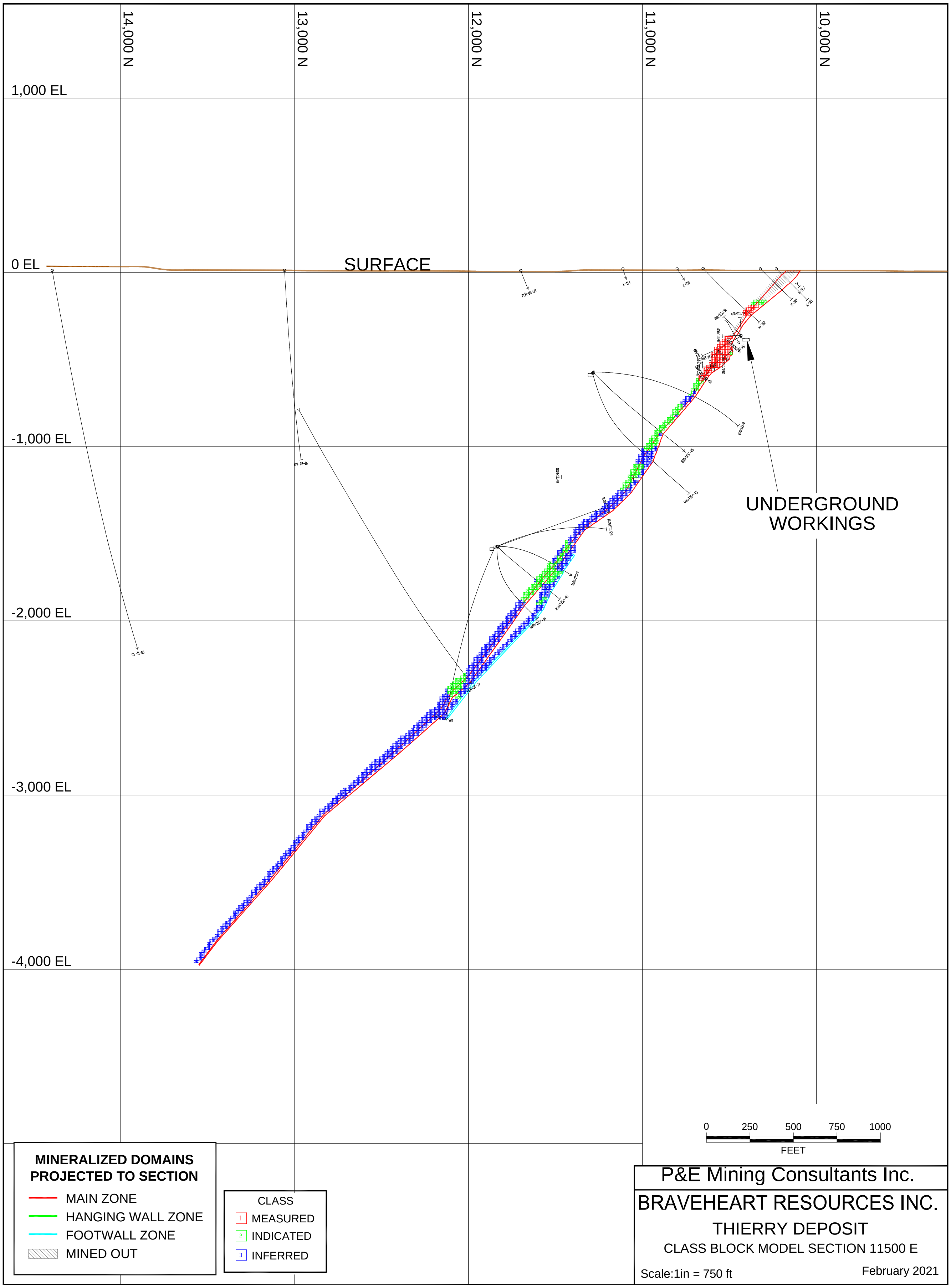
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 11000 E
Scale:1in = 750 ft
February 2021



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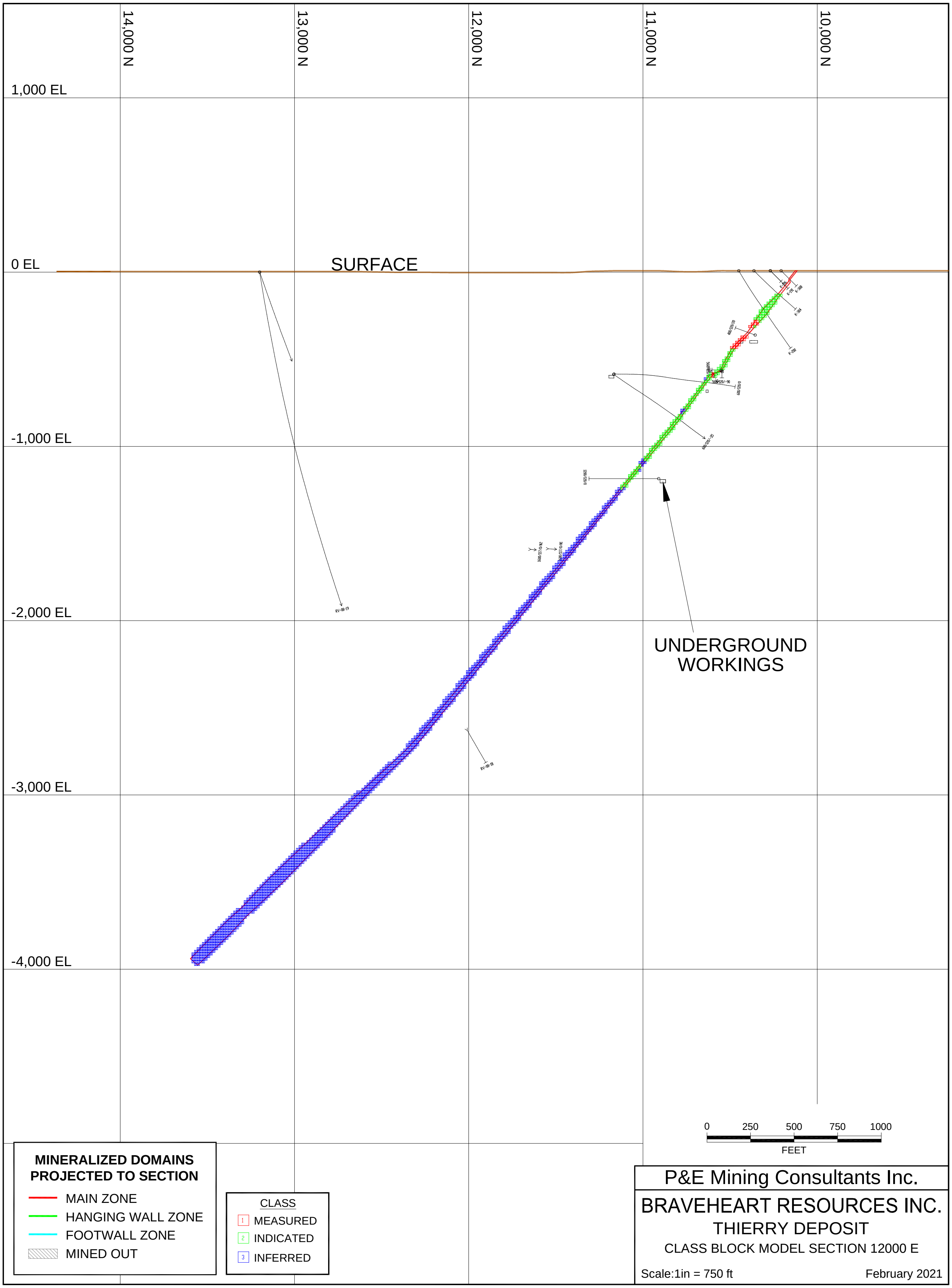
12,000 N
11,000 N
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 11500 E
Scale:1in = 750 ft
February 2021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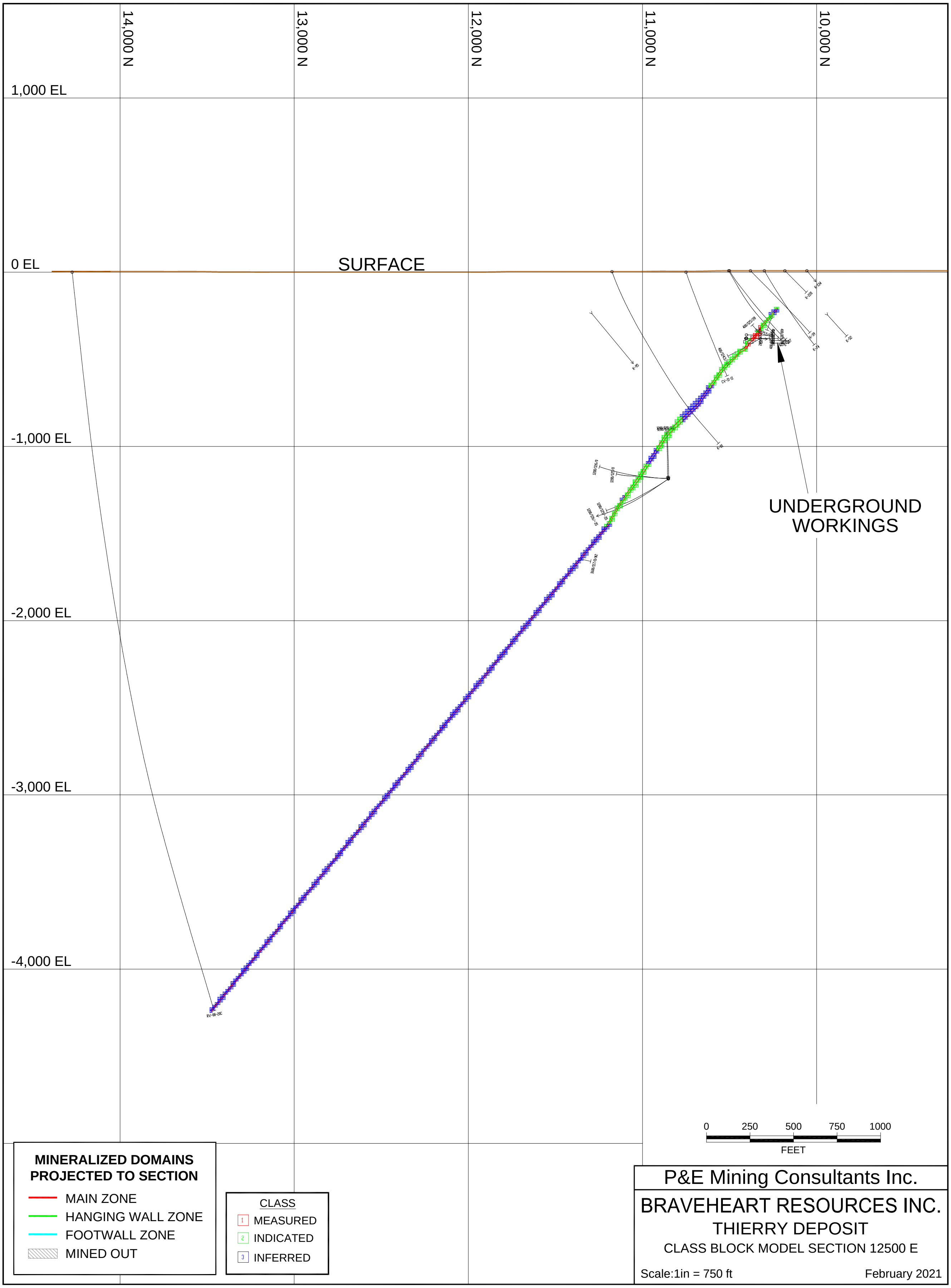
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 12000 E
Scale:1in = 750 ft
February 2021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12,000 N
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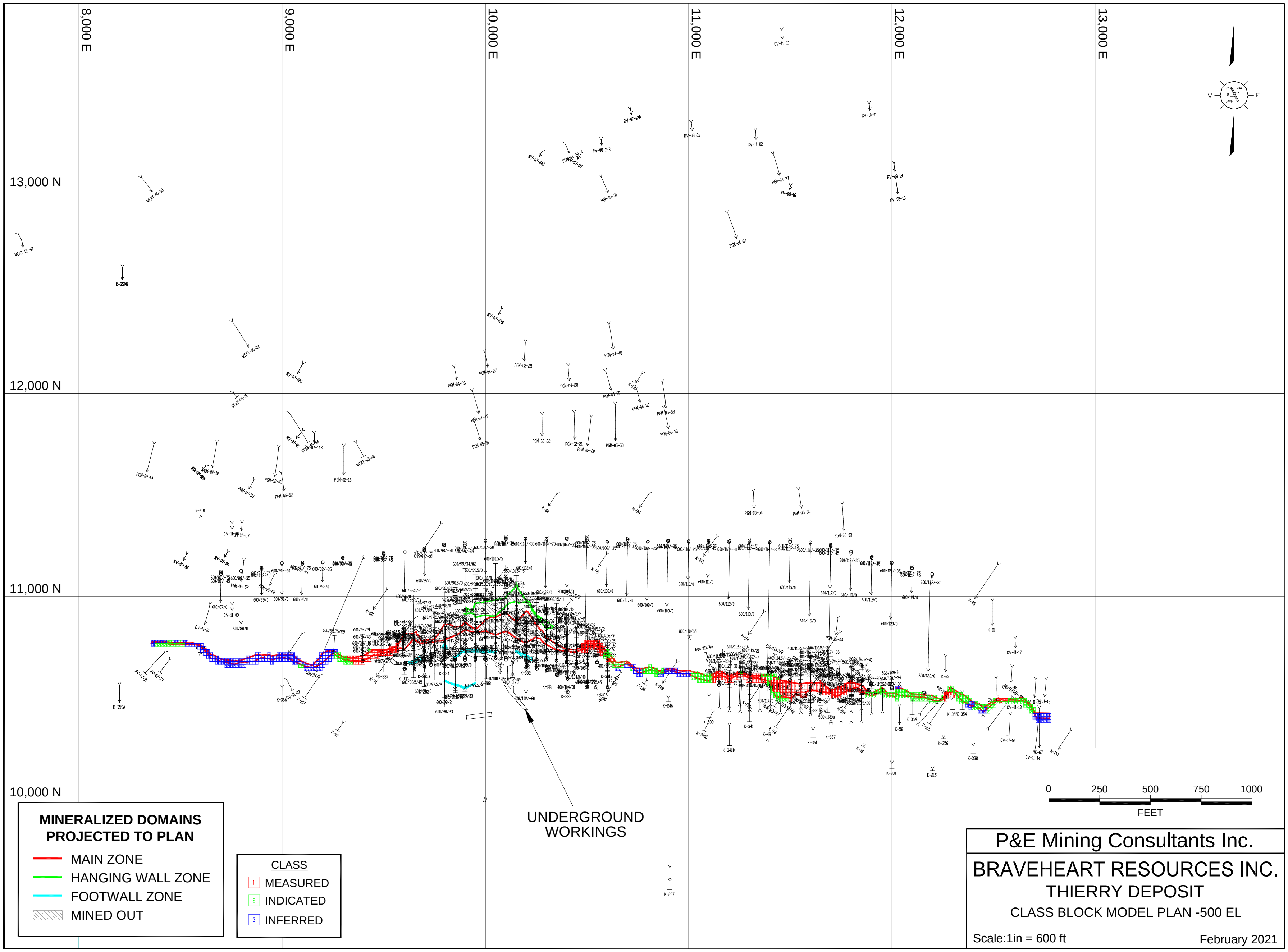
10,000 N
1,000 EL
0 EL
SURFACE
-1,000 EL
-2,000 EL
-3,000 EL
-4,000 EL
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO SECTION
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL SECTION 12500 E
Scale:1in = 750 ft
February 2021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9,000 E
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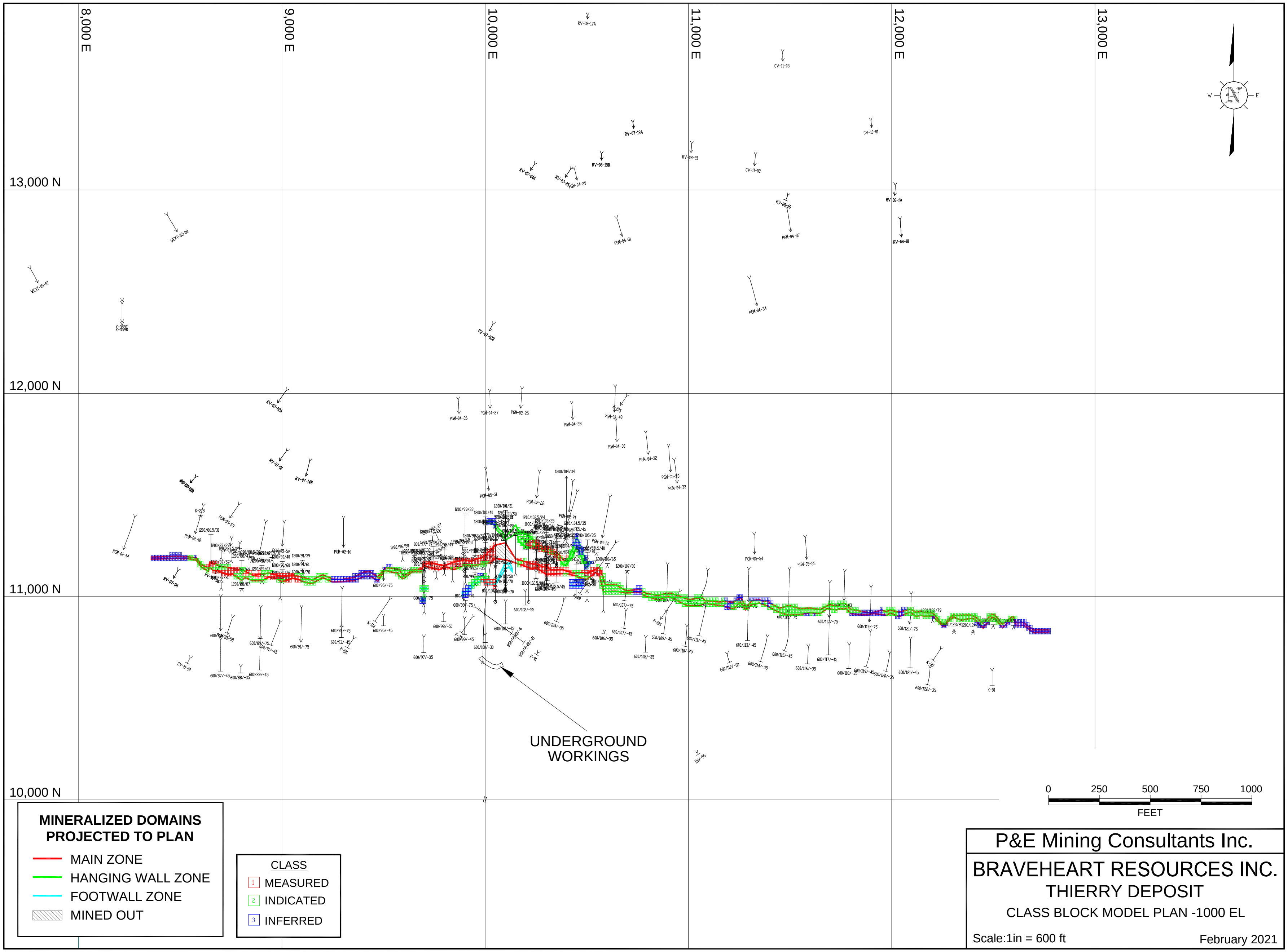
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND WORKINGS
0 250 500 750 1000
FEET
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL PLAN -500 EL
Scale:1in = 600 ft
February 2021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12,000 E
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12,000 N
11,000 N
10,000 N
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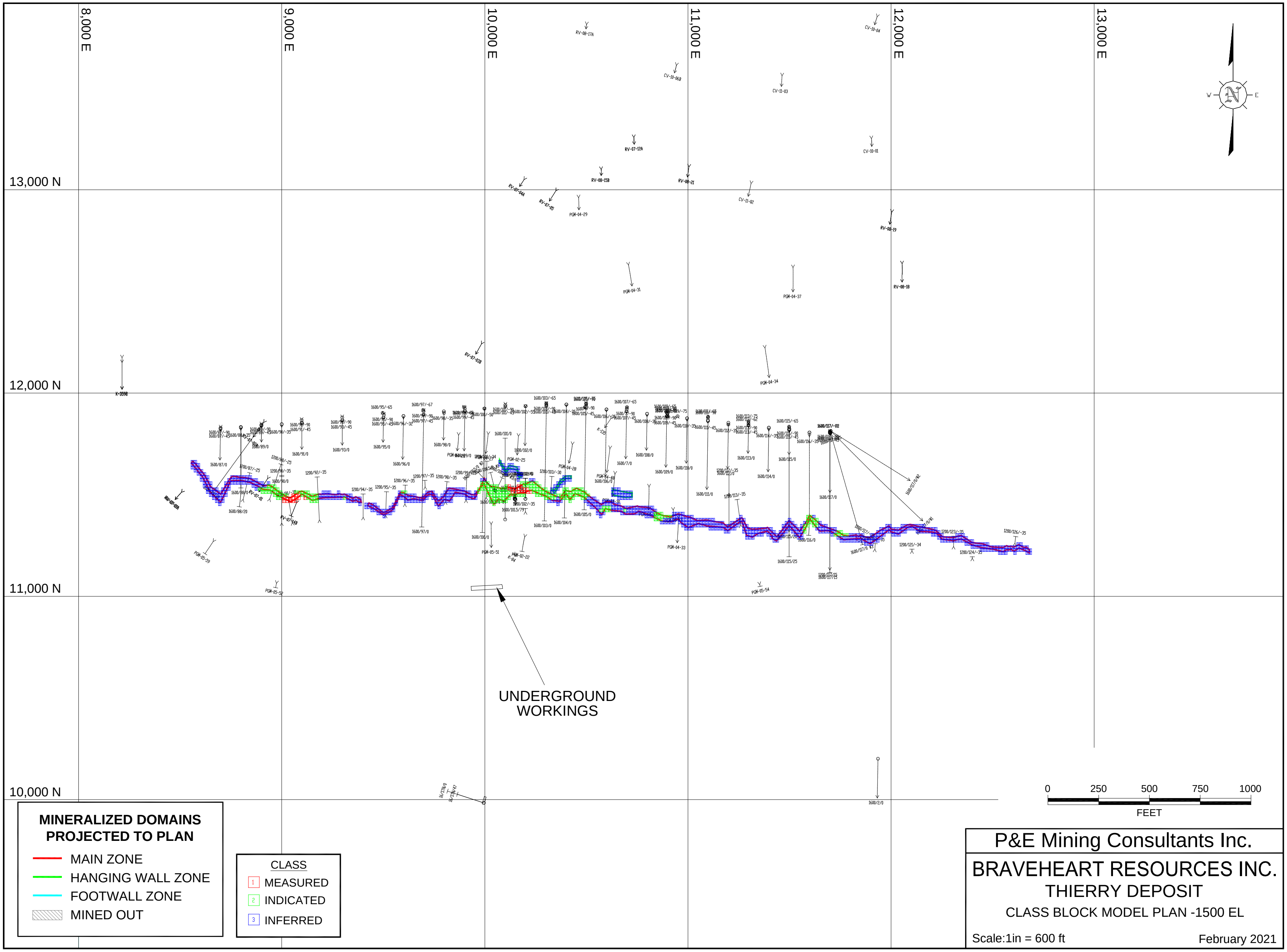
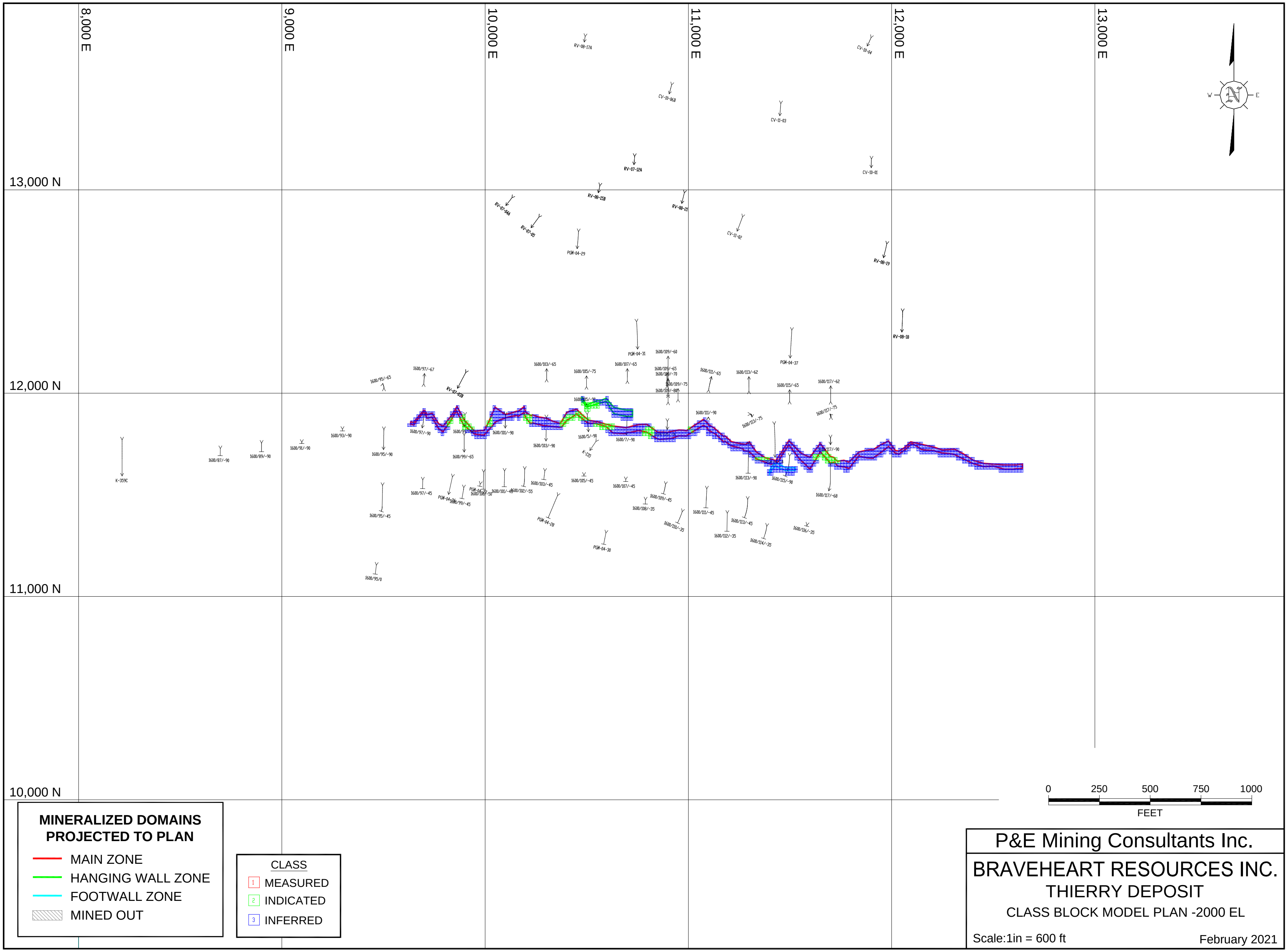
WORKINGS
0
250
500
750
1000
FEET
MINERALIZED DOMAINS
PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL PLAN -1000 EL
Scale:1in = 600 ft
February 2021



8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
UNDERGROUND WORKINGS
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL PLAN -1500 EL
Scale:1in = 600 ft
February 2021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11,000 E
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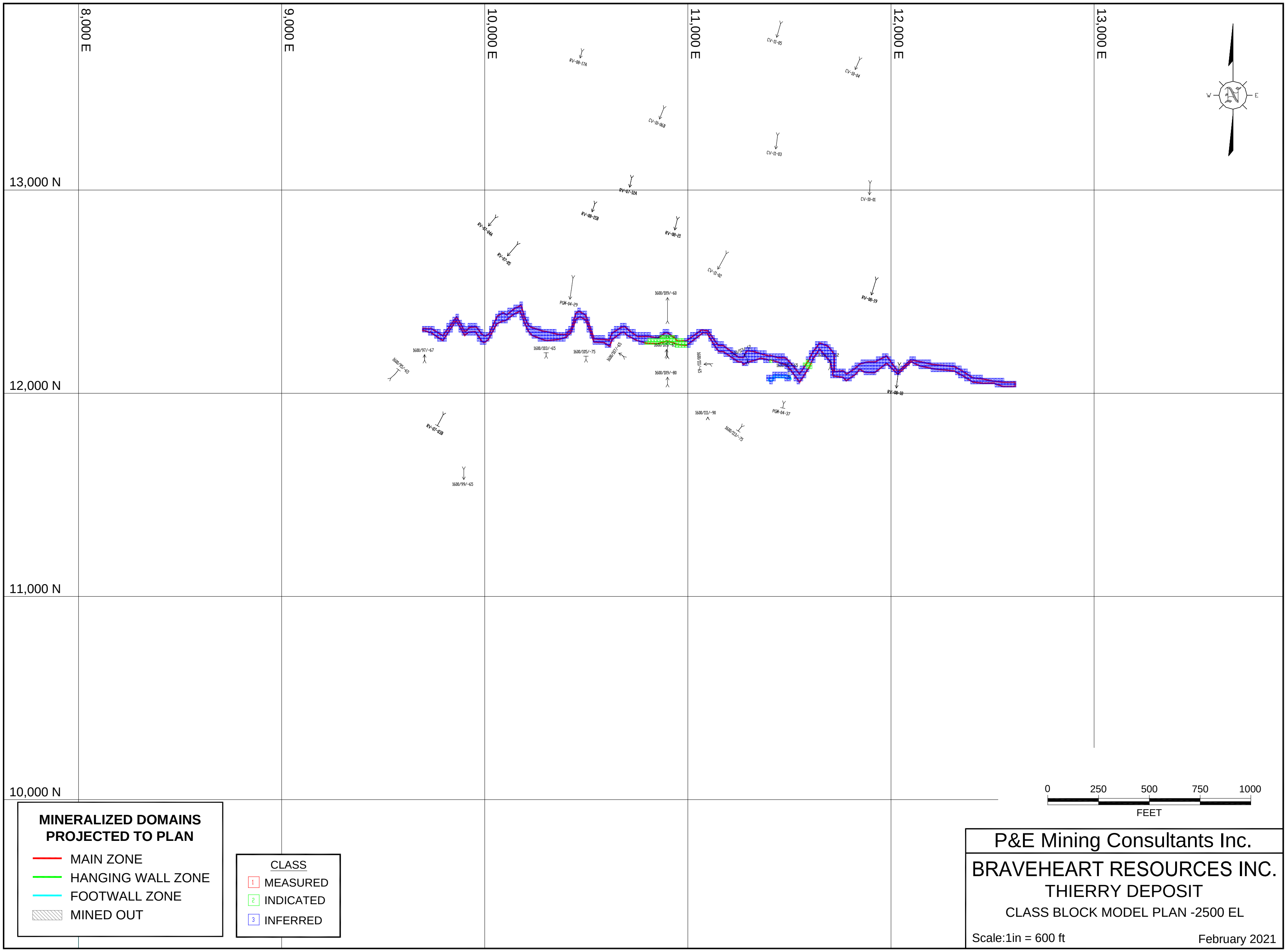
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
0
250
500
750
1000
FEET
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL PLAN -2000 EL
Scale:1in = 600 ft
February 2021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13,000 N
12,000 N
11,000 N
10,000 N
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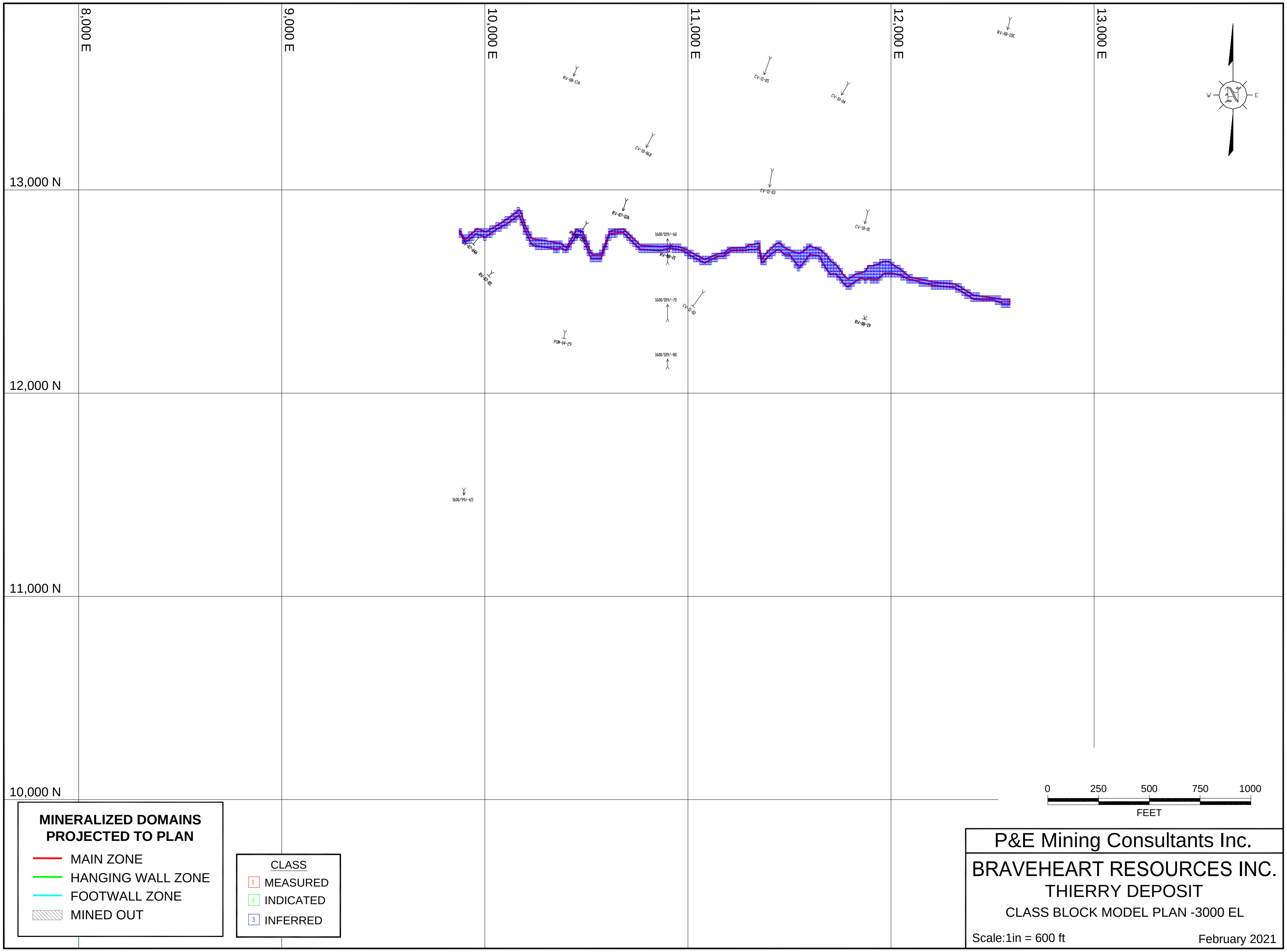
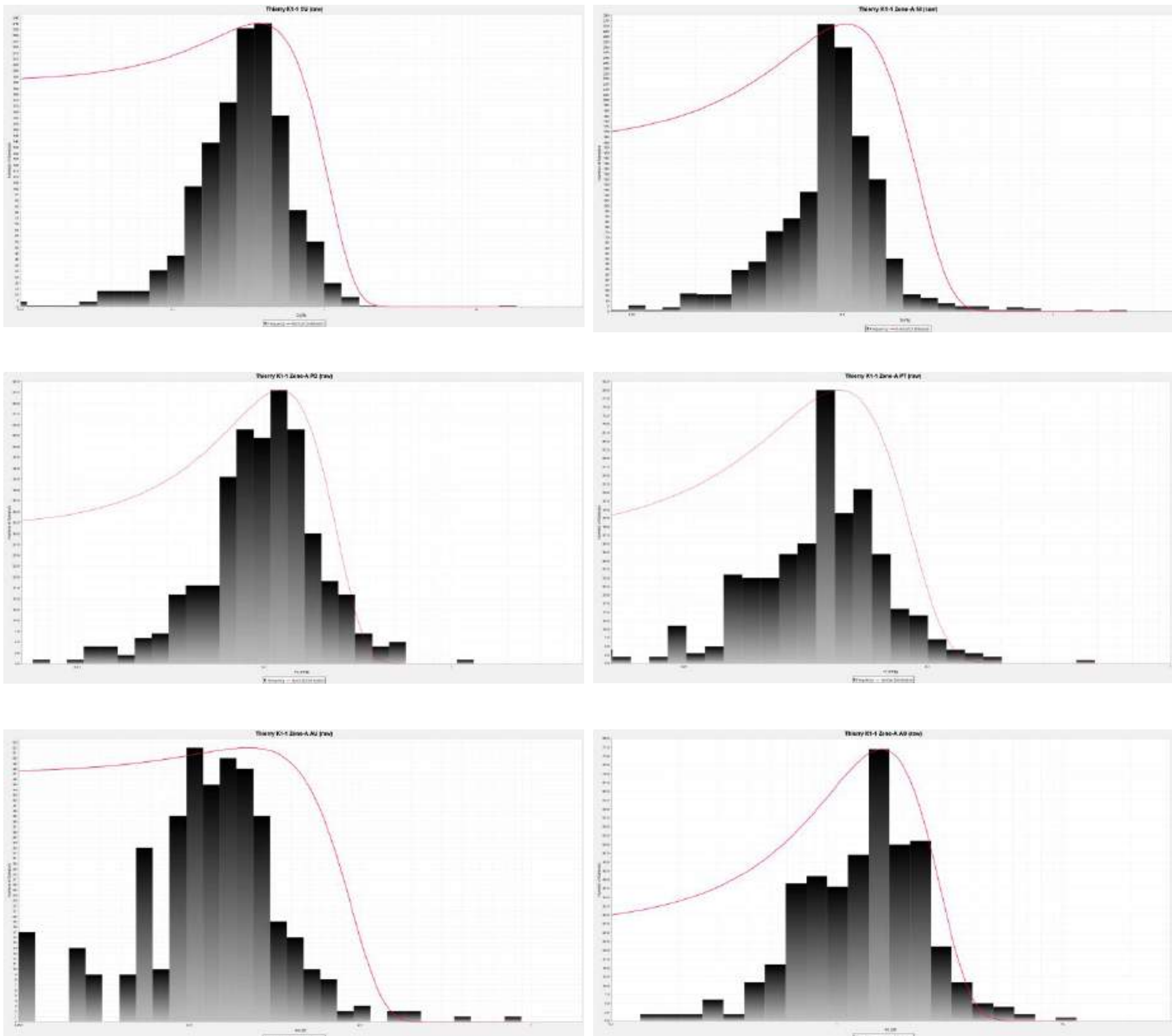
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
0
250
500
750
1000
FEET
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL PLAN -2500 EL
Scale:1in = 600 ft
February 2021



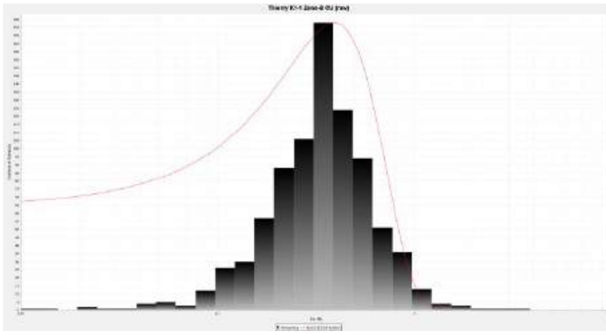
8,000 E
9,000 E
10,000 E
11,000 E
12,000 E
13,000 E
13,000 N
12,000 N
11,000 N
10,000 N
0
250
500
750
1000
FEET
MINERALIZED DOMAINS PROJECTED TO PLAN
MAIN ZONE
HANGING WALL ZONE
FOOTWALL ZONE
MINED OUT
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
BRAVEHEART RESOURCES INC.
THIERRY DEPOSIT
CLASS BLOCK MODEL PLAN -3000 EL
Scale:1in = 600 ft
February 2021

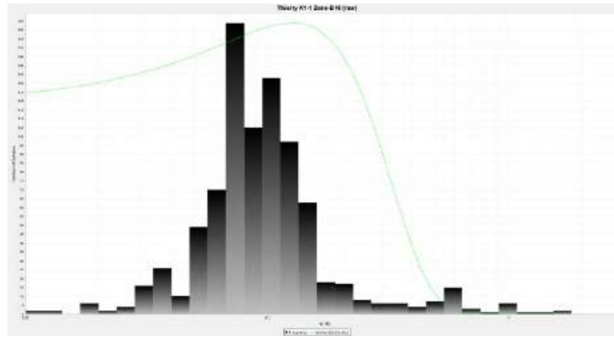
APPENDIX H HISTOGRAMS BY METAL FOR K1-1 DOMAIN ASSAYS

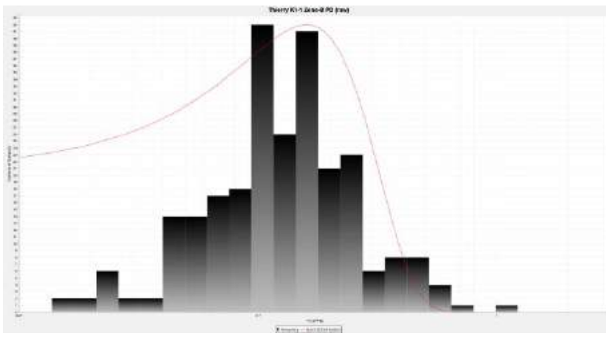
Zone A: Cu, Ni, Pd, Pt, Au, Ag

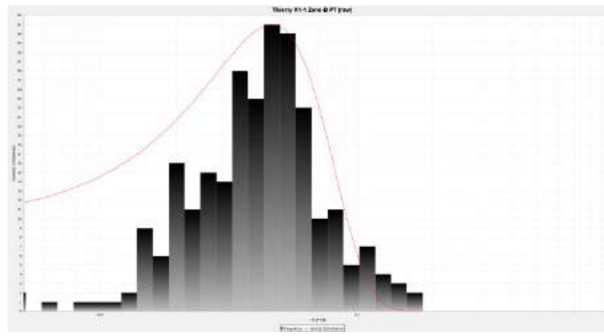


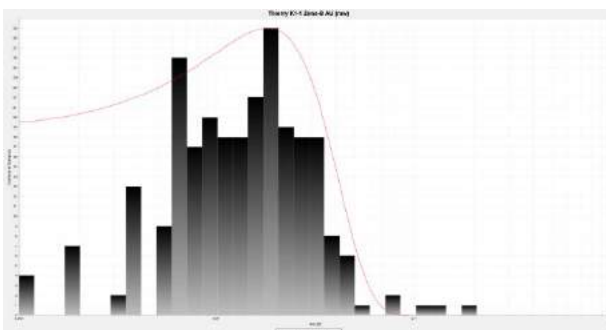
Zone B: Cu, Ni, Pd, Pt, Au, Ag

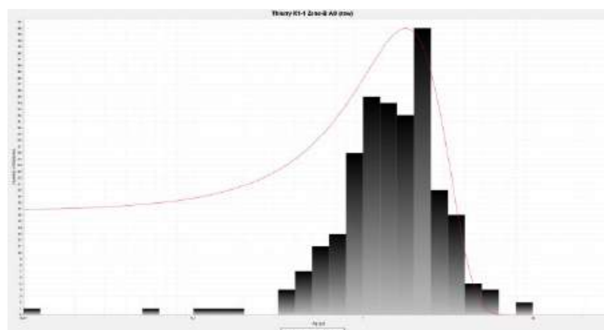




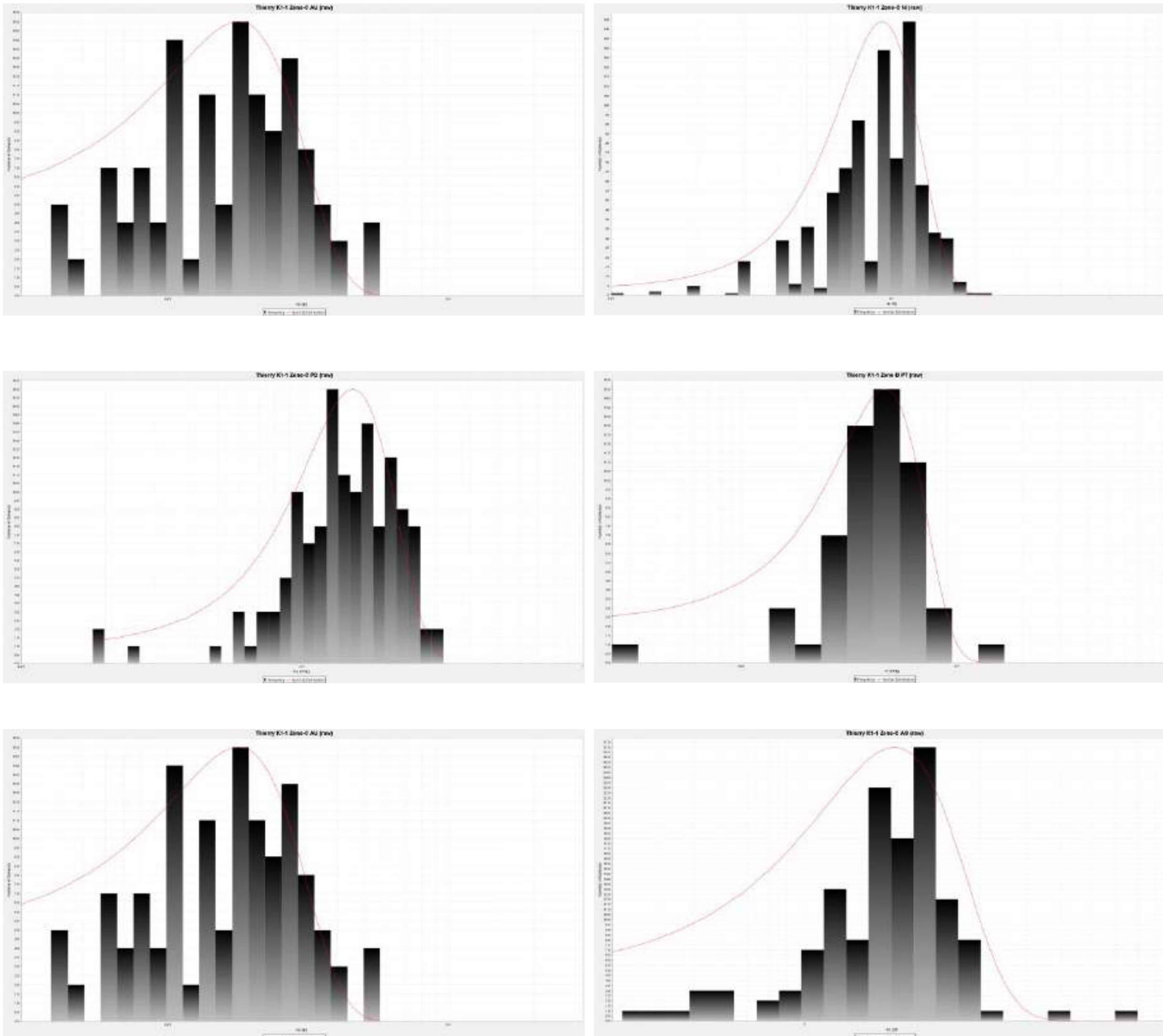




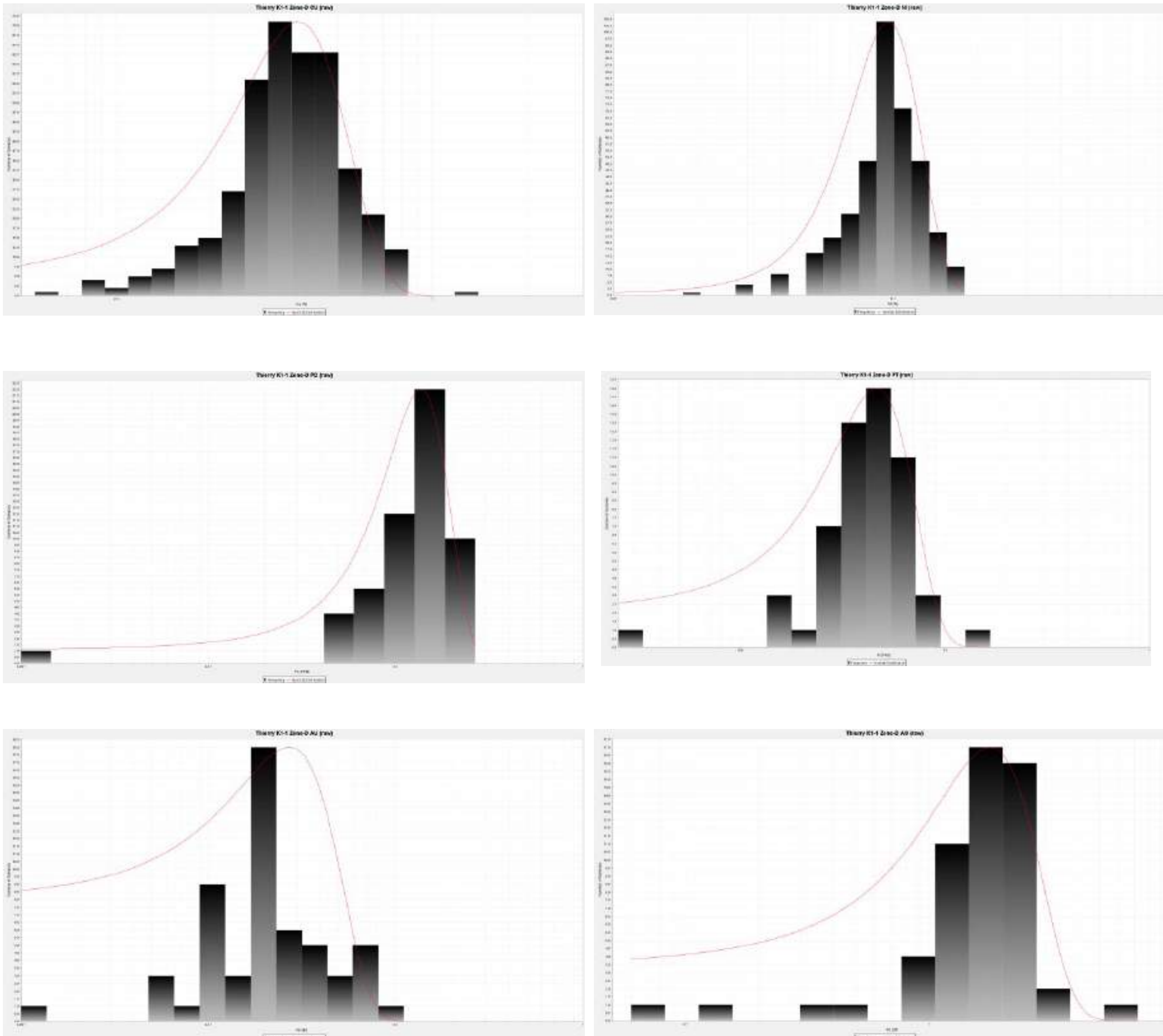




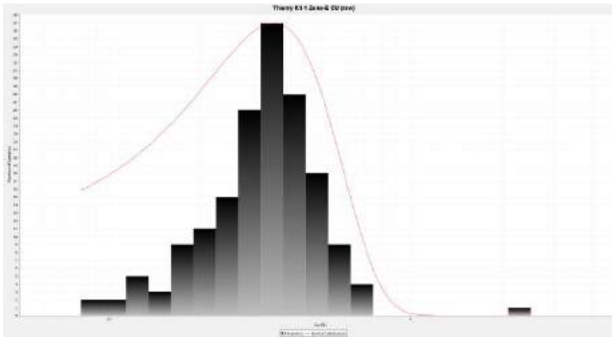
Zone C: Cu, Ni, Pd, Pt, Au, Ag

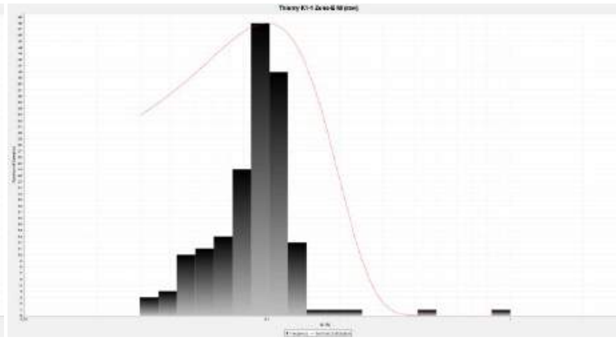


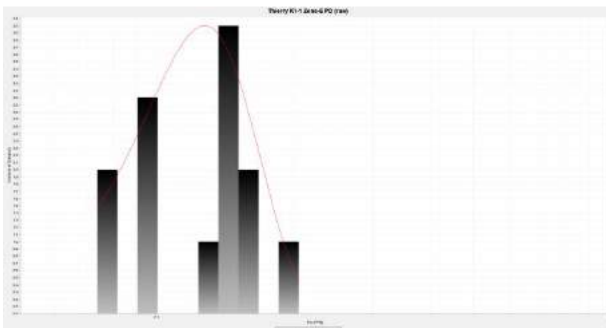
Zone D: Cu, Ni, Pd, Pt, Au, Ag

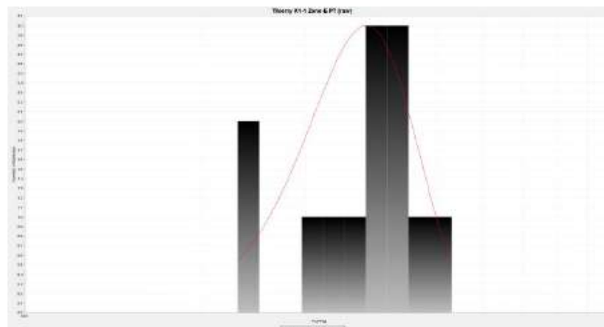


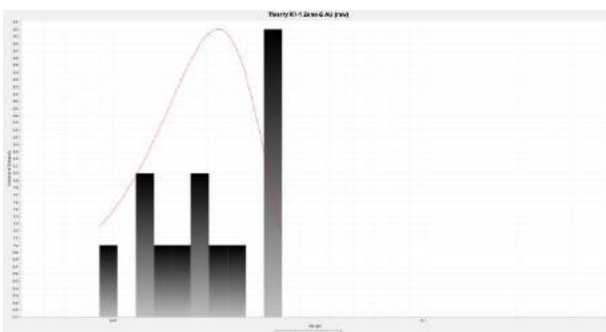
Zone E: Cu, Ni, Pd, Pt, Au, Ag

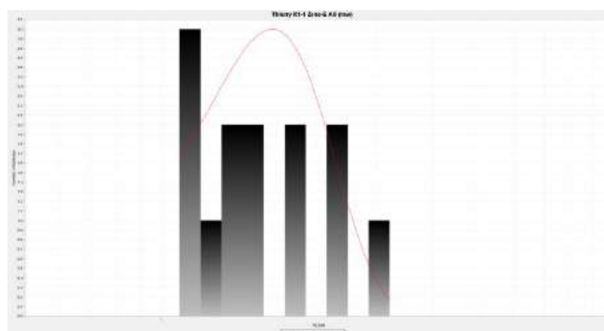




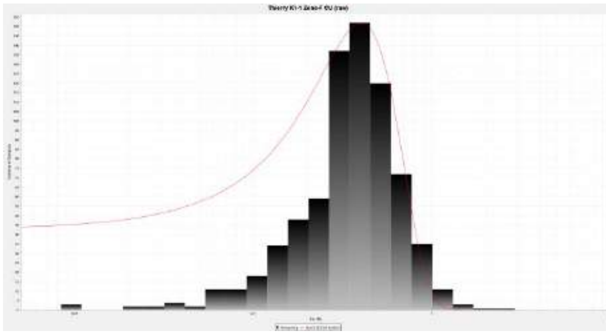


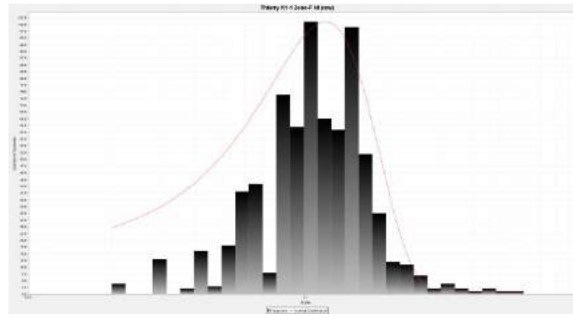


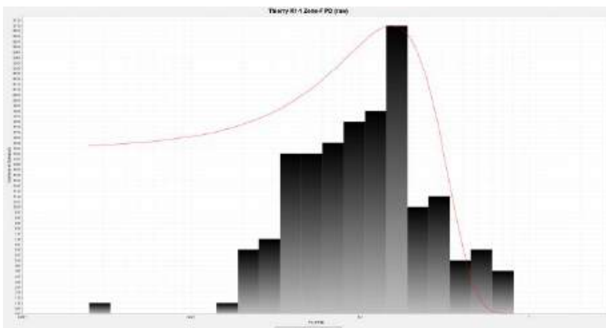


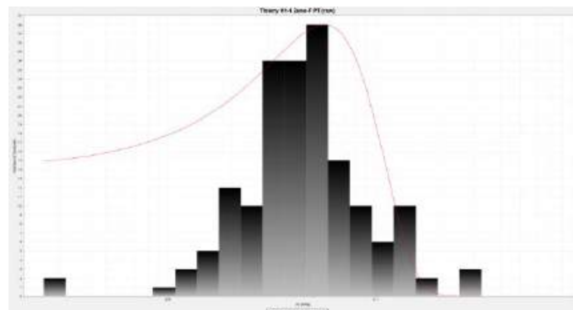


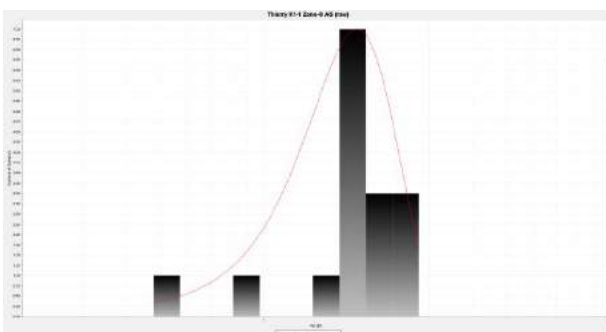
Zone F: Cu, Ni, Pd, Pt, Au, Ag

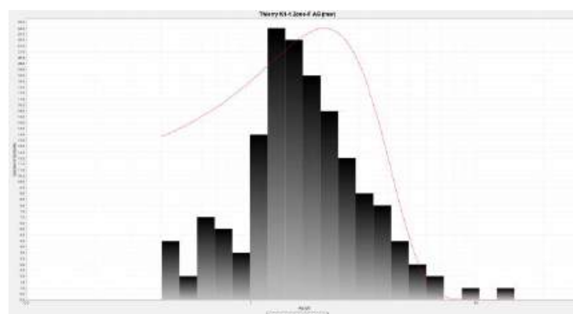




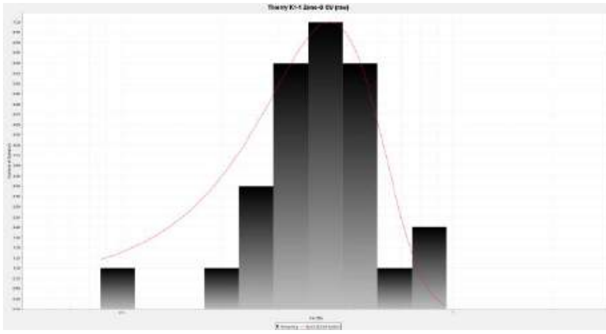


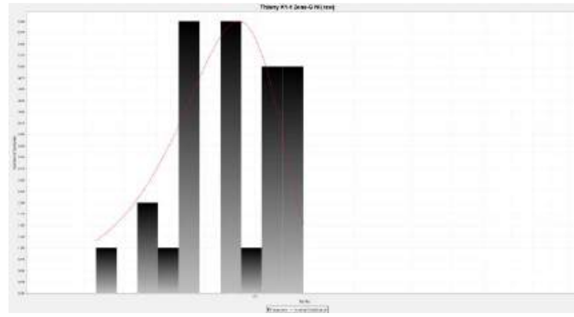


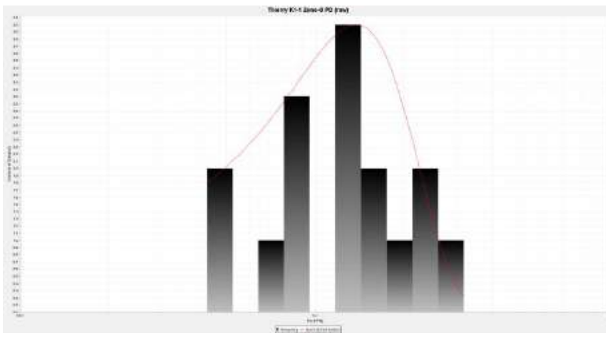


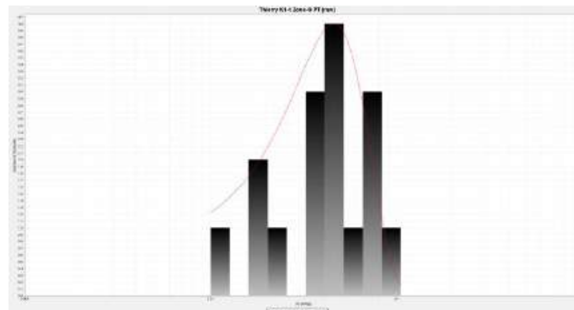


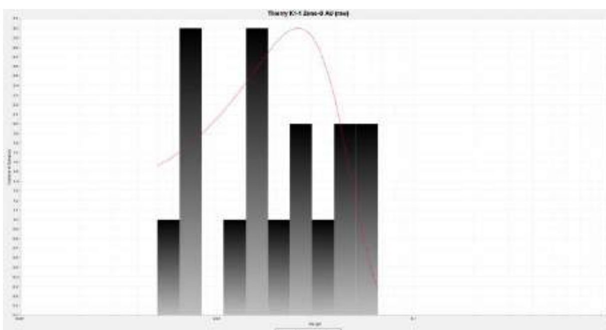
Zone G: Cu, Ni, Pd, Pt, Au, Ag

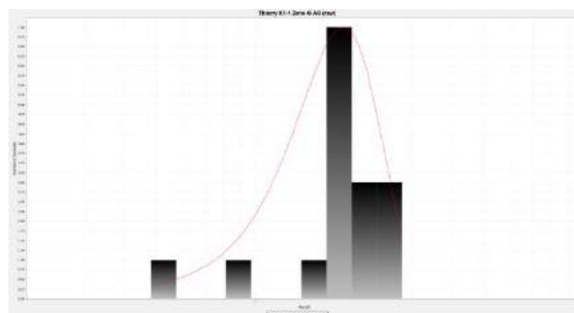








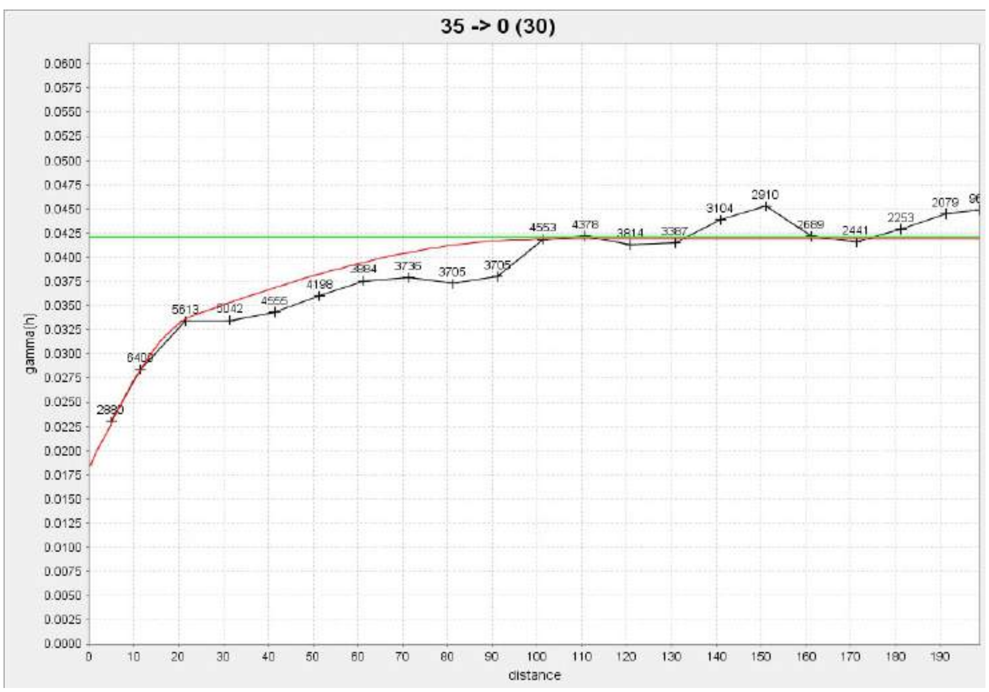




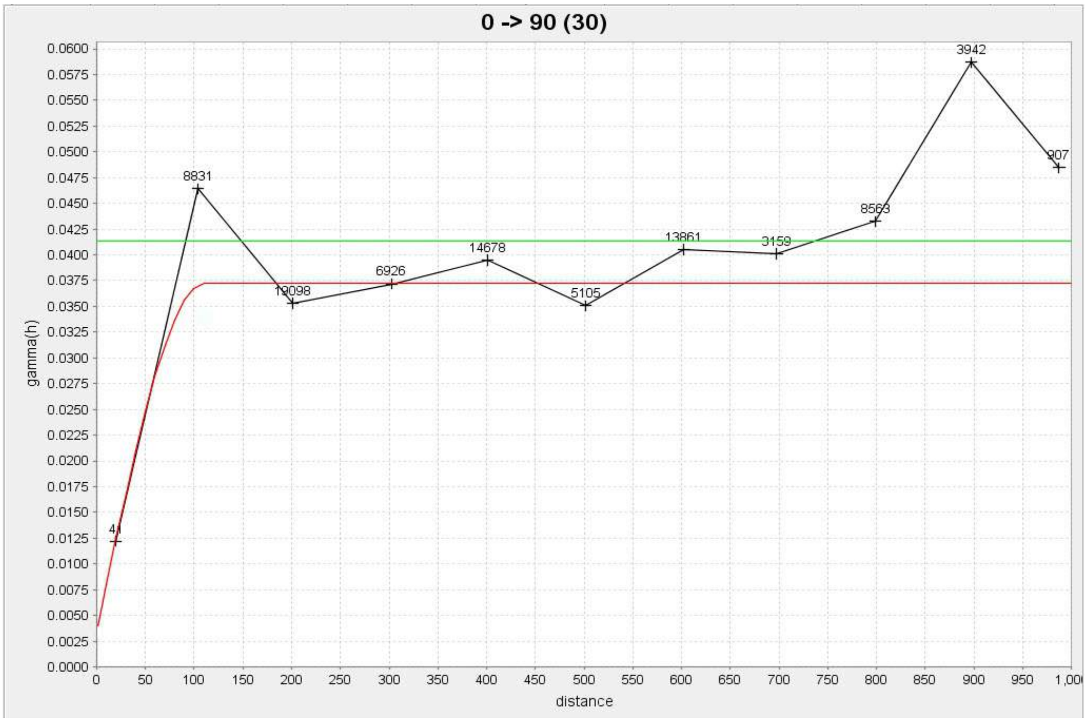
APPENDIX I K1-1 VARIOGRAPHY CU COMPOSITES 5 FT (1.5 M)

Down Hole Linear Variogram (Nested Spherical Model; 1.5 m Lags for Nugget Effect)

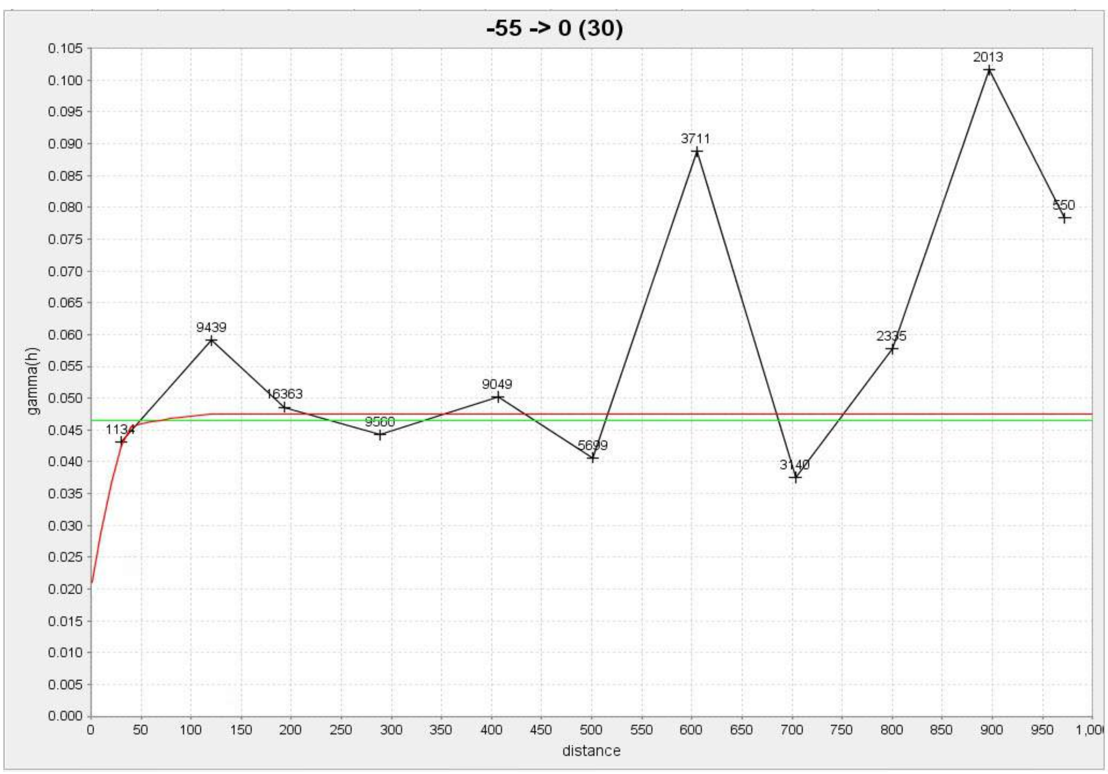
3-D Variogram 360°/+35° (Thickness; Nested Spherical Model ; 3 m lags; 30° Spread Angle)



3-D Variogram on Strike 090°/0° (Spherical Model; 100 m Lags; 30° Spread Angle)

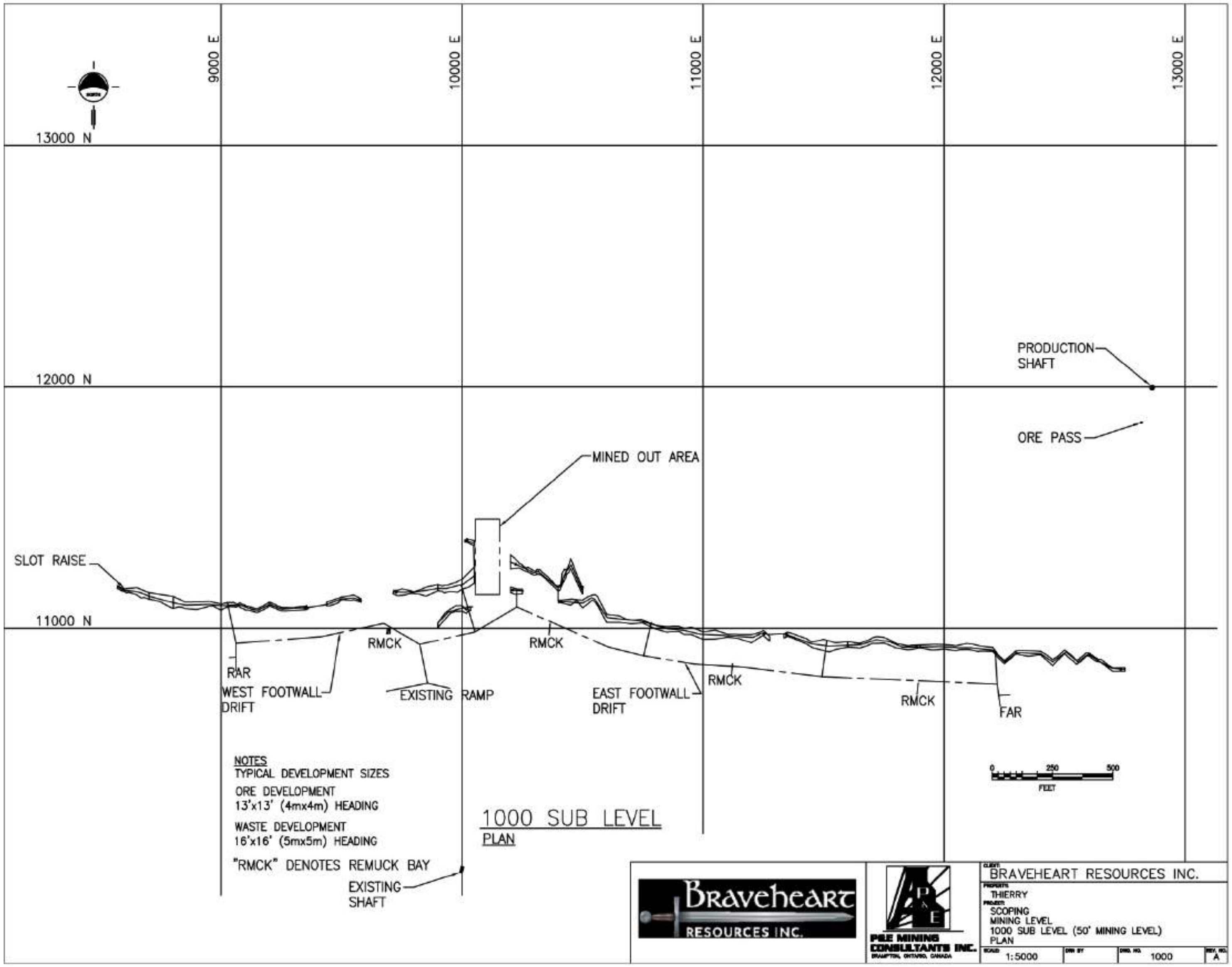


3-D Variogram on Dip 360°/-55° (Nested Spherical Model; 100 m Lags; 30° Spread Angle)

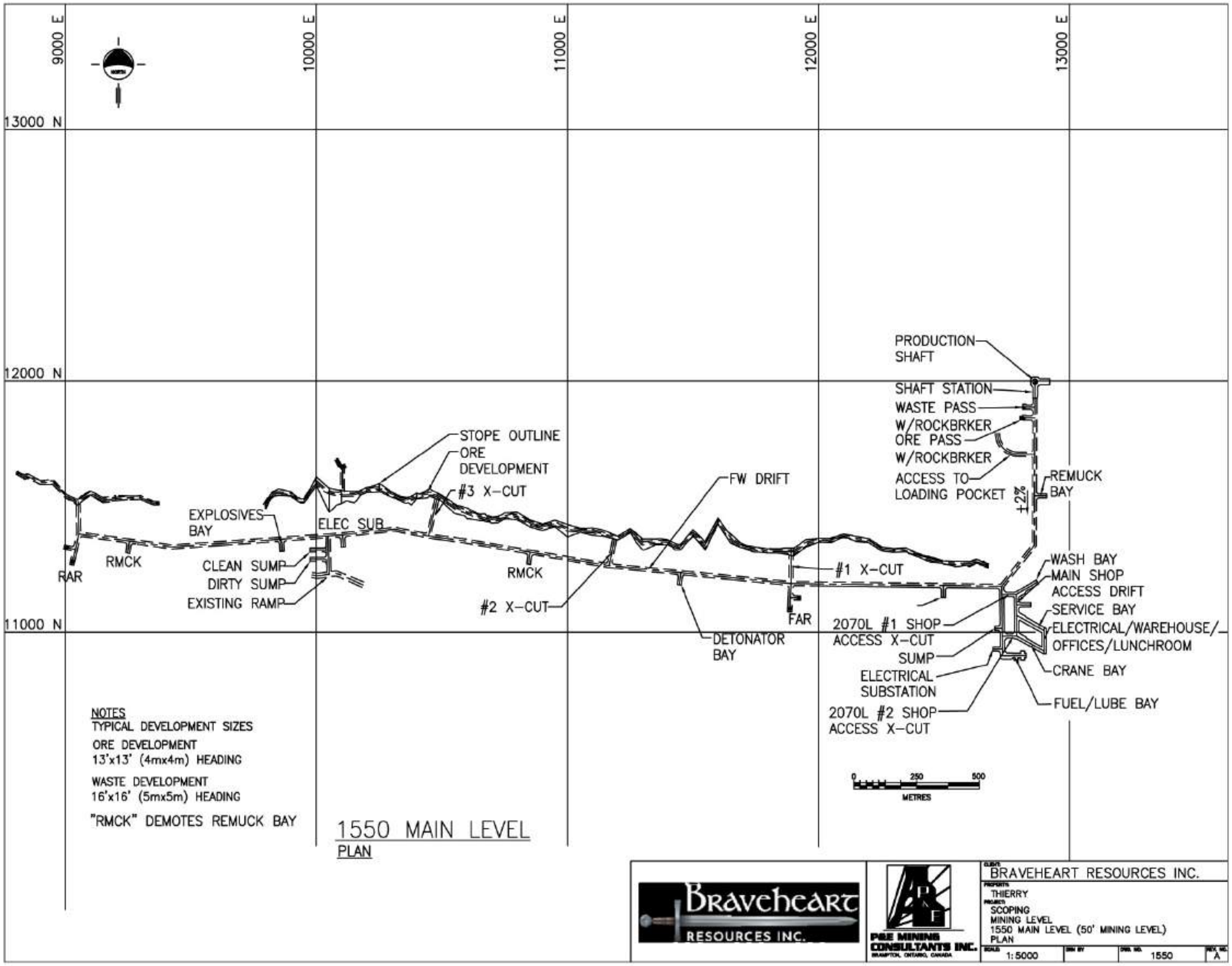


APPENDIX J THIERRY MINE PLAN DRAWINGS

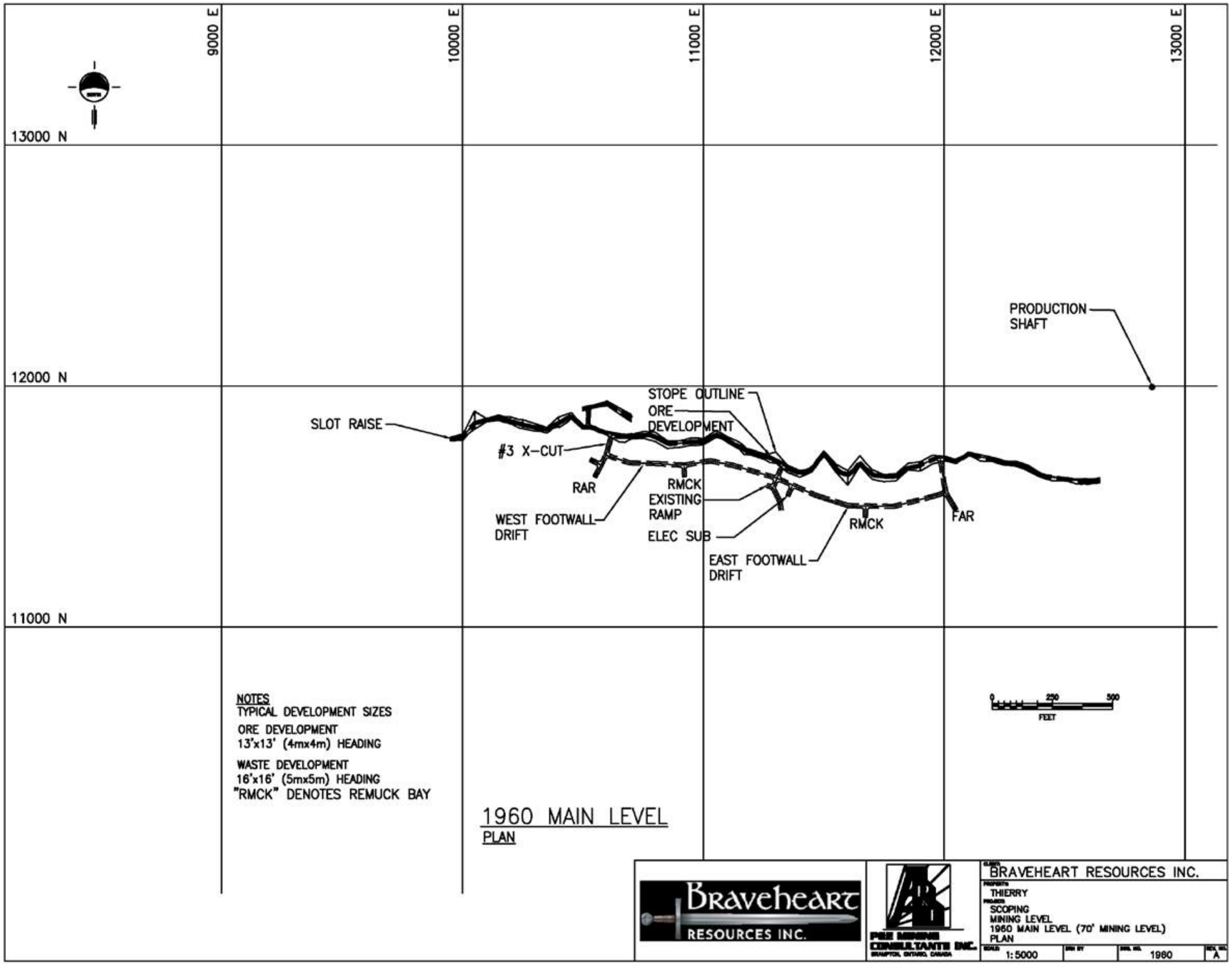
Underground 305 m (1,000 ft) Mining Sublevel



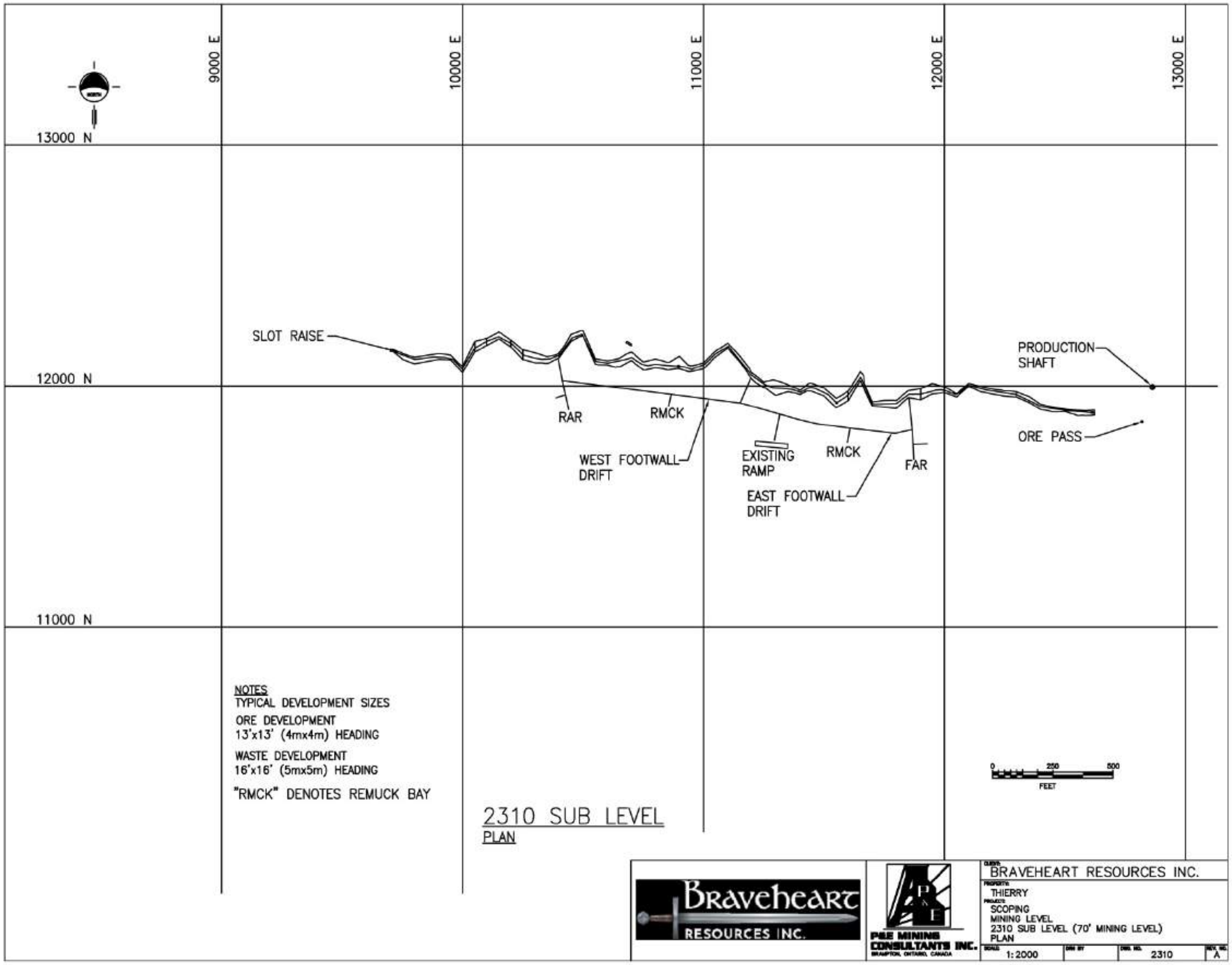
Underground 472 m (1,550 ft) Mining Level



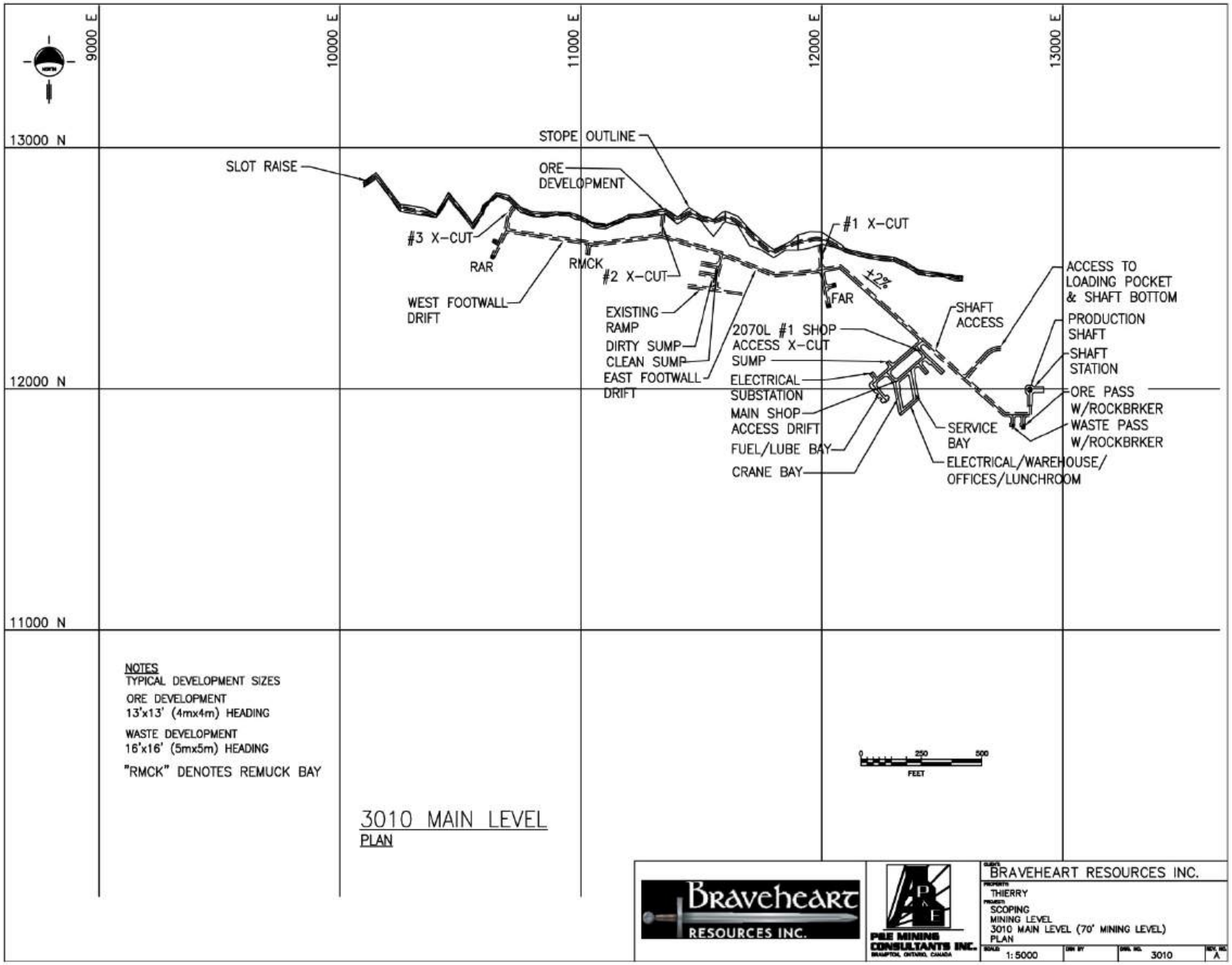
Underground 597 m (1,960 ft) Mining Level



Underground 704 m (2,310 ft) Mining Sublevel



Underground 917 m (3,010 ft) Mining Level



Underground 1067 m (3,500 ft) Mining Sublevel