TECHNICAL REPORT
AND
UPDATED RESOURCE ESTIMATE
ON THE
BUFFALO ANKERITE, FULLER, PAYMASTER,
AND
DAVIDSON TISDALE
GOLD DEPOSITS
PORCUPINE MINING DIVISION
NORTH-EASTERN ONTARIO, CANADA

NTS 42E 12/SW

FOR

Lexam VG Gold Inc.

Prepared By

P & E Mining Consultants Inc.

AND

Roscoe Postle Associates Inc.

NI 43-101F1 TECHNICAL REPORT
P&E Report No. 268

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TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

1.0 SUMMARY

Lexam VG Gold ("Lexam VG" or the "Company") is currently exploring four properties located in Tisdale and Deloro Townships within the city limits of Timmins, Ontario. Lexam owns a 100% interest in the Fuller and Buffalo Ankerite properties, a 60% interest in the Paymaster property (the remaining 40% JV interest is held by Goldcorp Inc.), and a 68.5% interest in the Davidson Tisdale Property (the remaining 31.5% is held by SGX Resources Inc.).

The Lexam Timmins project is situated in the south-western part of the Abitibi Greenstone Belt within the Archean Superior Province (Figure 7.1). The geology of the Timmins Camp comprises a thick sequence of Archean volcanic and sedimentary rocks that have been intruded by synvolcanic and post tectonic felsic dykes (Figure 7.1). The volcanic-sedimentary sequence has been subdivided into three main groups, the Deloro, Tisdale and Porcupine Groups. The Porcupine syncline separates the camp into a north limb and a south limb with three Lexam properties, Fuller, Buffalo Ankerite and Paymaster on the south limb and Davidson Tisdale on the north limb.

Gold mineralization on the Lexam properties belongs to the structurally controlled Archean lode gold class of deposits. Mineralization occurs in highly carbonate-altered zones often in or adjacent to porphyry bodies. Gold occurs in a quartz vein zones associated with a strong shear and sericite-carbonate alteration halos. Commonly, the quartz conforms to the shearing along strike, but often cross cuts the shearing down dip. Locally the stringer zones are very irregular and contain very erratic gold values.

The majority of the Lexam exploration on the four (4) properties has been diamond drilling. Historically, the properties have all been underground gold producers and therefore historic surface and underground drilling is also available in limited details. Lexam has conducted extensive surface drilling on all four projects during the period 2003 through 2012.

1.1 BUFFALO ANKERITE

P & E Mining Consultants Inc., ("P&E") prepared Mineral Resource estimates for the Buffalo Ankerite North and South zones including the portions of these zones that lie on the east-adjacent Paymaster Property. The estimates have an effective date of June 1, 2013. The P&E estimate was carried out by 3D computer block modelling in contrast to a previous estimate and NI 43-101 Technical Report which was completed in October 2012 by Bevan and Guy (2012) for Lexam and which was based on the same drill hole database but utilized different parameters and the sectional polygonal method, the latter not well suited to open pit design and optimization.

Lexam and its consultants provided P&E with the drill hole and assay database, specific gravity data and wireframes of stopes and mine workings for the North and South zones. Preliminary wireframes for mineralization were also provided but these were extensively modified by P&E. The P&E estimates include data from both historical underground and surface drilling, and recent surface drilling by Lexam and its predecessor company. However, none of the historic underground chip/channel sampling data was available for the P&E estimates.

The drilling, sampling and assay data and the coordinate system are in Imperial Measure consequently P&E retained Imperial units for the resource estimate and resource reporting. Gold grades are in troy ounces per short ton and length measure is in feet.

The Mineral Resource estimates for the North and South zones on the Buffalo Ankerite Mine property are based entirely on surface and underground diamond drilling, core sampling and assaying. The drill hole database, which includes holes drilled in the Paymaster and Fuller deposits, contains 5,734 diamond drill holes totalling 1,615,789.77 ft (492,492.72 m). Mineral Resources in the North Zone were intersected by 735 holes totalling 240,418.21 ft (73,279 m) whereas those in the South Zone were intersected by 692 holes for 241,422.3 ft (73,586 m).

The Mineral Resources for the North and South zones were estimated by conventional 3D computer block modelling using GEMS™ 6.3 and 6.4 mining software (GEMS) by GEMCOM Software International Inc. Mineral Resource estimation was constrained by mineral zone wireframes and used multiple search ellipses mapped to the zones' orientations and inverse distance to the power of 3 (ID^3) for grade interpolation. Resources were estimated for open pit and underground mining based on wireframe cut-offs of 0.015 oz/ton Au (0.5 g/t Au) for open pit and 0.045 oz/ton Au (1.5 g/t Au) for underground. Preliminary open pits, with 45° slopes, were designed from the respective zones' resource block models using Whittle™ software.

P&E classified Mineral Resources as Indicated Resources and Inferred Resources. Indicated Resources are outlined where surface and/or underground drill hole spacing is in the order of 150 ft for the North Zone and 100 ft for the South Zone.

The Indicated and Inferred Resources within the Whittle optimized pits are summarized in Table 1.1. Resources outside the pits are considered as underground Mineral Resources. Table 1.2 summarizes the underground resources for a cut-off grade of 0.075 oz/t Au.

Mineral resources are reported net of past production based on modelled stopes and drifts located in the mineral zones.

At a 0.015 oz/ton Au (0.51 g/t Au) cut-off, the Buffalo Ankerite Property contains an open pit Indicated Mineral Resource of 3.15 million tons (2.86 million tonnes) at 0.074 oz/ton Au (2.54 g/t Au) totalling 234,600 contained gold ounces, and an Inferred Mineral Resource of 2.90 million tons (2.63 million tonnes) at 0.068 oz/ton Au (2.33 g/t Au) totalling 197,800 contained gold ounces. At a 0.075 oz/ton Au (2.57 g/t Au) cut-off, the Buffalo Ankerite Property contains Indicated Mineral Resource of 3.58 million tons (3.25 million tonnes) at 0.139 oz/ton Au (4.77 g/t Au) totalling 499,000 contained gold ounces, and an underground Inferred Mineral Resource of 3.10 million tons (2.81 million tonnes) at 0.118 oz/ton Au (4.41 g/t Au) totalling 367,000 ounces.

The Buffalo Ankerite Property has a total Indicated Mineral Resource of 6.73 million tons (6.11 million tonnes) at 0.109 oz/ton Au (3.74 g/t Au) totalling 733,600 contained gold ounces, and a total Inferred Mineral Resource of 6.00 million tons (5.44 million tonnes) at 0.094 oz/ton Au (3.22 g/t Au) totalling 564,800 contained gold ounces.

At a 0.015 oz/ton Au (0.51 g/t Au) cut-off, the Paymaster portion of the North and South zones contains an open pit Indicated Mineral Resource of 1.76 million tons (1.60 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 96,250 contained gold ounces, and an open pit Inferred

Mineral Resource of 0.192 million tons (0.174 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 10,460 contained gold ounces. At a 0.075 oz/ton Au (2.57 g/t Au) cut-off, the Paymaster portion of the North and South zones contains an underground Indicated Mineral Resource of 0.020 million tons (0.018 million tonnes) at 0.100 oz/ton Au (3.43 g/t Au) totalling 1,970 contained gold ounces, and an underground Inferred Mineral Resource of 0.002 million tons (0.0018 million tonnes) at 0.082 oz/ton Au (2.18 g/t Au) totalling 157 contained gold ounces.

The Paymaster portion of the North and South zones has a total Indicated Mineral Resource of 1.78 million tons (1.61 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 98,220 contained gold ounces, and an Inferred Mineral Resource of 0.193 million tons (0.175 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 10,620 contained gold ounces.

Lexam holds 100% interest in the Buffalo Ankerite property and 60% interest in the Paymaster Property and respectively in the properties' Mineral Resources. From the above Mineral Resources, Lexam's attributable gold ounces contained in the North and South zones on the Buffalo Ankerite and Paymaster properties are 792,000 ounces in Indicated Mineral Resources and 571,000 ounces in Inferred Mineral Resources. The Qualified Person for the Buffalo Ankerite Mineral Resource estimate is Richard Routledge, P.Geo., of P&E. The effective date of the estimate is June 1, 2013.

1.2 MINERAL RESOURCES

1.2.1 P&E 2013 Buffalo Ankerite Project Mineral Resource Estimate

TABLE 1.1
BUFFALO ANKERITE OPEN PIT RESOURCES AT A CUT-OFF GRADE OF 0.015 OZ/TON AU

Zone	Indicated Resources				Inferred Resources			
	Tons (000's)	**Au oz/ton**	**Au Ounces (000's)**	**Lexam Ounces (000's)**	**Tons (000's)**	**Au oz/ton**	**Au Ounces (000's)**	**Lexam Ounces (000's)**
North	532	0.071	37.6	37.6	198	0.07	13.8	13.8
South	2,622	0.075	197	197	2,707	0.068	183	183
Total	**3,154**	**0.074**	**235**	**235**	**2,905**	**0.068**	**197**	**197**
Paymaster Open Pit Resources at a Cut-Off Grade of 0.015 oz/ton Au								
North	1,702	0.054	92.1	55.3	78.5	0.046	3.74	2.24
South	57.8	0.072	4.15	2.49	113	0.061	6.88	4.13
Total	**1,760**	**0.055**	**96.3**	**57.8**	**192**	**0.055**	**10.6**	**6.37**

**P&E 2013*

TABLE 1.2 BUFFALO ANKERITE UNDERGROUND RESOURCES AT A CUT-OFF GRADE OF 0.075 OZ/TON AU								
Zone	**Indicated Resource**				**Inferred Resource**			
	Tons (000's)	**Au oz/ton**	**Au Ounces (000's)**	**Lexam Ounces (000's)**	**Tons (000's)**	**Au oz/ton**	**Au Ounces (000's)**	**Lexam Ounces (000's)**
North	1,779	0.149	266	266	1,017	0.122	124	124
South	1,818	0.128	233	233	2,082	0.117	243	243
Total	**3,597**	**0.139**	**499**	**499**	**3,099**	**0.118**	**367**	**367**
Paymaster Underground Resources at a Cut-Off Grade of 0.075 oz/ton Au								
North	19.8	0.100	1.97	1.18	1.78	0.079	0.140	0.0840
South	-	-	-	-	0.141	0.117	0.017	0.0102
Total	**19.8**	**0.100**	**1.97**	**1.18**	**1.92**	**0.082**	**0.157**	**0.0942**

**P&E 2013*

1.3 RPA 2013 FULLER PROJECT MINERAL RESOURCE ESTIMATE

Roscoe Postle Associates, Inc., ("RPA") prepared an updated Mineral Resource estimate for the Fuller Property with the effective date of May 22, 2013. The previous Mineral Resource estimate was completed by Wardrop Engineering Inc. (Wardrop, now Tetra Tech) in 2007 and reported in a Technical Report on the property prepared for VG Gold Corporation (Wardrop, 2007). Fifty-three additional drill holes have been completed on the property since the Wardrop estimate.

Lexam provided RPA with the current drill hole database as well as density measurements. Lithology and mineralization wireframes interpreted by Wardrop in the previous estimate, as well as composite samples used in the Wardrop estimation, were also provided to RPA. The current estimate includes data from both historical and recent drilling and underground sampling. Assay results for all drilling had been received at the time of the estimate.

The updated Mineral Resource estimate is based on 3D block modelling utilizing Datamine Studio 3 and Gemcom GEMS 6.5 software. The Mineral Resources are unconstrained by wireframes: the block model was constrained by dynamic search angles and a constrained ellipse in the across strike direction. Dynamic angles used to dictate the orientation of the axes of the search ellipse were created by means of structural wireframe surfaces and strike and dip polylines representing the strike and dip of the main mineralized structural fabric. The block model and drill holes were domained coincidently with grade interpolation by means of a probabilistic constraining technique to aid the validation of resulting estimates.

The Fuller open pit (OP) and underground (UG) Mineral Resource estimate is summarized in Table 1.3. The open pit resource is constrained within a preliminary pit shell. Resources located outside the pit shell are reported as underground resources. The Qualified Person for the Fuller Mineral Resource estimate is Katharine Masun, P.Geo., Senior Geologist with RPA. The effective date of the estimate is May 22, 2013.

Note that all measurements stated herein are imperial measurements, i.e., tonnage in short tons, metal content in ounces per short tons, coordinates in feet, density in short tons per cubic foot.

TABLE 1.3 MINERAL RESOURCE ESTIMATE – MAY 22, 2013				
Lexam VG Gold Inc. - Fuller Project				
Classification	**Cut-off Grade (opt Au)**	**Tonnage (000 tons)**	**Grade (opt Au)**	**Contained Metal (000 oz Au)**
OP				
Indicated	≥0.015	5,878	0.049	290
Inferred	≥0.015	2,981	0.038	112
UG				
Indicated	≥0.075	361	0.168	61
Inferred	≥0.075	930	0.145	135
Total Indicated		**6,239**	**0.056**	**351**
Total Inferred		**3,911**	**0.063**	**247**

Notes:
(1) CIM definitions were followed for Mineral Resources.
(2) Mineral Resources are estimated at a cut-off grade of 0.015 opt Au for OP and 0.075 opt Au for UG.
(3) Mineral Resources are estimated using a gold price of US$1,600/oz, and a US$/C$ exchange rate 1:1.
(4) Numbers may not add due to rounding

1.4 RPA 2013 PAYMASTER PROJECT MINERAL RESOURCE ESTIMATE

1.4.1 Project Summary

RPA prepared an updated Mineral Resource estimate for the Paymaster Property. The previous Mineral Resource estimate was completed by Kenneth Guy and Peter Bevan in 2010 and reported in a Technical Report on the property prepared for VG Gold Corporation (Guy and Bevan, 2010). Twenty-four additional drill holes have been completed on the property since the 2010 estimate.

The updated Mineral Resource estimate for the Paymaster Project is summarized in Table 1.4. The estimate was carried out using Gemcom GEMS 6.4 in two stages. Initially, an open pit (OP) resource was estimated using a lower gold cut-off grade, and then an underground (UG) resource was defined below the pit shell, at a higher gold cut-off grade. The Mineral Resources were classified as Indicated and Inferred, with all of the Indicated Resources located within the open pit. The Qualified Person for the Paymaster Mineral Resource estimate is Tudorel Ciuculescu, M.Sc., P.Geo., Senior Geologist with RPA. The effective date of the Paymaster Mineral Resource estimate is May 22, 2013.

Note that all measurements stated in this section are imperial measurements, i.e., tonnage is in short tons, metal content in ounces per short tons, coordinates in feet, density in short tons per cubic foot.

TABLE 1.4 MINERAL RESOURCE ESTIMATE – MAY 22, 2013					
Lexam VG Gold Inc. - Paymaster Project					
Classification	**Cut-off Grade (opt Au)**	**Tonnage (tons)**	**Grade (opt Au)**	**Gold (ounces)**	**Lexam Ounces**
OP					
Indicated	≥0.015	5,135,000	0.047	242,000	145,000
Inferred	≥0.015	1,542,000	0.047	72,000	43,000
UG					
Indicated	-	-	-	-	
Inferred	≥0.075	239,000	0.179	43,000	26,000
Total Indicated		**5,135,000**	**0.047**	**242,000**	**145,000**
Total Inferred		**1,781,000**	**0.065**	**115,000**	**69,000**

Notes:

(1) CIM definitions were followed for Mineral Resources.

(2) Mineral Resources are estimated at a cut-off grade of 0.015 opt Au for OP and 0.075 opt Au for UG.

(3) Mineral Resources are estimated using a gold price of US$1,600/oz, and a US$/C$ exchange rate 1:1.

(4) A minimum mining width of approximately 20 ft was used for OP and approximately 5 ft for UG.

(5) Numbers may not add due to rounding.

A nominal minimum horizontal width of 20 feet was used as a guide for the OP and five feet for the UG. The largest OP mineralized wireframe straddles the existing stopes, while the rest of the mineralized wireframes are mostly parallel to the former. The UG wireframes are narrower and some of them represent the higher grade core of the OP wireframes situated below the pit shell.

1.5 P&E 2013 DAVIDSON TISDALE PROJECT MINERAL RESOURCE ESTIMATE

P&E prepared an updated Mineral Resource estimate for the Davidson Tisdale Property with an effective date of April 2, 2013.

The updated Mineral Resource estimate for the Davidson Tisdale Project was carried out in two stages using the Gemcom GEMS 6.4 software package. Initially, an open pit (OP) resource was estimated using a lower gold cut-off grade of 0.5 g/t Au (0.015 opt Au), and then an underground (UG) resource was defined below the pit shell, at a higher gold cut-off grade of 2.6 g/t Au (0.75 opt Au). The Mineral Resources were classified as Measured, Indicated or Inferred, with all of the open pit resources being either Measured or Indicated. The Qualified Persons for the Davidson Tisdale resource estimate were Yungang Wu, P.Geo., Eugene Puritch, P.Eng., and Antoine Yassa, P.Geo. of P&E.

In order for the constrained open pit mineralization in the Davidson Tisdale Deposit resource model to be considered potentially economic, a first pass Whittle 4X pit optimization was carried out to create a pit shell utilizing the criteria below:

Waste mining cost per tonne	$1.85
Ore mining cost per tonne	$1.85
Overburden Mining cost per tonne	$1.35
Process cost per tonne	$18
General & Administration cost per ore tonne	$5
Process production rate (ore tonnes per year)	525,000
Pit slopes (overall wall angle)	45 degrees
Mineralized Rock Bulk Density	$2.87 t/m^3$
Waste Rock Bulk Density	$2.90 t/m^3$
Overburden Density	$1.80 t/m^3$

The resulting resource estimate for the Davidson Tisdale project is summarized in the Table 1.5.

TABLE 1.5
MINERAL RESOURCE ESTIMATE FOR DAVIDSON TISDALE[(1-4)]

	Cut-Off (Au g/t)	Category	Tonnes	Au g/t	Contained Au Oz	68.5% Attributable OZ to Lexam
In-Pit	0.5	Measured	452,000	2.44	35,500	24,300
	0.5	Indicated	173,000	2.43	13,500	9,300
	0.5	Total	625,000	2.44	49,000	33,600
UG	2.6	Measured	18,000	6.64	3,800	2,600
	2.6	Indicated	41,000	4.91	6,500	4,400
	2.6	M+I	59,000	5.43	10,300	7,000
	2.6	Inferred	71,000	4.20	9,600	6,600
Total	0.5+2.6	Measured	470,000	2.60	39,300	26,900
	0.5+2.6	Indicated	214,000	2.90	20,000	13,700
	0.5+2.6	M+I	684,000	2.70	59,300	40,600
	2.6	Inferred	71,000	4.20	9,600	6,600

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

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

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

(4) The historical mined tonnage was not depleted as the mined tonnage was insignificant.

(5) The Davidson Tisdale project is a joint venture between Lexam as operator (68.5%) and SGX Resources Inc. (31.5%). The contained Au oz reflects the 68.5% of the resource attributable to Lexam.

1.6 RECOMMENDATIONS AND PROPOSED BUDGET

Recommendations include limited diamond drilling and a preliminary ecomonic assessment (PEA).

1.6.1 Drilling

<u>Buffalo Ankerite</u>

An exploration programme is proposed for the property with emphasis on exploring the depth extent of the North Zone, in particular the area below 2,000 feet below surface where historic records indicate a mineralized Quartz-Feldspar Porphyry (QFP) body with gold mineralization. The proposed programme will concentrate on the continuation of the exploration of the Buffalo Ankerite North Zone, following up on the success of the drilling to date. Emphasis will be placed on expanding the depth extent of the gold mineralization. Upon completion of the drill programme an examination of the data should be conducted in order to determine where to explore the porphyry at depth as information indicates that economic values have been obtained at the 2,200 to 3,500 foot levels.

No further drilling is necessary on the South Zone as the eastern portion has been drilled to the plunge line and in the western section the better potential at depth is in the North Zone area.

<u>Fuller</u>

Two targets at depth remain not adequately tested on the Fuller property:

- Follow-up on holes VG96-26 and 96-30
- Below the 1500 foot level.

VG-96-26 intersected 3.9 gpt Au/63.5 m including 7.5 gpt Au/8.9 m and 8.3 gpt Au/3.2 m and 4.8 gpt Au/16.7 m at 1,300 feet below surface.

VG-96-30 intersected 5.0 gpt Au/10.9 m and 9.6 gpt Au/8.8m at 1,600 feet below surface.

Two holes were drilled in 2009 - VGF-09-111 and VGF-09-112 to test this target. They intersected respectively 70 m west/90 m below and 40 m east/200 m below the previous drilling. The best result was 09-111 with 8.2 gpt Au/2.0 m corresponding to the previous intersections. These holes are excessively distant from the previous drilling to eliminate this target.

Three holes are recommended to intersect closer to the previous drilling to verify the mineralization.

<u>Paymaster</u>

No drilling is recommended at this time at the Paymaster property

<u>Davidson Tisdale</u>

No drilling is recommended at this time at the Tisdale property

Budget

TABLE 1.6 PROPOSED BUDGET				
Project	**Activity**	**Units**	**$/Unit**	**Cost**
Drilling				
Buffalo Ankerite	drilling	5,000	$160	$800,000
Fuller	drilling	1,000	$160	$160,000
PEA				$300,000
Subtotal				$1,260,000
Admin - 15%				$190,000
Total				**$1,450,000**

2.0 INTRODUCTION

2.1 TERMS OF REFERENCE

This report titled, "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Properties, Porcupine Mining Division, North-Eastern Ontario, Canada" (the "Technical Report") with an effective date of June 01, 2013, was prepared to provide a NI 43-101 compliant technical report and updated resource estimate of the gold mineralization on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Properties, Porcupine Mining Division, Timmins Mining Camp, Ontario (the Lexam "Properties").

The four gold Properties are located in Tisdale and Deloro Townships and are situated within the city limits of Timmins, Ontario. Lexam VG Gold Inc, ("Lexam VG" or the "Company") owns a 100% interest in the Fuller and Buffalo Ankerite properties, a 60% interest in the Paymaster property (the remaining 40% JV interest is held by Goldcorp Inc.), , and a 68.5% interest in the Davidson Tisdale Property (the remaining 31.5% is held by SGX Resources Inc.)

This Technical Report has been prepared pursuant to the guidelines of Canadian National Instrument ("NI") 43-101F1 and as such is compliant with NI 43-101 regulations which require that all resource estimates be prepared in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy ad Petroleum ("CIM") Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council and in force as of the effective date of the report.

This Technical Report was prepared jointly by P & E Mining Consultants Inc., ("P&E") and by Roscoe Postle Associates, Inc., ("RPA") at the request of Mr. Ken Guy, Exploration Manager of Lexam, which is a Toronto, based company trading on the TSX under the symbol "LEX" and on the OTCQX in the United States as "LEXVF". It also trades in Europe on the Frankfurt Exchange as "VN3A" The head office address is:

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This Report is current as of June 01, 2013.

The purpose of the current Report is to provide an independent, NI 43-101 compliant, Technical Report that includes updated mineral resource estimates on the Properties. This Report will be filed on SEDAR as required under NI 43-101 disclosure regulations and will be used to provide technical support for the advancement of the Properties through the initiation of higher level engineering studies.

The Company has accepted that the qualifications, expertise, experience, competence and professional reputation of P&E's and RPA's Principals and Associate Geologists and Engineers are appropriate and relevant for the preparation of this Report. The Company has also accepted that the Principals of P&E and RPA are members of professional bodies that are appropriate and relevant for the preparation of this Report.

2.2 SITE VISITS

Mr. Antoine Yassa, P.Geo. of P&E, a 'qualified person' under the terms of NI 43-101, conducted a site visit to both the Buffalo Ankerite and Davidson Tisdale properties during the period November 03 and 04 and on November 06, 2012 (3 days). During that period a validation sampling program was completed as part of a larger, more inclusive QA/QC sampling program.

Mr. Tudorel Ciuculescu P.Geo of RPA, a 'qualified person' under the terms of NI 43-101, conducted a site visit to the Paymaster Property during the period November 29-30, 2012. No validation sampling was completed at that time.

Ms. Katharine Masun, P.Geo. of RPA, a 'qualified person' under the terms of NI 43-101, conducted a site visit to the Fuller Property on November 20th and 21st, 2012. No validation sampling was completed at that time.

2.3 UNITS AND CURRENCY

Unless otherwise stated all units used in this report are metric unless Imperial measurements are specifically noted. Au assay values are reported in grams per metric tonne ("g/t") unless some other unit such as ounces per short ton ("opt") is specifically stated. The Canadian $ is used throughout this report unless otherwise indicated. At the effective date of this report the CDN $ was at par with the US$.

2.4 SOURCES OF INFORMATION

This 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 27.0 at the conclusion of this Technical Report. Several sections from reports authored by other consultants may be directly quoted in this Technical Report, and are so indicated in the appropriate sections.

This report has drawn heavily upon material presented in the following recent Technical Reports on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Properties.

Guy, K.W., And Bevan, P.A., 2012: Ni 43-101 Technical Report Resource Estimate on the Buffalo Ankerite Property, Porcupine Mining Division, North-Eastern Ontario, Canada. Prepared for Lexam VG Gold Inc. and Dated October 20, 2012. ("Guy and Bevan, 2012")

Naccashian, S., 2006: Mineral Resource Estimate of the Fuller Gold Property. Prepared for Vedron Gold Inc. by Wardrop Engineering Inc. and Dated May 3, 2006, ("Naccashian, 2006")

Guy, K.W, and Bevan, P.A., 2010: Summary Report on Exploration and Resource Technical Report on the Paymaster Option, Porcupine Mining Division, North-eastern Ontario, Canada. Prepared for VG Gold Corp. and dated December 20, 2010 ("Guy and Bevan, 2010")

Guy, K.W, and Puritch, E.J., 2007: Exploration Report 2003 - 05 and Resource Estimate Technical Report on the Tisdale Project, Porcupine Mining Division, North-eastern Ontario, Canada. Prepared for Vedron Gold Inc. and dated March 23, 2007. ("Guy and Puritch, 2007")

2.5 GLOSSARY OF TERMS

In this document, in addition to the definitions contained heretofore and hereinafter, unless the context otherwise requires, the following terms have the meanings set forth below.

"CDN $"	means the currency of Canada
"AA"	is an acronym for Atomic Absorption, a technique used to measure metal content subsequent to fire assay
"Ag"	means silver
"asl"	means above sea level
"Au"	means gold
"AusIMM"	mean Australian Institute of Mining and Metallurgy
"Azi"	means azimuth
"BLK"	means Bulk Leachable Au
"Buffalo Ankerite"	Means the Buffalo Ankerite Gold Property in Deloro Township, Timmins, Ontario"
"CIM"	means the "Canadian Institute of Mining, Metallurgy and Petroleum"
"CSA"	means the Canadian Securities Administrators
"Davidson Tisdale"	Means the Davidson Tisdale Gold Property in Tisdale Township, Timmins, Ontario
"DDH"	means diamond drillhole
"E"	means east
"el"	means elevation level
"FS"	means Feasibility Study
"ft"	means feet
"Fuller"	means the Fuller Gold Property in Tisdale and Deloro Townships, Timmins, Ontario
"Ga"	means billions of years
"g/cm^3"	means grams per cubic centimetre
"g/m^3"	means grams per cubic metre
"GNP"	means Gross National Product
"g/t"	means grams per tonne
"g/t Au"	means grams of Au per tonne of rock
"ha"	means Hectare
"in"	means inches
"IP"	means Induced Polarization
"IRR"	means Internal Rate of Return
"JORC code"	means Australian Joint Ore Reserve Committee
"kg"	means kilogram
"km"	means kilometre equal to 1,000 metres or approx. 0.62 statute miles
"m"	means metric metre distance measurement equivalent to approximately 3.27 feet
"M"	means million
"Ma"	means millions of years
"MDRU"	means the Mineral Deposits Research Unit

"mi"	means miles
"mm/an"	means millimetres per annum
"Mt"	means millions of tonnes
"N"	means North
"NE"	means North-east
"NI 43-101"	means Canadian Securities Administrators National Instrument 43-101
"NN"	means Nearest Neighbour
"NW"	means North-west
"OP"	means open pit
"orpailleur"	means a traditional local miner using rudimentary means to collect oxidized surface/sub-surface material from which Au is extracted
"orpaillage"	means the act of collecting surface/subsurface oxidized material from which Au is extracted
"oz/T"	means Troy ounces per short ton
"P&E"	means P & E Mining Consultants Inc. a resource consulting firm
"Paymaster"	means the Paymaster Gold Property in Deloro and Tisdale Townships, Timmins, Ontario.
"PEA"	means a Preliminary Economic Assessment study
"ppb"	mean parts per billion
"ppm"	means parts per million
"Property"	means Riverstone's property holdings in Burkina Faso
"RPA"	means RPA a resource consulting firm
"S"	means south
"SE"	means south-east
"SEDAR"	means the System for Electronic Document Analysis and Retrieval
"SG"	means specific gravity
"SW"	means south-west.
"t"	means metric tonne equivalent to 1,000 kilograms or approximately 2,204.62 pounds
"T"	means Short Ton (standard measurement), equivalent to 2,000 pounds
"t/a"	means tonnes per year
"tpd"	means tonnes per day
"UG"	means underground
"US$"	means the currency of the United States of America
"W"	means west

3.0 RELIANCE ON OTHER EXPERTS

The authors have assumed, and relied on the fact, that all the information and existing technical documents listed in the References section of this Technical Report are accurate and complete in all material aspects. While we carefully reviewed all the available information presented to us, we cannot guarantee its accuracy and completeness. We reserve 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 Report.

Although copies of the tenure documents, operating licenses, permits, and work contracts were reviewed, an independent legal assessment of land title and tenure was not performed. Neither P&E nor RPA has verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has relied on Lexam's solicitor(s) to have conducted the proper legal due diligence in this regard. Information on tenure and permits was obtained from Lexam and relied upon by the authors who did independently compare the information supplied with available data from public records using the Ministry of Ontario's CLAIMaps III website (for unpatented and leased claims only, patented claim information is not available).

The authors are not aware of any outstanding environmental, socio-political or permitting issues and have relied on opinions provided by Lexam in this regard.

A draft copy of this Technical Report has been reviewed for factual errors by the Company and the authors have relied on the Company's historical and current knowledge of the Properties in this regard. Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Technical Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 INTRODUCTION

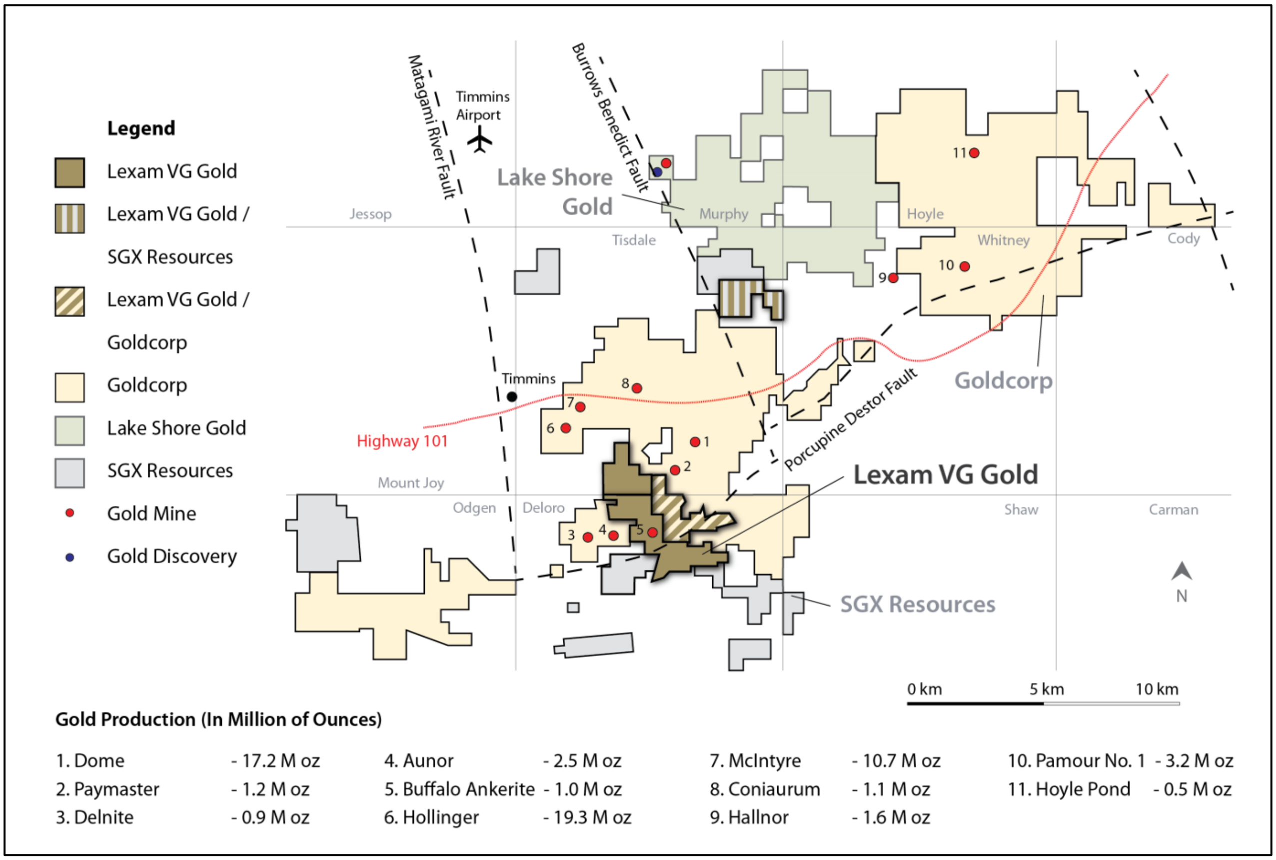
The Lexam properties are located in Tisdale and Deloro Townships within the municipal boundaries of the city of Timmins in north-eastern Ontario (Figure 4.1). The properties are past producers and part of the historic Porcupine Gold Camp.

Gold production from the Porcupine camp over the past 100 years is greater than 67 million ounces at a Camp average grade of 0.213 ounces gold per ton (Atkinson et al, 2005). Ten different mines have produced in excess of 1.0 million ounces and account for 91% of the gold recovered in the Camp.

Figure 4.1 Regional Location Map

The claims form 2 groups - Davidson Tisdale in the north part of Tisdale Township and the contiguous package comprising the Buffalo Ankerite, Fuller and Paymaster Properties (Figure 4.2).

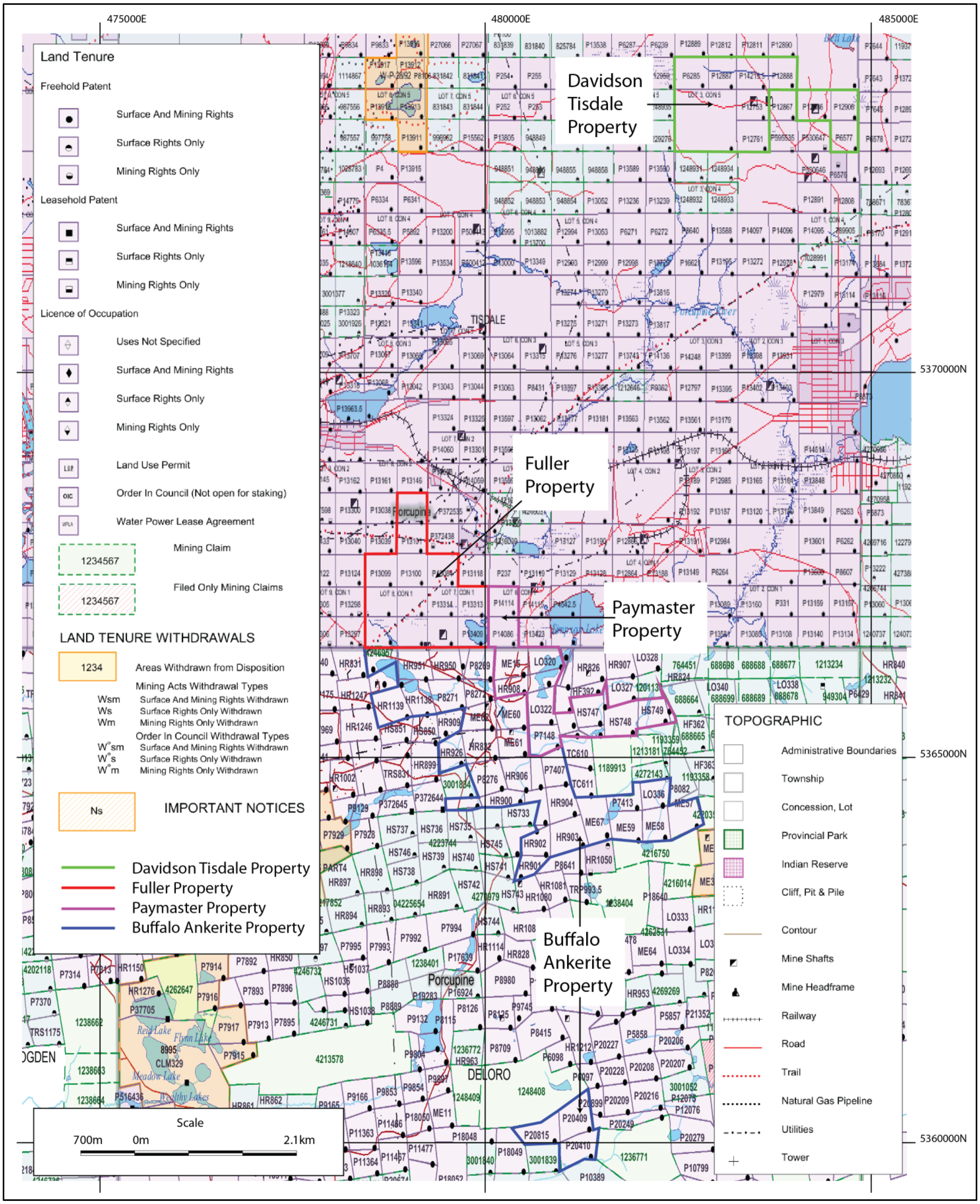
Figure 4.2 Buffalo Ankerite, Davidson Tisdale, Fuller, Paymaster Property Location Map



Source: *modified from Initerra mapping services 2008*

A claim map of the area that covers all the Properties is shown in Figure 4.3.

Figure 4.3 Buffalo Ankerite, Davidson Tisdale, Fuller and Paymaster Properties Claim Map



Source: *ClaimMaps Website*

4.2 BUFFALO ANKERITE PROPERTY

The Buffalo Ankerite Property consists of 36 patented mining rights/claims with an area of 491.4 hectares (1,214.2 acres) in the north half of Deloro Township.

The Buffalo Ankerite Property is located approximately 3.5 miles (5.6 km) southeast of the city of Timmins in northern Ontario. The property, a past producer, is situated in the historic Porcupine Gold Camp (see Figure 4.1, Figure 4.2 and Figure 4.3). Its boundaries were located either in the field, by the use of historic records, or from the claim map issued by the Minister of Northern Development, Mines and Forestry (MNDMF). The property contains an area of previous surface mining (fenced off with security) dating from the early 1980's, as well as a few historic shaft collars. An historic tailings pile is also found on the southern section of the property.

A list of claims appears in Table 4.1 and a claim location map is shown in Figure 4.4.

TABLE 4.1
BUFFALO ANKERITE PROPERTY LIST OF CLAIMS, DELORO TOWNSHIP

Patent #/ License	Parcel
ME60 15 SEC	23816 SEC
ME61 16 SEC	23816 SEC
ME62 17 SEC	23816 SEC
P7407 54 SEC	23816 SEC
P7406 55 SEC	23816 SEC
P7426 (HR 905) 183 SEC	23816 SEC
P7426A	23816 SEC
P7413 (ME73) 186 SEC	23816 SEC
HR906 1321 SEC	23816 SEC
P7934 (HR 952) 2725 SEC	23816 SEC
P8269 (ExpPt2:6R-1903) 3275 SEC	23816 SEC
P8271 (ME50) 3276 SEC	23816 SEC
P8271 (ME50) 3276 SEC	23816 SEC
PT P8272 (ExpPt2:6R-1903) 3279 SEC	23816 SEC
P8204 (LO 336) 3377 SEC	23816 SEC
P9598 (HR 904) 4155 SEC	23816 SEC
P9600 (HR 902) 4156 SEC	23816 SEC
P9599 (HR 903) 4157 SEC	23816 SEC
P9605 (ME 57) 4158 SEC	23816 SEC
P9604 (ME 58) 4161 SEC	23816 SEC
P9603 (ME 59) 4162 SEC	23816 SEC
P9602 (ME 67) 4163 SEC	23816 SEC
P9601 (HR 901) 4164 SEC	23816 SEC
HR832 5038 SEC	23816 SEC
HR823 5039 SEC	23816 SEC
P8276 5040 SEC	23816 SEC
HR926 5041 SEC	23816 SEC
HR900 5042 SEC	23816 SEC
HR950 (TRS 775) 5998 SEC	23816 SEC

Table 4.1 Buffalo Ankerite Property List of Claims, Deloro Township	
Patent #/ License	**Parcel**
HR951 (TRS 774) 22957 SEC	23816 SEC
HR1138 (TRS 1280) 2525 SEC	23816 SEC
HR951 (TRS 774) 22957 SEC	23817 SEC
HR1138 (TRS 1280) 2525 SEC	23817 SEC
HR1139 (TRS 1281) 2566 SEC	23817 SEC
HR830 (P7251)	23817 SEC
P24590	23817 SEC
P20410	23818 SEC
P20815	23818 SEC
P20409	23818 SEC

Lexam holds the mineral rights through either Lexam VG Gold or VG Holdings, a wholly owned subsidiary of Lexam. Numerous subdivided land parcels (surface rights) are owned and/or occupied by homeowners in the area. The remainder of the surface rights are held by either RiLoro Corporation or Lexam. A small neighbourhood of approximately 20 homes is located on the historic Buffalo Ankerite property.

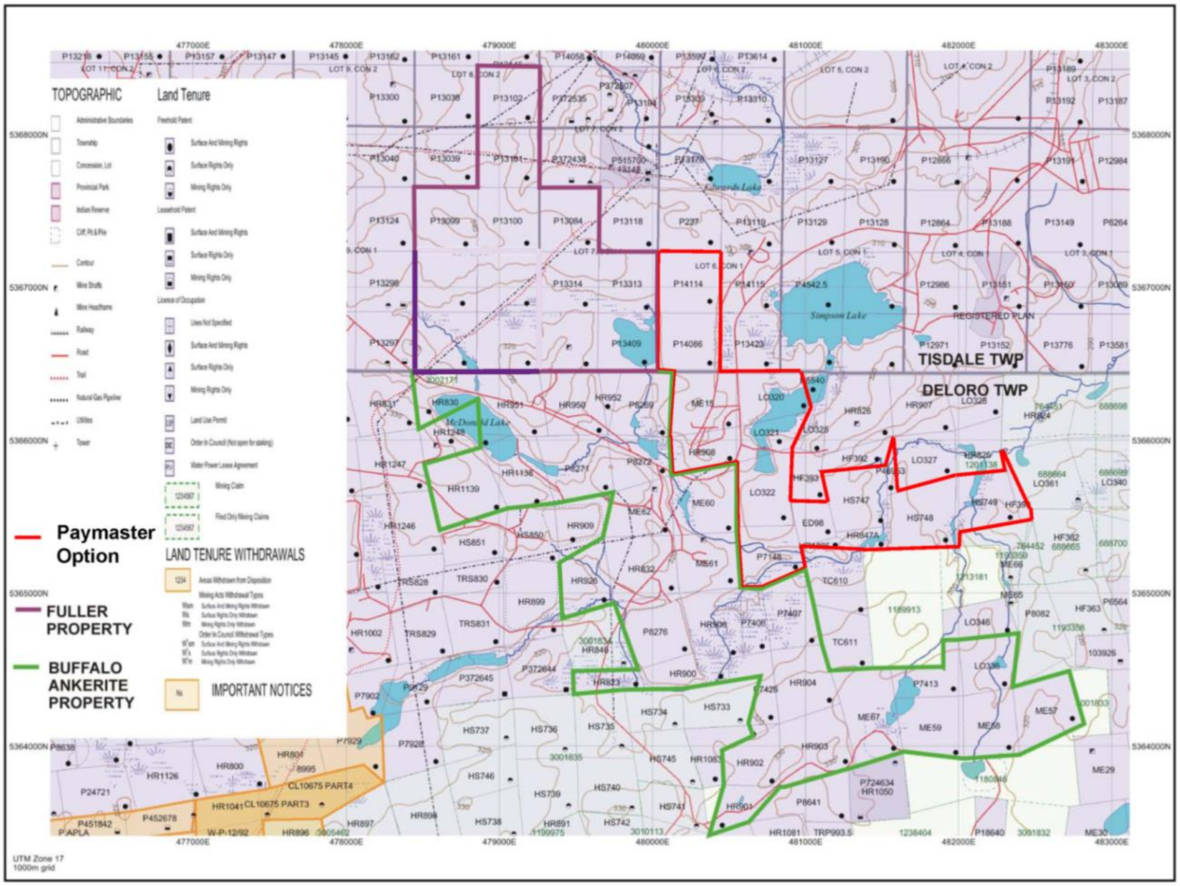
The Buffalo Ankerite property is subject to a net profits interest (NPI) royalty as follows: 10% NPI on all claims except for the south pit area of the South Mine (claims ME61, HR906, HR832, P8276) , where the NPI is 20% on ore mined by open pit mining methods. The NPI is payable to The Summit Organization Inc. To the authors' knowledge no additional encumbrances exist on the property.

To the authors' knowledge there are no outstanding environmental liabilities with the MNDMF for the Buffalo Ankerite property.

Lexam has received the permit to take water for the Buffalo Ankerite property. A closure plan application is in progress (75% completed). The permits for sewage and for air/ noise emissions are in the final stages of application. A closure plan for an open pit mining plan is in final stages of completion and a Site Plan Control Agreement will need to be arranged with the City of Timmins when planning reaches the pre-development exploitation stage.

The authors know of no other significant factors and risks that may affect access, title, or the ability to perform work on the Buffalo Ankerite property.

Figure 4.4 Buffalo Ankerite, Fuller and Paymaster Property Claim Map



Source: *ClaimMaps Website*

4.3 FULLER PROPERTY

The Fuller Property consists of 13 patented mining claims covering an area of 210.183 hectares (519.373 acres) in Tisdale Township within the Porcupine Mining Division.

The Fuller Property is located in the heart of the Timmins gold mining district, about 3 kilometres southeast of Timmins city center (see Figure 4.1, Figure 4.2 and Figure 4.3). A paved road to central Timmins, locally referred to as the "Gold Mine Road", crosses by the southwest corner of the property. The dormant underground workings on the Fuller claim are accessible by gravel covered side-roads. The south-eastern portion of the property is undeveloped, but there are roads within 2 kilometres of any point. There is an electrical power line to the Fuller underground workings, an old warehouse building and a small coreshack building.

Lexam holds 100% interest in the Fuller Property. The claims comprise parts of surveyed lots as shown in Figure 4.4 and as outlined in Table 4.2.

TABLE 4.2 FULLER PROPERTY LIST OF PATENT CLAIMS, TISDALE TOWNSHIP			
Description	**Claim No**	**Township**	**Hectares**
SW1/4, N1/2 , Lot 7, Con 1	P13084	Tisdale	16.137
SW1/4 , N1 /2, Lot 8, Con 1	P13099	Tisdale	16.187
SE1 /4, N1 /2 , Lot 8, Con 1	P13100	Tisdale	16.187
NE1 /4, N1 /2, Lot 8, Con 1	P13101	Tisdale	16.187
SE PI., S PI, Lot 8. Con 2	P13102	Tisdale	16.187
NE1/4. S1/2 , Lot 7, Con 1	P13313	Tisdale	16.137
NW1/4. S1 /2, Lot 7, Con 1	P13314	Tisdale	16.137
SE1/4, S1/2, Lot 7, Con 1 (Dobie claim)	P13409	Tisdale	16.137
SW1/4, S1/2. Lot 7, Con 1 (Fuller claim)	P13189	Tisdale	16.137
S1/2 Lot 8, Con 1 (Chisholm property)	P44835, P44836, P44837, P44838	Tisdale	64.750

A ramp excavated in the 1980's is collared on a single patented claim, called the "Fuller claim" (P13189). Lexam owns both the surface and the mineral rights for this claim. In 2008 Lexam acquired from Goldcorp the mineral rights for a parcel consisting of four claim units, called the “Chisholm property" (S1/2 of Lot 8, Con 1). In exchange for the mineral rights on the Chisholm property, Lexam granted Goldcorp the surface rights to five Fuller claims (P13099, P13100, P13313, P13314 and P13084).

The Fuller claim and the Chisholm property are free of net profits interest (“NPI”). The remaining claims of the Fuller property are subject to a 10% NPI royalty, which is payable to the Summit Organization Inc.

Lexam has received the permit to take water for the Fuller property. A closure plan application is in progress (75% completed). The permits for sewage and for air/ noise emissions are in the final stages of the application process. A closure plan is in final stages of completion and a Site Plan Control Agreement with the City of Timmins is required when the Property reaches the pre-development exploitation stage.

The authors have no knowledge of any outstanding environmental liabilities on the Fuller property, or of any other significant factors and risks that may affect access, title, or the ability to perform work on the property.

4.4 PAYMASTER PROPERTY

The Paymaster Property consists of 15 contiguous claim units covering 179.2 hectares (442.8 acres), with two claim units located in the south central part of Tisdale Township and the remaining 13 claim units in the north central part of Deloro Township.

The Paymaster property is located on Gold Mine Road, locally known as the “Back Road”. The road connects South Porcupine and Timmins and bisects the property (see Figure 4.1, Figure 4.2 and Figure 4.3). Numerous mine roads are found throughout the remainder of the Paymaster property. The property lies to the east of the Buffalo Ankerite and Fuller properties (100% owned by Lexam VG Gold) and to the west of the Dome Mine (owned by Goldcorp). The Paymaster property has gold mineralization, which are continuations of zones previously explored by Lexam VG Gold on the Buffalo Ankerite and Fuller properties.

The claims of the Paymaster Property are as shown in Figure 4.4 and as outlined in Table 4.3.

TABLE 4.3
PAYMASTER PROPERTY LIST OF PATENT CLAIMS, TISDALE AND DELORO TOWNSHIPS

Parcel Description	Claim No	Township	Hectares
PIN65398-0146(LT), Parcel 4455SWS (Mining rights from Surface to 4075 foot Level only, 12006.8 elevation)	P14114	Tisdale	16.390
PIN65398-0147(LT) Parcel 4456SWS (Mining rights from Surface to 4075 foot Level only, 12006.8 elevation)	P14086	Tisdale	16.390
PIN65442-0212(LT), Parcel 2441SEC	ME15	Deloro	17.604
P1N65442-0214(LT) Parcel 2440SEC	HR908	Deloro	10.117
PIN65442-0206(LT) Parcel 2477SEC	L0320	Deloro	14.326
PIN65442-0624(LT) Parcel 2526SEC	L0321	Deloro	8.984
PIN65442-0204(LT) Parcel 2527SEC	L0322	Deloro	16.511
PIN65442-0203(LT) Parcel12SEC	L0323	Deloro	15.479
PIN65442-0218(LT) Parcel 2512SEC	ED98	Deloro	7.203
PIN65442-0202(L T) Parcel 275SEC	HR1085, HR847A	Deloro	5.059
PIN65442-0219(LT) Parcel 15188SEC	HS747	Deloro	15.419
PIN65442-0580(LT) Parcel 13257SEC	HR1010	Deloro	1.416
PIN65442-0231 (LT) Parcel 15187SEC	HS748	Deloro	17.887
PIN65442-0236(LT) Parcel 15189SEC	HS749	Deloro	14.690
PIN65442-0239(LT) Parcel 3877SEC	HF390	Deloro	1.700

Lexam optioned the Paymaster property from Goldcorp in 2008 and earned 60% ownership interest in mineral rights in 2012. The remaining 40% of the mineral rights on the property remain with Goldcorp. The authors know of no royalties, back-in rights, payments or other agreements and encumbrances to which the property may be subject.

Goldcorp has a fully funded Closure Plan for the Paymaster property.

The authors know of no environmental liabilities on the property, nor of other significant factors and risks that may affect access, title, nor the ability to perform work on the Paymaster Property.

4.5 DAVIDSON TISDALE PROPERTY

The Davidson Tisdale property consists of 10 claims covering 207.7 hectares (513.2 acres) in Tisdale Township in the Timmins Mining Camp.

The project is accessible via an all-weather road off of Crawford Street in South Porcupine and it is located approximately 3 km along strike from three mines which have produced over 31 million ounces of gold (see Figure 4.1, Figure 4.2 and Figure 4.3).

The claims of the Davidson Tisdale Property are as shown in Figure 4.5 and as outlined in Table 4.4.

TABLE 4.4
DAVIDSON TISDALE LIST OF HOLDINGS, TISDALE TOWNSHIP

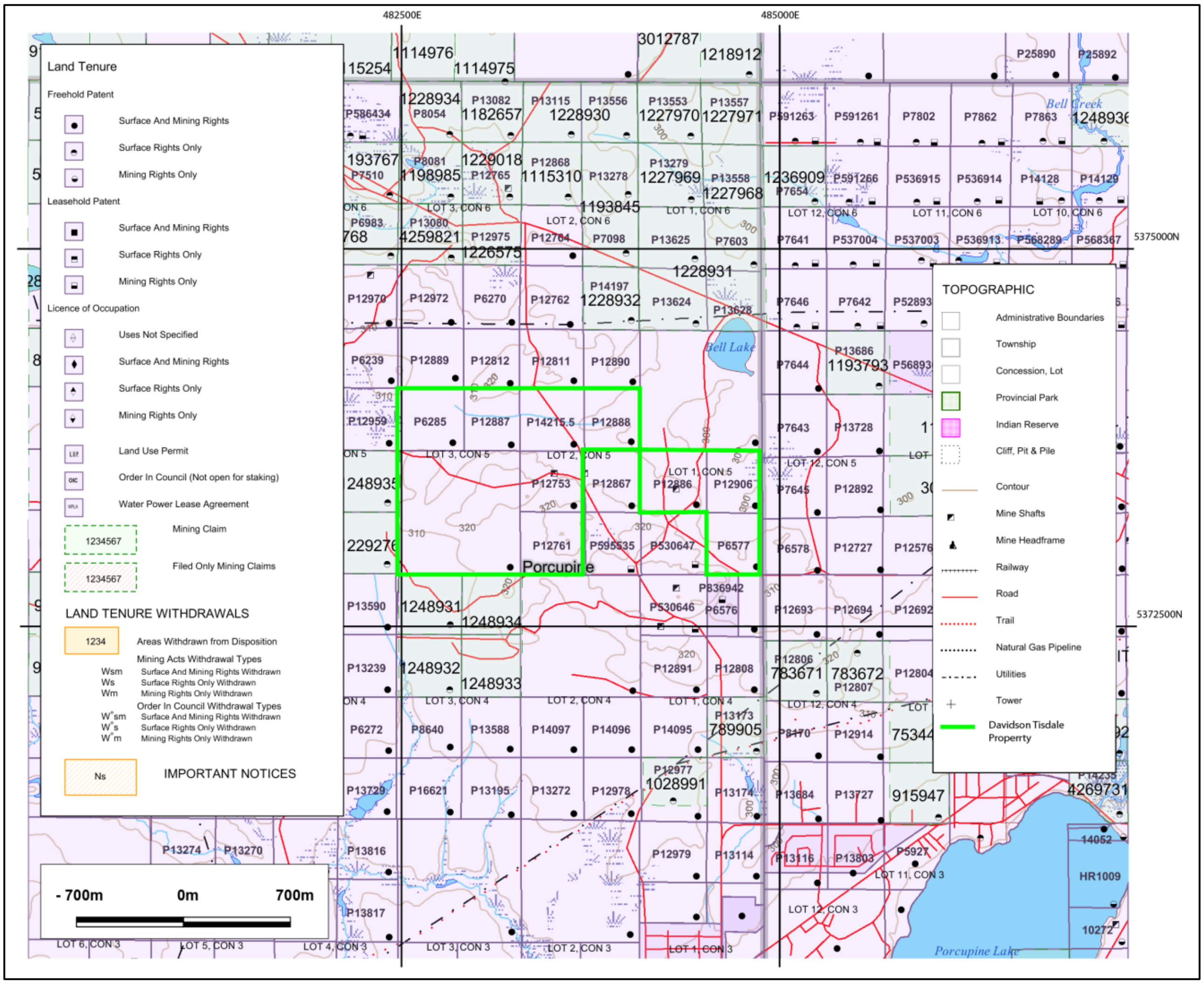
PIN Number	Parcel	Claim Number	Size hectares	¼	½	Lot	Concession
65399-0130 (LT)	3848WT	P12761	15.934	SW	S	2	5
65399-0129 (LT)	3847WT	P12753	15.934	NW	S	2	5
65399-0157 (LT)	14003WT	P12886	15.378	NW	S	1	5
		P12906	15.378	NE	S	1	5
		P6577	16.39	SE	S	1	5
65399-0134 (LT)	3853WT	P14125 ½	15.934	SW	N	2	5
65399-0133 (LT)	3852WT	Vet lot - all	64.547		S	3	5
65399-0155 (LT)	14004WT	P6285	16.137	SW	N	3	5
		P12887	16.137	SE	N	3	5
		P12888	15.934	SE	N	2	5

In 2009 Lexam earned a 68.5% interest in mining rights on the Davidson Tisdale property from Laurion Mineral Exploration Inc (see Figure 4.1, Figure 4.2 and Figure 4.3 for property maps). The remaining 31.5% interest is currently held by SGX Resources Inc. The surface rights to these claims are held by either one of ERG Resources, Davidson Tisdale Mines or the City of Timmins.

Lexam has a permit to take water, a permit for sewage, a permit for air/noise emissions and a closure plan for the Davidson Tisdale property.

The authors have no knowledge of any outstanding environmental liabilities, of any royalties, back-in rights, payments or other agreements and encumbrances on the property, that may affect access, title, or the ability to perform work on the Davidson Tisdale property.

Figure 4.5 Davidson Tisdale Property Claim Map



Source: *ClaimMaps Website*

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, PHYSIOGRAPHY & INFRASTRUCTURE

The claims comprising the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale properties are all located within the Municipality of Timmins and are accessible by either provincial or municipal roads. The contiguous Paymaster, Buffalo Ankerite and Fuller properties are all located proximal to or straddling a secondary highway, known locally as Gold Mine Road (Figure 4.2).

The Tisdale property is located at the end of a gravel road north of Crawford Street in South Porcupine (Figure 4.5).

The Dome Mine/Mill Complex is located approximately 1 kilometre east of the Paymaster Property.

The climate is typical of North East Ontario with temperatures in the range of 30°C in the summer to -30°C in the winter. The City of Timmins is about 100 years old and one of the most famous gold mining camps still productive in Canada.. The local community offers all the services for exploration and mine production both underground and open pit.

Timmins has a modern airport and it is connected to the province's major network of highways, including the Trans-Canada Highway.

The physiography is typical of glacial regions where the landscape is made up of low hills and numerous rivers and lakes.

The forest industry is also very active in the area, second only to mining.

Timmins is also host to the Kidd Creek mine, a polymetallic mine containing zinc, copper, gold and silver, owned and operated by Xstrata. It has a concentrator located in Timmins.

6.0 HISTORY

Prospecting in the Porcupine area began in 1907. The Dome, McIntyre and Hollinger mines were discovered in 1909 and the City of Timmins was founded in 1912 as a consequence of the Porcupine Gold Rush. The Porcupine gold camp is the largest historical gold producing district in North America, having produced over 70 million ounces of gold.

6.1 BUFFALO ANKERITE

Prior to 1935 the property was developed by two separate owners and operations which covered two separate mineralized zones, North Zone and South Zone. In 1935 the property was consolidated under Buffalo Ankerite Mines Limited the original North Zone operator.

6.1.1 Buffalo Ankerite North

Shaft to 50 feet and shaft to 120 feet by Dobie Mines Ltd.

1916 – 1921 Surface exploration, shafts to 350 feet, 130 feet and 3 shafts to 50 feet; work by Coniagas Mines Ltd. under an option from the owners, Ankerite Mining Company Ltd.

Mine workings re-sampled by United States Refining and Smelting Co. under an option from the owners, North American Gold Corporation.

Under an option agreement, underground development resumed by Porcupine Goldfields Development and Finance Co. Ltd. Lateral work on 200- and 300-foot levels, 3,438 feet; surface drilling 17 holes, 7,739 feet and underground drilling 21 holes, 4,630 feet.

1925-29 Ankerite Gold Mines Ltd. operated the mine and a mill of 250 tons per day. Total development on the property: shafts 1,302 feet; lateral work 12,696 feet.

Lateral work 1,254 feet; drilling 2,421 feet and mill operated by Ankerite Gold Mines Syndicate.

1932: Development, mining and milling continued by Buffalo Ankerite Gold Mines Ltd.

The adjoining March (Marbuan) Mine was taken over and the two mines operated as one property by Buffalo Ankerite Gold Mines Ltd. Ankerite No. 1 shaft 367 feet, Ankerite No. 2 shaft 1,200 feet; Ankerite No. 5 (main) shaft 3,996 feet; No. 8 (Imperial) shaft 109 feet; 27 levels with the deepest at 3,750 feet, drifting 63,000 feet approximately, crosscutting 47,000 feet approximately, mill capacity 400 tons per day.

The Buffalo Ankerite mine closed. The mine operated from 1926 to 1953 and produced approximately five million tons of ore at a recovered grade of 0.193 oz Au/ ton, by underground stoping methods, yielding 983,885 ounces of gold. From 1936 to 1953, production from Buffalo-Ankerite South was included in the total.

Buffalo Ankerite South

Claim HR833 (P8276) two shafts each 50 feet deep by Maidens Macdonald.

1916-17: Claim HR833 vertical shaft deepened to 107 feet and shaft inclined 65˚, deepened to 100 feet by LaRose Mines Limited under an option from Coniagas Mines Ltd.

1921-25: March No. 1 shaft claim HR823 (P7955) was sunk 800 feet with levels at 100 and 321 feet, surface drilling 2 holes 2,260 feet; work by March Gold Mines Ltd.

1917-26: Claim HR833 March No.2 inclined shaft deepened to 190 feet, March No.3 shaft deepened to 330 feet, another shaft to 115 feet by Porcupine Gold fields Development and March Gold Mines Ltd.

1926-32: Mill of 150 tons per day in operation; March No. 3 shaft deepened to 425 feet, South Winze (No. 4) from 425 to 675 feet, levels 170, 200, 300, 425, 550 and 675 feet; work by March Gold Mines Ltd.

1933-34: Mill operated, March South Winze deepened to 1,050 feet, levels at 800, 925 and 1,050 feet; work by Marbuan Gold Mines Ltd.

1935-53: Property consolidated with Buffalo Ankerite Gold Mines Limited. No. 6 Winze extends from 1,050 to 2,020 feet with levels at 1,250, 1,400, 1,550, 1,700, 1,850 and 2,000 feet. The production shaft for both mines was Buffalo Ankerite No. 5 shaft with the mines connected by haulage drives on the 1,050- and 2,000-foot levels.

Historical Drilling

In 2002 Placer Dome/Porcupine Joint Venture (PJV) optioned the Buffalo Ankerite and Fuller properties from Vedron Gold Corp, a precursor company to Lexam VG Gold Inc, and undertook exploration programs on the project area.

Drilling at the Buffalo Ankerite North zone by Placer Dome totalled 15 holes, 8,949.5 feet. Drilling by Lexam is reported under the section on “Drilling”. Placer Dome drilling on the South Zone consisted of fifty-nine holes, drilled in a two-phase effort by Placer Dome. Phase One consisted of 26 holes totalling 7,549 feet. Phase Two drilled footage was approximately 12,455 feet in 33 holes. Core logging was done directly into Placer Dome’s RDBMS Sample Management System/Laboratory Information Management System (SMS/LIMS). Drill hole information was then merged with the main data set in the master Access database. Drill hole data was then accessed in Vulcan through an ODBC Link.

Drill hole collar locations were spotted on a local grid by measurements from nearest grid markers. After completion of drilling, hole collars were surveyed using differential GPS for accurate locations.

As this model covers an area of historic mining activities, some drill holes broke through into previously mined (stopes/drifts) areas. Where possible, drill rods were pushed through these voids, and drilling continued.

Core samples from Phase One were cut by rock saw with half of the core being sent to the Dome Mine Lab of South Porcupine for assaying. The remaining half core is stored on the Dome Mine property. Phase Two drilling was whole core sampled and sent to the Dome Mine Lab.

Samples were analyzed by Fire Assay with an Atomic Absorption Finish. A 30 gram assay charge was used for the analysis. All diamond drill core assays are reported electronically with SMS/LIMS and worksheets remain in a separate folder in the Hoyle Pond administration office. All samples with results greater than 13.7 g/t Au were re-assayed in duplicate using the gravimetric method of determination.

The internal laboratory quality control program, which consists of one standard and one duplicate sample per board of 20 samples, was consistently used with no quality control issues arising.

Production

Years	**Gold**	**Silver**	**Ore Milled**
Ounces	**Ounces**	**Tons**	
1925-35	61,039	5,400	317,769
1936-53	Production included with Ankerite Mine North		
ODM Statistical Files, March (Marbuan Gold Mines Ltd.)			

Total production from Buffalo Ankerite South is believed to be in the order of 500,000 tons, yielding 100,000 oz gold.

In the 1970s, Pamour Mines open pit mined a portion of the crown pillars of the Buffalo Ankerite South deposit. Though no production records could be found, volumes developed by 3-dimensional models by Dome Mines suggests some 350,000 tons of rock were excavated by Pamour.

6.1.2 Historical Mineral Resource Estimates

An estimate of the Buffalo-Ankerite South deposit carried out for Placer Dome by R. Calhoun in July 2002 reports a mineralized zone containing up to 120,000 ounces of gold. Tonnage appears to be derived from polygons (length x average width x height ÷ tonnage factor) and the grade of 0.095 oz Au/ ton appears to be the average grade across the entire mineralized domain (Placer Dome created three mineralized domain solids – Tourmaline Breccia Zone, North Tourmaline Breccia Zone and Conglomerate).

A resource model for the Buffalo Ankerite South was developed by the Resource Evaluation Group of the PJV. A rotated block model was used, at 065° azimuth, following principal lithological and mineralization boundaries. Following a first pass three dimensional modeling campaign, the PJV re-visited the project in August, 2002. It was stated that insufficient drill data was available to generate a resource model. A composite average grade based on the modelled mineralized domains yielded 1.4 million tons at a grade of 0.07 oz Au/ ton for 98,000 ounces contained to a depth of 500 feet. This estimate crudely removed unmodelled underground workings by subtracting 1.0 million tons of material (Parolin, 2003).

Despite the above, on further review, Placer Dome carried out a geostatistical estimate of the resource at Buffalo Ankerite South and arrived at the following results:

At a 0.025 oz Au/ ton cut-off grade:

- Measured: 855,035 tons grading 0.080 oz Au/ ton containing 68,830 oz Au
- Indicated: 190,346 tons grading 0.075 oz Au/ ton containing 14,276 oz Au
- Inferred: 80,334 tons grading 0.058 oz Au/ ton containing 4,659 oz Au

And at a 0.05 oz Au/ ton cut-off grade:

- Measured: 517,673 tons grading 0.109 oz Au/ ton containing 56,426 oz Au
- Indicated: 97,522 tons grading 0.112 oz Au/ ton containing 10,971 oz Au
- Inferred: 33.307 tons grading 0.088 oz Au/ ton containing 2,931 oz Au

Estimates were carried out using Ordinary Kriging.

The authors are concerned that Placer Dome elected to put in a measured category in their resource estimate. The database comprises 59 diamond drill holes only and does not have the benefit of sampled underground levels and raises. Also, some of the drill intersections are immediately adjacent to underground openings – drifts, raises or stopes. As such, there is speculation about these mineralized blocks, whether they would be available or are in fact present.

The authors therefore discount these estimates as they would not comply with NI 43-101 rules and regulations.

6.2 FULLER

The current Vedron property has a long history of exploration and limited gold production. Since 1910. periodic surface exploration. underground and limited production has been carried out on the Fuller deposit on the Fuller claim. A 228-ft inclined shaft (the "Edwards shaft") was sunk sometime after 1924. In 1940-1941 Nakhodas Mining Company Limited ("Nakhodas") developed 2.100 ft of lateral workings at the 160 ft level and mined 44,028 tons grading 0.149 opt Au (recovered) above the level in four stopes. The rock was processed in the Faymar mill in Deloro Township (Ferguson, 1968). The mill was destroyed by fire in 1942, and was not rebuilt. In addition to 83 underground drillholes, Nakhodas also did at least 10,919 ft of surface drilling in 41 drillholes (S 1 to S41).

In the late 1930's, Hollinger Consolidated Gold Mines conducted a 14,000 ft drilling program in 25 holes on the claims in Tisdale Township, previously referred to as the "Tisdale Ankerite claims" (Ferguson and others, 1968). Drilling was also done by Pamour (Noranda Explorations Ltd.) from 1974 to 1976. One of these drillholes deviated onto the (then) single Vedron claim, and intersected three well-mineralized zones at a depth of about 500 ft, encouraging Vedron to drill beneath the 160 ft level (Vedron, 1979).

6.2.1 1983-1998

In 1983, Vedron did more than 4,000 ft of drilling in 15 holes to test beneath the Fuller workings.

In the period 1986 to 1989, Belmoral Mines Ltd. (Belmoral), under an agreement with Vedron, put in 4,611 ft of ramp-decline, established five levels to 650 ft below surface (150, 275, 375, 500 and 650 ft levels), opened the levels with 5,362 ft of drifts, and also established 3,505 ft of workings such as crosscuts and raises. In addition, more than 108,000 ft of core was drilled from surface and underground and metallurgical tests were completed. Resources were estimated near the end of this campaign, and a mining plan was proposed on behalf of Vedron (McKay, 1988). Other studies were carried out, including the review of old Buffalo Ankerite mine data and the drill testing of an IP anomaly on the northern part of the property.

The property lay dormant from early 1989 until the second half of 1996, when Vedron initiated a drilling program to explore the down-dip extensions of the mineralization of the Fuller Deposit beneath the 650 ft level to the depth of the upper Buffalo Ankerite workings (1,550 ft level). Vedron completed 105.047 ft of drilling in 63 holes by September 1, 1997. The drillholes intercepted the Fuller Deposit, which consists of a number of sub-parallel mineralized zones, along a strike length of approximately 2,400-ft, and to depths slightly in excess of the 1,550 ft level. A second phase of drilling (holes VG 97-64 to VG 98-83) to outline resources between the 1,550-1t and 2,550-1t levels, was completed during 1997-98 but was not included in Bevan's 1997 resource estimate. In 1996 and 1997 backhoe trenching programs were done to expose mineralization for the first time.

Limited trenching was also completed north of the Fuller Deposit in an attempt to expose mineralization in the new northern part of the property. IP and magnetic surveys from the Buffalo Ankerite north shaft to the northern part of the property were completed for Vedron in early 1997 by Exsics Exploration Limited.

Vedron instituted a program of computerizing historical and recent exploration data that included the digitizing of all available level plans and drillhole data prior to the 1996-1998 drilling campaigns. Completion of the project was stopped when the Property was optioned to Placer Dome in 2001.

6.2.2 2002

Upon optioning the property, Placer Dome carried out both field exploration and office database management activities on the Fuller Property commencing in early 2002. All of the underground and surface diamond drillhole logs available from the Fuller deposit were re-coded to Dome Mine terminology, which included lithology, assays, alteration, percent sulphides and percent veining. The downhole surveys and X, Y, Z drillhole collar co-ordinates for these holes were converted to the Dome Mine co-ordinate system. A re-interpretation of the Fuller Dobie Deposit consisting of re-modeling of the lithology and gold mineralization was then carried out by Placer Dome in April and May 2002. A final 3-D lithology and mineralization domain model was constructed.

The 3-D mineralization domain model was created utilizing percent quartz veining and percent pyrite content from the drill hole database. Modelling was done primarily on the underground level plans, utilizing the historic data, at an approximate 30 metres (100 ft), elevation spacing. This data currently remains proprietary to Placer Dome.

Based on their re-interpretation of the Fuller property, Placer carried out a surface diamond drill program on the North Vedron Zone between January 16th and February 19th, 2002. The four hole, 2,050.4 metres (6,727 It) drill program was designed to test for gold mineralization along the extension of the highly prospective Central and Vipond Series contact on the Fuller property.

The Paymaster 31 Zone Extension surface diamond drill program was carried out by Placer between April 7th and May 10th, 2002. The five hole, 2,032.7 metres (6,669.0 ft) drill program was designed to test: 1) a porphyry trend with associated gold values intersected in hole VG 98-82, and 2) the western strike projection of the 31 Vein structure.

6.2.3 Historical Mineral Resource Estimates

Peter Bevan estimated the Fuller resource in 1997. At 0.075 opt Au cut-off, Bevan estimated an un-capped resource of 2,207,260 tons at 0.260 opt Au, equivalent to 574,016 ounces of Au, and capped resource at 1 opt, 2,207,260 tons at 0.206 opt - 453,599 ounces Au (Bevan, 97). The historic estimates have not been reported under the guidelines of National Instrument 43-101 and as such are order of magnitude resource figures and are not classified resources. The authors therefore discount these estimates as they would not comply with NI 43-101 rules and regulations.

6.3 PAYMASTER

The discovery of the Dome Mine in 1909 resulted in the property adjacent to the Dome being staked and extensively explored. A spectacular discovery of gold was made on claim HR 908 in 1910. From 1910 to 1924 mining was conducted from the 1, 2 and 3 shafts with workings extending down to the 800 foot level. Lateral work was carried out on 6 levels. Several stocks of porphyry occur on the property, dipping north and plunging east. Considerable lateral work was done in the porphyry body which contained quartz veinlets and disseminated pyrite, both carrying gold values.

1910: Claims TRS776 (HR908), TRS975 (ME15), shaft 83 feet, 40 feet of crosscutting by Standard Gold Mines Ltd.

1910-11: No. l shaft 123 feet, No. 2 shaft 28 feet, No. 3 shaft 114 feet, No. 4 shaft 76 feet, 204 feet of drifting on 105-foot level; No. l shaft by West Dome Mines Ltd.

1915-28: No. l (Paymaster No. 6) shaft 1,350 feet, No. 2 shaft 30 feet, No. 3 shaft 595 feet, No. 4 shaft 113 feet. At No. l shaft levels at 120, 180, 300, 400, 500, 600, 750, 900, 1,050, 1,200, 1,325 feet and 18,866 feet of drifting and 7,365 feet of crosscutting. Mill operated

1915-1930: work by Consolidated West Dome Mines Ltd. and West Dome Lake Gold Mines Ltd.

1930-66: No. 5 (Main) shaft 4,462 feet, No. 2 winze 2,046 feet to 4,202 feet, No. 6 winze 4,059 feet to 6,157 feet, 9 shafts, 6 winzes, 197,294 feet of drifting, 82,577 feet of crosscutting, mill treated 365 tons per day; work by Paymaster Consolidated Mines Ltd. and Porcupine Paymaster Ltd. The mine closed April 1966.

Production

Years	Gold	Silver	Total Value	Ore Milled
ounces	ounces	dollars	dollars	tons
1915-66	1,192,206	325,088	42,146,614	5,607,402

(ODM Statistical Files, Dome Lake (West Dome Lake, Consolidated West Dome Lake), Paymaster (Paymaster Consolidated Mines Ltd., Porcupine Paymaster Ltd.).

The present day Paymaster property was formed with the amalgamation of several claim groups in 1930 by Paymaster Consolidated Mines Ltd.

The property was acquired by Placer Dome in 1989.

In 1989 Placer conducted surface mapping, lithogeochemistry, magnetic survey, power stripping and channel sampling.

The historical gold production and tonnage of the Paymaster property by its various former owners was 1,192,206 oz gold from 5,607,402 tons mined, resulting in an overall grade of 0.211 oz/ton. A total of 19 shafts (many of which were shallow exploration shafts) and 6 winzes were developed in the extraction of the ore. The main production shaft was the #5 Shaft with the #6 Shaft playing a subordinate role in the development of the northern portion of the property. Shafts #2 and #3 developed the south central portion of the property. Shaft #4 was relatively shallow developing a fuchsitic carbonate altered zone in the south-eastern portion of the property.

The main producing area of the Paymaster is associated with the "Paymaster" porphyry stock and other small porphyry bodies to the north and northwest with quartz ankerite veins occurring to the north, west and southwest of the porphyry. In general the tenure of Au in the quartz ankerite veins appears to increase with increased silicification and quartz impregnation partially replacing the ankerite. North-south white quartz veins are barren. The #10 vein is on the north porphyry contact on the upper levels, dipping 65 degrees away from the porphyry at depth. There is some folding and faulting along this vein. The #1 vein lies north of #10 vein with the two veins joining to the west. This vein extends west through a barren section and is again productive as #5 Shaft, #14 vein. The #7 vein is at the top of the Key flow (sheared variolitic flow top breccia). The #22 vein is on the strike of the #7 vein to the west, but is in a younger flow due to a drag fold. The large #20 or Kurts vein to the northwest contains erratic gold values but has not produced any ore. In the #5 Shaft area, #3-24 vein, lying near the bottom of the #99 flow (chloritized, lightly to moderately foliated uniformed textured basalt) has been a substantial producer The #8 vein lies southwest (SW) of the porphyry with the fracture entering the porphyry. The #8 vein terminates at the #18 vein which follows a strong fault.

The #36 ore zone, lies in volcanics just north of the talc-chlorite carbonate fault zone. This zone is separated from the large ultramafic intrusive sill (or flows) by 30 feet of basalt. There is one narrow quartz carbonate vein in the ore zone and only a few stringers. The sulphide content is as high as 10% combined pyrrhotite, pyrite and locally chalcopyrite. The ore zone extends east, and a short distance west, from a tongue of the Edward Shaft Porphyry. The 36 ore body is the down plunge extension of the Fuller ore body on the VG Gold Fuller property to the west.

Paymaster 2/3 Shaft Area

In 1995 Placer drilled 47 holes to outline a near surface resource.

In 1994-95 a preliminary pit resource estimate was completed – 1,327,215 tons at 0.066 opt to a depth of 500 feet = 87,600 ounces Au.

In 1996 Placer drilled 28 holes along the ultramafic / mafic contact south of the 2/3 shaft pit outline.

The West Porphyries were also mined from both the Preston and Dome mines. Production was mainly from narrow east/west trending, steeply dipping, shear hosted quartz veins.

The "Porphyry Greenstone" mineralization is associated with the fringes of Porphyry bodies located immediately south/south west of the Preston Porphyry. Mineralization consists of strong alteration of the greenstone that may make it difficult to distinguish greenstone from porphyry. Veining is not always present.

Paymaster 4 Shaft Area

In 1924-1925 United Mineral Lands sank a 253 ft shaft and developed on levels at 116 ft and 232 ft. No records of stope development or underground sampling are available. Gold values were reported to occur in a fuchsite-carbonate zone with several small porphyries intruding the zone. Selected historical drill results include 0.27 oz/t over 11.5 ft, 1.37 oz/t over 3 ft, and 0.26 oz/t over 6 ft. Results from underground sampling of drift zones include 80 ft X 5 ft of 0.217 oz/t, 35 ft X 5 ft of 0.06 oz/t, and 60 ft X 5 ft of 0.145 oz/t. Results from surface sampling of the zones include 70 ft X 10 ft of 0.254 oz/t and 24 ft X 2.1 ft of 0.300 oz/t.

The property was later acquired by Paymaster Consolidated Mines and data compilation was done. The only drill logs available on file at the Dome Mine are on Paymaster letterhead. The original drill core is therefore interpreted to have been re-logged.

The property was acquired by Placer Dome in 1989. In 1989, Dome Exploration conducted a small surface mapping, power stripping and channel sampling program. A total of four areas were power stripped. The best values from channel sampling were from a trench at the Preston claim boundary: 0.076 oz/t over 20.5 ft (including 0.442 oz/t over 2.6 ft and 0.129 oz/t over 2.2 ft). A 1990 program of additional power stripping followed by shallow diamond drilling on 100 ft centers with pierce points 50 ft below surface was proposed, but never implemented.

Placer Dome conducted a two phase diamond drill program totaling 13,236 ft in 17 holes in 1999-2000. The 1999 fall drill program was designed as an exploration phase to test the Paymaster 4 Shaft Carb Rock-Highly Altered Rock lithologies and coincident resistivity-high geophysical feature on approximately 400 ft centers. The 2000 winter drill program was designed to: 1) follow-up on a significant gold intercept; 2) test the north-eastern and south-western strike extents of the Carb Rock–Highly Altered Rock package and coincident resistivity-high geophysical feature, and 3) test a number of magnetic low features interpreted to represent potential structural-alteration zones.

6.3.1 Historical Mineral Resource Estimates

The resources reported here are historic in nature and should not be relied upon as their accuracy has not been verified by a Qualified Person. The company is not treating the historic estimates as current resources as defined by NI43-101. The resource estimates were conducted prior to the introduction of NI 43-101, but were carried out in accordance with established practise at that time. The historic classes used differ from current CIM classes however the estimates were conducted in a professional manner and might be comparable to the CIM indicated resource class.

Historic resource calculations on the Paymaster 2/3 shaft area or the West Porphyry were calculated by Placer Dome in 1994 and 1996. The resource calculations employed both polygonal and block model methods.

The resource estimates were carried out by both polygonal (sections and plans) and block modeling techniques using the inverse distance (ID) method.

The cut-off value used for both methods was 0.96 gram Au per tonne (0.028 oz. / ton)

Based on experience at the Dome Mine a density of 2.79 (tonnage factor of 11.5, or 0.087 tons per cubic foot) was utilized.

TABLE 6.1
RESULTS

Author	Year	Method	Tonnes	gpt	Total gm	Total oz.
Skeries	1994	polygonal	957,327	2.34	2,240,145	72,023
D.Hunt	1996	Inverse distance	1,053,916	2.5	2,634,790	84,712

Usually the block model produces a higher overall tonnage at a lower grade as compared with polygonal estimations. As can be seen this is not the case here. The anisotropic search ellipse with axes 15.2 m (x-axis) X 4.6 m (y-axis) X 18.3 m (z-axis) resulted in both increased grade and tonnage. The orientation and dimensions of this search ellipsoid approximated the major orientation of gold mineralized zones in the Main Porphyry.

The resources are historic in nature and therefore not NI43-101 compliant, however both of these resources were conducted in a professional manner by competent geologists and the author judges them to be reliable. The resource estimates would correspond to an indicated resource.

6.4 DAVIDSON TISDALE

The property was incorporated as Davidson Gold Mines Limited in 1911 land was succeeded in 1919 by Davidson Consolidated Gold Mines Limited. In 1921 Porcupine Davidson Mines Limited was formed as a 50/50 joint venture with British interests. Due to disagreement between the partners, prolonged litigation resulted in the property being tied up for several years. In 1925 control reverted to Davidson Consolidated Mines Limited who then sold the mineral rights to Mining Contracting and Supply Company (Ventures Limited - the forerunner of Falconbridge). In 1945 Ventures sold the rights to Davidson Tisdale Mines Limited and, though various joint ventures have been undertaken with several parties over the years, title has remained with the Company.

6.4.1 Work History 1911-1924

Between 1911 and 1924 exploration on the property comprised surface drilling and underground development through a small exploration shaft. Thirteen surface holes totaling 4,070 m were drilled and a 2-compartment vertical shaft (Main Shaft) was sunk to a depth of 95 m. Levels were established at 30 m, 60 m and 90 m with approximately 700 m of lateral development. An internal vertical winze was sunk a further 67 m from the 90 m Level with some 490 m of drifts and cross-cuts on the 3 new levels developed. A limited amount of underground drilling was carried out.

In 1918 electricity was brought to the site and a 10- stamp mill operated at 30 tons per day till it burned down in 1924. A reported total of 8501 tonnes @ 8.9 g Au/t was milled and 2,438 ounces of gold recovered using mercury amalgamation. It is noted that about 20% of the gold content was lost using this process.

In 1923/24 the 3-compartment Horseshoe Shaft was sunk at a site 180 m west of the Main Shaft. Inclined at 72 degrees to the northwest, the shaft was intended to go to a depth of 300 m in order to develop the deeper gold-bearing veins encountered by surface diamond drilling. Due to withdrawal of the British financial backers in late 1924, the shaft stopped at 247 m and stations were established at 60 m, 120 m, and 167 m along the incline.

Ventures (1933-1945) - Between 1933 and 1945 Ventures drilled 11 holes into and below the old workings in an attempt to locate vein extensions and to verify the high-grade results reported from previous drilling. They drilled a total of 1,557 m but the results did not meet their requirements and they returned the property to Davidson Tisdale Mines.

1945-1981 A report by Ed Hart in 1977/8 indicates that the tonnage and grade are understated while Kirwan reported positively on the property.

Dome Mines -1981 In 1981 Dome Mines drilled 10 holes (1,118 m) with only one deep hole in the vicinity of the old workings. Dome regarded the old mine as exhausted and quoted the results of Ventures' underground sampling. Kirwan notes that Ventures had NOT done any underground sampling as they had not dewatered the mine. In fact the old workings remained flooded from 1924 - 1983. Dome drilled an 11th hole to test the strike continuity based on "the old 70°:70° model" i.e. a mineralized zone dipping at 70° and striking 070°.

6.4.2 Modern Exploration

The period from 1983 to 1987 witnessed the most extensive and integrated exploration of the property. Efforts were concentrated in the known areas of old showings and workings, with very little property-wide exploration. It was during this period that resources and reserves were developed and the potential of the property quantified for the first time.

Work History 1983-1987

In early 1983 a new group assumed control of Davidson Tisdale Mines Limited and an extensive surface and underground exploration program was carried out. New grids were established, ground geophysical surveys (magnetics, VLF-EM, Maxmin II HEM and Pulse EM) were completed A thorough compilation of all available data was completed by Kirwan who recommended an extensive, though flexible, program.

The following program was completed during 1983:

- Extensive stripping in the Main Shaft area uncovered numerous occurrences of free gold (VG) over an area greater than 600 ft long. , Smith Vet & South Shaft areas were stripped but not mapped while trenching and stripping at Cal's Dome showed high gold values in quartz veins in sediments. Kirwan notes that VLF surveys show this sedimentary horizon to extend across the property.
- Stripping uncovered "an exciting occurrence" at the intersection of NW and NE trending quartz vein systems (the T-Zone). Gold occurs in thin quartz veins underlain by highly carbonated volcanics with VG.
- Extensive percussion drill sampling was carried out in the Main Shaft and T-Zone areas to test for open pit potential.
- Twenty-three holes comprising 2,125 m were drilled in the Main Shaft area.
- The underground workings were de-watered and rehabilitated followed by extensive sampling, assaying and geological mapping. No underground drilling was undertaken.

One of the most significant conclusions from the 1983 program was the demonstration that the major vein system in the Main Shaft area strike at 030° with a 45° north-westerly dip rather than the 70° striking and 70° dipping structure as previously thought and which had guided previous exploration.

During early 1984, 11 drillholes (2,080 m) were completed in the vicinity of the Main Shaft area and some underground mapping and sampling was completed. Getty Canadian Metals Limited became operator of the project on March 1, 1984. Getty Canadian Metals (1984-1987) is a subsidiary of Getty Oil of the USA.

Getty 1984 Program

The stated objectives of the 1984 program were as follows:

- To indicate by drilling the tonnage potential of the known quartz vein zones
- To establish the inferred continuation of these systems to the southwest along a strike length of 700 m and to a maximum depth of 230 m.
- To assess the potential for a 1 to 3 million ton deposit
- To explore for additional vein systems
- To outline sufficient tonnage to justify an underground exploration and development program

The program achieved the following:

Main Shaft Zone

- Drill testing the Main Shaft Zone on 50m centres and to a depth of 250 m between the Main Shaft and the S-Zone, a distance of 450 m. In-fill drilling at 25 m centres was completed in selected areas of the main Zone.
- Two en echelon auriferous vein systems striking 030° and dipping at 30 to 45° NW were identified.
- 45% of the drillholes encountered VG, with 45% of the holes returning 1.7 g Au/t or greater over the full width of the vein system.

Smith Vet-T Zone

- Exploration for 400 m to west of Smith Vet-T Zone to a vertical depth between 50 and 200 m.
- At least 2 parallel vein systems identified with the main auriferous structure (S-Zone) striking at 090° and dipping north at 25°.
- Limited in-fill drilling on 25 m centres was completed.
- 36% of the holes intersecting the S- Zone quartz vein system encountered VG with 25% of the holes returning 1.7 g Au/t or greater over the full width of the vein.

The status at the end of 1984 was as follows:

An in-house ore reserve calculation by Getty for the main Shaft and South Zone demonstrated 747,600 tonnes in the drill indicated category. The average uncut and in-situ grade averaged 12.39 g Au/t over an average true vein width of 3 m and to a depth of approximately 200 m.
Potential to significantly increase the reserves was identified.

Getty also identified open pit potential for the S-Zone. (It should be noted that the parameters used by Getty in their definition of "Reserves" have not been stated. A more appropriate wording may be to regard these as resources as the drill spacing of 25 m may be too wide for this type of deposit)

Getty 1985 Program

This program comprised 2 phases. Phase 1 program was to evaluate the potential for near-surface open pittable reserves in the S-Zone in the Smith Vet- T Zone area, while Phase 2 involved mining a bulk sample from underground to validate the drill-indicated reserves between the 4th and 5 Levels.

1985 Program-Phase 1

This program comprised 835 m in ten diamond drill holes. Though the vein system was encountered where anticipated, the lack of significant assays together with budget constraints caused the program to be terminated and efforts to be focused on the Phase 2 underground program.

1985 Program - Phase 2

This program consisted of 4 surface and 8 underground pilot core holes (761 m), site preparation, headframe installation and underground rehabilitation. Ninety-seven metres of crosscutting and 53 m of raises were completed and a 2,885 tonnes bulk sample was obtained. Systematic chip and muck sampling comprised approximately 4,000 samples. An important part of this program was the comparison of grade as indicated from drillholes with that achieved from the various sampling methods employed.

The principle conclusions from this program were:

- The quartz vein systems are very irregular and erratically mineralized. The vein systems are complex rather than being a simple sheet-type system.
- Comparison of assays from drill core with those from various sampling experiments underground suggest that whole core rather than split core should be sent for assay.
- Cutting individual assays > 34.28 g in drill core is indicated.
- Muck, panel and channel samples correlate very well with sample tower results, suggesting that these be used for grade estimation.

Getty Program - 1986/87 - Phase 1 Underground Program

The main component of this program was a bulk sample from the 4th Level to test the Lower Vein System. A total of 7,270 tonnes was mined and some 6,970 tonnes, of which 1,750 tonnes were classified as waste, were brought to the surface. Though the bulk of the material (75%) was mined on the 4th Level, mineralization was also recovered from the 3rd and 5th Levels. An additional 55 short diamond drillholes were completed for 1,337 m and the excavations mapped. The material was panel and muck sampled.

As slashing began on the 4th level it became apparent that the high-grade areas were visually identifiable so that the mining of this sample was effectively under geological control. Material was divided into stockpiles of varying grades as determined using chip samples from underground on one-metre squares and surface grab samples.

TABLE 6.2
ASSAYS OF DIFFERENT STOCKPILES

Pile#	Tonnes	Grade		Gram x Tonnes		Grade g Au/t	
		Uncut	Cut	Uncut	Cut	Uncut	Cut
1	3,313.4	17.63	8.58	58,408	28,426		
2	740.0	4.60	4.60	3,404	3,404		
3	1,200.0	2.09	2.09	2,508	2,508		
	5,253.4			64,320	34,338		
Total	**5,253.4**					**12.24**	**6.54**

Underground panel sampling completed subsequent to the mining indicated a grade of 7.31 g Au/t for the area of bulk sample.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Lexam Timmins project is situated in the south-western part of the Abitibi Greenstone Belt within the Archean Superior Province (Figure 7.1). The geology of the Timmins Camp comprises a thick sequence of Archean volcanic and sedimentary rocks that have been intruded by synvolcanic and post tectonic felsic dykes (Figure 7.1). The volcanic-sedimentary sequence has been subdivided into three main groups, the Deloro, Tisdale and Porcupine Groups. In the Timmins area, the greenstone belt is subdivided by Brisbin (1997) into four volcano-sedimentary groups. From oldest to youngest they comprise the Deloro Group, Tisdale Group, Hoyle Group and the Timiskaming Group. The South Tisdale Anticline (STA) area (see Figure 7.1) is underlain predominantly by volcanic rocks of the Tisdale Group. The Tisdale Group has been subdivided by Brisbin (1997) into four formations, from youngest to oldest these are: 1) the Hersey Lake Formation comprised of ultramafic and less common mafic flows; 2) the Central Formation consisting of mafic flows; 3) the Vipond Formation, and 4) the Gold Centre Formation.

The lowermost Deloro Group comprises mafic to ultramafic flows overlain by a series of pyroclastic rocks and with a well-developed regional iron formation near the top. No significant gold production is associated with the Deloro Group in the Timmins Camp.

The overlying Tisdale Group comprises ultramafic volcanic flows in the lowermost formation overlain by a sequence of high iron basaltic flows containing a number of carbonatized sedimentary units. The top of the Group comprises felsic pyroclastics. This Group hosts the majority of gold production in the Timmins Camp.

The youngest rocks in the Camp are the clastic sedimentary rocks of the Porcupine Group. In addition to overlying the volcanics these sediments may be laterally equivalent to the volcanics and distal to the major centres of volcanism.

A number of quartz-feldspar porphyry intrusions have been emplaced into the supracrustal rocks (see Figure 7.1). In the south-western part of the Timmins gold camp, gold deposits and occurrences are often spatially associated with porphyry intrusions. Examples include the Pearl Lake Porphyry at the Hollinger-McIntyre deposit, Preston Porphyry at the Dome-Preston deposit, 2&3 Shaft porphyries, West Porphyry and the Edwards Porphyry at the Fuller/Dobie deposit.

The Timmins gold camp exhibits a complex structural pattern with at least three major periods of deformation being recognized. This tectonic activity has resulted in a series of doubly plunging, upright, isoclinal folds which are offset by major fault structures and related secondary faults.

The Porcupine-Destor fault is a major structural zone which can probably be traced for a total distance of 440 km. It is at least 150 m wide, shows evidence of left-lateral displacement, and is offset by younger, brittle faults. On the north side of the fault zone, there have been at least two periods of folding, consisting of a north trending series of folds and folds with an east-northeast axis such as the Porcupine syncline. Hattori and Hodgson (1991) have interpreted five periods of deformation; the Porcupine Destor fault zone and the Porcupine syncline are thought to be relatively old.

The Porcupine syncline separates the camp into a north limb and a south limb. The south limb of the sycline includes three Lexam properties, Fuller, Buffalo Ankerite and Paymaster. The north limb includes the Davidson Tisdale property.

The key structures in the area include: 1) the northeast-southwest trending regional Destor-Porcupine Fault Zone that lies immediately south of the project area; 2) the northwest trending South Tisdale antiformal structure that is defined by the folded mafic-ultramafic contact in the vicinity of the Edwards Porphyry, and 3) the mafic-ultramafic contact, also referred to as the Paymaster Shear (Spracklin, 2001). Spracklin (2001) has traced the Paymaster Shear from the Aunor-Delnite area in the west to the Burrows-Benedict Fault in the east, which offsets the shear. Most of the gold mineralization in the south-western part of the Timmins area, apart from the Hollinger-McIntyre deposit, occurs at or near the mafic-ultramafic contact or Paymaster Shear. Examples include the Dome-Preston deposit, Paymaster 36 Zone, Fuller/Dobie deposit, the Buffalo-Ankerite North and South Mines, Aunor Mine and the Delnite Mine.

The following are the main characteristics of gold occurrence in the Timmins Camp:

- The dominant source of gold is within quartz vein lodes containing locally spectacular free gold;
- The quartz vein lode deposits are structurally controlled areas of dilatancy which permitted the development of vein zones;
- The majority of gold production in the Camp is hosted by rocks of the Tisdale Group;
- Some gold production is hosted in pyrite-bearing pyroclastics within the mafic volcanics of the Tisdale Group;
- Some production comes from quartz veins within the sediments of the Porcupine Group where they unconformably overlie productive portions of the Tisdale Group.

The bulk of the more than 62 million ounces of gold production in the Porcupine district in the Timmins area (Atkinson et al. 2005) has been from well-known deposits in Tisdale Township north of the Porcupine - Destor fault zone (see Figure 7.1). These include the producing Dome Mine (more than 13 million ounces Au) and the past-producing Hollinger (19.35 million ounces Au), McIntyre (10.36 million ounces Au), Coniaurum (1.11 million ounces Au), Paymaster (1.19 million ounces Au) and Preston East Dome mines (1.54 million ounces Au). In the northern part of Deloro Township are the former Buffalo Ankerite, Aunor (2.11 million ounces Au) and Delnite (0.92 million ounces Au) mines, as well as one of the Paymaster shafts on the Placer Dome property. Most of the production has been from deposits which occur within or immediately adjacent to the Tisdale group. The average historical underground recovered grade of Timmins area deposits has been about 0.25 oz Au/ ton. The Dome mine is completed production from an open pit (the "superpit") at a grade of about oz Au/ ton, and is presently producing from underground operations at a grade of about 0.13 oz Au/ ton (Goldcorp Website).

7.2 DETAILED GEOLOGY OF THE BUFFALO ANKERITE, FULLER AND PAYMASTER PROPERTIES

7.2.1 Geology

The contiguous Properties lie within the South Tisdale Anticline (STA) and are underlain by a sequence of un-subdivided ultramafic and mafic flows of the Hershey Lake Formation and locally subdivided mafic flows (flows C7-C17 of Longley, 1959) of the Central Formation (Figure 7.1). The contact between these two formations, referred to here as the north mafic-ultramafic contact or Paymaster Shear, is a major structure based upon its discordant nature observed both on surface and 3-D modeling (Pope, 2000).

General Setting

The properties form part of a continuous sequence of mines that extend from the Dome Mine in the east to the Delnite Mine in the west.

Stratigraphy

The mineralization is located primarily within a narrow pillowed mafic volcanic flow unit of the Central Series, Tisdale Assemblage. The volcanic rocks are complexly folded around the South Tisdale Anticline and Kayorum Syncline resulting in an S-shaped flexure in the stratigraphy. The pillowed mafic volcanic unit, which hosts the main mineralized domains of the Buffalo Ankerite South property, is flanked to the north and south by Hershey Lake Series ultramafic flow units. In the area of the property, the volcanic flows strike between 065 and 070°, and dip at approximately 60° to the north and thicken to the west. A discontinuous conglomerate unit is located along the contact between a flow textured mafic volcanic unit and the south ultramafic unit. The conglomerate is interpreted as Timiskaming in age containing mainly bleached mafic volcanic clasts with occasional porphyry and ultramafic clasts and typically follows this contact and is similarly oriented for dip. Quartz-feldspar porphyries intrude the volcanic units and late northwest-trending diabase dykes cut all the above rock types.

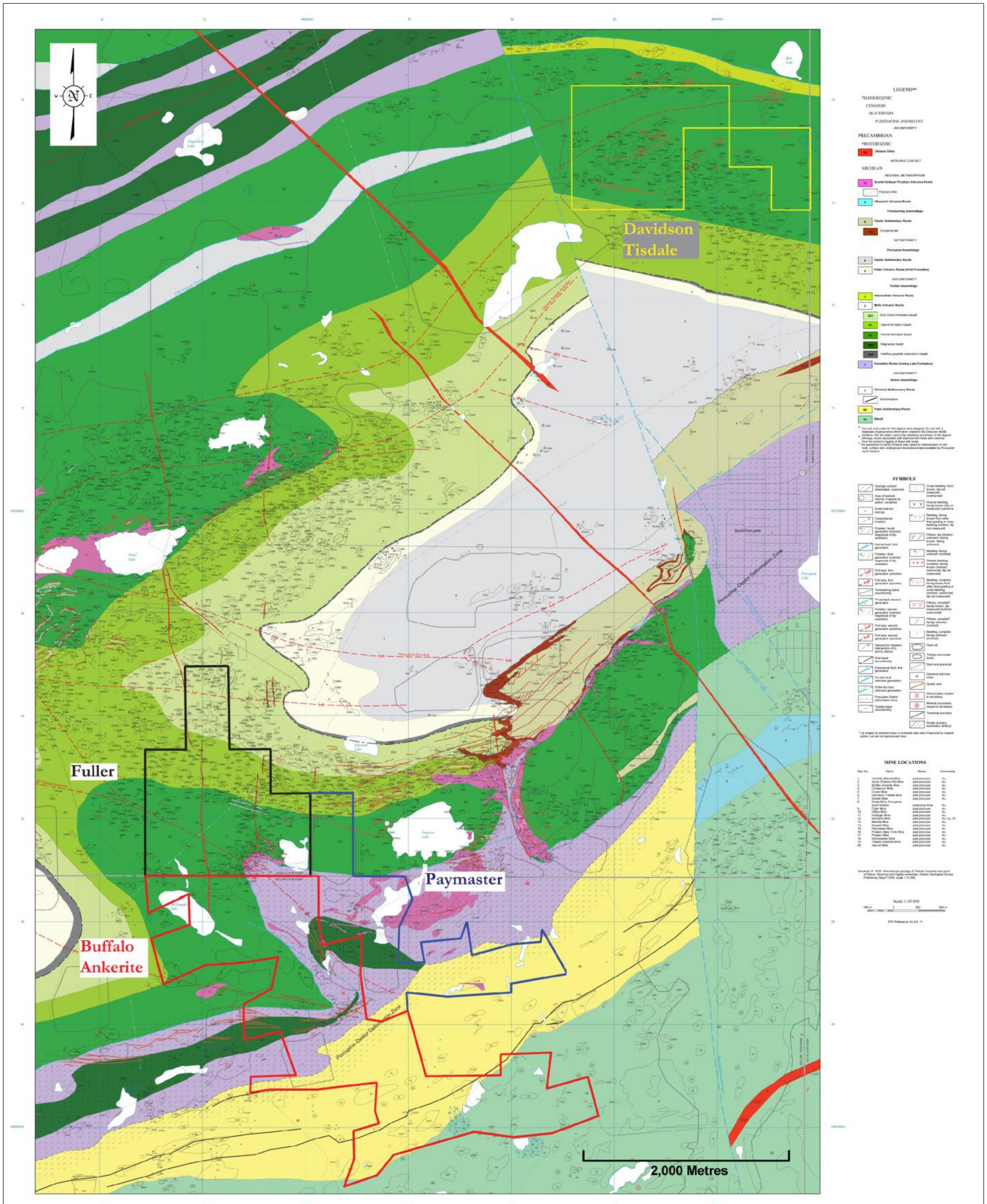
Structure

The F3 axial plane of the South Tisdale Anticline trends 300° azimuth and plunges steeply to the southeast and Kayorum Syncline is 275-280° azimuth plunging to the west-northwest. The regional Destor-Porcupine fault is located immediately to the south within the south ultramafic unit and varies from 200-300 metres in width. The southwest limb of the South Tisdale Anticline and northeast limb of the Kayorum Syncline are close to the hinge of the Kayorum Syncline and terminate against the Buffalo Ankerite 135 Fault Zone (Spur Fault). This left-lateral fault separates the Buffalo Ankerite South Mine from the Buffalo Ankerite North Mine, which were at one time one continuous deposit. The Buffalo Ankerite 135 Fault Zone is northwest/southeast trending at approximately 115° azimuth and dips at about 80° to the southwest.

Alteration

The pillowed mafic volcanic rocks show moderate ankerite and weak sericite alteration while the flanking ultramafic rocks show moderate to strong ankerite alteration with minor local fuchite. The ultramafic rocks are in fault contact with the mafic volcanic rocks as evidenced by talc fault gouge at the contacts. The degree of alteration intensity varies from east to west, with the strongest alteration in the east where the mafic volcanic rocks are narrow towards the Buffalo Ankerite 135 Fault and weakest in the west, where the thickness of the mafic volcanics increases.

Figure 7.1 Property Geology



7.3 DAVIDSON TISDALE PROPERTY

The Davidson Tisdale property was mapped in 1936 by D.R. Derry on behalf of Ventures and again in 1984 and 1985 by D.W. Broughton and R.G. Roberts on behalf of Getty.

The property is located in the Porcupine Gold Camp, along the possible offset easterly projection of the Hollinger-Macintyre trend (Figure 7.1). The property is underlain by a sequence of overturned easterly striking, northward dipping, pillowed and massive, magnesium tholeiitic volcanic flows of the Tisdale Group. In the southernmost part of the property there are outcrops of the distinctive V8 variolitic flows, underlain by a massive flow ("99"), which forms the basal member of the iron tholeiitic group. Minor graphitic sediments containing some pyrite and pyrrhotite have been noted locally on the property.

Alteration on the property is widespread, consisting of a low-grade calcite-chlorite envelope enclosing a mre intense quartz-sericite-ferro dolomite or ankerite core. Alteration was not well documented in the drill log database and has been recorded as somewhat patchy at the margins. The alteration is largely, if not entirely pre-faulting.

The abundance of faults is one of the most prominent features of the Davidson Tisdale property. Three distinct fault sets have been identified from underground workings on the property (Watts, Griffis, and McOuat, 1988). The faults are moderate to strong shear zones up to two metres in thickness. All known mineralized blocks lie within or very close to these faults. The "Main Fault" strikes 060º and dips 50º to the north. There is a set of faults, which generally parallel the main fault, but dip at 60º to 75º to the north. The second set of faults strikes 025º and dips northwest at 60º to 65º. These have been noted between fourth and fifth levels, representing a dilatant zone between two 060º structures. They contain prominent short veins, locally with gold mineralization. The third set trends 080º, dipping 30º to the north. These are limited to the east end of the workings and contain large "blow-outs" of quartz with erratic gold grains.

7.4 GOLD MINERALOGY

The main minerals of the gold-bearing zones are quartz, carbonates, alkali feldspar (most commonly albite), sericite, pyrite, tourmaline, arsenopyrite, scheelite, and molybdenite. Pyrrhotite is common in the deep parts of deposits, as well as in deposits hosted in banded iron formation. Arsenopyrite seems to be common in deposits hosted in sedimentary rocks.

The concentration of gold may be considered to be a product of the alteration process, as well as the concentrations of B, W, Sb, Te, Mo, and As. Although gold in quartz veins is the most distinctive occurrence, the gold in some deposits is also or largely in the altered wall rock. The mineralogy of the individual Properties is discussed in the following sub-sections

7.4.1 Buffalo Ankerite

The majority of the mineralization is associated with tourmaline-quartz-carbonate breccia zones (TBX) located within a pillowed mafic volcanic unit. Breccia fragments are comprised of ankerite-sericite altered pillowed mafic volcanics within a tourmaline-ankerite rich matrix. The finer the size of the carbonatized mafic fragments within the vein, the higher the gold grade.

Pyrite is widespread within these veins and ranges from 5-10% with a halo of 3-5% pyrite within the highly carbonatized pillowed volcanic flow. Visible gold is generally not observed but a good correlation of pyrite content with gold grade indicates a close association with the gold occurring probably in fractures within the pyrite or along boundaries of the pyrite grains.

Gold values within the conglomerate lithology are associated with quartz and quartz-tourmaline veins with 2-5% pyrite content at the vein margins.

7.4.2 Fuller

Mineralization on the Fuller property has been found in several settings. Most of the mineralization is within the Contact Zone, which is located along the contact between massive and pillowed basalt units. Mineralization is characterized by numerous parallel to subparallel quartz-carbonate veinlets hosted within a suite of volcanic rocks Pyrite is often abundant, both as very fine-grained disseminations and small pyrite trains roughly conformable to the stringers. The Contact Zone meanders along the contact between the pillowed and massive volcanic units, and frequently occurs entirely within one of the units. The boundaries of the zone are locally gradational.

The HW Zones are located in the structural hangingwall side of the Contact Zone, partly within the pillowed basalt sequence and partly within breccia. The zones are similar to the Contact Zone, but the quartz tends to reflect a pervasive silicification rather than discrete quartz veining.

Mineralization also occurs in highly carbonate-altered zones, and in porphyry bodies with quartz-tourmaline veinlets near the core of the synclinal structure and the Contact and HW Zones. Quartz-tourmaline-calcite veins with minor sulfides occur irregularly distributed throughout the massive volcanic unit; they generally vary in width from 0.3 to 2 ft.

A significant type of mineralization which has recently been evaluated is porphyry gold-pyrite --quartz mineralization in the Fuller deposit area where the porphyry has been relatively strongly deformed, particularly near the core of the Fuller syncline. Underground drillholes outlined, around the 500 ft level, a possibly continuous zone of mineralization which may extend laterally for more than 400 ft and vertically about 200 ft. Intercepts vary from 0.090 oz/t Au over 86.4 ft to 0.175 oz/t Au over 20.0 ft. Recent drill programs by Lexam have concentrated on evaluating this mineralization. Due to its' local importance in the Porcupine gold district; particular interest has been paid to its bulk, low-grade open pit potential property.

In the eastern part of the deposit, there are three zones which occur north of the Contact Zone; they are designated as footwall zones, or F1, F2 and F3 Zones. They are very similar to the Contact Zone, but are less silicified, sericitized, carbonatized and pyritized; they contain quartz veins.

The Green Carbonate #1 and #2 Zones occur at or near the contacts of feldspar porphyry structurally above the HW Zone. They are similar to carbonate zones found elsewhere, but contain more fuchsite and pyrite. Because they are related to lenses of porphyry, their continuity is somewhat uncertain.

7.4.3 Paymaster

The gold mineralization found in the porphyry appears to be related to various combinations of the following parameters. Tectonized porphyry with variable silicification, tourmalinization and sericitization seems to define corridors of low level gold mineralization. Within these the density of quartz-pyrite-tourmaline stringers and microveins appears to determine both the elevated and peaky pattern of the gold mineralization. These low grade zones are believed to generally trend east north-east but locally show more northerly orientations. There is some suggestion that a late stage of folding or deformation has modified a more linear and primary mineralization trend in addition to generating some of the narrow but often very high grade veins/veinlets. Their orientation is more north-north-westerly. The veinlets/stringers define steeply northerly plunging rod-like zones as suggested by several of the stopes.

From the gold distribution, the southern half and the central section of the porphyry appear to have the more pervasive gold mineralization, which is also where most of the past mining has taken place.

Most of the development took place on the 100, 200, 300, 400, and 600 Levels. There is no 500 Level. The two upper sublevels and a portion of the 100 Level, are more relevant to the 2 Shaft Porphyry. Most of the large cavern type stopes narrow and become raises and ore passes at approximately the 250 Level. Three stopes of significance are found between the 300 and 400 Level, of which one only starts at the 330 Level. Two of these continue on to the 600 Level. One of these mined out an adjacent Porphyry in the footwall of the Main Porphyry and only a thin porphyry veneer is believed to surround this stope in places.

Mineralization in the 2 Shaft Porphyry is similar to that in the Main Porphyry although the alteration is heavily weighted in favour of silicification and potassic alteration. Sericitization is generally weak and erratic. The porphyry is laced with quartz veins of varying intensities and orientations. Rubble or blocky zones are common, reflecting the more brittle nature of the tectonization. Pyrite is the common sulphide and can reach 10% locally. Low grade gold mineralization is nearly pervasive, although there seem to be linear zones of more continuous mineralization with a general northerly trend. To the south the porphyry body turns west and mineralization decreases rapidly. A similar situation exists to the north where the porphyry system turns to the east. The overall shape of the porphyry suggests a strong shear or deformation zone sub-parallel to the central and mineralized portion.

7.4.4 Davidson Tisdale

Two types of quartz veins were identified on the property (Brooks, 1987): type 1 - continuous tabular veins striking generally east-west and dipping 15º to 55º to the north, and type 2 - discontinuous, irregular, sub-vertical and steep north dipping to shallow south dipping lenses of quartz stringers and veins, striking 40º to 70º azimuth.

Examples of the type 1 veins are the “S” vein and the shallow vein stoped on the first level. They are gently undulating in strike and dip, vary in thickness from 0.5 to 7 metres, banded with seams of tourmaline, and mineralized with minor amounts of pyrite and chalcopyrite in areas of gold enrichment. Drifting and drilling to date indicate extensive barren veining with small high-grade pockets of native gold, the structural significance and predictability of which are unknown (Brooks, 1987). Type 1 veins are uncommon in the drill hole database for the pit area.

The type 2 vein quartz vein systems appear to be lenses of quartz veinlets and stringers which are oriented sub parallel to and separated by faults. These vein systems coalesce in places to form "blow-outs" of quartz breccia up to fifteen metres wide. These quartz veins often give way to shallowly south dipping auriferous quartz-filled tension gashes, which are abruptly terminated at faults. Most gold in the type 2 veins occurs near vein margins or xenoliths and is associated with patches of talc/muscovite and serpentine (often logged as chlorite), and a local increase in fine to coarse pyrite and chalcopyrite.

Following the Phase 1 underground program Getty personnel commented on the nature of the mineralized zones:

- In the vicinity of the Main Shaft gold occurs in a quartz stringer zone associated with a strong shear and sericite-carbonate alteration halo
- Though the quartz conforms to the shearing along strike, it cross cuts the shearing down dip
- Locally the stringer zones are very irregular and contain very erratic gold values
- Individual veins dip steeply to 90º at the center of the system and locally flatten to 0º, suggesting a sigmoidal pattern
- Interpretation of surface drilling had suggested a "sheet-like" vein system dipping about 45º NW.
- Underground, the gold mineralization was seen to be largely confined to a series of steeply dipping, en echelon quartz vein fracture systems occurring within the overall 45º dipping structure.

The geometry of the mineralized zones is as follows:

- Strike lengths of up to 40 m
- Widths of 2 to 4 m
- Dip lengths of about 12 m
- Upper and lower contacts plunging 20º and 70º to the west
- Dip is near vertical
- Mineralized zones are en echelon lining up within an envelope having a dip of 45º N and striking 060º

8.0 DEPOSIT TYPES

Gold mineralization on the Lexam properties belongs to the structurally controlled Archean lode gold class of deposits. Structurally hosted, low-sulphide, lode gold vein systems in metamorphic terrains from around the world possess many characteristics in common, spatially and through time; they constitute a single class of mesothermal precious metal deposits, formed during accretionary tectonics or continental delamination.

The Superior Province is the largest exposed Archean Craton in the world, and has accounted for more gold production than any other Archean Craton, with the 25 largest known deposits having produced more than 1 million ounces (30 tonnes) of gold.

The Timmins camp of the Abitibi Greenstone Belt is the most prolific gold camps in Canada with production in excess of 70 million ounces Au.

The majority of lode gold deposits formed proximal to regional terrane-boundary structures that acted as vertically extensive hydrothermal plumbing systems. Major mining camps are sited near deflections, strike slip or dilational jogs on the major structures. In detail, most deposits are situated in second or third order splays, or fault intersections, that define domains of low mean stress and correspondingly high fluid fluxes. Accordingly, the mineralization and associated alteration is most intense in these flanking domains. The largest lode gold mining camps are in terrains that possess greenschist facies hydrothermal alteration assemblages developed in cyclic ductile to brittle deformation. Smaller deposits are present in amphibolite to granulite facies terranes characterized by amphibolite to granulite facies alteration assemblages, ductile shear zones, and ductilely deformed veins (McCuaig and Kerrich, 1998).

The Abitibi belt is clearly the most prolific gold-producing greenstone terrain in the Superior Province.

Characteristically the largest gold deposits of the district are spatially associated with, but not in, porphyries similar to those exposed at the Dome mine. This association has led to considerable speculation regarding the genetic relationship of felsic porphyry emplacement to ore formation.

At a greenstone-belt scale, Archean gold camps are most commonly related to large-scale (>100 km long), transcrustal fault zones. However, on a camp scale, most of the world-class (>100 t) gold deposits are hosted in second- and third-order fault zones, whereas the first-order transcrustal faults are largely barren. There are many examples of transcrustal faults that are believed to penetrate into the lower crust or even into the mantle. Both the close spatial relationship of world-class gold deposits and trans crustal fault zones, and the deep penetration of the latter, stimulated the model that transcrustal fault zones represent the main conduits for gold-bearing hydrothermal fluids from mantle and lower-crustal levels to make their way into dilatant second- and third-order shear zones that host ore bodies in the upper crust (Kerrich, 1993)

This model requires that the trans-crustal fault zones and the gold-hosting second- and third-order shear zones were structurally and hydraulically connected at the time of gold mineralization. However, because most Archean trans-crustal fault zones worldwide are poorly exposed, and their location, strike, and orientation are typically interpreted from aeromagnetic data, there is a general lack of precise structural and fluid chemistry data. In the Abitibi greenstone belt in Canada, however, significant structural and fluid chemistry information does exist for the trans-crustal Cadillac tectonic zone.

8.1.1 Characteristics of Lode Gold Deposits

This class of Au-Ag deposits has variously been named lode or reef type, terms that include veins in shear zones, through stockworks, to mineralized wall rocks. The term mesothermal or mesozonal has also been used in view of their predominance in mid-crustal, greenschist facies environments. However, the deposits are now known to have formed over a large range of crustal depths from > 25 km to the near surface environment; hence those earlier terms are not appropriate.

Most lode gold deposits occur in terrains that experienced greenschist facies metamorphism, and the deposits feature greenschist-facies alteration assemblages. Recently, it has been recognized that Archean lode gold deposits in amphibolite and granulite facies terrains share numerous characteristics, such as structural hosting, metal inventory, element association, ore fluid properties and likely source, in common with their greenschist hosted mesothermal counterparts. Accordingly, this class of structurally hosted Au-Ag vein deposits may be viewed as forming over a crustal depth range, or 'crustal continuum', extending from granulite to sub-greenschist facies environments.

Studies of lode gold deposits of all ages have revealed a number of common characteristics which can be summarized as follows:

- Rich lode Au metallogenic provinces are associated with external super-continent cycles, or external domains of internal super-continent aggregation cycles.
- The timing of mineralization is late-accretion, within the larger time frame of erogenic belts involving accretion of one or multiple allochthonous terrains.
- Deposits are sited proximal to major accretionary structures within, or at the boundaries of, composite metamorphosed volcanic-plutonic or sedimentary terrains.
- Lode gold deposits are distributed in belts of great geological complexity and display gradients of lithology, strain, fluid flow and metamorphic grade typical of accretionary environments.
- Deposits are structurally hosted, associated with second or higher order splays of translithospheric faults.
- The alteration mineral paragenesis in greenschist facies domains is dominated by quartz, carbonate, mica, (albite), chlorite, pyrite, scheelite and tourmaline.
- There is a distinctive element association characterized by enrichment in Au, Ag (minor As, Sb, Te, W, Mo, Bi, B), with low enrichments of Cu, Pb, Zn relative to the background abundances. In Phanerozoic deposits Mo and Te are only enriched where veins cut felsic intrusions.
- Ore forming hydrothermal fluids are dilute aqueous carbonic fluids, with uniformly low fluid salinities.
- Lode systems may have vertical extents of up to 2 km, with a lack of zoning, or weak zoning, within deposits, albeit with some zoning of metal content at the scale of an entire mining district.

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8.3 FULLER

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8.5 DAVIDSON TISDALE

Two types of quartz veins were identified on the property (Brooks, 1987): type 1 - continuous tabular veins striking generally east-west and dipping 15º to 55º to the north, and type 2 - discontinuous, irregular, sub-vertical and steep north dipping to shallow south dipping lenses of quartz stringers and veins, striking 40º to 70º azimuth.

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- In the vicinity of the Main Shaft gold occurs in a quartz stringer zone associated with a strong shear and sericite-carbonate alteration halo
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- Interpretation of surface drilling had suggested a "sheet-like" vein system dipping about 45º NW.
- Underground, the gold mineralization was seen to be largely confined to a series of steeply dipping, en echelon quartz vein fracture systems occurring within the overall 45º dipping structure.

The geometry of the mineralized zones is as follows:

- Strike lengths of up to 40 m
- Widths of 2 to 4 m
- Dip lengths of about 12 m
- Upper and lower contacts plunging 20º and 70º to the west
- Dip is near vertical
- Mineralized zones are en echelon lining up within an envelope having a dip of 45º N and striking 060º

9.0 EXPLORATION

Lexam exploration on all the properties has been predominately diamond drilling (see chapter 10) and resource estimation reports.

10.0 DRILLING

The majority of the Lexam exploration on the four (4) properties has been diamond drilling. Historically, the properties have all been underground gold producers and therefore historic surface and underground drilling is also available in limited details. Lexam has conducted extensive surface drilling on all four projects during the period 2003 through 2012.

All drill holes were drilled with NQ size core. Norex Drilling of Timmins, Ontario was the contractor for most of the drilling. If the hole hit an underground working (drift, raise or stope) and the drillers were successful in being able to continue drilling, this was normally done with a BQ bit. Not all holes were successful in continuing, though most were completed to target depth. Most holes hit no opening at all. Collars were surveyed by a differential corrected GPS instrument accurate to within 0.1 m. Down hole surveys were carried out using a Reflex Early Shot instrument with readings taken every 50 m.

Core was picked up twice per day by Lexam core technicians and taken to the core shack located at the Davidson Tisdale property. Core was logged by the geologist with altered and mineralized sections marked for sampling.

Table 10.1 indicates the lithology legend used for logging in the Geotic Log drill logging program as well as the legend used for plotting in Gemcom.

10.1 BUFFALO ANKERITE

Lexam completed significant surface diamond drilling on the Buffalo Ankerite property during the period 2005 through 2012. The majority of the drilling was conducted on the Buffalo Ankerite South Zone, expanding on the exploration and resource work completed by Placer Dome during the period 2002 through 2005. Other drilling included in the resource is a limited amount of underground drilling completed by Buffalo Ankerite Mines Ltd. The majority of the drill hole database is exploration completed by Lexam or its predecessor company.

- Diamond Drilling to date by Lexam VG and predecessor companies, 2005-2012, has totalled of 81,225.5 metres of diamond core data from 225 drill holes.
- North Zone - 66 drill holes, 87,946.7 ft or 26,806.1 m
- South Zone - 159 drill holes, 178,541.2 ft or 54,419.4

A Surface Diamond Drill Table with hole locations and collar details as well as significant assays is seen in Table 10.2.

10.1.1 Geology and Mineralization

Both the Buffalo Ankerite South Zone and the North Zone gold mineralization is located primarily within a narrow pillowed mafic volcanic flow unit of the Central Series, Tisdale Assemblage. The volcanics are complexly folded around the South Tisdale Anticline and Kayorum Syncline resulting in an S-shaped flexure in the stratigraphy. The pillowed mafic volcanic unit, which hosts the main mineralized domains of the Buffalo Ankerite South property, is flanked to the north and south by "Hershey Lake Series" Mg-rich ultramafic flow units. In this area of the property, the volcanic flows strike between 065 and 070°, and dip at approximately 60° to the north and thicken to the west. The mineralization is found on both sides of the fold, the south side has seen the majority of exploration both recent and historic. The north side is truncated by a fault trending az335 and cutting off the mineralization on the north limb of the fold.

TABLE 10.1
LEXAM GOLD ROCK CODE LEGEND FOR PLANS AND SECTIONS

Lexam-Rock Code Legend		
Code	**Description**	**Colour**
CAS*	CASING	
OB	OVERBURDEN	
UGO*	UNDERGROUND OPENING	
FTZ-alt*	FAULT ZONE - Altered	
FTZ*	FAULT ZONE	
QVS*	Quartz Vein System	
QV*	Quartz Vein	
QToBrf	Quartz Tourmaline Breccia	
QToV	Quartz Tourmaline Vein	
MZ	Mineralized Zone	
I4*	Ultramafic Intrusive	
QFP-alt*	Altered Quartz Feldspar Porphyry	
QFP*	Quartz Feldspar Porphyry	
FP-alt*	Altered Feldspar Porphyry	
FP*	Feldspar Porphyry	
POR*	Porphyry (undifferentiated)	
V3Fe*	Fe tholeiitic mafic volcanic	
V3-alt*	Altered Mafic volcanic	
V3*	Mafic volcanic (Undifferentiated)	
V4a/V3b	Komatiitic Basalt	
V4a*	Ultramafic Volcanic - unaltered	
V4-alt-ak-fc*	Altered Ultramafic volcanic -ankerite/fuchsite alteration	
V4-alt*	Altered Ultramafic volcanic	
V4-tc*	Ultramafic volcanic-talc alteration	
V4-srp*	Ultramafic volcanic-serpentine alteration	
V4*	Ultramafic volcanic- (Undifferentiated)	

Typical cross sections of the geology and mineralization are shown in Figure 10.1 and Figure 10.2.

The southern margin of the Buffalo Ankerite South Zone mineralization is proximal to the Porcupine Destor Fault Zone. At the eastern end of the zone a discontinuous conglomerate unit is located along the contact between the mafic flows and the south ultramafic unit. The conglomerate is interpreted as Timiskaming in age containing mainly bleached mafic volcanic clasts with occasional porphyry and ultramafic clasts and typically follows this contact and is similarly oriented for dip. The conglomerate is commonly mineralized with Quartz-tourmaline breccia zones and gold values

Quartz-feldspar porphyries intrude the volcanic units and late northwest-trending diabase dykes cut all the above rock types.

The South Zone has a very pronounced plunge of -30 degrees to the west. This plunge represents the bottom of the synclinal structure which hosts the gold mineralization. In the vicinity of section 4200 E the North Zone and the South Zone merge at approximately 1800 feet below surface. The South Zone structure continues to the west and the North Zone continues to depth. The structural complexity of this area has not been fully explored, however it is known that porphyry bodies are common in this area and contain elevated intervals of gold mineralization.

The North Zone is hosted in similar geology but is approximately north-south striking with a steeply to slightly west dip.

10.1.2 Alteration

The pillowed mafic volcanics are carbonatized to moderately ankerite and weak sericite alteration, while the flanking ultramafic rocks are moderate to strong ankerite altered with minor local fuchsite. The ultramafic rockss are in fault contact with the volcanics and reflected in talc fault gouge at the contacts. The degree of alteration intensity varies from east to west, with the strongest alteration in the east, where the mafic volcanics are narrow towards the Buffalo Ankerite 135 Fault and the weakest in the west, where the thickness of the mafic volcanics increases.

10.1.3 Description of Mineralization

The majority of the mineralization is associated with quartz-tourmaline breccia veins and is located within all lithologies with the pillowed mafic volcanic being the primary host. The breccia fragments are comprised of ankerite-sericite altered pillowed mafic volcanics within a tourmaline-ankerite rich matrix. The greater the pyrite content the higher the gold grade.

Pyrite is widespread within these veins and ranges from 5-10% with a halo of 3-5% pyrite within the highly carbonatized pillowed volcanic flow. Visible gold is generally not observed, but a good correlation of pyrite content with gold grade indicates a close association with the gold occurring probably in fractures within the pyrite or along the boundaries of the pyrite grains.

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
VBA-05-01	4613.2	7399.8	10918.2	400.3	122.0	-45	90		242.8	252.6	9.8	0.154	74.0	77.0	3.0	5.26	15.8
									274.4	285.2	10.8	0.067	83.6	86.9	3.3	2.31	7.6
									317.5	326.8	9.3	0.143	96.8	99.6	2.8	4.89	13.9
VBA-05-02	4685.5	7199.9	10909.7	295.3	90.0	-45	90		162.5	177.2	14.7	0.066	49.5	54.0	4.5	2.27	10.2
									236.0	246.0	10.0	0.097	71.9	75.0	3.0	3.33	10.1
VBA-05-03	4745.5	7400.6	10892.0	242.8	74.0	-45	90		50.9	163.0	112.1	0.038	15.5	49.7	34.2	1.30	44.5
								incl	50.9	67.3						2.73	
VBA-05-04	4686.0	7465.0	10892.0	390.4	119.0	-60	90		209.2	227.8	18.6	0.207	63.8	69.4	5.7	7.11	40.2
								incl	214.1	223.1	9.0	0.384	65.3	68.0	2.7	13.16	36.1
VBA-05-05	4839.9	7200.2	10889.1	183.7	56.0	-45	90		100.5	114.8	14.3	0.065	30.6	35.0	4.4	2.25	9.8
									136.9	147.4	10.5	0.043	41.7	44.9	3.2	1.47	4.7
VBA-05-06	4708.6	7000.3	10897.2	400.3	122.0	-45	90		201.4	222.0	20.6	0.032	61.4	67.7	6.3	1.10	6.9
VBA-05-07	4814.5	6900.0	10894.5	360.9	110.0	-45	90		160.7	215.7	55.0	0.076	49.0	65.7	16.8	2.60	43.6
								incl	190.8	215.7	24.9	0.102	58.2	65.7	7.6	3.50	26.6
VBA-05-07									290.4	322.5	32.1	0.118	88.5	98.3	9.8	4.03	39.4
VBA-05-08	4796.6	6800.3	10904.5	370.7	113.0	-45	90		270.6	287.7	17.1	0.041	82.5	87.7	5.2	1.40	7.3
VBA-05-09	4865.3	6599.7	10913.5	390.4	119.0	-45	90		51.2	59.3	8.1	0.065	15.6	18.1	2.5	2.22	5.5
VBA-05-10	4925.3	6496.8	10918.5	439.6	134.0	-45	90		nsv								
VBA-05-11	5090.0	6300.0	10921.0	420.0	128.0	-45	90		nsv								
VBA-05-12	4952.8	6602.3	10905.2	183.7	56.0	-45	90		nsv								
VBA-05-13	4815.0	6900.0	10896.0	117.6	35.8	-45	90		55.8	65.6	9.8	0.023	17.0	20.0	3.0	0.77	2.3
VBA-06-14	3200.1	5600.6	10909.0	383.9	117.0	-50	180		239.1	292.1	53.0	0.036	72.9	89.0	16.2	1.22	19.7
VBA-06-15	3200.2	5600.6	10908.1	413.4	126.0	-75	180		323.5	332.6	9.1	0.047	98.6	101.4	2.8	1.61	4.5
									347.8	356.4	8.6	0.054	106.0	108.6	2.6	1.86	4.9
VBA-06-16	4025.4	5373.5	10934.0	813.7	248.0	-45	180		213.2	228.0	14.8	0.022	65.0	69.5	4.5	0.76	3.4
									660.9	692.8	31.9	0.033	201.4	211.2	9.7	1.13	11.0
									777.5	786.7	9.2	0.027	237.0	239.8	2.8	0.92	2.6
VBA-06-17	3800.3	5450.4	10929.4	528.2	161.0	-45	180		240.7	245.9	5.2	0.017	73.4	75.0	1.6	0.58	0.9
VBA-06-18	3800.3	5453.4	10929.5	351.0	107.0	-75	180		282.1	287.0	4.9	0.021	86.0	87.5	1.5	0.72	1.1
VBA-06-19	3399.7	5501.1	10912.7	380.6	116.0	-45	180		176.5	227.0	50.5	0.034	53.8	69.2	15.4	1.17	18.1
									262.0	277.0	15.0	0.023	79.9	84.4	4.6	0.78	3.6
VBA-06-20	3399.6	5504.1	10912.6	400.3	122.0	-75	180		263.0	280.5	17.5	0.036	80.2	85.5	5.3	1.23	6.6
VBA-06-21	3100.5	5497.2	10907.4	242.8	74.0	-45	180		173.5	178.9	5.4	0.027	52.9	54.5	1.6	0.93	1.5
VBA-06-22	3100.4	5500.6	10907.4	301.8	92.0	-75	180		182.9	192.0	9.1	0.024	55.7	58.5	2.8	0.84	2.3
									222.0	252.0	30.0	0.043	67.7	76.8	9.1	1.46	13.3
VBA-06-23	3000.0	5497.2	10909.7	223.1	68.0	-45	180		162.2	210.0	47.8	0.074	49.4	64.0	14.6	2.52	36.8
								incl	162.2	167.0	4.8	0.285	49.4	50.9	1.5	9.77	14.3
VBA-06-24	3000.2	5500.3	10909.8	272.3	83.0	-75	180		208.6	223.6	15.0	0.021	63.6	68.2	4.6	0.73	3.3
									233.6	268.5	34.9	0.055	71.2	81.8	10.6	1.87	19.9

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
VBA-06-25	2800.0	5600.0	10916.0	292.0	89.0	-45	180		nsv								
VBA-06-26	2800.0	5600.0	10916.0	331.4	101.0	-75	180		307.9	311.5	3.6	0.010	93.8	94.9	1.1	0.34	0.4
VBA-06-27	2600.5	5500.6	10921.1	252.6	77.0	-45	180		219.0	225.1	6.1	0.084	66.8	68.6	1.9	2.88	5.4
VBA-06-28	2600.4	5503.7	10920.5	272.3	83.0	-75	180		216.0	231.8	15.8	0.129	65.8	70.7	4.8	4.42	21.3
VBA-06-29	3394.9	4952.3	10902.6	557.7	170.0	-45	180		435.0	440.0	5.0	0.007	132.6	134.1	1.5	0.24	0.4
VBA-06-30	3641.2	5437.1	10924.4	292.0	89.0	-45	195		184.0	223.4	39.4	0.036	56.1	68.1	12.0	1.25	15.0
VBA-06-31	3642.4	5440.0	10924.0	318.3	97.0	-75	200		255.5	276.0	20.5	0.029	77.9	84.1	6.2	1.00	6.3
VBA-06-32	4157.3	5313.8	10942.9	282.2	86.0	-45	180		209.0	216.5	7.5	0.034	63.7	66.0	2.3	1.18	2.7
VBA-06-33	4157.7	5317.1	10943.0	331.4	101.0	-75	180		252.0	262.9	10.9	0.023	76.8	80.1	3.3	0.77	2.6
VBA-06-34	4414.0	7870.3	10890.4	538.1	164.0	-48	90		417.0	427.0	10.0	0.037	127.1	130.1	3.0	1.25	3.8
VBA-06-35	4365.5	8007.1	10888.7	653.5	199.2	-58	90		514.5	522.0	7.5	0.026	156.8	159.1	2.3	0.89	2.0
									537.0	636.5	99.5	0.050	163.7	194.0	30.3	1.73	52.4
								incl	617.4	636.5	19.1	0.109	188.2	194.0	5.8	3.74	
VBA-06-36	4356.8	8199.8	10887.5	695.5	212.0	-55	90		666.7	687.6	20.9	0.075	203.2	209.6	6.4	2.56	16.3
VBA-06-37	4251.0	8399.7	10888.8	823.5	251.0	-52	90		797.9	814.1	16.2	0.031	243.2	248.1	4.9	1.06	5.2
VBA-06-38	4099.3	8704.5	10889.2	902.3	275.0	-45	90		nsv								
VBA-06-39	4386.5	8493.6	10896.2	551.2	168.0	-52	90		248.9	265.0	16.1	0.032	75.9	80.8	4.9	1.08	5.3
VBA-06-40	4422.0	8305.8	10891.7	597.1	182.0	-55	90		nsv								
VBA-06-41	5328.0	6902.1	10888.4	757.9	231.0	-45	270		557.0	562.0	5.0	0.032	169.8	171.3	1.5	1.10	1.7
									686.0	741.5	55.5	0.053	209.1	226.0	16.9	1.81	30.6
VBA-06-42	5299.5	7204.6	10889.7	682.3	208.0	-45	270		646.3	669.3	23.0	0.037	197.0	204.0	7.0	1.27	8.9
VBA-06-43	4516.9	7607.3	10902.8	451.0	137.5	-57	90		283.7	292.0	8.3	0.040	86.5	89.0	2.5	1.38	3.5
									299.5	322.7	23.2	0.010	91.3	98.4	7.1	0.34	2.4
VBA-06-44	4558.3	7501.6	10904.0	518.4	158.0	-49	90		253.0	263.0	10.0	0.064	77.1	80.2	3.0	2.19	6.7
									351.9	368.5	16.6	0.053	107.3	112.3	5.1	1.82	9.2
									406.0	421.2	15.2	0.039	123.7	128.4	4.6	1.32	6.1
VBA-06-45	4683.6	7319.3	10914.3	370.7	113.0	-53	90		185.3	205.9	20.6	0.041	56.5	62.8	6.3	1.41	8.9
									291.0	301.0	10.0	0.029	88.7	91.7	3.0	1.00	3.0
VBA-06-46	4632.6	6804.2	10926.6	666.0	203.0	-48	90		436.5	442.7	6.2	0.143	133.0	134.9	1.9	4.89	9.2
									577.0	628.0	51.0	0.028	175.9	191.4	15.5	0.94	14.7
VBA-06-47	4689.4	6705.4	10929.9	803.8	245.0	-55	90		648.0	692.4	44.4	0.084	197.5	211.0	13.5	2.89	39.2
								incl	665.3	684.0	18.7	0.159	202.8	208.5	5.7	5.46	31.1
									725.9	765.0	39.1	0.033	221.3	233.2	11.9	1.14	13.6
VBA-06-48	4617.7	7107.7	10915.1	607.0	185.0	-48	90		327.7	336.6	8.9	0.106	99.9	102.6	2.7	3.63	9.9
									351.0	366.0	15.0	0.017	107.0	111.6	4.6	0.58	2.7
VBA-06-49	4433.0	7002.1	10943.9	843.2	257.0	-45	90		674.8	717.0	42.2	0.070	205.7	218.5	12.9	2.39	30.7

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
								incl	679.7	690.0	10.3	0.123	207.2	210.3	3.1	4.23	13.3
									747.4	755.8	8.4	0.039	227.8	230.4	2.6	1.34	3.4
									771.0	838.0	67.0	0.023	235.0	255.4	20.4	0.77	15.8
VBA-06-50	4493.4	7199.0	10932.0	439.6	134.0	-45	90		433.0	439.6	6.6	0.138	132.0	134.0	2.0	4.74	9.6
VBA-06-51	4297.5	7401.4	10953.5	601.0	183.2	-45	90		586.0	591.0	5.0	0.016	178.6	180.1	1.5	0.55	0.8
VBA-06-52	4496.4	6516.3	10938.5	1020.3	311.0	-45	90		959.0	967.5	8.5	0.048	292.3	294.9	2.6	1.65	4.3
VBA-06-53	4500.0	6500.0	10930.0	636.5	194.0	-45	45		622.3	632.0	9.7	0.046	189.7	192.6	3.0	1.58	4.7
VBA-06-54	4480.4	5341.4	10935.9	508.5	155.0	-45	217		185.5	190.0	4.5	0.025	56.5	57.9	1.4	0.86	1.2
VBA-06-55	4460.0	5350.0	10955.0	400.3	122.0	-65	202		237.9	247.7	9.8	0.088	72.5	75.5	3.0	3.02	9.0
									290.0	299.9	9.9	0.026	88.4	91.4	3.0	0.88	2.7
VBA-06-56	4459.0	5350.5	10935.9	370.7	113.0	-45	180		207.7	217.0	9.3	0.032	63.3	66.1	2.8	1.08	3.1
VBA-06-57	4486.8	5350.0	10936.0	449.5	137.0	-65	180		235.7	248.0	12.3	0.040	71.8	75.6	3.7	1.38	5.2
VBA-06-58	4618.8	5464.3	10945.3	528.2	161.0	-58	180		410.1	415.0	4.9	0.010	125.0	126.5	1.5	0.34	0.5
VBA-06-59	4827.1	5462.7	10942.6	496.8	151.4	-62	180		347.4	357.0	9.6	0.030	105.9	108.8	2.9	1.03	3.0
VBA-06-60	4924.1	5531.6	10932.7	646.3	197.0	-64	180		454.0	478.2	24.2	0.115	138.4	145.8	7.4	3.96	29.2
								incl	468.0	478.2	10.2	0.220	142.6	145.8	3.1	7.54	23.5
VBA-06-61	6034.8	5740.5	10898.8	429.8	131.0	-58	180		264.8	272.6	7.8	0.148	80.7	83.1	2.4	5.07	12.0
									318.0	330.0	12.0	0.026	96.9	100.6	3.7	0.90	3.3
									389.0	405.0	16.0	0.034	118.6	123.4	4.9	1.17	5.7
VBA-06-62	6042.0	5887.9	10889.0	587.3	179.0	-59	180		396.8	413.2	16.4	0.122	120.9	125.9	5.0	4.17	20.9
VBA-06-63	5926.4	5852.5	10914.2	656.2	200.0	-52	180		378.2	396.4	18.2	0.090	115.3	120.8	5.5	3.07	17.0
									616.0	630.3	14.3	0.019	187.8	192.1	4.4	0.65	2.8
VBA-06-64	5629.4	5644.1	10927.8	547.9	167.0	-63	180		425.5	434.0	8.5	0.142	129.7	132.3	2.6	4.85	12.6
									449.0	460.4	11.4	0.024	136.9	140.3	3.5	0.84	2.9
VBA-06-65	5529.4	5644.1	10927.8	474.4	144.6	-65	180		312.5	319.0	6.5	0.062	95.3	97.2	2.0	2.12	4.2
									426.0	442.4	16.4	0.045	129.8	134.8	5.0	1.55	7.7
VBA-06-66	5325.0	5518.0	10935.0	597.1	182.0	-65	180		318.7	360.6	41.9	0.061	97.1	109.9	12.8	2.09	26.7
									538.8	570.7	31.9	0.072	164.2	173.9	9.7	2.46	23.9
								incl	543.8	555.1	11.3	0.154	165.8	169.2	3.4	5.28	18.2
VBA-06-67	5230.1	5447.1	10925.8	528.2	161.0	-64	180		337.6	351.0	13.4	0.057	102.9	107.0	4.1	1.95	8.0
									372.3	409.0	36.7	0.042	113.5	124.7	11.2	1.44	16.1
									494.9	511.1	16.2	0.022	150.8	155.8	4.9	0.76	3.8
VBA-06-68	5115.3	5473.7	10925.7	675.9	206.0	-64	180		431.6	446.3	14.7	0.026	131.6	136.0	4.5	0.89	4.0
									612.7	638.0	25.3	0.073	186.8	194.5	7.7	2.51	19.3
VBA-06-69	6119.8	5906.9	10885.4	656.2	200.0	-72	180		476.6	492.0	15.4	0.055	145.3	150.0	4.7	1.90	8.9
									508.0	555.0	47.0	0.055	154.8	169.2	14.3	1.88	26.9
VBA-06-70	5301.1	5221.2	10921.1	252.6	77.0	-45	180		24.0	106.0	82.0	0.047	7.3	32.3	25.0	1.60	40.1
VBA-07-71	3449.0	6655.0	10930.0	3534.2	1077.2	-73	160		2,318.5	2,327.5	9.0	0.104	706.7	709.4	2.7	3.55	9.7

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
VBA-07-72	5197.0	5108.0	10940.0	114.8	35.0	-45	172		36.0	45.9	9.9	0.030	11.0	14.0	3.0	1.03	3.1
VBA-07-73	5616.0	5404.0	10930.0	164.0	50.0	-50	140		85.0	96.0	11.0	0.030	25.9	29.3	3.4	1.03	3.4
VBA-07-74	5718.0	5420.0	10930.0	160.8	49.0	-45	180		74.5	104.5	30.0	0.398	22.7	31.9	9.1	13.63	124.6
								incl	95.0	104.5	9.5	0.779	29.0	31.9	2.9	26.71	77.3
VBA-07-75	5718.0	5420.0	10930.0	234.8	71.6	-55	180		51.0	69.0	18.0	0.063	15.5	21.0	5.5	2.17	11.9
									80.0	140.0	60.0	0.081	24.4	42.7	18.3	2.78	50.9
									85.0	97.0	12.0	0.258	25.9	29.6	3.7	8.85	32.4
									192.9	203.0	10.1	0.224	58.8	61.9	3.1	7.69	23.7
VBA-07-76	5818.0	5379.0	10930.0	242.8	74.0	-45	180		15.0	34.0	19.0	0.105	4.6	10.4	5.8	3.59	20.8
VBA-07-77	6017.0	5549.0	10925.0	1166.0	355.4	-45	180		37.5	55.7	18.2	0.070	11.4	17.0	5.5	2.40	13.3
									117.8	124.6	6.8	0.062	35.9	38.0	2.1	2.11	4.4
VBA-07-78	5915.0	5568.0	10925.0	216.5	66.0	-50	180		96.0	106.0	10.0	0.022	29.3	32.3	3.0	0.75	2.3
									159.4	198.5	39.1	0.135	48.6	60.5	11.9	4.63	55.1
								incl	188.7	198.5	9.8	0.380	57.5	60.5	3.0	13.04	39.0
VBA-07-79	3094.0	5939.0	10923.0	1246.7	380.0	-45	230		619.0	643.5	24.5	0.065	188.7	196.1	7.5	2.24	16.7
									1,062.5	1,072.0	9.5	0.056	323.9	326.7	2.9	1.90	5.5
VBA-07-80	3229.0	6019.0	10923.0	1128.6	344.0	-57	240		743.8	755.8	12.0	0.047	226.7	230.4	3.7	1.60	5.8
									766.5	848.4	81.9	0.065	233.6	258.6	25.0	2.21	55.2
								incl	816.2	834.0	17.8	0.157	248.8	254.2	5.4	5.39	29.2
VBA-07-81	3229.0	6019.0	10923.0	1146.3	349.4	-50	215		678.0	685.0	7.0	0.117	206.7	208.8	2.1	4.02	8.6
									1,014.0	1,021.8	7.8	0.043	309.1	311.4	2.4	1.47	3.5
VBA-07-82	3229.0	6019.0	10923.0	961.3	293.0	-50	185		614.4	622.4	8.0	0.079	187.3	189.7	2.4	2.71	6.6
VBA-07-83	3414.0	6018.0	10923.0	1484.9	452.6	-50	180		629.5	658.2	28.7	0.061	191.9	200.6	8.7	2.09	18.3
									1,097.4	1,122.0	24.6	0.029	334.5	342.0	7.5	1.00	7.5
									1,182.1	1,188.6	6.5	0.074	360.3	362.3	2.0	2.52	5.0
VBA-07-84	3414.0	6018.0	10923.0	1443.6	440.0	-65	180		672.3	720.9	48.6	0.148	204.9	219.7	14.8	5.08	75.2
								incl	689.9	703.0	13.1	0.273	210.3	214.3	4.0	9.35	37.4
									995.9	1,006.0	10.1	0.035	303.6	306.6	3.1	1.21	3.7
									1,144.0	1,160.0	16.0	0.033	348.7	353.6	4.9	1.13	5.5
									1,217.0	1,237.6	20.6	0.115	370.9	377.2	6.3	3.95	24.8
									1,274.1	1,308.7	34.6	0.080	388.3	398.9	10.5	2.73	28.8
									1,333.8	1,366.0	32.2	0.030	406.5	416.4	9.8	1.04	10.2
VBA-07-85	3719.0	6061.0	10923.0	1305.8	398.0	-50	180		691.0	710.5	19.5	0.038	210.6	216.6	5.9	1.30	7.7
									725.5	745.4	19.9	0.077	221.1	227.2	6.1	2.65	16.1
									1,129.9	1,146.0	16.1	0.082	344.4	349.3	4.9	2.80	13.8
VBA-07-86	5823.0	6032.0	10910.0	961.3	293.0	-60	180		630.1	652.2	22.1	0.044	192.1	198.8	6.7	1.50	10.1
									675.0	683.0	8.0	0.054	205.7	208.2	2.4	1.84	4.5
									702.5	727.6	25.1	0.045	214.1	221.8	7.7	1.53	11.7
									842.2	895.0	52.8	0.053	256.7	272.8	16.1	1.82	29.3
VBA-07-87	5823.0	6032.0	10910.0	1197.5	365.0	-82	180		594.1	607.7	13.6	0.163	181.1	185.2	4.1	5.59	23.2
									622.6	636.0	13.4	0.039	189.8	193.9	4.1	1.33	5.4
VBA-07-88	6021.0	5943.0	10900.0	813.7	248.0	-75	180		256.0	267.0	11.0	0.032	78.0	81.4	3.4	1.10	3.7
									277.0	287.0	10.0	0.027	84.4	87.5	3.0	0.93	2.8

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
									330.0	346.4	16.4	0.111	100.6	105.6	5.0	3.79	19.0
									369.4	382.2	12.8	0.068	112.6	116.5	3.9	2.34	9.1
									560.0	587.6	27.6	0.023	170.7	179.1	8.4	0.79	6.6
									626.6	636.6	10.0	0.030	191.0	194.0	3.0	1.02	3.1
VBA-07-89	5616.0	6036.0	10920.0	1108.9	338.0	-62	180		671.5	692.0	20.5	0.063	204.7	210.9	6.2	2.16	13.5
									746.0	752.8	6.8	0.047	227.4	229.5	2.1	1.61	3.3
									875.0	908.0	33.0	0.044	266.7	276.8	10.1	1.50	15.1
									923.0	933.0	10.0	0.021	281.3	284.4	3.0	0.70	2.1
VBA-07-90	5520.0	6010.0	10930.0	1030.2	314.0	-54	180		610.0	638.3	28.3	0.029	185.9	194.6	8.6	1.00	8.6
									822.0	847.0	25.0	0.026	250.5	258.2	7.6	0.88	6.7
									862.0	919.0	57.0	0.045	262.7	280.1	17.4	1.54	26.8
								incl	888.8	895.7	6.9	0.131	270.9	273.0	2.1	4.50	9.4
VBA-07-91	5314.0	5893.0	10950.0	1089.2	332.0	-54	180		575.1	588.0	12.9	0.036	175.3	179.2	3.9	1.23	4.9
									949.3	970.4	21.1	0.052	289.3	295.8	6.4	1.77	11.4
VBA-07-92	5314.0	5893.0	10950.0	1151.6	351.0	-73	180		647.8	654.2	6.4	0.056	197.4	199.4	2.0	1.92	3.7
									669.0	679.7	10.7	0.197	203.9	207.2	3.3	6.77	22.1
									699.1	725.1	26.0	0.055	213.1	221.0	7.9	1.87	14.8
									826.7	835.0	8.3	0.046	252.0	254.5	2.5	1.57	4.0
									1,128.0	1,142.0	14.0	0.022	343.8	348.1	4.3	0.77	3.3
VBA-07-93	5144.0	6073.0	10950.0	1315.6	401.0	-50	180		718.2	741.6	23.4	0.028	218.9	226.0	7.1	0.95	6.8
									1,170.4	1,185.0	14.6	0.047	356.7	361.2	4.5	1.60	7.1
VBA-07-94	5144.0	6073.0	10950.0	1394.4	425.0	-79	170		1,059.0	1,077.0	18.0	0.018	322.8	328.3	5.5	0.61	3.4
									1,155.9	1,163.0	7.1	0.039	352.3	354.5	2.2	1.35	2.9
									1,233.4	1,241.0	7.6	0.105	375.9	378.3	2.3	3.60	8.3
VBA-07-95	5144.0	6073.0	10950.0	1365.9	416.3	-55	190		873.0	884.0	11.0	0.076	266.1	269.4	3.4	2.61	8.8
									896.0	905.0	9.0	0.062	273.1	275.8	2.7	2.11	5.8
									1,250.8	1,260.0	9.2	0.024	381.2	384.0	2.8	0.83	2.3
									1,289.0	1,305.0	16.0	0.033	392.9	397.8	4.9	1.14	5.6
VBA-07-96	5144.0	6073.0	10950.0	1473.1	449.0	-50	210		1,024.5	1,035.0	10.5	0.028	312.3	315.5	3.2	0.97	3.1
									1,427.5	1,455.5	28.0	0.076	435.1	443.6	8.5	2.60	22.2
VBA-07-97	5720.0	6500.0	10905.0	1404.2	428.0	-75	173		354.0	359.0	5.0	0.010	107.9	109.4	1.5	0.34	0.5
VBA-07-98	5320.0	6700.0	10950.0	1443.6	440.0	-63	173		818.0	833.0	15.0	0.068	249.3	253.9	4.6	2.33	10.7
									858.0	888.0	30.0	0.052	261.5	270.7	9.1	1.78	16.3
									913.0	979.6	66.6	0.108	278.3	298.6	20.3	3.71	75.2
								incl	913.0	958.0	45.0	0.148	278.3	292.0	13.7	5.09	69.8
									1,029.0	1,077.0	48.0	0.050	313.6	328.3	14.6	1.71	25.1
									1,152.0	1,255.5	103.5	0.018	351.1	382.7	31.5	0.62	19.5
VBA-07-99	5320.0	6700.0	10950.0	1551.8	473.0	-80	165		307.0	312.0	5.0	0.015	93.6	95.1	1.5	0.51	0.8
VBA-07-100	3872.0	6063.0	10925.0	1254.5	382.4	-50	170		708.0	715.5	7.5	0.048	215.8	218.1	2.3	1.63	3.7
									732.3	784.2	51.9	0.034	223.2	239.0	15.8	1.18	18.6
									1,143.8	1,150.6	6.8	0.059	348.6	350.7	2.1	2.04	4.2
VBA-07-101	3872.0	6063.0	10925.0	1729.0	527.0	-75	170		1,158.0	1,166.0	8.0	0.033	353.0	355.4	2.4	1.11	2.7
									1,206.0	1,214.0	8.0	0.234	367.6	370.0	2.4	8.02	19.6
									1,530.0	1,545.0	15.0	0.035	466.3	470.9	4.6	1.20	5.5
									1,564.0	1,571.0	7.0	0.050	476.7	478.8	2.1	1.71	3.7
									1,643.0	1,661.1	18.1	0.069	500.8	506.3	5.5	2.36	13.0
									1,709.3	1,719.2	9.9	0.227	521.0	524.0	3.0	7.77	23.5

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
VBA-07-102	4064.0	6119.0	10925.0	1463.3	446.0	-50	170		805.4	860.8	55.4	0.064	245.5	262.4	16.9	2.18	36.8
								incl	805.4	818.0	12.6	0.137	245.5	249.3	3.8	4.71	18.1
									1,165.7	1,175.0	9.3	0.176	355.3	358.1	2.8	6.02	17.1
VBA-07-103	3411.0	6322.0	10925.0	2329.4	710.0	-68	155		1,054.0	1,082.0	28.0	0.038	321.3	329.8	8.5	1.29	11.0
									1,146.0	1,157.0	11.0	0.024	349.3	352.7	3.4	0.82	2.7
									1,652.0	1,680.5	28.5	0.101	503.5	512.2	8.7	3.48	30.2
									1,908.0	1,924.0	16.0	0.065	581.6	586.4	4.9	2.24	10.9
VBA-07-104	3516.0	6850.0	10930.0	646.3	197.0	-74	180		1,933.0	1,960.0	27.0	0.077	589.2	597.4	8.2	2.65	21.8
VBA-07-104-B									2,430.2	2,449.3	19.1	0.071	740.7	746.5	5.8	2.42	14.1
VBA-07-105	4020.0	6900.0	10930.0	2951.5	899.6	-75	170		2,019.7	2,027.0	7.3	0.045	615.6	617.8	2.2	1.53	3.4
									2,311.0	2,332.0	21.0	0.025	704.4	710.8	6.4	0.85	5.5
VG-07-106ext	2750.0	12020.0	11060.0	1662.8	506.8	-60	175		nsv								
VBA-08-107	3520.0	7552.0	10985.0	2860.9	872.0	-76	160		2,598.0	2,648.0	50.0	0.018	791.9	807.1	15.2	0.62	9.5
VBA-08-108	3520.0	7552.0	10985.0	3623.2		-80	175		3,010.0	3,018.0	8.0	0.043	917.4	919.9	2.4	1.46	3.6
									3,080.0	3,093.0	13.0	0.030	938.8	942.7	4.0	1.02	4.0
									3,117.8	3,126.5	8.7	0.050	950.3	953.0	2.7	1.72	4.6
									3,256.0	3,286.0	30.0	0.358	992.4	1,001.6	9.1	12.28	112.3
								incl	3,281.0	3,286.0	5.0	2.072	1,000.0	1,001.6	1.5	71.04	108.3
									3,330.0	3,340.0	10.0	0.032	1,015.0	1,018.0	3.0	1.08	3.3
									3,475.2	3,504.0	28.8	0.029	1,059.2	1,068.0	8.8	0.99	8.7
VBA-08-109	3662.9	7733.4	11060.9	4622.7	1409.0	-85	213		34.0	51.0	17.0	0.019	10.4	15.5	5.2	0.67	3.5
									2,905.0	2,915.0	10.0	0.032	885.4	888.5	3.0	1.10	3.3
									3,447.0	3,454.6	7.6	0.044	1,050.6	1,053.0	2.3	1.50	3.5
									3,798.5	3,807.5	9.0	0.029	1,157.8	1,160.5	2.7	0.98	2.7
VBA-08-110	4081.4	7580.8	11048.4	3120.8	951.2	-48	172		100.0	110.0	10.0	0.164	30.5	33.5	3.0	5.62	17.1
									2,608.0	2,626.5	18.5	0.036	794.9	800.6	5.6	1.25	7.0
VBA-08-111	3937.0	6798.0	11034.9	2831.3	863.0	-64	170		1,753.0	1,765.0	12.0	0.080	534.3	538.0	3.7	2.74	10.0
									1,789.0	1,813.0	24.0	0.024	545.3	552.6	7.3	0.81	5.9
									2,307.0	2,316.0	9.0	0.029	703.2	705.9	2.7	0.99	2.7
VBA-08-112	3728.7	6612.5	11028.1	2526.2	770.0	-66	165		2,099.0	2,139.0	40.0	0.039	639.8	652.0	12.2	1.32	16.1
									2,194.0	2,214.0	20.0	0.022	668.7	674.8	6.1	0.74	4.5
VBA-08-113	3425.5	6209.0	11016.4	2414.0	735.8	-68	175		873.0	878.0	5.0	0.075	266.1	267.6	1.5	2.57	3.9
									958.0	1,013.0	55.0	0.035	292.0	308.8	16.8	1.21	20.3
									1,073.0	1,083.0	10.0	0.039	327.1	330.1	3.0	1.34	4.1
									1,314.1	1,323.0	8.9	0.386	400.5	403.3	2.7	13.24	35.9
									1,375.0	1,508.2	133.2	0.041	419.1	459.7	40.6	1.40	56.7
								incl	1,314.1	1,323.0	8.9	0.386	400.5	403.3	2.7	13.24	35.9
									1,526.0	1,538.0	12.0	0.039	465.1	468.8	3.7	1.33	4.9
VBA-08-114	4077.0	7630.2	11044.8	2093.2	638.0	-75	174		437.0	457.0	20.0	0.033	133.2	139.3	6.1	1.12	6.8
									1,707.0	1,717.0	10.0	0.034	520.3	523.3	3.0	1.16	3.5
									1,759.0	1,808.0	49.0	0.062	536.1	551.1	14.9	2.14	32.0
									1,798.0	1,808.0	10.0	0.124	548.0	551.1	3.0	4.26	13.0
									1,828.0	1,843.0	15.0	0.038	557.2	561.7	4.6	1.31	6.0
									1,913.0	1,940.0	27.0	0.052	583.1	591.3	8.2	1.79	14.7
VBA-08-115	4077.1	7630.8	11044.5	1797.9	548.0	-84	165		1,095.0	1,105.0	10.0	0.034	333.8	336.8	3.0	1.17	3.6

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
									1,247.0	1,255.4	8.3	0.099	380.1	382.6	2.5	3.38	8.6
VBA-08-120	4854.4	6620.8	11011.9	1984.9	605.0	-55	170		1,409.8	1,418.0	8.2	0.029	429.7	432.2	2.5	0.98	2.4
									1,488.0	1,500.5	12.5	0.024	453.5	457.4	3.8	0.82	3.1
VBA-08-121	5318.1	6409.4	10994.5	1532.2	467.0	-60	173		618.0	626.0	8.0	0.028	188.4	190.8	2.4	0.94	2.3
									1,099.0	1,105.7	6.7	0.107	335.0	337.0	2.0	3.66	7.5
VBA-08-122	5523.0	6310.0	10997.8	1286.1	392.0	-58	177		951.0	958.0	7.0	0.020	289.9	292.0	2.1	0.68	1.4
				8283.5	6620.7				995.0	1,012.0	17.0	0.083	303.3	308.5	5.2	2.85	14.8
VBA-08-123	5724.4	6206.5	11000.5	1109.0	338.0	-45	178		752.0	774.0	22.0	0.081	229.2	235.9	6.7	2.78	18.7
									948.0	959.0	11.0	0.030	289.0	292.3	3.4	1.03	3.5
VBA-08-124	5724.6	6208.9	11000.6	1364.9	416.0	-67	178		438.6	455.0	16.4	0.020	133.7	138.7	5.0	0.68	3.4
									788.6	824.0	35.4	0.054	240.4	251.2	10.8	1.86	20.1
VBA-08-125	5724.5	6210.1	11000.2	626.6	191.0	-82	178		258.6	377.0	118.4	0.047	78.8	114.9	36.1	1.60	57.9
								incl	258.6	277.3	18.7	0.137	78.8	84.5	5.7	4.68	26.7
VBA-08-126	4125.8	6448.6	11031.4	1789.0	545.3	-61	178		1,232.0	1,241.0	9.0	0.095	375.5	378.3	2.7	3.26	9.0
									1,344.6	1,355.0	10.4	0.032	409.8	413.0	3.2	1.08	3.4
									1,510.0	1,560.0	50.0	0.020	460.2	475.5	15.2	0.68	10.3
									1,615.0	1,639.0	24.0	0.036	492.3	499.6	7.3	1.25	9.1
VBA-08-127	3997.8	6601.3	11030.9	1865.0	568.5	-55	176		1,276.0	1,282.0	6.0	0.059	388.9	390.8	1.8	2.02	3.7
									1,455.0	1,470.0	15.0	0.027	443.5	448.1	4.6	0.94	4.3
									1,515.0	1,530.0	15.0	0.158	461.8	466.3	4.6	5.42	24.8
									1,667.0	1,681.0	14.0	0.046	508.1	512.4	4.3	1.57	6.7
									1,798.2	1,812.2	14.0	0.114	548.1	552.4	4.3	3.90	16.6
VBA-08-128	4318.0	6507.5	11046.1	2221.1	677.0	-65	178		1,676.0	1,700.0	24.0	0.033	510.8	518.2	7.3	1.12	8.2
									1,787.0	1,812.0	25.0	0.013	544.7	552.3	7.6	0.45	3.4
									1,906.2	1,918.0	11.8	0.041	581.0	584.6	3.6	1.40	5.1
									2,118.8	2,136.8	18.0	0.035	645.8	651.3	5.5	1.20	6.6
VBA-08-129	4324.0	6499.0	11043.3	1976.0	602.3	-45	172		1,158.1	1,187.7	29.5	0.020	645.8	651.3	5.5	1.20	6.6
									1,252.0	1,264.0	12.0	0.023	353.0	362.0	9.0	0.68	6.1
									1,276.0	1,302.6	26.6	0.020	381.6	385.3	3.7	0.79	2.9
									1,627.0	1,639.0	12.0	0.107	388.9	397.0	8.1	0.67	5.4
									1,651.0	1,663.0	12.0	0.058	495.9	499.6	3.7	3.67	13.4
VBA-08-130	4421.1	6498.7	11052.8	1847.1	563.0	-59	176		1,277.0	1,302.0	25.0	0.019	503.2	506.9	3.7	1.99	7.3
									1,485.0	1,495.0	10.0	0.018	389.2	396.8	7.6	0.65	5.0
									1,625.0	1,635.0	10.0	0.077	469.4	477.0	7.6	1.41	10.8
									1,715.0	1,740.0	25.0	0.040	495.3	498.3	3.0	2.62	8.0
VBA-08-131	4422.9	6501.4	11059.8	1934.0	589.5	-45	165		1,195.5	1,207.4	11.8	0.125	522.7	530.4	7.6	1.36	10.3
VBA-08-132	5922.7	6023.9	10990.9	539.4	164.4	-64	5		191.0	245.0	54.0	0.041	58.2	74.7	16.5	1.40	23.0
VBA-08-133	5922.9	6025.7	10990.8	720.0	219.5	-86	178		345.0	360.0	15.0	0.040	105.2	109.7	4.6	1.36	6.2
									388.8	445.0	56.2	0.041	118.5	135.6	17.1	1.42	24.3
									505.0	514.8	9.8	0.035	153.9	156.9	3.0	1.19	3.5
VBA-08-134	5925.1	6012.5	10990.8	912.1	278.0	-65	178		304.3	315.0	10.7	0.157	92.8	96.0	3.3	5.39	17.6
									790.0	845.0	55.0	0.037	240.8	257.6	16.8	1.27	21.3
VBA-10-135W	5049.8	10485.1	8513.9	4830.0	1472.2	-50	193		6,550.0	6,555.0	5.0	0.087	1,996.4	1,998.0	1.5	2.98	4.5
				final	start				7,353.0	7,358.0	5.0	0.041	2,241.2	2,242.7	1.5	1.42	2.2

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
				7897.0	3067.0			7,687.5	7,695.5	8.0	0.071	2,343.2	2,345.6	2.4	2.45	6.0
VBA-10-136	6121.4	5890.1	10984.6	492.3	150.1	-80	0	nsv								
VBA-10-137	6026.4	5911.2	10988.8	587.3	179.0	-80	0	263.0	293.4	30.4	0.025	80.2	89.4	9.3	0.84	7.8
VBA-10-138B	6026.4	5911.2	10988.8	410.1	125.0	-60	354	302.1	307.1	5.0	0.598	92.1	93.6	1.5	20.50	31.2
VBA-10-139	6026.4	5911.2	10988.8	479.0	146.0	-43	354	170.5	188.3	17.8	0.063	52.0	57.4	5.4	2.17	11.8
VBA-10-140	6226.8	5984.4	10976.9	528.2	161.0	-70	180	291.8	327.7	35.9	0.035	88.9	99.9	10.9	1.19	13.0
VBA-10-141	6226.8	5984.4	10976.9	459.3	140.0	-85	180	nsv								
VBA-10-142	5820.0	6065.0	11020.1	439.6	134.0	-85	3	303.3	306.8	3.5	0.042	92.4	93.5	1.1	1.44	1.5
VBA-10-143	5820.0	6065.0	11020.0	289.5	88.2	-45	3	152.4	158.1	5.7	0.035	46.5	48.2	1.7	1.21	2.1
								167.4	171.7	4.3	0.074	51.0	52.3	1.3	2.52	3.3
VBA-10-144	5620.0	6075.0	11020.0	705.4	215.0	-86	0	505.0	519.2	14.2	0.161	153.9	158.3	4.3	5.53	23.9
VBA-10-145	5620.0	6075.0	11020.0	390.4	119.0	-45	0	250.5	254.0	3.5	0.080	76.4	77.4	1.1	2.75	2.9
VBA-10-146	5720.0	5844.0	11030.0	675.9	206.0	-70	3	556.9	562.2	5.3	0.166	169.7	171.4	1.6	5.69	9.2
								600.0	610.0	10.0	0.036	182.9	185.9	3.0	1.23	3.7
VBA-10-147	5520.0	5905.0	11014.1	1161.4	354.0	-85	348	885.7	889.4	3.7	0.044	270.0	271.1	1.1	1.50	1.7
VBA-10-148	5520.0	5925.0	11014.1	803.8	245.0	-65	355	601.8	606.3	4.5	0.059	183.4	184.8	1.4	2.01	2.8
VBA-10-149	5520.0	5925.0	11014.1	674.5	205.6	-45	355	572.3	577.3	5.0	0.018	174.4	176.0	1.5	0.61	0.9
VBA-10-150	5420.0	5900.0	11029.8	1108.9	338.0	-75	355	814.5	822.5	8.0	0.050	248.3	250.7	2.4	1.71	4.2
								845.5	858.3	12.8	0.030	257.7	261.6	3.9	1.02	4.0
								888.3	904.7	16.4	0.117	270.8	275.8	5.0	4.02	20.1
								983.3	1,001.5	18.2	0.030	299.7	305.3	5.5	1.03	5.7
								1,018.0	1,031.0	13.0	0.043	310.3	314.2	4.0	1.46	5.8
VBA-10-151	5420.0	5900.0	11030.0	725.1	221.0	-50	355	670.4	675.4	5.0	0.061	204.3	205.9	1.5	2.09	3.2
VBA-10-152	5420.0	5900.0	11000.0	1198.2	365.2	-75	175	671.1	680.1	9.0	0.022	204.6	207.3	2.7	0.74	2.0
								1,012.4	1,058.9	46.5	0.061	308.6	322.8	14.2	2.09	29.6
								1,123.5	1,128.5	5.0	0.072	342.4	344.0	1.5	2.46	3.7
VBA-10-153	5429.8	5898.7	11033.8	981.5	299.2	-45	178	486.0	498.9	12.9	0.065	148.1	152.1	3.9	2.22	8.7
								592.4	627.5	35.1	0.055	180.6	191.3	10.7	1.90	20.3
								689.5	708.0	18.5	0.052	210.2	215.8	5.6	1.77	10.0
								726.0	749.6	23.6	0.036	221.3	228.5	7.2	1.23	8.8
								765.0	849.4	84.4	0.047	233.2	258.9	25.7	1.62	41.7
VBA-10-154	5201.1	5877.0	11038.5	1286.1	392.0	-70	355	1,139.9	1,155.6	15.7	0.079	347.4	352.2	4.8	2.70	12.9
								1,206.0	1,223.0	17.0	0.031	367.6	372.8	5.2	1.06	5.5
VBA-10-155	5201.0	5879.2	11038.5	1056.0	321.9	-45	350	774.3	777.9	3.6	0.076	236.0	237.1	1.1	2.60	2.9
VBA-11-156	5002.7	5682.0	11030.2	1522.3	464.0	-85	355	903.0	914.0	11.0	0.059	275.2	278.6	3.4	2.01	6.7
								960.7	966.3	5.6	0.044	292.8	294.5	1.7	1.51	2.6
								1,026.0	1,032.1	6.1	0.062	312.7	314.6	1.9	2.13	4.0
VBA-11-157	5002.4	5683.2	11030.6	1492.8	455.0	-65	355	1,331.5	1,343.1	11.6	0.015	405.8	409.4	3.5	0.51	1.8

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
								1,415.9	1,426.3	10.4	0.019	431.6	434.7	3.2	0.64	2.0
VBA-11-158	5002.5	5686.1	11031.0	1581.4	482.0	-45	355	1,039.0	1,049.7	10.7	0.020	316.7	319.9	3.3	0.70	2.3
VBA-11-159	5004.2	5680.9	11031.0	867.0	264.3	-50	175	512.0	530.5	18.5	0.180	156.1	161.7	5.6	6.16	34.7
VBA-11-159								551.6	554.0	2.4	0.117	168.1	168.9	0.7	4.01	2.9
VBA-11-160	5099.9	5899.2	11046.8	1404.2	428.0	-80	355	1,203.6	1,209.6	6.0	0.061	366.9	368.7	1.8	2.10	3.8
								1,219.7	1,223.3	3.6	0.087	371.8	372.9	1.1	2.99	3.3
								1,346.0	1,374.0	28.0	0.033	410.3	418.8	8.5	1.12	9.6
VBA-11-161	5099.9	5899.2	11046.8	1197.5	365.0	-60	355	1,046.1	1,072.0	25.9	0.039	318.9	326.7	7.9	1.34	10.6
VBA-11-162	4902.5	6571.6	11017.0	1857.0	566.0	-65	175	1,261.5	1,265.8	4.3	0.144	384.5	385.8	1.3	4.95	6.5
								1,305.8	1,310.3	4.5	0.046	398.0	399.4	1.4	1.56	2.1
								1,317.9	1,334.0	16.1	0.049	401.7	406.6	4.9	1.67	8.2
								1,381.3	1,386.0	4.7	0.063	421.0	422.5	1.4	2.17	3.1
VBA-11-163	4902.6	6569.2	11016.5	1857.0	566.0	-50	175	1,301.5	1,306.1	4.6	0.055	396.7	398.1	1.4	1.89	2.6
								1,363.2	1,384.0	20.8	0.101	415.5	421.8	6.3	3.46	21.9
								incl				incl		1.2	8.38	10.1
								1,405.6	1,485.1	79.5	0.172	428.4	452.7	24.2	5.89	142.7
								incl		20.8	0.572	incl		6.3	19.62	124.4
								incl			2.234	incl		1.5	76.60	114.9
								1,534.3	1,538.0	3.7	0.077	467.7	468.8	1.1	2.65	3.0
								1,568.2	1,573.4	5.2	0.101	478.0	479.6	1.6	3.47	5.5
								1,606.0	1,610.1	4.1	0.098	489.5	490.8	1.2	3.37	4.2
VBA-11-164	4901.2	6572.4	11015.1	2171.9	662.0	-45	350	36.7	46.1	9.4	0.138	11.2	14.1	2.9	4.74	13.6
								419.0	423.7	4.7	0.085	127.7	129.1	1.4	2.90	4.2
								442.1	534.5	92.4	0.053	134.8	162.9	28.2	1.83	51.6
								include		8.0	0.219	incl		2.4	7.50	18.3
								1,729.0	1,732.4	3.4	0.160	527.0	528.0	1.0	5.50	5.7
VBA-11-165	4757.7	6476.1	11040.0	1851.2	564.2	-55	175	1,337.5	1,341.6	4.1	0.078	407.7	408.9	1.2	2.67	3.3
								1,365.0	1,368.1	3.1	0.060	416.1	417.0	0.9	2.04	1.9
								1,398.9	1,438.5	39.6	0.090	426.4	438.5	12.1	3.08	37.2
								incl		11.1	0.257	incl		3.4	8.82	29.9
								1,599.5	1,605.4	5.9	0.059	487.5	489.3	1.8	2.04	3.7
VBA-11-166	4650.7	6814.0	11024.5	2251.6	686.3	-55	180	1,562.0	1,567.1	5.1	0.034	476.1	477.7	1.6	1.15	1.8
								1,821.0	1,825.1	4.1	0.028	555.0	556.3	1.2	0.96	1.2
VBA-11-167	4708.9	6999.1	10997.5	1755.2	535.0	-70	174	1,290.8	1,304.6	13.8	0.044	393.4	397.6	4.2	1.50	6.3
VBA-11-168	4708.9	6999.1	10997.5	2306.4	703.0	-57	174	1,397.3	1,433.8	36.5	0.039	425.9	437.0	11.1	1.35	15.1
								incl		4.9	0.094	incl		1.5	3.21	4.8
VBA-11-169	4651.0	6816.2	11024.4	2155.4	657.0	-68	184	105.0	110.0	5.0	0.068	32.0	33.5	1.5	2.33	3.6
								1,392.1	1,397.0	4.9	0.064	424.3	425.8	1.5	2.21	3.3
								1,419.6	1,434.0	14.4	0.093	432.7	437.1	4.4	3.20	14.0
								incl		5.3		incl		1.6	5.12	8.3
VBA-11-170	4503.9	7191.9	11029.3	331.4	101.0	-45	174	nsv								
VBA-11-171	4503.9	7191.9	11029.3	2366.9	721.4	-70	174	694.1	700.1	6.0	0.066	211.6	213.4	1.8	2.25	4.1
								708.1	714.1	6.0	0.064	215.8	217.7	1.8	2.18	4.0
								1,454.9	1,460.9	6.0	0.050	443.5	445.3	1.8	1.70	3.1

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
VBA-11-172	3403.3	6995.6	11032.5	2575.4	785.0	-53	90		1,651.4	1,663.4	12.0	0.163	503.3	507.0	3.7	5.58	20.4
									incl		5.0	0.272	incl		1.5	9.31	14.2
									1,695.1	1,701.1	6.0	0.092	516.7	518.5	1.8	3.17	5.8
									1,781.6	1,798.5	16.9	0.050	543.0	548.2	5.2	1.72	8.9
									2,250.9	2,260.9	10.0	0.029	686.1	689.1	3.0	1.00	3.0
									2,367.6	2,401.0	33.4	0.021	721.6	731.8	10.2	0.72	7.3
									2,440.0	2,445.0	5.0	0.052	743.7	745.2	1.5	1.77	2.7
VBA-11-173	4426.8	8298.2	10989.6	1092.5	333.0	-45	88		688.3	698.0	9.7	0.034	209.8	212.8	3.0	1.15	3.4
									713.6	725.5	11.9	0.107	217.5	221.1	3.6	3.65	13.3
VBA-11-174	3824.5	6992.7	11057.8	1545.3	471.0	-45	90		1,399.5	1,407.5	8.0	0.053	426.6	429.0	2.4	1.81	4.4
									1,407.5	1,411.1	3.6	UGO	429.0	430.1	1.1	UGO	
									1,411.1	1,420.6	9.5	0.077	430.1	433.0	2.9	2.63	7.6
									1,420.6	1,473.0	52.4	UGO	433.0	449.0	16.0	UGO	
									1,473.0	1,477.5	4.5	0.121	449.0	450.3	1.4	4.16	5.7
VBA-11-175	4450.0	8100.0	10988.9	925.2	282.0	-45	88		600.2	633.9	19.7	0.062	182.9	193.2	10.3	2.13	21.9
									incl		16.1	0.105	incl		4.9	3.60	17.7
									incl		2.1	0.407	incl		0.6	13.95	8.9
VBA-11-176	3320.0	5498.0	11006.0	1131.9	345.0	-45	180		166.1	173.0	6.9	0.034	50.6	52.7	2.1	1.17	2.5
									190.0	202.6	12.6	0.042	57.9	61.8	3.8	1.43	5.5
VBA-11-177	3804.0	7200.4	11055.5	1260.1	384.1	-45	90		877.1	882.1	5.0	0.100	267.3	268.9	1.5	3.41	5.2
									1,252.5	1,260.1	7.6	0.078	381.8	384.1	2.3	2.68	6.2
VBA-11-178	4483.0	8167.0	10993.0	1504.4	458.5	-70	88		605.7	627.5	21.8	0.110	184.6	191.3	6.6	3.77	25.1
									incl		6.8		incl		2.1	8.00	16.6
									886.0	889.0	3.0	0.388	270.1	271.0	0.9	13.30	12.2
									925.0	930.0	5.0	0.059	281.9	283.5	1.5	2.03	3.1
									935.0	940.0	5.0	0.057	285.0	286.5	1.5	1.97	3.0
VBA-11-179	3320.0	5497.8	11005.6	1279.5	390.0	-58	180		202.0	222.9	20.9	0.034	61.6	67.9	6.4	1.17	7.5
									761.2	768.5	7.3	0.048	232.0	234.2	2.2	1.65	3.7
VBA-06-25e	2800.0	5600.0	11017.6	561.0	171.0	-45	180		675.8	688.0	12.2	0.034	206.0	209.7	3.7	1.18	4.4
	Extended								794.0	799.0	5.0	0.030	242.0	243.5	1.5	1.03	1.6
	from:	292.0	to:	853.0													
VBA-06-23e	3000.0	5497.2	11008.4	836.7	255.0	-45	180		162.2	184.0	21.8	0.099	49.4	56.1	6.6	3.39	22.5
	Extended								189.3	200.0	10.7	0.096	57.7	61.0	3.3	3.30	10.8
	from:	219.7	to:	1056.4					205.0	210.0	5.0	0.054	62.5	64.0	1.5	1.85	2.8
VBA-06-24e	3000.2	5500.3	9993.2	1082.7	330.0	-75	180		208.6	223.6	15.0	0.021	63.6	68.2	4.6	0.73	3.3
	Extended								233.6	259.8	26.2	0.060	71.2	79.2	8.0	2.07	16.5
	from:	272.3	to:	1355.0				incl	255.7	259.8	4.1	0.192	77.9	79.2	1.2	6.58	8.2
									265.0	268.5	3.5	0.088	80.8	81.8	1.1	3.02	3.2
VBA-06-21e	3100.5	5497.2	11006.1	780.8	238.0	-45	180		173.5	178.9	5.4	0.027	52.9	54.5	1.6	0.93	1.5
	Extended	from:	242.8	to:	1023.6				192.0	196.0	4.0	0.023	58.5	59.7	1.2	0.79	1.0
VBA-07-80e	3233.5	6028.0	11014.5	862.8	263.0	240	-57		747.2	755.8	8.6	0.057	227.7	230.4	2.6	1.96	5.1
	Extended								766.5	784.1	17.6	0.014	233.6	239.0	5.4	0.47	2.5
	from:	1128.6	to:	1991.4					791.1	848.4	57.3	0.088	241.1	258.6	17.5	3.02	52.7
								incl	791.1	800.8	9.7	0.132	241.1	244.1	3.0	4.53	13.4
								and	811.0	844.0	33.0	0.107	247.2	257.3	10.1	3.68	37.0
								incl	816.2	822.0	5.8	0.360	248.8	250.5	1.8	12.34	21.8
									1,186.0	1,188.6	2.6	0.147	361.5	362.3	0.8	5.03	4.0
									1,221.0	1,228.5	7.5	0.076	372.2	374.4	2.3	2.61	6.0

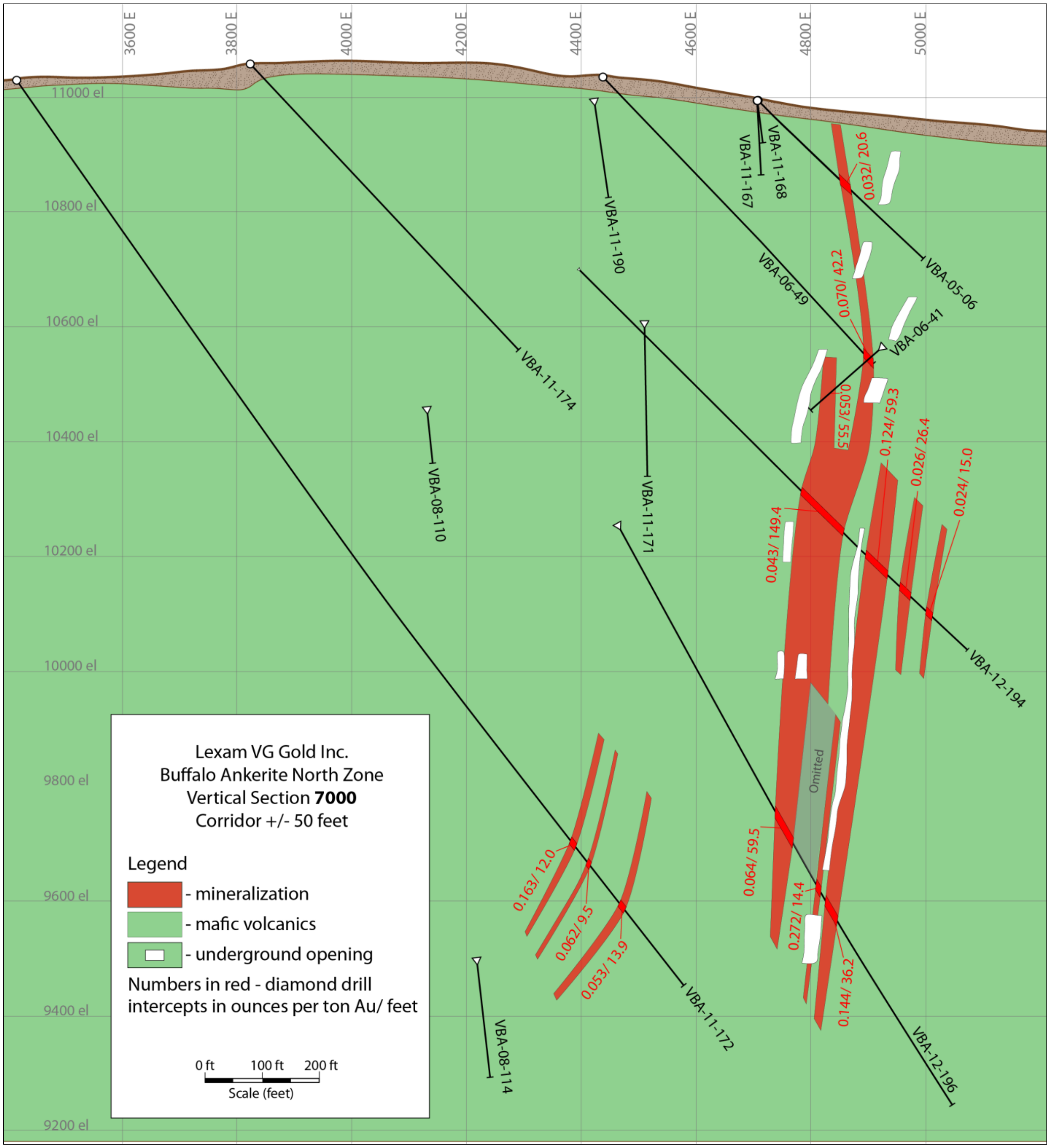
TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
								1,329.1	1,332.0	2.9	0.112	405.1	406.0	0.9	3.84	3.4
								1,366.5	1,393.7	27.2	0.025	416.5	424.8	8.3	0.87	7.2
VBA-07-82e	3233.0	6025.7	11014.2	410.1	125.0	185	-50	614.4	618.3	3.9	0.135	187.3	188.5	1.2	4.63	5.5
	Extended							967.8	989.1	3.9	0.135	295.0	301.5	6.5	1.11	7.2
	from:	961.3	to:	1371.4				1,081.0	1,084.7	3.9	0.135	329.5	330.6	1.1	3.06	3.5
								1,241.2	1,247.8	3.9	0.135	378.3	380.3	2.0	2.56	5.1
VBA-06-14e	3198.7	5599.6	11008.0	675.8	206.0	-50	180	239.1	276.2	37.1	0.042	72.9	84.2	11.3	1.44	16.3
	Extended							286.5	292.1	5.6	0.049	87.3	89.0	1.7	1.68	2.9
	from:	383.9	to:	1059.7				926.4	931.3	5.6	0.049	282.4	283.9	1.5	1.59	2.4
VBA-11-180	3803.0	7200.0	11055.0	1839.3	560.6	-65	90	950.1	988.5	38.4	0.060	289.6	301.3	11.7	2.07	24.2
								incl		5	0.206	incl		1.5	7.05	10.7
								1,008.5	1,013.5	5.0	0.207	307.4	308.9	1.5	7.11	10.8
								1,584.1	1,594.0	9.9	0.087	482.8	485.9	3.0	2.99	9.0
								1,641.5	1,692.3	50.8	0.052	500.3	515.8	15.5	1.77	27.4
								incl		5.6	0.177	incl		1.7	6.08	10.4
								1,712.3	1,745.0	32.7	0.047	521.9	531.9	10.0	1.60	15.9
								incl		8.6	0.103	incl		2.6	3.52	9.2
								1,810.4	1,814.3	3.9	0.069	551.8	553.0	1.2	2.35	2.8
								1,839.3	UGO	eoh						
VBA-11-181	3539.0	5611.0	11020.0	1200.8	366.0	-54	179	313.7	338.0	24.3	0.063	95.6	103.0	7.4	2.16	16.0
								356.5	380.3	23.8	0.150	108.7	115.9	7.3	5.15	37.4
								incl		3.9	0.790	incl		1.2	27.10	32.2
								811.7	815.3	3.6	0.057	247.4	248.5	1.1	1.94	2.1
VBA-11-181a	3539.0	5611.0	11020.0	39.4	12.0	-54	179	nsv								
VBA-11-182	3539.0	5611.0	11020.0	1009.2	307.6	-76	178	368.5	379.2	10.7	0.038	112.3	115.6	3.3	1.31	4.3
								399.4	420.5	21.1	0.085	121.7	128.2	6.4	2.93	18.8
								incl		4.5	0.340	incl		1.4	11.64	16.0
								508.1	518.0	9.9	0.031	154.9	157.9	3.0	1.05	3.2
								962.9	972.7	9.8	0.039	293.5	296.5	3.0	1.33	4.0
								1,009.2	UGO	eoh						
VBA-11-183	3892.3	7385.8	11054.2	1486.2	453.0	-45	90	467.0	472.0	5.0	0.078	142.3	143.9	1.5	2.68	4.1
								1,005.2	1,015.2	10.0	0.033	306.4	309.4	3.0	1.12	3.4
								1,045.2	1,124.2	79.0	0.052	318.6	342.7	24.1	1.79	43.2
								incl		10.0	0.140	incl		3.0	4.80	14.6
								and		9.0	0.104	and		2.7	3.58	9.8
								1,134.6	1,143.9	9.3	UGO	345.8	348.7	2.8	UGO	
								1,204.4	1,210.7	6.3	0.059	367.1	369.0	1.9	2.02	3.9
VBA-11-184	3528.5	5674.8	11030.0	3097.1	944.0	-68	90	2,279.5	2,289.5	6.3	0.041	694.8	697.8	3.0	1.41	4.3
								2,383.5	2,388.5	5.0	0.187	726.5	728.0	1.5	6.40	9.8
								3,083.0	3,095.0	12.0	UGO	939.7	943.4	3.7	UGO	
VBA-11-185	3726.6	7651.5	11056.2	1438.8	438.5	-50	82	1,141.6	1,160.8	19.2	0.018	348.0	353.8	5.9	0.63	3.7
								1,207.9	1,212.9	5.0	0.046	368.2	369.7	1.5	1.58	2.4
								1,259.9	1,264.2	4.3	0.093	384.0	385.3	1.3	3.19	4.2
								1,312.8	1,318.0	5.2	0.072	400.1	401.7	1.6	2.47	3.9
								1,357.3	1,359.0	1.7	0.082	413.7	414.2	0.5	2.82	1.5
								1,359.0	1,379.5	20.5	UGO	414.2	420.5	6.2	UGO	
								1,379.5	1,383.3	3.8	0.018	420.5	421.6	1.2	0.62	0.7
								1,390.8	1,403.5	12.7	UGO	423.9	427.8	3.9	UGO	
VBA-11-185b	3726.6	7651.5	11056.2	2303.1	702.0	-50	82	1,809.8	1,813.8	4.0	0.080	551.6	552.8	1.2	2.75	3.4

TABLE 10.2
BUFFALO ANKERITE DIAMOND DRILL TABLE (CONT'D)

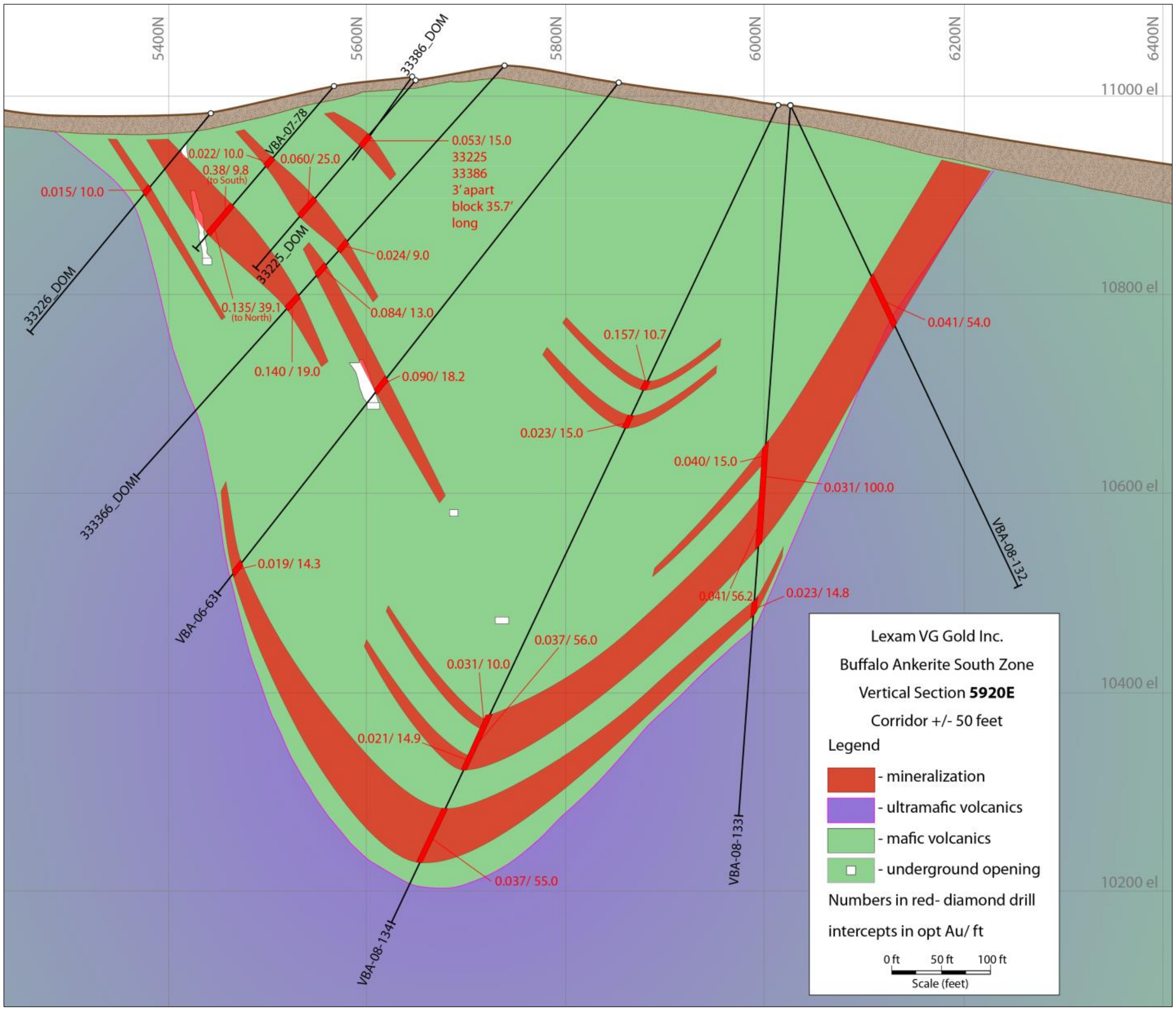
Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az		From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW (m)
									1,834.0	1,847.5	13.5	0.370	559.0	563.1	4.1	12.69	52.2
									incl		3.5	1.403	incl		1.1	48.10	51.3
									1,933.5	1,939.6	6.1	UGO	589.3	591.2	1.9	UGO	
									1,939.6	1,944.2	4.6	0.025	591.2	592.6	1.4	0.84	1.2
									2,010.3	2,019.0	8.7	0.045	612.7	615.4	2.7	1.54	4.1
									2,163.4	2,168.0	4.6	0.042	659.4	660.8	1.4	1.44	2.0
VBA-11-186	3639.9	7899.6	11032.8	1338.6	408.0	-45	84		1,182.2	1,186.6	4.4	0.033	360.3	361.7	1.3	1.12	1.5
									1,186.6	1,238.0	51.4	UGO	361.7	377.3	15.7	UGO	
									1,238.0	1,260.5	22.5	0.039	377.3	384.2	6.9	1.33	9.1
									1,300.2	1,328.7	28.5	UGO	396.3	405.0	8.7	UGO	
VBA-11-187	3638.8	7899.1	11034.1	2411.4	735.0	-60	80		1,333.7	1,413.6	79.9	0.035	406.5	430.9	24.4	1.20	29.2
									incl		38.6	0.056	incl		11.8	1.92	22.6
									1,508.0	1,515.5	7.5	0.034	459.6	461.9	2.3	1.17	2.7
									1,559.2	1,573.1	13.9	0.041	475.2	479.5	4.2	1.42	6.0
									1,591.5	1,619.5	28.0	0.124	485.1	493.6	8.5	4.25	36.3
									1,933.4	1,962.0	28.6	0.040	589.3	598.0	8.7	1.36	11.8
									2,180.3	2,183.5	3.2	0.128	664.6	665.5	1.0	4.38	4.3
VBA-11-188	4104.6	7301.8	11050.6	1151.6	351.0	-50	85		51.0	65.1	14.1	0.071	15.5	19.8	4.3	2.43	10.5
									822.0	825.0	3.0	0.103	250.5	251.5	0.9	3.53	3.2
									862.3	894.3	32.0	UGO	262.8	272.6	9.8	UGO	
									862.3	879.3	17.0	0.141	262.8	268.0	5.2	4.83	25.0
									935.7	945.2	9.5	0.029	285.2	288.1	2.9	0.98	2.8
									1,018.7	1,022.5	3.8	0.204	310.5	311.7	1.2	7.00	8.1
VBA-11-189	2820.0	7297.0	11024.0	3578.9	1090.8	-60	81		3,129.1	3,142.5	13.4	0.083	953.7	957.8	4.1	2.86	11.7
									3,193.5	3,205.5	12.0	0.088	973.4	977.0	3.7	3.00	11.0
									3,218.9	3,228.9	10.0	0.019	981.1	984.2	3.0	0.64	2.0
									3,243.0	3,246.0	3.0	0.079	988.5	989.4	0.9	2.71	2.5
									3,434.9	3,442.4	7.5	0.047	1,047.0	1,049.2	2.3	1.60	3.7
VBA-11-190	4416.6	7075.4	11043.3	2276.9	694.0	-60	166		61.5	70.0	8.5	0.088	18.7	21.3	2.6	3.03	7.8
									1,666.6	1,703.4	36.8	0.099	508.0	519.2	11.2	3.41	38.2
									incl		20.7	0.122	incl		6.3	4.18	26.4
VBA-11-191	3409.1	7574.0	11069.1	3211.9	979.0	-60	81		1,852.9	1,875.4	22.5	0.031	564.8	571.6	6.9	1.06	7.3
VBA-11-192	3010.4	6983.2	11027.9	2736.7	834.1	-55	78		2,086.5	2,093.8	7.3	0.219	636.0	638.2	2.2	7.52	16.7
									2,144.4	2,157.8	13.4	0.031	653.6	657.7	4.1	1.06	4.3
									2,188.5	2,230.1	41.6	0.022	667.1	679.7	12.7	0.77	9.8
									2,345.0	2,350.0	5.0	0.054	714.8	716.3	1.5	1.84	2.8
VBA-11-193	4202.6	7094.5	11047.2	939.1	286.2	-45	85		682.5	689.0	6.5	0.030	208.0	210.0	2.0	1.04	2.1
									793.0	800.5	7.5	0.090	241.7	244.0	2.3	3.09	7.1
									800.5	810.4	9.9	UGO	244.0	247.0	3.0	UGO	
									820.8	835.5	14.7	0.113	250.2	254.7	4.5	3.89	17.4
									incl		9.5	0.158	incl		2.9	5.40	15.6
									866.1	874.1	8.0	UGO	264.0	266.4	2.4	UGO	
									874.1	878.7	4.6	0.140	266.4	267.8	1.4	4.80	6.7
									889.1	939.1	50.0	UGO	271.0	286.2	15.2	UGO	
VBA-12-194	4056.0	6899.3	11042.7	1440.9	439.2	-45	82		634.0	644.0	10.0	0.028	193.2	196.3	3.0	0.97	2.9
									1,034.6	1,183.9	149.3	0.043	315.3	360.9	45.5	1.49	67.6
									1,074.6	1,082.6	8.0	0.172	327.5	330.0	2.4	5.89	14.3
									1,190.9	1,245.2	54.3	0.124	363.0	379.5	16.6	4.24	70.1
								incl	1,224.8	1,233.2	8.4	0.537	373.3	375.9	2.6	18.42	47.2
									1,274.1	1,300.5	26.4	0.026	388.3	396.4	8.0	0.89	7.2
									1,335.5	1,350.5	15.0	0.024	407.1	411.6	4.6	0.82	3.7

Figure 10.1 Buffalo Ankerite North Zone - Section 7000N



Source: *Lexam 2013*

Figure 10.2 Buffalo Ankerite South Zone - Section 5920E



Source: *Lexam 2013*

10.2 FULLER

Lexam completed significant surface diamond drilling on the Fuller property during the period 2004 through 2012. The majority of the drilling was conducted on the Contact Zone/Edwards porphyry area. Other drilling included in the resource is a significant amount of both surface and underground drilling completed by various operators during the period 1984-1987 and 1996-1999.

Diamond Drilling to date by Lexam VG and predecessor companies, 2004-2012, has totalled of 19,087.0 metres/62,621.2 feet of diamond core data from 65 drill holes.

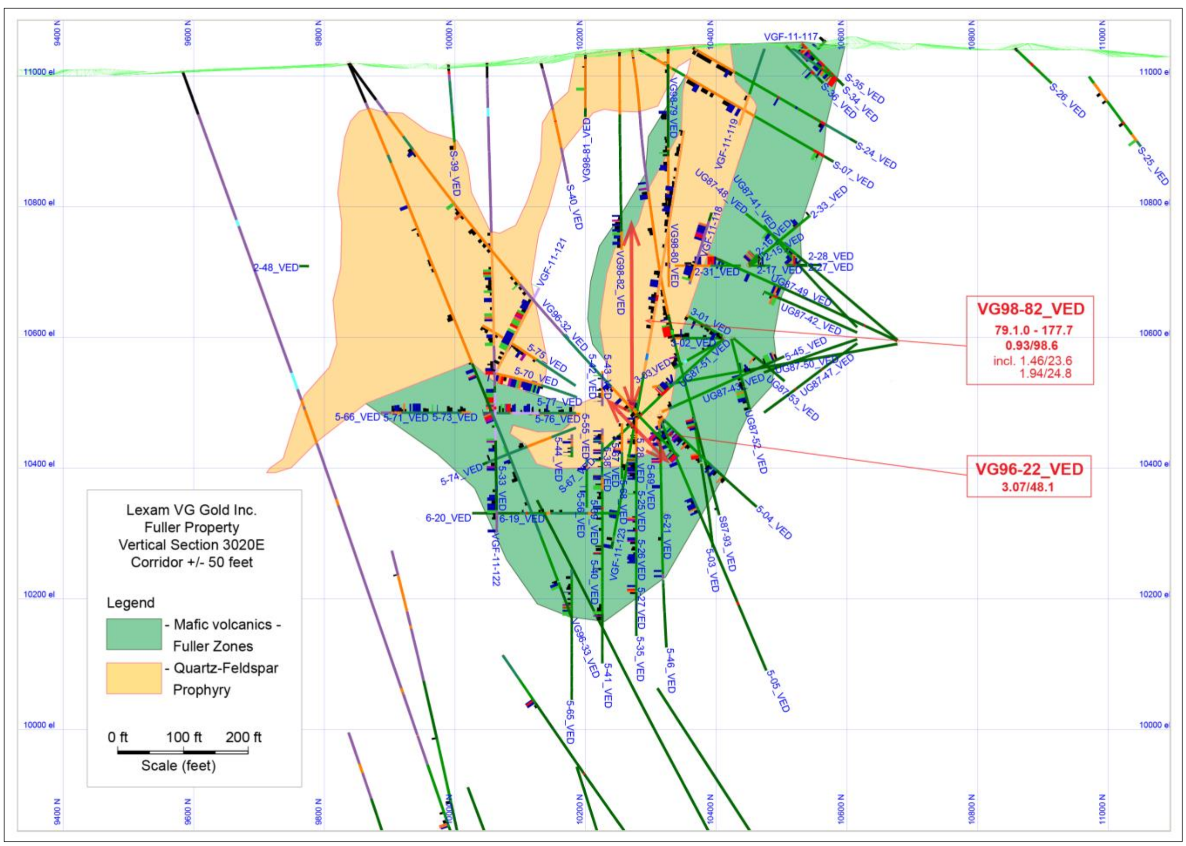
A Surface Diamond Drill Table with hole locations and collar details as well as significant assays is seen in Table 10.3.

The core or central area of the Fuller zones were well tested by the earlier drilling, therefore the focus of Lexam drilling was to expand the Fuller mineralization along strike or to depth

The Company tested targets including the east Paymaster boundary area and the south to southeast extension of Contact Zone mineralization on the south limb of the "Fuller syncline" towards the Buffalo Ankerite No. 5 shaft area. Assay results, in general, were mixed. Several mineralized/carbonate-altered zones were intersected, but gold assay results associated with these altered zones were predominantly not anomalous.

Additional drilling was completed to test for extensions of the Green Carb Zones 1 and 2 of the Fuller deposit. The holes tested for extensions of the Green Carb zone within 50 m of the existing ramp and underground workings, 50 to 100 m along strike from historic green carb zone intersects. Every hole intersected gold mineralization within or adjacent to the fuchsite and ankerite altered quartz-feldspar porphyry, host rock to the Fuller Green Carb zone mineralization.

Figure 10.3 Fuller Geological cross section - Section 3020E



Source: *Lexam 2013*

TABLE 10.3
FULLER LEXAM DIAMOND DRILL HOLE SUMMARY

HOLE-ID	East	North	Elev	Length (ft)	Length (m)	Az	Dip	incl	FROM (ft)	TO(ft)	ft	opt	FROM(m)	TO(m)	m	gpt	g x w
VG04-90	4424.7	9698.4	11016.1	912.1	278.0	2	-65		714.9	754.8	39.9	0.016	217.9	230.1	12.2	0.55	6.7
VG04-91	4428.2	9802.9	11016.9	607.0	185.0	0	-48		nsv								
VG04-92	4190.5	10660.3	11018.0	1,049.9	320.0	180	-48		949.2	988.4	39.2	0.056	289.3	301.3	11.9	1.92	22.9
VG04-93	2731.2	9323.6	11034.4	832.8	253.8	285	-45		584.3	592.0	7.7	0.031	178.1	180.4	2.3	1.05	2.5
VG04-94	2731.2	9323.6	11034.4	823.5	251.0	255	-45		655.6	660.6	5.0	0.020	199.8	201.4	1.5	0.69	1.0
VG-06-95	4300.0	12375.0	11049.8	1,069.6	326.0	180	-45		nsv								
VG-06-96	4308.1	11598.7	11055.2	1,302.5	397.0	180	-45		697.5	713.1	15.6	0.035	212.6	217.4	4.8	1.21	5.7
VG-06-97	3801.3	11182.3	11035.9	902.2	275.0	180	-45		234.8	239.2	4.4	0.034	71.6	72.9	1.3	1.17	1.6
									364.2	369.6	5.4	0.051	111.0	112.7	1.6	1.75	2.9
VG-06-98	3411.4	11187.1	11022.9	803.8	245.0	180	-45		198.6	203.1						1.51	
VG-06-99	3625.4	10024.8	11015.6	321.5	98.0	360	-53		264.0	294.0	30.0	0.091	80.5	89.6	9.1	3.13	28.6
VG-06-100	3604.3	9877.4	11015.5	725.1	221.0	360	-62		435.0	447.0	12.0	0.032	132.6	136.2	3.7	1.09	4.0
									668.8	722.0	53.2	0.068	203.9	220.1	16.2	2.33	37.8
VG-06-101	3717.1	9856.7	11011.2	823.5	251.0	360	-65		764.2	799.0	34.8	0.044	232.9	243.5	10.6	1.50	15.9
VG-06-102	3724.1	9932.7	11011.6	508.5	155.0	360	-54		96.5	112.0	15.5	0.037	29.4	34.1	4.7	1.26	6.0
									197.0	209.6	12.6	0.037	60.0	63.9	3.8	1.27	4.9
									384.0	404.9	20.9	0.030	117.0	123.4	6.4	1.04	6.6
									450.8	482.0	31.2	0.055	137.4	146.9	9.5	1.90	18.0
VG-06-103	3823.1	9862.0	11009.2	695.5	212.0	360	-57		505.0	509.0	4.0	0.048	153.9	155.1	1.2	1.65	2.0
									545.0	549.0	4.0	0.055	166.1	167.3	1.2	1.89	2.3
VG-06-104	2396.0	9757.5	11060.3	557.7	170.0	200	-45		439.0	453.9	14.9	0.062	133.8	138.3	4.5	2.13	9.7
VG-06-105	2396.6	9758.5	11059.8	616.8	188.0	175	-55		413.0	424.5	11.5	0.085	125.9	129.4	3.5	2.93	10.3
VG-07-107	3246.9	9199.5	11010.0	843.2	257.0	220	-45		nsv								
VG-07-108	3246.9	9199.5	11010.0	971.1	296.0	220	-75		nsv								
VG-07-109	3857.1	8998.1	11003.8	924.1	281.7	220	-45		nsv								
VG-07-110	2593.4	9156.3	11039.2	547.9	167.0	40	-45		nsv								
VGF-09-111	3219.1	11449.4	11046.2	2,782.2	848.0	166	-62		618.0	622.0	4.0	0.044	188.4	189.6	1.2	1.52	1.9
									2,306.0	2,312.0	6.0	0.241	702.9	704.7	1.8	8.27	15.1
VGF-09-112	3329.3	11394.3	11043.0	2,585.3	788.0	155	-60		250.0	255.0	5.0	0.029	76.2	77.7	1.5	1.00	1.5
									695.0	697.9	2.9	0.055	211.8	212.7	0.9	1.88	1.7
									910.0	918.0	8.0	0.117	277.4	279.8	2.4	4.03	9.8
VGF-10-113	3516.5	11400.0	11043.0	2,144.0	653.5	154	-59		605.0	608.0	3.0	0.084	184.4	185.3	0.9	2.87	2.6
									655.0	659.0	4.0	0.163	199.6	200.9	1.2	5.58	6.8
									758.0	764.0	6.0	0.095	231.0	232.9	1.8	3.24	5.9
VGF-11-114	4429.3	12379.9	11042.2	2,136.0	651.1	160	-65		551.5	565.5	14.0	0.045	168.1	172.3	4.3	1.55	6.6
									1,660.2	1,674.2	14.0	0.083	506.0	510.3	4.3	2.84	12.1
VGF-11-115	4429.9	12378.2	11042.9	1,893.1	577.0	160	-45		1,145.7	1,150.1	4.4	0.039	349.2	350.6	1.3	1.32	1.8
									1,552.4	1,558.5	6.1	0.033	473.2	475.0	1.9	1.14	2.1
VGF-11-116	4623.2	12390.1	11046.1	3,756.6	1,145.0	160	-65		1,540.9	1,544.9	4.0	0.109	469.7	470.9	1.2	3.72	4.5
									2,334.8	2,346.7	11.9	0.174	711.6	715.3	3.6	5.95	21.6
									2,637.4	2,648.0	10.6	0.054	803.9	807.1	3.2	1.84	6.0
VGF-11-117	2978.1	10558.5	11053.8	1,650.2	503.0	208	-45		17.0	33.0	16.0	0.042	5.2	10.1	4.9	1.43	7.0
									77.0	85.0	8.0	0.152	23.5	25.9	2.4	5.22	12.7
									181.3	451.4	270.1	0.022	55.3	137.6	82.3	0.75	61.9
								incl	318.0	338.0	20.0	0.231	96.9	103.0	6.1	7.93	48.3
								incl	322.0	334.0	12.0	0.358	98.1	101.8	3.7	12.27	44.9
									619.0	661.5	42.5	0.079	188.7	201.6	13.0	2.71	35.2
								incl	649.0	657.0	8.0	0.239	197.8	200.3	2.4	8.20	20.0
									836.0	844.0	8.0	0.033	254.8	257.3	2.4	1.13	2.7
									1,173.9	1,179.0	5.1	0.1502	357.8	359.4	1.6	5.15	8.0
									1,546.1	1,554.0	7.9	0.025	471.3	473.7	2.4	0.86	2.1
VGF-11-118	2681.4	10482.8	11056.7	1,525.6	465.0	108	-45		44.0	48.0	4.0	0.114	13.4	14.6	1.2	3.91	4.8
									180.0	210.0	30.0	0.051	54.9	64.0	9.1	1.74	15.9
									302.0	345.0	43.0	0.013	92.0	105.2	13.1	0.44	5.8
									371.0	374.6	3.6	0.077	113.1	114.2	1.1	2.64	2.9
									408.0	442.9	34.9	0.160	124.4	135.0	10.6	5.48	58.3
								incl	428.0	438.0	10.0	0.340	130.5	133.5	3.0	11.67	35.6
									494.0	527.7	33.7	0.056	150.6	160.8	10.3	1.91	19.7
									558.4	572.0	13.6	0.104	170.2	174.3	4.1	3.56	14.8

TABLE 10.3
FULLER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

HOLE-ID	East	North	Elev	Length (ft)	Length (m)	Az	Dip	incl	FROM (ft)	TO(ft)	ft	opt	FROM(m)	TO(m)	m	gpt	g x w
									804.0	817.0	13.0	0.051	245.1	249.0	4.0	1.74	6.9
									883.0	901.8	18.8	0.032	269.1	274.9	5.7	1.11	6.3
									912.1	937.0	24.9	0.076	278.0	285.6	7.6	2.60	19.7
									1,112.0	1,129.0	17.0	0.022	338.9	344.1	5.2	0.74	3.8
VGF-11-119	2961.7	10470.0	11048.5	1,515.7	462.0	111	-45		355.6	368.0	12.4	0.131	108.4	112.2	3.8	4.49	17.0
									659.0	663.0	4.0	0.222	200.9	202.1	1.2	7.62	9.3
									1,015.0	1,035.0	20.0	0.052	309.4	315.5	6.1	1.78	10.9
									1,048.6	1,072.8	24.2	0.015	319.6	327.0	7.4	0.53	3.9
VGF-11-120	3143.2	10404.8	11034.6	698.8	213.0	111	-45		67.2	71.0	3.8	0.032	20.5	21.6	1.2	1.11	1.3
VGF-11-121	2657.6	10314.1	11062.7	1,013.8	309.0	120	-47		86.1	132.0	45.9	0.027	26.2	40.2	14.0	0.94	13.2
									86.1	109.3	23.2	0.017	26.2	33.3	7.1	0.59	4.2
									529.0	537.7	8.7	0.082	161.2	163.9	2.7	2.81	7.4
									555.7	720.3	164.6	0.080	169.4	219.5	50.2	2.73	137.1
									569.7	670.0	100.3	0.123	173.6	204.2	30.6	4.23	129.3
									596.3	625.0	28.7	0.248	181.8	190.5	8.7	8.49	74.3
VGF-11-122	3034.7	10040.8	11015.3	718.5	219.0	0	-90		276.0	280.0	4.0	0.036	84.1	85.3	1.2	1.25	1.5
									316.0	718.5	402.5	0.074	96.3	219.0	122.7	2.53	310.4
								incl	316.0	581.0	265.0	0.098	96.3	177.1	80.8	3.37	272.3
								incl	320.0	328.0	8.0	0.233	97.5	100.0	2.4	7.98	19.5
								and	467.0	477.0	10.0	0.347	142.3	145.4	3.0	11.90	36.3
								and	504.5	530.4	25.9	0.533	153.8	161.7	7.9	18.28	144.3
								incl	511.8	520.0	8.2	1.362	156.0	158.5	2.5	46.70	116.7
VGF-11-123	3082.9	10383.6	11041.1	780.8	238.0	180	-80		0.0	77.0	77.0	0.037	0.0	23.5	23.5	1.26	29.6
									155.0	169.0	14.0	0.017	47.2	51.5	4.3	0.59	2.5
									216.5	278.0	61.5	0.025	66.0	84.7	18.7	0.85	16.0
								incl	216.5	256.0	39.5	0.030	66.0	78.0	12.0	1.04	12.5
									348.0	408.0	60.0	0.030	106.1	124.4	18.3	1.03	18.8
								incl	348.0	373.0	25.0	0.047	106.1	113.7	7.6	1.62	12.4
									549.5	568.5	19.0	0.093	167.5	173.3	5.8	3.18	18.4
									722.0	734.0	12.0	0.041	220.1	223.7	3.7	1.40	5.1
VGF-11-124	3520.8	10198.0	11017.5	1,190.9	363.0	0	-90		113.0	336.5	223.5	0.026	34.4	102.6	68.1	0.87	59.5
									131.9	157.0	25.1	0.023	40.2	47.9	7.7	0.80	6.1
									182.0	207.9	25.9	0.096	55.5	63.4	7.9	3.30	26.1
									182.0	196.0	14.0	0.108	55.5	59.7	4.3	3.70	15.8
									460.6	515.0	54.4	0.168	140.4	157.0	16.6	5.76	95.5
								incl	472.0	495.7	23.7	0.222	143.9	151.1	7.2	7.61	55.0
									649.8	714.0	64.2	0.044	198.1	217.6	19.6	1.52	29.7
									991.4	1,038.5	47.1	0.023	302.2	316.5	14.4	0.79	11.4
									1,084.0	1,107.8	23.8	0.016	330.4	337.7	7.3	0.55	4.0
VGF-12-125	2810.6	10329.2	11052.5	777.5	237.0	177	-45		20.8	48.2	27.4	0.030	6.3	14.7	8.4	1.02	8.5
									272.0	297.0	25.0	0.079	82.9	90.5	7.6	2.72	20.7
									332.0	359.0	27.0	0.043	101.2	109.4	8.2	1.47	12.1
									450.2	454.0	3.8	0.054	137.2	138.4	1.2	1.84	2.1
									590.3	595.2	4.9	0.072	179.9	181.4	1.5	2.48	3.7
									609.3	616.0	6.7	0.039	185.7	187.8	2.0	1.34	2.7
VGF-12-126	3921.9	9890.4	11014.5	738.2	225.0	357	-45		210.0	216.0	6.0	0.047	64.0	65.8	1.8	1.63	3.0
									370.0	384.0	14.0	0.018	112.8	117.0	4.3	0.60	2.6
VGF-12-127	3929.0	9787.0	11011.2	1,131.9	345.0	358	-55		513.0	518.4	5.4	0.048	156.4	158.0	1.6	1.66	2.7
									607.1	622.5	15.4	0.025	185.0	189.7	4.7	0.85	4.0
									721.0	771.0	50.0	0.024	218.1	235.0	16.9	0.81	13.7
									788.0	791.0	3.0	0.082	240.2	241.1	0.9	2.82	2.6
VGF-12-128	3812.3	9864.6	11010.2	885.8	270.0	358	-50		399.0	433.0	34.0	0.045	121.6	132.0	10.4	1.53	15.9
									636.0	640.0	4.0	0.051	193.9	195.1	1.2	1.75	2.1
									711.0	714.0	3.0	0.067	216.7	217.6	0.9	2.28	2.1
									730.0	748.0	18.0	0.071	222.5	228.0	5.5	2.43	13.3
									777.6	783.0	5.4	0.156	237.0	238.7	1.6	5.36	8.8
VGF-12-129	3615.5	10038.4	11022.8	885.8	270.0	358	-45		167.0	175.0	8.0	0.046	50.9	53.3	2.4	1.59	3.9
									230.0	234.0	4.0	0.065	70.1	71.3	1.2	2.24	2.7
									283.0	288.0	5.0	0.063	86.3	87.8	1.5	2.16	3.3

TABLE 10.3
FULLER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

HOLE-ID	East	North	Elev	Length (ft)	Length (m)	Az	Dip	incl	FROM (ft)	TO(ft)	ft	opt	FROM(m)	TO(m)	m	gpt	g x w
									323.0	331.0	8.0	0.061	98.5	100.9	2.4	2.10	5.1
									401.0	407.2	6.2	0.079	122.2	124.1	1.9	2.72	5.1
VGF-12-130	3710.6	9876.2	11009.9	1,427.2	435.0	354	-57		428.8	453.3	24.5	0.013	130.7	138.2	7.5	0.45	3.4
									614.0	617.0	3.0	0.038	187.1	188.1	0.9	1.30	1.2
									640.0	648.0	8.0	0.097	195.1	197.5	2.4	3.34	8.1
									658.0	665.0	7.0	0.096	200.6	202.7	2.1	3.28	7.0
									724.0	729.0	5.0	0.150	220.7	222.2	1.5	5.13	7.8
									794.0	839.0	45.0	0.079	242.0	255.7	13.7	2.72	37.3
								incl	794.0	807.0	13.0	0.137	242.0	246.0	4.0	4.70	18.6
VGF-12-131	3516.7	10005.2	11020.6	931.8	284.0	358	-48		200.0	273.0	73.0	0.054	61.0	83.2	22.3	1.85	41.3
								incl	228.0	236.0	8.0	0.271	69.5	71.9	2.4	9.30	22.7
									583.4	594.3	10.9	0.062	177.8	181.1	3.3	2.11	7.0
VGF-12-132	3313.1	10183.0	11021.2	502.0	153.0	358	-45		152.0	182.0	30.0	0.061	46.3	55.5	9.1	2.11	19.2
									322.0	327.0	5.0	0.077	98.1	99.7	1.5	2.65	4.0
									401.0	422.4	21.4	0.091	122.2	128.7	6.5	3.10	20.3
VGF-12-133	3316.4	9817.3	11016.0	957.9	292.0	358	-45		490.0	510.0	20.0	0.185	149.4	155.4	6.1	6.33	38.6
									547.8	563.0	15.2	0.029	167.0	171.6	4.6	1.01	4.7
									841.0	849.3	8.3	0.036	256.3	258.9	2.5	1.23	3.1
									928.8	933.0	4.2	0.051	283.1	284.4	1.3	1.74	2.2
VGF-12-134	3011.8	10136.1	11028.4	541.3	165.0	360	-45		55.3	60.0	4.7	0.057	16.9	18.3	1.4	1.94	2.8
									231.0	258.0	27.0	0.044	70.4	78.6	8.2	1.51	12.4
									315.0	323.0	8.0	0.030	96.0	98.5	2.4	1.03	2.5
									494.0	498.0	4.0	0.099	150.6	151.8	1.2	3.41	4.2
VGF-12-135	2621.4	9925.2	11052.3	580.2	176.8	359	-45		180.0	190.0	10.0	0.026	54.9	57.9	3.0	0.89	2.7
									252.0	262.0	10.0	0.032	76.8	79.9	3.0	1.10	3.4
									293.0	313.0	20.0	0.038	89.3	95.4	6.1	1.31	8.0
									334.9	350.0	15.1	0.116	102.1	106.7	4.6	3.98	18.3
VGF-12-136	2827.8	10483.8	11052.6	902.2	275.0	178	-45		23.5	28.8	5.3	0.033	7.2	8.8	1.6	1.14	1.8
									40.5	52.0	11.5	0.067	12.3	15.8	3.5	2.31	8.1
									130.0	164.0	34.0	0.100	39.6	50.0	10.4	3.44	35.6
								incl	136.0	144.0	8.0	0.237	41.5	43.9	2.4	8.13	19.8
									362.0	411.3	49.3	0.025	110.3	125.4	15.0	0.86	12.9
									483.2	513.0	29.8	0.047	147.3	156.4	9.1	1.62	14.7
									483.2	495.8	12.6	0.032	147.3	151.1	3.8	1.09	4.2
									503.0	510.0	7.0	0.127	153.3	155.4	2.1	4.36	9.3
									776.0	786.0	10.0	0.024	236.5	239.6	3.0	0.83	2.5
VGF-12-137	3037.2	10640.4	11043.5	1,036.7	316.0	180	-45		117.2	134.0	16.8	0.042	35.7	40.8	5.1	1.43	7.3
									303.0	308.0	5.0	0.046	92.4	93.9	1.5	1.59	2.4
									389.0	416.0	27.0	0.049	118.6	126.8	8.2	1.67	13.8
									389.0	404.0	15.0	0.067	118.6	123.1	4.6	2.29	10.5
									511.0	580.0	69.0	0.071	155.8	176.8	21.0	2.44	51.3
								incl	535.0	557.0	22.0	0.139	163.1	169.8	6.7	4.77	32.0
								and	562.0	573.0	11.0	0.125	171.3	174.7	3.4	4.28	14.3
									647.0	655.0	8.0	0.028	197.2	199.6	2.4	0.95	2.3
									776.0	781.0	5.0	0.085	236.5	238.0	1.5	2.91	4.4
									790.0	811.0	21.0	0.074	240.8	247.2	6.4	2.54	16.3
									798.0	807.0	9.0	0.157	243.2	246.0	2.7	5.37	14.7
VGF-12-138	3113.9	10017.2	11013.0	708.7	216.0	355	-47		400.0	412.0	12.0	0.033	121.9	125.6	3.7	1.12	4.1
									430.0	458.0	28.0	0.020	131.1	139.6	8.5	0.68	5.8
									486.0	499.0	13.0	0.016	148.1	152.1	4.0	0.55	2.2
									590.0	608.0	18.0	0.175	179.8	185.3	5.5	6.01	33.0
VGF-12-139	3220.0	10075.0	11030.0	659.4	201.0	358	-47		303.7	318.0	14.3	0.104	92.6	96.9	4.4	3.58	15.6
									350.0	390.0	40.0	0.026	106.7	118.9	12.2	0.89	10.8
									419.0	434.0	15.0	0.031	127.7	132.3	4.6	1.06	4.8
									552.0	577.0	25.0	0.035	168.2	175.9	7.6	1.19	9.1
VGF-12-140	3416.8	10038.3	11019.2	708.7	216.0	356	-48		297.0	341.0	44.0	0.080	90.5	103.9	13.4	2.74	36.7
									542.0	545.0	3.0	0.257	165.2	166.1	0.9	8.82	8.1
									606.0	609.0	3.0	0.073	184.7	185.6	0.9	2.50	2.3
VGF-12-141	3123.7	9844.9	11010.9	649.6	198.0	357	-45		488.0	495.8	7.8	0.068	148.7	151.1	2.4	2.31	5.5

TABLE 10.3
FULLER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

HOLE-ID	East	North	Elev	Length (ft)	Length (m)	Az	Dip	incl	FROM (ft)	TO(ft)	ft	opt	FROM(m)	TO(m)	m	gpt	g x w
									510.8	536.8	26.0	0.028	155.7	163.6	7.9	0.97	7.7
VGF-12-142	3025.7	9944.2	11011.8	472.4	144.0	357	-45		428.5	436.5	8.0	0.142	130.6	133.0	2.4	4.86	11.9
VGF-12-143	3225.6	9883.3	11011.1	846.4	258.0	357	-45		421.3	426.3	5.0	0.073	128.4	129.9	1.5	2.49	3.8
									509.6	521.6	12.0	0.026	155.3	159.0	3.7	0.90	3.3
									532.3	537.3	5.0	0.048	162.2	163.8	1.5	1.64	2.5
									567.6	587.6	20.0	0.029	173.0	179.1	6.1	1.00	6.1
									637.0	650.0	13.0	0.102	194.2	198.1	4.0	3.49	13.8
									734.6	781.6	47.0	0.037	223.9	238.2	14.3	1.25	18.0
VGF-12-144	3225.6	9881.2	11010.8	698.8	213.0	357	-70		nsv								
VGF-12-145	2723.3	9778.4	11039.0	600.3	183.0	357	-48		455.1	459.3	4.2	0.037	138.7	140.0	1.3	1.28	1.6
									459.3	469.3	10.0	UGO	140.0	143.0	3.0	UGO	
									481.3	544.1	62.8	0.093	146.7	165.8	19.1	3.19	61.1
								incl	493.3	505.3	12.0	0.250	150.4	154.0	3.7	8.58	31.4
VGF-12-146	2723.3	9778.4	11039.0	679.1	207.0	357	-68		112.8	129.0	16.2	0.021	34.4	39.3	4.9	0.73	3.6
									159.0	180.5	21.5	0.086	48.5	55.0	6.6	2.95	19.3
									334.6	358.4	23.8	0.028	102.0	109.2	7.3	0.96	7.0
									515.3	521.3	6.0	0.037	157.1	158.9	1.8	1.28	2.3
VGF-12-147	2722.6	9780.5	11038.9	403.5	123.0	260	-45		101.2	118.1	16.9	0.014	30.8	36.0	5.2	0.48	2.5
									141.3	152.4	11.1	0.023	43.1	46.5	3.4	0.77	2.6
VGF-12-148	2722.6	9780.5	11038.9	574.4	175.1	260	-68		116.7	129.0	12.3	0.060	35.6	39.3	3.7	2.05	7.7
VGF-12-149	2720.4	9780.3	11038.4	826.7	252.0	227	-55		205.4	209.4	4.0	0.102	62.6	63.8	1.2	3.50	4.3
									637.4	642.4	5.0	0.028	194.3	195.8	1.5	0.95	1.5
VGF-12-150	2717.1	9774.6	11038.9	607.6	185.2	187	-45		nsv								
VGF-12-151	4124.0	9728.3	11015.3	846.4	258.0	357	-48		585.0	639.7	54.7	0.022	178.3	195.0	16.7	0.76	12.7
								incl	585.0	599.0	14.0	0.046	178.3	182.6	4.3	1.58	6.8
VGF-12-152	4320.9	9802.5	11012.4	590.5	180.0	357	-48		470.0	491.0	21.0	0.034	143.3	149.7	6.4	1.16	7.4
								inc	484.8	491.0	6.2	0.074	147.8	149.7	1.9	2.52	4.8
VGF-12-153	4321.5	9801.2	11013.1	767.7	234.0	357	-62		591.9	596.6	4.7	0.078	180.4	181.8	1.4	2.69	3.9
									727.0	745.0	18.0	0.023	221.6	227.1	5.5	0.78	4.3
VGF-12-154	4426.8	10122.2	11015.3	452.7	138.0	289	-45		132.7	137.7	5.0	0.081	40.4	42.0	1.5	2.79	4.3
65 holes				62,621.2	19,087.0												

A drill program was completed to test for extensions of the ML Zone of the Fuller deposit. The ML zone has not been included in the earlier resource estimates of the Fuller deposit. The ML zone likely represents the extension of the more extensively drilled Green Carb Zones that are located on the east side of the fold from the Fuller deposit underground workings. Exploration data is sparse on the western extensions of the Fuller deposit situated on the west side of the fold. Historic interpretation of the Fuller deposit was that of a single fold, however evidence existed that in fact the Fuller deposit was an "S" fold with the stratigraphy and mineralization extending to the west rather than trending southeast. Earlier exploration had just begun to confirm this. Drilling confirmed the existence of the Fuller zones with wide intersections of fuchsite ankerite alteration corresponding to the extension of the Green Carb 1 and 2 Zones.

In 2011 Lexam switched the focus of drilling at the Fuller property to evaluate the potential of a near surface or open pit model of mineralization, focusing on the mineralization within and proximal to the altered quartz-feldspar Porphyry within the synclinal structure of the Fuller zones (Figure 10.3). Drilling was targeted at testing the potential of the Edwards porphyry combined with the Fuller Zones to host near surface Au mineralization. Most of the holes intersected significant widths of Quartz-feldspar porphyry (QFP), mostly well altered, with intermittent sections of good pyrite mineralization.

Results include:

- Hole VGF-12-145 assayed 3.19 grams per tonne ("gpt") Au over 19.1 m including 8.58 gpt Au over 3.7 m at approximately 100 m vertical below surface.
- Hole VGF-12-137 assayed 2.44 gpt Au over 21.0 m including 8.58 gpt Au over 3.7 m at approximately 100 m vertical below surface.

The results indicated the potential for the Edwards porphyry to host near surface low grade Au mineralization. The potential here is in the near surface environment and re-examination of the resource with an open pit model and therefore a lower cut-off grade being used in the calculation of the resource.

10.2.1 Geology and Mineralization

Geologically the Fuller property is underlain by a generally eastwest trending assemblage of massive and pillowed mafic metavolcanic flows with minor variolitic flows. These have been traced onto the adjacent Placer Dome's Paymaster mine and Dome mine properties to the east. To the west the units are traceable into complex fold structures; part of the package is believed to be folded to the south around the South Tisdale anticline, while the northerly part of the package appears to trend onto the Hollinger mine property.

The structure from the Edwards shaft to the Buffalo Ankerite south zone is dominated by an S-shaped fold pattern expressed by the contact between an assemblage of largely massive to pillowed metavolcanic flows on the west, and talc-chlorite schist (meta-ultramafic rocks) with lesser mafic volcanic rock to the east and south. The mineralization on the property occurs stratigraphically above what appears to be the contact between the older ultramafic lower formation and the basaltic middle formation of the Tisdale group.

The geology is best known in the immediate area of the Fuller deposit in and around the ramp, where the most thorough geological data has been collected in recent years.

The ramp was put down in the vicinity of a proposed hinge of an interpreted easterly plunging, local synclinal fold (the "Fuller syncline") (Figure 10.3). In a general south to north direction, the succession of rocks includes talc-chlorite schist (metamorphosed ultramafic rocks), quartz-feldspar porphyry, pillowed amygdaloidal basaltic flows, massive basaltic flows, and a series of alternating units of massive, pillowed and amygdaloidal volcanic rocks.

The porphyry is interpreted to have been intruded prior to folding. Hydrothermally altered volcanic rocks, including a strongly altered unit with more than 50% quartz flooding, green mica and pyrite mineralization, are spatially associated with the porphyry; there are also large folded zones of highly carbonate altered volcanic rocks in contact with the porphyry stocks.

10.3 PAYMASTER

In June 2008, a 4-year option agreement with Goldcorp Inc. ("Goldcorp") was entered into to acquire a 60% interest in 16 patented mining claims that are adjacent to and on strike with the gold mineralized zones situated on the Fuller Property and the Buffalo Ankerite Property. Lexam VG has now completed the option phase and acquired a 60% interest in the property. The exploration will now be managed by Lexam VG on a 60:40 joint venture base with Goldcorp.

Acquisition of the Paymaster Property option gives Lexam VG the opportunity to expand the Fuller Property gold resource to the east, the Buffalo Ankerite South Mine and the Buffalo Ankerite North Mine mineralizations to the east, and to follow up on additional opportunities.

During the latter half of 2009 and through 2010, Lexam VG focused on exploration on the Paymaster West Porphyry Zone, which is situated entirely on the Paymaster property, and was mined in 1915-1928 from surface to 600 ft. Lexam VG sees potential for a near-surface open pit resource and for a higher-grade underground resource below the 600 ft current underground workings.

In June 2012, Lexam VG completed its earn-in requirements and elected to exercise the option to acquire 60% interest in the Paymaster Property. Goldcorp's back-in right expired in December 2012 and Lexam VG remains the manager of the joint venture.

A total of 136 drill holes for a total of 46,016.1 m have been drilled by Lexam on the Paymaster property. The majority of the program consisted of drilling at the 2/3 Shaft Porphyry area and the Buffalo Ankerite North Zone.

West Porphyry Zone or 2/3 Shaft Porphyry: The majority of the Lexam VG holes during the option period have been drilled on the West Porphyry or 2/3 Shaft area of the Paymaster Property. This area of the property was mined during the period 1915-1924 from the Paymaster 1, 2 and 3 shafts. The historic mining took place predominately between sections 5800E and 6300E, 500 ft (150 m) strike length, to a maximum depth of 600 ft (182 m). Most of the extraction was from above the 300 ft (91 m) level.

The program was following up on the success of the historic drilling as well as on drilling conducted by Placer Dome.

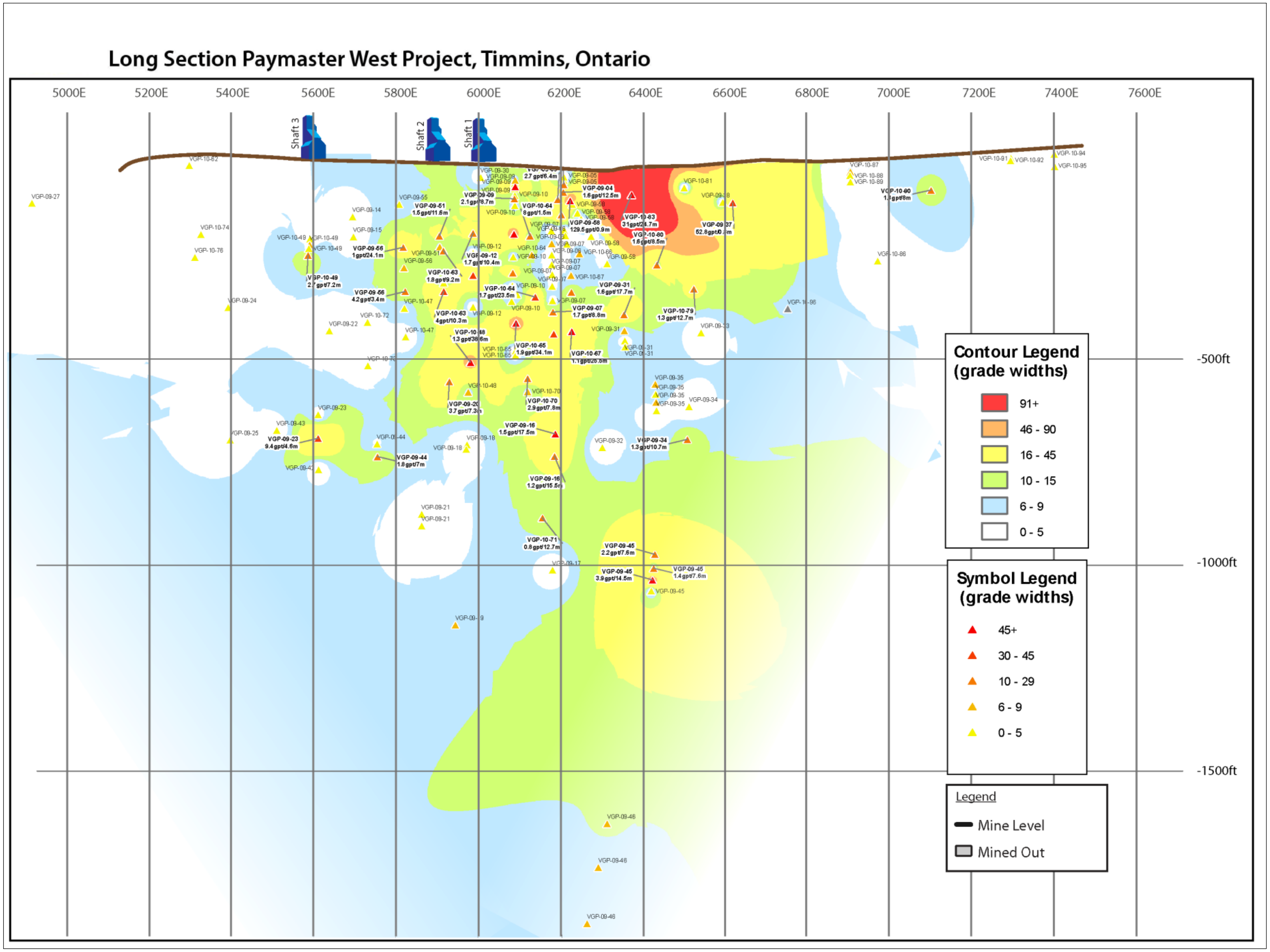
The nature of the program was two-fold:

- Expand and further delineate the near surface mineralization to advance the project toward a resource estimate
- Explore the zone to depth and down plunge.

The following drill summary (Table 10.4) indicates the significant assays from the drilling completed by Lexam as well as the collar detail. The drilling has been successful in outlining significant near surface material of a potential bulk mining nature (Figure 10.4). As well mineralized higher-grade chutes have been identified at depth that may indicate a potential higher-grade underground mining scenario.

The Main Zone Porphyry has been drill traced for a strike length of 2,500 ft (760 m) in an east-west direction, and for a depth of 1,900 ft (580 m) below surface. It dips at 45° to 70° to the north.

Figure 10.4 Paymaster Longitudinal Section



Source: *Lexam 2013*

The shallower dips are in the central and shallow areas with steeper dips to the east and at depth. Figure 10.4 indicates a plunge to the east at approximately 70° to 80°. This corresponds to the plunge as indicated in the mined-out workings.

The comparison of the Lexam VG results to the high-grade historic results is favourable, with similar width values for both sets of data. The historic results have a few zones of greater than 50 m, while the Lexam VG results have a maximum width of 38.6 m, however the two data sets have similar numbers of zones greater than 25 m. The earlier historic data (West Dome, circa 1920) shows an extremely large number of composites greater than 50 gpt x m. This data is probably skewed by a higher background of gold contamination in the sample preparation stage. The Dome data and the data at depth from the 1500 (Buffalo Ankerite) and 2500 (Paymaster) levels is comparable to the Lexam VG results. It would therefore not be prudent to place a great emphasis on the West Dome assay results and they should not be included in any resource calculations. However, the length of the composites and the lithological data is comparable to the Lexam VG data and could be utilized in the geological modeling.

Sectional information from the central area of the West Porphyry Zone indicates the geometry of the quartz-feldspar units. They are all sub-parallel and moderately north dipping. Widths can vary greatly both down dip and along strike. The main porphyry on these two sections is quite well mineralized with virtually every hole having fairly homogenous gold mineralization. Occasionally a high grade nugget is intersected, which skews the composite to the high side. For example hole VGP-10-83 on section 6,400E intersected 30.99 gpt Au over 24.7 m. The intersection is heavily weighted by a sample which assayed 366 gpt Au over 2.0 m. However, cutting the high grade sample to 30 gpt stills yields an impressive result of 4.03 gpt Au over 24.7 m. As can be seen on the sections the porphyry unit and the Au values are quite respect to the gold mineralization, being more erratically mineralized. However, the adjacent units appear to be geologically continuous and with fewer pierce points they represent valid exploration targets.

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
Year 1																
BA																
VGP-08-01	6305.8	5894.7	10980.6	902.2	275.0	-85	330									
VGP-08-02	6468.3	5859.7	10972.8	1,414.0	431.0	-77	125									
VGP-08-03	5540.7	8017.6	11038.4	1,404.2	428.0	-55	265	95.1	100.0	4.9	0.031	29.0	30.5	1.5	1.06	1.6
								259.2	284.0	24.8	0.048	79.0	86.6	7.6	1.63	12.3
								307.0	319.0	12.0	0.120	93.6	97.2	3.7	4.13	15.1
								276.1	277.7	1.6	UGO					
								1,076.0	1,099.0	23.0	0.060	328.0	335.0	7.0	2.06	14.4
								1,114.0	1,129.0	15.0	0.016	339.5	344.1	4.6	0.53	2.4
								1,169.0	1,174.0	5.0	0.035	356.3	357.8	1.5	1.20	1.8
VBA-06-39B	4386.5	8493.6	10896.2	626.6	191.0	-52	90	695.4	697.3	1.9	0.046	212.0	212.5	0.6	1.58	0.9
Paymaster																
VGP-09-04	6209.9	8278.3	11003.1	154.3	47.0	-65	180	51.0	69.7	18.7	0.043	15.5	21.2	5.7	1.46	8.3
								85.0	106.0	21.0	0.056	25.9	32.3	6.4	1.92	12.3
								121.0	126.0	5.0	0.106	36.9	38.4	1.5	3.63	5.5
VGP-09-05	6209.9	8278.3	11003.1	124.8	38.0	-45	180	44.0	62.0	18.0	0.074	13.4	18.9	5.5	2.52	13.8
								95.1	120.0	24.9	0.042	29.0	36.6	7.6	1.43	10.8
VGP-09-06	6175.2	8402.1	11009.3	470.8	143.5	-45	180	158.0	240.0	82.0	0.049	48.2	73.2	25.0	1.69	42.3
									includes	7.0	0.205		includes	2.1	7.02	15.0
								300.0	305.0	5.0	0.053	91.4	93.0	1.5	1.82	2.8
VGP-09-07	6174.7	8406.1	11009.6	498.7	152.0	-80	180	195.0	198.0	3.0	0.048	59.4	60.4	0.9	1.65	1.5
								238.0	243.0	5.0	0.084	72.5	74.1	1.5	2.88	4.4
								303.0	308.0	5.0	0.033	92.4	93.9	1.5	1.13	1.7
								386.0	411.0	25.0	0.054	117.7	125.3	7.6	1.85	14.1
								451.0	460.0	9.0	0.417	137.5	140.2	2.7	14.31	39.3
VGP-09-08	6091.6	8277.0	11010.6	77.6	23.7	-45	180	hole ends in opening prior to zone								
VGP-09-09	6091.9	8278.6	11010.5	225.0	68.6	-65	180	79.0	80.8	1.8	0.354	24.1	24.6	0.5	12.14	6.7
								121.0	144.4	23.4	0.071	36.9	44.0	7.1	2.44	17.4
VGP-09-10	6092.1	8280.6	11010.8	390.4	119.0	-85	180	85.0	91.0	6.0	1.215	25.9	27.7	1.8	41.67	76.2
								132.0	135.6	3.6	0.042	40.2	41.3	1.1	1.44	1.6
								168.0	172.0	4.0	0.116	51.2	52.4	1.2	3.98	4.9
								179.5	184.0	4.5	0.059	54.7	56.1	1.4	2.02	2.8
								194.0	199.0	5.0	0.067	59.1	60.7	1.5	2.30	3.5
								204.0	238.7	34.7	0.153	62.2	72.8	10.6	5.25	55.6
								260.0	264.4	4.4	0.090	79.2	80.6	1.3	3.09	4.1
								283.0	318.0	35.0	0.074	86.3	96.9	10.7	2.55	27.2
								343.0	348.0	5.0	0.052	104.5	106.1	1.5	1.78	2.7
								368.0	372.8	4.8	0.100	112.2	113.6	1.5	3.43	5.0
VGP-09-11	5989.4	8332.4	11023.8	176.8	53.9	-45	180	hole ends in opening prior to zone								
VGP-09-12	5989.4	8334.5	11023.8	469.2	143.0	-75	180	205.0	239.0	34.0	0.050	62.5	72.8	10.4	1.70	17.6
								314.0	347.0	33.0	0.099	95.7	105.8	10.1	3.39	34.0
								408.0	411.4	3.4	0.040	124.4	125.4	1.0	1.37	1.4
VGP-09-13	5889.9	8281.2	11029.6	158.9	48.4	-45	180	hole ends in opening prior to zone								
VGP-09-14	5700.0	8414.7	11029.0	498.7	152.0	-45	180	40.0	44.5	4.5	0.054	12.2	13.6	1.4	1.85	2.5
								151.0	156.0	5.0	0.071	46.0	47.5	1.5	2.43	3.7
VGP-09-15	5700.0	8416.0	11028.8	489.0	149.0	-60	180	157.8	176.1	18.3	0.131	48.1	53.7	5.6	4.49	25.0
VGP-09-16	6184.6	9161.6	11022.9	1,355.0	413.0	-50	172	578.8	582.1	3.3	0.177	176.4	177.4	1.0	6.06	6.1
								852.7	928.0	75.3	0.041	259.9	282.9	23.0	1.40	32.2
								966.3	971.5	5.2	0.066	294.5	296.1	1.6	2.26	3.6
VGP-09-17	6184.5	9163.0	11022.5	1,169.6	356.5	-68	174	352.6	358.5	5.9	0.069	107.5	109.3	1.8	2.37	4.3
VGP-09-18	5998.3	9108.0	11033.1	1,000.7	305.0	-50	180	130.0	135.0	5.0	0.040	39.6	41.1	1.5	1.39	2.1
								774.7	780.0	5.3	0.035	236.1	237.7	1.6	1.19	1.9
								933.0	938.0	5.0	0.030	284.4	285.9	1.5	1.03	1.6
VGP-09-19	5998.4	9110.5	11033.5	1,335.3	407.0	-69	180	376.0	382.6	6.6	0.338	114.6	116.6	2.0	11.60	23.3
								684.0	689.0	5.0	0.090	208.5	210.0	1.5	3.07	4.7
								1,257.8	1,264.5	6.7	0.104	383.4	385.4	2.0	3.56	7.3
VGP-09-20	5797.5	8864.4	11036.6	206.0	62.8	-50	165	136.2	139.2	3.0	40.542	41.5	42.4	0.9	1,390	1271.0
extended yr 2								508.5	513.4	4.9	0.048	155.0	156.5	1.5	1.66	2.5
VGP-09-21	5797.1	8866.1	11035.8	233.0	71.0	-70	165	520.0	532.4	12.4	0.035	158.5	162.3	3.8	1.20	4.5
extended yr 2								583.5	596.2	12.7	0.048	177.9	181.7	3.9	1.66	6.4
VGP-09-22	5599.3	8989.6	11054.5	1,084.3	330.5	-45	175					no significant assays				
VGP-09-23	5598.4	8992.7	11055.1	912.2	278.0	-70	175	510.0	514.9	4.9	0.047	155.4	156.9	1.5	1.63	2.4
								729.0	732.0	3.0	0.068	222.2	223.1	0.9	2.34	2.1
								785.0	800.1	15.1	0.274	239.3	243.9	4.6	9.40	43.3

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
VGP-09-24	5396.6	9014.3	11066.7	1,049.9	320.0	-45	175	455.3	469.0	13.7	0.043	138.8	143.0	4.2	1.46	6.1
VGP-09-25	5396.4	9017.5	11067.2	1,217.2	371.0	-65	175					no significant assays				
VGP-09-26	5204.2	8810.1	11066.7	1,591.2	485.0	-45	175	701.0	717.1	16.1	0.033	213.7	218.6	4.9	1.13	5.6
VGP-09-27	4908.1	8808.8	11064.9	2,408.1	734.0	-56	175	266.1	270.0	3.9	0.038	81.1	82.3	1.2	1.31	1.6
								752.5	768.2	15.7	0.061	229.4	234.1	4.8	2.11	10.1
								1,236.0	1,240.0	4.0	0.282	376.7	378.0	1.2	9.66	11.8
								1,992.0	1,998.0	6.0	0.055	607.2	609.0	1.8	1.90	3.5
VGP-09-28	6092.1	8071.9	11027.8	87.3	26.6	-45	0					hole ends in opening prior to zone				
VGP-09-29	6092.1	8069.6	11027.8	574.3	175.0	-75	0					hole ends in opening prior to zone				
VGP-09-30	5995.5	8071.7	11028.2	116.0	35.4	-45	0					hole ends in opening prior to zone				
VGP-09-31	6312.1	8866.8	11012.7	1,257.9	383.4	-45	175	481.2	500.5	19.3	0.071	146.7	152.6	5.9	2.42	14.2
								includes		7.5	0.120	includes		2.3	4.11	9.4
								545.1	690.0	144.9	0.029	166.1	210.3	44.2	1.00	44.3
									includes	7.9	0.156		includes	2.4	5.37	12.9
VGP-09-32	6311.7	8868.9	11012.2	1,532.2	467.0	-68	175	119.0	125.0	6.0	0.058	36.3	38.1	1.8	2.00	3.7
								650.0	670.6	20.6	0.052	198.1	204.4	6.3	1.79	11.3
								710.0	715.7	5.7	0.047	216.4	218.1	1.7	1.60	2.8
								1,219.0	1,223.6	4.6	0.167	371.6	373.0	1.4	5.72	8.0
VGP-09-33	6497.8	8880.6	11004.0	1,364.7	416.0	-45	180	253.0	256.0	3.0	0.165	77.1	78.0	0.9	5.66	5.2
								670.0	750.0	93.4	0.030	204.2	228.6	28.5	1.03	29.3
								1,187.0	1,197.0	10.0	0.048	361.8	364.8	3.0	1.64	5.0
								1,211.0	1,225.0	14.0	0.074	369.1	373.4	4.3	2.54	10.8
VGP-09-34	6497.3	8883.4	11004.1	1,660.4	506.1	-70	180	283.0	291.7	8.7	0.038	86.3	88.9	2.7	1.31	3.5
								611.0	616.0	5.0	0.491	186.2	187.8	1.5	16.85	25.7
								744.0	764.0	20.0	0.049	226.8	232.9	6.1	1.70	10.3
								1,330.0	1,334.9	4.9	0.123	405.4	406.9	1.5	4.23	6.3
								1,433.6	1,535.3	101.7	0.047	437.0	468.0	31.0	1.61	49.9
									includes	10.0	0.119		includes	3.0	4.09	12.5
VGP-09-35	6397.6	9109.7	11018.6	1,108.9	338.0	-45	180	790.5	798.0	7.5	0.099	240.9	243.2	2.3	3.39	7.8
								828.0	858.0	30.0	0.032	252.4	261.5	9.1	1.09	10.0
VGP-09-36	6397.2	9112.6	11018.9	1,542.0	470.0	-70	180	147.0	152.0	5.0	0.068	44.8	46.3	1.5	2.32	3.5
								450.0	489.0	39.0	0.145	137.2	149.0	11.9	4.98	59.2
									includes	5.0	0.933		includes	1.5	32.00	48.8
								1,289.0	1,301.0	12.0	0.052	392.9	396.5	3.7	1.79	6.5
								1,325.0	1,331.0	6.0	0.140	403.9	405.7	1.8	4.79	8.8
VGP-09-37	6585.0	8760.6	11005.1	1,374.7	419.0	-45	165	171.1	173.8	2.7	1.540	52.2	53.0	0.8	52.80	43.5
								246.0	252.0	6.0	0.100	75.0	76.8	1.8	3.42	6.3
								930.0	934.5	4.5	0.438	283.5	284.8	1.4	15.00	20.6
VGP-09-38	6583.9	8762.3	11004.9	1,601.0	488.0	-68	165	730.0	736.0	6.0	0.055	222.5	224.3	1.8	1.88	3.4
								864.0	869.0	5.0	0.052	263.3	264.9	1.5	1.80	2.7
								1,166.0	1,202.0	36.0	0.035	355.4	366.4	11.0	1.21	13.3
								1,440.3	1,458.0	17.7	0.036	439.0	444.4	5.4	1.22	6.6
VGP-09-39	5813.7	8817.1	11026.4	213.3	65.0	-45	180							nsv		
VGP-09-40	5813.7	8817.1	11026.4	213.3	65.0	-50	165							nsv		
VGP-09-41	5813.7	8817.1	11026.4	213.3	65.0	-50	195							nsv		
VGP-09-42	5603.3	9406.3	11049.6	1,246.7	380.0	-56	175									
VGP-09-43	5504.4	9313.6	11062.1	1,108.9	338.0	-52	175	907.4	911.0	3.6	0.033	276.6	277.7	1.1	1.15	1.3
VGP-09-44	5717.5	9325.0	11049.7	1,148.3	350.0	-50	170	932.9	938.6	5.7	0.058	284.3	286.1	1.7	1.99	3.5
								967.8	971.6	7.5	0.115	295.0	296.1	2.3	3.95	9.0
VGP-09-45	6406.6	9587.8	11008.4	1,453.4	443.0	-45	180	1,305.0	1,427.5	122.5	0.067	397.8	435.1	37.3	2.29	85.5
									includes	25.0	0.163		includes	7.6	5.57	42.5
VGP-09-46	6406.6	9590.1	11008.4	1,108.9	338.0	-60	175					nsv				
VGP-09-51	5903.4	8117.3	11017.7	449.5	137.0	-45	0	293.0	348.8	55.8	0.040	89.3	106.3	17.0	1.36	23.1
VGP-09-52	5902.9	8114.9	11018.8	139.1	42.4	-60	0	hole ends in opening prior to zone								
VGP-09-53	5902.7	8113.5	11018.9	459.3	140.0	-75	0	230.5	329.6	99.1	0.037	70.3	100.5	30.2	1.28	38.6
VGP-09-54	5810.6	8129.3	11018.3	557.7	170.0	-45	0						nsv			
VGP-09-55	5810.4	8127.4	11018.2	511.8	156.0	-75	0	137.9	142.6	4.7	0.083	42.0	43.5	1.4	2.85	4.1
VGP-09-56	5810.3	8125.9	11019.2	407.1	124.1	-60	0	236.0	241.0	5.0	0.063	71.9	73.5	1.5	2.15	3.3
								256.0	261.0	5.0	0.064	78.0	79.6	1.5	2.21	3.4
								306.0	315.9	9.9	0.046	93.3	96.3	3.0	1.56	4.7
								326.3	337.9	11.6	0.081	99.4	103.0	3.5	2.78	9.8
								390.6	393.6	3.0	0.127	119.1	120.0	0.9	4.37	4.0
								403.7	406.9	3.2	0.107	123.0	124.0	1.0	3.67	3.6
VGP-09-57	5805.1	7978.2	11025.9	636.5	194.0	-45	0	527.3	552.3	25.0	0.074	160.7	168.3	7.6	2.55	19.4
								602.3	606.1	3.8	0.112	183.6	184.7	1.2	3.83	4.4
VGP-09-58	6143.1	8070.7	11026.6	479.0	146.0	-45	35	191.8	194.8	3.0	3.777	58.5	59.4	0.9	129.50	118.4

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
								232.8	236.8	4.0	0.065	71.0	72.2	1.2	2.22	2.7
VGP-09-59	6141.2	8071.1	11023.0	449.5	137.0	-60	35	371.7	377.7	6.0	0.029	113.3	115.1	1.8	1.01	1.8
Year 2																
VGP-09-20ext	5797.5	8864.4	11036.6	314.9	96.0	-50	165	GP-09-20								
final	990.8	302.0														
original	675.9	206.0														
VGP-09-21ext	5797.1	8866.1	11035.8	512.1	156.1	-70	165	GP-09-21								
final	1276.5	389.1														
original	764.4	233.0														
VGP-09-46ext	6406.6	9590.1	11008.4	1,151.8	351.1	-60	175	GP-09-46								
final	2260.7	689.1														
original	1108.9	338.0														
VGP-10-47	5807.6	8742.2	11028.2	754.6	230.0	-50	178	398.0	402.0	4.0	0.251	121.3	122.5	1.2	8.62	10.5
								518.0	529.4	11.4	0.046	157.9	161.4	3.5	1.57	5.4
								557.7	568.3	10.6	0.026	170.0	173.2	3.2	0.90	2.9
VGP-10-48	6001.3	8917.6	11024.8	902.2	275.0	-50	180	119.9	124.0	4.1	0.093	36.5	37.8	1.2	3.17	4.0
								223.5	288.0	64.5	0.023	68.1	87.8	19.7	0.78	15.3
								440.0	450.0	10.0	0.018	134.1	137.2	3.0	0.62	1.9
								631.4	758.0	126.6	0.039	192.5	231.0	38.6	1.34	51.9
								763.0	769.0	6.0	0.025	232.6	234.4	1.8	0.86	1.6
								784.0	809.5	25.5	0.029	239.0	246.7	7.8	0.99	7.7
VGP-10-49	5620.6	8618.9	11036.7	597.1	182.0	-45	185	168.7	182.0	13.3	0.025	51.4	55.5	4.1	0.84	3.4
								365.5	371.5	6.0	0.098	111.4	113.2	1.8	3.36	6.1
								396.0	419.7	23.7	0.080	120.7	127.9	7.2	2.73	19.7
VGP-10-50	5620.6	8620.1	11035.5	597.1	182.0	-61	186	275.8	279.4	3.6	0.056	84.1	85.2	1.1	1.91	2.1
VGP-10-60	5495.3	8536.2	11044.4	695.5	212.0	-45	175	403.4	435.4	32.0	0.033	123.0	132.7	9.8	1.11	10.9
								441.0	465.8	24.8	0.085	134.4	142.0	7.6	2.91	22.0
VGP-10-61	5495.5	8539.4	11045.1	616.8	188.0	-67	175	nsv								
VGP-10-62	5296.4	8301.7	11040.9	508.5	155.0	-45	180	143.0	148.0	5.0	0.061	43.6	45.1	1.5	2.09	3.2
VGP-10-63	5899.1	8599.9	11028.2	734.9	224.0	-45	175	251.0	254.1	3.1	0.024	76.5	77.4	0.9	0.83	0.8
								363.0	393.2	30.2	0.051	110.6	119.8	9.2	1.76	16.2
								499.0	532.8	33.8	0.115	152.1	162.4	10.3	3.96	40.8
VGP-10-64	6097.8	8572.6	11017.8	754.6	230.0	-52	170	263.7	268.7	5.0	0.234	80.4	81.9	1.5	8.01	12.2
								307.7	339.0	31.3	0.030	93.8	103.3	9.5	1.02	9.7
								414.0	491.0	77.0	0.048	126.2	149.7	23.5	1.65	38.8
VGP-10-65	6098.5	8573.8	11018.6	793.9	242.0	-67	178	309.0	314.0	5.0	0.048	94.2	95.7	1.5	1.64	2.5
								376.0	396.0	20.0	0.020	114.6	120.7	6.1	0.67	4.1
								403.1	515.0	111.9	0.055	122.9	157.0	34.1	1.89	64.6
VGP-10-66	6216.4	8567.8	11011.5	606.9	185.0	-42	168	281.0	285.4	4.4	0.020	85.6	87.0	1.3	0.69	0.9
								350.5	380.0	29.5	0.029	106.8	115.8	9.0	0.98	8.8
VGP-10-67	6216.2	8570.9	11012.5	656.2	200.0	-67	171	324.1	344.0	19.9	0.028	98.8	104.9	6.1	0.95	5.8
								359.0	388.0	29.0	0.044	109.4	118.3	8.8	1.50	13.3
								439.0	522.6	83.6	0.033	133.8	159.3	25.5	1.15	29.2
VGP-10-68	5296.4	8303.0	11040.5	577.4	176.0	-60	175	376.2	383.2	7.0	0.072	114.7	116.8	2.1	2.47	5.3
								397.2	400.9	3.7	0.058	121.1	122.2	1.1	1.99	2.2
VGP-10-69	5402.7	8310.0	11039.4	921.9	281.0	-45	175	90.0	97.6	7.6	0.107	27.4	29.7	2.3	3.65	8.5
								387.6	398.9	11.3	0.050	118.1	121.6	3.4	1.72	5.9
								828.0	833.0	5.0	0.039	252.4	253.9	1.5	1.33	2.0
VGP-10-70	6097.3	9042.6	11024.2	961.3	293.0	-45	175	282.0	287.0	5.0	0.047	86.0	87.5	1.5	1.62	2.5
								402.0	411.9	9.9	0.026	122.5	125.5	3.0	0.89	2.7
								770.9	796.6	25.7	0.084	235.0	242.8	7.8	2.88	22.6
								816.0	836.2	20.2	0.035	248.7	254.9	6.2	1.19	7.3
VGP-10-71	6100.2	9222.0	11031.2	1,227.1	374.0	-60	173	685.0	696.2	11.2	0.017	208.8	212.2	3.4	0.59	2.0
								1,034.0	1,075.8	41.8	0.024	315.2	327.9	12.7	0.82	10.5
VGP-10-72	5694.0	8942.5	11050.5	1,207.4	368.0	-45	175	1,128.6	1,132.8	4.2	0.056	344.0	345.3	1.3	1.93	2.5
VGP-10-73	5694.5	8940.0	11049.3	1,177.8	359.0	-62	175	326.5	333.9	7.4	0.024	99.5	101.8	2.3	0.82	1.8
VGP-10-74	5297.8	8761.8	11058.9	764.4	233.0	-45	173	65.3	68.1	2.8	53.958	19.9	20.8	0.9	1,850	1,579
VGP-10-75	5003.6	12213.1	11050.0	5,219.8	1,591.0	-56	158	1,678.0	1,682.0	4.0	0.016	511.5	512.7	1.2	0.54	0.7
								1,778.5	1,783.5	5.0	0.057	542.1	543.6	1.5	1.94	3.0
								1,796.0	1,820.0	24.0	0.024	547.4	554.7	7.3	0.81	5.9
								1,841.8	1,849.0	7.2	0.050	561.4	563.6	2.2	1.73	3.8
								2,705.5	2,710.5	5.0	0.111	824.6	826.2	1.5	3.80	5.8
								3,028.0	3,052.9	24.9	0.050	922.9	930.5	7.6	1.71	13.0
VGP-10-76	5297.4	8757.7	11058.0	853.0	260.0	-62	175	148.5	160.0	11.5	0.023	45.3	48.8	3.5	0.80	2.8

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
								342.0	354.7	12.7	0.021	104.2	108.1	3.9	0.73	2.8
VGP-10-77	5298.0	8757.0	11058.2	223.1	68.0	-45	143	131.0	137.0	6.0	0.018	39.9	41.8	1.8	0.63	1.1
								155.0	167.0	12.0	0.032	47.2	50.9	3.7	1.11	4.1
VGP-10-78	5299.1	8761.0	11058.4	164.1	50.0	-45	210	116.0	128.0	12.0	0.019	35.4	39.0	3.7	0.66	2.4
VGP-10-79	6495.4	8654.8	11003.4	1,271.1	387.4	-47	175	380.0	387.0	7.0	0.019	115.8	118.0	2.1	0.65	1.4
								402.0	409.7	7.7	0.065	122.5	124.9	2.3	2.22	5.2
								422.0	463.7	41.7	0.038	128.6	141.3	12.7	1.30	16.6
								510.3	556.9	46.6	0.070	155.5	169.7	14.2	2.38	33.8
								914.0	959.3	45.3	0.023	278.6	292.4	13.8	0.77	10.7
								973.0	1,009.5	36.5	0.031	296.6	307.7	11.1	1.07	11.9
VGP-10-80	6413.2	8568.5	11002.8	961.3	293.0	-41	175	335.2	352.0	16.8	0.117	102.2	107.3	5.1	4.02	20.6
								391.0	419.0	28.0	0.046	119.2	127.7	8.5	1.58	13.5
VGP-10-81	6499.7	8218.4	11009.7	557.8	170.0	-45	177	149.9	161.0	11.1	0.049	45.7	49.1	3.4	1.69	5.7
								454.4	461.7	7.3	0.026	138.5	140.7	2.2	0.90	2.0
VGP-10-82	5607.3	12119.1	11058.6	278.9	85.0	-70	156	nsv								
VGP-10-83	6374.0	8239.1	11012.1	754.6	230.0	-45	178	114.0	195.0	81.0	0.904	34.7	59.4	24.7	30.99	765.1
									incl	6.5	10.675			2.0	366.00	
								363.7	370.0	6.3	0.021	110.9	112.8	1.9	0.73	1.4
VGP-10-84	5604.1	12121.7	11059.4	3,345.5	1,019.7	-70	156	1,088.7	1,092.7	4.0	0.035	331.8	333.1	1.2	1.20	1.5
								2,260.0	2,263.0	3.0	0.085	688.8	689.8	0.9	2.91	2.7
								2,986.3	2,996.3	10.0	0.016	910.2	913.3	3.0	0.55	1.7
								2,999.3	3,006.1	6.8	0.043	914.2	916.3	2.1	1.48	3.1
								3,041.0	3,046.0	5.0	0.033	926.9	928.4	1.5	1.13	1.7
VGP-10-85	7000.9	8142.2	11009.0	1,246.7	380.0	-44	174	60.0	72.9	12.9	0.074	18.3	22.2	3.9	2.54	10.0
								88.7	129.6	40.9	0.053	27.0	39.5	12.5	1.80	22.4
								893.0	898.0	5.0	0.069	272.2	273.7	1.5	2.38	3.6
Year 3																
VGP-10-64e	6097.8	8572.6	11017.8	1,433.7	437.0	-52	170	1,220.2	1,223.8	3.6	0.021	371.9	373.0	1.1	0.72	0.8
final	2188.3	667.0														
original	754.6	230.0														
VGP-10-65e	6098.5	8573.8	11018.6	764.5	233.0	-67	180									
final	1558.4	475.0														
original	793.9	242.0														
VGP-10-71e	6100.2	9222.0	11027.5	902.2	275.0	-60	173	1,503.5	1,523.0	19.5	0.044	458.3	464.2	5.9	1.51	9.0
final	2129.3	649.0														
original	1227.1	374.0														
VGP-10-86	7001.6	8135.5	11012.8	961.3	293.0	-45	353	88.3	116.5	28.2	0.045	26.9	35.5	8.6	1.54	13.2
								362.2	370.5	8.3	0.020	110.4	112.9	2.5	0.67	1.7
								510.0	514.0	4.0	0.014	155.4	156.7	1.2	0.49	0.6
								817.0	823.0	6.0	0.173	249.0	250.9	1.8	5.92	10.8
VGP-10-87	6904.7	8351.3	11013.1	1,177.8	359.0	-48	176	8.0	13.2	5.2	0.069	2.4	4.0	1.6	2.35	3.7
								64.5	70.0	5.5	0.095	19.7	21.3	1.7	3.24	5.4
								294.1	320.3	26.2	0.548	89.6	97.6	8.0	18.80	150.1
								incl		13.9	1.018			4.2	34.90	147.9
								incl		5.2	2.605			1.6	89.30	141.5
								784.1	794.4	10.3	0.039	239.0	242.1	3.1	1.34	4.2
								1,085.2	1,092.0	6.8	0.059	330.8	332.8	2.1	2.02	4.2
VGP-10-88	6904.5	8351.7	11012.1	1,387.1	422.8	-65	175	3.6	8.7	5.1	0.041	1.1	2.7	1.6	1.40	2.2
								71.9	77.2	5.3	0.052	21.9	23.5	1.6	1.78	2.9
								369.0	386.1	17.1	0.055	112.5	117.7	5.2	1.89	9.8
								519.2	523.9	4.7	0.045	158.3	159.7	1.4	1.55	2.2
								1,237.3	1,246.7	9.4	0.064	377.1	380.0	2.9	2.18	6.3
								1,298.1	1,307.5	9.4	0.061	395.7	398.5	2.9	2.10	6.0
VGP-10-88A	6904.9	8348.4	11009.2	36.1	11.0	-55	175	6.4	25.9	19.5	0.019	2.0	7.9	5.9	0.65	3.9
VGP-10-89	6904.4	8353.3	11013.1	906.2	276.2	-81	175	23.5	27.9	4.4	0.061	7.2	8.5	1.3	2.09	2.8
								70.2	86.4	16.2	0.023	21.4	26.3	4.9	0.80	4.0
								97.0	105.0	8.0	0.023	29.6	32.0	2.4	0.79	1.9
								528.0	537.2	9.2	0.023	160.9	163.7	2.8	0.79	2.2
								649.0	654.0	5.0	0.075	197.8	199.3	1.5	2.57	3.9
								725.1	740.1	15.0	0.018	221.0	225.6	4.6	0.63	2.9
								758.7	762.5	3.8	0.059	231.3	232.4	1.2	2.01	2.3
								818.0	836.5	18.5	0.040	249.3	255.0	5.6	1.39	7.8
VGP-10-90	7090.4	8383.1	11007.7	469.2	143.0	-55	175	96.6	123.0	26.4	0.038	29.4	37.5	8.0	1.30	10.4

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
								348.0	353.0	5.0	0.050	106.1	107.6	1.5	1.70	2.6
VGP-10-91	7297.0	8303.0	11012.9	1,089.3	332.0	-45	181	13.6	24.0	10.4	0.021	4.1	7.3	3.2	0.71	2.2
								205.0	218.9	13.9	0.043	62.5	66.7	4.2	1.48	6.3
								412.0	418.4	6.4	0.038	125.6	127.5	2.0	1.32	2.6
								795.8	805.2	9.4	0.026	242.6	245.4	2.9	0.91	2.6
								812.3	817.9	5.6	0.038	247.6	249.3	1.7	1.31	2.2
VGP-10-92	7298.3	8289.0	11011.4	1,168.0	356.0	-45	358	24.7	32.0	7.3	0.026	7.5	9.8	2.2	0.88	2.0
								463.0	472.6	9.6	0.023	141.1	144.0	2.9	0.79	2.3
VGP-10-93	7403.8	8283.5	11010.3	301.8	92.0	-45	358	20.7	26.6	5.9	0.109	6.3	8.1	1.8	3.74	6.7
								74.8	86.0	11.2	0.028	22.8	26.2	3.4	0.95	3.2
								220.5	224.1	3.6	0.057	67.2	68.3	1.1	1.96	2.2
								253.0	265.1	12.1	0.050	77.1	80.8	3.7	1.71	6.3
VGP-10-94	7406.0	8292.9	11009.8	1,187.7	362.0	-45	178	9.9	18.0	8.1	0.019	3.0	5.5	2.5	0.84	2.1
								388.0	427.0	39.0	0.029	118.3	130.1	11.9	1.01	12.0
								425.1	427.0	1.9	0.272	129.6	130.1	0.6	9.31	5.4
VGP-10-94A	7406.0	8292.9	11009.8	65.6	20.0	-45	182			(no significant assays)						
VGP-10-95	7406.4	8295.4	11009.6	803.8	245.0	-70	175	38.8	43.0	4.2	0.119	11.8	13.1	1.3	4.09	5.2
								199.5	211.0	11.5	0.029	60.8	64.3	3.5	0.98	3.4
								520.8	593.7	72.9	0.046	158.7	181.0	22.2	1.57	34.9
									incl	13.9	0.079		incl	4.2	2.72	11.5
VGP-10-96	6680.0	9093.5	11023.2	1,916.0	584.0	-47	168	199.5	211.0	11.5	0.029	60.8	64.3	3.5	0.98	3.4
								429.0	439.0	10.0	0.033	130.8	133.8	3.0	1.12	3.4
								446.0	456.0	10.0	0.029	135.9	139.0	3.0	0.99	3.0
								1,013.1	1,039.9	26.8	0.090	308.8	317.0	8.2	3.07	25.1
								1,347.8	1,363.5	15.7	0.017	410.8	415.6	4.8	0.59	2.8
								1,388.8	1,413.0	24.2	0.057	423.3	430.7	7.4	1.96	14.4
								1,746.1	1,749.2	3.1	0.430	532.2	533.2	0.9	14.75	13.9
VGP-10-97	6314.0	9180.1	11021.4	2,378.6	725.0	-70	175	284.6	300.0	15.4	0.022	86.7	91.4	4.7	0.75	3.5
								396.0	404.5	8.5	0.224	120.7	123.3	2.6	7.67	19.9
								424.0	446.0	22.0	0.016	129.2	135.9	6.7	0.53	3.6
								544.0	558.2	14.2	0.016	165.8	170.1	4.3	0.56	2.4
								578.5	592.5	14.0	0.019	176.3	180.6	4.3	0.65	2.8
								606.5	612.5	6.0	0.031	184.9	186.7	1.8	1.07	2.0
								1,729.0	1,733.0	4.0	0.062	527.0	528.2	1.2	2.13	2.6
VGP-10-98	6769.3	9390.2	11008.0	2,318.2	706.6	-56	172	633.9	639.8	5.9	0.055	193.2	195.0	1.8	1.89	3.4
								1,148.2	1,151.2	3.0	0.101	350.0	350.9	0.9	3.46	3.2
								1,236.0	1,245.8	9.8	0.021	376.7	379.7	3.0	0.72	2.2
								1,425.0	1,451.6	26.6	0.036	434.3	442.4	8.1	1.23	10.0
								2,079.6	2,083.9	4.3	0.045	633.9	635.2	1.3	1.54	2.0
VGP-10-99a	6606.6	9391.4	11023.9	39.4	12.0	-59	176									
VGP-10-99	6602.8	9392.7	11023.9	2,470.5	753.0	-69	170	1,733.5	1,736.3	2.8	0.130	528.4	529.2	0.9	4.46	3.8
								2,044.8	2,076.4	31.6	0.033	623.3	632.9	9.6	1.13	10.9
								2,404.0	2,413.9	9.9	0.021	732.7	735.8	3.0	0.72	2.2
VGP-10-100	6506.1	9599.4	11016.8	2,198.2	670.0	-65	168	43.0	48.0	5.0	0.041	13.1	14.6	1.5	1.41	2.1
								164.5	169.0	4.5	0.044	50.1	51.5	1.4	1.51	2.1
								2,095.3	2,098.2	2.9	0.193	638.6	639.5	0.9	6.61	5.8
VGP-10-101	6617.4	9686.1	11012.7	1,427.2	435.0	-77	172	964.6	995.0	30.4	0.016	294.0	303.3	9.3	0.53	4.9
VGP-10-102a	7029.9	9706.4	11006.6	68.9	21.0	-48	180	24.4	27.9	3.5	0.064	7.4	8.5	1.1	2.21	2.4
VGP-10-102	7034.1	9698.5	11006.6	2,380.2	725.5	-48	175	819.4	853.7	34.3	0.016	249.8	260.2	10.5	0.54	5.6
VGP-10-103A	7204.1	9731.9	11031.7	98.4	30.0	-50	179									
VGP-11-103	7193.0	9743.8	11013.3	2,040.9	622.1	-50	172	238.6	248.1	9.5	0.020	72.7	75.6	2.9	0.68	2.0
								839.1	853.0	13.9	0.022	255.8	260.0	4.2	0.74	3.1
								885.4	898.9	13.5	0.053	269.9	274.0	4.1	1.82	7.5
VGP-11-104	7196.4	9353.7	11006.6	1,663.2	506.9	-49	175	563.0	568.0	5.0	0.029	171.6	173.1	1.5	1.00	1.5
								808.0	813.0	5.0	0.036	246.3	247.8	1.5	1.23	1.9
								1,196.7	1,201.7	5.0	0.074	364.8	366.3	1.5	2.54	3.9
VGP-11-105	7387.5	9410.9	11010.1	1,880.8	573.3	-50	175	198.0	203.0	5.0	0.272	60.4	61.9	1.5	9.33	14.2
								523.0	533.0	10.0	0.178	159.4	162.5	3.0	6.09	18.6
VGP-11-107	5304.3	7976.6	11041.8	390.4	119.0	-50	273	148.3	151.3	3.0	0.107	45.2	46.1	0.9	3.68	3.4
								205.9	211.1	5.2	0.036	62.8	64.3	1.6	1.25	2.0
								326.4	358.9	32.5	0.203	99.5	109.4	9.9	6.96	68.9

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
								incl		8.3	0.588			2.5	20.16	51.0
VGP-11-113	4838.9	8598.4	11051.2	1,276.2	389.0	-55	180	81.2	89.1	7.9	0.029	24.7	27.2	2.4	0.98	2.4
								120.1	124.1	4.0	0.152	36.6	37.8	1.2	5.22	6.4
								275.3	284.6	9.3	0.023	83.9	86.7	2.8	0.78	2.2
								644.8	658.0	13.2	0.021	196.5	200.6	4.0	0.72	2.9
								661.0	670.5	9.5	0.065	201.5	204.4	2.9	2.23	6.5
								687.8	693.6	5.8	0.181	209.6	211.4	1.8	6.20	11.0
								770.0	774.1	4.1	0.043	234.7	235.9	1.2	1.47	1.8
								802.0	806.0	4.0	0.067	244.4	245.7	1.2	2.29	2.8
								842.0	854.0	12.0	0.022	256.6	260.3	3.7	0.75	2.7
BA North Zone - Partial holes on Pay																
VGP-11-106	5303.1	7975.7	11041.6	2,112.9	644.0	-55	262	298.8	302.3	3.4	0.057	91.1	92.1	1.1	1.95	2.1
collared on Pay				Pay	Lex			776.9	785.6	8.7	0.022	236.8	239.5	2.7	0.77	2.0
				914.4	1,198.5			837.0	841.4	4.4	0.047	255.1	256.5	1.3	1.60	2.1
								1,331.2	1,334.2	3.0	1.397	405.7	406.7	0.9	47.9	43.8
								1,542.0	1,589.6	47.6	0.045	470.0	484.5	14.5	1.53	22.2
								incl			0.142			2.1	4.88	10.4
								1,640.6	1,652.6	12.0	0.046	500.1	503.7	3.7	1.59	5.8
								1,716.0	1,813.2	97.2	0.109	523.0	552.7	29.6	3.74	110.8
								incl			0.696			2.4	23.85	58.2
								1,829.2	1,836.0	6.8	0.030	557.5	559.6	2.1	1.02	2.1
VGP-11-108	5441.3	8200.1	11038.7	1,860.2	567.0	-52	278	31.0	39.0	8.0	0.039	9.4	11.9	2.4	1.32	3.2
collared on Pay				Pay	Lex			328.5	334.8	6.3	0.039	100.1	102.0	1.9	1.34	2.6
				1,214.0	646.2			528.7	536.5	7.8	0.086	161.1	163.5	2.4	2.93	7.0
								1,327.7	1,332.5	4.8	0.039	404.7	406.1	1.5	1.34	2.0
								1,748.0	1,773.8	25.8	0.048	532.8	540.7	7.9	1.64	12.9
									incl	4.5	0.142		incl	1.4	4.88	6.7
VGP-11-109	5235.2	7815.8	11031.8	1,102.3	336.0	-45	270	95.2	136.0	40.8	0.151	29.0	41.5	12.4	5.18	64.4
collared on Pay				Pay	Lex				incl	5.0	0.496		incl	1.5	17.00	25.9
				631.1	471.2			270.4	275.2	4.8	0.040	82.4	83.9	1.5	1.36	2.0
								307.5	448.8	141.3	0.039	93.7	136.8	43.1	1.33	57.1
								394.0	408.6	14.6	UGO	120.1	124.5	4.5	UGO	
								492.0	498.0	6.0	0.043	150.0	151.8	1.8	1.48	2.7
								557.0	566.0	9.0	0.050	169.8	172.5	2.7	1.72	4.7
								668.2	685.0	16.8	0.037	203.7	208.8	5.1	1.26	6.4
								721.5	729.0	7.5	0.110	219.9	222.2	2.3	3.78	8.6
								784.0	859.0	75.0	0.111	239.0	261.8	22.9	3.80	86.9
									incl	15.0	0.311		incl	4.6	10.66	48.7
								808.9	831.7	22.8	UGO	246.6	253.5	6.9	UGO	
								883.0	989.0	106.0	0.049	269.1	301.4	32.3	1.67	54.1
									incl	24.0	0.090		incl	7.3	3.08	22.6
VGP-11-110	5193.5	7587.1	10098.0	1,128.6	344.0	-45	267	261.0	270.4	9.4	0.029	79.6	82.4	2.9	0.98	2.8
collared on Pay				Pay	Lex			500.8	512.7	11.9	0.038	152.6	156.3	3.6	1.31	4.8
				497.1	631.5			587.3	592.0	4.7	UGO	179.0	180.4	1.4	UGO	
								592.0	602.0	10.0	0.092	180.4	183.5	3.0	3.15	9.6
								625.0	638.7	13.7	0.033	190.5	194.7	4.2	1.12	4.7
								638.7	646.3	7.6	UGO	194.7	197.0	2.3	UGO	
								757.0	761.0	4.0	0.098	230.7	232.0	1.2	3.35	4.1
								834.1	868.0	33.9	0.066	254.2	264.6	10.3	2.25	23.2
									incl	4.7	0.264		incl	1.4	9.05	13.0
VGP-11-112	5456.1	8004.6	11045.5	1,305.8	398.0	-45	271	302.0	354.5	52.5	0.046	92.0	108.1	16.0	1.59	25.4
collared on Pay				Pay	Lex				incl	7.3	0.121		incl	2.2	4.15	9.2
end in ugo				947.4	358.4			462.2	466.0	3.8	0.055	140.9	142.0	1.2	1.89	2.2
								888.0	898.0	10.0	0.075	270.7	273.7	3.0	2.58	7.9
Fuller																
VGF-11-115	4422.0	12385.2	11047.9	1,893.1	577.0	-50	160	1,145.7	1,150.1	4.4	0.039	349.2	350.6	1.3	1.33	1.8
				Pay	Lex											
				1,535.4	357.7											
VGF-11-116	4615.3	12397.8	11050.4	3,756.6	1,145.0	-65	160	1,367.5	1,370.8	3.3	0.049	416.8	417.8	1.0	1.69	1.7

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
				Pay	Lex			1,540.9	1,544.9	4.0	0.109	469.7	470.9	1.2	3.72	4.5
				2,235.5	1,521.1			2,334.8	2,346.7	11.9	0.174	711.6	715.3	3.6	5.95	21.6
								incl		3.1	0.589	incl		0.9	20.20	
								2,397.1	2,401.4	4.3	0.038	730.6	731.9	1.3	1.32	1.7
								2,637.4	2,648.0	10.6	0.054	803.9	807.1	3.2	1.84	6.0
Year 4																
Paymaster N																
VGP-11-114	4839.2	8594.6	11051.7	833.3	254.0	-45	135	463.0	471.0	8.0	0.027	141.1	143.6	2.4	0.94	2.3
VGP-11-116	5249.4	7711.7	11019.1	108.2	33.0	-50	315		nsv							
VGP-11-117	5541.3	7822.1	11035.2	908.1	276.8	-45	268	49.0	90.0	41.0	0.080	14.9	27.4	12.5	2.73	34.2
									incl	9.0			incl	2.7	5.69	15.6
VGP-11-118	4968.2	7820.5	10998.1	787.4	240.0	-45	88		nsv							
test NS 109																
VGP-11-119	4966.5	7819.6	10998.5	275.6	84.0	-50	61	120.0	124.0	4.0	0.075	36.6	37.8	1.2	2.57	3.1
test NS 109								131.0	140.0	9.0	0.080	39.9	42.7	2.7	2.74	7.5
								206.0	232.1	26.1	0.088	62.8	70.7	8.0	3.01	24.0
									incl	7.2	0.192		incl	2.2	6.58	14.4
VGP-11-120	4972.1	7813.6	10998.8	344.5	105.0	-65	61	131.0	135.0	4.0	0.081	39.9	41.1	1.2	2.79	3.4
test NS 109								157.0	162.0	5.0	0.023	47.9	49.4	1.5	0.78	1.2
1 ugo								171.0	176.8	5.8	UGO					
								198.5	207.7	9.2	0.161	60.5	63.3	2.8	5.51	15.5
VGP-11-121	4967.4	7810.0	10998.7	816.9	249.0	-45	115	85.5	95.0	9.5	0.041	26.1	29.0	2.9	1.39	4.0
test NS 109																
VGP-11-125	6032.5	7781.9	11012.9	1,204.1	367.0	-55	270	844.0	848.0	4.0	0.067	257.3	258.5	1.2	2.31	2.8
4 shaft DD																
VGP-12-127	9013.2	7206.8	10966.3	502.0	153.0	-45	150	43.7	49.0	5.3	0.013	13.3	14.9	1.6	0.44	0.7
								124.4	133.8	9.4	0.019	37.9	40.8	2.9	0.65	1.9
VGP-12-128	8898.4	7522.2	10962.9	895.7	273.0	-45	150	263.5	283.0	19.5	0.015	80.3	86.3	5.9	0.51	3.1
								291.0	347.0	56.0	0.023	88.7	105.8	17.1	0.78	13.3
								351.0	364.0	13.0	0.013	107.0	110.9	4.0	0.45	1.8
								509.2	515.0	5.8	0.025	155.2	157.0	1.8	0.84	1.5
								624.5	630.4	5.9	0.026	190.3	192.1	1.8	0.89	1.6
VGP-12-129	8646.8	7598.1	10972.5	230.9	70.4	-50	148									
VGP-12-129B	8646.8	7598.1	10972.5	147.6	45.0	-52	148	e. No core								
VGP-12-130	8386.1	7493.5	10980.1	1,501.7	457.7	-57	149	1,182.0	1,186.5	4.5	0.081	360.3	361.6	1.4	2.78	3.8
								1,245.0	1,251.0	6.0	0.019	379.5	381.3	1.8	0.65	1.2
								1,266.0	1,270.0	4.0	0.212	385.9	387.1	1.2	7.27	8.9
								1,280.0	1,293.5	13.5	0.014	390.1	394.3	4.1	0.48	2.0
								1,280.0	1,293.5	13.5	0.014	390.1	394.3	4.1	0.48	2.0
VGP-12-131	9193.7	7917.5	10962.4	1,663.4	507.0	-55	147	nsv								
VGP-12-132	8675.3	7484.2	10968.7	1,505.9	459.0	-57	148	809.0	814.0	5.0	0.024	246.6	248.1	1.5	4.03	6.1
								865.0	873.0	8.0	0.023	263.7	266.1	2.4	4.03	9.8
								886.0	891.0	5.0	0.083	270.1	271.6	1.5	4.03	6.1
								961.1	965.0	3.9	0.019	292.9	294.1	1.2	4.03	4.8
								973.4	1,045.1	71.7	0.027	296.7	318.5	21.9	4.03	88.1
								1,133.0	1,164.3	31.3	0.099	345.3	354.9	9.5	4.03	38.4
								1,137.0	1,153.0	16.0	0.156	346.6	351.4	4.9	4.03	19.7
								1,194.0	1,217.5	23.5	0.014	363.9	371.1	7.2	4.03	28.9
								1,239.2	1,275.0	35.8	0.086	377.7	388.6	10.9	4.03	44.0
								1,269.0	1,275.0	6.0	0.275	386.8	388.6	1.8	4.03	7.4
								1,312.0	1,324.0	12.0	0.052	399.9	403.6	3.7	4.03	14.7
								1,359.0	1,364.0	5.0	0.047	414.2	415.7	1.5	4.03	6.1
VGP-12-133	7567.3	7261.6	10991.6	2,062.6	628.7	-50	148	1,508.0	1,513.5	5.5	0.020	459.6	461.3	1.7	4.03	6.8
								1,547.1	1,552.2	5.1	0.015	471.6	473.1	1.6	4.03	6.3
VGP-12-134	7774.5	5794.0	10975.6	801.1	244.2	-50	345	235.0	240.0	5.0	0.028	71.6	73.2	1.5	4.03	6.1
								257.0	271.0	14.0	0.031	78.3	82.6	4.3	4.03	17.2
								306.9	318.3	11.4	0.034	93.5	97.0	3.5	4.03	14.0
								370.0	374.0	4.0	0.036	112.8	114.0	1.2	4.03	4.9
								403.5	407.9	4.4	0.020	123.0	124.3	1.3	4.03	5.4
BA North Zone - Partial																
VGP-11-115	4837.9	8598.8	11052.7	2,352.3	717.0	-45	225	179.0	182.0	3.0	0.039	54.6	55.5	0.9	1.34	1.2
				Pay	Lex			187.0	239.0	52.0	0.029	57.0	72.8	15.8	0.99	15.7
				151.5	2,200.8			335.0	339.0	4.0	0.042	102.1	103.3	1.2	1.43	1.7

TABLE 10.4
PAYMASTER LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

Hole #	East	North	Elev	Length (ft)	Length (m)	Dip	Az	From (ft)	To(ft)	Width (ft)	Grade (opt)	From (m)	To (m)	Width (m)	Au-gpt	GxW
								1,537.4	1,542.0	4.6	0.045	468.6	470.0	1.4	1.56	2.2
VGP-11-122	5391.0	7604.0	11015.3	987.5	301.0	-45	267	913.1	921.0	7.9	0.182	278.3	280.7	2.4	6.24	15.0
end in ugo				Pay	Lex			947.0	954.0	7.0	0.043	288.6	290.8	2.1	1.46	3.1
				567.6	419.9			954.0	987.5	33.5	UGO					
VGP-11-123	5165.2	7683.2	10997.7	1,276.2	389.0	-45	266	473.5	479.6	6.1	0.152	144.3	146.2	1.9	5.20	9.7
1 ugo				Pay	Lex			577.4	583.4	6.0	UGO					
				526.2	750.0			561.8	590.7	28.9	0.053	171.2	180.0	8.8	1.80	15.9
								incl		7.3		incl		2.2	4.69	10.4
								601.8	608.2	6.4	0.052	183.4	185.4	2.0	1.77	3.5
								788.5	831.5	43.0	0.092	240.3	253.4	13.1	3.15	41.3
									incl	17.4	0.198		incl	5.3	6.78	36.0
									incl	3.5	0.513		incl	1.1	17.60	18.8
								863.9	869.0	5.1	0.035	263.3	264.9	1.6	1.20	1.9
VGP-11-124	6031.9	7781.5	11012.6	3,303.8	1,007.0	-55	265	863.9	869.0	5.1	0.035	263.3	264.9	1.6	1.20	1.9
				Pay	Lex			722.2	727.2	5.0	0.223	220.1	221.7	1.5	7.63	11.6
				1,078.9	2,224.9			785.5	790.2	4.7	0.037	239.4	240.9	1.4	1.27	1.8
								2,065.4	2,075.4	10.0	0.226	629.5	632.6	3.0	7.75	23.6
								2,598.7	2,615.7	17.0	0.084	792.1	797.3	5.2	2.88	14.9
VGP-11-126	6343.0	7984.6	11011.9	3,379.2	1,030.0	-60	270	311.0	315.0	4.0	0.048	94.8	96.0	1.2	1.65	2.0
				Pay	Lex			324.0	333.0	9.0	0.033	98.8	101.5	2.7	1.12	3.1
				3,184.1	195.1			462.0	467.2	5.2	0.076	140.8	142.4	1.6	2.59	4.1
								484.0	494.5	10.5	0.062	147.5	150.7	3.2	2.11	6.7
								1,190.3	1,197.3	7.0	0.035	362.8	364.9	2.1	1.21	2.6
								2,814.9	2,824.1	9.2	0.066	858.0	860.8	2.8	2.28	6.4
VGP-11-126W	6343.0	7984.6	11011.9	5,022.0	1,530.7	-60	270	3,371.0	3,381.0	10.0	0.031	1,027.5	1,030.5	3.0	1.06	3.2
				Pay	Lex			3,495.0	3,500.0	5.0	0.038	1,065.3	1,066.8	1.5	1.30	2.0
				175.2	4,846.8			3,846.0	3,851.0	5.0	0.088	1,172.3	1,173.8	1.5	3.00	4.6
								4,276.0	4,301.0	25.0	0.073	1,303.3	1,310.9	7.6	2.49	19.0
VBA-11-164	4901.2	6572.4	11015.1	2,171.9	662.0	-45	349	36.7	46.1	9.4	0.138	11.2	14.1	2.9	4.74	13.6
				Pay	Lex			130.0	142.5	12.5	0.019	39.6	43.4	3.8	0.67	2.5
				891.9	1,280.0			419.0	423.7	4.7	0.085	127.7	129.1	1.4	2.90	4.2
								442.1	464.0	21.9	0.044	134.8	141.4	6.7	1.51	10.1
								472.0	480.0	8.0	0.219	143.9	146.3	2.4	7.49	18.3
								490.0	500.0	10.0	0.020	149.4	152.4	3.0	0.67	2.0
								511.0	534.5	23.5	0.083	155.8	162.9	7.2	2.84	20.4
								1,729.0	1,732.4	3.4	0.160	527.0	528.0	1.0	5.50	5.7
								2,031.4	2,040.0	8.6	0.027	619.2	621.8	2.6	0.93	2.4
VBA-11-173	4427.3	8298.1	10989.7	1,092.5	333.0	-45	88	633.4	644.4	11.0	0.017	193.1	196.4	3.4	0.57	1.9
				Pay	Lex			688.3	698.0	9.7	0.034	209.8	212.8	3.0	1.15	3.4
				605.4	487.1			713.6	725.5	11.9	0.107	217.5	221.1	3.6	3.65	13.3
VBA-11-175	4478.8	8174.0	10989.8	925.2	282.0	-45	88	600.2	619.9	19.7	0.064	182.9	188.9	6.0	2.19	13.2
				Pay	Lex			626.9	633.9	7.0	0.115	191.1	193.2	2.1	3.93	8.4
				527.9	397.3											
VBA-11-178	4478.8	8174.0	10989.8	1,504.4	458.5	-70	88	605.7	627.5	21.8	0.110	184.6	191.3	6.6	3.77	25.1
				Pay	Lex			886.0	889.0	3.0	0.388	270.1	271.0	0.9	13.30	12.2
				702.8	801.6			925.0	930.0	5.0	0.059	281.9	283.5	1.5	2.03	3.1
								935.0	940.0	5.0	0.057	285.0	286.5	1.5	1.96	3.0
								1,391.0	1,426.0	35.0	0.024	424.0	434.6	10.7	0.83	8.9
VBA-11-185b	3726.6	7651.5	11056.2	2,303.1	702.0	-50	82	1,809.8	1,813.8	4.0	0.080	551.6	552.8	1.2	2.75	3.4
				Pay	Lex			1,844.0	1,847.5	3.5	1.403	562.1	563.1	1.1	48.10	51.3
				612.3	1,690.8			2,010.3	2,019.0	8.7	0.045	612.7	615.4	2.7	1.54	4.1
								2,163.4	2,168.0	4.6	0.042	659.4	660.8	1.4	1.44	2.0
VBA-11-187	3638.8	7899.1	11034.1	2,411.4	735.0	-60	80	1,328.4	1,363.4	35.0	0.020	404.9	415.6	10.7	0.68	7.3
				Pay	Lex			1,375.0	1,413.6	38.6	0.056	419.1	430.9	11.8	1.92	22.5
				316.1	2,095.3			1,508.0	1,515.5	7.5	0.034	459.6	461.9	2.3	1.17	2.7
								1,559.2	1,573.1	13.9	0.041	475.2	479.5	4.2	1.42	6.0
								1,591.5	1,619.5	28.0	0.124	485.1	493.6	8.5	4.25	36.3
								1,933.4	1,962.0	28.6	0.040	589.3	598.0	8.7	1.35	11.8
								2,094.9	2,098.7	3.8	0.061	638.5	639.7	1.2	2.08	2.4
								2,180.3	2,183.5	3.2	0.128	664.6	665.5	1.0	4.38	4.3

10.4 DAVIDSON TISDALE

Lexam VG has a 68.5% earned interest in The David Tisdale Property, with SGX Resources Inc. holding the remainder of 31.5%. The Davidson Tisdale Property consists of 10 claims covering approximately 513 acres (208 hectares) in Tisdale Township in the Timmins Mining Camp.

Lexam VG conducted surface drilling on the property during the period 2003 to 2010. The distribution of the drilling is as seen below:

TABLE 10.5
DISTRIBUTION OF DRILLING

Total		91	holes	23123.2	m
Main	Zone	46	holes	18467.2	m
S	Zone	45	holes	4656	m

Drilling was primarily focused on increasing the resource of the Main Zone, along strike and at depth. Previous exploration had been concentrated in the vicinity of the historic workings with little drilling below the 500-foot level of the mine. Typically, mines in the Timmins Camp have great depth extent, commonly exceeding 2,000 ft.

Results from the Lexam VG drilling were very favourable, indicating high grade gold mineralization below and peripheral to the historic exploration and resources.

Results indicate the high gold grades associated with the Main Zone mineralization when the vein system is encountered. The term Quartz Vein System ("QVS") refers to zones of quartz mineralization hosted within the altered Mafic Volcanic. The QVS forms bodies of quartz containing assimilated host rock with pyrite and visible gold. The attitude of the QVS within an envelope is striking 70º and dipping to the north at approximately 70º. The individual vein sets within this envelope can have various orientations, from steep to flat, and varying dimensions.

Table 10.6
Davidson Tisdale Lexam Diamond Drill Hole Summary

	Location										
Hole#	North	East	ele	Az	Dip	Depth	From	To	width	Au g/t	G x W
03-301	9939.1	10234.2	3300.2	150	-45	69.0	52.6	55.0	2.4	9.22	22.1
03-302	9920.0	10218.0	3302.0	150	-72	51.0	15.5	21.1	5.6	1.46	8.2
03-303	9920.0	10218.0	3301.9	150	-53	32.0	14.5	24.6	10.1	1.30	13.1
03-304	9936.4	10218.7	3301.5	150	-45	38.0	16.1	20.5	4.4	38.55	169.6
03-305	9936.2	10218.5	3301.6	150	-74	35.0	23.9	26.1	2.2	12.16	26.8
03-306	9868.0	10202.2	3308.7	150	-45	45.0	41.1	42.4	1.3	2.35	3.1
03-307	9869.3	10201.3	3308.7	150	-85	26.0	nsv				
03-308	9880.3	10168.6	3309.8	150	-45	53.0	nsv				
03-309	9879.4	10168.9	3309.5	150	-64	41.0	9.3	12.7	3.4	0.19	0.6
03-310	9925.0	10113.3	3306.2	324	-80	203.0	163.0	166.4	3.4	341.10	1159.7
03-311	9925.3	10113.4	3306.4	330	-70	326.4	263.0	265.5	2.5	1.56	3.9
03-312	9925.1	10113.5	3306.4	150	-63	162.0	67.3	70.8	3.5	29.86	104.5
03-313	9908.5	10064.8	3311.2	150	-74	143.0	138.5	141.5	3.0	20.18	60.5
03-314	9908.7	10064.7	3311.3	150	-85	147.4	nsv				
03-315	9955.4	10067.8	3309.5	150	-77	226.0	161.1	165.7	4.6	197.34	907.8
							207.4	210.8	3.4	121.15	411.9
03-316	9955.5	10067.8	3309.4	147	-50	218.0	40.7	45.3	4.6	0.06	0.3
03-317	9886.0	10136.5	3311.0	150	-76	132.0	119.4	125.2	5.8	1.24	7.2
04-318	9933.0	10080.2	3309.1	300	-85	272.0	41.8	46.1	4.3	0.08	0.4
04-319	9933.4	10080.1	3309.0	340	-88	35.0	nsv				
04-320	9933.5	10080.1	3308.6	4.1	-87	217.0	168.1	168.7	0.6	0.74	0.4
04-321	9886.3	10077.5	3311.7	352	-80	230.0	nsv				
04-322	9782.1	10151.3	3313.5	340	-59	284.0	138.7	144.8	6.1	6.36	38.8
04-323	9942.4	10094.0	3309.1	150	-46	254.0	147.6	152.6	5.0	3.97	19.9
04-324	9942.5	10089.5	3308.0	188	-86	257.0	159.0	161.7	2.7	9.18	24.8
04-325	9970.8	10043.0	3312.4	150	-71	248.0	195.5	218.5	23.0	20.10	462.3
04-326	9834.4	10122.2	3315.2	329	-56	317.0	251.9	260.4	8.5	30.05	255.4
04-327	9898.3	10158.6	3312.0	330	-65	356.0	170.7	180.5	9.8	11.45	112.2
							317.7	327.0	9.3	5.34	49.6
04-328	9832.6	10079.3	3317.1	324	-65	414.0	300.0	303.4	3.4	0.16	0.5
04-329	9832.0	10079.7	3317.2	332	-75	230.0	166.4	182.0	15.6	0.14	2.1
04-330	9832.3	10079.5	3317.1	334	-61	436.0	371.0	373.7	2.7	0.14	0.4
04-331	9911.8	10207.6	3303.4	330	-59	466.0	nsv				
04-332	9914.9	10290.4	3301.9	330	-55	371.0	167.8	176.4	8.6	7.54	64.9
							245.2	250.7	5.5	0.69	3.8
							189.2	206.2	17.0	0.21	3.6
04-333	9914.7	10290.5	3301.8	330	-62	200.0	nsv				
04-334	9890.8	10331.1	3301.0	330	-50	368.0	44.0	48.3	4.3	1.10	4.7
04-335	9936.9	10219.2	3300.4	330	-72	323.0	nsv				
04-336	9962.2	10231.9	3301.9	330	-78	299.0	144.1	154.4	10.3	1.27	13.1
04-337	10002.5	10265.5	3300.8	330	-85	188.0	124.3	131.1	6.8	2.71	18.4
04-338	10012.9	10319.5	3299.0	330	-77	170.0	nsv				
04-339	10040.2	10360.7	3300.3	150	-75	194.0	nsv				
04-340	10040.0	10360.9	3299.9	150	-45	146.0	nsv				
04-341	9986.9	10443.7	3299.7	330	-60	223.0	nsv				
04-342	9986.5	10443.9	3299.9	330	-88	131.0	101.4	105.0	3.6	0.65	2.3

TABLE 10.6
DAVIDSON TISDALE LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

	Location										
Hole#	North	East	ele	Az	Dip	Depth	From	To	width	Au g/t	G x W
04-343	10057.2	10462.9	3299.8	330	-70	299.0	nsv				
04-344	10062.3	10516.7	3299.2	150	-64	47.0	nsv				
04-345	10063.0	10516.0	3299.5	330	-64	236.0	nsv				
04-346	9908.1	10355.8	3300.5	330	-45	359.0	273.5	277.8	4.3	1.66	7.1
04-347	9705.3	10123.9	3312.3	330	-47	260.0	S	Zone			
04-348	9693.0	10101.9	3313.2	330	-49	431.0	S	Zone			
04-349	9855.3	10022.8	3318.7	330	-68	470.0	S	Zone			
04-350	9753.5	10136.9	3311.6	330	-66	224.0	S	Zone			
05-351	9959.3	10064.3	3310.0	300	-83	230.0	nsv				
05-352	9959.3	10064.1	3309.9	10	-75	341.0	293.3	298.3	5.0	31.87	159.4
							308.0	309.5	1.5	2.54	3.8
							321.6	323.0	1.4	1.38	1.9
05-353	10015.9	10091.2	3308.0	150	-87	330.0	nsv				
05-354	9960.8	10155.5	3303.9	322	-71	455.0	282.6	285.1		17.28	
05-355	9961.2	10156.3	3304.0	335	-74	443.0	290.0	308.0	18.0		
05-356	9961.1	10156.3	3303.9	324	-79	308.0	nsv				
07-357	9985.4	10150.1	3302.9	334	-72	415.0	S	Zone			
07-358	9545.9	9899.7	3316.8	180	-75	107.0	S	Zone			
07-359	9559.4	9874.5	3317.0	180	-72	110.0	S	Zone			
07-360	9561.8	9849.9	3317.3	180	-85	110.0	S	Zone			
07-361	9701.2	9724.6	3321.3	180	-82	260.0	S	Zone			
07-362	9720.1	9774.2	3321.0	180	-80	164.0	S	Zone			
VGT-10-363	9384.4	9724.8	3310.5	180	-45	251.0	S	Zone			
VGT-10-364	9384.7	9724.7	3310.3	180	-60	92.0	S	Zone			
VGT-10-365	9486.7	9849.2	3312.2	355	-45	176.0	S	Zone			
VGT-10-366	9463.5	9877.5	3311.7	355	-45	140.0	S	Zone			
VGT-10-367	9482.3	9924.5	3313.7	355	-45	119.0	S	Zone			
VGT-10-368	9485.7	9950.1	3314.0	345	-45	110.0	S	Zone			
VGT-10-369	9464.7	9999.5	3314.1	350	-45	131.0	S	Zone			
VGT-10-370	9509.1	9998.2	3312.7	355	-45	149.0	S	Zone			
VGT-10-371	9532.4	9948.5	3316.0	345	-45	140.0	S	Zone			
VGT-10-372	9527.9	9874.7	3315.4	355	-45	170.0	S	Zone			
VGT-10-373	9546.1	9847.9	3315.8	355	-45	191.0	S	Zone			
VGT-10-374	9649.7	9966.4	3324.2	355	-55	221.0	S	Zone			
VGT-10-375	9582.1	9926.8	3317.4	355	-45	215.0	S	Zone			
VGT-10-376	10051.0	10012.0	0.0	145	-64	251.0	nsv				
VGT-10-377	10110.0	10064.5	3315.0	139	-70	347.0	275.0	277.0	2.0	1.03	2.1
							285.5	291.5	6.0	3.91	23.4
VGT-10-378	10076.0	10055.5	3315.0	153	-62	296.0	216.0	217.5	1.5	0.70	1.0
VGT-10-379	10091.5	10032.0	3315.0	145	-57	299.0	226.0	227.5	1.5	2.27	3.4
							232.4	233.9	1.5	1.23	1.8
VGT-10-380	10091.5	10032.0	3315.0	145	-72	320.0	269.8	278.8	9.0	2.90	26.1
VGT-10-381	10036.0	10035.5	3315.0	145	-63	275.0	nsv				
VGT-10-382	9995.4	9973.3	3308.7	56	-58	440.0	347.3	348.8	1.5	1.85	2.8
VGT-10-383	9995.2	9972.6	3308.8	60	-85	404.4	311.4	313.7	2.3	2.99	6.7
VGT-10-384	9995.4	9972.9	3308.8	60	-70	398.0	290.2	293.9	3.7	3.27	12.1

TABLE 10.6
DAVIDSON TISDALE LEXAM DIAMOND DRILL HOLE SUMMARY (CONT'D)

	Location										
Hole#	**North**	**East**	**ele**	**Az**	**Dip**	**Depth**	**From**	**To**	**width**	**Au g/t**	**G x W**
							301.8	303.3	1.5	1.78	2.7
							311.7	312.8	1.1	3.12	3.3
							318.6	324.1	5.5	0.96	5.3
VGT-10-385	9995.7	9973.0	3308.7	55	-61	473.0	nsv				
VGT-10-386	9995.8	9972.8	3308.6	54	-65	434.0	47.8	48.8	1.0	3.02	3.0
							109.3	110.4	1.1	1.32	1.4
							305.0	306.3	1.3	8.77	11.7
VGT-10-387	9985.2	9968.6	3308.4	15	-83	419.0	312.0	314.0	2.0	3.35	6.7
							326.4	327.4	1.0	5.10	5.1
VGT-10-388	9983.4	9967.9	3308.5	325	-86	440.0	231.5	233.0	1.5	1.88	2.8
VGT-10-389	9983.4	9967.9	3307.0	332	-75	569.0	416.5	419.1	2.5	1.01	2.5
							447.8	453.4	5.6	1.50	8.4
VGT-10-390	9977.6	9898.8	3309.5	325	-72	674.0	nsv				
VGT-10-391	9977.3	9898.3	3309.4	306	-63	803.0	84.5	85.9	1.4	3.76	5.3
			91.0	**hole**		**23,123.2**	**m**				
	Main	**Zone**	**46.0**			**18,467.2**	**m**				
	S	**Zone**	**45.0**	**hole**		**4,656.0**	**m**				

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 BUFFALO ANKERITE

11.1.1 Drill Program Procedures

The sample preparation, analyses and security procedures for drilling carried out by VG Gold Corp., at the Buffalo Ankerite Property for the period from 2002 to 2006 have been described in the following reports:

- A February 11, 2009 report, authored by Peter A. Bevan, P.Eng., titled "Qualifying Report on the Buffalo-Ankerite Property, South Porcupine, Northern Ontario, Canada" and;
- An October 20, 2012 report, authored by Peter A. Bevan, P.Eng. and Kenneth W. Guy, P.Geo., titled "Resource Estimate on the Buffalo Ankerite Property, Porcupine Mining Division, Northeastern Ontario, Canada".

Both reports are available on the SEDAR website.

All drill core from the 2009 to 2012 drill programs at the Buffalo Ankerite Property, was picked up from the drill site and directly delivered to the Company's South Porcupine core logging facility by Lexam core technicians.

The core technicians then measured the drill core and stapled a metal tag to each of the core boxes with the hole number, box number and footage recorded on the tag. The technicians also took measurements from the drill core, including RQD, core recovery, and orientation of any structures, contacts and veins.

Company geologists logged the drill core, recording the lithological, structural, alteration and mineralogical features observed, as well as selected samples to be analyzed based on the alteration, mineralization and veining observed.

The core was then cut lengthways in half using a manual core splitter. One half of the core sample was placed in a plastic sample bag containing a sample tag for easy identification and then sealed for shipping to the assay laboratory.

Prior to shipment to the laboratory, samples were securely stored at the South Porcupine core facility, which is locked with a security system. The remaining half core was left in the core box and stored at the Company's South Porcupine core facility for future reference.

Ninety nine percent (99%) of the core had 100% core recovery and Lexam has stipulated that no drilling, sampling or recovery factors were encountered that would materially impact the accuracy and reliability of the analytical results. No factors were identified by P&E, which may have resulted in a sample bias.

11.1.2 Sampling Protocol

Samples were transported directly to the ALS Chemex Laboratory in Timmins, Ontario ("ALS") by Lexam core technicians for sample preparation and analyses.

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

ALS maintains ISO registrations and accreditations, and most ALS Minerals laboratories are registered or are pending registration to ISO 9001:2008. A number of analytical facilities, including the Timmins Laboratory, have received ISO 17025 accreditations for specific laboratory procedures.

At ALS the samples were dried and crushed to 70% passing minus ten (-10) mesh. A Jones riffle splitter was used to take a 250 g sub sample for pulverizing and the reject portion was bagged and stored. After reducing the 250 g sample to 85% passing -200 mesh, the sample was thoroughly blended and a 30 gram charge was assayed for gold by standard fire assay-ICP finish. Gold values in excess of 10 ppm were re-analyzed by fire assay with gravimetric finish for greater accuracy.

A secondary laboratory, Laboratoire Expert ("Expert") of Rouyn-Noranda, Quebec, was used to check the analyses obtained by ALS. Expert is registered under ISO 9001:2000 quality standard and participates in the CANMET PTP-MAL Laboratory Proficiency testing.

11.1.3 Diamond Drilling Quality Assurance/Quality Control

Lexam did not undertake their own Quality Assurance and Quality Control Program ("QA/QC Program") for the drilling carried out at their Buffalo Ankerite Property. They instead relied upon ALS Laboratory's own internal QA/QC Program, as well as submitting 10% of all pulp samples to a second laboratory (Expert) for check analysis. Both the internal laboratory QA/QC and the check analysis are discussed in Section 12.

It is P&E's opinion that the sampling methods, security and analytical procedures used were adequate to have provided sufficient geotechnical and geological information.

It is recommended that Lexam implement a QA/QC program for future drilling undertaken at the Buffalo Ankerite Property. These recommendations are detailed in the Recommendations Section of the report.

11.2 FULLER DEPOSIT

11.2.1 Pre-2007 Programs

Sample preparation, analysis, and security on the Fuller Project prior to 2009 were described by Wardrop in the previous technical report (Wardrop, 2007) and are summarized below.

Sampling procedures and protocols were similar for the 1996-1998, 2004, and 2006-2007 drilling campaigns. Drill core sections were sampled according to lithological and/or alteration boundaries and vein(s) width, with the maximum sample interval not exceeding five feet in length. The sections of core to be sampled were split, with half-core retained for assay and the remainder stored in core boxes. All of the sections of sampled core were cut using a manual core splitter.

The sample preparation and analytical work was carried out by Swastika Laboratories of Kirkland Lake, Ontario, for the 1996-1998 drilling campaigns and by Laboratoire Expert Inc. (Expert) in Rouyn Noranda, Quebec, for the 2004 and 2006-2007 drilling campaigns. Samples were crushed to 70% passing -2.0 mm and 250 g was collected and pulverized to 85% passing <75 µm in a ring mill. The pulverized sample was then split utilizing a riffle splitter. Analysis for gold was carried out using a one ton (30 g) fire assay with an atomic absorption spectroscopy (AAS) finish.

11.2.2 2009-2012 Programs

Sampling Methods

Drill core was laid out at the Lexam core logging facility and marked for sampling by a geologist based on observed geological units, alteration, and veining and/or mineralization intensity. All drill core was photographed after samples were marked. Whenever possible, four core boxes were photographed at a time and the images were downloaded to the Lexam server. Sample length varied from three to seven feet, however, in zones that were well mineralized sample length was limited to about three feet. When visible gold was observed, the drill core was marked for special laboratory analytical methods and photographed separately.

Each core box was labelled on the outside with an aluminum tag noting the drill hole number, box number, and hole depth of each box.

Geochemical results reported from the Fuller Project are from halved drill core samples collected by Lexam personnel on site at their South Porcupine core facility, located on the Tisdale property. Drill core assay samples were taken by a trained Lexam technician and half core splits were bagged and tagged. The core was halved with a manual core splitter. One half of the drill core was secured in a sample bag with a sample number tag. The remaining half was returned to the core box. A duplicate sample tag was stapled into the core box at the start of each sample. The samples were placed into larger bags which were stored in the core logging building until taken to the sample preparation laboratory.

Analysis

From 2009 to 2012, Lexam used ALS Canada Ltd. (ALS), an ISO 17025 accredited testing laboratory, as the primary assay laboratory. Expert, in Rouyn-Noranda, Quebec, was used as the secondary laboratory. Expert has no current accreditations or certifications.

The diamond drill core samples were delivered directly to the ALS sample preparation laboratory in Timmins by Lexam personnel, where the core was subjected to a standard sample preparation of crushing, splitting, and pulverizing. After ALS logged the sample into its tracking system, the sample was crushed to better than 70% passing a 2.0 mm (Tyler 9 mesh, US Std. No.10) screen. A riffle split of up to 1,000 g was pulverized to better than 85% passing 75 µm (Tyler 200 mesh) screen. The pulverized pulp was sent for assay to the ALS laboratory in Val d'Or, Quebec, where a 30 g aliquot was analyzed for gold by fire assay fusion, with an AAS finish. If the initial AAS finish returned an assay value greater than 10 ppm Au (the upper detection limit of AAS), a second 30 g aliquot from the sample pulp was automatically re-assayed by fire assay fusion using a gravimetric analytical method, with an upper detection limit of 1,000 ppm.

When the Lexam geologist observed visible gold during core logging, the sample was assayed by screen metallics. From the final prepared pulp, 1,000 g was passed through a 100 µm screen to separate the oversize fraction. Any +100 µm material was analyzed in its entirety by fire assay with a gravimetric finish and reported as the Au(+) fraction. The -100 µm fraction was homogenized and two subsamples were analyzed by fire assay with an AAS finish. The average of the AAS results was taken and reported as the Au(-) fraction result. All three values were used in calculating the combined gold content (AuTotal). The gold values for both the +100 µm and -100 µm fractions were reported together with the weight of each fraction as well as the calculated total gold content of the sample.

The Fuller drill programs from 2009-2012 relied primarily on the ALS internal sample blanks, standards, and duplicates for quality control. In addition, 10% of the pulp samples were sent to the second laboratory, Expert, in Rouyn-Noranda, Quebec, for assay verification. The pulps were re-assayed using the same methods used by ALS.

Dry Bulk Density Measurements

Dry bulk density samples were taken in 2011 and 2012 by Lexam personnel on altered and unaltered drill core, primarily in mineralized intersections. Measurements were made by weighing the core dry and then immersing the core in a bucket of distilled water, and weighing the core again. The dry bulk density was calculated using the following formula:

$$Dry\ Bulk\ Density = \frac{Weight\ of\ core\ dry\ (g)}{Weight\ of\ core\ dry\ (g) - Weight\ of\ core\ in\ water\ (g)}$$

The calculated measurement was converted to tonnes per cubic feet for use in the Mineral Resource estimate.

Security

Lexam does not have formal security procedures, however, all samples are collected and transported by company personnel. The core logging facility is locked, there is an alarm system, and the entire facility is fenced with access via a locked gate.

RPA does not believe that there is any problem with sample security.

In the opinion of RPA, sampling, sample preparation, assaying, and security procedures used by Lexam at the Fuller Project are reasonable and acceptable for generation of data to use for Mineral Resource estimation.

11.3 PAYMASTER

11.3.1 Placer Dome

RPA has no knowledge of the Placer Dome sample preparation, analysis and security procedures on the Paymaster project.

Placer Dome conducted comprehensive QA/QC programs during all of their drilling on the Paymaster Project to validate the assay results received. This included blind repeat assays at the primary laboratory and check assaying at a secondary laboratory. RPA has been unable to verify the Placer Dome QA/QC program and results.

11.3.2 Lexam

Sampling Methods

Drill core was laid out at the Lexam core logging facility and marked for sampling by a geologist based on observed geological units, alteration, and veining and/or mineralization intensity. All drill core was photographed after samples are marked. Whenever possible, four core boxes were photographed at a time and the images were downloaded to the Lexam server. Sample length varied from three to seven feet, however, in zones that are well mineralized sample length was limited to about three feet. When visible gold was observed, the drill core was marked for special laboratory analytical methods and photographed separately.

Each core box was labelled on the outside with an aluminum tag noting the drill hole number, box number, and hole depth of each box.

Geochemical results reported from the Paymaster Project are from halved drill core samples collected by Lexam personnel on site at their South Porcupine core facility, located on the Tisdale property. Drill core assay samples were taken by a trained Lexam technician and half core splits were bagged and tagged. The core was halved with a manual core splitter. One half of the drill core was secured in a sample bag with a sample number tag. The remaining half was returned to the core box. A duplicate sample tag was stapled into the core box at the start of each sample. The samples were placed into larger bags which were stored in the core logging building until taken to the sample preparation laboratory.

Analysis

From 2009 to 2012, Lexam used ALS Canada Ltd. (ALS), an ISO 17025 accredited testing laboratory, as the primary assay laboratory. Expert, in Rouyn-Noranda, Quebec, was used as the secondary laboratory. Expert has no current accreditations or certifications.

The diamond drill core samples were delivered directly to the ALS sample preparation laboratory in Timmins by Lexam personnel, where the core was subjected to a standard sample preparation of crushing, splitting, and pulverizing. After ALS logged the sample into its tracking system, the sample was crushed to better than 70% passing a 2.0 mm (Tyler 9 mesh, US Std. No.10) screen. A riffle split of up to 1,000 g was pulverized to better than 85% passing 75 µm (Tyler 200 mesh) screen. The pulverized pulp was sent for assay to the ALS laboratory in Val d'Or, Quebec, where a 30 g aliquot was analyzed for gold by fire assay fusion, with an AAS finish. If the initial AAS finish returned an assay value greater than 10 ppm Au (the upper detection limit of AAS), a second 30 g aliquot from the sample pulp was automatically re-assayed by fire assay fusion using a gravimetric analytical method, with an upper detection limit of 1,000 ppm.

When the Lexam geologist observed visible gold during core logging, the sample was assayed by screen metallics. From the final prepared pulp, 1,000 g was passed through a 100 µm screen to separate the oversize fraction. Any +100 µm material was analyzed in its entirety by fire assay with a gravimetric finish and reported as the Au (+) fraction. The -100 µm fraction was homogenized and two subsamples were analyzed by fire assay with an AAS finish. The average of the AAS results was taken and reported as the Au (-) fraction result. All three values were used in calculating the combined gold content (AuTotal). The gold values for both the +100 µm and -100 µm fractions were reported together with the weight of each fraction as well as the calculated total gold content of the sample.

The Paymaster drill programs from 2009-2012 relied primarily on the ALS internal sample blanks, standards, and duplicates for quality control. In addition, 10% of the pulp samples were sent to the second laboratory, Expert, in Rouyn-Noranda, Quebec, for assay verification. The pulps were re-assayed using the same methods used by ALS.

Dry Bulk Density Measurements

Dry bulk density samples were from 2006 to 2012 by Lexam personnel on altered and unaltered drill core, primarily in mineralized intersections. Measurements were made by weighing the core dry and then immersing the core in a bucket of distilled water, and weighing the core again. The dry bulk density was calculated using the following formula:

$$Dry\ Bulk\ Density = \frac{Weight\ of\ core\ dry\ (g)}{Weight\ of\ core\ dry\ (g) - Weight\ of\ core\ in\ water\ (g)}$$

The calculated measurement was converted to tonnes per cubic feet for use in the Mineral Resource estimate.

Security

Lexam does not have formal security procedures; however, all samples are collected and transported by company personnel. The core logging facility is locked, there is an alarm system, and the entire facility is fenced with access via a locked gate.

RPA does not believe that there is any problem with sample security.

In the opinion of RPA, sampling, sample preparation, assaying, and security procedures used by Lexam at the Paymaster Project are reasonable and acceptable for generation of data to use for Mineral Resource estimation.

11.4 DAVIDSON TISDALE DEPOSIT

11.4.1 Drill Program Procedures

The sample preparation, analyses and security procedures for drilling carried out by Vedron Gold Inc., at the Davidson-Tisdale Property for the period from 2003 to 2005 have been described in the following report:

- A March 23, 2007 report, authored by Kenneth Guy, P.Geo., titled “Exploration Report 2003 – 05 and Resource Estimate Technical Report on the Tisdale Project, Porcupine Mining Division, Northeastern Ontario, Canada”.

All drill core from the 2010 to 2012 drill programs at the Davidson-Tisdale Property, was picked up from the drill site and directly delivered to the Company’s South Porcupine core logging facility by Lexam core technicians.

The core technicians then measured the drill core and stapled a metal tag to each of the core boxes with the hole number, box number and footage recorded on the tag. The technicians also took measurements from the drill core, including RQD, core recovery, and orientation of any structures, contacts and veins.

Company geologists logged the drill core, recording the lithological, structural, alteration and mineralogical features observed, as well as selected samples to be analyzed based on the alteration, mineralization and veining observed.

The core was then cut lengthways in half using a manual core splitter. One half of the core sample was placed in a plastic sample bag containing a sample tag for easy identification and then sealed for shipping to the assay laboratory.

Prior to shipment to the laboratory, samples were stored at the South Porcupine core facility, which is locked with a security system. The remaining half core was left in the core box and stored at the Company's South Porcupine core facility for future reference.

Ninety nine percent (99%) of the core had 100% core recovery and Lexam has stipulated that no drilling, sampling or recovery factors were encountered that would materially impact the accuracy and reliability of the analytical results. No factors were identified by P&E, which may have resulted in a sample bias.

11.4.2 Sampling Protocol

Samples were transported directly to the ALS Chemex Laboratory in Timmins, Ontario (“ALS”) by Lexam core technicians for sample preparation and analyses.

ALS Minerals has developed a Quality Management System (“QMS”) designed to ensure the production of consistently reliable data and implemented this at each of its locations. The system covers all laboratory activities and takes into consideration the requirements of ISO standards.

ALS maintains ISO registrations and accreditations, and most ALS Minerals laboratories are registered or are pending registration to ISO 9001:2008. A number of analytical facilities, including the Timmins Laboratory, have received ISO 17025 accreditations for specific laboratory procedures.

At ALS the samples were dried and crushed to 70% passing minus ten (-10) mesh. A Jones riffle splitter was used to take a 250 g sub sample for pulverizing and the reject portion was bagged and stored. After reducing the 250 g sample to 85% passing -200 mesh, the sample was thoroughly blended and a 30 gram charge was assayed for gold by standard fire assay-ICP finish. Gold values in excess of 10 ppm were re-analyzed by fire assay with gravimetric finish for greater accuracy.

A secondary laboratory, Laboratoire Expert ("Expert") of Rouyn-Noranda, Quebec, was used to check the analyses obtained by ALS. Expert is registered under ISO 9001:2000 quality standard and participates in the CANMET PTP-MAL Laboratory Proficiency testing.

11.4.3 Diamond Drilling Quality Assurance/Quality Control

Lexam did not undertake their own Quality Assurance and Quality Control Program ("QA/QC Program") for the drilling carried out at their Davidson-Tisdale Property. They instead relied upon ALS Laboratory's own internal QA/QC Program, as well as submitting 10% of all pulp samples to a second laboratory (Expert) for check analysis. The internal laboratory QA/QC for ALS is discussed in Section 12.

It is P&E's opinion that the sampling methods, security and analytical procedures used were adequate to have provided sufficient geotechnical and geological information.

It is recommended that Lexam implement a QA/QC program for future drilling undertaken at the Davidson-Tisdale Property. These recommendations are detailed in the Recommendations Section of the report.

12.0 DATA VERIFICATION

12.1 BUFFALO ANKERITE

12.1.1 Site Visit and Due Diligence Sampling

The Buffalo Ankerite Property was visited by Mr. Antoine Yassa, P.Geo., an independent Qualified Person as defined by National Instrument NI 43-101 Standards of Disclosure for Mineral Projects, on November 6 and 7, 2012. Ninety-six (96) samples were collected from sixty-one (61) holes by sawing a ¼ split of the half core remaining in the box. The samples were documented, bagged, and sealed with packing tape and were taken by Mr. Yassa to Dicom in Rouyn-Noranda, QC. From there, the samples were shipped to the offices of P&E in Brampton, ON, and sent by courier to AGAT Laboratories in Mississauga, ON for analysis. Gold was analyzed using fire assay on a 30 gram aliquot with an AAS finish. Samples yielding values greater than 10 g/t Au were reassayed and quantitatively determined using the gravimetric method.

AGAT Laboratories employs a quality assurance system to ensure the precision, accuracy and reliability of all results. The best practices have been documented and are consistent with:

- The International Organization for Standardization's ISO/IEC 17025, "General Requirements for the Competence of Testing and Calibration Laboratories' and the ISO 9000 series of Quality Management standards";
- All principles of Total Quality Management (TQM);
- All applicable safety, environmental and legal regulations and guidelines;
- Methodologies published by the ASTM, NIOSH, EPA and other reputable organizations;
- The best practices of other industry leaders.

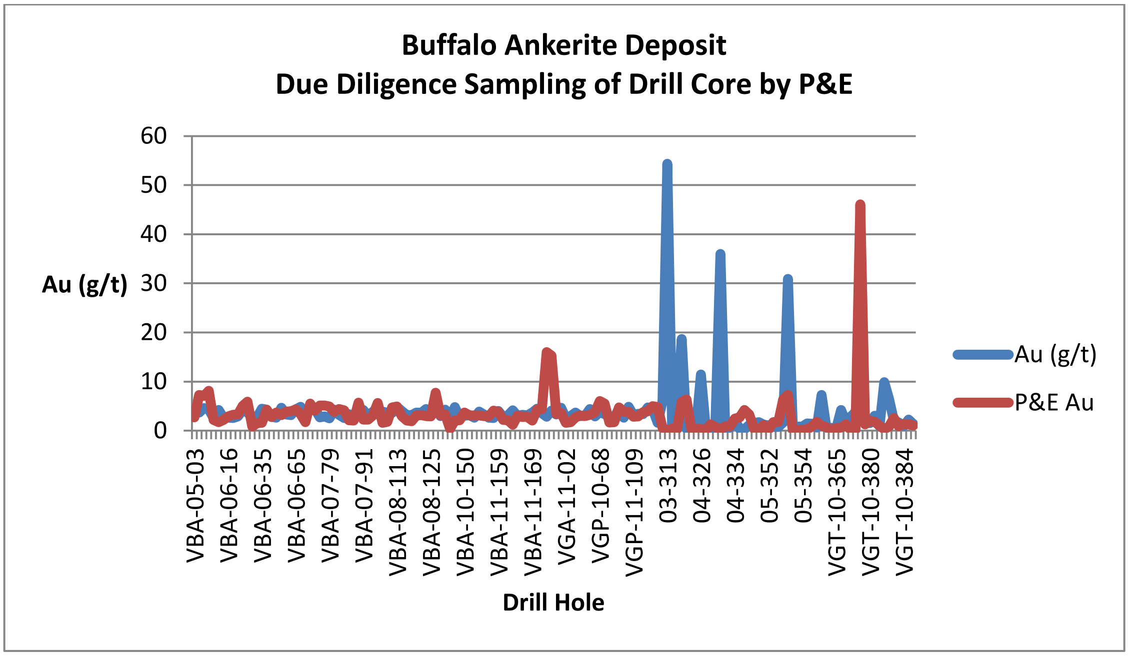
At no time, prior to the time of sampling, were any employees or other associates of Lexam advised as to the location or identification of any of the samples to be collected.

Lexam completed significant surface diamond drilling on the Buffalo Ankerite property during the period 2005 through 2012. The majority of the drilling was conducted on the Buffalo Ankerite South Zone, expanding on the exploration and resource work completed by Placer Dome during the period 2002 through 2005.

Lexam did not insert any of their own QC samples, and P&E felt that in order to validate the earlier drill holes in the database, (in addition to verifying all lab internal QC), a larger than normal number of samples would need to be collected. Holes drilled in years 2005 through 2012 were sampled by Mr. Yassa.

A comparison of the P&E independent sample verification results versus the original assay results for gold can be seen in Figure 12.1. The P&E results demonstrate that results obtained and reported by Lexam were reproducible, though there is evidence of a nugget effect.

Figure 12.1 Due Diligence Sample Results for Gold



12.1.2 Quality Assurance/Quality Control

Lexam did not implement a quality assurance/quality control ("QA/QC" or "QC") program for any of the drilling, however they did rely upon the ALS (principal lab) and Expert (secondary lab) internal QC. Each of these labs implemented strict quality assurance/quality control programs and all resulting data were obtained by P&E and verified for accuracy, precision and absence of contamination.

Holes drilled on the Buffalo Ankerite Property, as well as many holes drilled on the Paymaster Property contributed to the current mineral resource estimate, and as such, QC for each property was evaluated. For the Paymaster Property, all the QC was verified, in spite of the fact that not all holes pertained to Buffalo Ankerite.

<u>ALS Minerals Internal Lab QC for Buffalo Ankerite</u>

From 2009 through 2012, there were 18 different certified reference materials used at one point or another during the drill programs. No data prior to 2009 were available. All standards were purchased from either Ore Research and Pty in Australia, or Rocklabs in New Zealand, apart from one standard which was purchased from CDN Resource Labs in Vancouver (Langley), Canada.

All 18 standards were graphed, using the +/- 2 and +/- 3 standard deviation limits as warning and tolerance limits, respectively. There were over 800 standards pertaining to the Buffalo Ankerite holes analyzed at ALS during 2009 to 2012. The standards demonstrated excellent performance, with only a very minor number of failures outside the tolerance limits.

Expert Labs Internal QC for Buffalo Ankerite

Expert used three different certified reference materials during the drill programs, and all three were purchased from Rocklabs in New Zealand.

The three standards were graphed, using the +/- 2 and +/- 3 standard deviation limits as warning and tolerance limits, respectively. There were 76 standards pertaining to the Buffalo Ankerite holes analyzed at Expert during 2009 to 2012. The standards demonstrated excellent performance, with no failures outside the tolerance limits.

ALS Minerals Internal Lab QC for Paymaster

From 2009 through 2012, there were 21 different certified reference materials used at one point or another during the drill programs. No data prior to 2009 were available. All standards were purchased from either Ore Research and Pty in Australia, or Rocklabs in New Zealand, apart from two standards which were purchased from CDN Resource Labs in Vancouver (Langley), Canada.

All 21 standards were graphed, using the +/- 2 and +/- 3 standard deviation limits as warning and tolerance limits, respectively. There were over 1,200 standards pertaining to the Paymaster holes analyzed at ALS during 2009 to 2012. The standards demonstrated excellent performance, with only a very minor number of failures outside the tolerance limits.

12.1.3 Performance of Blank Material

ALS Minerals Internal Lab Blanks for Buffalo Ankerite

There were over 625 blanks inserted with the samples from 2009 to 2012 and all of them reported less than three times the detection limit, indicating an absence of contamination at the analytical level.

Expert Internal Lab Blanks for Buffalo Ankerite

There were 68 blanks inserted with the samples from 2009 to 2012 and all of them reported less than three times the detection limit, indicating an absence of contamination at the analytical level.

ALS Minerals Internal Lab Blanks for Paymaster

There were over 725 blanks inserted with the samples from 2009 to 2012 and all of them reported less than three times the detection limit, indicating an absence of contamination at the analytical level.

12.1.4 Duplicate Precision

ALS Minerals Internal Pulp Duplicates for Buffalo Ankerite

Out of a total of 54,078 (unconstrained) samples, ALS took a second pulp of 1,168 of them and re-assayed them as pulp duplicates. A filter of five times the detection limit of 0.005 g/t Au was applied, in order to get rid of values close to detection limit that would falsely influence the pulp duplicate precision. Of the 1,168 duplicates, 386 were greater than five times the detection limit of 0.005 g/t Au.

A Thompson-Howarth precision plot of the 386 pairs indicated that at the open-pit cut-off grade of 0.50 g/t Au (0.015 opt Au), precision of 20% can be expected, and for the underground resource cut-off grade of 1.5 g/t Au (0.045 opt Au), precision is approximately 17%. A corresponding Absolute Relative Difference versus Mean of the sample pair evaluation yielded comparative results.

Lexam should concentrate on "fine-tuning" the sampling and assaying protocol, in order to minimize the nugget effect and improve precision at the pulp duplicate level. Components of this exercise should include evaluating the collection of a larger sample volume, crushing to a higher percentage passing -10 mesh, and using a 50 gram aliquot for fire assay.

P&E declared the data suitable for use in a mineral resource estimate.

12.2 FULLER DEPOSIT

Katharine Masun, P.Geo., RPA's Senior Geologist, visited the Fuller Property site on November 20-21, 2012. During the site visit, RPA inspected the Fuller Property, including the location of drill collars VGF-11-113, VGF-11-117 and VGF-12-136, the underground decline opening, and the drill rig that was operating on the Property at the time.

RPA carried out a geological core review on lithology, mineralization and sampling, checking against drill logs of the following drill holes: VGF-12-131 and VGF-12-136 through VGF-12-140. During the core review, no notable discrepancies were found: footage tags were placed in the correct locations in the core boxes, samples were clearly and accurately marked, and core boxes were clearly labelled.

RPA reviewed the resource database that formed the basis of the Mineral Resource estimate presented in this Technical Report. This included results from the quality assurance/quality control (QA/QC) program and assay certificates for drill hole samples from 2009 to 2012. No drilling or sampling occurred on the Fuller Property from the time of the previous NI 43-101 report in 2007 to 2009.

12.2.1 Manual Database Verification

The review of the resource database included header, survey, lithology, assay, and density tables. Database verification was performed using tools provided within the Gemcom GEMS software package. As well, the assay and density tables were reviewed for outliers. Minor transcription errors and missing data were noted and promptly repaired by Lexam. A visual check on the drill hole collar elevations and topography was completed. No inconsistencies were noted.

RPA compared several thousand assay records for the current Mineral Resource estimate. This included comparison of 2,992 database assay values to the laboratory certificates (ALS) from drill holes completed from 2009 to 2012. The database was free of gross errors and minor mistakes were promptly corrected by Lexam.

- For records prior to 2009, RPA has relied on Data Verification completed by Wardrop in the 2007 technical report.
- Independent Assays of Drill Core
- RPA did not collect samples from drill core for independent assay during the site visit. Other Qualified Persons had previously sampled the mineralization, as discussed in the 2007 Technical Report by Wardrop. In RPA's opinion, the production history of the Fuller Property confirms that the deposit contains mineralization.

12.2.2 Quality Assurance and Quality Control

Quality assurance (QA) consists of evidence to demonstrate that the assay data has precision and accuracy within generally accepted limits for the sampling and analytical method(s) used in order to have confidence in future resource estimations. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of sampling, preparing, and assaying the exploration drilling samples. In general, QA/QC programs are designed to prevent or detect contamination and allow assaying (analytical) precision (repeatability) and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling–assaying variability of the sampling method itself.

For the Fuller Project QA/QC program, Lexam uses ALS Canada Ltd. (ALS) in Val D'Or, Quebec, an ISO 17025 Accredited Testing Laboratory, as the primary laboratory. Laboratoire Expert Inc. (Expert) in Rouyn-Noranda, Quebec, is used as the secondary laboratory.

12.2.3 Certified Reference Material (Standards)

Results of the regular submission of certified reference materials (CRM or standards) are used to identify problems with specific sample batches and long-term biases associated with the regular assay laboratory. Lexam has not incorporated the use of CRMs into its QA/QC program. Lexam monitors ALS's internal quality control CRM results. RPA reviewed this data and found the results acceptable.

12.2.4 Blanks

Contamination and sample numbering errors are assessed through blank samples, on which the presence of the elements undergoing analysis has been confirmed to be below the corresponding detection limit. A significant level of contamination is identified when the blank sample yields values exceeding three times detection limit of the analyzed element. In order to be effective, the matrix of the blank sample should be similar to the matrix of the material being routinely analyzed and inserted at a rate of 5%, or one per batch of twenty samples. Lexam's QA/QC protocol does not include blank sample insertion in the sample stream.

Lexam monitors ALS's internal quality control blank results. RPA reviewed this data and found the results acceptable.

12.2.5 Duplicates

Field duplicates assess the variability introduced by sampling the same interval. The duplicate splits are bagged separately with separate sample numbers so as to be blind to the sample preparation laboratory. The duplicates contain all levels of sampling and analytical error and are used to calculate field, sample preparation, and analytical precision. They are also a check on possible sample over selection; that is, the sampler has either purposely or inadvertently sampled the geological material (usually drill core) so as to preferentially place visible mineralization in the sample bag going for analysis.

Coarse duplicates (or coarse reject duplicates) are duplicate samples taken immediately after the first crushing and splitting step. The coarse duplicates will inform about the sub-sampling precision. In order to ensure repeatability conditions, both the original and the coarse duplicate samples should be submitted to the same laboratory (primary laboratory), in the same sample batch and under a different sample number, so that pulverization and assaying follow the same procedure.

Pulp duplicates consist of second splits of final prepared pulverized samples, analyzed by the same laboratory as the original samples under different sample numbers. The pulp duplicates are indicators of the analytical precision, which may also be affected by the quality of pulverization and homogenization (i.e., sub-sampling variance introduced during pulp preparation from splits of the original coarse reject material).

One component of ALS's QA/QC program is the regular measurement of laboratory duplicate samples. Lexam has relied on the results of ALS's pulp duplicates to ensure that analytical precision meets project requirements.

Out of a total of 2,292 samples, ALS took second splits of 284 final pulps (12%) and re-assayed them as pulp duplicates. Results showed an acceptable correlation between the gold pulp duplicates for this type of deposit (Figure 12.2). Thompson-Howarth plots illustrated in Figure 12.3 and Figure 12.4 serves to highlight the high nugget effect of the Fuller deposit, indicating that near the open pit cut-off grade of 0.015 opt Au, an analytical precision of only ±40% can be expected. RPA has converted all analytical results from parts per million (ppm) to ounces per ton (opt).

Figure 12.2 Fuller Pulp Duplicate Samples - 2009-2012-ALS 2009-2012 Pulp Duplicate Scatter Plot

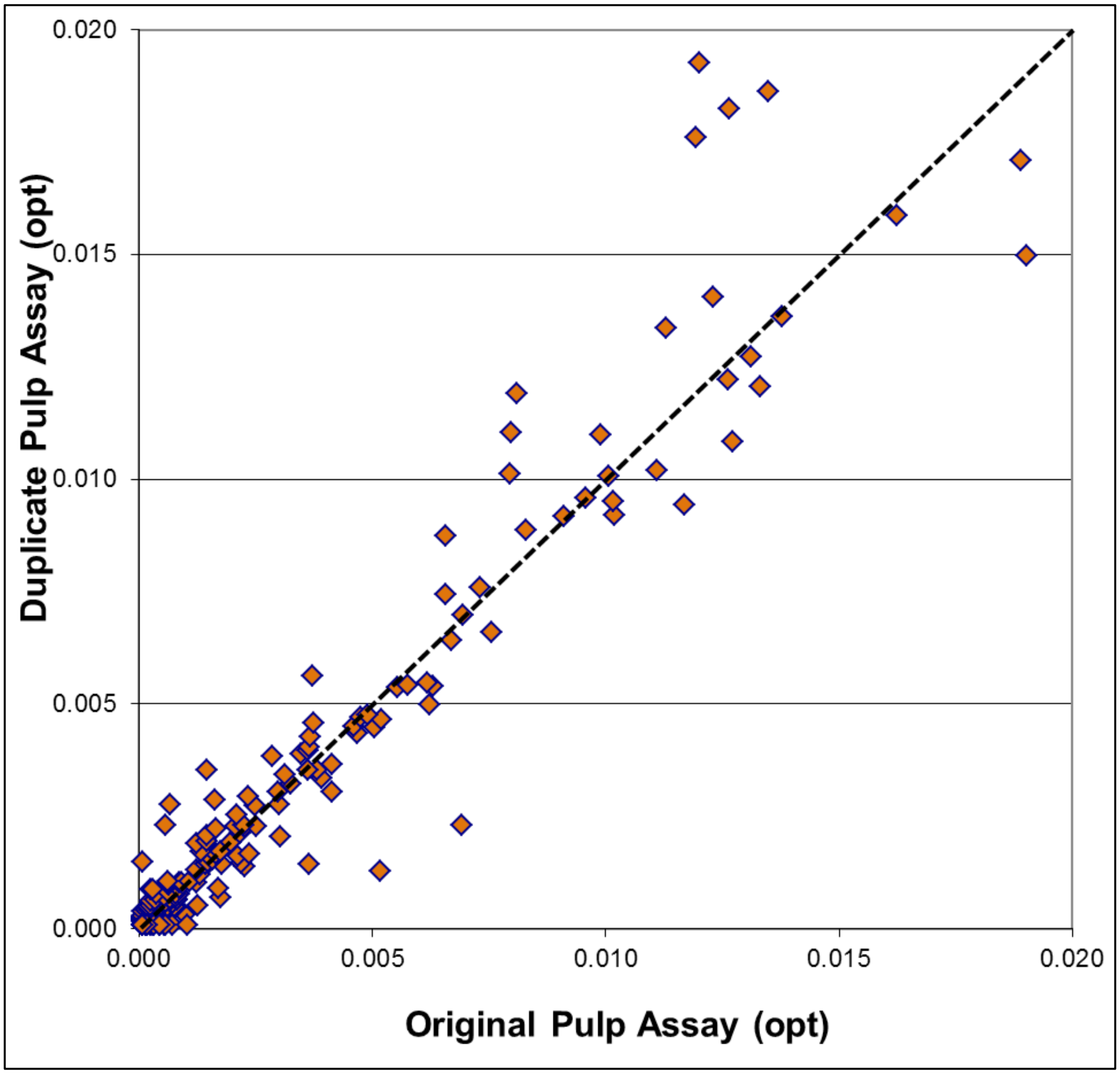


Figure 12.3 Fuller Pulp Duplicate Samples – 2009-2012-ALS 2009-2012 Pulp Duplicate Thompson-Howarth Plot

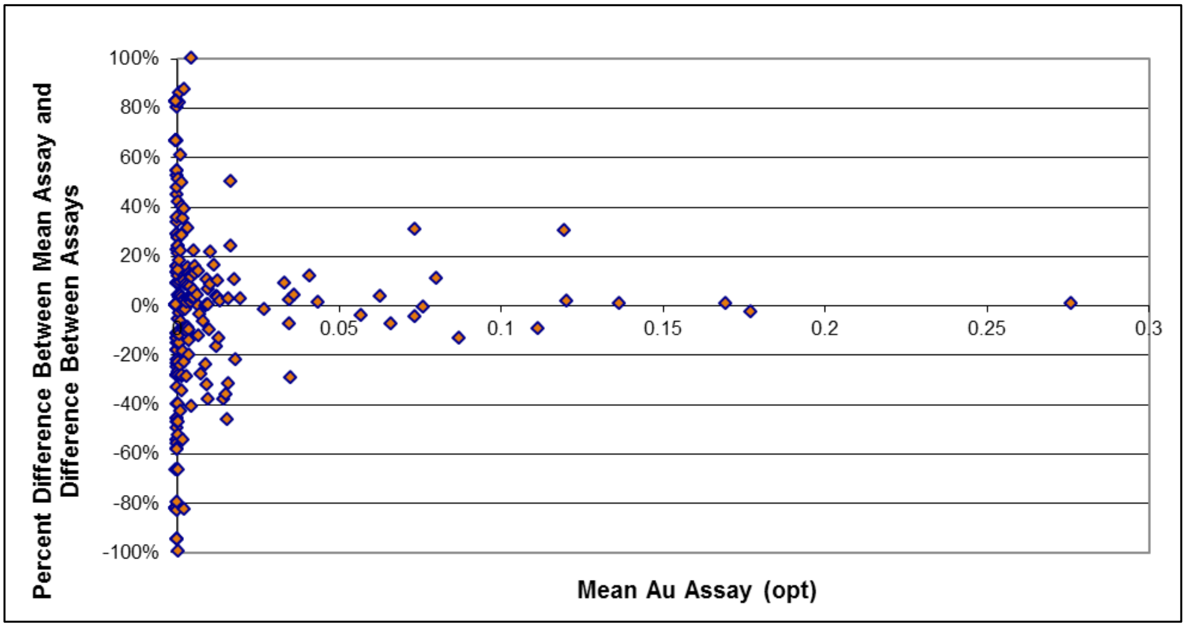
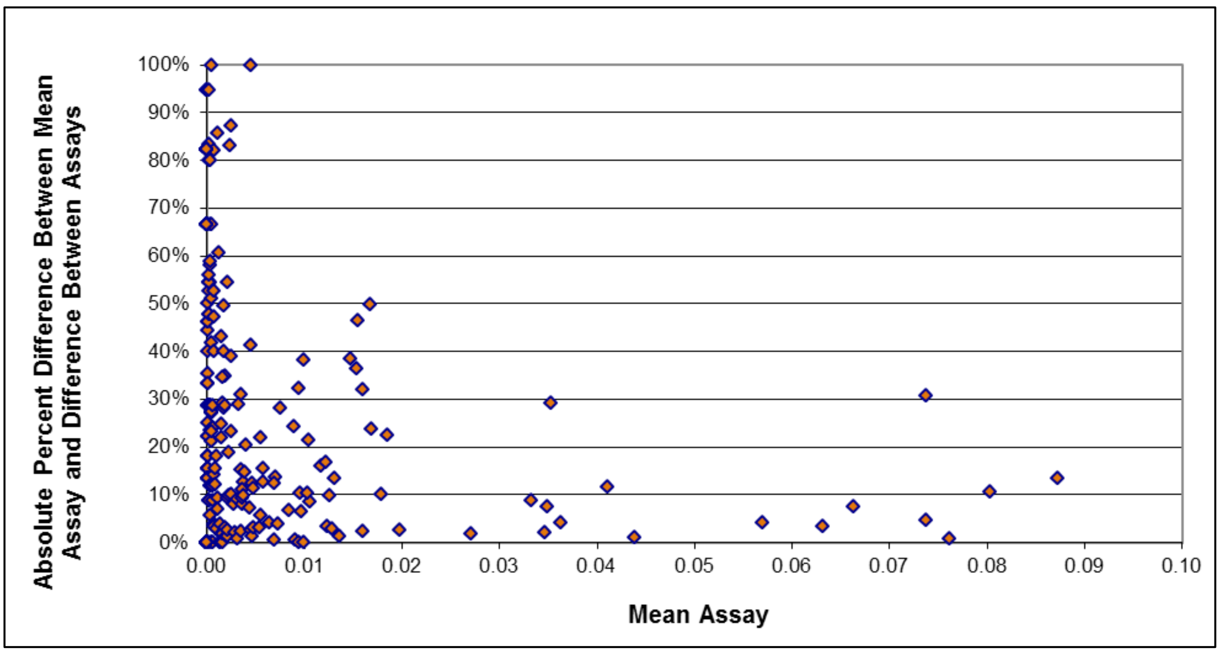


Figure 12.4 Als 2009-2012 Pulp Duplicate Results Showing Modified Thompson-Howarth Plot Below 0.10 Opt



Lexam has not incorporated field or coarse reject duplicates into the Fuller Project QA/QC program.

It is RPA's opinion that Lexam should incorporate field and coarse duplicates into the Fuller QA/QC program. RPA recommends that field and coarse reject duplicates be taken in the future and submitted to the primary laboratory for analysis. If practicable, duplicates should be chosen to represent the range of grades expected and for a better representation of the precision. Field, coarse reject, and pulp duplicates will provide information about the precision at three different stages in the sampling stream.

Check Samples

Check samples consist of second splits of the final prepared pulverized samples routinely analyzed by the primary laboratory (ALS) and resubmitted to a secondary laboratory (Expert) under a different sample number. These samples are used to assess the assay accuracy of the primary laboratory relative to the secondary laboratory.

Lexam's QA/QC protocol calls for check samples to be taken at a rate of one in every ten to twenty samples and submitted to a secondary laboratory. RPA received the results for 181 duplicate pairs, which covered drilling completed from 2009 to 2012. There are no check samples for drilling completed prior to 2009.

Table 12.1 summarizes the statistics for gold check assay pairs. After removing a single outlier pair, the results from the external check assays indicate good correlation for drilling completed from 2009-2012.

TABLE 12.1
CHECK SAMPLE SUMMARY FOR GOLD

Lexam VG Gold Inc. - Fuller Project		
	2009-2012	
	Primary (ALS)	**Secondary (Expert)**
Number of Samples > DL (N)	180	180
Number of assays removed	1	1
Mean Assay (opt)	0.308	0.322
Maximum Assay (opt)	5.398	5.990
Minimum Assay (opt)	0.003	0.003
Median Assay (opt)	0.019	0.019
Variance	0.676	0.792
Standard Deviation	0.822	0.890
Coefficient of Variation	2.674	2.765
Correlation Coefficient	0.988	
% Difference Between Means	-4.6%	

Gold check assay pairs have good correlation coefficients for both sets of data (0.988) and the percent difference between means is within acceptable limits.

A scatter plot and a quantile-quantile (Q-Q) plot for the gold pulp assay check pairs are shown in Figure 12.5 and Figure 12.6respectively. In general, there is good correlation and no obvious bias in assay results. As expected in a high nugget deposit such as Fuller, low gold grades show poor reproducibility (Figure 12.7).

Figure 12.5 Scatter Plot for Gold check samples for assays <1.0 opt (2009-2012)

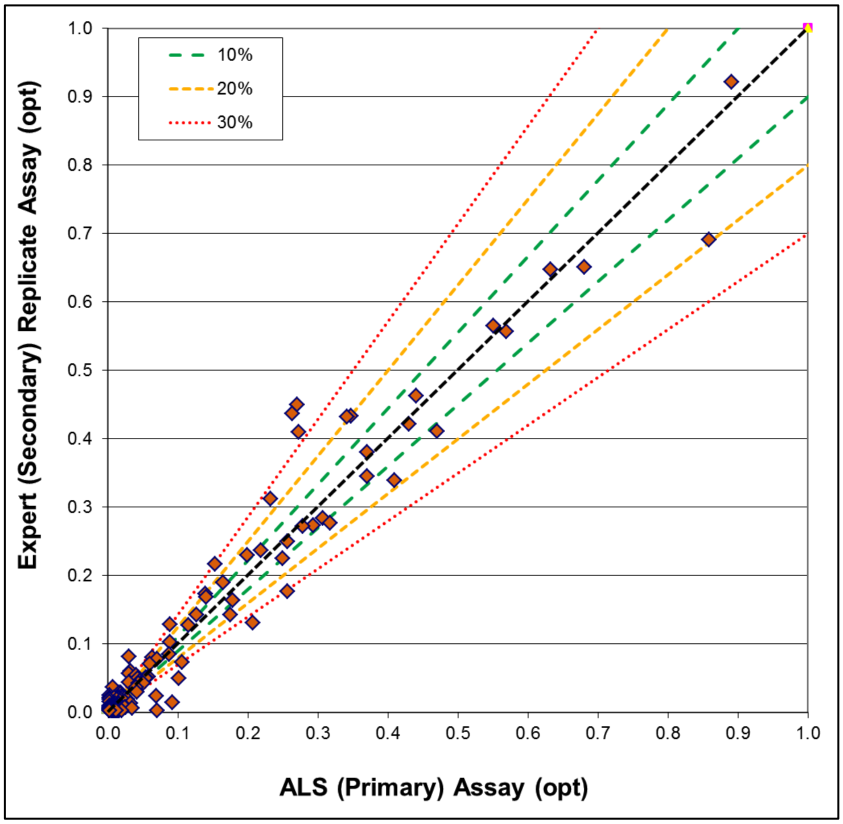


Figure 12.6 Q-Q Plot for Gold Check Samples (2009-2012)

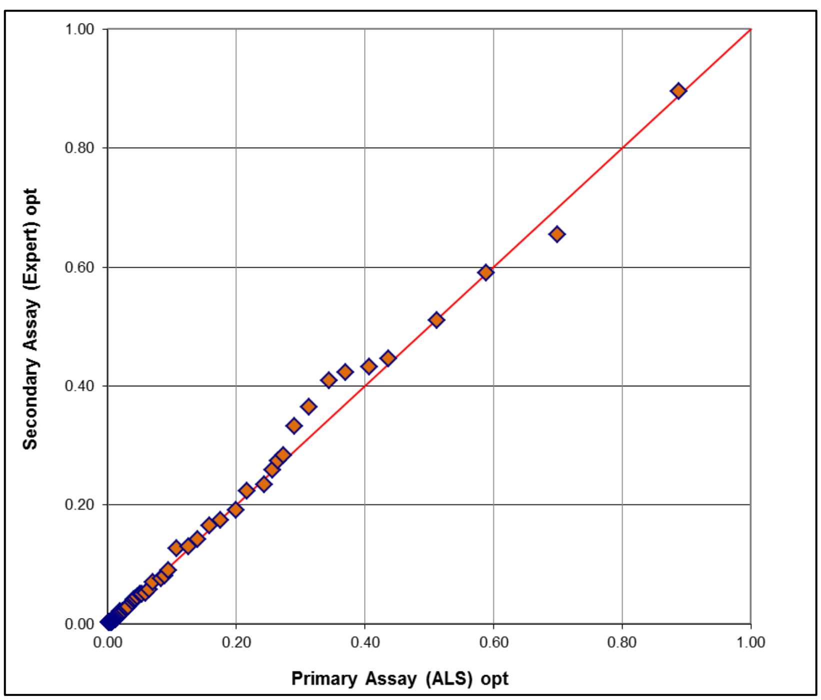
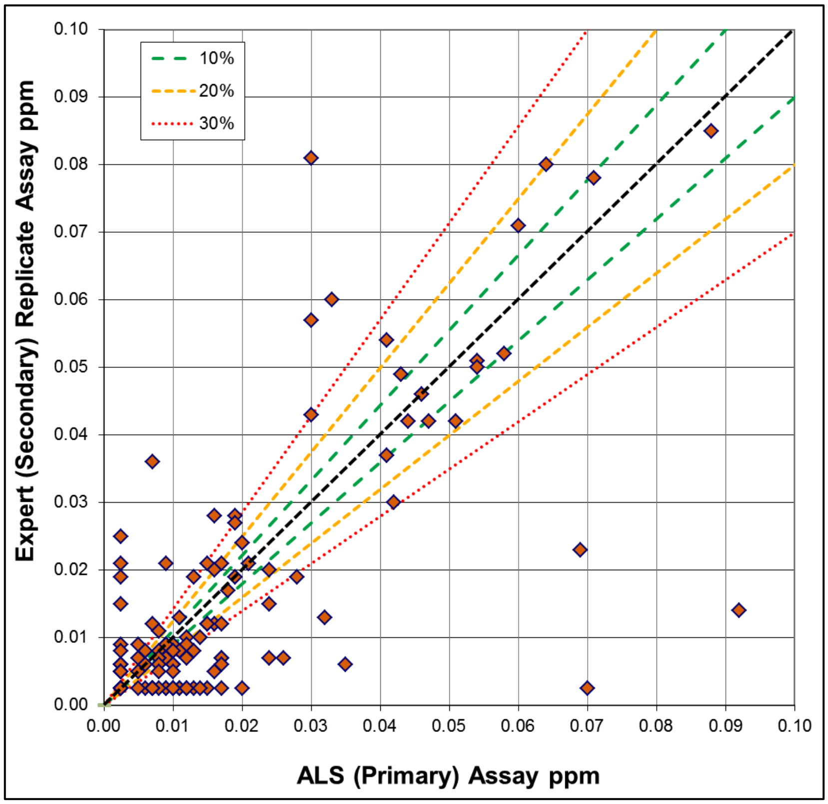


Figure 12.7 Scatter Plot for Gold check samples for assays <0.1 opt (2009-2012)



It is RPA's opinion that Lexam's program of check sampling is rigorous and meets industry standards. Results of the check sampling for the 2009-2012 drilling programs show acceptable correlation between the primary and secondary laboratories for pulps from a high nugget gold deposit.

12.2.6 Enhancements to QA/QC Program

RPA recommended enhancements to Lexam's QA/QC protocol including the regular submission of field and coarse reject duplicate samples to the primary laboratory, the inclusion of at least three different grades of CRMs into the sampling stream to monitor the accuracy of analysis for potentially economic elements, and the routine inclusion of blank samples with each batch submitted for analysis. RPA further recommends that Lexam use an accredited secondary laboratory to assess the assay accuracy of the primary laboratory and implement a QA monitoring system to detect failed batches, and in turn, identify sample batches for reanalysis.

Based on RPA's data verification of the drill hole database, drill core review, and site visit, RPA is of the opinion that the resource database is reliable and accurate and is suitable for Mineral Resource estimation.

12.3 PAYMASTER DEPOSIT

Tudorel Ciuculescu, P.Geo., RPA's Senior Geologist, visited the Paymaster Property site on November 29-30, 2012. During the site visit, RPA inspected the Paymaster Property and located drill collars VGP-10-86 and VGP-10-85. There was no drilling on the Property at the time of the site visit.

RPA carried out a geological core review of lithology, mineralization and sampling, checking against drill logs of the following drill holes: VGP-09-12, VGP-09-21, VGP-09-31, VGP-09-45, VGP-10-74, VGP-10-83, and VGP-10-95. During the core review, no notable discrepancies were found: footage tags were placed in the correct locations in the core boxes, samples were clearly and accurately marked, and core boxes were clearly labelled.

Lexam provided a GEMS project containing drill hole data for the Paymaster Property and adjacent properties. The drilling database contained recent records from 1986 to 2012, as well as older records from 1920s and 1950s.

RPA compared assay results from the older records, related to the No. 2, 3, and 5 shafts, against assays from recent drill holes. The drill holes were composited from collar to toe at fixed five foot intervals. Composites from older drill holes were paired with composites from recent drill holes based on distance between composite midpoints. Pairs situated within zero to three feet and pairs situated within three to six feet showed an average absolute percent difference of more than 30%. Based on the low similarity between the two groups of data, RPA decided to use only the recent drill holes for the resource estimate. The previous resource estimate for the Paymaster Property was also based entirely on the recent drilling.

RPA flagged and selected recent Paymaster holes, drilled from 1986 to 2012, for geological modelling and resource estimate. RPA reviewed the collar locations, downhole deviation surveys, lithology, and assay tables. Visual checks against topography and older maps were made for collars, while the deviation surveys were compared on screen with stope intersects and underground openings crossed by the hole. Tools available in Gemcom GEMS were used for database table validation.

RPA reviewed the resource database that formed the basis of the Mineral Resource estimate presented in this Technical Report. This included results from the quality assurance/quality control (QA/QC) program and assay certificates for drill hole samples by VG Gold Corp. (now Lexam) from 2009 to 2012. Since the previous resource estimate in 2010, 24 new drill holes have been completed and used in the current estimate.

12.3.1 Manual Database Verification

Lexam provided a GEMS project containing drill hole data for the Paymaster Property and adjacent properties. RPA flagged and selected Paymaster drill holes for geological modelling and resource estimation.

RPA reviewed the collar locations, downhole deviation surveys, lithology, and assay tables. Visual checks against topography and older maps were made for collars, while the deviation surveys were compared on screen with stope intersects and underground openings crossed by the hole. Tools available in Gemcom GEMS were used for database table validation.

The assay table verification included comparison of database values against laboratory certificates. Gold assay values from 124 assay certificates for Lexam drilling from 2009 to 2012, representing approximately 25% of the database, were matched and no issues were identified.

12.3.2 Independent Assays of Drill Core

RPA did not collect drill hole core samples for independent assay testing during the site visit. Historical mining in the Paymaster No. 2 and 3 Shaft area confirms that the deposit contains mineralization.

12.3.3 Quality Assurance and Quality Control

Quality assurance (QA) consists of evidence to demonstrate that the assay data has precision and accuracy within generally accepted limits for the sampling and analytical method(s) used in order to have confidence in future resource estimations. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of sampling, preparing, and assaying the exploration drilling samples. In general, QA/QC programs are designed to prevent or detect contamination and allow assaying (analytical) precision (repeatability) and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling–assaying variability of the sampling method itself.

QA/QC protocols include insertion into the sample stream of certified reference materials of various grades, blanks, field sample duplicates, coarse rejects duplicates, and pulp replicates.

For the Paymaster Project QA/QC program, Lexam uses ALS Canada Ltd. (ALS) in Val D'Or, Quebec, an ISO 17025 Accredited Testing Laboratory, as the primary laboratory. Laboratoire Expert Inc. (Expert) in Rouyn-Noranda, Quebec, is used as the secondary laboratory.

RPA reviewed all assay certificates available from 2009-2012 and identified 1,006 ALS internal pulp duplicate pairs. RPA removed 170 pairs that were below detection limit from this selection and reviewed the results of 836 pairs.

12.3.4 Certified Reference Material (Standards)

Results of the regular submission of certified reference materials (CRM or standards) are used to identify problems with specific sample batches and long-term biases associated with the regular assay laboratory. Lexam has not incorporated the use of CRMs into its QA/QC program. Lexam monitors ALS's internal quality control CRM results. RPA reviewed this data and found the results acceptable.

12.3.5 Blanks

Contamination and sample numbering errors are assessed through blank samples, on which the presence of the elements undergoing analysis has been confirmed to be below the corresponding detection limit. A significant level of contamination is identified when the blank sample yields values exceeding three times detection limit of the analyzed element. In order to be effective, the matrix of the blank sample should be similar to the matrix of the material being routinely analyzed and inserted at a rate of 5%, or one per batch of twenty samples. Lexam's QA/QC protocol does not include blank sample insertion in the sample stream.

Lexam monitors ALS's internal quality control blank results. RPA reviewed this data and found the results acceptable.

12.3.6 Duplicates

Field duplicates assess the variability introduced by sampling the same interval. The duplicate splits are bagged separately with separate sample numbers so as to be blind to the sample preparation laboratory. The duplicates contain all levels of sampling and analytical error and are used to calculate field, sample preparation, and analytical precision. They are also a check on possible sample over selection; that is, the sampler has either purposely or inadvertently sampled the geological material (usually drill core) so as to preferentially place visible mineralization in the sample bag going for analysis.

Coarse duplicates (or coarse reject duplicates) are duplicate samples taken immediately after the first crushing and splitting step. The coarse duplicates will inform about the sub-sampling precision. In order to ensure repeatability conditions, both the original and the coarse duplicate samples should be submitted to the same laboratory (primary laboratory), in the same sample batch and under a different sample number, so that pulverization and assaying follow the same procedure.

Pulp duplicates consist of second splits of final prepared pulverized samples, analyzed by the same laboratory as the original samples under different sample numbers. The pulp duplicates are indicators of the analytical precision, which may also be affected by the quality of pulverization and homogenization (i.e., sub-sampling variance introduced during pulp preparation from splits of the original coarse reject material).

One component of ALS's QA/QC program is the regular measurement of laboratory duplicate samples. Lexam has relied on the results of ALS's pulp duplicates to ensure that analytical precision meets project requirements.

Out of a total of 10,564 samples, ALS took second splits of 1,008 final pulps (9.5%) and re-assayed them as pulp duplicates. Results showed an acceptable correlation between the gold pulp duplicates for this type of deposit (Figure 12.8). Thompson-Howarth plots illustrated in Figure 12.9 and Figure 12.10 serve to highlight the high nugget effect of the Paymaster deposit, indicating that near the open pit cut-off grade of 0.015 opt Au. An analytical precision of only ±40% can be expected. RPA has converted all analytical results from parts per million (ppm) to ounces per ton (opt).

Figure 12.8 Paymaster Pulp Duplicate Samples - 2009-2012- ALS 2009-2012 Pulp Duplicate Scatter Plot

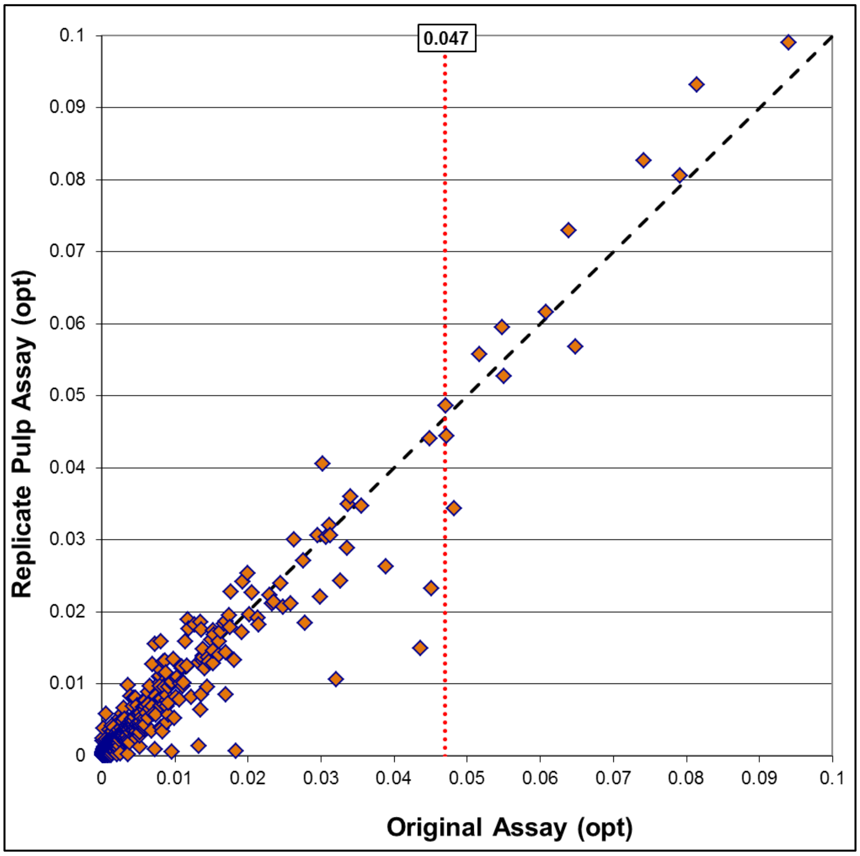


Figure 12.9 Paymaster Pulp Duplicate Samples – 2009-2012 - ALS 2009-2012 Pulp Duplicate Thompson-Howarth Plot

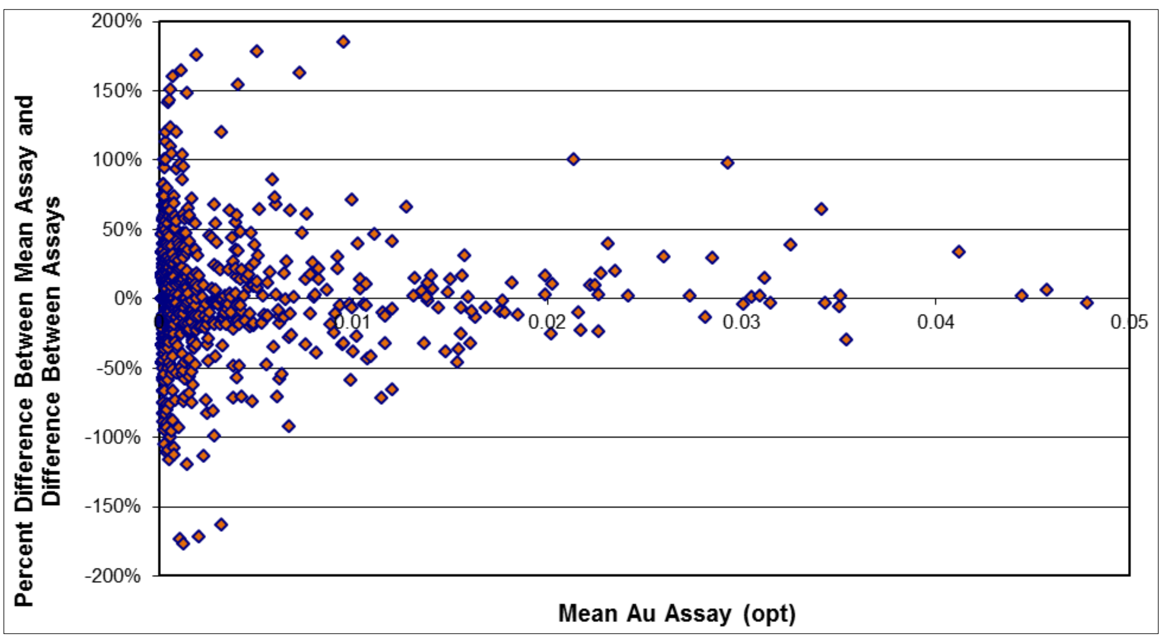
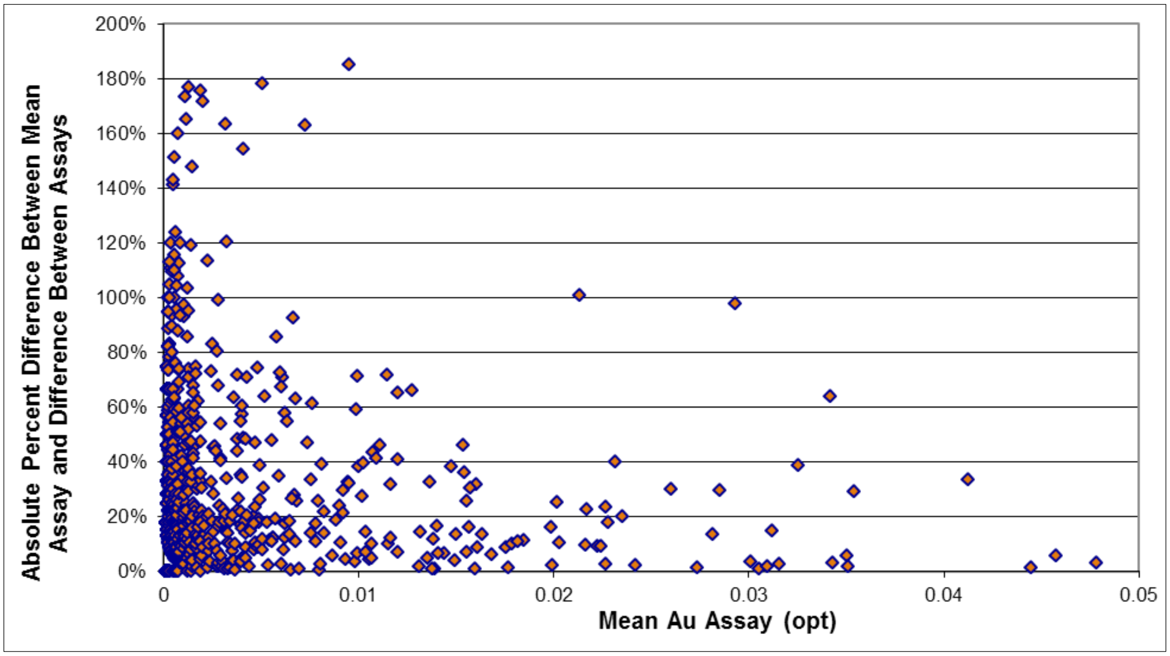


Figure 12.10 ALS 2009-2012 Pulp Duplicate Results Showing Modified Thompson-Howarth Plot Below 0.10 OPT



Lexam has not incorporated field or coarse reject duplicates into the Paymaster Project QA/QC program.

It is RPA's opinion that Lexam should incorporate field and coarse duplicates into the Paymaster QA/QC program. RPA recommends that field and coarse reject duplicates be taken in the future and submitted to the primary laboratory for analysis. If practicable, duplicates should be chosen to represent the range of grades expected and for a better representation of the precision. Field, coarse reject, and pulp duplicates will provide information about the precision at three different stages in the sampling stream.

12.3.7 Check Samples

Check samples consist of second splits of the final prepared pulverized samples routinely analyzed by the primary laboratory (ALS) and resubmitted to a secondary laboratory (Expert) under a different sample number. These samples are used to assess the assay accuracy of the primary laboratory relative to the secondary laboratory.

Lexam's QA/QC protocol calls for check samples to be taken at a rate of one in every ten to twenty samples and submitted to a secondary laboratory. RPA received the results for 425 duplicate pairs, which covered drilling completed from 2009 to 2012. There are no check samples for drilling completed prior to 2009.

Table 12.2 summarizes the statistics for gold check assay pairs. After removing a single outlier pair, the results from the external check assays indicate good correlation for drilling completed from 2009 to 2012.

TABLE 12.2 CHECK SAMPLE SUMMARY FOR GOLD - LEXAM VG GOLD INC. - PAYMASTER PROJECT		
	2009-2012	
	Primary (ALS)	**Secondary (Expert)**
Number of Samples	426	426
Number of assays removed	1	1
Mean Assay (opt)	0.0040	0.0035
Maximum Assay (opt)	0.1770	0.2018
Minimum Assay (opt)	0.0001	0.0001
Median Assay (opt)	0.0007	0.0006
Variance	0.0001	0.0001
Standard Deviation	0.0117	0.0118
Coefficient of Variation	2.9386	3.3686
Correlation Coefficient	0.968	
% Difference Between Means	11.9%	

Summary statistics suggest that gold check assay pairs have good correlation coefficients for both sets of data (0.968) and the percent difference between means is within acceptable limits. It is RPA's opinion, however, that the laboratory check samples did not successfully sample an appropriate grade range for the Paymaster deposit. The mean assay grade of Paymaster check samples is 0.004 opt, whereas the cut-off grade for the Paymaster open pit is 0.015 opt and the average grade of the deposit is 0.047 opt.

A scatter plot and a quantile-quantile (Q-Q) plot for the gold pulp assay check pairs are shown in Figure 12.11 and Figure 12.2 respectively. As expected in a high nugget deposit such as Paymaster, the low gold grades show poor reproducibility (Figure 12.13). Although there is an apparent low bias in the ALS gold assay data (see Figure 12.12), the assay ranges are not representative of the Paymaster deposit.

Figure 12.11 Scatter Plot for Gold check samples for assays <0.3 opt (2009-2012)

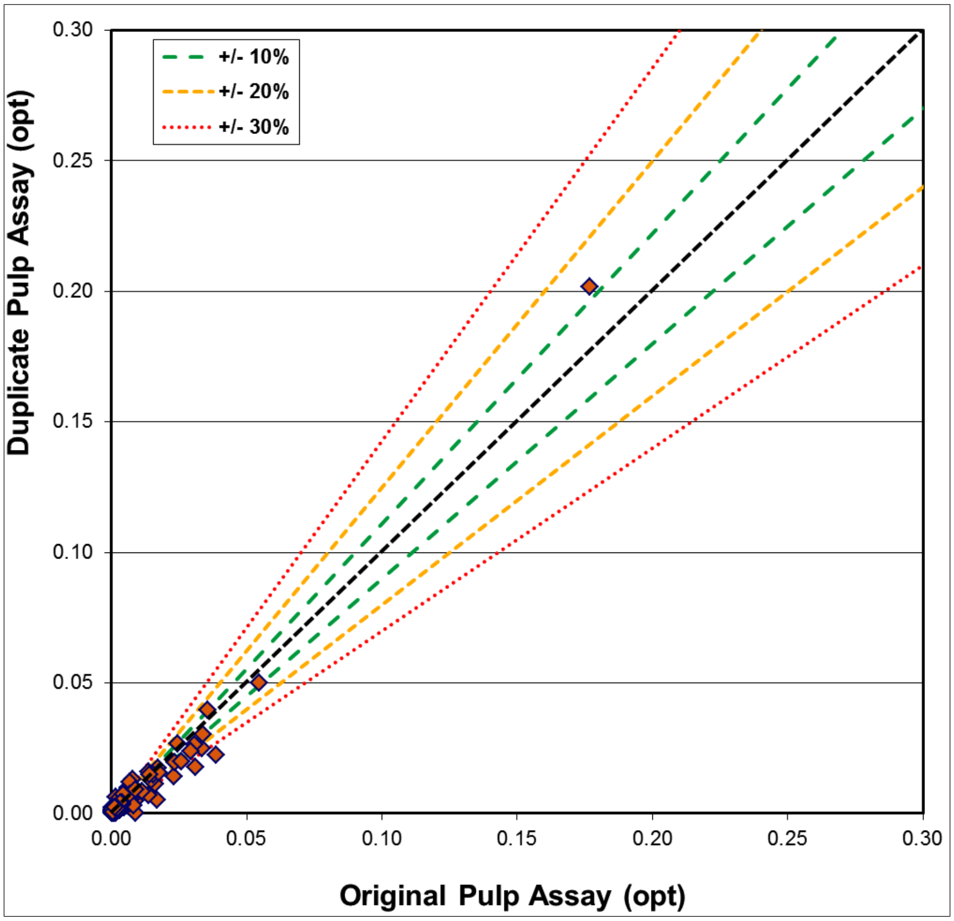


Figure 12.12 Q-Q Plot for Gold Check Samples (2009-2012)

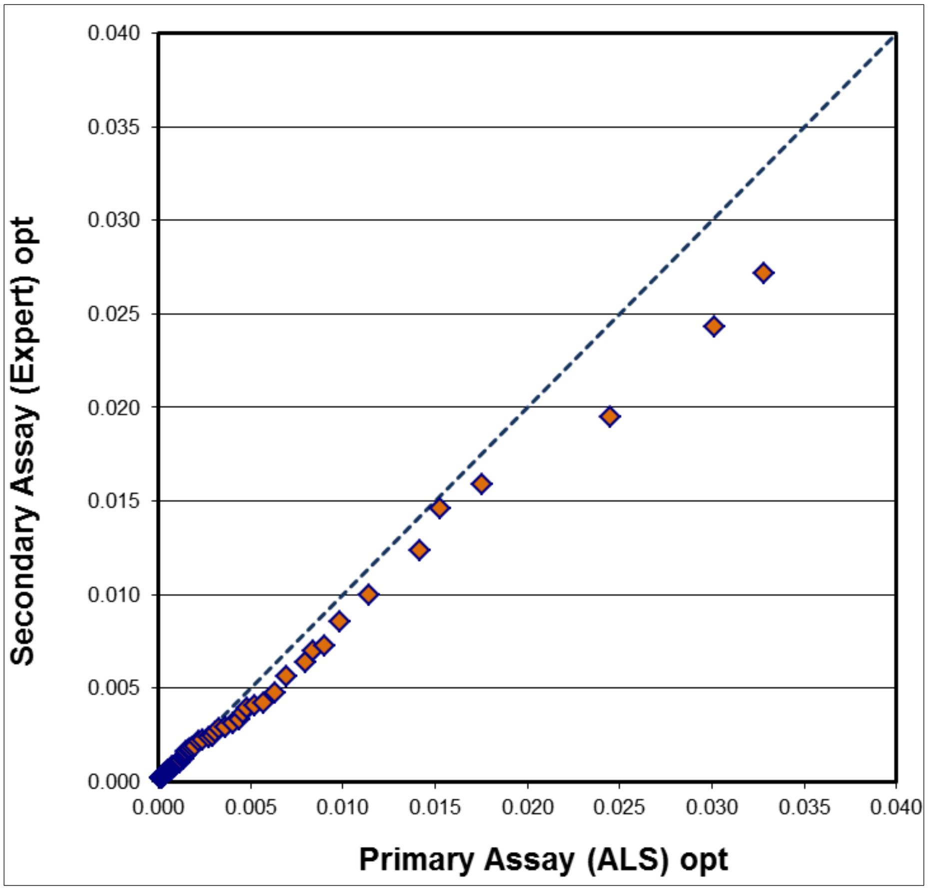
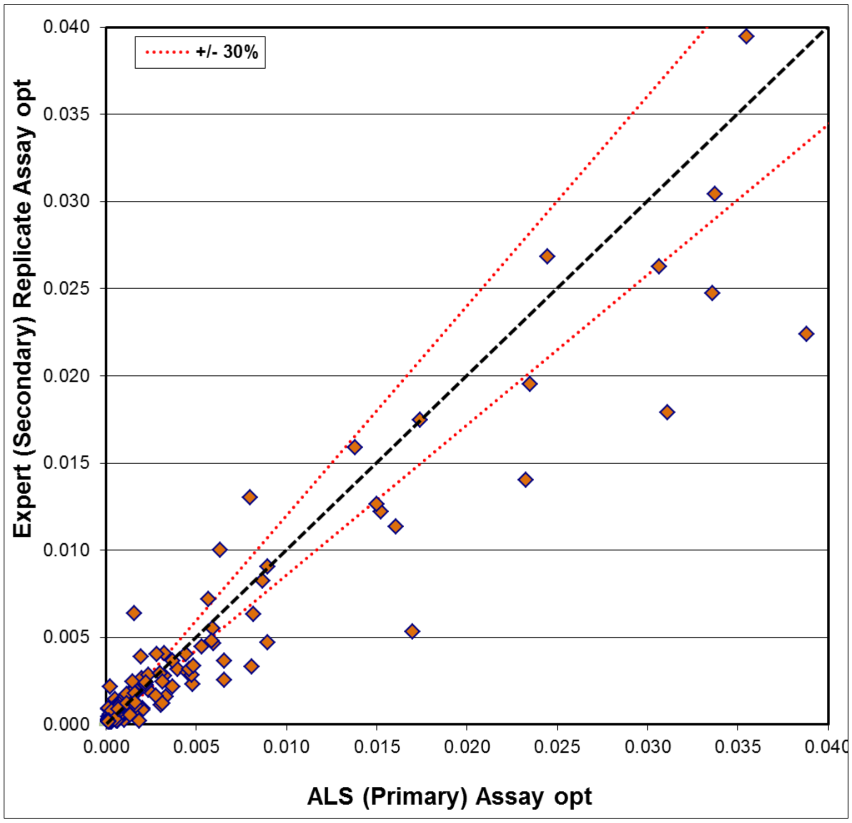


Figure 12.13 Scatter Plot for Gold check samples for assays <0.04 opt(2009-2012)



In RPA's opinion, Lexam's program of check sampling meets industry standards. Results of the check sampling for the 2009-2012 drilling programs show acceptable correlation between the primary and secondary laboratories for pulps from a high nugget gold deposit, however, the grade range is not representative of the Paymaster deposit.

12.3.8 Enhancements to QA/QC Program

RPA recommended enhancements to Lexam's QA/QC protocol including the regular submission of field and coarse reject duplicate samples to the primary laboratory, the inclusion of at least three different grades of CRMs into the sampling stream to monitor the accuracy of analysis for potentially economic elements, and the routine inclusion of blank samples with each batch submitted for analysis. Although Lexam has implemented a fairly rigorous check sample program, it is RPA's opinion that the expected grades of samples should be representative of the deposit. RPA further recommends that Lexam implement a QA monitoring system used to detect failed batches, and in turn, identify sample batches for reanalysis.

Based on RPA's data verification of the drill hole database, drill core review, and site visit, RPA is of the opinion that the resource database is reliable and accurate and is suitable for Mineral Resource estimation.

12.4 DAVIDSON TISDALE

12.4.1 Site Visit and Due Diligence Sampling

The Davidson Tisdale Property was visited by Mr. Antoine Yassa, P.Geo., an independent Qualified Person as defined by National Instrument NI 43-101 Standards of Disclosure for Mineral Projects, on November 6 and 7, 2012. Fifty-three (53) samples were collected from twenty (20) holes by sawing a ¼ split of the half core remaining in the box. The samples were documented, bagged, and sealed with packing tape and were taken by Mr. Yassa to Dicom in Rouyn-Noranda, QC. From there, the samples were shipped to the offices of P&E in Brampton, ON, and sent by courier to AGAT Laboratories in Mississauga, ON for analysis. Gold was analyzed using fire assay on a 30 gram aliquot with an AAS finish. Samples yielding values greater than 10 g/t Au were reassayed and quantitatively determined using the gravimetric method.

AGAT Laboratories employs a quality assurance system to ensure the precision, accuracy and reliability of all results. The best practices have been documented and are consistent with:

- The International Organization for Standardization's ISO/IEC 17025, "General Requirements for the Competence of Testing and Calibration Laboratories' and the ISO 9000 series of Quality Management standards";
- All principles of Total Quality Management (TQM);
- All applicable safety, environmental and legal regulations and guidelines;
- Methodologies published by the ASTM, NIOSH, EPA and other reputable organizations;
- The best practices of other industry leaders.

At no time, prior to the time of sampling, were any employees or other associates of Lexam advised as to the location or identification of any of the samples to be collected.

Lexam did not insert any of their own QC samples, and P&E felt that in order to validate the earlier drill holes in the database, a larger than normal number of due diligence samples would need to be collected. Holes drilled in years 2003 through 2010 were sampled by Mr. Yassa.

Prior to the 2010 drilling, the most recent drill programs on the Davidson Tisdale property were from 2003 to 2005 inclusive, and Laboratoire Expert ("Expert") of Rouyn-Noranda, QC was the principal lab for these earlier drill programs. Expert is registered under ISO 9001:2000 quality standard and participates in the CANMET PTP-MAL Laboratory Proficiency testing.

An attempt was made to procure the lab's internal QC for this drilling, however in spite of the lab inserting their own QC samples, no separate QC reports had been requested by Vedron Gold for the years 2003 to 2005.

A comparison of the P&E independent sample verification results versus the original assay results for gold can be seen in Figure 12.14 and Figure 12.15. Figure 12.14 displays all values received, while Figure 12.15 displays only the lower range of values from 0 to 10 g/t Au in order to reveal detail at the lower grades. The P&E results demonstrate that the results obtained and reported by Lexam were reproducible to a certain degree, however precision is poor due to the difference in sample size and the natural inhomogeneous distribution of gold, (nugget effect) in the deposit.

Figure 12.14 Due Diligence Sample Results for Gold – All Values

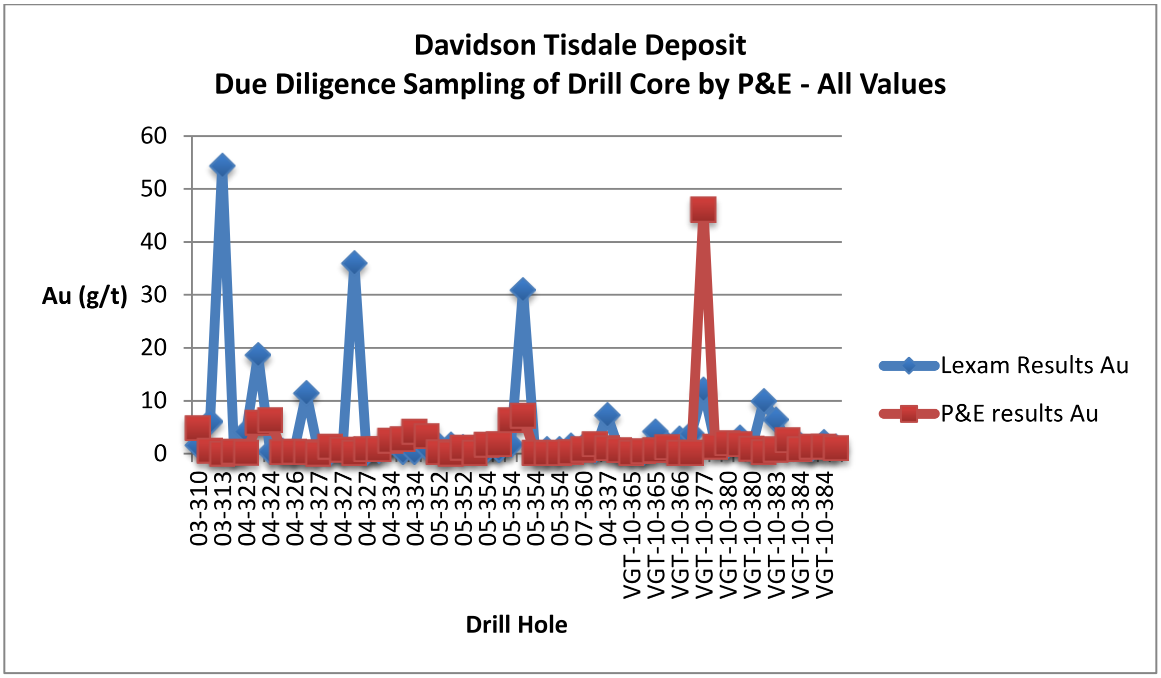
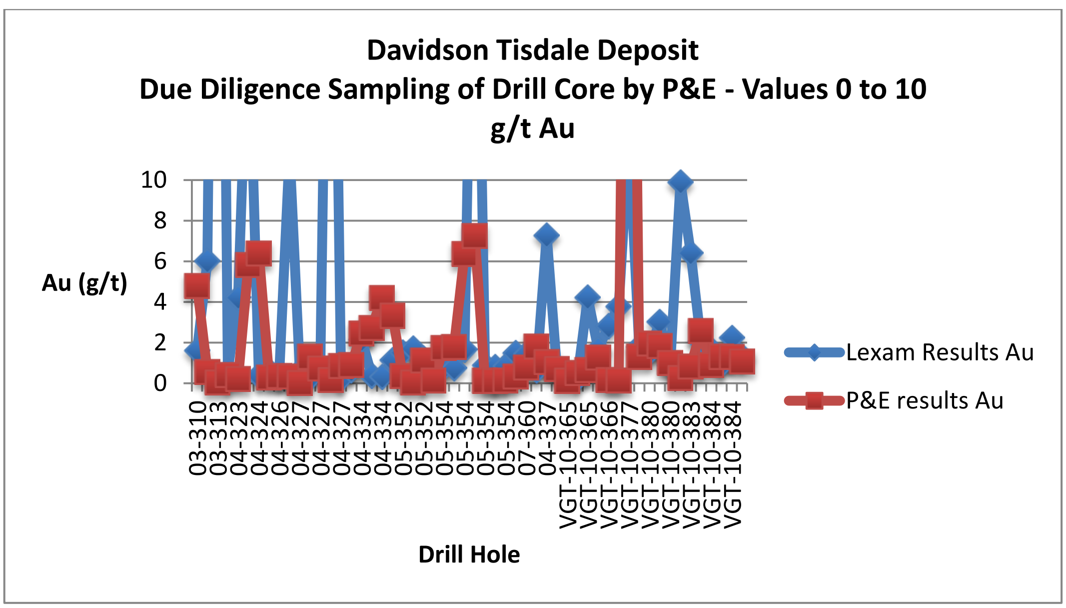


Figure 12.15 Due Diligence Sample Results for Gold – Values 0 to 10 g/t Au



12.4.2 Quality Assurance/Quality Control

Lexam did not implement a quality assurance/quality control ("QA/QC" or "QC") program for any of the drilling, however they did rely upon the Expert (principal lab 2003 to 2005) and ALS (principal lab 2010) internal QC. Each of these labs implemented strict quality assurance/quality control programs. Resulting QC data for 2010 from ALS were obtained by P&E and verified for accuracy, precision and absence of contamination.

12.4.3 Performance of Certified Reference Materials

ALS Minerals Internal Lab QC for Davidson Tisdale – 2010

In 2010 Lexam used ALS as the principal lab. There were seven different certified reference materials used at one point or another during the drill program. All standards were purchased from either Ore Research and Pty in Australia, or Rocklabs in New Zealand.

All seven standards were graphed, using the +/- 2 and +/- 3 standard deviation limits as warning and tolerance limits, respectively. There were 177 standards pertaining to the Davidson Tisdale holes analyzed at ALS during 2010. The standards demonstrated excellent performance, with only two failures outside the tolerance limits.

12.4.4 Performance of Blank Material

ALS Minerals Internal Lab Blanks for Davidson Tisdale -2010

There were 20 blank samples inserted with the samples in 2010 and all of them reported less than three times the detection limit, indicating an absence of contamination at the analytical level.

12.4.5 Duplicate Precision

ALS Minerals Internal Pulp Duplicates for Davidson Tisdale – 2010

There were 233 pulp duplicates prepared at the lab for Davidson Tisdale. A filter of five times the detection limit of 0.005 g/t Au was applied, in order to get rid of values close to detection limit that would falsely influence the pulp duplicate precision. Of the 233 duplicates, only 36 were greater than five times the detection limit of 0.005 g/t Au.

The 36 pairs were plotted on a simple scatter graph, and precision was in the order of 20%, indicative of an inhomogeneity in the pulps. With so few pairs, it was not possible to obtain precision at the resource cut-off grade.

Lexam should concentrate on "fine-tuning" the sampling and assaying protocol, in order to minimize the nugget effect and improve precision at the pulp level. Components of this exercise should include evaluating the collection of a larger sample volume, crushing to a higher percentage passing -10 mesh, and using a 50 gram aliquot for fire assay.

P&E declared the data suitable for use in a mineral resource estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Historical memoranda and reports by Pamour Mines Limited (1988) and are available for the Buffalo Ankerite and Fuller gold deposits and by Lakefield Research (1989) for the Fuller gold deposit. Historical production records in the public domain are of limited value in determination of recovery since most documents report only recovered grade and ounces. No recent work has been completed on mineral processing or metallurgical studies on the mineralization from the property. Based on the historical data, gold recoveries in the range of 90% are anticipated for the Buffalo Ankerite and Fuller mineralization. Due to the lack of available metallurgical testwork data on the Davidson Tisdale and Paymaster Deposits, it was assumed that these metallurgical recoveries would be similar in nature to those of Buffalo Ankerite and Fuller at 90%.

Pamour Mines Limited (1988) completed a number of combined flotation and cyanidation tests at their Metallurgical Laboratories in 1975 and 1988 on samples from the Tisdale Ankerite (Buffalo Ankerite) and Vedron (Fuller) properties. Tests involved grinding, recovery of a flotation rougher concentrate, regrinding, and cyanidation. Optimum recoveries were achieved with grinding to -200 mesh followed by regrinding of the flotation concentrate to -325 mesh. Overall recoveries for three tests averaged 90.9%. Results are tabulated in Table 13.1 below.

TABLE 13.1
METALLURGICAL TESTS BY PAMOUR MINES LIMITED 1975-1988

Test	Head grade (opt)	Flotation		Cyanidation		Overall recovery (%)
		Grind	Recovery (%)	Grind	Recovery (%)	
Tisdale Ankerite* (1975)	0.167	84.7% - 200 mesh	96.5	95.5% - 44 µ	96.3	92.9
Vedron** Hangingwall (1988)	0.762	69.2% - 200 mesh	95.6	95.6% - 325 mesh	97.1	92.8***
Vedron** Main (1988)	0.144	67.8% - 200 mesh	92.6	88.6% - 325 mesh	93.9	87.0

**Tisdale Ankerite (Buffalo Ankerite) **Vedron (Fuller) ***incorrectly reported as 93.8% in the Pamour memorandum*

Lakefield Research (1989) conducted test work on 5 samples of mineralization from the Vedron (Fuller) gold deposit. Four samples had head grades in the range of 4 to 5 g/t Au and one sample graded 50 g/t Au. Direct cyanidation of samples in the 4 to 5 g/t Au range, with grinds of 65% - 200 mesh, showed gold extractions of 85 to 90% after 48 hours. The single higher grade sample showed an extraction of 97.6%. Flotation followed by cyanidation of the flotation cleaner concentrate showed improved cyanidation extraction, but lower overall recoveries due to open circuit cleaning of the rougher concentrate.

14.0 MINERAL RESOURCE ESTIMATES

14.1 P&E 2013 BUFFALO ANKERITE MINERAL RESOURCE ESTIMATE FOR THE NORTH AND SOUTH ZONES

14.1.1 Summary

Mineral Resources for the North and South zones on the Buffalo Ankerite Mine property were estimated based entirely on surface and underground diamond drilling, core sampling and assaying. Portions of both zones were mined in the past and underground chip/channel sampling was carried out historically, however, these data were not available. The resource database is based on Imperial measure, consistent with the drill hole database that was developed during mining to 1953 and retained by subsequent operators on the property. Gold assay grades are in troy ounces per short ton. The drill hole database, which includes holes drilled in the Paymaster and Fuller deposits, contains 5,734 diamond drill holes totalling 1,615,789.77 ft (492,492.72 m). Of these, 4,841 holes for 777,361.45 ft (236,940 m) were collared underground and 893 holes for 838,428.32 ft (255,553 m) were drilled from surface. Resources in the North Zone were intersected by 735 holes totalling 240,418.21 ft (73,279 m) whereas resources in the South Zone were intersected by 692 holes for 241,422.3 ft (73,586 m).

The Mineral Resources for the North and South zones were estimated by conventional 3D computer block modelling using GEMS™ 6.3 and 6.4 mining software (GEMS) by GEMCOM Software International Inc, now Dassault Systèmes. Resources were estimated for open pit and underground mining based on wireframe cut-offs of 0.015 oz/ton Au (0.5 g/t Au) for open pit and 0.045 oz/ton Au (1.5 g/t Au) for underground. Grade interpolation was carried out by inverse distance cubed method (ID^3). Preliminary open pits, with 45° slopes, were designed from the respective zones' resource block models using Whittle™ software. The Indicated and Inferred Resources within the Whittle optimized pits are summarized in Table 14.1. Resources outside the pits are considered as underground Mineral Resources. Table 14.2 summarizes the underground resources for a cut-off grade of 0.075 oz/t Au.

TABLE 14.1
BUFFALO ANKERITE OPEN PIT RESOURCES AT A CUT-OFF GRADE OF 0.015 OZ/TON AU

Zone	Indicated Resources				Inferred Resources			
	Tons (000's)	Au oz/ton	Au Ounces (000's)	Lexam Ounces (000's)	Tons (000's)	Au oz/ton	Au Ounces (000's)	Lexam Ounces (000's)
North	532	0.071	37.6	37.6	198	0.07	13.8	13.8
South	2,622	0.075	197	197	2,707	0.068	183	183
Total	**3,154**	**0.074**	**235**	**235**	**2,905**	**0.068**	**197**	**197**
Paymaster Open Pit Resources at a Cut-Off Grade of 0.015 oz/ton Au								
North	1,702	0.054	92.1	55.3	78.5	0.046	3.74	2.24
South	57.8	0.072	4.15	2.49	113	0.061	6.88	4.13
Total	**1,760**	**0.055**	**96.3**	**57.8**	**192**	**0.055**	**10.6**	**6.37**

**P&E 2013*

Table 14.2 Buffalo Ankerite Underground Resources at a Cut-Off Grade of 0.075 oz/ton Au								
Zone	**Indicated Resource**				**Inferred Resource**			
	Tons (000's)	**Au oz/ton**	**Au Ounces (000's)**	**Lexam Ounces (000's)**	**Tons (000's)**	**Au oz/ton**	**Au Ounces (000's)**	**Lexam Ounces (000's)**
North	1,779	0.149	266	266	1,017	0.122	124	124
South	1,818	0.128	233	233	2,082	0.117	243	243
Total	**3,597**	**0.139**	**499**	**499**	**3,099**	**0.118**	**367**	**367**
Paymaster Underground Resources at a Cut-Off Grade of 0.075 oz/ton Au								
North	19.8	0.100	1.97	1.18	1.78	0.079	0.140	0.0840
South	-	-	-	-	0.141	0.117	0.017	0.0102
Total	**19.8**	**0.100**	**1.97**	**1.18**	**1.92**	**0.082**	**0.157**	**0.0942**

**P&E 2013*

14.1.2 Drill Hole Database

The drill hole database, which includes holes drilled in the Paymaster and Fuller deposits, contains 5,734 diamond drill holes totalling 1,615,789.77 ft (492,492.72 m). Of these, 4,841 holes for 777,361.45 ft (236,940 m) were collared underground and 893 holes for 838,428.32 ft (255,553 m) were drilled from surface (Figure 14.1).

Resources in the North Zone were intersected by 735 holes (plus one wedge hole) totalling 240,418.21 ft (73,279 m) whereas the South zones were intersected by 692 holes for 241,422.3 ft (73,586 m). Some 600 holes in the North Zone were drilled underground and 511 were collared underground for the South Zone. The balance of 135 holes in the North Zone and 181 holes in the South Zone were drilled from surface.

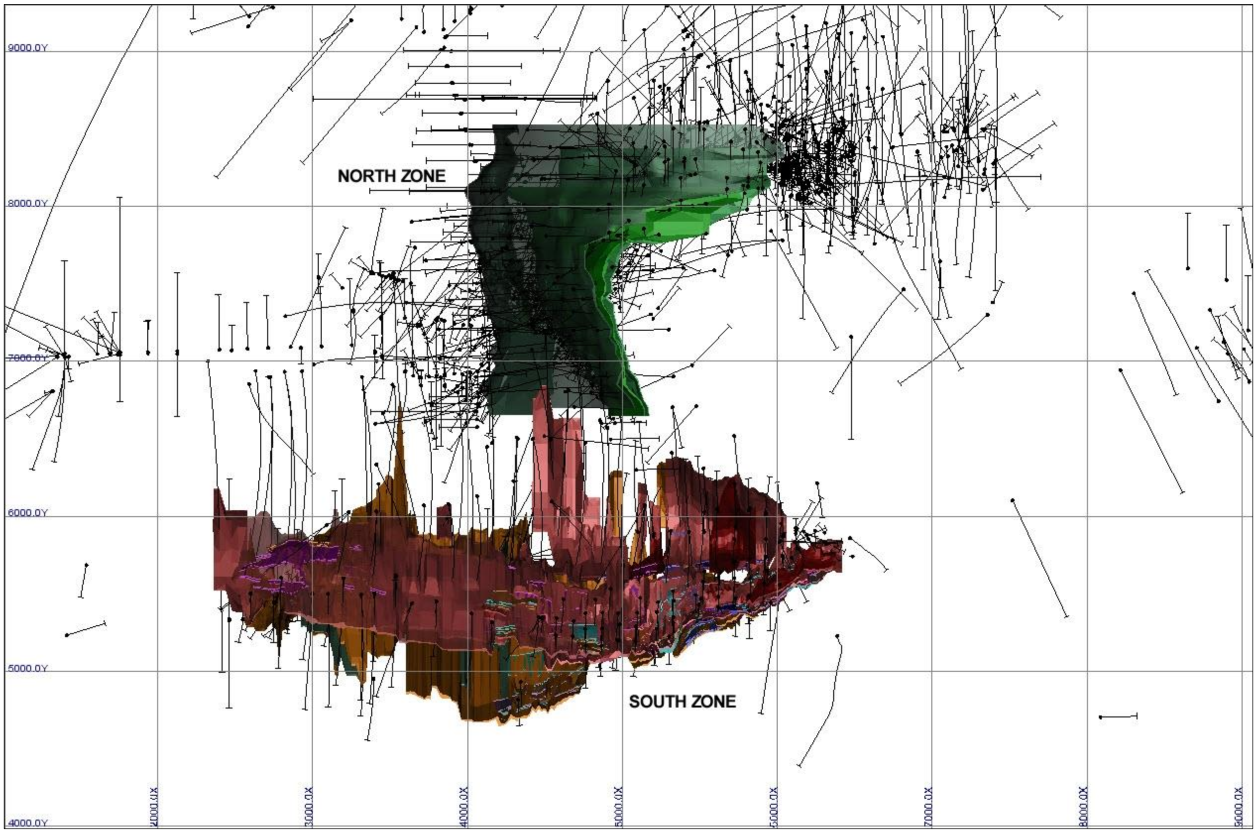
Surface drilling on the South Zone is generally on 100 ft sections and at variable pierce points along dip with intercepts wider at depth in part due to fanned drilling on section.

The core sampling interval, nominally at five feet, is appropriate to the deposit scale and mineralization continuity.

14.1.2.1 Down Hole Deviation Surveys

P&E examined the down hole surveys for the resource holes in the North and South zones and notes that 78% and 74% of the holes, respectively, were not surveyed and these mostly underground holes account for 44% and 25% respectively, of the drilled footage (Table 14.3). Of the surface and underground holes surveyed, high to implausible deviation readings were found in 32% of the North Zone holes and 45% of the South Zone holes. P&E recommends further review of the down hole surveys for the surface drill holes and to consider discarding some of the high to implausible deviation readings where practicable. The quality of the drill hole database has to be taken into account when assessing the risk in resource estimation and in the classification of the resources.

Figure 14.1 Drill Hole Location Plan and Mineral Zone Wireframes



Note: *Scale in feet*

TABLE 14.3 SUMMARY OF DOWN HOLE DEVIATION SURVEYS		
	North Zone	**South Zone**
Number of resource holes in survey file	734	756
Number of records	1,481	2,000
Total length drilled (ft)	240,248.2	275,267.7
Number of unsurveyed holes	570	556
Total length of unsurveyed holes (ft)	105,220.0	69,629.3
Percent of unsurveyed holes	78%	74%
Percent by length of unsurveyed holes	44%	25%
Number of surveyed holes with no azimuth change	1	1
Number of surveyed holes with no dip change	7	2
Number of azimuth readings >5°/100 ft	52	108
Number of dip readings >5°/100 ft	25	15
Number of azimuth readings >10°/100 ft	19	34
Number of dip readings >10°/100 ft	8	6
Number of holes with one or more readings >5°/100 ft	53	91
Number of holes with one or more readings >10°/100 ft	18	33

14.1.3 Assay Database

The Buffalo Ankerite resource estimate is based on some 6,100 assays over 21,705.28 ft (6,629.48 m) in the North Zone and 5,073 assays over 17,546.75 ft (5,348.25 m) in the South Zone.

P&E conducted verification of the drill hole assay database by comparison of the database entries with the assay certificates. The assay certificates were obtained in digital format directly from the assay laboratories and compiled. Both the North and South zones extend onto the Paymaster property consequently P&E also checked assays of Paymaster drill core.

54,077 drill hole database assay results entries were checked against certificates for Buffalo Ankerite and Paymaster assaying representing approximately 52% of the overall Buffalo Ankerite- Paymaster-Fuller assay database. Table 14.3 summarizes the results of database versus assay certificate verification. 11,885 matches of the Buffalo Ankerite database sample numbers with certificates were found. Of these, 3,188 values at less than the detection limit (LDL) of 0.005 g/t Au were entered as 0.003 g/t and 0.0001 oz/ton. One value was entered as 0 g/t and 0.0008 oz/ton with no impact on resource estimation. 8,636 matches for gold entries were confirmed and together with the LDL entries, this represents 99.5% of the sample number matches. Some 8,767 assays used in the resource estimate are from older drill holes and have no sample numbers available for verification.

Some 61 database entries differed from the certificates and of these 5 were higher than the certificates and 56 were lower. 53 of these database values were 0 g/t or 0.0008 to 0.0009 oz/ton. P&E notes that the treatment of LDL values in the database is variable; from one half the LDL to set values, larger or smaller than the certificate LDL's. Some 638 of the assays checked are included in the resource estimate.

Significant differences (>0.01 g/t) were found for 33 assays of which two are included in the resource estimate for the North Zone. These assays (94682 and 94690) are 12.55 g/t Au and 11.7 g/t Au in the certificates but zero in the drill hole database. Both assays are in hole VBA-12-196. This discrepancy should be further investigated.

Certificates for alternate laboratory LabExpert were also examined for assays reported in ounces per ton or in grams per tonne.

Lab Expert Certificates (oz/T)

Some 2,796 database entries reported as oz/ton were paired with certificates. Some 84 entries are also reported in g/t and have discrepancies versus certificates. The database entry exceeds the certificate value for 1,547 assays of which 1,524 are for LDL that is set at a higher value in the database. Some 290 assays in this portion of the database exceed the open pit cut-off grade of 0.015 oz/ton Au. 24 resource assays were found that differ >0.01 oz/ton with respect to the certificates. Of these, three samples were significantly lower in the database and should be checked.

Table 14.4 LabExpert Assays to Check			
Hole ID	**Sample#**	**DB oz/T Au**	**Certificate oz/T Au**
S-35_VED	4492	0.04	0.6785
VBA-06-23	8761	0.034	0.3400
VBA-08-114	33820	0.0008	0.3580

LabExpert Certificates (g/t)

106 matches were found for assays reported in g/t, most of which are low grade. Of these, 22 of the database entries exceed the certificate value, 82 are less than the certificate value and 92 appear to be LDL entries. Only 7 database entries exceed the pit cut-off of 0.5 g/t Au. Some 24 assays differ by >0.01 g/t and are included in the resource estimate. Of these there a number exceeding 1 g/t ranging up to 6.5 g/t (versus zero in the database) for sample 21850 in VBA-05-02.

Paymaster Certificates (g/t)

13,777 database entries were checked. Of these 13,397 were LDL and correctly reflected in the database but differences were noted for 380 non LDL assays. Some 364 certificate values were greater than the database entries for which 361 were 0 g/t. The latter also had entries for gold in oz/ton, however, they are inconsistent with the certificates and other assaying for these samples may have been involved. None of the assays with discrepancies were used in the resource estimate.

Table 14.5 Results of Database versus Assay Certificate Verification for Buffalo Ankerite									
Matches	**DB>**	**DB<**	**DB=**	**DB=0**	**DB≥COG**	**Domains**	**Significant Δ**	**Signif. Δ in Domains**	**LDL**
BA (g/t) n=11,885	5	56	11,824	53	996	638	33	2	3,188
LabExpert (oz/T) n=2,796	23[1]	186	1,063	40	290	341	68	32	1,524
LabExpert (g/t) n=106	22	82	2	77	7	29	68	24	92
Paymaster (g/t) n= 380	16	364	13,397	361	12	0	292	0	13,397

Notes:

(1) excludes 1,524 LDL values in the database that are slightly higher than the certificates.

In summary of the above work, P&E notes that:

- There are minor inconsistencies in the treatment of values below detection limit
- There are a small percentage of low grade assays within the resource domains that differ with respect to low grade certificate results
- The few high certificate values entered as zeros in the database likely result in some conservatism in localized resource block values
- The above items have little impact on the resource estimate.

P&E concludes that the assay database is acceptable for resource estimation.

14.1.3.1 Interpretation, Wireframes and Cut-Offs

The geologic interpretation of the mineralized structures or zones was guided by drill hole logging of lithology and gold-bearing mineralization, including quartz-tourmaline-carbonate-pyrite breccias, quartz veining, mineralized quartz-feldspar porphyries and shearing/alteration of volcanic rocks as well as by gold grades. The contact between metavolcanic host rocks and komatiites also guided interpretation of the zones particularly around the nose of the Kayorum syncline in the South Zone. The locations of stopes and drifts underground, voids logged in drilling and existing open pits at surface, were also important guides to interpreting the trend of the mineralized structures. Preliminary wireframes for mineralization were provided by AGP Mining Consultants Inc. (AGP), Lexam's consultant, but these were extensively modified by P&E based on the mine workings. Whilst the stopes are almost certainly in mineralization, the interpretation of vein trends based on stoping is not always unambiguous since mining locally appears to have changed from following main structures to splays, or vice-versa. In addition, the 3D spatial location of gold mineralized intervals and voids, as logged in drill holes is determined by down hole deviation surveys which for deeper holes, introduces some positioning error with respect to the surveyed mine openings.

Mineral wireframes were constructed based on a cut-off grade for open pit resources of 0.015 oz/ton Au (0.5 g/t Au). Gold price and costs used to determine the cut-off grades are listed in Table 14.6. The underground resource cut-off for wireframing was 0.045 oz/ton Au (1.5 g/t Au). The open pit cut-off was applied from surface to the 10,000 ft elevation at a depth of approximately 1,000 ft as the maximum depth expected for an open pit. Below the 10,000 ft elevation, the underground cut-off was applied to mineral zone wireframing. A minimum horizontal mining width, at a nominal 6 ft (1.8 m), was used in wireframing, however, this was relaxed where necessary to preserve zone continuity or for underground drilling where assaying in the zones is not continuous. The outline of the stopes and drifts on cross section generally determined the zone widths except where drilling indicated larger widths. Since the stopes are likely somewhat wider than the original mineralized zones mined, using the outlines accounts for the expectation of somewhat wider zones occurring due to the use of lower cut-off grades than employed during past mining.

Table 14.6 Determination of Wireframe Cut-off Grade	
Gold Price	$US1600.00/oz
Underground Mining Cost	$C46/tonne
Open Pit Ore/Waste Mining Cost	$C1.85/tonne
Overburden Stripping Cost	$C1.35/tonne
G&A	$C5.00/tonne
Process Cost	$C18.00/tonne
Process Recovery	90%

North Zone

The North Zone has been subdivided geologically into four gold mineralized zones for the purpose of resource estimation (Figure 14.2). The North Zone has been tested by surface and underground drilling. East-west vertical cross sections were generated at 100 ft intervals in GEMS consistent with the surface and underground drill holes section spacing, the latter also drilled on 150 ft (50 m) level spacing. The wireframes were developed on these sections from polylines enclosing drill hole core samples with assays at or exceeding open pit or underground cut-off grades.

The North Zone wireframes occupy and area extending on NS strike up to 1,865 ft (568 m) in length and in EW surface projection for up to 1,990 ft (607 m) including the near surface roll of the zones to the east. The wireframes extend from bedrock surface variously to elevation 7,300 ft, a depth up to 3,737 ft (1,139 m). Total volume of the South Zone wireframes is 200.2 million ft^3 or approximately 17.8 million tons at a bulk density of 0.0888 t/ft^3. Figure 14.3 shows a plan view of the South Zone mineral domain wireframes.

The mineralized structures and the wireframed mineral domains are reasonably continuous section to section. One domain is continuous to depth whereas two of the domains are composed of upper and lower lenses. The fourth domain is not well defined by drilling and dips moderately west, possibly a splay off the more steeply west dipping (-70° to -75°) North Zones.

North Zone grades exceeding cut-off are reasonably continuous, however to preserve zone continuity locally, some low grade material and non-assayed intervals were incorporated as internal dilution. This dilution accounts for approximately 37% of the global wireframe material.

Figure 14.2 3D Perspective Views of the North Zone Resource Wireframes (Looking NNW)

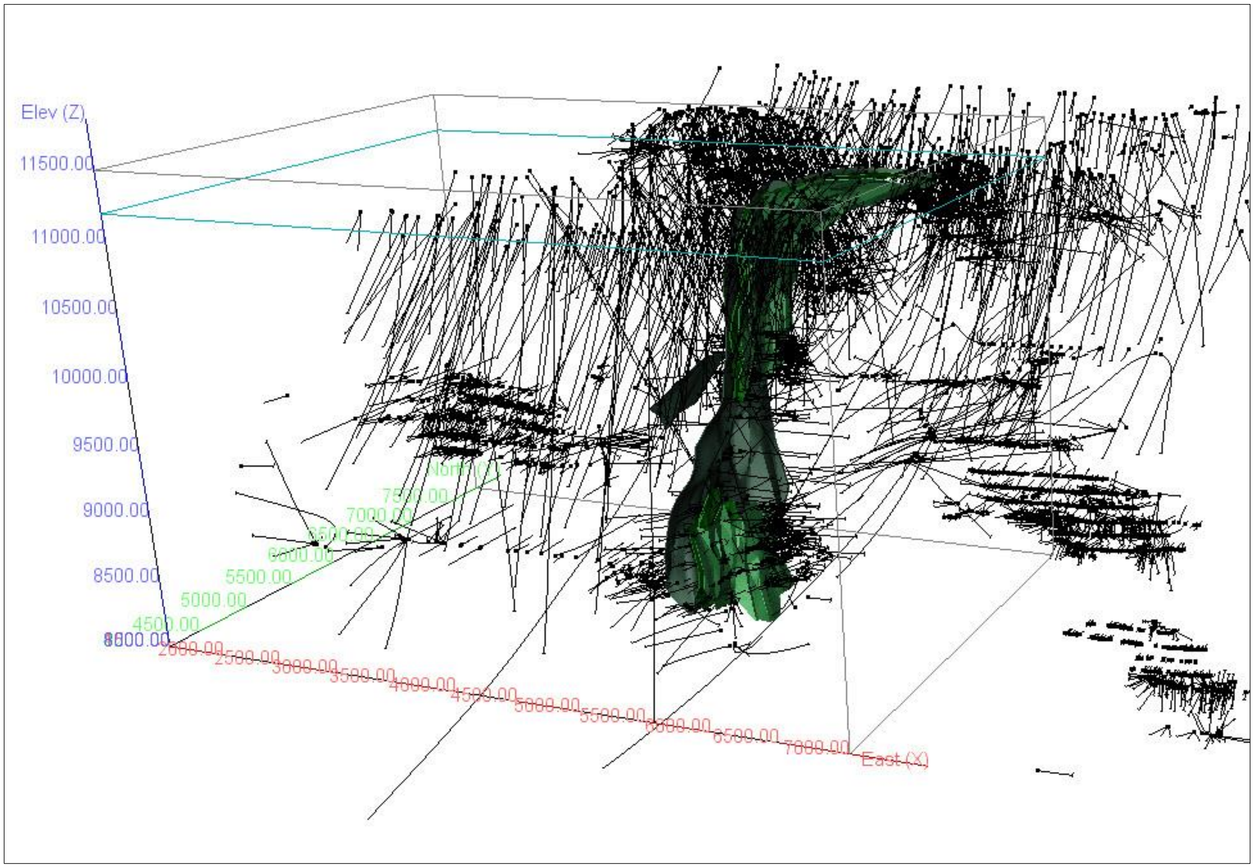
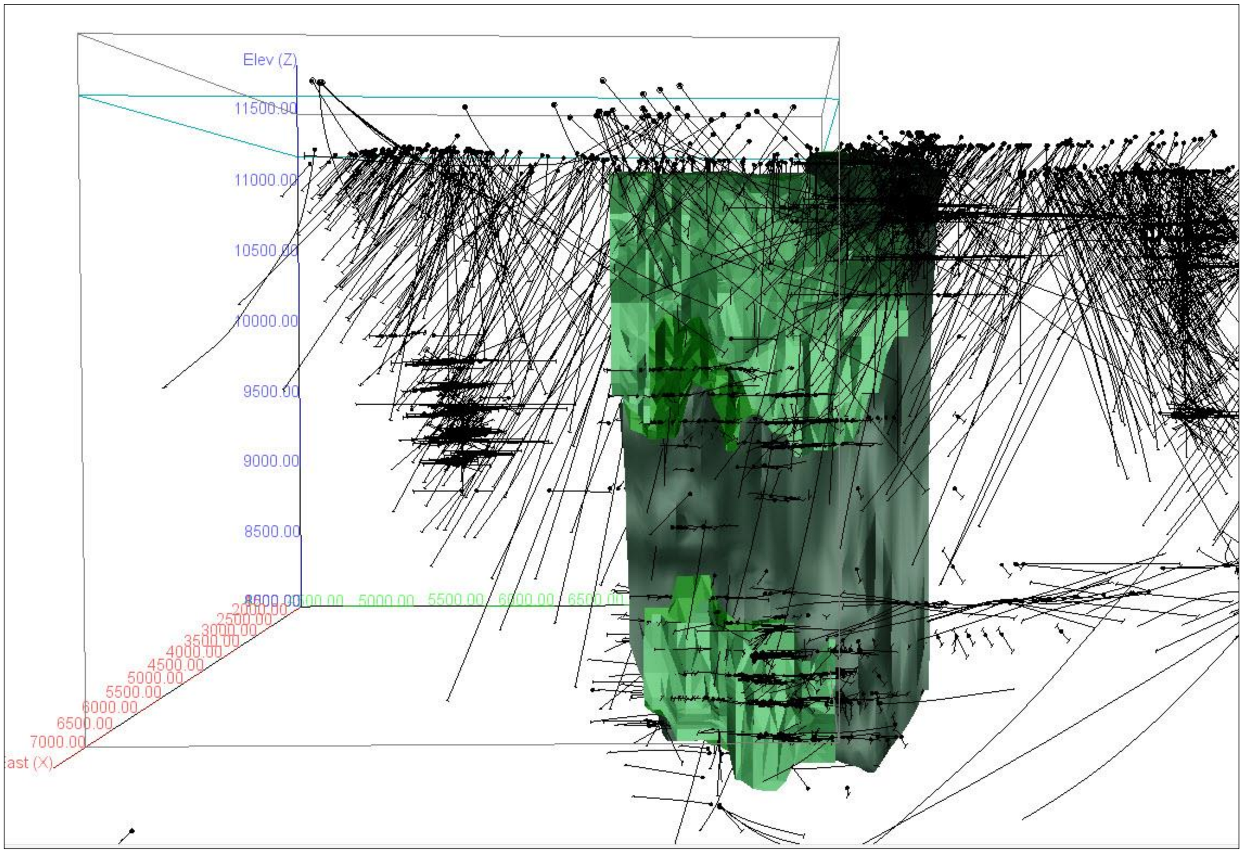


Figure 14.3 3D Perspective Views of the North Zone Resource Wireframes (Looking West)



Note: *Scale in feet*

South Zone

The South Zone has been subdivided geologically into 27 gold mineralized domains for the purpose of resource estimation. Some of the shallow depth domains have open pit resources only.

Above the 10,000 ft elevation, drilling is entirely from surface (200 exploration holes) whereas both underground and surface drilling tests the underground portion of the South Zone.

North-south vertical cross sections were generated at 100 ft intervals in GEMS consistent with the surface drill holes section spacing and filled in to 20 ft sections consistent with the detailed underground drilling and to follow the stopes and mine workings. The wireframes were developed on these sections from polylines enclosing drill hole core samples with assays at open pit or underground cut-off grades.

The South Zone wireframes extend within an area of EW strike up to 4,040 ft (1,231 m) in length and in NS surface projection for up to 2,148 ft (655 m). The wireframes extend from bedrock surface variously to the 8,069 ft elevation, a depth up to 4,716 ft (1,437 m). Total volume of the South Zone wireframes is 276.8 million ft^3 or approximately 24.6 million tons at a bulk density of 0.0888 t/ft^3. Figure 14.4 shows 3D perspective views of the South Zone mineral domain wireframes.

The mineralized structures are stacked and quite continuous section to section with the wireframed mineral domains reasonably continuous at the open pit and underground cut-off grades. Where necessary to preserve zone continuity, however, some low grade material and non-assayed intervals were incorporated as internal dilution. This dilution accounts for approximately 21% of the global wireframe material.

Figure 14.4 3D Perspective Views of the South Zone Resource Wireframes (Looking N)

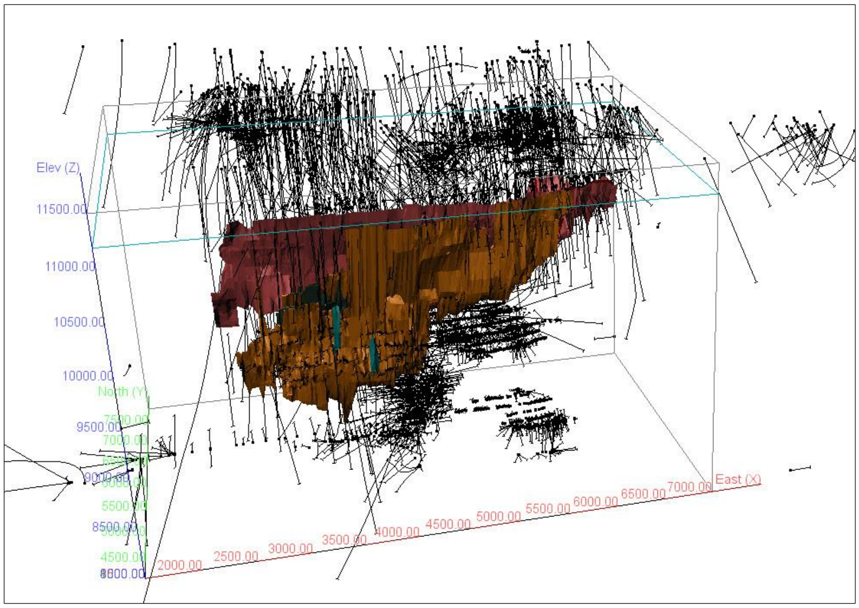


Figure 14.5 3D Perspective Views of the South Zone Resource Wireframes (Looking W)

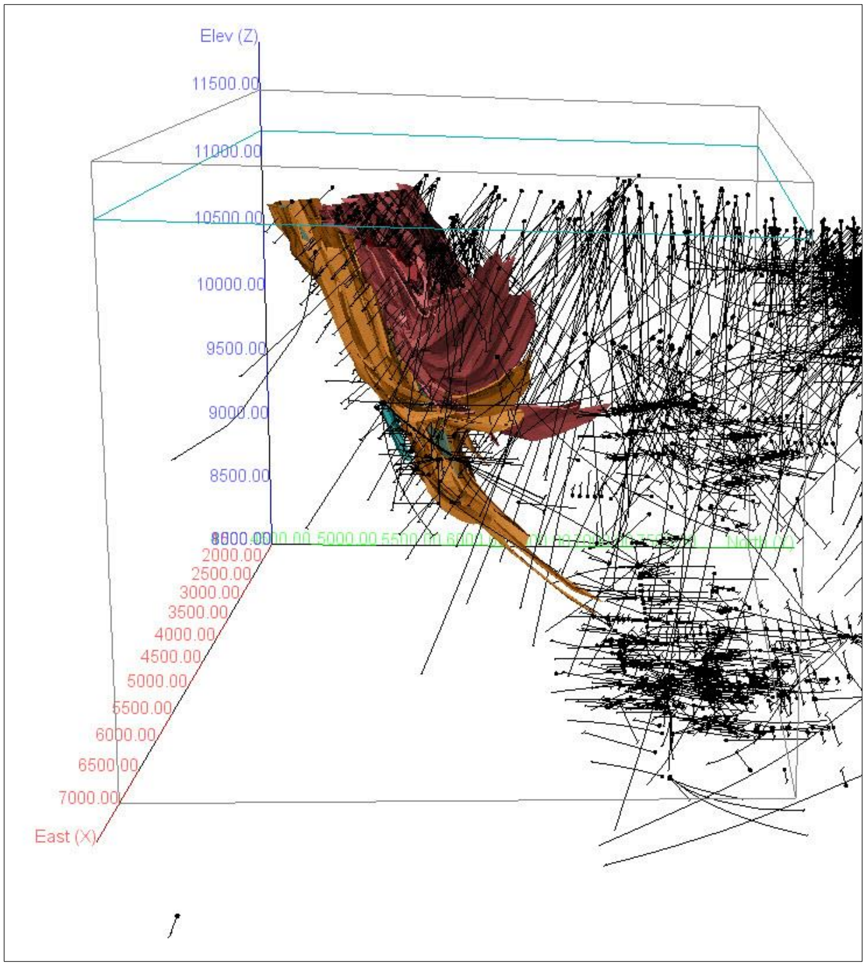
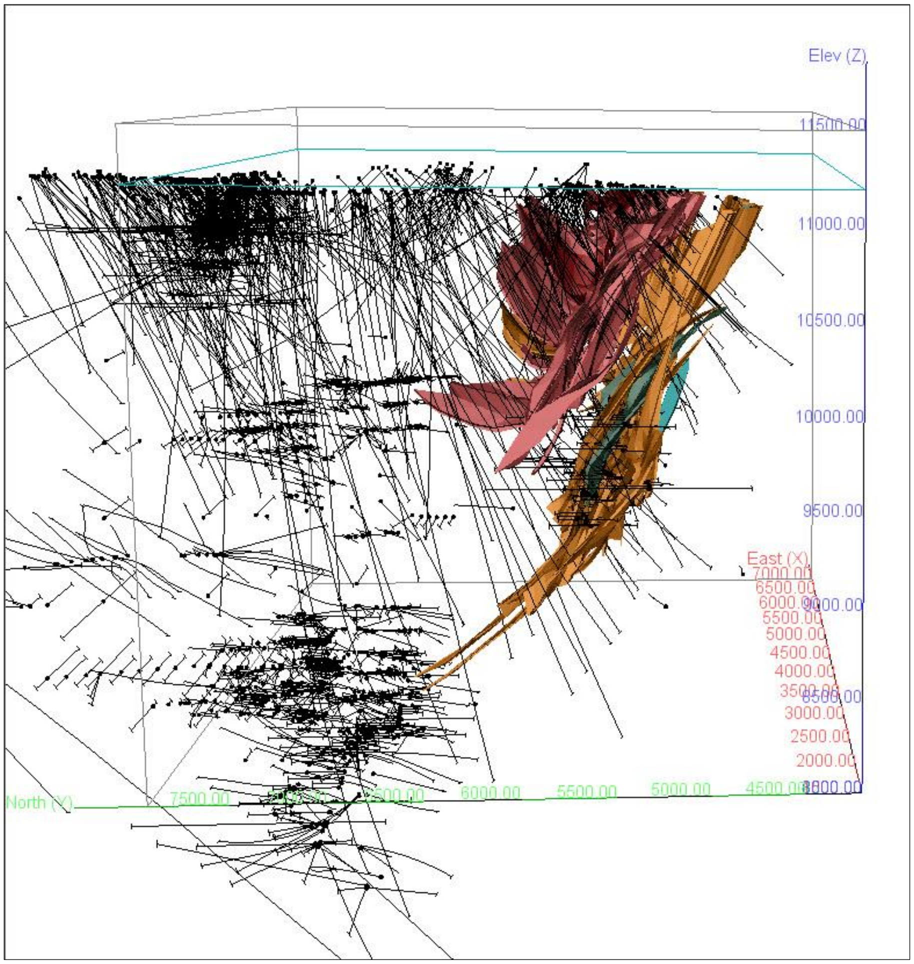


Figure 14.6 3D Perspective Views of the South Zone Resource Wireframes (Looking E)



Note: *Scale in feet*

<u>Workings Wireframes</u>

Wireframes for mine stopes were constructed by Lexam VG and AGP digitized outlines on level plans and cross sections at 1:120 and 1:240 scale and using various methods of polylines to solids and clipping techniques. Access and development workings (shafts, raises, ramps and drifts) were also digitized. Some 145 stope models were built for the North Zone and 181 stopes for the South Zone.

P&E consolidated the stopes into one model for each zone to facilitate determination of the mined volumes and for building block models of the mined material. Several stopes were invalid and could not be integrated with the consolidated stope solids. P&E rebuilt these stopes and united them with the main stope solids. Five to six stopes for both zones required minor location shifts (fraction of a foot) to allow union with the stope solids as they were progressively built. Access workings and drifts/levels in mineralization were also combined into single solids for each zone.

Figure 14.7 3D Perspective View of North Zone Modeled Stopes and Drifts (Looking WSW)

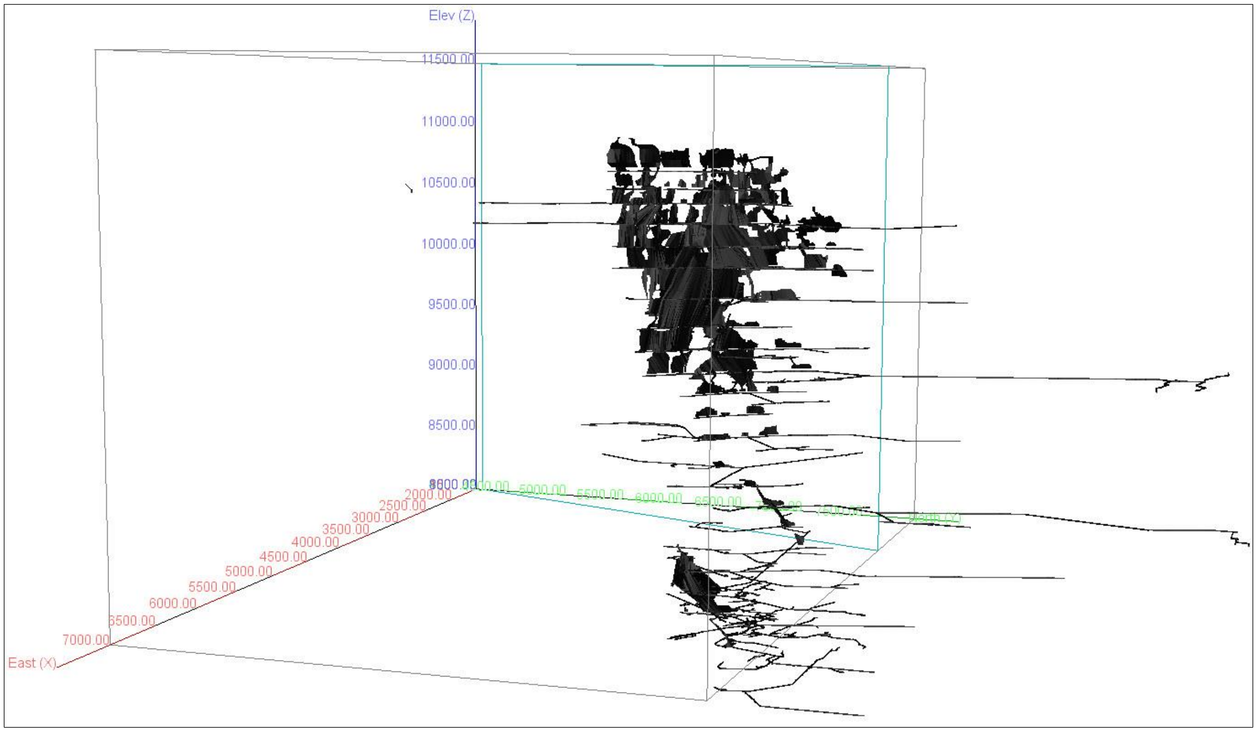
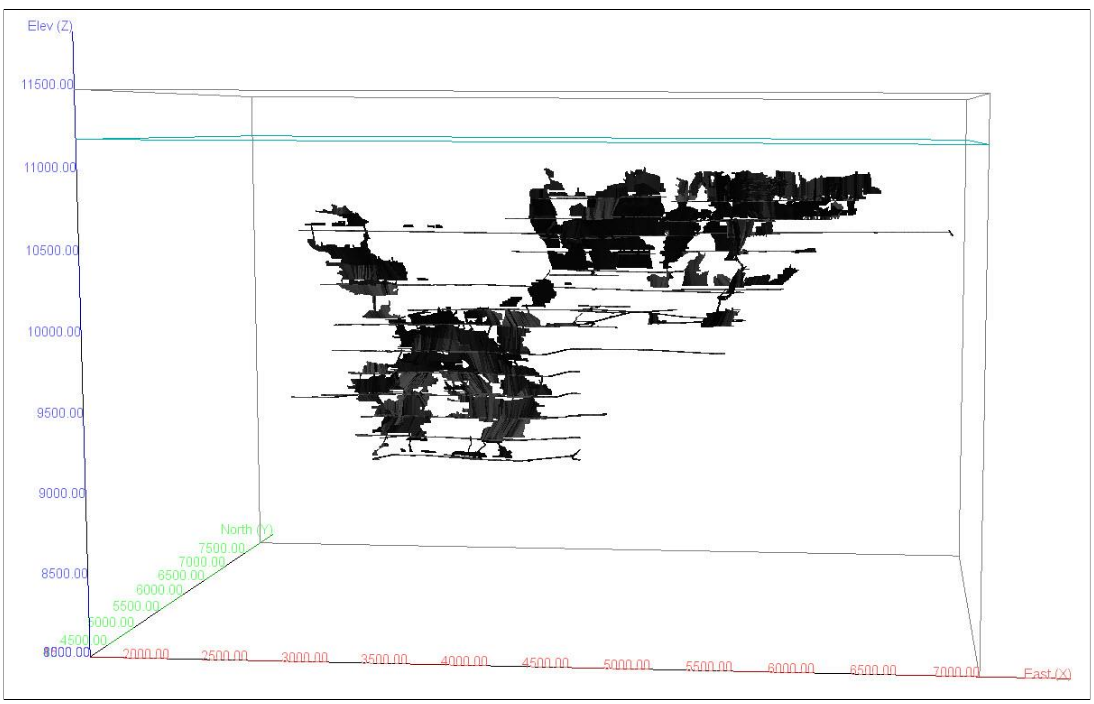


Figure 14.8 3D Perspective View of South Zone Modeled Stopes and Drifts (Looking N)



Note: *Scale in feet*

Surfaces

Excavation surfaces for topography, and the overburden-bedrock contact, were provided by Lexam VG and AGP. These surfaces cover both the North Zone and South Zone block models. The bedrock surface was used to "clip" the mineral wireframes where necessary and as an upper limit for reporting resources from the block model.

Wireframe Solids Volumetrics

The volumes of the various solids used for resource estimation are listed in Table 14.7 to Table 14.10. P&E cautions that the tabulated volumes of the mineral domain solids include the volumes for mined out stopes and drifts that are later removed for the purpose of reporting resources.

TABLE 14.7
NORTH ZONE DOMAIN'S VOLUMES

Wireframe Solid			Domain	Model Code	Volume (ft^3)	Tons[1]	Tonnes
Name 1	**Name 2**	**Name 3**			**(000's)**	**(000's)**	**(000's)**
2013	NZ-1	Clip2	NZ1-414	414	98,885	8,781	7,966
NZ-1-N	Splay	Mar-14	NZ1-415	415	2,150	191	173
NZ2	Clip2	Union3	NZ2-424	424	50,970	4,526	4,106
NZ3	Clip2	Union3	NZ3-434	434	48,218	4,282	3,884
Total					**200,223**	**17,780**	**16,130**

Notes:
(1) Tonnage factor of 11.26 ft^3/ton or bulk density of 0.0888 t/ft^3 (2.85 SG) used to convert volume to tons
(2) Solid clipped to overburden-bedrock contact
(3) Lower lenses regenerated and appended to clipped upper lenses

TABLE 14.8
SOUTH ZONE DOMAIN'S VOLUMES

Wireframe/Domain			Model	Volume ft^3	Tons[1]	Tonnes
Name 1	**Name 2**	**Name 3**	**Code**	**(000's)**	**(000's)**	**(000's)**
SZ1	A	Clip2	1010	1,678	149	135
SZ1	C	Jan28	1030	4,580	407	369
SZ2	A	Clip2	2010	44,024	3,909	3,546
SZ2	B	Clip2	2020	907	81	73
SZ2	C	Clip2	2030	2,010	178	161
SZ2	D	Jan28	2040	466	41	37
SZ2	E	Clip2	2050	8,477	753	683
SZ2	H	Clip2	2080	36,841	3,272	2,968
SZ2	J	Jan28	2100	6,338	563	511
SZ2	K	Jan29	2110	3,147	279	253
SZ2	N	Jan29	2140	701	62	56
SZ2	Q	Jan29	2170	6,601	586	532
SZ2	R	Jan29	2180	477	42	38
SZ3	A	Jan29	3010	103	9	8
SZ3	C	Feb1	3030	1,363	121	110

Table 14.8 South Zone Domain's Volumes						
Wireframe/Domain			**Model**	**Volume ft^3**	**Tons[1]**	**Tonnes**
Name 1	**Name 2**	**Name 3**	**Code**	**(000's)**	**(000's)**	**(000's)**
SZ3	D	Jan25	3040	23,468	2,084	1,891
SZ3	E	Clip2	3050	26,654	2,367	2,147
SZ3	F	Jan30	3060	552	49	44
SZ3	G	Jan30	3070	208	18	16
SZ3	H	Clip2	3080	85,515	7,594	6,889
SZ3	I	Clip2	3090	6,361	565	513
SZ3	J	Clip2	3100	4,947	439	398
SZ3	K	Clip2	3110	3,460	307	279
SZ3	L	Jan31	3120	2,557	227	206
SZ3	N	Feb13	3140	1,328	118	107
SZ3	O	Jan31	3150	824	73	66
SZ4	LG	Feb1	4000	3,248	288	261
Totals	**27**			**276,835**	**24,583**	**22,302**

Notes:
(1) Tonnage factor 11.26 ft3/ton or 0.0888 tons/ft^3 (2.85 SG) used to convert volumes to tons
(2) Clipped to bedrock surface-overburden base and by other solids

Table 14.9 Stopes and Workings Wireframe Solids Volumetrics						
Wireframe Solid			**Model Code**	**Volume (ft^3)**	**Tons[1]**	**Tonnes**
Name 1	**Name 2**	**Name 3**		**(000's)**	**(000's)**	**(000's)**
NZ	Stopes	Combined	999	27,642	2,455	2,227
NZ-UG	Workings	Combined	999	7,324	650	590
Subtotal				34,966	3,105	2,817
SZStopes	All	Valid	999	37,201	3,303	2,997
SZ	Drifts	Valid	999	5,336	474	430
Subtotal				42,538	3,777	3,427
Total				**77,503**	**6,882**	**6,244**

Notes:
(1) Tonnage factor of 11.26 ft^3/ton or bulk density of 0.0888 ton/ft^3 (2.85 SG) used to convert volume to tons

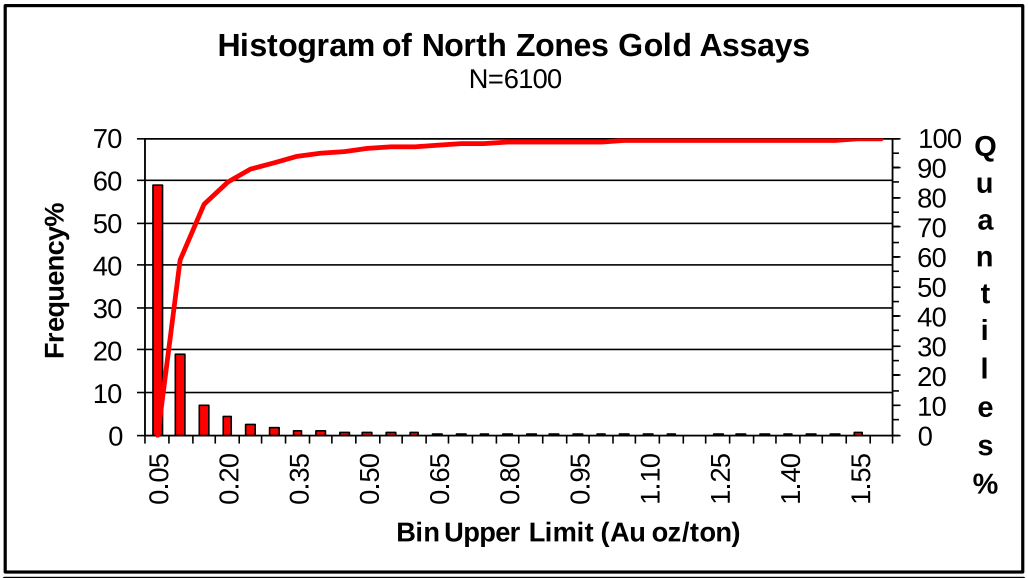
Table 14.10 Drill Hole Intercepts in the Mineral Wireframes					
Domain	**Model Code**	**Intercepts**	**Intercept (ft)**	**Assayed (ft)**	**% Assayed**
North Zone Intercepts					
All Domains	-	1,088	31,305.36	23,007.07	73%
NZ1-414	414	424	11,067.48	7,599.65	69%
NZ1-415	415	7	54.80	54.80	100%
NZ2-424	424	346	10,782.53	8,448.42	78%
NZ3-434	434	311	9,400.54	6,904.20	73%
South Zone Intercepts					

TABLE 14.10 DRILL HOLE INTERCEPTS IN THE MINERAL WIREFRAMES					
Domain	**Model Code**	**Intercepts**	**Intercept (ft)**	**Assayed (ft)**	**% Assayed**
Domain	Model Code	Intercepts	Intercept (ft)	Assayed (ft)	% Assayed
All Domains	-	1,193	27,130.09	18,624.94	69%
SZ1-A	1010	15	263.82	263.82	100%
SZ1-C	1030	11	194.26	194.26	100%
SZ2-A	2010	99	2,000.52	1,970.91	99%
SZ2-B	2020	6	267.28	266.48	100%
SZ2-C	2030	18	195.88	184.49	94%
SZ2-D	2040	3	31.80	31.80	100%
SZ2-E	2050	23	352.33	348.43	99%
SZ2-H	2080	58	1,147.30	1,084.65	95%
SZ2-J	2100	14	215.64	168.22	78%
SZ2-K	2110	7	197.75	130.25	66%
SZ2-N	2140	3	190.46	94.50	50%
SZ2-Q	2170	2	58.50	43.50	74%
SZ2-R	2180	1	19.70	19.70	100%
SZ3-A	3010	2	14.00	14.00	100%
SZ3-C	3030	25	291.65	171.75	59%
SZ3-D	3040	163	3,324.09	1,866.56	56%
SZ3-E	3050	247	6,030.14	3,208.27	53%
SZ3-F	3060	1	20.00	20.00	100%
SZ3-G	3070	1	10.00	10.00	100%
SZ3-H	3080	435	10,850.25	7,127.16	66%
SZ3-I	3090	17	349.64	341.12	98%
SZ3-J	3100	17	370.08	370.08	100%
SZ3-K	3110	14	222.91	222.91	100%
SZ3-L	3120	3	78.36	78.36	100%
SZ3-N	3140	3	198.50	198.50	100%
SZ3-O	3150	3	71.00	71.00	100%
SZ4-LG	4000	2	164.20	124.20	76%

14.1.4 Assay Grade Distributions and Statistics

The grade distribution for gold assays within the wireframes is positively skewed for the both the North and South zones and approaches log-normal to Poisson distribution as shown in the histograms in Figure 14.9 and Figure 14.10. Coefficient of variation for the North Zone exceeds 2 and is approaching 2 for the South Zone. Table 14.11 and Table 14.12 present the assay statistics for the North and South zones.

Figure 14.9 North Zone Assay Histograms



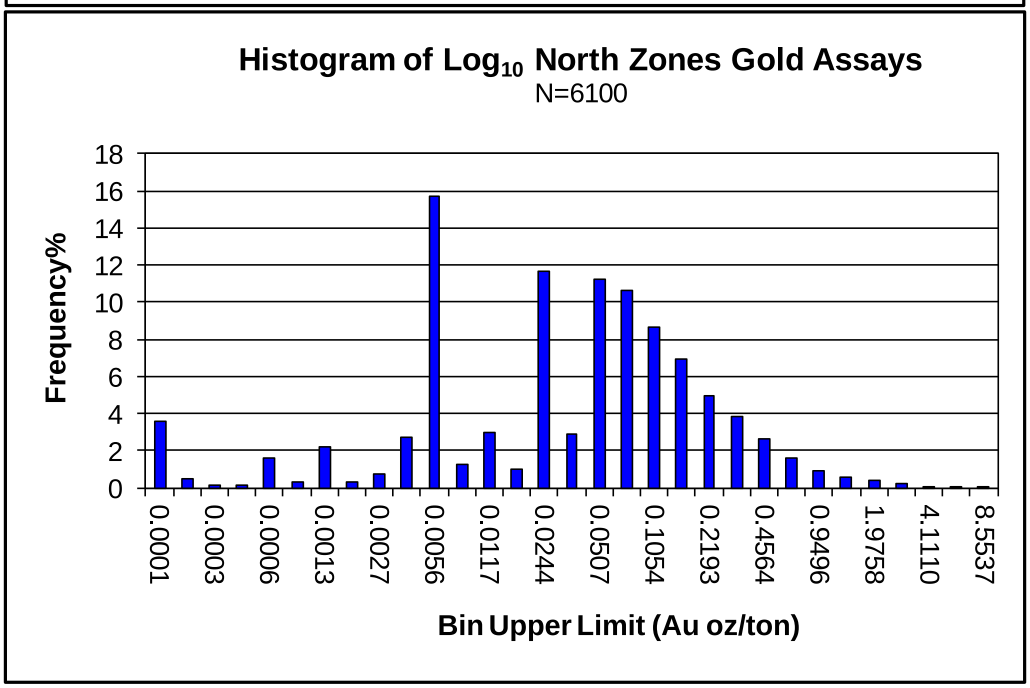
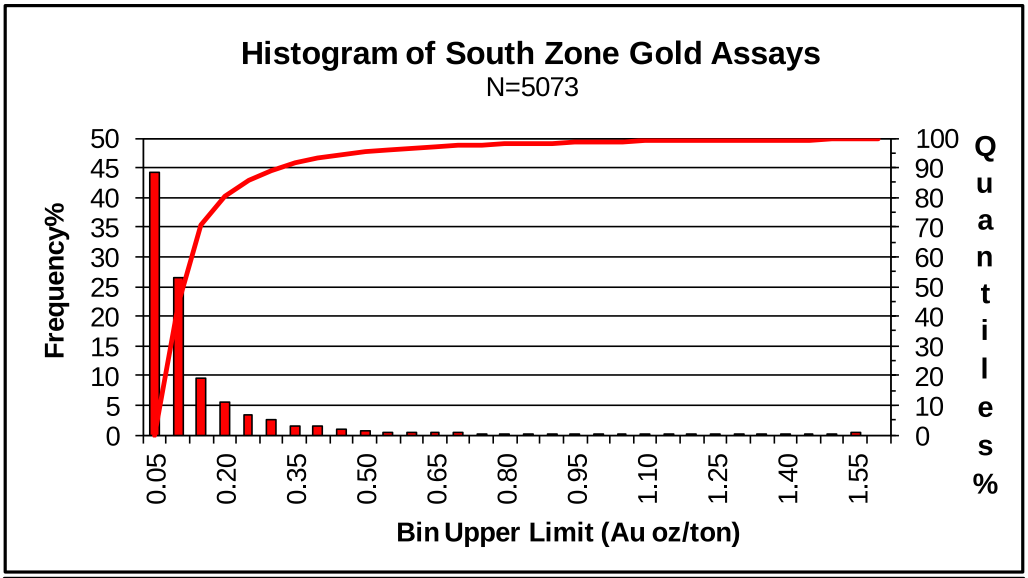


Figure 14.10 South Zone Assay Histograms



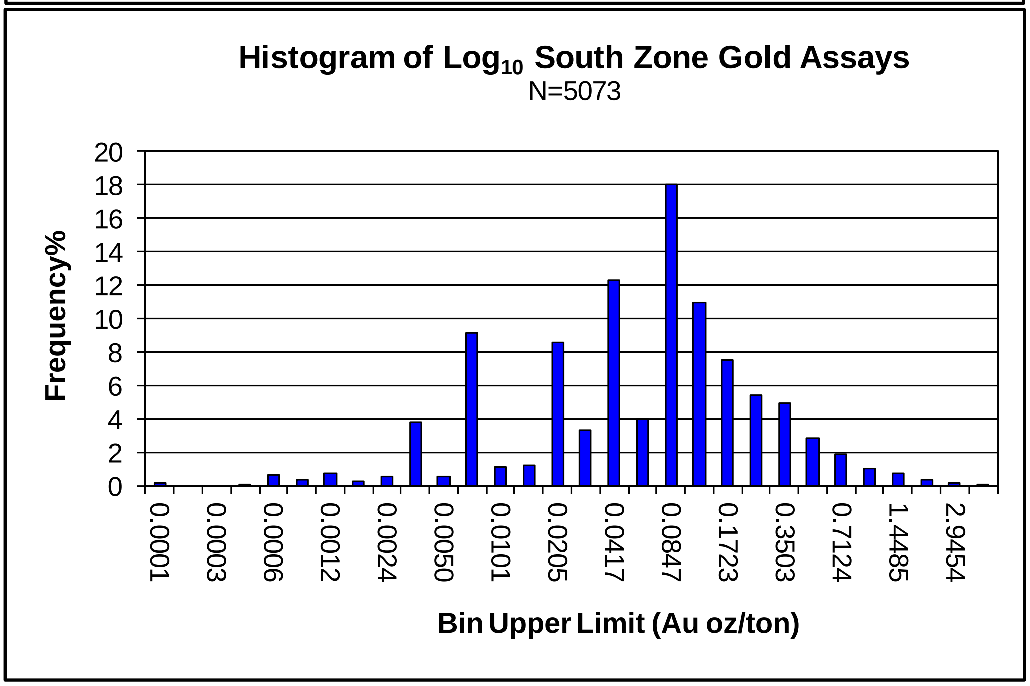


TABLE 14.11
NORTH ZONE ASSAY STATISTICS

Assays above 10,000 ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	1,959	1,959	1,959
Sum	7,977.48	-	-
Minimum	0.20	0.000	0.000
25th Percentile	2.60	0.010	0.010
Median	4.00	0.025	0.025
75th Percentile	5.00	0.065	0.065
Maximum	50.00	2.790	1.500
Mean	4.07	0.065	0.063
Weighted Mean	-	0.054	0.054
Variance	6.11	0.019	0.015
Standard Deviation	2.47	0.138	0.123
Coefficient of Variation	0.61	2.14	1.94
Skewness	6.30	8.64	5.94
Kurtosis	92.84	120.14	50.98
95th Percentile	7.50	0.230	0.230
98th Percentile	10.00	0.412	0.412
99th Percentile	10.33	0.590	0.590
99.5th Percentile	14.00	0.790	0.790

Assays below 10,000 ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	4,141	4,141	4,141
Sum	13,772.80	-	-
Minimum	0.50	0.000	0.000
25th Percentile	2.00	0.005	0.005
Median	3.00	0.040	0.040
75th Percentile	5.00	0.106	0.106
Maximum	15.00	5.930	1.500
Mean	3.33	0.111	0.105
Weighted Mean	-	0.103	0.099
Variance	2.57	0.069	0.040
Standard Deviation	1.60	0.262	0.200
Coefficient of Variation	0.48	2.37	1.91
Skewness	0.39	8.87	4.31
Kurtosis	0.61	128.40	22.55
95th Percentile	5.00	0.440	0.440
98th Percentile	6.00	0.762	0.762
99th Percentile	7.00	1.220	1.220
99.5th Percentile	8.00	1.700	1.500

TABLE 14.12
SOUTH ZONE ASSAY STATISTICS

Assays above 10,000 ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	1,360	1,360	1,360
Sum	5,308.62	-	-
Minimum	1.20	0.000	0.000
25th Percentile	3.00	0.016	0.016
Median	3.90	0.036	0.036
75th Percentile	5.00	0.074	0.074
Maximum	10.60	2.552	1.500
Mean	3.90	0.071	0.070
Weighted Mean	-	0.068	0.068
Variance	1.11	0.018	0.015
Standard Deviation	1.05	0.133	0.121
Coefficient of Variation	0.27	1.88	1.72
Skewness	0.53	8.05	5.48
Kurtosis	1.18	109.59	43.43
95th Percentile	5.40	0.248	0.248
98th Percentile	6.00	0.439	0.439
99th Percentile	6.28	0.609	0.609
99.5th Percentile	6.86	0.811	0.811

Assays below 10,000 ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	3,713	3,713	3,713
Sum	12,238.13	-	-
Minimum	0.30	0.001	0.001
25th Percentile	2.00	0.025	0.025
Median	3.00	0.070	0.070
75th Percentile	5.00	0.140	0.140
Maximum	16.00	4.200	1.500
Mean	3.30	0.137	0.133
Weighted Mean	-	0.130	0.127
Variance	2.31	0.059	0.043
Standard Deviation	1.52	0.243	0.206
Coefficient of Variation	0.46	1.77	1.55
Skewness	0.32	6.10	3.78
Kurtosis	0.78	58.58	17.71
95th Percentile	5.00	0.480	0.480
98th Percentile	6.00	0.858	0.858
99th Percentile	6.00	1.130	1.130
99.5th Percentile	7.00	1.658	1.500

14.1.5 Grade Capping

After review of the histograms and assay statistics, P&E prepared cumulative frequency% log-probability plots and cutting curves for all gold assays in the zones in order to examine the need for grade capping. The skewed nature of the gold grade distribution and moderate to high coefficient of variation indicates that grade capping is warranted to avoid over influencing the average grade. Based on similar grade distributions for the North and South zones, P&E selected 1.5 oz/ton Au as an appropriate grade capping level for both zones. Examination of assays >1.5 oz/ton Au in 3D space showed essentially random distribution, i.e. no clustering that would warrant independent wireframing.

The effect of capping at 1.5 oz/ton Au is shown in Table 14.13.

TABLE 14.13
IMPACT OF CAPPING AT 1.5 OZ/T AU

	Max. Au oz/t	Au oz/ton	Coeff. Var.	Au oz/t Capped	Coeff. Var.	No. Capped	Percentile	Metal Loss%	Data Loss%
North Zone	5.93	0.096	2.4	0.092	2.0	30	99.5	3%	0.49%
South Zone	4.20	0.118	1.8	0.115	1.6	13	99.5	2.5%	0.43%

Capping was applied to individual assays in the North and South zones prior to compositing.

Figure 14.11 North Zone Gold Assays Log-Probability Plot

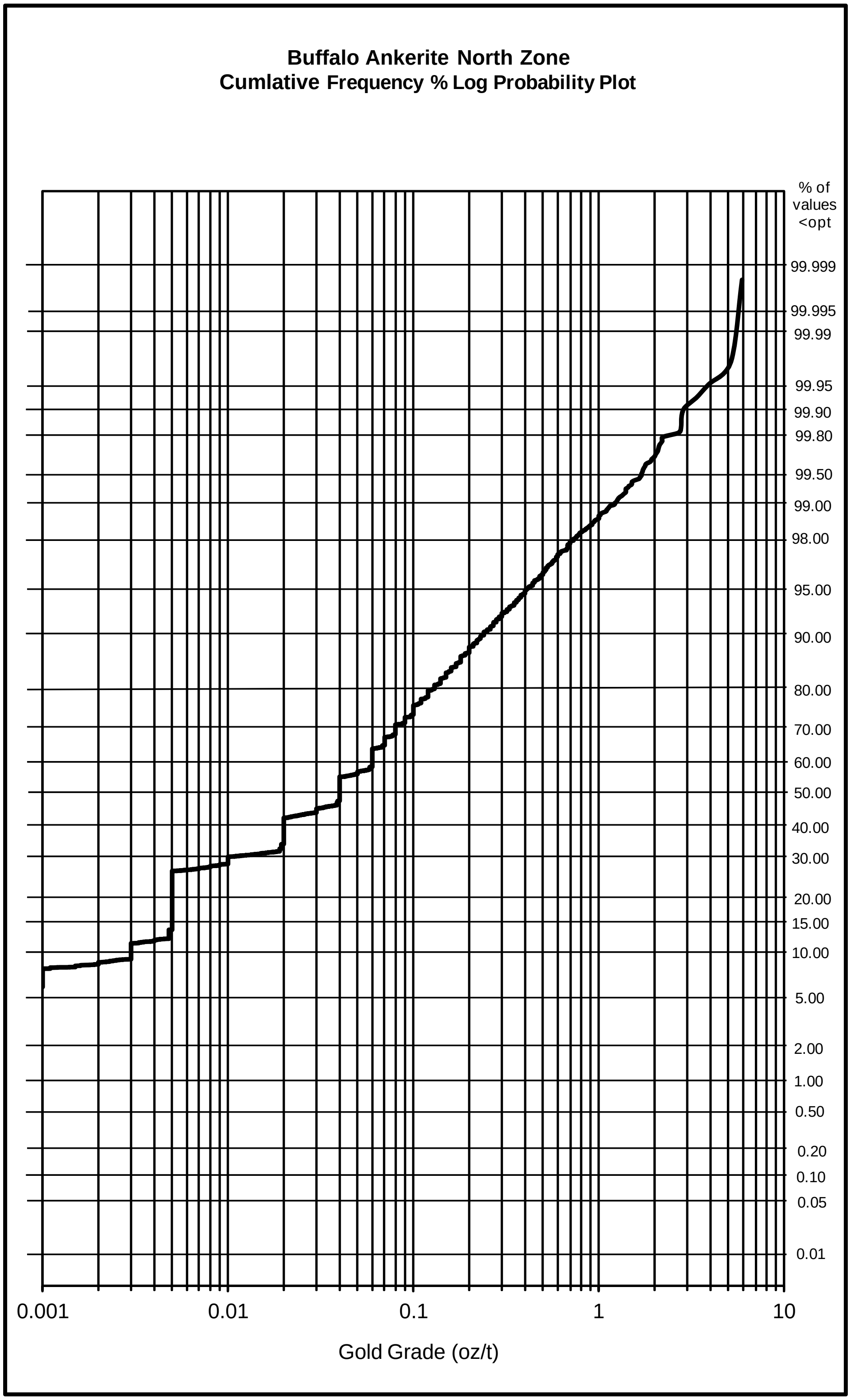


Figure 14.12 South Zone Gold Assays Log-Probability Plot

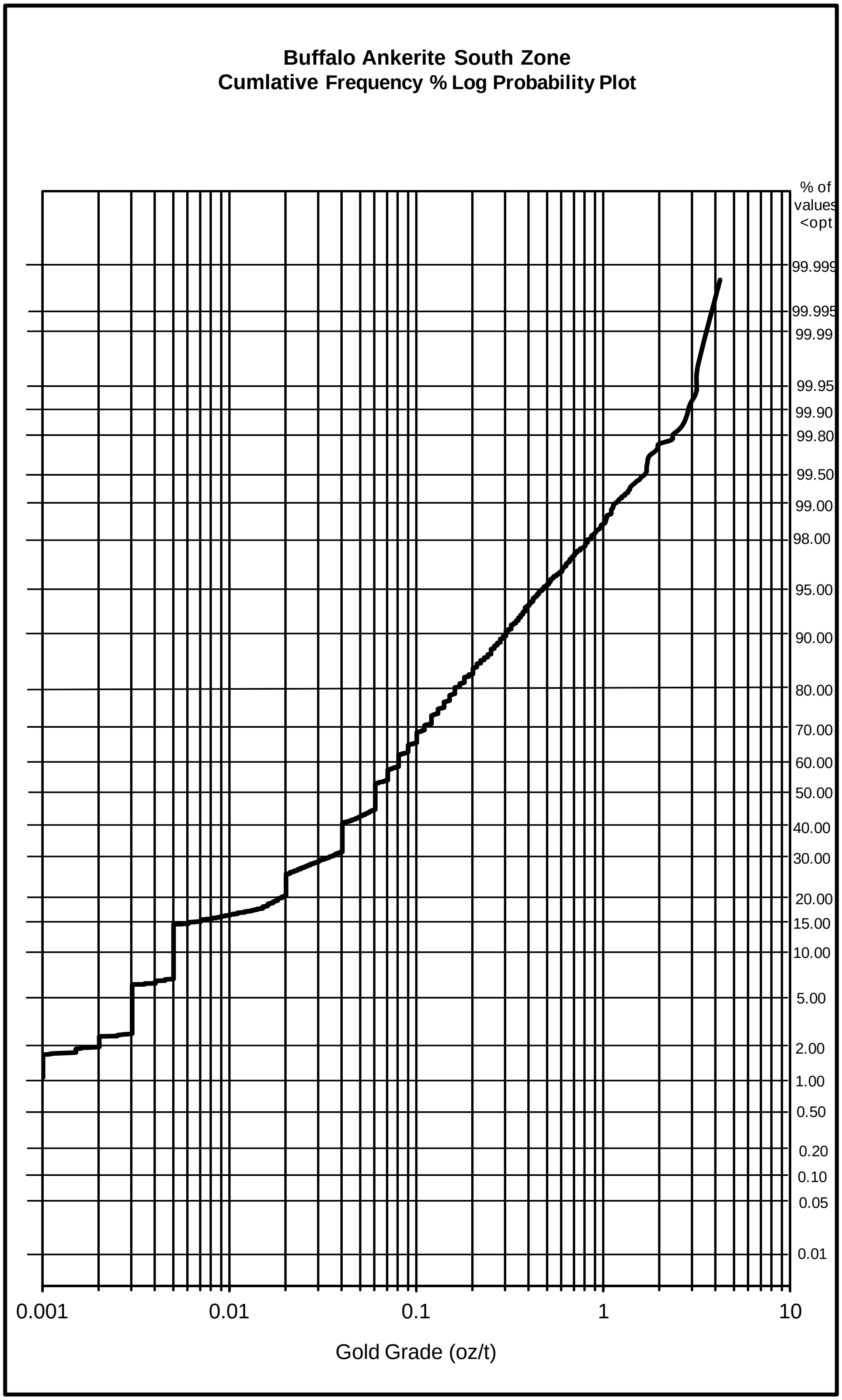


Figure 14.13 Assay Grade Cutting Curves (North Zone)

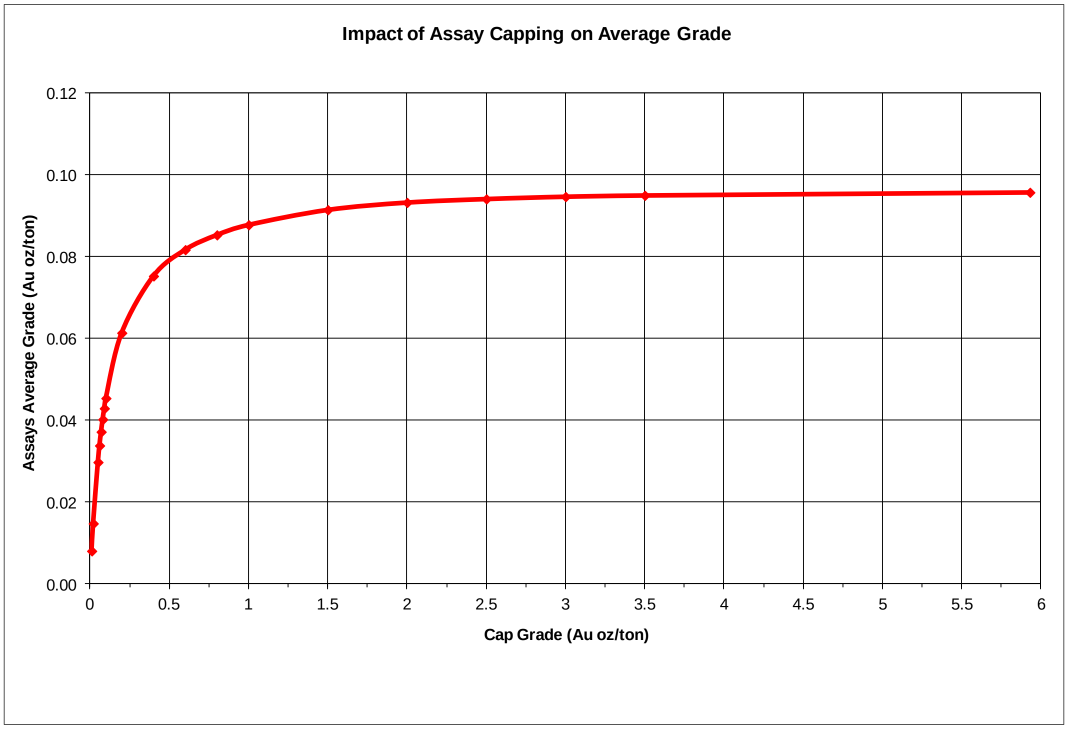
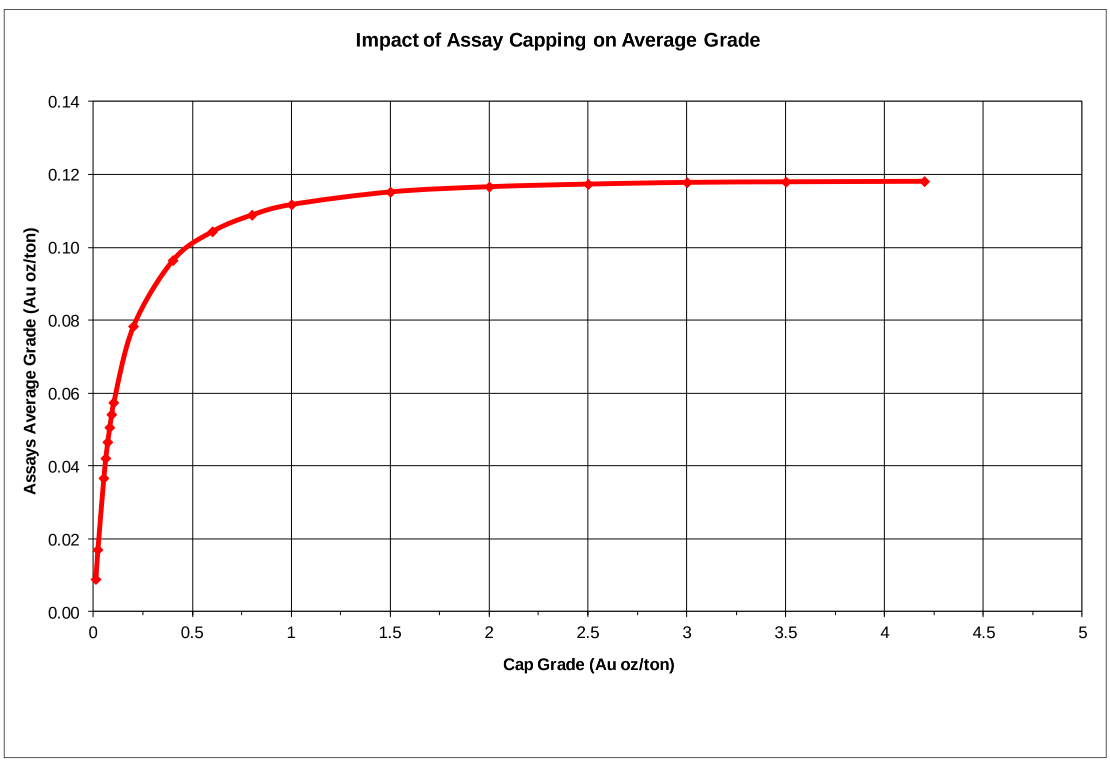


Figure 14.14 Assay Grade Cutting Curves (South Zone)



14.1.6 Bulk Density

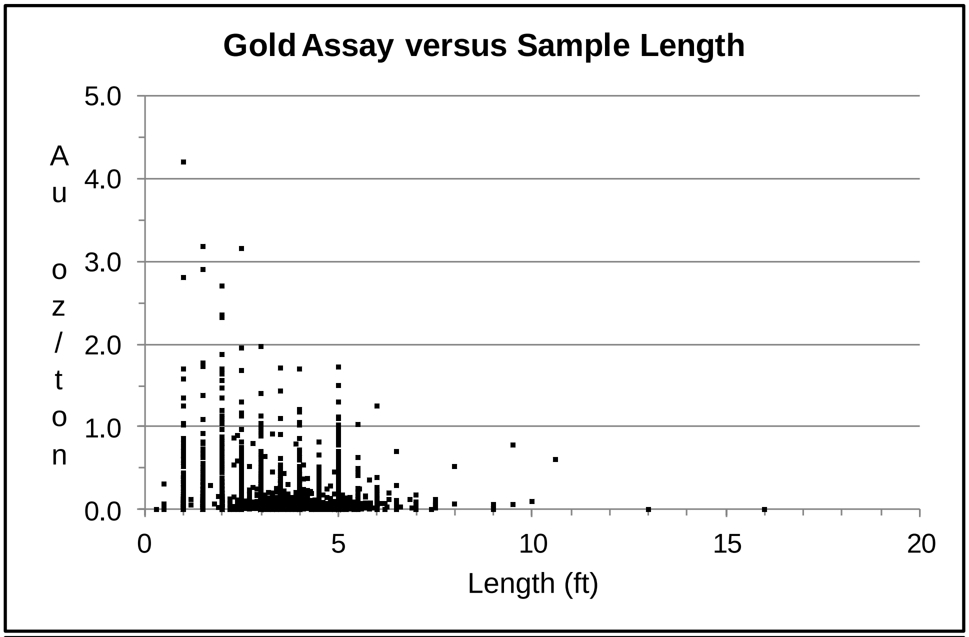
To convert volume to tons, a bulk density of 0.0888 tons/ft^3 (11.26 ft^3/ton), equivalent to a specific gravity (SG) of 2.85, was applied uniformly throughout the North and South zones. SG testing has been performed for 307 samples of various rock and mineral types. P&E's review of a subset (58) of these data related to tourmaline-quartz-carbonate-pyrite breccias, quartz veining and quartz feldspar porphyry indicates that an SG of 2.85 is appropriate for Buffalo Ankerite gold mineralization.

14.1.7 Assay Compositing

Assay composites at five foot lengths were generated down hole by length weighting the assays captured by GEMS in the zone wireframes. The five foot length is at the 94.4 percentile of the sample lengths distribution for the South Zone and 91.9 percentile for the (North Zone Figure 14.15 and Figure 14.16) Assaying is generally continuous in the zones for surface drilling with assays generally only missing where holes traversed the open voids of stopes and drifts. For underground drilling, assaying was generally carried out only for identified mineralization thereby resulting in incomplete assaying across zone widths. For compositing the underground hole assays, explicit and implicit missing assays were set to zero grade whereas for surface holes missing assays/voids were ignored for the purpose of grade interpolation.

To regularize the composite lengths and cull length artifacts arising from irregularities in zone intercepts by GEMS, composite lengths shorter than 2.5 ft were omitted from the estimate for surface holes and composites less than 2 ft were omitted for underground holes where zone widths may be defined by single, narrow assay intervals at the zone walls. No apparent grade bias was introduced by this practice. Table 14.14 and Table 14.15 present summary statistics for the 5 ft composites.

Figure 14.15 South Zone Core Sample Length Distribution



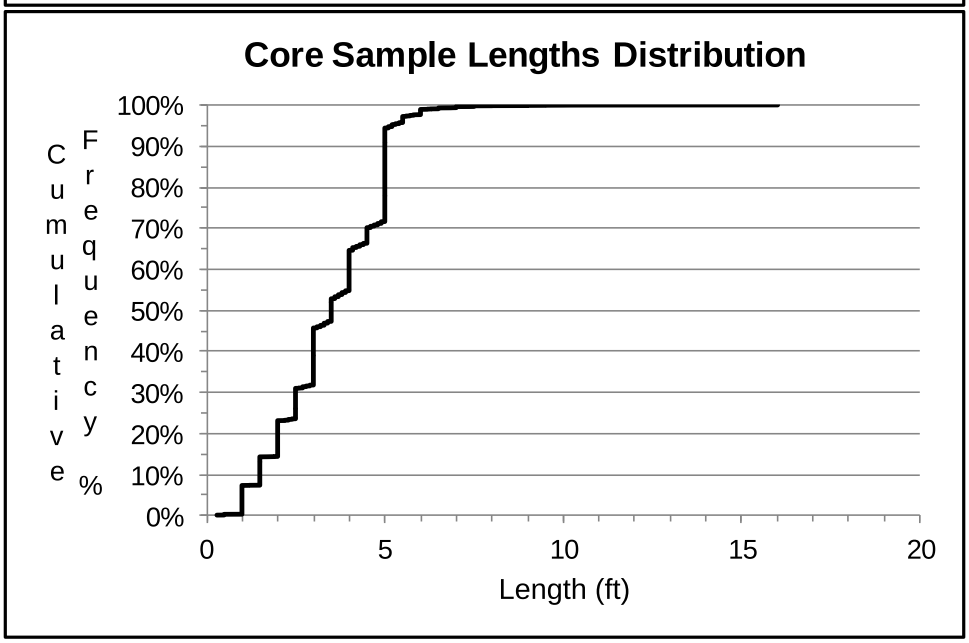
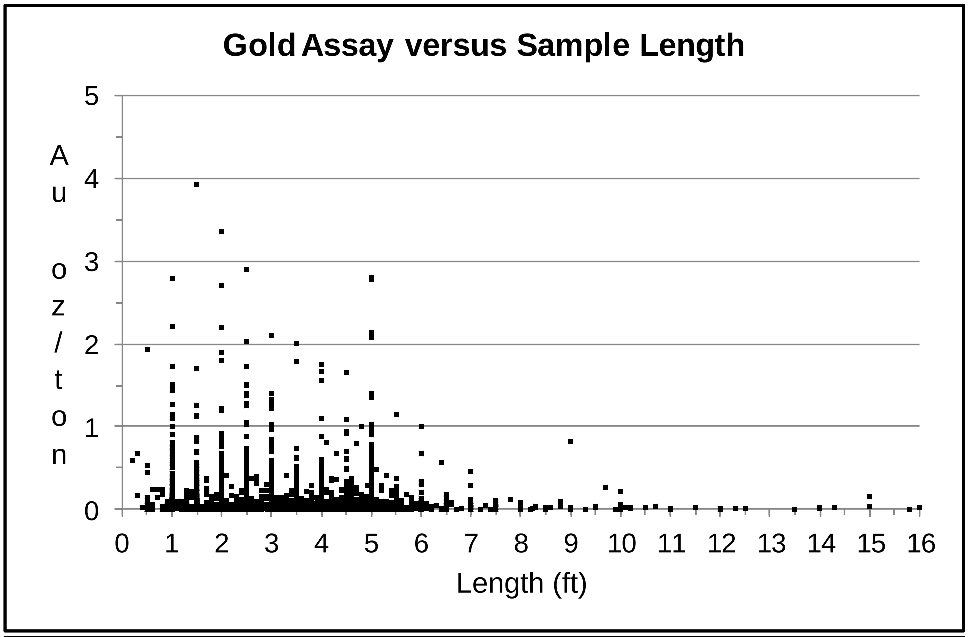


Figure 14.16 North Zone Core Sample Length Distribution



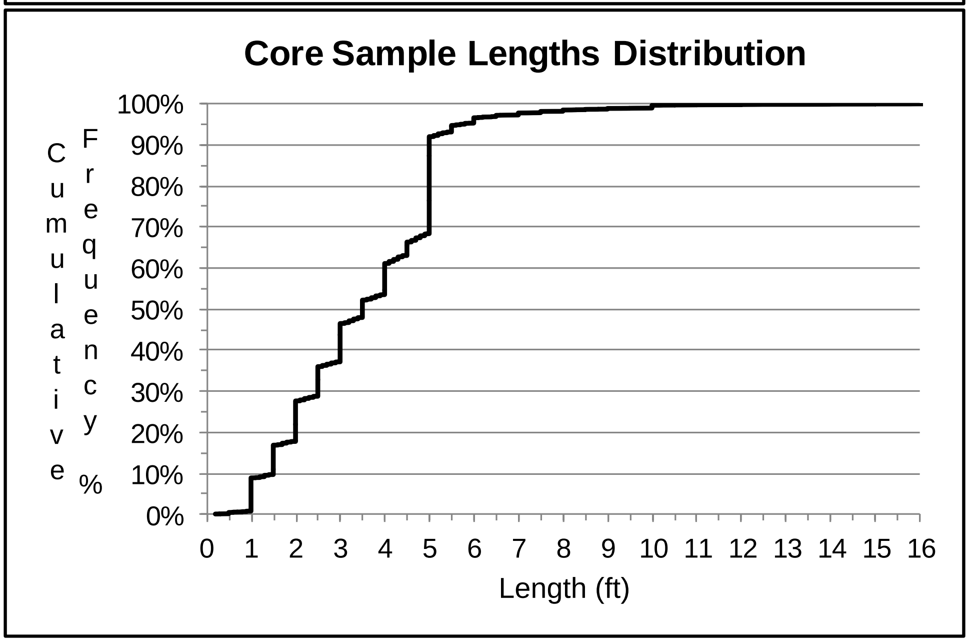


TABLE 14.14
NORTH ZONE COMPOSITE STATISTICS

Composites above 10,000 ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	1,986	1,986	1,986
Sum	9,710.45	-	-
Minimum	2.00	0.000	0.000
25th Percentile	5.00	0.003	0.003
Median	5.00	0.020	0.020
75th Percentile	5.00	0.050	0.050
Maximum	5.00	0.970	0.902
Mean	4.86	0.045	0.045
Weighted Mean	-	0.045	0.045
Variance	0.27	0.007	0.006
Standard Deviation	0.52	0.081	0.080
Coefficient of Variation	0.11	1.79	1.77
Skewness	-3.88	4.77	4.56
Kurtosis	14.67	32.71	29.56
95th Percentile	5.00	0.173	0.173
98th Percentile	5.00	0.288	0.282
99th Percentile	5.00	0.422	0.417
99.5th Percentile	5.00	0.570	0.524

Composites below 10,000 ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	4,232	4,232	4,232
Sum	20,485.46	-	-
Minimum	2.00	0.000	0.000
25th Percentile	5.00	0.000	0.000
Median	5.00	0.019	0.019
75th Percentile	5.00	0.072	0.072
Maximum	5.00	2.800	1.500
Mean	4.86	0.069	0.066
Weighted Mean	-	0.069	0.066
Variance	0.27	0.025	0.018
Standard Deviation	0.52	0.159	0.135
Coefficient of Variation	0.11	2.31	2.04
Skewness	-3.66	7.13	4.89
Kurtosis	12.85	78.89	33.24
95th Percentile	5.00	0.278	0.278
98th Percentile	5.00	0.488	0.482
99th Percentile	5.00	0.752	0.695
99.5th Percentile	5.00	1.025	0.913

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

TABLE 14.15
SOUTH ZONE COMPOSITE STATISTICS

Composites above 10,000 ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	1,181	1,181	1,181
Sum	5,726.45	-	-
Minimum	2.50	0.000	0.000
25th Percentile	5.00	0.019	0.019
Median	5.00	0.038	0.038
75th Percentile	5.00	0.076	0.076
Maximum	5.00	0.893	0.893
Mean	4.85	0.067	0.066
Weighted Mean	-	0.067	0.066
Variance	0.25	0.009	0.008
Standard Deviation	0.50	0.09	0.09
Coefficient of Variation	0.10	1.38	1.36
Skewness	-3.42	4.02	3.85
Kurtosis	10.54	22.39	20.41
95th Percentile	5.00	0.222	0.222
98th Percentile	5.00	0.363	0.363
99th Percentile	5.00	0.438	0.438
99.5th Percentile	5.00	0.606	0.604

Composites below 10,000ft Elevation			
Statistic	**Length (ft)**	**Au oz/ton**	**Au oz/t Capped**
Count	4,367	4,367	4,367
Sum	20,979.83	-	-
Minimum	2.00	0.000	0.000
25th Percentile	5.00	0.000	0.000
Median	5.00	0.033	0.033
75th Percentile	5.00	0.090	0.090
Maximum	5.00	3.150	1.500
Mean	4.86	0.077	0.075
Weighted Mean	-	0.076	0.075
Variance	0.27	0.02	0.02
Standard Deviation	0.52	0.14	0.13
Coeff. Variation	0.11	1.87	1.74
Skewness	-3.66	5.89	4.08
Kurtosis	12.85	69.17	23.78
95th Percentile	5.00	0.306	0.304
98th Percentile	5.00	0.481	0.480
99th Percentile	5.00	0.729	0.679
99.5th Percentile	5.00	0.896	0.858

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

14.1.8 Block Models

Separate block models were constructed for the North and South zones. Table 14.16 summarizes model origins and block parameters. The two models overlap in part with blocks in the shared space having centroids in common.

TABLE 14.16
BLOCK MODEL SET-UP AND PARAMETERS

North Zone	X	Y	Z
Origin	3,000	5,690	11,500
Block Size (ft)	10	10	10
Number of Blocks	320	385	450
Rotation		0	
South Zone			
Origin	1,500	4,200	11,500
Block Size (ft)	10	10	10
Number of Blocks	550	280	350
Rotation		0	

The block models are oriented with X axis at 090° azimuth i.e. non-rotated, and each have block dimensions at 10 ft EW x 10 ft NS x 10 ft vertical. Block dimensions take into account the drill hole spacing on 20 ft (6 m) to 100+ ft (30 m) sections, zone and stope widths. Since the boundaries of the zones are locally within several feet of one another and within the resource block dimensions (10 ft), three partial–percent models were created in GEMS for the South Zone and two partial–percent models for the North Zone so that varied percentages of the zones and waste could be coded into "Standard" models for GEMS reporting. In addition to the grade models, a mined block model for each zone was prepared from the workings solids in which the blocks occupying stopes, raises or drifts were coded and the percent mined subtracted from the resource blocks.

14.1.9 Variography

Linear semi-variograms (variograms) of the five foot resource composites were prepared down-hole for the North and South zones to assess the gold nugget effect which was found to be relatively moderate at 20% for both zones. Three-dimensional variography, using variance normalized, nested spherical models and 50 ft lags, was carried out for the zones' strikes and dips. The strike and dip variograms for the South Zone were not robust for low spread angles and omni-directional variograms at 90° spread angles were adopted for further variography which indicated ranges in the order of 100 ft. The North Zone variograms were reasonable for strike and dip and showed better continuity/longer ranges from 119 ft up to 183 ft. Table 14.17 shows results of the variography. Variograms are presented in Appendix IV.

TABLE 14.17 VARIOGRAM MAXIMUM RANGES	
Variogram	**Range (ft)**
North Zone	
Omni; composites above 10,000' EL	61
Omni; composites below 10,000' EL	85
On strike (135°) below 10,000' EL 45° spread angle	119
Domain 414 on strike (169°) ; 45° spread angle	145
Domain 414 dip; 45° spread angle	168
Domain 434 strike; 45° spread angle	183
South Zone	
Omni; composites above 10,000' EL	101
Omni; all composites	77
Domain 3080; composites below 10,000' EL	95

14.1.10 Block Model Grade Interpolation

14.1.10.1 Search Strategy and Grade Interpolation

Variography results, drill hole spacing and orientation of the domains within the North and South zones guided the interpolation search strategy. The upper northeast portion of the North zone has shallow westerly dips which steepen to -70° to -75° to the west. The South zone domains dip steeply north at -60° in the west on the south flank of the syncline, flatten to sub horizontal where they plunge to the west in the central area, and dip steeply south in the northeast area on the north limb of the syncline. Consequently the block models were "mapped" into areas of similar dip and grade was interpolated within the mapped areas using search ellipses matched to domain trends. Two search ellipse orientations were employed for the North Zones and three orientations (steep north dip, flat west dip, steep south dip) were used for the South domains. Rotation of the ellipses followed the GEMS ZYZ convention that links ellipse axis rotation to the block model orientation.

The block models were divided into an open pit portion above the 10,000 ft elevation and an underground portion below 10,000 ft elevation. Grade interpolation for the open pit portion was based on composites located above the 10,000 ft elevation. Similarly the underground portion of the block model was interpolated from composites lying below the 10,000 ft elevation.

Grade interpolation was carried out by the inverse distance cubed method (ID^3) in up to five passes (Table 14.18). Multiple passes were utilized to allow for progressive capture of sample data at widening spatial separation because of the variability of drill hole spacing from 20 ft in underground holes to several hundred feet for surface holes. This approach also suited the separation of open pit and underground composites at the 10,000 ft elevation. The fifth final pass was designed to complete the filling of the wireframe.

Small portions of four domains lying just above and below the 10,000 ft elevation had no available composites in either the underground or open pit sets, i.e. for the wireframe extending below the 10,000 ft elevation there were no underground composites and for the wireframe extending to above the 10,000 ft elevation, there were no open pit composites. Grade for these domains was populated in a sixth broad pass utilizing all composites. This did not impact on resources later determined to be in the Whittle designed open pit.

In P&E's opinion, the ID^3 method is reasonable since the nugget effect is relatively low (20%) and some smoothing is desirable. In addition, the variography is not particularly robust due to the low number of drill holes and samples for many South Zone domains, 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. Composite sample minimums and maximums, and multiple expanding expanded passes were adopted to avoid over-smoothing and preserve local grade variability.

Because of areas in the North and South zones have been subjected to more detailed and close spaced underground drilling, P&E examined South Zone domain SZ3-H, an extensive zone with underground and surface drilling and many intercepts, by means of cell declustering to assess whether sample clustering could affect ID interpolation. Declustered means varied mostly less than ±5% from the composites average indicating no significant impact from clustering.

TABLE 14.18
INTERPOLATION PARAMETERS AND SEARCH DISTANCES

North Zone	Pass 1	Pass 2	Pass 3	Pass 4	Pass 5
Minimum Composites	4	2	2	2	1
Maximum Composites	12	12	12	12	12
Maximum Composites per Hole	2	-	-	-	-
ZYZ 10°/-75°/0° Search X (ft)1	150	150	300	600	9999
ZYZ 10°/-75°/0° Search Y (ft)	125	125	250	500	9999
ZYZ 10°/-75°/0° Search Z (ft)	20	20	40	100	9999
ZYZ 10°/-20°/0° Search X (ft)2	150	150	300	-	-
ZYZ 10°/-20°/0° Search Y (ft)	125	125	250	-	-
ZYZ 10°/-20°/0° Search Z (ft)	20	20	50	-	-
ZYZ 15°/-45°/0° Search X (ft)3	150	150	300	600	9999
ZYZ 15°/-45°/0° Search Y (ft)	125	125	250	500	9999
ZYZ 15°/-45°/0° Search Z (ft)	20	20	40	80	9999
South Zone	**Pass 1**	**Pass 2**	**Pass 3**	**Pass 4**	**Pass 5**
Minimum Composites	4	2	2	2	1
Maximum Composites	12	12	12	12	12
Maximum Composites per Hole	2	-	-	-	-
ZYZ 0°/-61°/30° Search X (ft)4	80	80	160	320	9999
ZYZ 0°/-61°/30° Search Y (ft)	100	100	200	400	9999
ZYZ 0°/-61°/30° Search Z (ft)	40	40	80	120	9999
ZYZ 0°/-30°/0° Search X (ft)5	100	100	200	400	9999
ZYZ 0°/-30°/0° Search Y (ft)	80	80	160	320	9999
ZYZ 0°/-30°/0° Search Z (ft)	20	20	40	80	9999
ZYZ 20°/60°/10° Search X (ft)6	80	80	160	320	9999
ZYZ 20°/60°/10° Search Y (ft)	100	100	200	400	9999
ZYZ 20°/60°/10° Search Z (ft)	40	40	60	120	9999

Notes:
(1) Steep west dip
(2) Shallow west dip
(3) -45° west dip for domain 415

(4) Steep north dip
(5) Flat dip, west plunge
(6) Steep south dip

14.1.11 Resource Classification

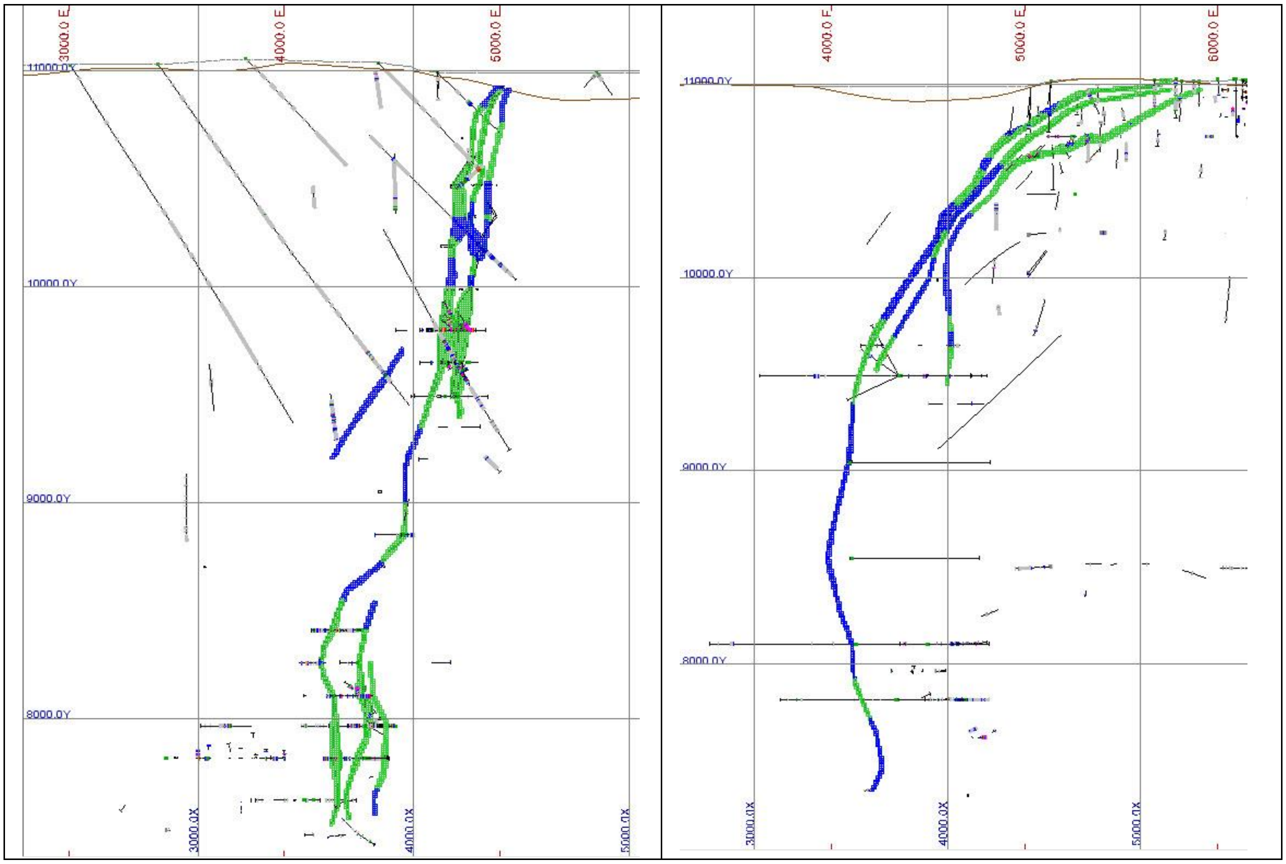
In P&E's opinion, the level of drilling, assaying and exploration work completed to 2012 is sufficient to show that the Buffalo Ankerite North and South zones have the size and average gold grades to indicate reasonable potential for economic extraction at current gold prices and thus qualify them as Mineral Resources under CIM definitions. Resources were classified as Indicated and Inferred based on the drill hole and sampling data spacing as well as geological interpretation of structure and grade continuity as indicated from variography. The continuity of the gold bearing structures has been well established by drilling and past mining and drifting on the structures, however, grade data is limited to underground and surface drilling with underground chip/channel sampling in stopes and drifts unavailable in digital format. Drill hole intercepts in the domains range from 20 ft to several hundred feet. Where fan drilling was undertaken from surface in the South Zone, domain intercepts are progressively wider at depth and confidence in the grade estimation for some zones is low resulting in Inferred Resource classification.

For the North Zone, P&E classified Indicated resources as blocks lying within 150 ft of a drill hole (variogram range) and where two holes are present in this distance. Blocks in the model were coded using a 150 ft spherical search radius and with the number of holes found in the search. This approach worked well for the North Zone. In the South Zone, a similar approach was employed based on a 100 ft distance to the nearest composite and the use of two holes from the first interpolation pass. The areas outlined were not as definitive as for the North zone so the results were used as a guide to build solids encompassing the Indicated Resources. The solids were built from polylines on 100 ft cross sections. Blocks contained within both the classification solids and the mineral wireframes were coded as Indicated Resources. Blocks within the mineral wireframes but outside the classifications solids were classified as Inferred Resources. The Indicated resource blocks formed well defined areas where drilling is spaced at 100 ft near surface in the east and central areas of the South Zone and where underground drill holes tested the zones in the west portion of the South Zone below the 10,000 ft elevation.

Figure 14.17 and Figure 14.18 illustrate Indicated Resources and Inferred Resources on two cross sections for each zone. Indicated Resource blocks are shown in green.

Figure 14.17 Cross Sections Showing Resource Classification for the North Zone

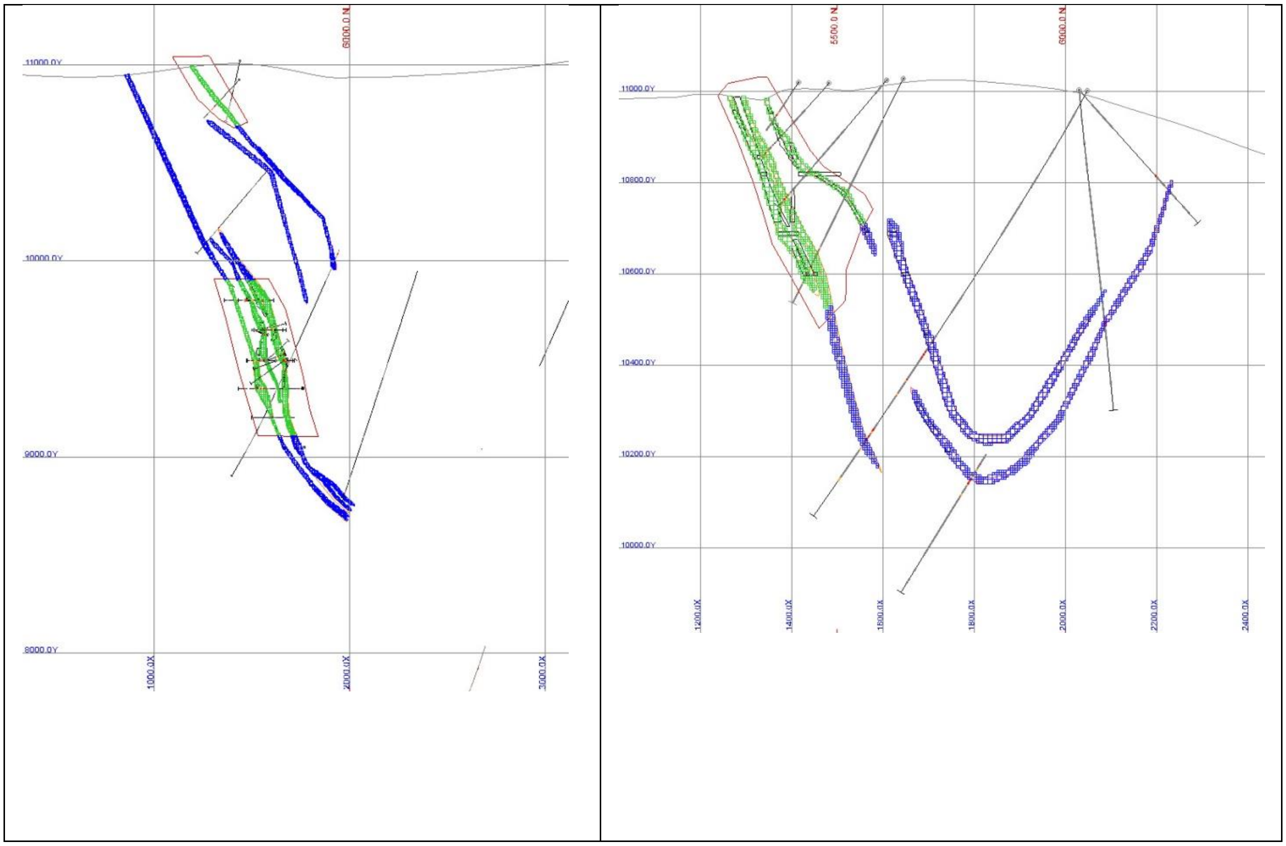
Cross Section 7000N **Cross Section 8100N**



Legend	
(green)	Indicated
(blue)	Inferred

Figure 14.18 Cross Sections Showing Resource Classification Solid and Blocks for the South Zone

Cross Section 3620E **Cross Section 5620E**



Legend	
	Indicated
	Inferred

14.1.12 Block Model Inventory

The solids representing the underground drifts and digitized mined-out stopes were used to prepare percent block models of the openings in GEMS. The percents of the mined blocks were subtracted from the resource zones' block percents to arrive at the post mining percent remaining in the resource zones and this percentage was used for resource reporting. P&E removed 1.79 million tons and 120,000 ounces from the North Zone, and 3.34 million tons and 238,000 ounces from the South Zone, on a global resource basis.

Table 14.19 and Table 14.20 provide the results of grade interpolation for the block models at various gold cut-off grades.

Table 14.19 Block Model Inventory North Zone (above 10,000 ft EL)						
Cut-Off Au oz/ton	**Indicated**			**Inferred**		
	Tons (000's)	**Au oz/ton**	**Au Ounces (000's)**	**Tons (000's)**	**Au oz/ton**	**Au Ounces (000's)**
Wireframe	4,379	0.045	195	2320	0.050	115
0.006	3,927	0.049	194	2113	0.054	114
0.009	3,724	0.052	193	1990	0.057	113
0.015	3,333	0.056	188	1836	0.061	112
0.030	2,299	0.072	165	1402	0.073	102
0.045	1,506	0.090	135	1025	0.086	87.9
0.050	1,313	0.096	126	911	0.090	82.4
0.075	697	0.127	88.6	456	0.118	54.0
0.100	398	0.158	63.0	197	0.162	31.8
0.125	245	0.188	46.0	115	0.198	22.8
0.150	151	0.220	33.1	72.4	0.234	16.9
North Zone (below 10,000 ft EL)						
	Indicated			**Inferred**		
Cut-Off Au oz/ton	**Tons (000's)**	**Au oz/ton**	**Au Ounces (000's)**	**Tons (000's)**	**Au oz/ton**	**Au Ounces (000's)**
Wireframe	6,395	0.060	385	2,888	0.053	153
0.006	5,729	0.067	383	2,613	0.059	153
0.009	5,470	0.070	381	2,508	0.061	152
0.015	4,909	0.076	375	2,264	0.066	149
0.030	3,744	0.093	349	1,781	0.078	138
0.045	2,830	0.111	315	1,433	0.088	126
0.050	2,571	0.118	302	1,322	0.091	120
0.075	1,593	0.152	242	637	0.124	79.2
0.100	1,028	0.189	194	276	0.176	48.5
0.125	724	0.221	160	126	0.251	31.5
0.150	531	0.252	134	72.0	0.337	24.3

Notes:

(1) The block model mineral inventory was estimated by conventional 3D block modelling based on wireframing at a 0.015 oz/ton (0.5 g/t Au) cut-off and ID^3 grade interpolation.

(2) The Qualified Persons for this Mineral Resource estimate are: Richard Routledge, P.Geo., and Eugene Puritch, P.Eng.

(3) A uniform bulk density of 0.0888 ton/ft^3 (2.85 tonnes/m^3) has been applied for volume to tons conversion.

(4) Mineral inventory in the block model is estimated from surface at approximately 11,050 ft elevation to the 7,290 ft elevation or ±3,740 ft depth (1,140 m depth).

(5) Block model mineral inventory does not constitute Mineral Resources or Mineral Reserves under CIM definitions and does not have demonstrated economic viability. The mineral inventory contains the Indicated and Inferred Mineral Resources that have been outlined by Whittle™ pit design.

Table 14.20 Block Model Inventory South Zone (above 10,000 ft EL)						
Cut-Off Au oz/ton	**Indicated Resources**			**Inferred Resources**		
	Tons (000's)	**Au oz/ton**	**Au Ounces (000's)**	**Tons (000's)**	**Au oz/ton**	**Au Ounces (000's)**
Wireframe	3,318	0.067	224	8,420	0.057	479
0.006	3,299	0.068	224	8,335	0.057	478
0.009	3,288	0.068	224	8,322	0.057	478
0.015	3,149	0.070	222	7,973	0.059	474
0.030	2,516	0.082	207	6,844	0.065	447
0.045	1,777	0.101	180	4,747	0.078	368
0.050	1,615	0.107	172	4,068	0.083	336
0.075	927	0.140	130	1,710	0.113	193
0.100	609	0.168	103	893	0.136	122
0.125	368	0.205	75.4	395	0.166	65.5
0.150	252	0.236	59.5	209	0.193	40.5
South Zone (below 10,000 ft EL)						
Cut-Off	**Indicated Resources**			**Inferred Resources**		
	Tons	**Au**	**Au Ounces**	**Tons**	**Au**	**Au Ounces**
Au oz/ton	(000's)	oz/ton	(000's)	(000's)	oz/ton	(000's)
Wireframe	5,335	0.069	367	4,348	0.064	280
0.006	5,140	0.071	367	4,272	0.065	280
0.009	5,026	0.073	366	4,244	0.066	280
0.015	4,778	0.076	363	4,129	0.067	278
0.030	4,037	0.086	346	3,588	0.074	266
0.045	3,231	0.098	316	2,539	0.089	226
0.050	2,973	0.102	304	2,262	0.094	213
0.075	1,786	0.129	230	1,387	0.114	159
0.100	1,039	0.159	166	593	0.155	91.6
0.125	660	0.187	123	294	0.202	59.3
0.150	195	0.219	88.9	195	0.235	45.8

Notes:

(1) The block model mineral inventory was estimated by conventional 3D block modelling based on wireframing at a 0.045 oz/ton (1.5 g/t Au) cut-off and ID^3 grade interpolation.

(2) The Qualified Persons for this Mineral Resource estimate are: Richard Routledge, P.Geo., and Eugene Puritch, P.Eng.

(3) A uniform bulk density of 0.0888 ton/ft3 (2.85 tonnes/m^3) has been applied for volume to tons conversion.

(4) Mineral inventory in the block model is estimated from the 10,000 ft elevation or ±1,000 ft depth (300 m depth) below surface to the 8,050 ft elevation, a depth of 2,971 ft.

(5) Block model mineral inventory does not constitute Mineral Resources or Mineral Reserves under CIM definitions and does not have demonstrated economic viability. The mineral inventory contains the Indicated and Inferred Mineral Resources that have been outlined by Whittle™ pit design.

14.1.13 Resource Reporting

The block models to the 10,000 ft elevation for the North and South zones were exported to Whittle™ open pit optimization software for preliminary open pit design and to enable the reporting of open pit resources.

The Whittle™ designed pit is based on parameters listed in Table 14.21.

TABLE 14.21 WHITTLE™ INPUT PARAMETERS	
Gold Price	US$1,600/oz
Process Recovery	90%
Pit Slopes	45°
Overburden Stripping	$1.22/ton
Ore/Waste Mining Cost	$1.68/ton
Process Cost	$11.35/ton
G&A	$3.16/ton
Pit Discard Cut-Off Grade	0.01 oz/ton

Ramp design and pit floor modifications were not done to finalize the pit at this resource estimation stage. Such work has an impact on the stripping ratio and on the in-pit 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 in-pit resources. Material in the block modeled mineral wireframes lying outside the pit shells is reported as underground resources where resource blocks meet the resource cut-off grade of 0.075 oz/ton Au.

The resulting dipper pits were used to report in-pit Mineral Resources (Figure 14.19 through Figure 14.22).

Figure 14.19 Plan and long Section of Whittle™ Dipper Pit and North Zone Mineral Wireframes-Plan View

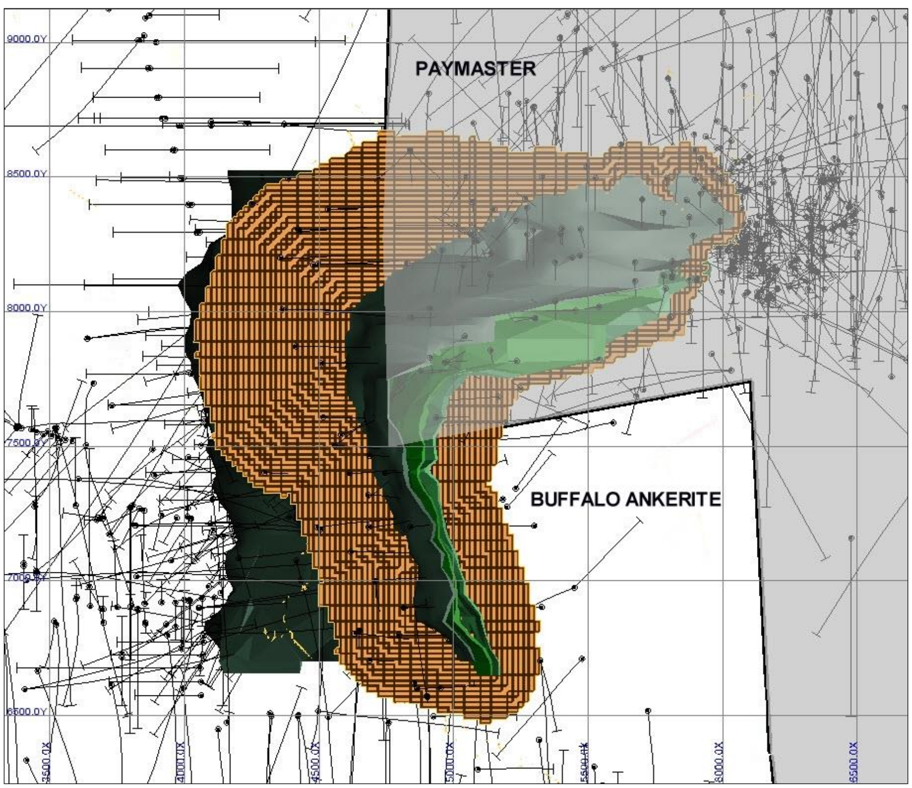
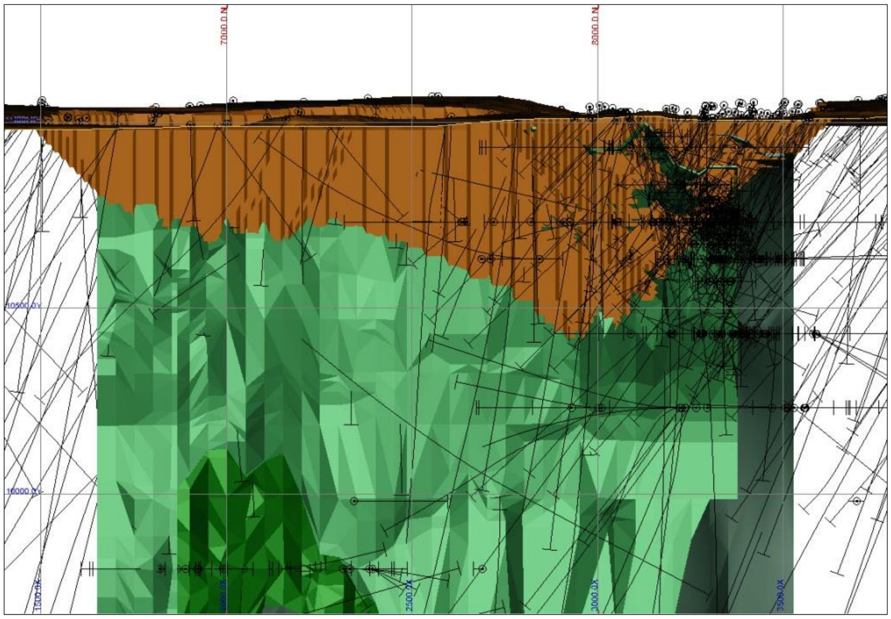


Figure 14.20 Composite Longitudinal Section Looking West



Note: *Scale in Feet*

Figure 14.21 Plan and long Section of Whittle™ Dipper Pit and South Zone Mineral Wireframes-Plan View

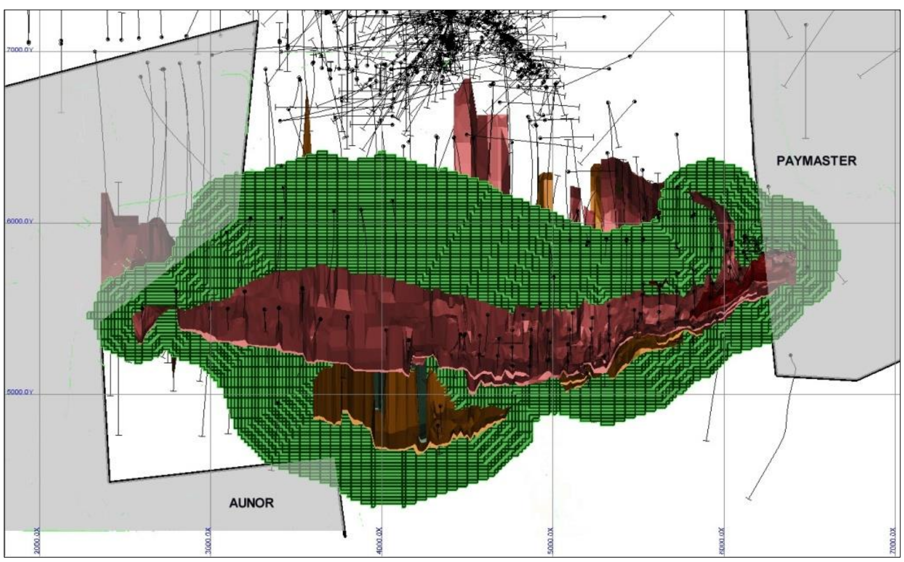
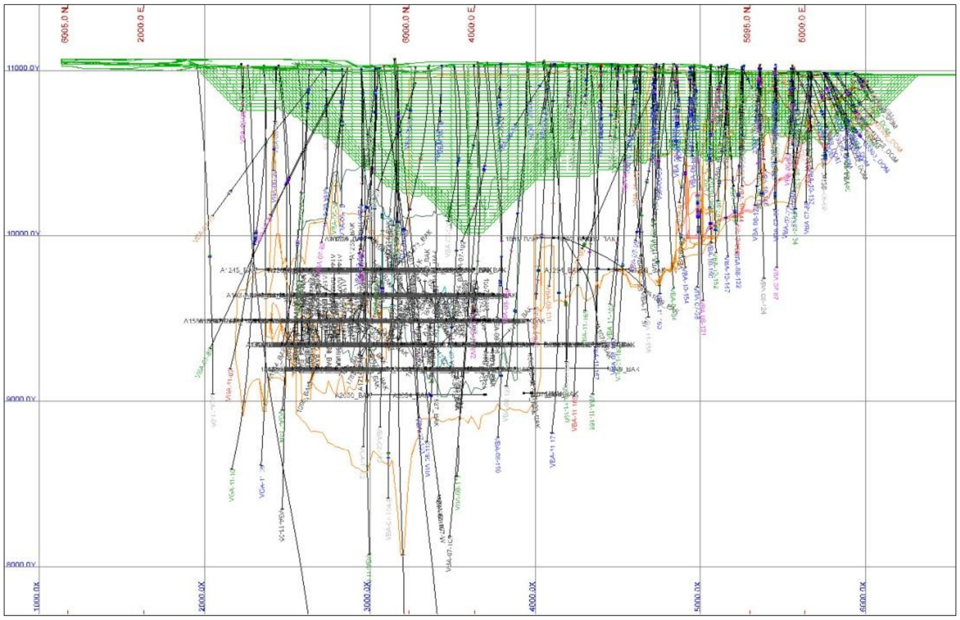


Figure 14.22 Composite Longitudinal Section Looking North



Note: *Scale in Feet*

Table 14.22 summarizes the in-pit and underground Mineral Resources for the North Zone at various cut-off grades for the Buffalo Ankerite property. Table 14.23 summarizes the in-pit and underground Mineral Resources for the North Zone at various cut-off grades on the Paymaster property. Table 14.24 summarizes in-pit and underground Mineral Resources for the South Zone on the Buffalo Ankerite property. P&E recommends that the in-pit resources at the 0.015 oz/ton Au cut-off, and the underground resource at 0.075 oz/ton Au cut-off, be used for public disclosure and further economic evaluation.

TABLE 14.22
BUFFALO ANKERITE NORTH ZONE MINERAL RESOURCES

North Zone Within Pit Resources						
Cut-Off	**Indicated Resources**			**Inferred Resources**		
	Tons	**Au**	**Au Ounces**	**Tons**	**Au**	**Au Ounces**
Au oz/ton	(000's)	oz/ton	(000's)	(000's)	oz/ton	(000's)
Wireframe	622	0.062	38.3	209	0.067	13.9
0.015	532	0.071	37.6	198	0.070	13.8
0.030	430	0.082	35.3	157	0.082	12.9
0.045	308	0.100	30.8	122	0.095	11.6
0.050	283	0.105	29.6	115	0.098	11.3
0.075	163	0.136	22.2	60.6	0.129	7.80
0.100	96.4	0.171	16.5	29.3	0.174	5.08
0.125	62.1	0.204	12.7	22.3	0.194	4.33
0.150	40.5	0.240	9.72	15.6	0.217	3.39
Buffalo Ankerite North Zone Underground Resources						
	Indicated Resources			**Inferred Resources**		
Cut-Off	**Tons**	**Au**	**Au Ounces**	**Tons**	**Au**	**Au Ounces**
Au oz/ton	(000's)	oz/ton	(000's)	(000's)	oz/ton	(000's)
Wireframe	7,535	0.058	435	4,837	0.051	249
0.015	5,748	0.074	423	3,780	0.064	242
0.030	4,338	0.090	392	2,947	0.076	223
0.045	3,232	0.108	350	2,295	0.087	199
0.050	2,919	0.115	336	2,082	0.091	189
0.075	1,779	0.149	266	1,017	0.122	124
0.100	1,219	0.186	210	440	0.170	74.8
0.125	780	0.219	171	217	0.229	49.7
0.150	562	0.251	141	128	0.294	37.7

Notes:

(1) CIM definitions were followed for Mineral Resources.

(2) The Qualified Persons for this Mineral Resource estimate are: Richard Routledge, P.Geo., and Eugene Puritch, P.Eng.

(3) Mineral Resources are estimated by conventional 3D block modelling and ID^3 grade interpolation based on wireframing at a 0.015 oz/ton Au cut-off for open pit and 0.045 oz/ton Au cut-off for underground.

(4) Gold price for the estimate is US$1,600/oz Au.

(5) A uniform bulk density of 0.0888 tons/ft^3 (2.85 tonnes/m^3) has been applied for rock volume to tons conversion. Overburden bulk density is 0.0562 tons/ft3 (1.80 tonnes/m^3).

(6) Open pit Mineral Resources are estimated from surface to the 10,360 ft elevation (650 ft or198 m depth) and underground resource below the pit to the 7,290 ft elevation, a depth of (3,740 ft (1,140 m).

(7) Classification of Indicated Resources is based on variogram ranges which indicate grade continuity of 100 ft to 150 ft, the drill hole spacing and geologic interpretation.

(8) The open pit Mineral Resource was determined within a Whittle™ pit shell with 45 degree slopes utilizing ore and waste mining costs of C$1.68/ton and $1.32/ton for overburden.

(9) Costs used to determine the resource cut-off grades were processing at $11.35/ton and G&A $3.16/ton. Gold process recovery is 90%.

TABLE 14.23
PAYMASTER NORTH ZONE MINERAL RESOURCES WITHIN PIT RESOURCES

Cut-Off Au oz/ton	Indicated Resources				Inferred Resources			
	Tons (000's)	Au oz/ton	Au Ounces (000's)	Lexam Ounces (000's)	Tons (000's)	Au oz/ton	Au Ounces (000's)	Lexam Ounces (000's)
Wireframe	2,147	0.044	95.4	57.2	106	0.036	3.78	2.27
0.015	1,702	0.054	92.1	55.3	78.5	0.046	3.58	2.15
0.030	1,141	0.07	79.5	47.7	52.8	0.057	3.03	1.82
0.045	732	0.088	64.4	38.6	28.2	0.076	2.13	1.28
0.050	633	0.094	59.6	35.8	24.9	0.079	1.97	1.18
0.075	329	0.126	41.3	24.8	13.1	0.094	1.23	0.738
0.100	195	0.135	29.7	17.8	2.96	0.133	0.393	0.236
0.125	125	0.176	22.1	13.3	2.01	0.143	0.288	0.173
0.150	78.6	0.199	15.7	9.4	0.592	0.169	0.100	0.060
Paymaster North Zone Underground Resources								
Wireframe	470	0.024	11.3	6.78	56.0	0.035	1.80	1.08
0.015	259	0.038	9.72	5.83	43.5	0.039	1.70	1.02
0.030	134	0.053	7.02	4.21	26.2	0.049	1.29	0.774
0.045	64	0.070	4.46	2.68	13.3	0.061	0.816	0.490
0.050	48.5	0.077	3.72	2.23	11.5	0.064	0.729	0.437
0.075	20.0	0.100	1.97	1.18	1.79	0.079	0.140	0.084
0.100	5.68	0.133	0.757	0.454	0.031	0.110	0.003	0.002
0.125	1.87	0.180	0.337	0.202	-	-	-	-
0.150	0.923	0.227	0.210	0.126	-	-	-	-

Notes:

(1) CIM definitions were followed for Mineral Resources.

(2) The Qualified Persons for this Mineral Resource estimate are: Richard Routledge, P.Geo., and Eugene Puritch, P.Eng.

(3) Mineral Resources are estimated by conventional 3D block modelling and ID^3 grade interpolation based on wireframing at a 0.015 oz/ton Au cut-off for open pit and 0.045 oz/ton Au cut-off for underground.

(4) Gold price for the estimate is US$1,600/oz Au.

(5) A uniform bulk density of 0.0888 tons/ft^3 (2.85 tonnes/m^3) has been applied for rock volume to tons conversion. Overburden bulk density is 0.0562 tons/ft3 (1.80 tonnes/m^3).

(6) Open pit Mineral Resources are estimated from surface to the 10,360 ft elevation (650 ft or198 m depth) and underground resource below the pit to the 7,290 ft elevation, a depth of (3,740 ft (1,140 m).

(7) Classification of Indicated Resources is based on variogram ranges which indicate grade continuity of 100 ft to 150 ft, the drill hole spacing and geologic interpretation.

(8) The open pit Mineral Resource was determined within a Whittle™ pit shell with 45 degree slopes utilizing ore and waste mining costs of C$1.68/ton and $1.32/ton for overburden.

(9) Costs used to determine the resource cut-off grades were processing at $11.35/ton and G&A $3.16/ton. Gold process recovery is 90%.

TABLE 14.24 PAYMASTER SOUTH ZONE MINERAL RESOURCES WITHIN PIT RESOURCES								
Cut-Off	**Indicated Resources**				**Inferred Resources**			
	Tons	**Au**	**Au Ounces**	**Lexam Ounces**	**Tons**	**Au**	**Au Ounces**	**Lexam Ounces**
Au oz/ton	(000's)	oz/ton	(000's)	(000's)	(000's)	oz/ton	(000's)	(000's)
Wireframe	59.3	0.070	4.17	2.50	117	0.059	6.91	4.15
0.015	57.8	0.072	4.15	2.49	113	0.061	6.88	4.13
0.030	50.9	0.079	4.00	2.40	101	0.065	6.58	3.95
0.045	34.4	0.098	3.37	2.02	67.9	0.078	5.28	3.17
0.050	32.2	0.102	3.27	1.96	62.8	0.080	5.05	3.03
0.075	21.1	0.122	2.57	1.54	37.3	0.093	3.26	1.96
0.100	9.7	0.159	1.55	0.930	6.82	0.134	0.914	0.548
0.125	6.67	0.182	1.21	0.726	3.34	0.162	0.54	0.324
0.15	4.64	0.200	0.930	0.558	2.64	0.167	0.441	0.265
South Zone Underground Resources								
Au oz/ton	(000's)	oz/ton	(000's)	(000's)	(000's)	oz/ton	(000's)	(000's)
Wireframe	0.189	0.026	0.005	0.0030	22.6	0.032	0.716	0.430
0.015	0.189	0.026	0.005	0.0030	21.0	0.033	0.695	0.417
0.030	0.017	0.037	0.001	0.0006	12.5	0.038	0.480	0.288
0.045	-	-	-		2.07	0.058	0.121	0.073
0.050	-	-	-		1.46	0.063	0.0920	0.055
0.075	-	-	-		0.141	0.117	0.0170	0.0102
0.100	-	-	-		0.131	0.120	0.0160	0.0096
0.125	-	-	-		0.088	0.129	0.0110	0.0066
0.150	-	-	-		-	-	-	-

Notes:

(1) CIM definitions were followed for Mineral Resources.

(2) The Qualified Persons for this Mineral Resource estimate are: Richard Routledge, P.Geo., and Eugene Puritch, P.Eng.

(3) Mineral Resources are estimated by conventional 3D block modelling and ID^3 grade interpolation based on wireframing at a 0.015 oz/ton Au cut-off for open pit and 0.045 oz/ton Au cut-off for underground.

(4) Gold price for the estimate is US$1,600/oz Au.

(5) A uniform bulk density of 0.0888 tons/ft^3 (2.85 tonnes/m^3) has been applied for rock volume to tons conversion. Overburden bulk density is 0.0562 tons/ft3 (1.80 tonnes/m^3).

(6) Open pit Mineral Resources are estimated from surface to the 10,360 ft elevation (650 ft or198 m depth) and underground resource below the pit to the 7,290 ft elevation, a depth of (3,740 ft (1,140 m).

(7) Classification of Indicated Resources is based on variogram ranges which indicate grade continuity of 100 ft to 150 ft, the drill hole spacing and geologic interpretation.

(8) The open pit Mineral Resource was determined within a Whittle™ pit shell with 45 degree slopes utilizing ore and waste mining costs of C$1.68/ton and $1.32/ton for overburden.

(9) Costs used to determine the resource cut-off grades were processing at $11.35/ton and G&A $3.16/ton. Gold process recovery is 90%.

14.1.14 Block Model Validation

Although mining has been carried out previously at Buffalo Ankerite, no reconciliation studies or data are available for validation of the current resource estimate. As such, estimated tonnages, grades, and contained metal have not been compared to actual production, nor has the sensitivity of the grade estimate to the drill hole density been evaluated.

The block model was validated using a number of industry standard methods including visual and statistical methods and review of the volumetrics of wireframes versus reported resources.

These methods included visual examination of assay, composite and block grades on plans and sections on-screen and review of the reasonableness of estimation parameters such as:

- Number of composites used for estimation
- Number of holes used for estimation
- Distance to the nearest composite
- Number of passes used to estimate grade.

Comparison was also made of mean grades between assays, composites, and ID^3 and NN model blocks on a global basis. The mean grades agree between resource assays and composites reasonably well when explicit and implicit missing assays/non-assayed intervals are accounted for at zero grade. The block model global grade is generally lower than the resource assay and composite grades which is expected as part of the volume variance effect and also reflects spatial distribution of holes in low grade material. The nearest neighbour block model global average grade is ±10% of the ID^3 average grade except for the South Zone below the 10,000 ft elevation. P&E notes that swath plots of south zone levels show consistently higher NN grades versus ID^3 from the 9820 ft EL to 9020 ft EL which includes the area of underground drilling, production stopes and declining number of interpolated blocks. The smoothing of ID^3 in an area of mixed higher grades and zero grade composites (due to lack of assaying), likely accounts for the lower ID^3 model grade and variance with the NN model grade.

TABLE 14.25
VALIDATION COMPARISONS

	North Zone	**North Zone**	**South Zone**	**South Zone**
	Above 10,000 ft EL Au (oz/ton)	**Below 10,000 ft EL Au (oz/ton)**	**Above 10,000 ft EL Au (oz/ton)**	**Below 10,000 ft EL Au (oz/ton)**
Assays[1]	0.054/0.044	0.099/0.066	0.068/0.063	0.127/0.074
Composites	0.045	0.066	0.066	0.075
ID^3 Blocks[2]	0.047	0.060	0.062	0.069
NN Blocks[2]	0.052	0.059	0.063	0.079
Variance[3]	-10%	2%	-2%	-13%

Notes:
(1) Lower grade from dilution at zero grade for gaps in assaying
(2) Domains not adjusted for mined stopes
(3) ID^3 versus NN

14.2 FULLER DEPOSIT

14.2.1 Mineral Resource Estimate

Project Summary

RPA prepared an updated Mineral Resource estimate for the Fuller Property with the effective date of May 22, 2013. The previous Mineral Resource estimate was completed by Wardrop Engineering Inc. (Wardrop, now Tetra Tech) in 2007 and reported in a Technical Report on the property prepared for VG Gold Corporation (Wardrop, 2007). Fifty-three additional drill holes have been completed on the property since the Wardrop estimate.

Lexam provided RPA with the current drill hole database as well as density measurements. Lithology and mineralization wireframes interpreted by Wardrop in the previous estimate, as well as composite samples used in the Wardrop estimation, were also provided to RPA. The current estimate includes data from both historical and recent drilling and underground sampling. Assay results for all drilling had been received at the time of the estimate.

The updated Mineral Resource estimate is based on 3D block modelling utilizing Datamine Studio 3 and Gemcom GEMS 6.5 software. The Mineral Resources are unconstrained by wireframes: the block model was constrained by dynamic search angles and a constrained ellipse in the across strike direction. Dynamic angles used to dictate the orientation of the axes of the search ellipse were created by means of structural wireframe surfaces and strike and dip polylines representing the strike and dip of the main mineralized structural fabric. The block model and drill holes were domained coincidently with grade interpolation by means of a probabilistic constraining technique to aid the validation of resulting estimates.

The Fuller open pit (OP) and underground (UG) Mineral Resource estimate is summarized in Table 14.26. The open pit resource is constrained within a preliminary pit shell. Resources located outside the pit shell are reported as underground resources. The Qualified Person for the Fuller Mineral Resource estimate is Katharine Masun, P.Geo., Senior Geologist with RPA. The effective date of the estimate is May 22, 2013.

Note that all measurements stated herein are imperial measurements, i.e., tonnage in short tons, metal content in ounces per short tons, coordinates in feet, density in short tons per cubic foot.

TABLE 14.26
MINERAL RESOURCE ESTIMATE – MAY 22, 2013

Lexam VG Gold Inc. - Fuller Project				
Classification	**Cut-off Grade (opt Au)**	**Tonnage (000 tons)**	**Grade (opt Au)**	**Contained Metal (000 oz Au)**
OP				
Indicated	≥0.015	5,878	0.049	290
Inferred	≥0.015	2,981	0.038	112
UG				
Indicated	≥0.075	361	0.168	61
Inferred	≥0.075	930	0.145	135
Total Indicated		**6,239**	**0.056**	**351**
Total Inferred		**3,911**	**0.063**	**247**

Notes:
(1) CIM definitions were followed for Mineral Resources.
(2) Mineral Resources are estimated at a cut-off grade of 0.015 opt Au for OP and 0.075 opt Au for UG.
(3) Mineral Resources are estimated using a gold price of US$1,600 per ounce, and a US$/C$ exchange rate 1:1.
(4) Numbers may not add due to rounding.

14.2.2 Mineral Resource Database

Lexam provided RPA with an up-to-date Gemcom project for the Fuller Property in November 2012. Data for drill holes completed on the property subsequent to this was received as Excel spreadsheets and imported by RPA into the Gemcom project. Lithological modelling and volumetrics were completed in Gemcom GEMS 6.5 and the block modelling and grade estimation, in Datamine Studio 3.

The drill hole database contains underground and surface holes completed by various operators throughout the history of exploration and limited gold production on the Fuller Property (see Section 6). Holes were drilled in a variety of orientations depending on the local orientation of the foliation. Drill holes typically range from less than 100 feet to more than 2,000 feet in length, the longer holes belonging to more recent drilling campaigns. The shorter holes generally were drilled from underground.

Drilling on the Fuller Property covers an approximate area of 2,400 feet (E-W) by 1,600 feet (N-S), with an average drill hole spacing of less than 50 feet within the hinge of the syncline to more than 150 feet along the eastern and western flanks. Further detail on drilling can be found in Section 10 Drilling. A drill location plan map for the Fuller deposit is contained in Appendix I.

The current Fuller Project database comprises 657 drill holes, which includes 280 surface and 377 underground drill holes, totalling approximately 378,000 feet of drilling and 144,284 assay feet (35,672 samples). Of these, 558 drill holes (206 surface and 352 underground) for approximately 291,024 feet of drilling and 290,803 assayed feet (33,393 samples) were used in Mineral Resource estimation. From the original database supplied by Lexam, RPA discarded 18 records related to underground drift back samples and sludge test hole samples, ranging in length from four feet to 41 feet (approximately 292 feet), which resulted in the removal of 19 samples with an average grade of 0.182 opt from the database used for Mineral Resource estimation.

14.2.3 Dynamic Anisotropy and Probabilistic Domains

The Mineral Resource estimation for the Fuller Property utilized dynamic anisotropy to constrain the interpolation. The ellipse minor axis was restricted to marginally less than one block width during the first search pass so as not to oversmooth block estimates by the inclusion of grades from neighbouring samples in the across strike direction (minor axis). Dynamic anisotropy relies on the interpretation by the resource modeller to derive angles that will subsequently orient the search ellipse on a per block basis. The dynamic anisotropy models are often visually more appealing and realistic than techniques such as “unwrinkling” or “unfolding” as they avoid artifacts of back-transformation. The dynamic search and estimation strategies are detailed later in this section.

Since the constraints used for modelling are purely directional and are not hard-coded in the block model, RPA created a probability model based off of an indicator at a cut-off grade near the pit discard cut-off grade. A probability of being above the 0.015 opt open pit cut-off grade of 0.2 was chosen as a domain boundary. Blocks with probabilities greater than 0.2 were flagged as Zone 1, while blocks with probabilities less than 0.2 were flagged as Zone 0. The raw assays and composites were then back-flagged by the interpolated probabilities. The technique allowed RPA to validate the block model within spatial, geological, and statistically homogeneous domains as opposed to a globally unconstrained validation exercise.

14.2.4 Geological and Structural Models

RPA used dynamic anisotropy to mimic the tightly folded stratigraphy on the Fuller resource. RPA created wireframe surfaces (structural or trend surfaces) from polylines on plan view, using drill hole assays, guided by the structural interpretation completed by Wardrop in 2007. In addition, RPA adjusted the interpretation to honour the digitized underground drifts and stopes. The dip and dip direction angles calculated for each wireframe triangle were interpolated into the block model using nearest neighbour and a large spherical ellipse. For a first pass, gold grades were interpolated into the block model based on these angles and the same search strategy was used for grade interpolation for the final model. The grade model created was then used to refine the dynamic angles by means of polylines on plan views and vertical sections perpendicular to the apparent local strike of the deposit.

The structural wireframes and polylines used to calculate the dynamic angles are shown in Figure 14.23 to Figure 14.24.

Figure 14.23 Structural Wireframe Surfaces for Dynamic Angles

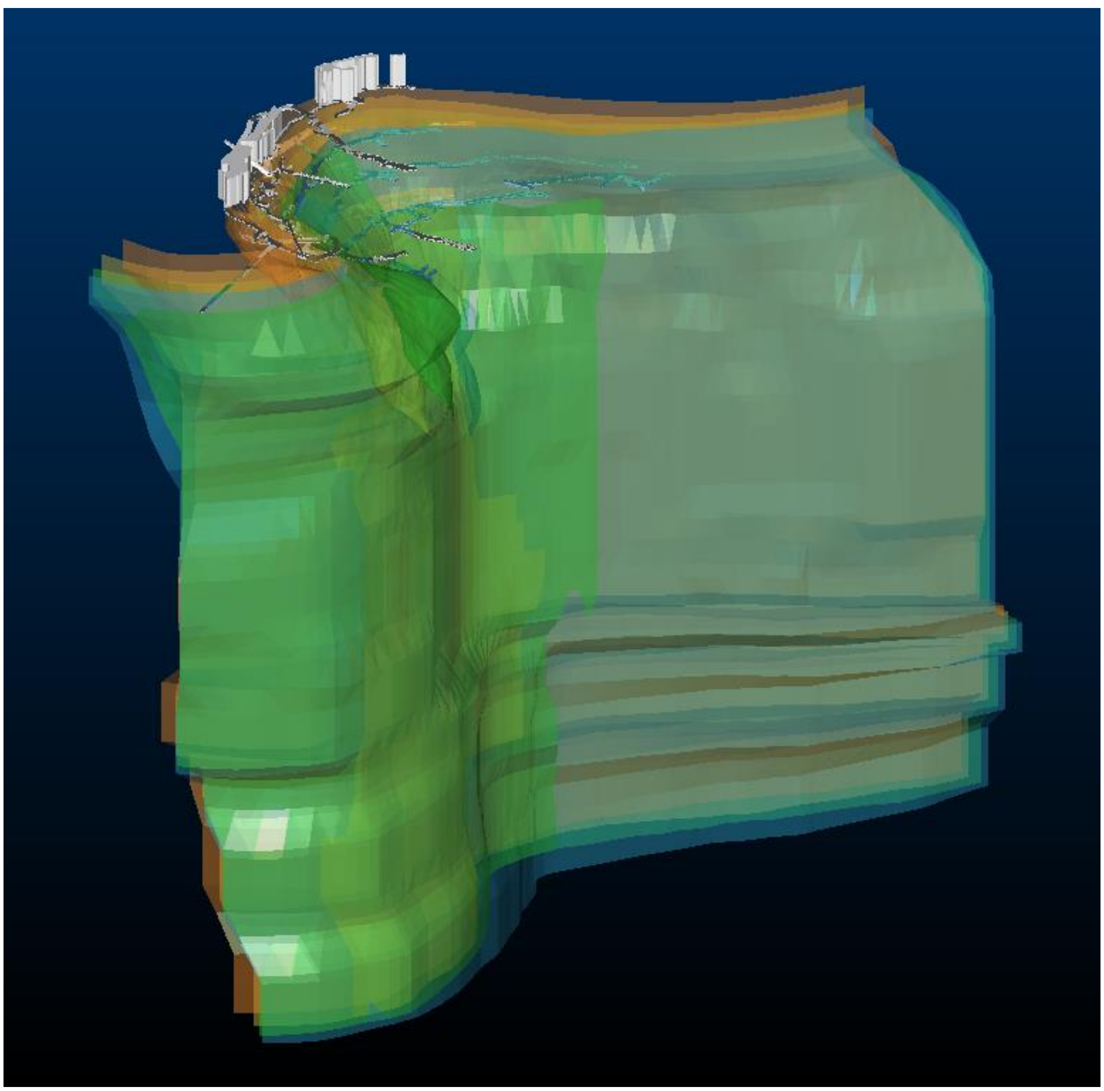
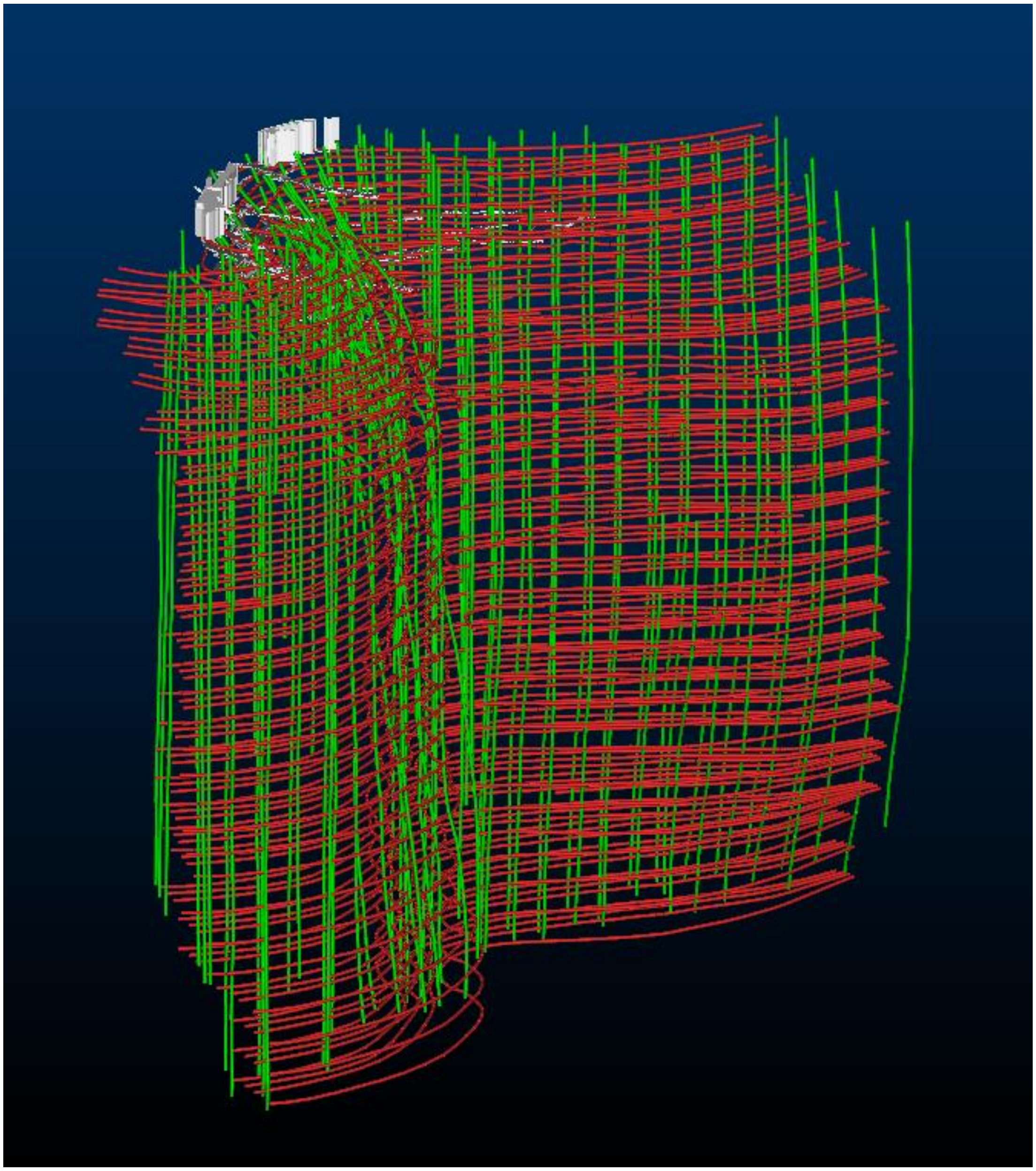


Figure 14.24 Structural Polylines for Dynamic Angles



14.2.5 Assay Statistics

The probabilistic domaining technique was used to flag assays falling inside (Zone 1) and outside (Zone 0) of predominantly mineralized areas. Length weighted univariate statistical analysis was performed on the uncut resource assay values and summarized in Table 14.27.

Table 14.27 Descriptive Statistics of Uncut Gold Resource Assays		
Lexam VG Gold Inc. – Fuller Project		
	Length (feet)	**Au (opt)**
	Zone 0	
No. of Cases	23,563	23,563
Minimum	0.030	0.000
Maximum	645.570	7.130
Mean	9.444	0.008
Median	5.000	0.001
Standard Deviation	22.017	0.088
Coefficient of Variation	2.331	11.065
	Zone 1	
No. of Cases	8,126	8,126
Minimum	96.450	0.000
Maximum	3.964	21.650
Mean	3.500	0.069
Median	4.121	0.010
Standard Deviation	1.040	0.444
Coefficient of Variation	0.050	6.397
	Other*	
No. of Cases	1,704	1,704
Minimum	0.100	0.000
Maximum	227.330	4.160
Mean	21.168	0.008
Median	5.000	0.001
Standard Deviation	39.513	0.105
Coefficient of Variation	1.866	12.835

**assays too far away from blocks and not flagged*

14.2.6 Cutting of High Assays

Where the assay distribution is skewed positively or approaches lognormal, erratic high grade assay values can have a disproportionate effect on the average grade of a deposit. One method of treating these outliers in order to reduce their influence on the average grade is to cut or cap them at a specific grade level. In the absence of production data to calibrate the cutting level, inspection of the assay distribution can be used to estimate a first pass cutting level.

The distribution of high grade outliers in the Fuller Mineral Resource assay population warranted the application of high grade cutting. Decile analysis, cumulative density function, log histogram, and inspection of the spatial distribution of the 90th to the 95th and the 95th to the 100th cumulative probabilities of the resource assays, including inspection of the outliers within the entire sample distribution as well as within the Zone 1 probability domain, indicated that a capping value of 1.0 opt was appropriate. Resource assays were cut to 1.0 opt prior to compositing, which had the effect of reducing the coefficient of variation and mean average assay grade.

Table 14.28 summarizes statistics on the resource assay data after cutting to 1.0 opt.

TABLE 14.28 SUMMARY STATISTICS OF CUT GOLD RESOURCE ASSAYS		
Lexam VG Gold Inc. – Fuller Project		
Domain	**Number of Cut Assays to 1.0 opt**	**% Metal Cut**
Zone 0	9	7%
Zone 1	58	18%
Total	67	15%
Zone 1		
No. of Cases	8,126	
Minimum	0.000	
Maximum	1.000	
Mean	0.054	
Median	0.010	
Standard Deviation	0.128	
Coefficient of Variation	2.358	
Zone 0		
No. of Cases	23,563	
Minimum	0.000	
Maximum	1.000	
Mean	0.007	
Median	0.001	
Standard Deviation	0.036	
Coefficient of Variation	5.067	

14.2.7 Compositing

Capped assays were composited over the full length of the drill holes. Composite lengths were chosen based on the selectivity at the proposed scale of mining. For the OP portion of the block model, samples were composited to 12 foot lengths, while for the UG block model, samples were composited to six feet. RPA treated unsampled core intervals as having a zero gold grade.

Table 14.29 summarizes statistics on the 12 foot composite data and Table 14.30 summarizes statistics on the six foot composite data.

Table 14.29 Open Pit 12 foot Composite Statistics	
Lexam VG Gold Inc. – Fuller Project	
	Au (opt)
	Zone 0
No. of Cases	18,340
Minimum	0.0
Maximum	0.5134
Mean	0. 0024
Standard Deviation	0. 0129
Coefficient of Variation	5.37
	Zone 1
No. of Cases	2,863
Minimum	0.0
Maximum	0.745
Mean	0.037
Standard Deviation	0.061
Coefficient of Variation	1.63

Table 14.30 Underground 6 foot Composite Statistics	
Lexam VG Gold Inc. – Fuller Project	
	Au (opt)
	Zone 0
No. of Cases	36,722
Minimum	0
Maximum	0.846
Mean	0. 002
Standard Deviation	0. 015
Coefficient of Variation	6.24
	Zone 1
No. of Cases	5,691
Minimum	0.0
Maximum	0.835
Mean	0.037
Standard Deviation	0.077
Coefficient of Variation	2.05

14.2.8 Variography

RPA used six foot composites capped at 1.0 opt for variography. Downhole variograms indicated a very high nugget effect (C_0=0.53) and a range of approximately 40 feet. RPA used the nugget effect from the downhole variogram and attempted to model the west and east limbs of the Fuller syncline and the hinge separately. The direction of the longest range was generally in the order of 100 feet, but the resulting models were unsatisfactory as they conflicted with geology and the orientations of the mined-out stopes. When determining search ellipse neighbourhoods, RPA chose to use visual inspection of grade continuity on plans and sections and the size, shape, and orientation of the mined-out stopes.

14.2.9 Block Model and Grade Estimation Procedures

Dimensions and Coding

Two block models were created to cover the area of interest within the Fuller deposit. A block size of 12 feet x 12 feet x 12 feet was used to estimate Mineral Resources within the open pit and a block size of 6 feet x 6 feet x 6 feet was chosen for the block model used to estimate Mineral Resources outside the open pit, i.e., for the underground portion of the Fuller deposit. These two block sizes reflect the proposed scale of mining for open pit and underground. The two block model extents are summarized in Table 14.31 and descriptions of the block model attributes are given in Table 14.32.

TABLE 14.31
OPEN PIT AND UNDERGROUND BLOCK MODEL EXTENTS

Lexam VG Gold Inc. – Fuller Project		
Attribute	**OP**	**UG**
Block Model Name	FULLERDA	FULLER_RPA
Block Size (feet)	12 x 12 x 12	6 x 6 x 6
X Origin*	1,541	1,901
Y Origin*	8,677	8,797
Z Origin*	11,149	11,149
Number In X	287	503
Number In Y	222	396
Number In Z	250	499
Number Of Blocks	15,928,500	99,394,812

**origin location given in the Fuller local grid coordinate system*

TABLE 14.32
OPEN PIT AND UNDERGROUND BLOCK MODEL ATTRIBUTES

Lexam VG Gold Inc. – Fuller Project	
FULLERDA (OP)	
Attribute Name	**Description**
Rock Type OP	Coded 100 for all blocks inside pit shell with Au grade >0 (used in volumetrics)
Rock Type	Rock codes from lithology model
Density	Bulk density coded from lithology model
Percent	Percent block within underground openings
AU_C	Capped estimated gold values
AU_UC	Uncapped estimated gold values
AU_NN	Nearest neighbour estimated gold values
Class_Ind	Classification (1=Indicated; 2=Inferred; 0=no class)
IND	Interpolated indicator
ZONE	Probabilistic domain
SVOL	Estimation pass number

TABLE 14.32 OPEN PIT AND UNDERGROUND BLOCK MODEL ATTRIBUTES	
Lexam VG Gold Inc. – Fuller Project	
FULLER_RPA (UG)	
Attribute Name	Description
Rock Type Minzone	Coded 100 for all blocks outside of pit shell with Au grade >0 (used in volumetrics)
Rock Type	Rock codes from lithology model
Density	Bulk density coded from lithology model
Percent	Percent block within underground openings
AU_C	Underground capped estimated gold values for 6 foot composites
Class_Ind	Classification (1=Indicated; 2=Inferred; 0=no class)
UG	Coded 1 for all blocks outside of pit shell
OP	Coded 1 for all blocks within pit shell
Zone	Probabilistic domain
SVOL	Estimation pass number

Grade Interpolation

The estimation of gold grades was carried out using inverse distance to the 5th power (ID5) while the probabilistic indicator was estimated using inverse distance squared. Grades were interpolated using 12 ft composites for OP and 6 ft composites for UG, and both assays capped at 1 opt. The dynamic angles facilitated grade estimation that honours orientations of the geology while a limited across-strike or minor axis reduced the potential for over-smoothing of grades laterally. The primary search ranges for the major and semi-major axes was determined from 100 foot average spacing of drill sections. A nearest neighbour (NN) estimate was run in parallel to the ID5 estimate for validation purposes.

The search and estimation strategy for the three pass interpolation is given in Table 14.33.

TABLE 14.33 INTERPOLATION PARAMETERS		
Lexam VG Gold Inc. – Fuller Project		
Estimation Pass	**OP**	**UG**
Estimation Pass 1:		
Samples		
Min samples used	2	2
Max samples used	10	10
Max samples per hole	1	1
Distances		
Range Major	100	100
Range Semi-Major	100	100
Range Minor	5	2.5
Ellipsoid Orientation		
Principal Azimuth (degrees)	Dynamic	Dynamic
Principal Dip (degrees)	Dynamic	Dynamic

TABLE 14.33 INTERPOLATION PARAMETERS		
Lexam VG Gold Inc. – Fuller Project		
Estimation Pass	**OP**	**UG**
Intermediate Azimuth (degrees)	0	0
Estimation Pass 2:		
Samples		
Min samples used	2	2
Max samples used	10	10
Max samples per hole	1	1
Distances		
Range Major	200	200
Range Semi-Major	200	200
Range Minor	10	5
Ellipsoid Orientation		
Principal Azimuth (degrees)	Dynamic	Dynamic
Principal Dip (degrees)	Dynamic	Dynamic
Intermediate Azimuth (degrees)	0	0
Estimation Pass 3:		
Samples		
Min samples used	2	2
Max samples used	10	10
Max samples per hole	1	1
Distances		
Range Major	300	300
Range Semi-Major	300	300
Range Minor	15	7.5
Ellipsoid Orientation		
Principal Azimuth (degrees)	Dynamic	Dynamic
Principal Dip (degrees)	Dynamic	Dynamic
Intermediate Azimuth (degrees)	0	0

Block Model Validation

RPA validated block estimates by a global comparison between NN and ID5 grades, visual inspection of vertical and plan sections, and swath plots of ID5 versus NN estimates.

The global comparison of NN and ID5 grades indicates the effectiveness of the overall block estimates at reproducing the declustered composite statistics (Table 14.34). Screen shots of sections and plan views of composite versus block grades are shown in Figure 14.25 and Figure 14.26.

Table 14.34 ID^5 Versus NN Comparison				
Lexam VG Gold Inc. – Fuller Project				
Domain	**Method**	**Average (opt)**	**Variance (opt)**	**Std Dev (opt)**
OP Model				
Zone 0	NN	0.0007	0.0000	0.0033
	ID5	0.0011	0.0000	0.0033
Zone 1	NN	0.0321	0.0029	0.0541
	ID5	0.0296	0.0016	0.0397
UG Model				
Zone 0	NN	0.0010	0.0001	0.0084
	ID5	0.0014	0.0000	0.0068
Zone 1	NN	0.0282	0.0043	0.0654
	ID5	0.0263	0.0024	0.0491

Figure 14.25 Plan view of Block Estimates at the 10,800 foot elevation

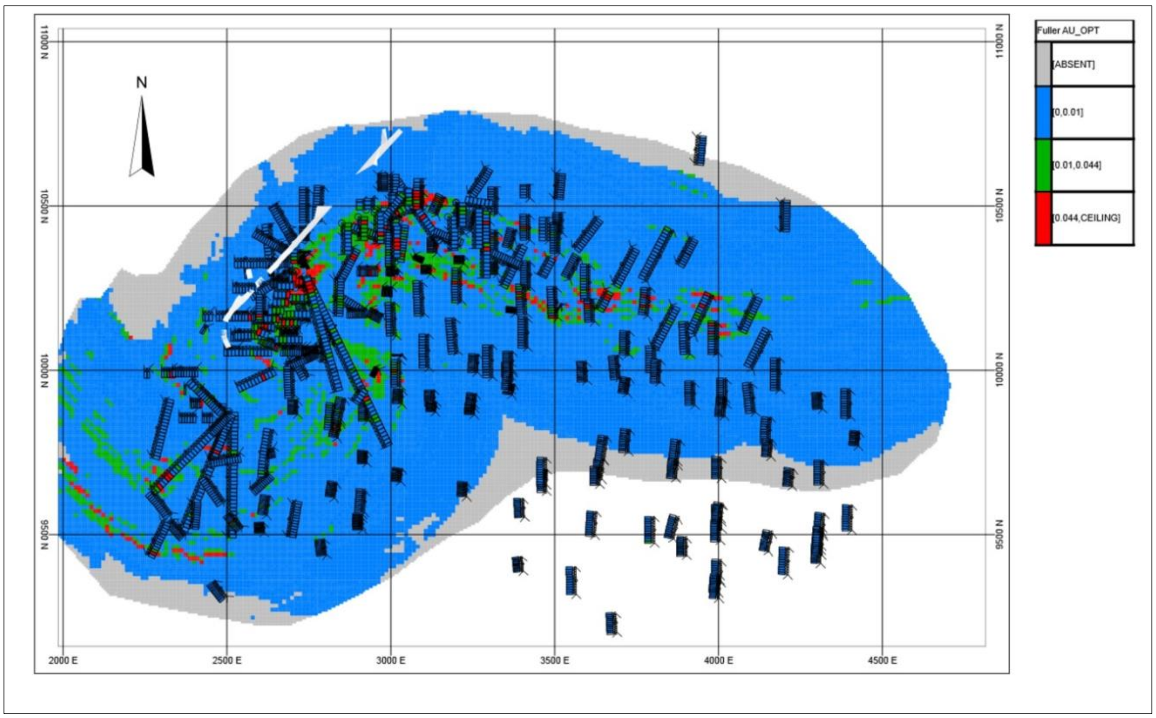
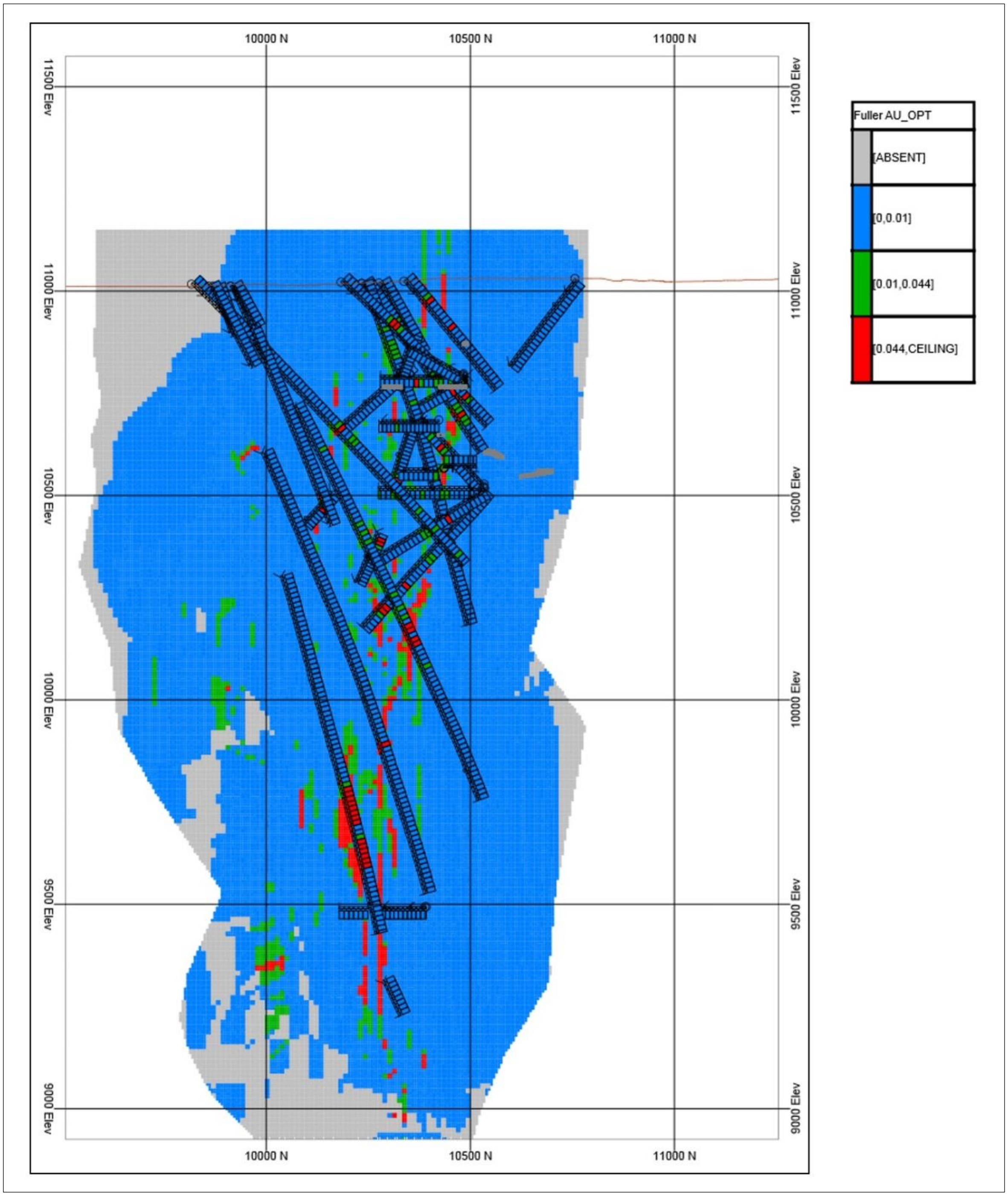


Figure 14.26 Vertical section showing Block Estimates at 3,220 E



14.2.10 Bulk Density

To convert volumes to tons, a simplified lithological model was created in GEMS with the following rock types: Porphyry (901/902), V3 (Mafic – 903), V4 (Ultramafic – 904), and Diabase (905). A bulk density factor was assigned for each lithology by determining the mean value of each rock type from bulk density testing carried out on drill core by Lexam from 2011 and 2012. Each block in the two block models was coded with a lithology rock type by majority rules. Blocks that fell outside of the lithological model were coded as Waste (99) and a default bulk density of 0.0885 ton/ft^3 (the average of V3 and V4) was applied. Resource bulk density statistics are summarized in Table 14.35.

TABLE 14.35 RESOURCE BULK DENSITY MEASUREMENTS	
Lexam VG Gold Inc. – Fuller Project	
	Porphyry
No. of Cases	45
Minimum	0.082 ton/ft^3
Maximum	0.090 ton/ft^3
Mean	0.084 ton/ft^3
Standard Deviation	0.002 ton/ft^3
Coefficient of Variation	0.018 ton/ft^3
	V3
No. of Cases	89
Minimum	0.083 ton/ft^3
Maximum	0.104 ton/ft^3
Mean	0.088 ton/ft^3
Standard Deviation	0.002 ton/ft^3
Coefficient of Variation	0.026 ton/ft^3
	V4
No. of Cases	40
Minimum	0.084 ton/ft^3
Maximum	0.091 ton/ft^3
Mean	0.089 ton/ft^3
Standard Deviation	0.002 ton/ft^3
Coefficient of Variation	0.017 ton/ft^3
	Diabase
No. of Cases	7
Minimum	0.086 ton/ft^3
Maximum	0.095 ton/ft^3
Mean	0.091 ton/ft^3
Standard Deviation	0.004 ton/ft^3
Coefficient of Variation	0.044 ton/ft^3

RPA recommends that more work be undertaken on determination of bulk density for the mineralized zones and wall rock within the Fuller deposit.

14.2.11 Classification

The Mineral Resource classification used in this report is consistent with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2010) as incorporated by reference in NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction". Mineral Resources are classified into Measured, Indicated, and Inferred categories according to the level of confidence in the geological information available on the mineral deposit.

Classification for the Fuller Mineral Resource estimate was guided by search distance criteria, relative continuity of grade, overall sample density, and confirmation of the geological interpretation by the existing underground drifts and stopes.

The first pass minimum criteria for Indicated Resources are as follows:

- The block is estimated during the first search pass corresponding to the average sectional drill spacing of 100 feet.
- The block falls within the Zone 1 probability domain suggesting continuity of grade.
- The estimation of grade locally is consistent with the orientation of underground openings.

Using the above criteria, RPA visually inspected the classification of blocks on vertical and plan sections both within the pit for the open pit portion of the Mineral Resource and outside the pit for the underground portion of the Mineral Resource. RPA concluded that the preliminary classification continuity was not acceptable (see Figure 14.27 and Figure 14.28). Using the first pass criteria outlined above as a guide and incorporating both drill hole spacing (average of 100 foot spacing at a minimum) and visual continuity of grade above cut-off, RPA manually wireframed classification solids for Indicated Mineral Resources. This exercise was completed for both the open pit and the underground Mineral Resources. The blocks in each block model that fell within the Indicated classification solids were flagged as Indicated and the remaining classified blocks were flagged as Inferred. Figure 14.29 and Figure 14.30 highlight the final classified blocks in the 12 foot (OP) and six foot (UG) block models using the Indicated classification solids.

Figure 14.27 Preliminary Classified Blocks at the 10,800 Foot Level Using Probabilistic Domaining

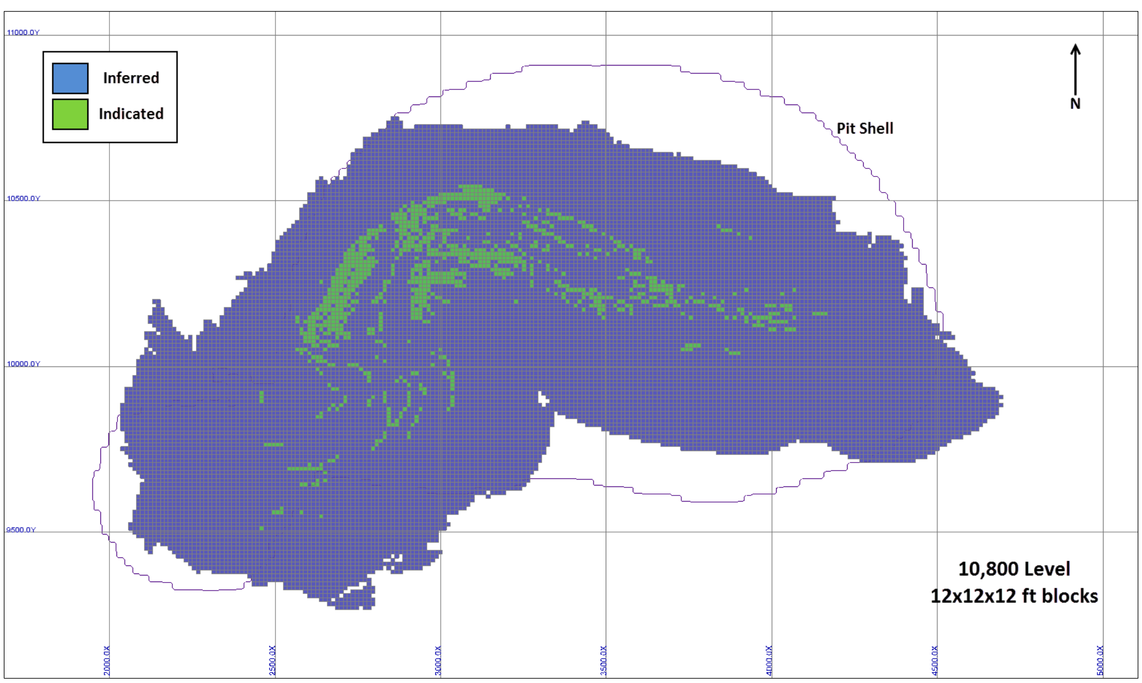


Figure 14.28 Classified blocks at the 9,700 Foot Level Using Probabilistic Domaining

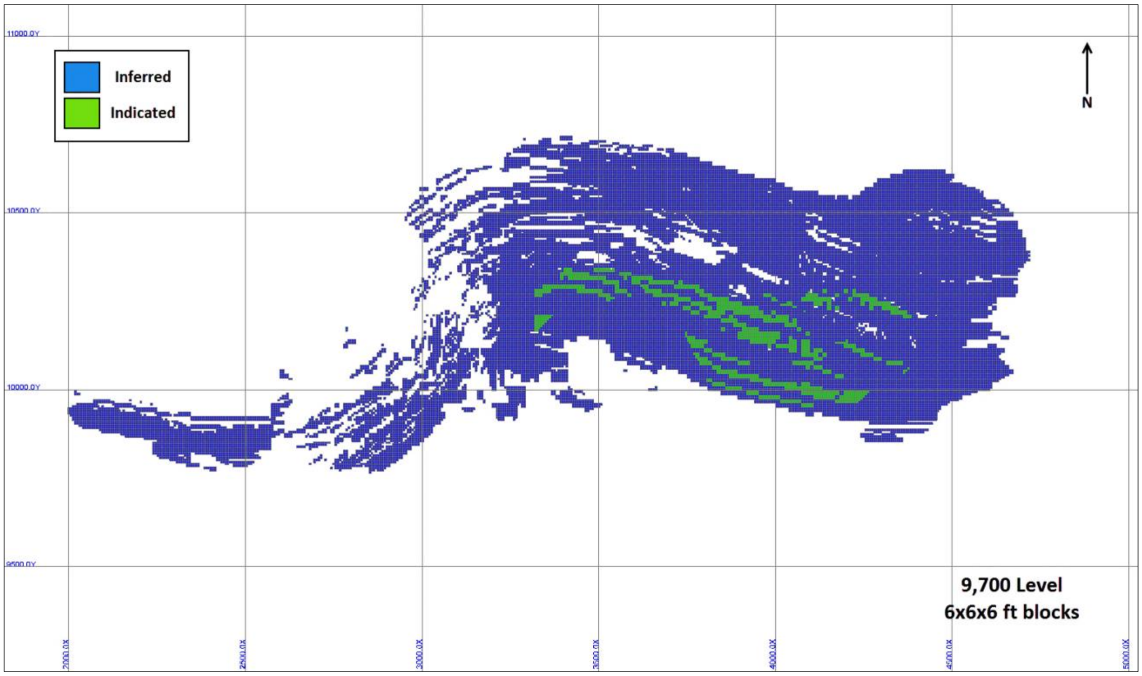
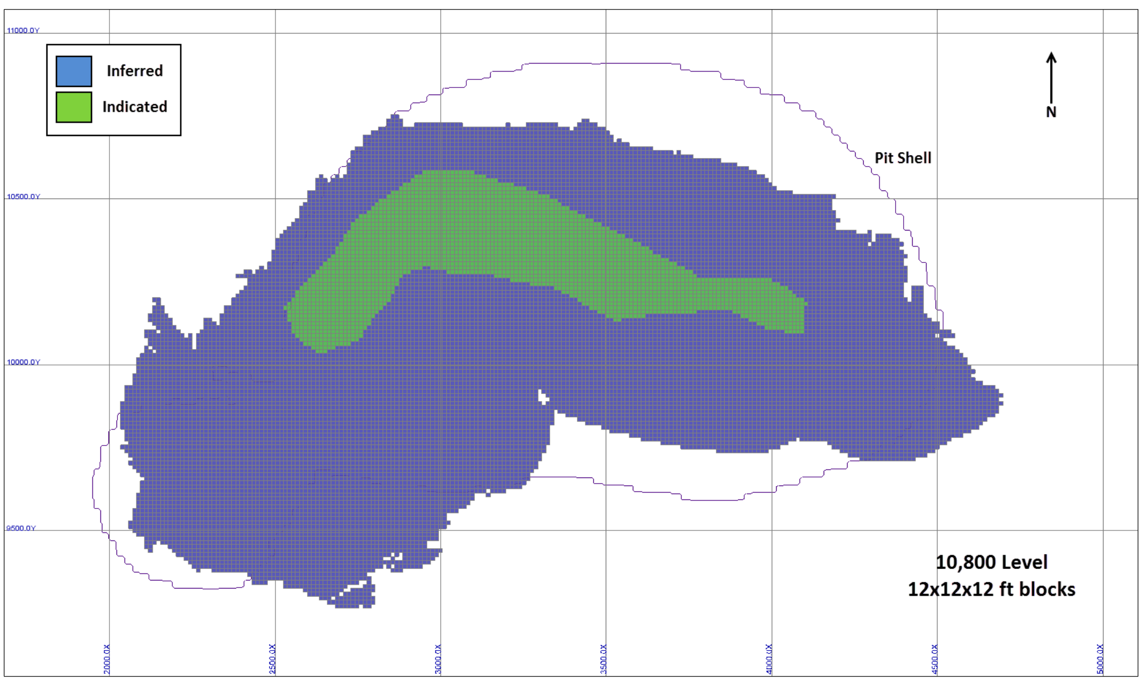
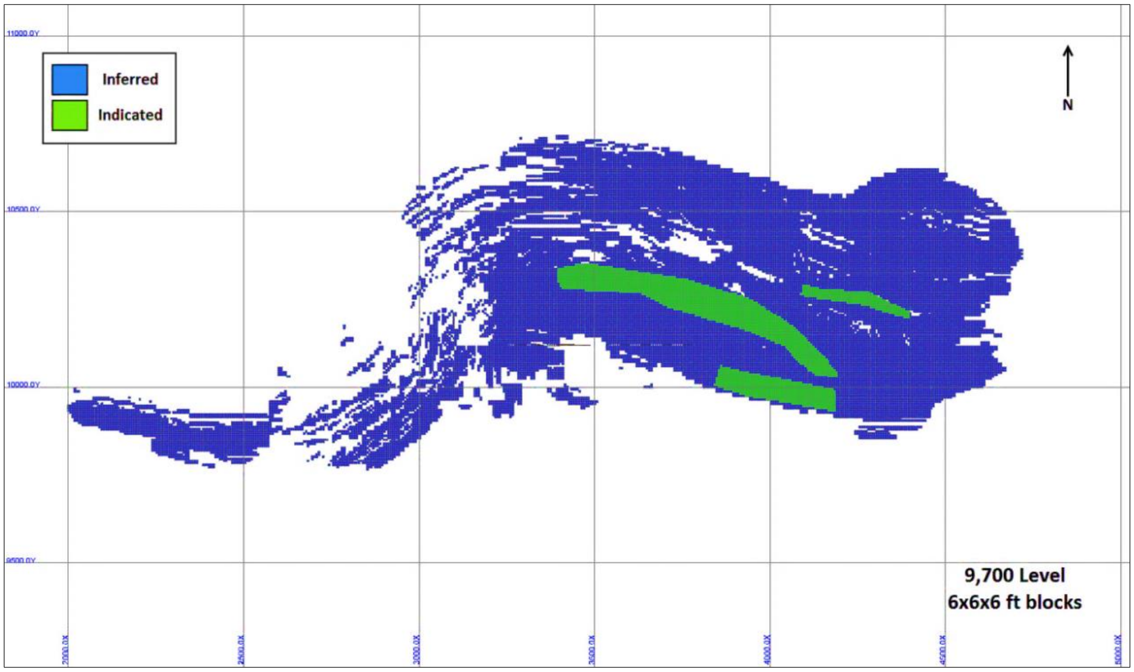


Figure 14.29 Final Classified Blocks at the 10,800 Foot Level Using Classification Solids



Note. *All classified 12x12x12 foot blocks are shown, regardless of grade.*

Figure 14.30 Classified Blocks at the 9,700 Foot Level Using Classification Solids



Note: *All classified 12x12x12 foot blocks are shown, regardless of grade*

14.2.12 Removal of Mined-out Areas

RPA received solids representing the underground workings and digitized mined-out stopes from plan views and longitudinal sections. RPA subtracted the mined-out portions from the Mineral Resources but was unable to independently validate the extent of underground mining. Using a cut-off grade of 0.015 opt Au, RPA removed 105,000 tons and 8,314 ounces of gold from the Fuller Mineral Resource.

14.2.13 Open Pit Optimization

The block model generated in Gemcom GEMS was transferred to Whittle software for the preliminary pit optimization work. The block model was reblocked to a block size of 24 feet x 24 feet x 24 feet to reduce the number of blocks and implicitly the processing time. The parameters used in the preliminary optimization process are listed in Table 14.36.

TABLE 14.36
PRELIMINARY PIT OPTIMIZATION FACTORS

Lexam VG Gold Inc. - Fuller Project	
Pit Slopes	-50°
Mining Cost	$1.68/ton (converted from $1.85/tonne)
Process Cost	$16.33/ton (converted from $18.00/tonne)
G&A	$4.54/ton (converted from $5.00/tonne)
Recovery	90%
Gold Price	US$1,600/oz
Block Size (reblocked)	24 x 24 x 24 feet

The revenue factor 1 pit was then transferred to Gemcom GEMS for open pit resource reporting and served as a limit for underground resource reporting.

14.2.14 Cut-off Grades

The open pit resources were reported using a 0.015 opt cut-off grade. The underground resources were reported using a 0.075 opt cut-off grade based on the costs in Table 14.37 and US$1,600/oz gold price and an underground mining cost of approximately $88/ton.

14.2.15 Sensitivity Analysis

The OP and UG Mineral Resources are sensitive to the cut-off grade. For the OP, sensitivity is most pronounced in the 0.015 opt to 0.03 opt Au range, then moderately sensitive for grades higher than 0.03 opt Au. UG Mineral Resources are most sensitive in the 0.015 opt Au to 0.06 opt Au range, then moderately sensitive for grades higher than 0.06 opt Au. Table 14.37 presents the tonnage, grade, and ounces at various cut-off values. Figure 14.31 and Figure 14.32 show the grade-tonnage curves for the OP and UG Indicated Resources and for the OP and UG Inferred Resources.

Figure 14.31 OP and UG Indicated Resources – Grade–Tonnage Curves

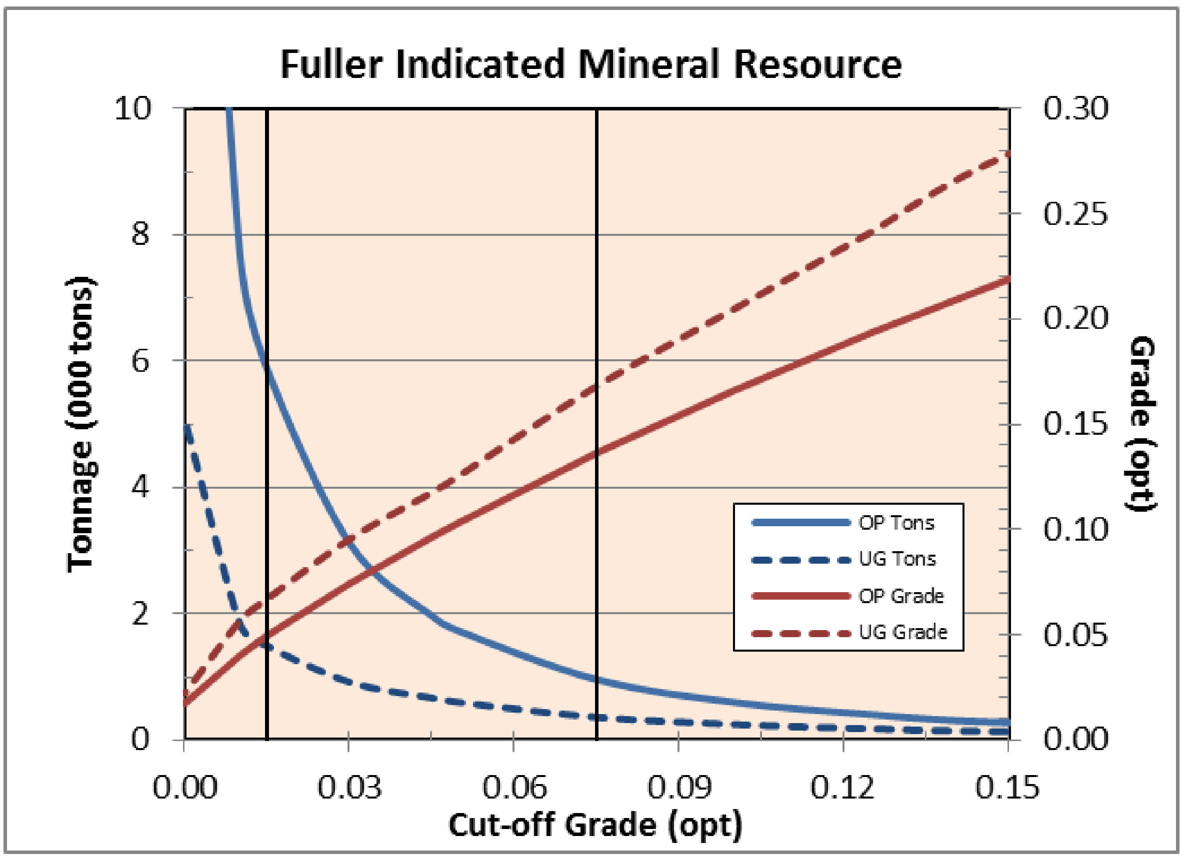


Figure 14.32 OP and UG Inferred Resources – Grade–Tonnage Curves

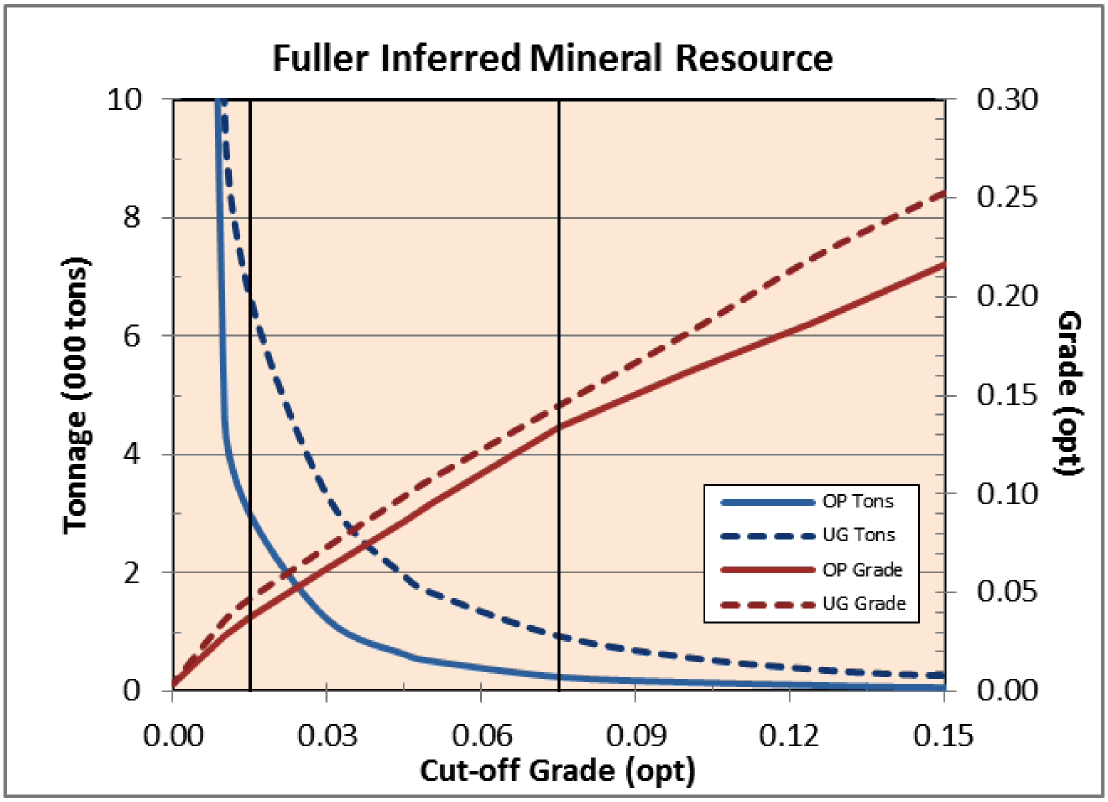


TABLE 14.37 GRADE AND TONNAGE AT VARIOUS CUT-OFFS				
Lexam VG Gold Inc. - Paymaster Project				
Classification	**Cut-off (opt Au)**	**Tonnage (tons)**	**Grade (opt Au)**	**Gold (ounces)**
OP Indicated	≥0.320	24,851	0.636	9,327
	≥0.150	276,800	0.375	60,698
	≥0.125	402,752	0.219	77,925
	≥0.100	598,081	0.193	99,594
	≥0.075	957,507	0.167	130,732
	≥0.050	1,723,742	0.137	177,403
	≥0.045	1,966,405	0.103	188,908
	≥0.030	3,117,676	0.096	231,011
	≥0.015	5,878,200	0.074	289,920
OP Inferred	≥0.320	4,726	0.350	1,653
	≥0.150	64,979	0.216	14,068
	≥0.125	100,591	0.188	18,898
	≥0.100	152,374	0.162	24,663
	≥0.075	242,723	0.134	32,491
	≥0.050	513,551	0.095	48,613
	≥0.045	623,502	0.086	53,827
	≥0.030	1,212,894	0.062	75,489
	≥0.015	2,980,525	0.038	111,953

TABLE 14.37 **GRADE AND TONNAGE AT VARIOUS CUT-OFFS**				
Lexam VG Gold Inc. - Paymaster Project				
Classification	**Cut-off (opt Au)**	**Tonnage (tons)**	**Grade (opt Au)**	**Gold (ounces)**
UG Indicated	≥0.320	35,349	0.467	16,509
	≥0.150	131,983	0.279	36,758
	≥0.125	178,517	0.242	43,121
	≥0.100	248,725	0.205	50,961
	≥0.075	360,760	0.168	60,699
	≥0.050	596,976	0.126	75,089
	≥0.045	666,044	0.118	78,369
	≥0.030	918,197	0.096	87,700
	≥0.015	1,492,259	0.067	100,039
UG Inferred	≥0.320	53,500	0.459	24,562
	≥0.150	265,454	0.253	67,141
	≥0.125	366,994	0.221	81,023
	≥0.100	571,895	0.182	103,896
	≥0.075	930,400	0.145	134,826
	≥0.050	1,672,367	0.107	179,740
	≥0.045	1,945,892	0.099	192,696
	≥0.030	3,302,817	0.073	242,243
	≥0.015	6,642,912	0.047	312,534

14.2.16 Comparison with Previous Resource Estimate

The Mineral Resource estimate for the Fuller Project reported in the August 2007 Technical Report on the Fuller Gold Property by Wardrop is compared with the current estimate in Table 14.38.

TABLE 14.38 **COMPARISON WITH AUGUST 2007 RESOURCE ESTIMATE**					
Lexam VG Gold Inc. - Fuller Project					
Year	**Classification**	**Cut-off Grade (opt Au)**	**Tonnage (000 tons)**	**Grade (opt Au)**	**Contained Metal (000 oz Au)**
2013	OP				
	Indicated	≥0.015	5,878	0.049	290
	Inferred	≥0.015	2,981	0.038	112
	UG				
	Indicated	≥0.075	361	0.168	61
	Inferred	≥0.075	930	0.145	135
	Total Indicated		6,239	0.056	351
	Total Inferred		3,911	0.063	247
2007	Total Indicated	≥0.075	1,475	0.160	236
	Total Inferred	≥0.075	1,813	0.165	300

The 2007 estimation was done by interpolating grades using Ordinary Kriging (OK) into 14 geological solids in seven zones. Wardrop used composite lengths of up to 10 feet and did not discard any composites, regardless of length. The capping value applied in the 2007 estimate was 1.0 opt Au and the reporting cut-off value was 0.075 opt Au.

Additional drilling was available for the 2013 resource estimate. Mineralized intercepts from 53 new drill holes for both open pit and underground domains were used in the current estimate.

Total Indicated and Inferred Mineral Resources in the current estimate at a cut-off grade of 0.075 opt average 0.145 opt and 0.143 opt, respectively, or 9% and 13% lower than the 2007 estimate. The current Indicated and Inferred Mineral Resources, at a 0.075 opt cut-off grade, total 191,000 ounces and 167,000 ounces of gold, or 19% and 44% fewer ounces, respectively, than the 2007 estimate.

The difference represents the cumulative effect of different interpolation methods, different compositing strategies, and the extra drilling.

In order to support mine planning, areas of the Fuller block models constructed by RPA for underground and open pit resources require refinement and RPA recommends adding trend lines and surfaces locally to better align the orientation of the search ellipse.

Infill drilling should be carried out on the periphery of the deposit, where drilling is more widely spaced, to upgrade the Mineral Resource classification in these areas.

14.3 PAYMASTER DEPOSIT

Project Summary

RPA prepared an updated Mineral Resource estimate for the Paymaster PropertyThe previous Mineral Resource estimate was completed by Kenneth Guy and Peter Bevan in 2010 and reported in a Technical Report on the property prepared for VG Gold Corporation (Guy and Bevan, 2010). Twenty-four additional drill holes have been completed on the property since the 2010 estimate.

The updated Mineral Resource estimate for the Paymaster Project is summarized in Table 14.39. The estimate was carried out using Gemcom GEMS 6.4 in two stages. Initially, an open pit (OP) resource was estimated using a lower gold cut-off grade, and then an underground (UG) resource was defined below the pit shell, at a higher gold cut-off grade. The Mineral Resources were classified as Indicated and Inferred, with all of the Indicated Resources located within the open pit. The Qualified Person for the Paymaster Mineral Resource estimate is Tudorel Ciuculescu, M.Sc., P.Geo., Senior Geologist with RPA. The effective date of the Paymaster Mineral Resource estimate is May 22, 2013.

Note that all measurements stated in this section are imperial measurements, i.e., tonnage is in short tons, metal content in ounces per short tons, coordinates in feet, density in short tons per cubic foot.

TABLE 14.39 MINERAL RESOURCE ESTIMATE – MAY 22, 2013					
Lexam VG Gold Inc. - Paymaster Project					
Classification	**Cut-off Grade (opt Au)**	**Tonnage (tons)**	**Grade (opt Au)**	**Gold (ounces)**	**Lexam Ounces**
OP					
Indicated	≥0.015	5,135,000	0.047	242,000	145,000
Inferred	≥0.015	1,542,000	0.047	72,000	43,000
UG					
Indicated	-	-	-	-	
Inferred	≥0.075	239,000	0.179	43,000	26,000
Total Indicated		**5,135,000**	**0.047**	**242,000**	**145,000**
Total Inferred		**1,781,000**	**0.065**	**115,000**	**69,000**

Notes:

(1) CIM definitions were followed for Mineral Resources.

(2) Mineral Resources are estimated at a cut-off grade of 0.015 opt Au for OP and 0.075 opt Au for UG.

(3) Mineral Resources are estimated using a gold price of US$1,600 per ounce, and a US$/C$ exchange rate 1:1.

(4) A minimum mining width of approximately 20 ft was used for OP and approximately 5 ft for UG.

(5) Numbers may not add due to rounding.

A nominal minimum horizontal width of 20 feet was used as a guide for the OP and five feet for the UG. The largest OP mineralized wireframe straddles the existing stopes, while the rest of the mineralized wireframes are mostly parallel to the former. The UG wireframes are narrower and some of them represent the higher grade core of the OP wireframes situated below the pit shell.

14.3.1 Mineral Resource Database

The drill hole database for the resource estimate contains the Placer Dome drilling (1995-1996) and the VG Gold/Lexam drilling (2009-2012). The database contains 263 drill holes with a total drilled length of 217,977.56 ft and has a total of 21,439 samples representing 91,734.43 feet. The resource data consists of 13,052.08 feet from 145 drill holes for the open pit and 826.2 feet from 35 drill holes for the underground. There is some overlap between the underground intercepts and the open pit intercepts.

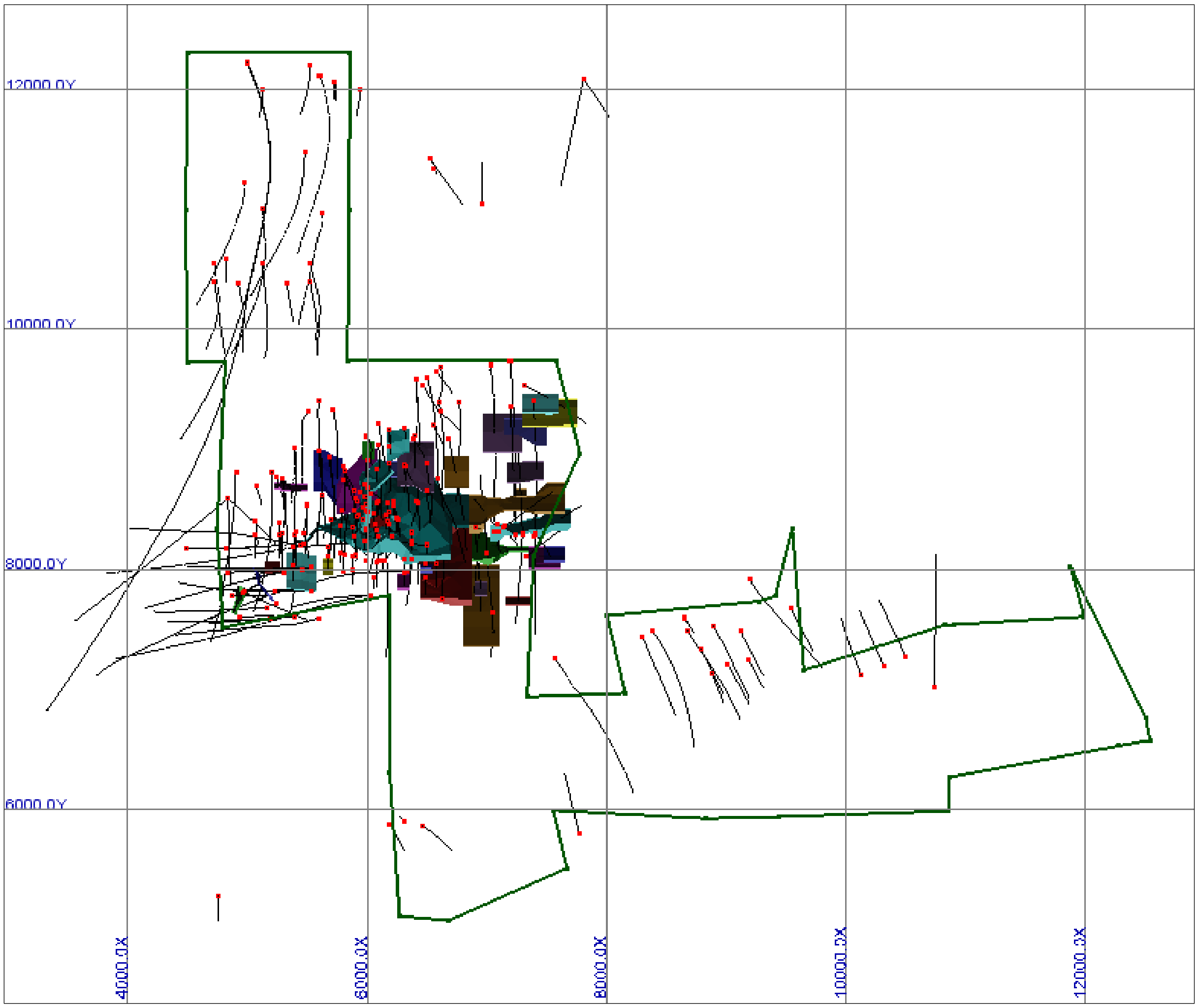
For the current resource estimate, RPA did not consider the data from historical holes drilled in the 1920s and 1950s on the Paymaster property. Closely spaced pairs of older and recent composites indicated a low similarity between the two different generations of drill holes, as described in Section 12 Data Verification.

A drill location plan map for the Paymaster deposit is contained in Appendix I.

14.3.2 Geological Interpretation and 3D Solids

For the open pit exercise, the focus was on the central part of the property. Mineralized wireframes were built at approximately a 0.010 opt Au cut-off value over a nominal 20 feet horizontal width. The main mineralized solid is oriented along the existing underground stopes, oriented east-west (solid 114). Mineralized solids with various orientations were also modelled. The stope and underground development solid was provided by Lexam.

Figure 14.33 OP Resource Wireframes, Drill Holes and Property Boundary

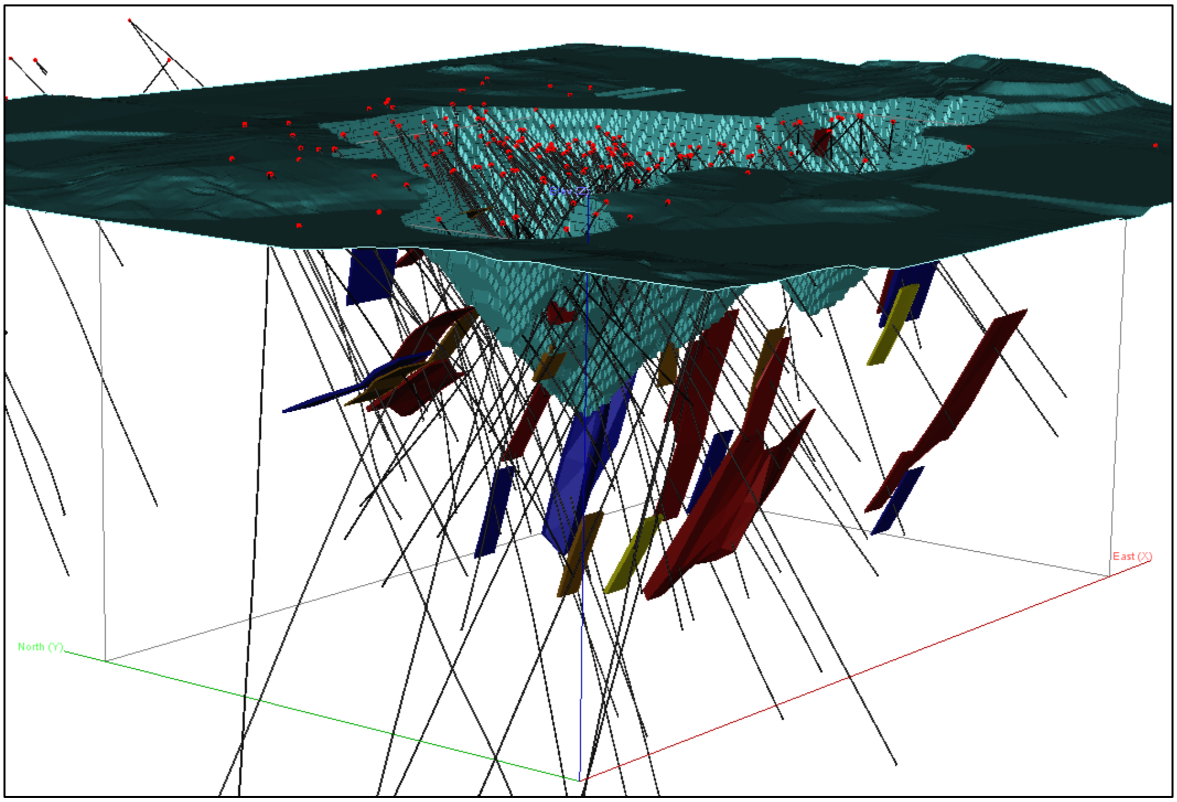


Quartz feldspar porphyry, mafic, and ultramafic lithological domains were modelled, as well as the overburden.

The underground wireframes were modelled for the mineralized material below the pit shell developed for the open pit estimate. A cut-off value of approximately 0.043 opt Au over nominal five foot horizontal width was used as a guide for wireframing. A few lower grade intercepts were included to preserve continuity. The resulting wireframes were either higher grade cores of the deep OP solids or new, thin, higher-grade lenses.

The pit shell reaches 900 foot depth below surface, while the UG wireframes reach a depth of 1,500 feet below surface. Figure 14.33 shows the OP wireframes, resource drill holes, and property boundary. The pit shell and UG wireframes are shown in Figure 14.34.

Figure 14.34 UG Resource Wireframes, Drill Holes and the Pit Shell



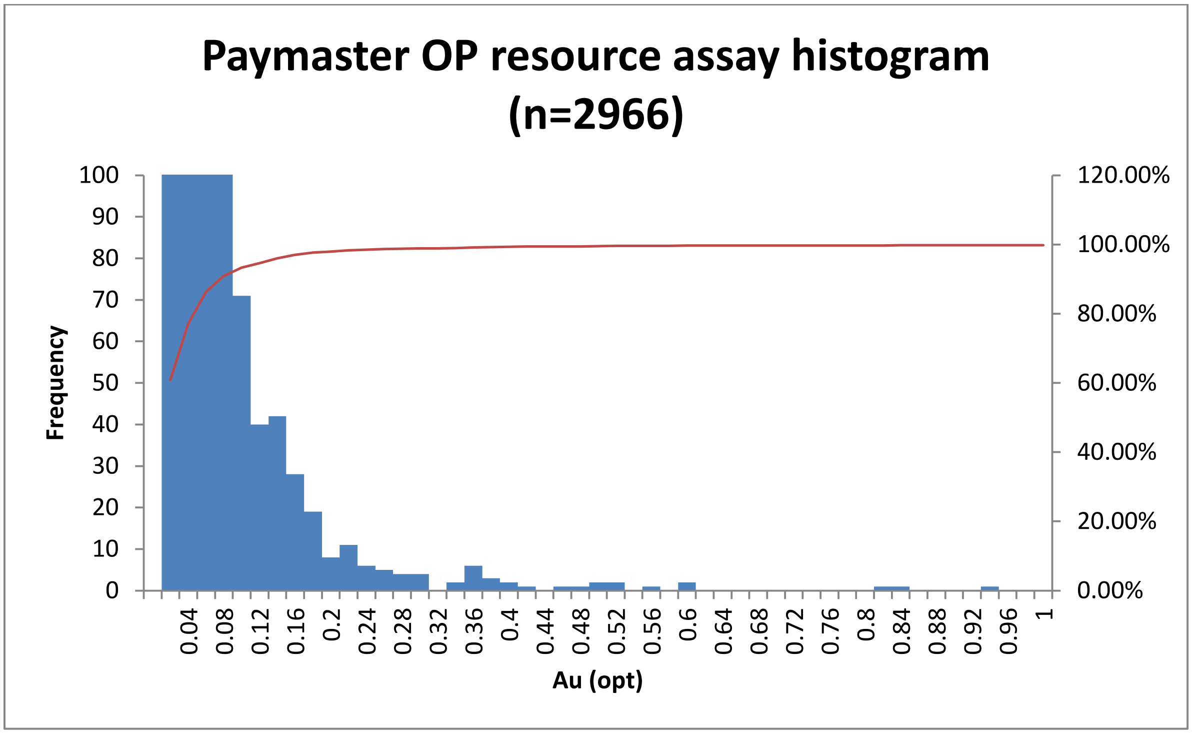
14.3.3 Basic Statistics

<u>OP Resource Assays</u>

The descriptive statistics of the OP resource assays are presented in Table 14.40. The histogram of the OP resource assays is shown in Figure 14.35.

TABLE 14.40 OPEN PIT ASSAYS		
Lexam VG Gold Inc. - Paymaster Project		
	AU (opt)	**AU (opt)**
Count	2,966	2,966
Mean	0.041	0.035
Median	0.015	0.015
Std Dev	0.223	0.064
Minimum	0.000	0.000
Maximum	10.675	0.700

Figure 14.35 OP Resource Assay Histogram

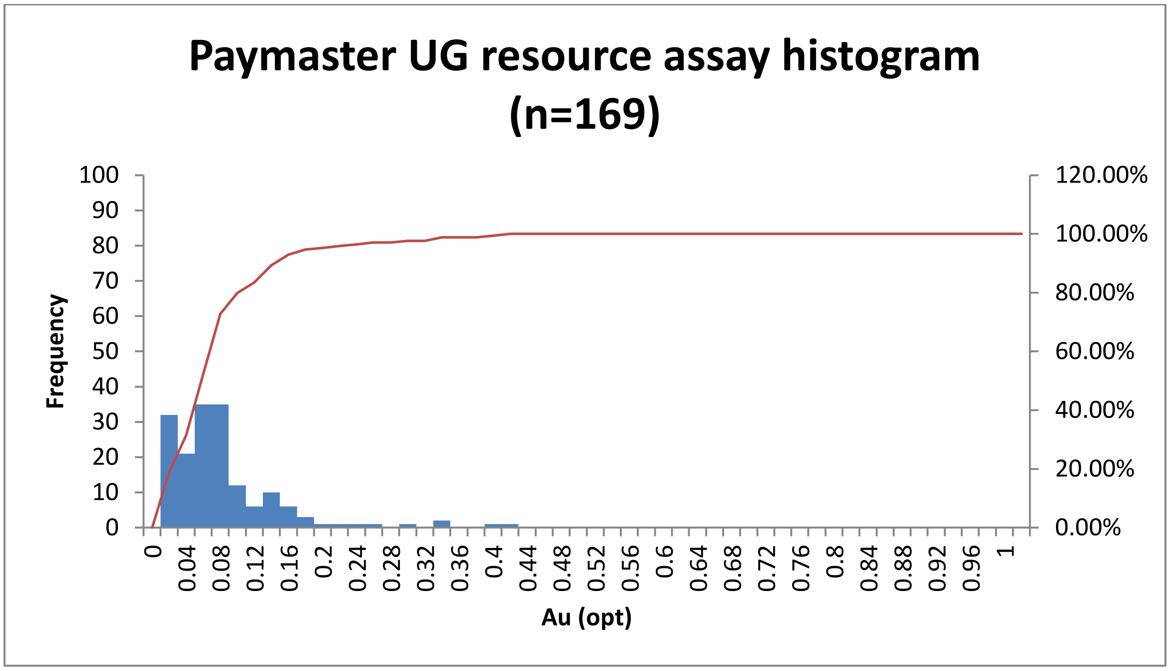


14.3.4 UG Resource Assays

The descriptive statistics of the UG resource assays are presented in Table 14.41. The histogram of the UG resource assays is shown in Figure 14.36.

TABLE 14.41 UNDERGROUND ASSAYS		
Lexam VG Gold Inc. - Paymaster Project		
	AU(opt)	**CAU(opt)**
Count	169	169
Mean	0.071	0.071
Median	0.059	0.059
Std Dev	0.069	0.069
Minimum	0.000	0.000
Maximum	0.410	0.410

Figure 14.36 UG Resource Assay Histogram



14.3.5 Capping of High Assays

Decile analysis, cumulative frequency log probability plots, and histograms of the resource assays for the OP indicate a capping value of 0.7 opt Au. A total of 10 samples in the OP intercepts were capped and this reduced the average gold grade of the OP resource assays by approximately 20%.

The UG resource samples had assay values below the OP capping level. Consequently, it was considered that no capping was necessary for the UG.

14.3.6 Composites

OP Composites

The open pit intercepts were composited at five foot fixed lengths, from collar to toe. The intercepts have internal dilution representing samples with grades below the wireframing cut-off, resulting in some very low grade full length composites inside the wireframe and a number of higher grade orphans at the footwall of the mineralized wireframe. The descriptive statistics (Table 14.42) indicate that orphans shorter than 50% composite length have mean values for capped gold grade larger by approximately 50% than all the composites and orphans together. By filtering out the composites grading less than 0.010 opt Au, representing internal dilution, the mean values for all composites greater than 0.010 opt Au and for orphans shorter than 50% composite length become similar, indicating that removal of short orphans is not necessary. Consequently, RPA used all 2,803 composites.

TABLE 14.42 COMPOSITES AND ORPHANS						
Lexam VG Gold Inc. - Paymaster Project						
	Composites 2.5 to 5.0 ft			**Orphans 0 to 2.5 ft**		
	Length (ft)	**Au (opt)**	**Cut Au (opt)**	**Length (ft)**	**Au (opt)**	**Cut Au (opt)**
Count	2,704	2,704	2,704	99	99	99
Mean	4.95	0.040	0.032	1.21	0.051	0.049
Median	5.00	0.017	0.017	1.20	0.026	0.026
Std Dev	0.29	0.21	0.05	0.68	0.10	0.08
Minimum	2.50	0.000	0.000	0.02	0.000	0.000
Maximum	5.00	8.561	0.700	2.40	0.933	0.700
CV	0.06	5.25	1.56	0.56	1.96	1.63
	All composites 0 to 5.0 ft			**Composites Au>0.010 opt**		
	Length (m)	**Au (opt)**	**Cut Au (opt)**	**Length (m)**	**Au (opt)**	**Cut Au (opt)**
Count	2803	2803	2803	1912	1912	1912
Mean	4.81	0.040	0.033	4.76	0.057	0.046
Median	5.00	0.018	0.018	5.00	0.030	0.030
Std Dev	0.76	0.21	0.05	0.86	0.25	0.06
Minimum	0.02	0.000	0.000	0.02	0.010	0.010
Maximum	5.00	8.561	0.700	5.00	8.561	0.700
CV	0.16	5.25	1.52	0.18	4.39	1.30

14.3.7 UG Composites

The UG resource composites are full drill hole intercepts inside the higher grade set of wireframes. Table 14.43 presents descriptive statistics of the UG composites.

TABLE 14.43 UNDERGROUND COMPOSITES		
Lexam VG Gold Inc. - Paymaster Project		
	AU(opt)	**CAU(opt)**
Count	63	63
Mean	0.088	0.088
Median	0.067	0.067
Std Dev	0.069	0.069
Minimum	0.003	0.003
Maximum	0.398	0.398

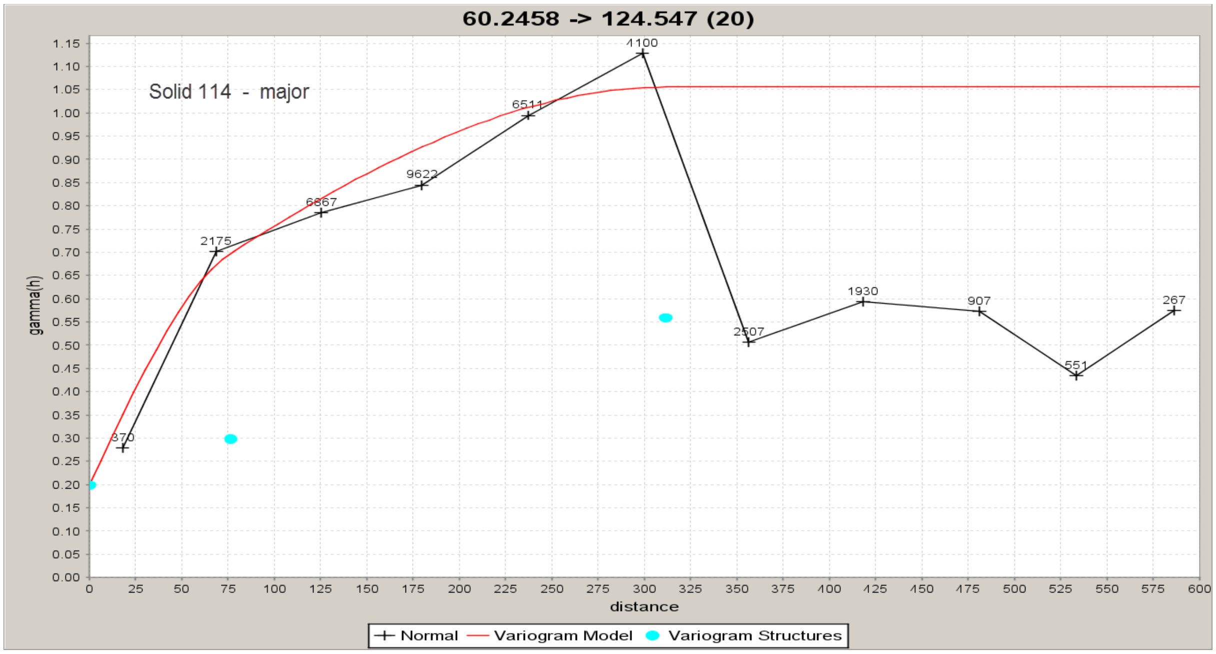
14.3.8 Variography

Variographic analysis was performed on Paymaster composites for both OP and UG domains to determine the maximum ranges of the search ellipses used in the interpolation stages.

OP Variography

Composites from the main mineralized body on the Paymaster property and straddling the underground developments, labelled solid 114, were used for variographic analysis. RPA interpreted ranges of up to 300 ft (Figure 14.37) in the plane of the solid, however, the two longest ranges were not orthogonal, most likely due to the drilling pattern.

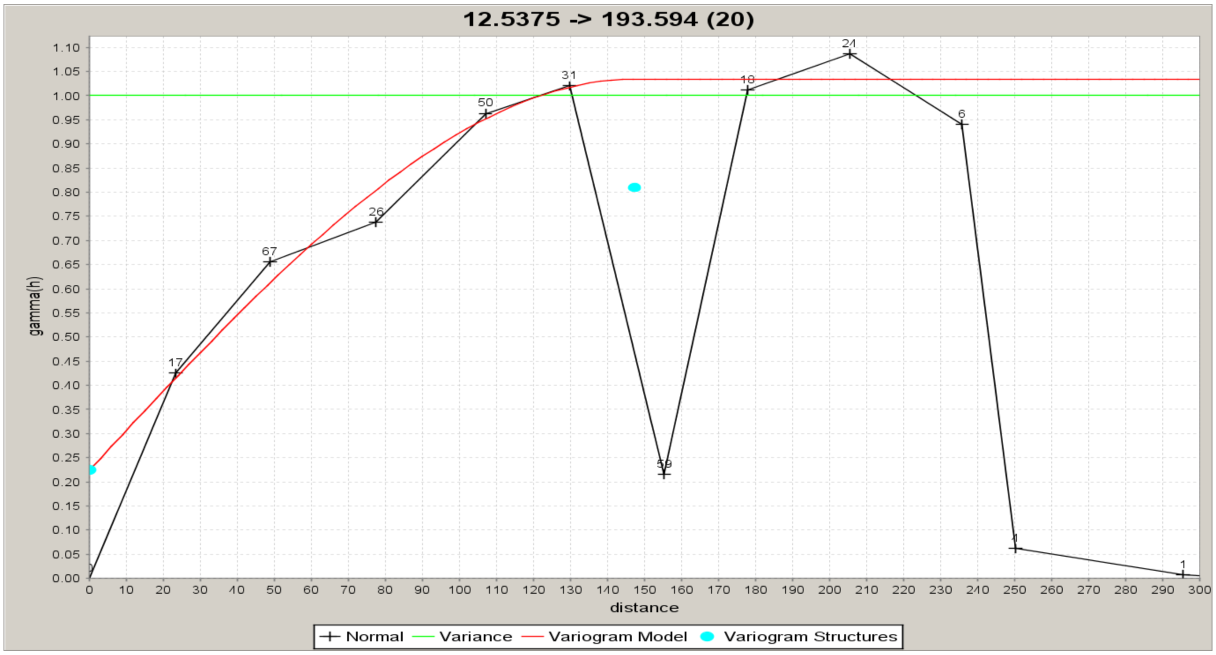
Figure 14.37 OP Major Axis for Solid 114



UG Variography

The modelled UG solids had a small number of intercepts and generally had large drill hole spacing. The variogram using the composites from the main mineralized solid (114) with gold values higher than 0.04 opt Au showed ranges of up to 150 feet (Figure 14.38).

Figure 14.38 UG Major Axis for Higher Grade Composites in Solid 114



14.3.9 Specific Gravity

Specific gravity (SG) measurements were performed on core from the 2009-2012 drilling on the Paymaster Property. The average values by lithology are presented in Table 14.44.

TABLE 14.44 DENSITY TESTWORK			
Lexam VG Gold Inc. - Paymaster Project			
Lithology	**SG (g/cm^3)**	**SG (ton/ft^3)**	**Count**
Ultramafic	2.87	0.089584	33
Mafic	2.82	0.088023	39
Quartz feldspar porphyry	2.71	0.084590	105
Overburden	1.80	0.056185	-

Blocks were flagged with the SG value based on the lithological domains. All the mineralized blocks were designated as quartz feldspar porphyry.

14.3.10 Block Model Setup

The OP and UG block models have the same dimensions, origins, and block sizes and have been defined in GEMS as two separate workspaces. The block model origin, extent, and block size are listed in Table 14.45.

Table 14.45 Block model Definition	
Lexam VG Gold Inc. - Paymaster Project	
Origin	
X (ft)	4,200
Y (ft)	6,600
Z (ft)	11,300
Rotation	0
Block size (ft)	10x10xw0
Column (ft)	10
Row (ft)	10
Level (ft)	10
Number of Blocks	
Columns	440
Rows	380
Levels	220

14.3.11 Interpolation Parameters

The OP model used search ellipses with major and semi-major radii of 150 feet for the first pass and 300 feet for the second and inverse distance to the 5th power (ID5) for grade interpolation, while the UG model used search radii of 150 feet combined with an inverse distance cubed (ID^3) method. RPA used a higher exponent to interpolate the OP model in order to help reduce the lateral interaction between the mineralization and internal waste bands included in some of the OP resource wireframes. Numerous search ellipsoid orientations were used to ensure that the search neighbourhood was consistent with the orientation of the resource wireframes. Details of the interpolation parameters for OP and UG block models are listed in Table 14.46.

The OP interpolated block grades, based on capped gold grades, are presented in Figure 14.39. The UG interpolated blocks are shown in Figure 14.40.

TABLE 14.46
BLOCK MODEL INTERPOLATION PARAMATERS

Lexam VG Gold Inc. - Paymaster Project											
Solid	**Ellipse**	**Anisotropy**			**Rotation**			**Sample selection**			**Interpolation Method**
		Major	**Semimajor**	**Minor**	**Azimuth**	**Dip**	**Interm Azim**	**Min**	**Max**	**Max per drill hole**	
133, 134, 135	EW_35	150	150	25	180	35	90	1	10	2	ID5
	EW_35_2	300	300	60	180	35	90	1	10	2	ID5
116, 120, 131, 132	EW_45	150	150	25	180	45	90	1	10	2	ID5
	EW_45_2	300	300	60	180	45	90	1	10	2	ID5
103, 104, 106, 107, 108, 110, 111, 113, 114, 117, 119, 121, 126, 127, 128, 129, 130	EW_65	150	150	25	180	65	90	1	10	2	ID5
	EW_65_2	300	300	60	180	65	90	1	10	2	ID5
101, 102, 105, 109, 112, 118, 122, 136, 137	EW_75	150	150	25	180	75	90	1	10	2	ID5
	EW_75_2	300	300	60	180	75	90	1	10	2	ID5
115	NNE_50	150	150	25	160	50	30	1	10	2	ID5
	NNE_50_2	300	300	60	160	50	30	1	10	2	ID5
124	N_80	150	150	25	160	80	15	1	10	2	ID5
	N_80_2	300	300	60	160	80	15	1	10	2	ID5
123	SSW_45	150	150	25	150	45	120	1	10	2	ID5
	SSW_45_2	300	300	60	150	45	120	1	10	2	ID5
2114	EW2_90	150	150	25	180	90	50	1	10	2	ID5
	EW2_90_2	300	300	60	180	90	50	1	10	2	ID5
3114	EW3_60	150	150	25	180	60	95	1	10	2	ID5
	EW3_60_2	300	300	60	180	60	95	1	10	2	ID5
4114	EW4_75	150	150	25	180	75	100	1	10	2	ID5
	EW4_75_2	300	300	60	180	75	100	1	10	2	ID5
UG1, UG2, UG3, UG4	UG	150	150	150	0	0	0	2	12	1	ID^3
	UG	150	150	150	0	0	0	1	12	1	ID^3

Figure 14.39 OP Blocks - Gold Grades inside the Pit

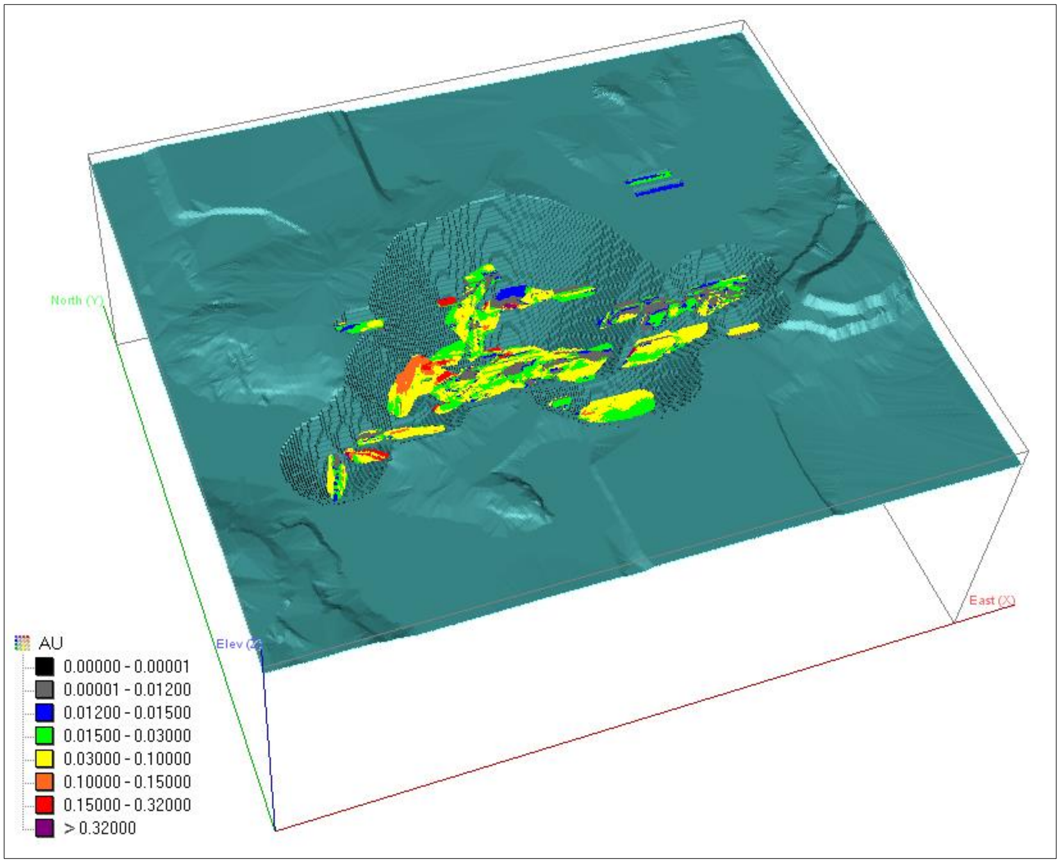
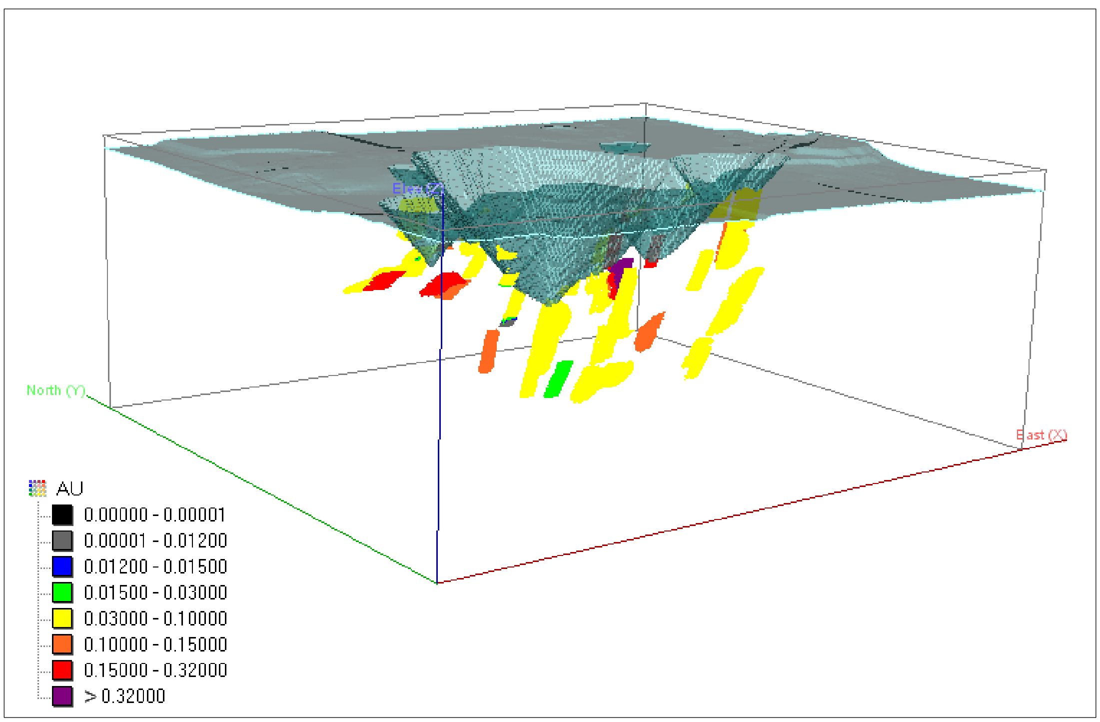


Figure 14.40 UG Blocks - Gold Grades below the Pit



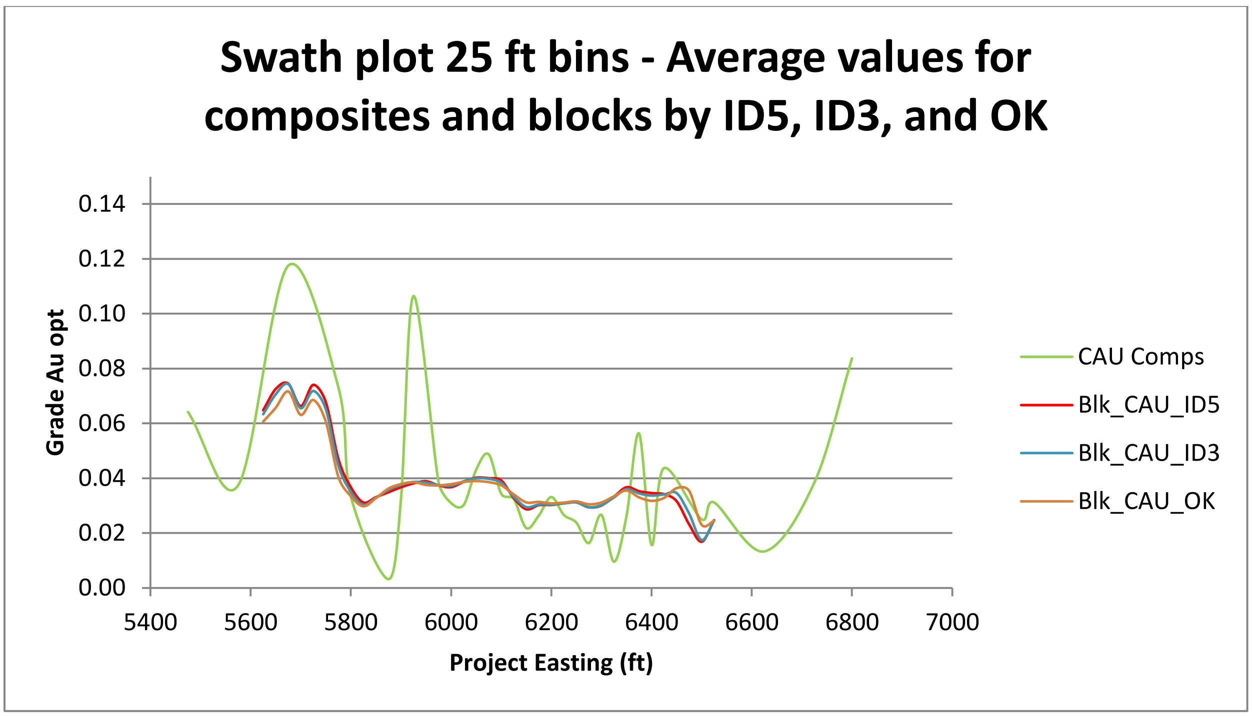
14.3.12 Block Model Validation

The interpolated block grades were visually compared in section and plan view with the grades of the composites, for both the OP and the UG. In general, the block grades correlate well spatially with the composite grades. There are some areas with visible, slightly misaligned grade bands that are due to local differences between the search ellipsoid and resource wireframe orientations. In RPA's opinion, this is a minor issue that could be improved in future models by adding more search domains.

For the OP resource estimate, along with the ID5 interpolation method, RPA also used ID^3 and OK methods, which rendered similar grades. Swath plots showing interpolated block gold grades using various methods versus the composite gold grades are presented in Figure 14.41.

In the opinion of RPA, the block model is a reasonable representation of the tonnage and grade for the gold mineralization of the Paymaster Project.

Figure 14.41 OP Swath Plot – Comparison of Various Interpolation Methods and Composites for Capped-Au Opt

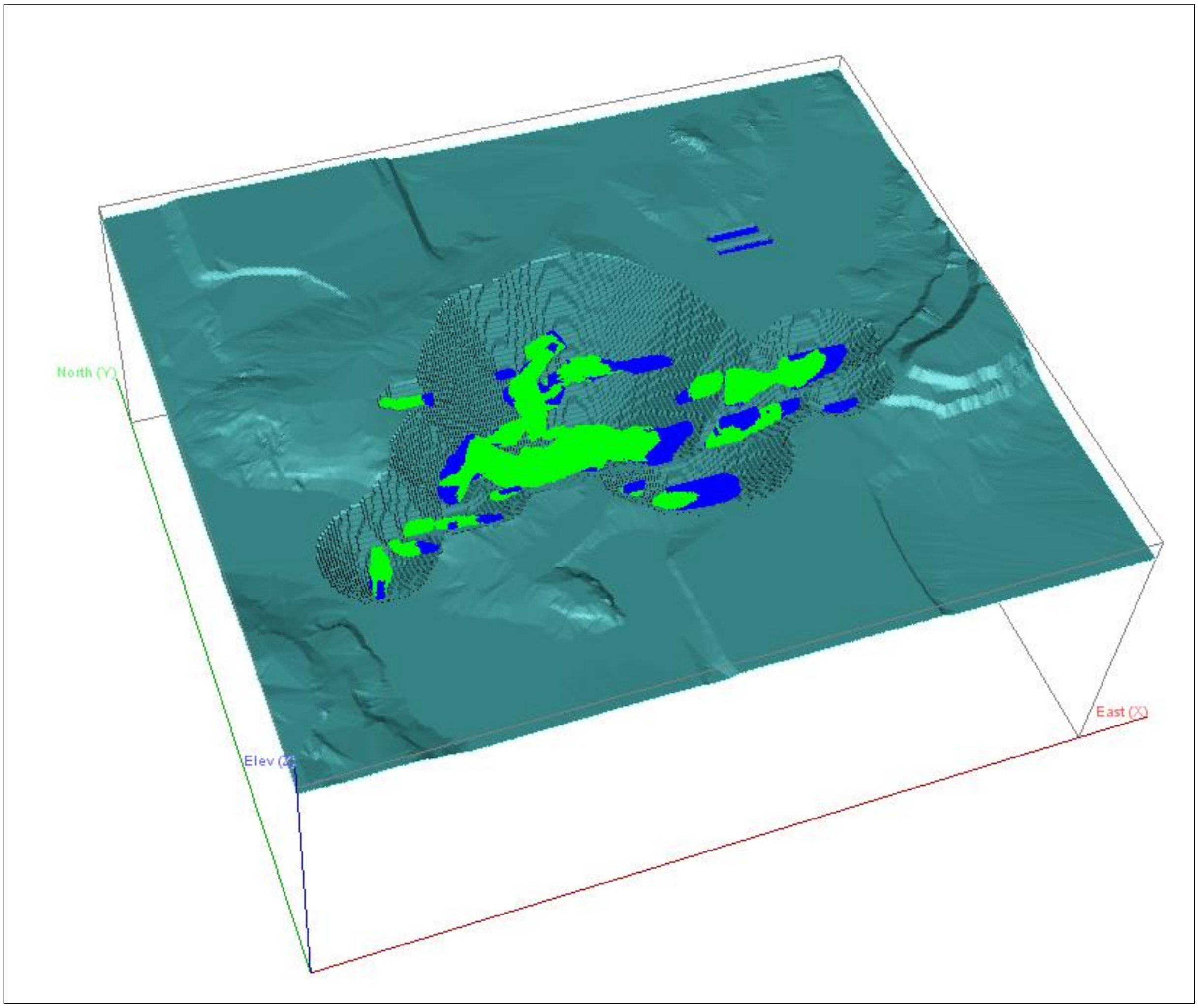


14.3.13 Classification

The OP blocks interpolated in the first and second passes were classified as Inferred, and then classification of the blocks that were within 150 feet from two drill holes, for each mineralized wireframe, was changed to Indicated category. Figure 14.42 shows the Indicated and Inferred blocks in the OP.

All the UG resource blocks were classified as Inferred. The drilling was focused on the potential OP target, resulting in relatively sparse drilling in the UG domain.

Figure 14.42 OP Indicated (Green) and Inferred (Blue) Blocks



14.3.14 Open Pit Optimization

The block model generated in Gemcom GEMS was transferred to Whittle for the pit optimization exercise. The block model was reblocked to a block size of 20 feet x 20 feet x 20 feet to reduce the number of blocks and implicitly the processing time. The parameters used in the preliminary optimization process are listed in Table 14.47.

TABLE 14.47 PRELIMINARY PIT OPTIMIZATION FACTORS	
Lexam VG Gold Inc. - Paymaster Project	
Pit Slopes	-50º
Mining Cost	$1.68/ton (converted from $1.85/tonne)
Process Cost	$14.52/ton (converted from $16.00/tonne)
Recovery	90%
Au Price	$1,600/oz
Block Size (Reblocked)	20 x 20 x 20 feet

The revenue factor 1 pit was then transferred to Gemcom GEMS for OP resource reporting and served as the upper limit for the UG domain. The OP resources were reported using a 0.015 opt cut-off grade.

14.3.15 Sensitivity Analysis

The OP Mineral Resources are sensitive to the cut-off grade in the 0.015 opt Au to 0.05 opt Au range and moderately sensitive to cut-off grades higher than 0.05 opt Au. Table 14.48 presents the tonnage, grade, and ounces at various cut-off values. Figure 14.43 and Figure 14.44 show the grade-tonnage curves for the OP Indicated and Inferred Resources.

TABLE 14.48
GRADE AND TONNAGE AT VARIOUS CUT-OFF GRADES

Lexam VG Gold Inc. - Paymaster Project

Classification	Cut-off Grade (opt Au)	Tonnage (tons)	Grade (opt Au)	Gold (ounces)	Lexam Ounces
OP Indicated	≥0.320	48,836	0.485	23,697	14,218
	≥0.150	140,203	0.301	42,217	25,330
	≥0.125	205,273	0.249	51,025	30,615
	≥0.100	312,292	0.201	62,905	37,743
	≥0.075	573,880	0.148	84,973	50,984
	≥0.050	1,336,024	0.098	131,314	78,788
	≥0.045	1,660,044	0.088	146,684	88,011
	≥0.030	2,937,641	0.066	193,856	116,313
	≥0.015	5,134,849	0.047	242,383	145,430
OP Inferred	≥0.320	8,299	0.583	4,834	2,901
	≥0.150	27,823	0.334	9,281	5,569
	≥0.125	39,485	0.275	10,864	6,519
	≥0.100	57,199	0.225	12,887	7,732
	≥0.075	132,264	0.144	19,040	11,424
	≥0.050	469,876	0.084	39,554	23,732
	≥0.045	567,099	0.078	44,163	26,498
	≥0.030	962,366	0.061	58,585	35,151
	≥0.015	1,541,873	0.047	72,495	43,497
UG Inferred	≥0.320	12,093	0.393	4,752	2,851
	≥0.150	129,284	0.242	31,282	18,769
	≥0.125	155,146	0.226	35,037	21,022
	≥0.100	189,110	0.205	38,788	23,273
	≥0.075	239,066	0.179	42,857	25,714
	≥0.050	836,503	0.097	81,072	48,643
	≥0.045	909,353	0.093	84,529	50,718
	≥0.030	1,059,626	0.086	90,771	54,463
	≥0.015	1,098,545	0.083	91,624	54,974

Figure 14.43 OP Indicated Resources – Grade–Tonnage Curve

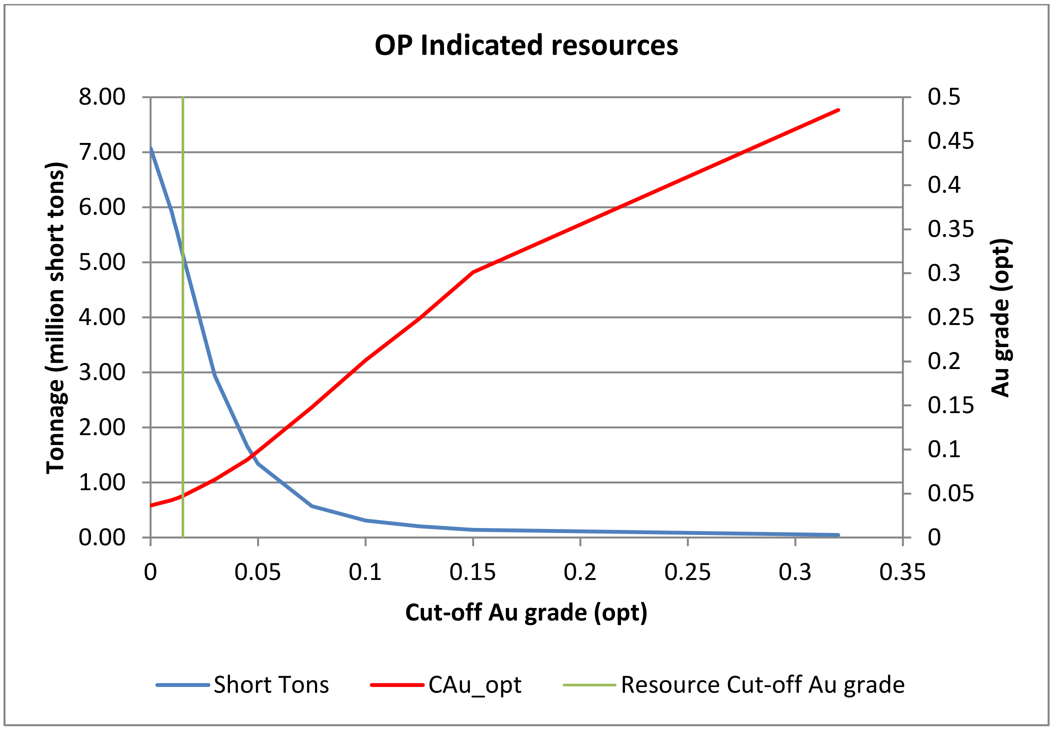
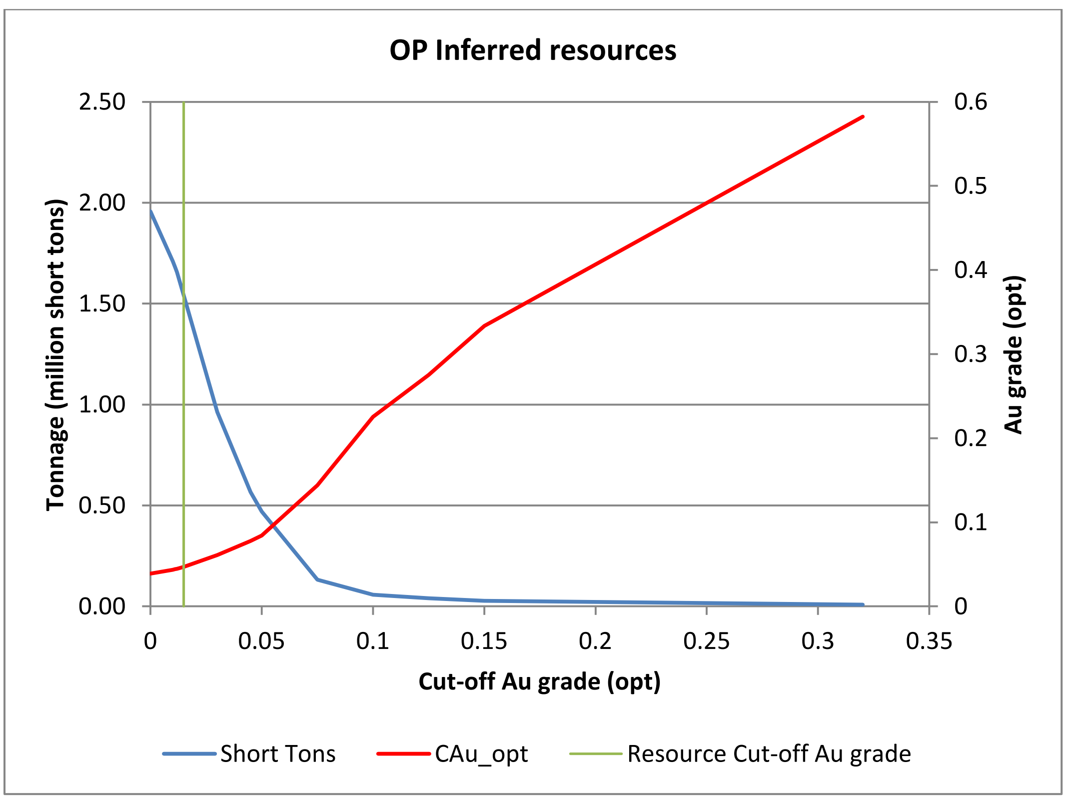


Figure 14.44 OP Inferred Resources – Grade–Tonnage Curve



14.3.16 Comparison with Previous Resource Estimate

The current 2013 estimate is compared with the previous, 2010 Mineral Resource estimate (Guy and Bevan, 2010) in Table 14.49.

TABLE 14.49 COMPARISON WITH DECEMBER 2010 RESOURCE ESTIMATE					
Lexam VG Gold Inc. - Paymaster Project					
Year	**Classification**	**Cut-off (opt Au)**	**Tonnage (tons)**	**Grade (opt Au)**	**Gold (ounces)**
2013	**OP**				
	Indicated	≥0.015	5,135,000	0.047	242,000
	Inferred	≥0.015	1,542,000	0.047	72,000
	UG				
	Indicated	-	-	-	-
	Inferred	≥0.075	239,000	0.179	43,000
	Total Indicated		5,135,000	0.047	242,000
	Total Inferred		1,781,000	0.065	115,000
2010	Total Indicated	≥0.015	3,562,511	0.053	189,082
	Total Inferred	≥0.015	3,071,876	0.050	154,404

The historical estimation was done in panels spanning half distance between sections, while on section the intercept grade was projected halfway between drill holes. The capping value applied in the 2010 estimate was 1 opt Au and the reporting cut-off value was 0.015 opt Au.

Additional drilling was available for the 2013 resource estimate. Mineralized intercepts from 24 new drill holes for both OP and UG domains were used in the current estimate.

In the current estimate, the OP grades are lower by 11% for Indicated and by 9% for Inferred Resources with respect to the 2010 estimate. The difference represents the cumulative effect of different capping levels, extra drilling, 3D wireframing, as well as constraining the resources inside a pit shell.

The total resource ounces show almost a 30% increase for Indicated Resources and a 10% increase for Inferred Resources (OP and UG).

14.4 DAVIDSON TISDALE

14.4.1 Introduction

The mineral resource estimate presented herein is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 and has been performed in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. This resource estimate was prepared by Yungang Wu, P.Geo. , Eugene Puritch, P.Eng., and Antoine Yassa, P.Geo. of P&E. The effective date of this resource estimate is April 2, 2013.

14.4.2 Database

All drill data were provided by Lexam, in the form of a Gemcom GEMS 6.4 database as a Microsoft Access .mdb file. The Gemcom database consisted of a total of 398 surface and 287 underground drill holes, of which 430 holes intersected the mineralized domains and were utilized for this mineral resource estimation. The remaining data were not in the area that was modeled for this resource estimate. Surface and underground drill hole plans are shown in Appendix - I.

The database for the Davidson Tisdale Project contained 13,852 Au assays of which 4,878 assays were utilized for this resource estimate. P&E validated the database in Gemcom and corrections were made in order to bring it to an error free status. All drill hole survey assay values are expressed in metric units, while grid coordinates are in a local metric system.

14.4.3 Data Verification

Au assays were validated by P&E against original laboratory certificates of analysis. A total of 4,493 assay results from the recent drilling were verified. This comprised 73.8% of the recent database that was verified and 18.6% of the entire database.

14.4.4 Domain Interpretation

Four mineralized domains were created with computer screen digitizing on drill hole sections in Gemcom by the authors of this report. The domain outlines were determined from lithology, structure and Au grade by visually inspecting the drill hole cross sections. Nineteen (19) drill cross sections were developed on a 25-metre spacing looking northeast on an azimuth of 60^{o}. The digitized outlines were influenced by the selection of mineralized material above a cut-off 0.3g/t Au that demonstrated zonal continuity along strike and down dip. In some cases, mineralization below 0.3 g/t Au was included for the purpose of maintaining zonal continuity. On each section, polyline interpretations were digitized from drill hole to drill hole but not extended nominally more than 25 metres into untested territory. Minimum constrained true width for interpretation was approximately 2.0 metres. The interpreted polylines from each section were “wireframed” in Gemcom into a 3-dimensional domain. The resulting domain was utilized for statistical analysis, grade interpolation, rock coding and resource reporting purposes. Wireframes of the mineralized domains are displayed in Appendix-II.

A Topographic and overburden surface were created based on the drill hole collars and logs. A void solid of underground workings was provided by the client.

14.4.5 Rock Code Determination

All mineralized domain solids were assigned rock codes respectively for purpose of resource estimating. The domain geometric volumes and rock codes applied for the resource modeling are presented in Table 14.50.

Table 14.50 Geometric Volume and Rock Code Description for Davidson Tisdale Project		
Domains	**Rock Codes**	**Volume (m^3)**
A-Zone	10	46,022
B-Zone	20	497,046
C-Zone	30	114,284
Misc	40	34,134
Air	0	
OVB	5	
Waste	99	
Void	100	

14.4.6 Statistical Analysis

The statistical analysis of the constrained Au raw assay and sample length is summarized in Table 14.51, which gives a mean of 4.97g/t Au with a maximum value of 1,649.2 g/t Au.

Table 14.51 Summary Statistics of the Constrained Au Raw Assays		
Variable	**Au (g/t)**	**Length (m)**
Number of samples	4,876	4876
Minimum value	0.001	0.03
Maximum value	1649.2	2.20
Mean	4.973	0.82
Median	0.480	0.90
Geometric Mean	0.202	0.75
Variance	1432.05	0.126
Standard Deviation	37.842	0.354
Coefficient of variation	7.609	0.430
Skewness	24.941	0.748
Kurtosis	875.779	3.018
Natural Log Mean	-1.602	-0.285
Log Variance	9.294	0.192

14.4.7 Grade Capping

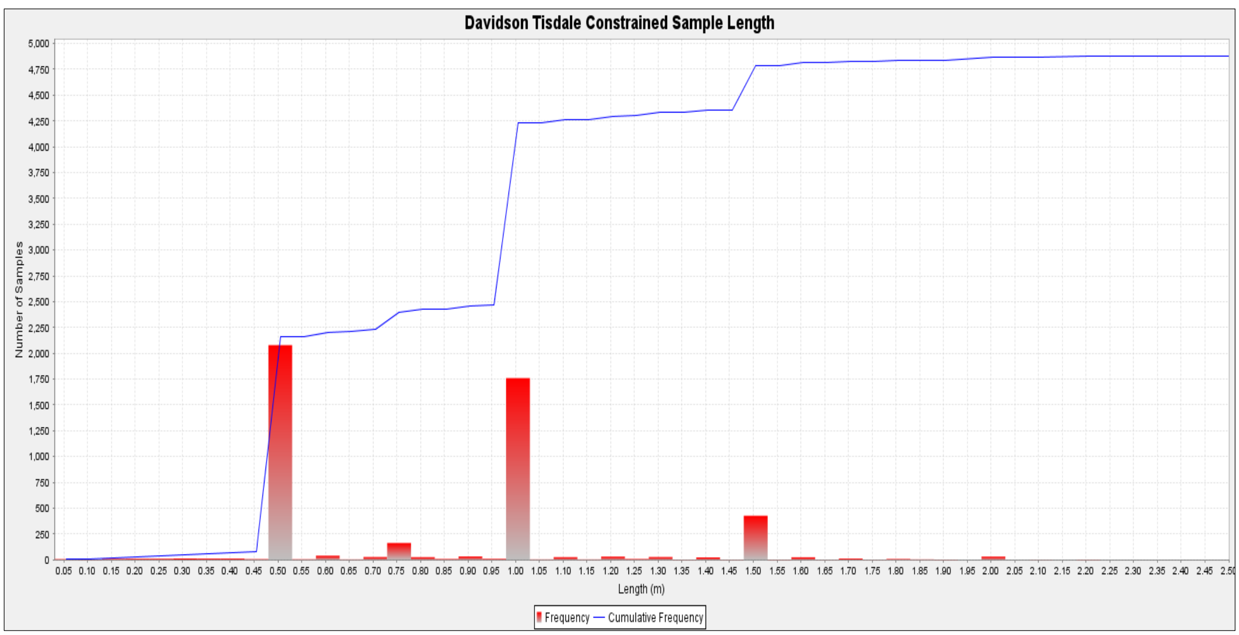
Grade capping was investigated on the Au assays within the constraining domains to ensure that the possible influence of erratic high values did not bias the database. Au assay Log-normal histograms were generated and resulting graphs are exhibited in Appendix-III. Table 14.52 details the grade capping values. The capped Au assays were utilized for the compositing.

TABLE 14.52 AU GRADE CAPPING VALUES						
Domains	**Total # of Assays**	**Capping Value Au (g/t)**	**Number of Assays Capped**	**Raw Coefficient of Variation**	**Capped Coefficient of Variation**	**Capping Percentile**
A-Zone	201	50	2	3.04	2.49	99.0%
B-Zone	3,355	50	57	4.724	2.51	98.3%
C-Zone	1,216	50	17	4.54	2.49	98.6%
Misc	104	20	5	3.01	1.96	95.2%

14.4.8 Compositing

Table 14.51 depicts that the average constrained sample length was 0.82 m, and approximately 36% of the sample lengths within the constrained wireframe were one metre in length (Figure 14.45). In order to regularize the assay sample lengths for grade interpolation, assay compositing to one metre in length was carried out down hole within the constraints of the above mentioned domains. The composites were calculated for Au capped over 1.0 metre lengths starting at the first point of intersection between drill hole and hanging wall of the 3-D zonal constraint. The compositing process was halted upon exiting from the footwall of the aforementioned constraint. Un-assayed intervals and below detection limit assays were set to 0.001 g/t Au. Any composites that were less than 0.25 metres in length were discarded so as not to introduce any short sample bias in the Au grade interpolation process.

Figure 14.45 Drill Hole Assay Sample Length Distribution



14.4.9 Semi-Variograms

The variography investigation was attempted on the constrained composites. Reasonable variograms were developed along strike, down dip and across dip for the combination of all domains, B-Zone and C-Zone. The variogram ranges were used as the spherical search ellipse parameters for grade interpolation. The variograms of the B-Zone are demonstrated in Appendix-IV.

14.4.10 Bulk Density

A total of 55 core samples were taken by Antoine Yassa, P.Geo. of P&E and analyzed at AGAT Laboratories in Mississauga, Ontario. The average bulk density 2.87t/m^3 of the 55 samples was applied to this resource estimate. P&E suggests that systematic density testing program should be undertaken in future drilling programs.

14.4.11 Block Modeling

The Davidson Tisdale resource block model was constructed using Gemcom modeling software. The block model is oriented with X axis at 60^o azimuth (rotated 30^o counter clockwise) parallel to the strike of the main mineralized domains. The block model parameters are shown in Table 14.53.

TABLE 14.53
DAVIDSON TISDALE BLOCK MODEL DEFINITIONS

Direction	Origin	# of Blocks	Block Size
X	9,985	166	5
Y	9,260	240	5
Z	3,330	88	5
Rotation	30^o (Counter Clockwise)		

Block models for rock type, density, percent, Au, and class were created. All blocks in the rock type block model were initially assigned a waste rock code of 99, corresponding to country rocks. The mineralized domains were used to update all blocks within the rock type block model that contained by a volume of 1% or greater Au mineralization. These blocks were assigned their appropriate individual rock codes as indicated in Table 14.50. The overburden surface and topographic surface were subsequently utilized to assign rock code 5 for overburden and 0 for air to all blocks 50% or greater above the surfaces. A void provided by the client was employed to deplete the historic mined area by coding 100 to the blocks 50% or greater within the void wireframe.

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

The bulk density models were initialized to 2.87t/m^3 for all Au mineralized blocks.

Ordinary Kriging (OK) grade interpolation was utilized for grade interpolation based on the Au composites which were extracted from drill hole profiles into point profiles. Negative weights were set to zero. Grade blocks were interpolated using the parameters in Table 14.54.

TABLE 14.54 BLOCK MODEL INTERPOLATION PARAMETERS							
Pass	**Strike Range (m)**	**Dip Range (m)**	**Across Dip Range (m)**	**Max # per Hole**	**Min # Sample**	**Max # Sample**	**% of Interpolated Blocks**
1	15	12	8	2	5	12	29%
2	25	20	15	2	3	12	37%
3	50	40	30	2	1	12	34%

The search ellipsoid orientation was aligned with the trend of each domain or sub-domain which was established according to the variation of the domain trend. The resulting Au grade blocks are presented on the block model cross-sections and plans in Appendix-V.

14.4.12 Resource Classification

In P&E's opinion, the drilling, assaying and exploration work supporting this resource estimate are sufficient to indicate reasonable potential for economic extraction and thus qualify it as a Mineral Resource under CIM definition standards. Based on geology, semi-variogram performance and density of the drilling data, the Measured resource category was justified for blocks interpolated by the pass one (Table 14.54) which was using at least five composites from a minimum of three drill holes within spacing of 15 m on strike, 12 m down dip and 8m on across dip direction. Indicated resources were classified for blocks interpolated with pass 2 which was using at least three composites from a minimum of two drill holes within spacing of 25 m on strike, 20 m down dip and 15 m on across dip direction. Inferred resources were classified by pass 3 on all remaining grade populated blocks. The classifications of some blocks have been manually adjusted to represent the resource classification more reasonably. Classification block cross-sections and plans are attached in Appendix VI.

14.4.13 Resource Estimate

The resource estimate was derived from applying an Au cut-off grade to the block model and reporting the resulting tonnes and grade for potentially mineable areas. The following calculation demonstrates the rationale supporting the Au cut-off grade that determines Open Pit and Underground potentially economic portions of the constrained mineralization.

14.4.13.1 Au Grade Cut-off Calculation

Open Pit Au Cut-Off Grade Calculation CDN$

Au Price	US$1,600/oz (Approx. 30 month trailing average price Mar 31/13)
$US/$CDN Exchange Rate	$1.00
Au Recovery	90%
Process Cost (1,500 tpd)	$18/tonne milled
General & Administration	$5/tonne milled

Therefore, the Au cut-off grade for the open pit resource estimate calculated as follows:

Operating costs per ore tonne = ($18+ $5) = $23/tonne
[($23)/[($1,600/oz/31.1035 x 90% Recovery)] = 0.35 g/t, used 0.5 g/t

Underground Au Cut-Off Grade Calculation CDN$

Au Price	US$1,600/oz (24 month trailing avg. price as of Mar 31/13)
$US/$CDN Exchange Rate	1:1
Au Recovery	90%
Mining Cost	$97/tonne mined
Process Cost (1,500 tpd)	$18/tonne milled
General & Administration	$5/tonne milled

Therefore, the Au cut-off grade for the underground resource estimate is calculated as follows:

Operating costs per ore tonne = ($97+$18+ $5) = $120/tonne
[($120)/[($1,600/oz/31.1035 x 90% Recovery)] = 2.60 g/t

14.4.14 Pit Optimization

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

Waste mining cost per tonne	$1.85
Ore mining cost per tonne	$1.85
Overburden Mining cost per tonne	$1.35
Process cost per tonne	$18
General & Administration cost per ore tonne	$5
Process production rate (ore tonnes per year)	525,000
Pit slopes (overall wall angle)	45 degrees
Mineralized Rock Bulk Density	2.87t/m^3
Waste Rock Bulk Density	2.90t/m^3
Overburden Density	1.80t/m^3

14.4.15 Mineral Resource Estimate

The resulting resource estimate for the Davidson Tisdale project is summarized in the Table 14.55.

TABLE 14.55
MINERAL RESOURCE ESTIMATE FOR DAVIDSON TISDALE(1-5)

	Cut-Off (Au g/t)	Category	Tonnes	Au g/t	Contained Au oz	68.5% Attributable oz to Lexam
In-Pit	0.5	Measured	452,000	2.44	35,500	24,300
	0.5	Indicated	173,000	2.43	13,500	9,300
	0.5	Total	625,000	2.44	49,000	33,600
UG	2.6	Measured	18,000	6.64	3,800	2,600
	2.6	Indicated	41,000	4.91	6,500	4,400
	2.6	M+I	59,000	5.43	10,300	7,000
	2.6	Inferred	71,000	4.20	9,600	6,600
Total	**0.5+2.6**	**Measured**	**470,000**	**2.60**	**39,300**	**26,900**
	0.5+2.6	**Indicated**	**214,000**	**2.90**	**20,000**	**13,700**
	0.5+2.6	**M+I**	**684,000**	**2.70**	**59,300**	**40,600**
	2.6	**Inferred**	**71,000**	**4.20**	**9,600**	**6,600**

(1) Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
(2) The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.
(3) The mineral resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
(4) The historical mined tonnage was not depleted as the mined tonnage was insignificant.
(5) The Davidson Tisdale project is a joint venture between Lexam as operator (68.5%) and SGX Resources Inc. (31.5%). The contained Au oz reflects the 68.5% of the resource attributable to Lexam.

The Au cut-off sensitivities to the resource estimate are demonstrated in Table 14.56 and Table 14.59.

Table 14.56 **In-Pit Au Cut-Off Sensitivity to Resource Estimate of Davidson Tisdale**				
Category	**Cut-Off (Au g/t)**	**Tonnes**	**Au g/t**	**Contained Au Oz**
MEASURED	1.00	328,445	3.09	32,609
	0.90	348,626	2.96	33,222
	0.80	372,502	2.83	33,872
	0.70	394,463	2.71	34,408
	0.60	422,907	2.57	35,001
	0.50	452,147	2.44	35,520
	0.40	484,114	2.31	35,980
	0.30	524,347	2.16	36,432
	0.20	569,030	2.01	36,794
INDICATED	1.00	137,823	2.87	12,716
	0.90	143,277	2.80	12,883
	0.80	150,496	2.70	13,079
	0.70	157,386	2.62	13,247
	0.60	163,220	2.55	13,369
	0.50	173,305	2.43	13,547
	0.40	182,442	2.33	13,680
	0.30	194,725	2.21	13,817
	0.20	203,890	2.12	13,888

Table 14.57 **Out-of-Pit (Underground) Au Cut-Off Sensitivity to Resource Estimate of the Davidson Tisdale**				
Category	**Cut-off (Au g/t)**	**Tonnes**	**Au g/t**	**Contained Au Oz**
MEASURED	3.0	13,964	7.70	3,458
	2.6	17,811	6.64	3,801
	2.0	23,319	5.61	4,208
	1.5	32,918	4.49	4,754
	1.0	47,153	3.50	5,314
INDICATED	3.0	33,056	5.41	5,750
	2.6	40,904	4.91	6,454
	2.0	66,704	3.91	8,381
	1.5	85,907	3.43	9,463
	1.0	132,817	2.64	11,290
INFERRED	3.0	49,212	4.83	7,636
	2.6	71,135	4.20	9,607
	2.0	100,058	3.67	11,800
	1.5	122,433	3.30	13,006
	1.0	198,347	2.50	15,963

14.4.16 Confirmation of Estimate

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

These methods included visual examination of composite and block grades on plans and sections on-screen and review of estimation parameters such as:

- Number of composites used for estimation;
- Number of holes used for estimation;
- Distance to the nearest composite;
- Number of interpolation passes used to estimate grade;
- Mean value for composites used.

As a test of the reasonableness of the resource estimates, the block model grade was also interpolated with Inverse Distance Cubed ($1/d^3$) and Nearest Neighbour (NN) methods. The average interpolated grades for the block model were compared to the mean of composites used for grade interpolation. As shown in Table 14.58, the average grades of block model among the different interpolation methods and mean of composites are similar.

TABLE 14.58
COMPARISON OF AVERAGE GRADE OF COMPOSITES WITH AVERAGE GRADES OF THE TOTAL BLOCK MODEL

Data Type	Au g/t	Interpolation Method
Composites	1.50	
Block Model	1.51	Ordinary Kriging (OK)
Block Model	1.48	Inverse Distance Cubed ($1/d^3$)
Block Model	1.49	Nearest Neighbour (NN)

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

Block Model Volume = 690,419 m^3
Geometric Domain Volume = 691, 486 m^3
Difference = 0.15%

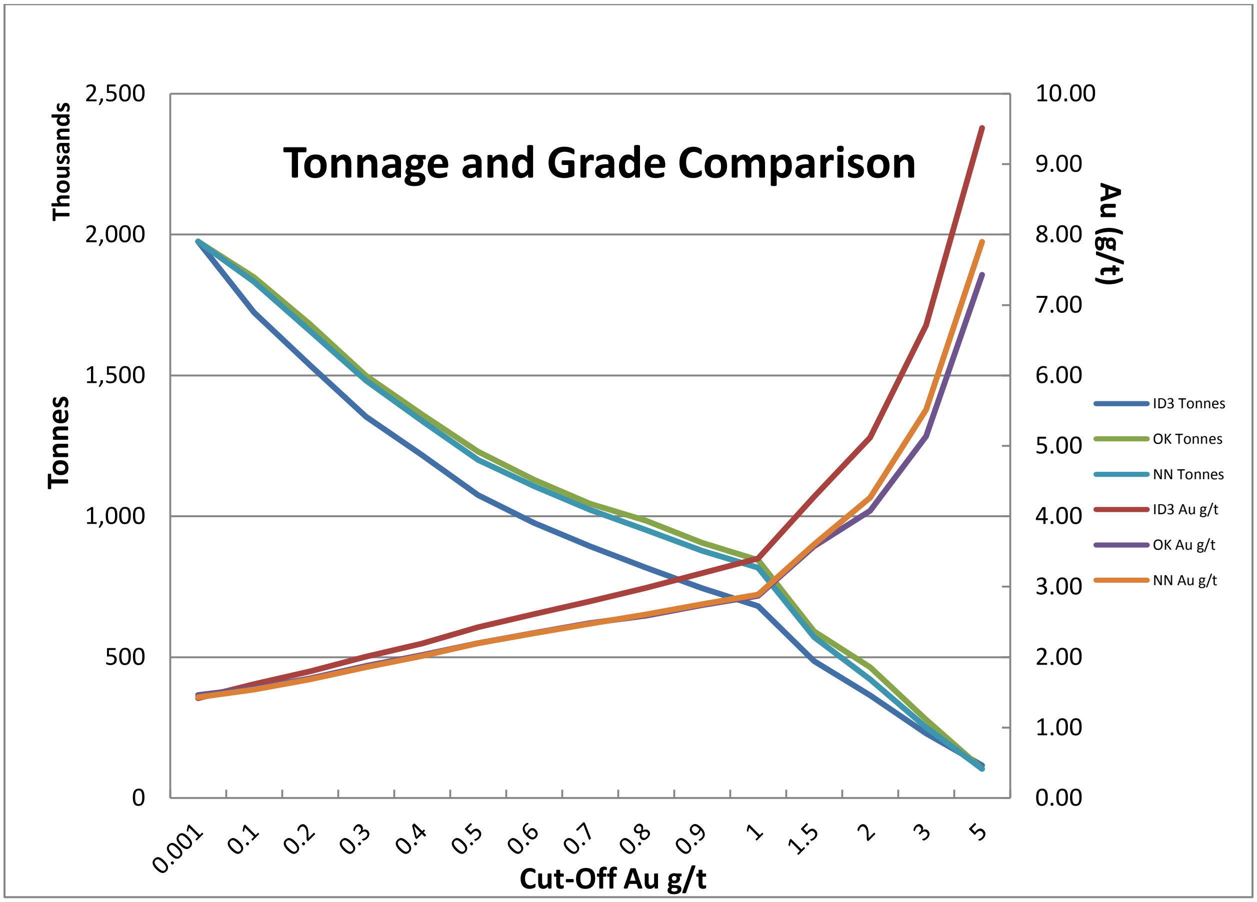
Comparison of grade models interpolated with OK, Inverse Distance Cubed ($1/d^3$) and Nearest Neighbour (NN) was conducted on the global resources as shown in Table 14.59 and Figure 14.45.

In P&E's opinion, the grade interpolation utilizing Ordinary Kriging is reasonable.

TABLE 14.59
COMPARISON OF RESOURCE ESTIMATE AT CUT-OFF 1.0G/T AU INTERPOLATED WITH OK, $1/D^3$ AND NN METHOD

Interpolation Method	OK	$1/d^3$	NN
Average Grade Au g/t	1.88	2.01	1.86
Tonnes (000s)	1,499	1,354	1,481
Contained OZ of Au (000s)	90	87	88

Figure 14.46 Comparison of Resource Estimate at Au Cut-Off Grade among OK, 1/d^3 and NN Method



15.0 MINERAL RESERVE ESTIMATES

This section is not applicable to the current Technical Report.

16.0 MINING METHODS

This section is not applicable to the current Technical Report.

17.0 RECOVERY METHODS

This section is not applicable to the current Technical Report.

18.0 PROJECT INFRASTRUCTURE

This section is not applicable to the current Technical Report.

19.0 MARKET STUDIES AND CONTRACTS

This section is not applicable to the current Technical Report.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section is not applicable to the current Technical Report.

21.0 CAPITAL AND OPERATING COSTS

This section is not applicable to the current Technical Report.

22.0 ECONOMIC ANALYSIS

This section is not applicable to the current Technical Report.

23.0 ADJACENT PROPERTIES

The authors have reviewed available historic assessment reports and private property reports for the preparation of this report.

The Buffalo Ankerite Property lies on the south limb of the Porcupine Syncline. This limb encompasses much of the production from the Timmins Gold camp including the past producers that lie adjacent to the property. The largest producer on the south limb is the Dome Mine, with more than 17 million ounces of Au production and the longest producer in the camp, having recently celebrated its 100th anniversary, and being still in production.

23.1 DELNITE MINE

It was in production from 1937 to 1964 when 920,404 oz gold was recovered from 3,847,364 tons of ore. Gold bearing veins parallel to the strike of country rocks (carbonatized basalt) dip at 60° or steeper. Ore shoots developed near surface were 200 feet long and 5 feet wide, developed by 2 surface shafts and 1 internal sub-shaft from 2,888 feet to 5,395 feet.

23.2 AUNOR MINE

The ore zone is a band of pillow volcanics enclosed in talc-chlorite schist (serpentinite), striking N80°E and dipping 50-80°N. Quartz forms 50% of vein material associated with carbonate, tourmaline and scheelite and small amounts of pyrite and chalcopyrite. Individual veins are about 3.5 feet wide but closely spaced veins and stringers or zones 50-75 feet in width. Production was 2,502,000 oz gold from 8,482,000 tons of ore.

23.3 DOME MINE

The mineralized ankerite and quartz-tourmaline veins persist over a strike length of 9,000 feet and to a depth of 5,000 feet. Sulphides present in the veins and adjacent wall rocks are up to 3% and consist mainly of pyrite and pyrrhotite with minor sphalerite and galena. The gold is generally coarse and most of it can be recovered by gravity concentration. Production has been achieved by open pit and underground operations and began in 1910. Total production to date has been in excess of 15 million ounces of gold.

23.4 PRESTON EAST DOME MINE

Developed by 4 shafts, production from this property has been 1,539,400 oz gold from 6,284,400 tons of ore. Veins are present in 2 porphyries and stockworks occur in association with porphyries and volcanic rocks. The veins consist of quartz, ankerite and tourmaline. Pyrite, pyrrhotite, sphalerite, galena and native gold are present in the veins. Individual veins are up to 6 feet wide, 700 feet long and extend for 600 feet in depth.

23.5 PAYMASTER MINE

The main Paymaster past producing area was not included in the Lexam/Goldcorp Paymaster Joint Venture. The #5 shaft was the source of the majority of the production from the Paymaster property and lies to the east of the Fuller property and immediately north of the Paymaster Joint Venture.

The main vein zone is 2,000 feet long with a depth of 3,000 feet. Vein material consists of quartz and carbonate with small amounts of sericite and scheelite. On the 1,200-foot level an ore shoot on the No. 3 vein was 182 feet long, 3.6 feet wide and averaged 0.87 oz Au/ ton. The mine was developed principally by the No.5 (main) Shaft to 4,462 feet, the No.2 Winze 2,046 - 4,202 feet and the No.6 winze 4,059 - 6157 feet, 197,294 feet of drifting and 82,577 feet of crosscutting. This mine adjoins the Fuller property to the east and is north of the Paymaster Joint Venture. Paymaster production from 1915 to 1966 was 1,192,000 oz gold from 5,607,000 tons of ore.

23.6 HOLLINGER MINE

Most of the veins are in the basalts adjacent to the porphyries and a few are within the porphyries themselves. The deposit is a composite vein zone 5,000 feet long, 3,000 feet wide and 2,000 feet deep. The average stoping width was about 10 feet which might consist of a vein 5 feet wide with a zone of stringers and mineralized wall rock adjacent to the vein. Quartz and carbonate are the most abundant vein minerals (chalcopyrite, sphalerite, galena and tellurides). Pyrite occurs in the wall rock adjacent to the veins and gold occurs in veinlets in the darker coloured parts of the vein and fractures in the pyrite adjacent to the vein. The property has numerous shafts and 380 miles of underground lateral workings. Production from this mine was prolific and amounted to 19,354,500 oz gold from 65,890,400 tons of ore.

24.0 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other relevant data or information that is not included in the Technical Report.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 BUFFALO ANKERITE

Significant deposits of gold mineralization have been delineated by drilling in the North and South zones on the Buffalo Ankerite Property and these zones' extensions onto the Paymaster Property to the east. Diamond drilling has been carried out throughout a long history of exploration and underground production. Both surface and underground drill holes were included in the Mineral Resource estimate.

The procedures for drilling, collection of data, sampling, assaying and check assaying carried out by Dome and Lexam have produced a drill hole database that is acceptable for Mineral Resource estimation, in P&E's opinion. The surface drill hole database for the resource estimate contains the Placer Dome drilling (2001-2004) and the VG Gold/Lexam drilling (2005-2012). Resources in the North Zone were intersected by 735 holes totaling 240,418.21 ft (73,279 m) whereas resources in the South Zone were intersected by 692 holes for 241,422.3 ft (73,586 m). Of the total resource holes, 337 were drilled from surface and 1,154 were drilled from underground.

Continuity of host structures on the Buffalo Ankerite Property were readily amenable to wireframe constraints. The Mineral Resource estimation was constrained by mineral zone wireframes and used multiple search ellipses mapped to the zones' orientations and inverse distance to the power of 3 (ID^3) for grade interpolation. Wireframes were constructed at a cut-off grade of 0.015 oz/ton Au for potentially open pittable gold mineralization to the 10,000 ft relative elevation at approximately 1,000 ft depth and at 0.045 oz/ton Au cut-off grade for underground mineable mineralization below 10,000 ft relative elevation. The latter cut-off grade is based on underground breakeven costs and a gold price of $1,600/oz. The resources above the 10,000 ft elevation were estimated from 5 ft assay composites based on a 0.015 oz/ton Au cut-off whereas the resources below the 10,000 ft elevation were estimated from 5 ft assay composites based on a 0.045 oz/ton cut-off. Resources are reported net of past production based on modelled stopes and drifts located in the mineral zones.

P&E notes that:

- No assay data were available for the stopes and drifts, however, construction of vein domains was guided by the location of stopes and exploration drifts which imply good continuity of the structures hosting the gold-bearing quartz-tourmaline breccias and veining. Structures appear to merge or splay locally as interpreted from stoping, drifting and drilling intercepts and interpretation of the structures is locally not unambiguous.
- Indicated Resources are outlined where surface and/or underground drill hole spacing is in the order of 150 ft for the North Zone and 100 ft for the South Zone. Grade data based on core sampling in areas of wide spaced surface drilling, is insufficient with respect to the grade variability, as noted in variography, to classify resources any higher than Inferred Mineral Resources.
- There is some uncertainty in the widths of modelled stopes since the tonnage attributable to all the modelled stopes exceeds past production.
- Surface drill hole deviation results in spatial uncertainty of the zone interpreted from drilling with respect to the stopes and drifts and therefore not all mined/developed material may have been subtracted from the estimated

resources. Underground resources bordering stopes, arising from larger zone widths because of lower cut-off grade, may only be partly recoverable or may not be recoverable, subject to engineering study.

At a 0.015 oz/ton Au (0.51 g/t Au) cut-off, the Buffalo Ankerite Property contains an open pit Indicated Mineral Resource of 3.15 million tons (2.86 million tonnes) at 0.074 oz/ton Au (2.54 g/t Au) totalling 234,600 contained gold ounces, and an Inferred Mineral Resource of 2.90 million tons (2.63 million tonnes) at 0.068 oz/ton Au (2.33 g/t Au) totalling 197,800 contained gold ounces. At a 0.075 oz/ton Au (2.57 g/t Au) cut-off, the Buffalo Ankerite Property contains Indicated Mineral Resource of 3.58 million tons (3.25 million tonnes) at 0.139 oz/ton Au (4.77 g/t Au) totalling 499,000 contained gold ounces, and an underground Inferred Mineral Resource of 3.10 million tons (2.81 million tonnes) at 0.118 oz/ton Au (4.41 g/t Au) totalling 367,000 ounces.

The Buffalo Ankerite Property has a total Indicated Mineral Resource of 6.73 million tons (6.11 million tonnes) at 0.109 oz/ton Au (3.74 g/t Au) totalling 733,600 contained gold ounces, and a total Inferred Mineral Resource of 6.00 million tons (5.44 million tonnes) at 0.094 oz/ton Au (3.22 g/t Au) totalling 564,800 contained gold ounces.

At a 0.015 oz/ton Au (0.51 g/t Au) cut-off, the Paymaster portion of the North and South zones contains an open pit Indicated Mineral Resource of 1.76 million tons (1.60 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 96,250 contained gold ounces, and an open pit Inferred Mineral Resource of 0.192 million tons (0.174 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 10,460 contained gold ounces. At a 0.075 oz/ton Au (2.57 g/t Au) cut-off, the Paymaster portion of the North and South zones contains an underground Indicated Mineral Resource of 0.020 million tons (0.018 million tonnes) at 0.100 oz/ton Au (3.43 g/t Au) totalling 1,970 contained gold ounces, and an underground Inferred Mineral Resource of 0.002 million tons (0.0018 million tonnes) at 0.082 oz/ton Au (2.18 g/t Au) totalling 157 contained gold ounces.

The Paymaster portion of the two zones has a total Indicated Mineral Resource of 1.78 million tons (1.61 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 98,220 contained gold ounces, and an Inferred Mineral Resource of 0.193 million tons (0.175 million tonnes) at 0.055 oz/ton Au (1.89 g/t Au) totalling 10,620 contained gold ounces.

Lexam holds 100% interest in the Buffalo Ankerite property and 60% interest in the Paymaster Property and respectively in the properties' Mineral Resources. From the above Mineral Resources, Lexam's attributable gold ounces contained in the North and South zones on the Buffalo Ankerite and Paymaster properties are 792,000 ounces in Indicated Mineral Resources and 571,000 ounces in Inferred Mineral Resources.

There are no Mineral Reserves estimated on the Buffalo Ankerite or Paymaster properties.

In P&E's opinion, the completion of a scoping study or a Preliminary Economic Assessment (PEA) is warranted for the Buffalo Ankerite project with the objective of evaluating the potential viability of an open pit and underground mine and determining the work required to advance the projects to definitive Pre-feasibility or Feasibility study.

25.2 FULLER

A significant deposit of gold mineralization has been delineated at the Fuller Project by diamond drilling throughout the long history of exploration and limited production. Both underground and surface drill holes were included in the Mineral Resource estimate, after RPA discarded a small number of underground drift back samples and sludge test hold samples that are present in the drill hole database.

The procedures for drilling, collection of data, sampling, assaying and check assaying carried out by Lexam have produced a drill hole database that is acceptable for Mineral Resource estimation, in the opinion of RPA. Results of 558 diamond drill holes to December 31, 2012, have been used by RPA to estimate Mineral Resources. This includes 206 surface diamond drill holes and 352 underground diamond drill holes. Continuity of mineralization on the Fuller Property is not amenable to wireframe constraint. Consequently, the Mineral Resource estimation utilized a dynamic anisotropy interpolation approach to constrain the interpolation and mimic the tightly folded stratigraphy.

At a 0.015 ounces per ton gold cut-off, the Fuller Property contains an open pit Indicated Mineral Resource of 5,878,000 tons at 0.049 ounces per ton Au totalling 290,000 ounces, and an Inferred Mineral Resource of 2,981,000 tons at 0.038 ounces per ton Au totalling 112,000 ounces. At a 0.075 ounces per ton gold cut-off, the Fuller Property contains an underground Indicated Mineral Resource of 361,000 tons at 0.168 ounces per ton Au totalling 61,000 ounces, and an Inferred Mineral Resource of 930,000 tons at 0.145 ounces per ton Au totalling 135,000 ounces. The Fuller Property has a total Indicated Mineral Resource of 6,239,000 tons at 0.056 ounces per ton Au totalling 351,000 ounces, and an Inferred Mineral Resource of 3,911,000 tons at 0.063 ounces per ton Au totalling 247,000 ounces. There are no Mineral Reserves estimated on the Fuller Property.

Further advancement of the project aimed at evaluating at a conceptual level the viability of an open pit and underground mine and completing a Preliminary Economic Assessment (PEA) is warranted, in RPA's opinion.

25.3 PAYMASTER

A significant deposit of gold mineralization has been delineated at the Paymaster Project by diamond drilling throughout the long history of exploration and underground production. Only surface drill holes were included in the Mineral Resource estimate. RPA discarded data from historical holes drilled in the 1920s and 1950s after a comparison of the assay results to recent drill holes showed low similarity.

The procedures for drilling, collection of data, sampling, assaying and check assaying carried out by Lexam have produced a drill hole database that is acceptable for Mineral Resource estimation, in the opinion of RPA. The drill hole database for the resource estimate contains the Placer Dome drilling (1995-1996) and the VG Gold/Lexam drilling (2009-2012). The database contains 263 drill holes drilled from surface. The resource related data consists of 145 drill holes for the open pit and 35 drill holes for the underground. There is some overlap between the underground intercepts and the open pit intercepts. Continuity of mineralization on the Paymaster Property was proved amenable to wireframe constraints.

The Mineral Resource estimation in the open pit used inverse distance to the fifth power for grade interpolation, while the underground model used inverse distance cubed. RPA used a higher exponent to interpolate the open pit model to help reduce the lateral interaction between the mineralization and internal waste bands included in some of the open pit resource wireframes. The underground wireframes had virtually no internal dilution.

At a 0.015 ounces per ton gold cut-off, the Paymaster Property contains an open pit Indicated Mineral Resource of 5,135,000 tons at 0.047 ounces per ton Au totalling 242,000 ounces, and an Inferred Mineral Resource of 1,542,000 tons at 0.047 ounces per ton Au totalling 72,000 ounces. At a 0.075 ounces per ton gold cut-off, the Paymaster Property contains an underground Inferred Mineral Resource of 239,000 tons at 0.179 ounces per ton Au totalling 43,000 ounces. The Paymaster Property has a total Indicated Mineral Resource of 5,135,000 tons at 0.047 ounces per ton Au totalling 242,000 ounces, and an Inferred Mineral Resource of 1,781,000 tons at 0.065 ounces per ton Au totalling 115,000 ounces. There are no Mineral Reserves estimated on the Paymaster Property.

Further advancement of the project aimed at evaluating at a conceptual level the viability of an open pit and underground mine and completing a Preliminary Economic Assessment (PEA) is warranted in RPA's opinion.

25.4 DAVIDSON TISDALE

The Davidson Tisdale gold deposit has been delineated by substantial historical exploration and diamond drill programs. A total of 430 holes. out of the 398 surface and 287 underground drill holes, intersected the mineralized domains, and were utilized for the mineral resource estimation.

The procedures for drilling, collection of data, sampling, assaying and check assaying carried out by Dome and Lexam have produced a drill hole database that is acceptable for Mineral Resource estimation, in P&E’s opinion.

Continuity of host structures on the Davidson Tisdale Property were readily amenable to wireframe constraints. The Mineral Resource estimation was constrained by mineral zone wireframes and used multiple search ellipses mapped to the zones’ orientations and Ordinary Kriging for grade interpolation. a cut-off grade of 0.5 g/t Au (0.015 oz/ton Au) was applied for potentially open pittable gold mineralization and 2.6 g/t Au (0.045 oz/ton Au) cut-off grade for underground mineable mineralization. The latter cut-off grade is based on underground breakeven costs and a gold price of $1,600/oz.

At a 0.5 g/t Au (0.015 oz/ton Au) cut-off, the Davidson Tisdale Property contains an open pit Indicated and Measured Mineral Resource of 625,000 tonnes at 2.44 g/t Au. This resource contains a total of 33,600 oz of gold that are 68.5% attributable to Lexam.

At a 2.6 g/t Au (0.075 oz/ton Au) cut-off, the Davidson Tisdale Property contains an underground Indicated and Measured Mineral Resource of 59,000 tonnes at 5.43 g/t Au that contains 7,000 oz of gold that are 68.5% attributable to Lexam. In addition the deposit contains 71,000 tonnes of Inferred Resources grading 4.2 g/t Au that contains 6,600 ounces of contained gold that are 68.5% attributable to Lexam.

In total, the Davidson Tisdale Property has an open pit and underground Indicated plus Measured Mineral Resource of 684.000 tonnes grading 2.7 g/t Au along with an underground Inferred Mineral Resource of 71,000 tonnes grading 4.20 g/t Au. The total contained ounces of gold that are 68.5 % attributable to Lexam are 40,600 oz in the Indicated+Measured category and 6,600 oz in the Inferred category.

There are no Mineral Reserves estimated on the Davidson Tisdale property.

In P&E's opinion, the completion of a scoping study or a Preliminary Economic Assessment (PEA) is warranted for the Davidson Tisdale project with the objective of evaluating the potential viability of an open pit and underground mine.

26.0 RECOMMENDATIONS

26.1 BUFFALO ANKERITE AND DAVIDSON TISDALE

P&E has the following recommendations:

26.1.1 Buffalo Ankerite

With respect to the Buffalo Ankerite project, P&E has the following recommendations:

- Further advance the project by initiating engineering, metallurgical, geotechnical environmental, permitting, and other studies aimed at evaluating the viability of an open pit and underground mine and completing a Preliminary Economic Assessment (PEA).
- With respect to future resource estimates, Lexam should tie the mine grid into the Universal Transverse Mercator (UTM) NAD 83 grid system and Lexam should convert all units in the database to metric.
- P&E identified a number of instances during assay verification wherein resource assays in the drill hole database were much lower or zero with respect to laboratory certificates. P&E recommends investigating these discrepancies with the objective of incorporating correct values in future resource estimation updates.
- P&E identified a number of drill holes for which implausible deviation is recorded in the down hole surveys. Lexam should review available documentation for these surveys and correct or delete problematic azimuth or dip readings.
- Begin a program of regular submission of blank material with regular drill core samples.
- Use certified commercial standards, with certified gold values to monitor the laboratory performance. RPA recommends the inclusion of at least three different grades of CRMs into the sampling stream: low grade (near cut-of), average grade and high grade.
- Begin to incorporate duplicate samples (field, pulp and coarse reject material) into the Fuller and Paymaster Project QA/QC protocol for drill programs.
- Implement a QA monitoring system used to detect failed batches, and in turn, identify sample batches for reanalysis.
- The underground resources above 10,000 ft elevation were wireframed at the open pit cut-off grade of 0.015 oz/ton Au and these resources were determined at a block cut-off grade of 0.075 oz/ton Au. Once the open pit shell is finalized in a scoping/PEA study, the resources below the pit to the 10,000 ft elevation should be re-wireframed at the higher underground cut-off grade and re-estimated with the expectation of improving the average grade of these resources.
- The widths of stopes modelled may be over stated resulting in excess volume, for past production and development, being removed from resources and thus a more conservative resource tonnage estimate made. Lexam should review the stope modelling to confirm if this is the case and remodel problematic openings where necessary.
- Resource classification can be improved with infill drilling along the more widely spaced drilling areas. P&E recommends a spacing of 150 ft by 150 ft (50 m x 50 m) in the North Zone and 100 x 100 feet (30 m x 30 m) in the South Zone. The

scoping study/PEA should determine which portions of the Inferred Mineral Resources are to be prioritized for additional drilling.

- Preliminary rock geotechnical investigations are needed for open pit slope optimization so geotechnical data should be routinely collected when logging drill core.
- Bulk density determinations should be routinely carried out in mineralization and waste in any future drilling.

26.1.2 Davidson Tisdale

With respect to the Davidson Tisdale project, P&E has the following recommendations:

- A systematic density testing program should be undertaken in future drilling programs.
- Further advance the project by initiating engineering, metallurgical, geotechnical environmental, permitting, and other studies aimed at evaluating the viability of an open pit and underground mine and completing a Preliminary Economic Assessment (PEA).
- Resource classification in the underground portion (Inferred upgraded to Indicated) can be potentially accomplished with infill drilling along the more widely spaced drilling areas. P&E recommends drill hole spacing of 25m by 20 m. The scoping study/PEA should determine which portions of the Inferred Mineral Resources are to be prioritized for additional drilling.

26.2 FULLER AND PAYMASTER

26.2.1 General

RPA recommends that Lexam further advance the project by initiating engineering, metallurgical, environmental, permitting, and other studies aimed at evaluating at a conceptual level the viability of an open pit and underground mine and completing a Preliminary Economic Assessment (PEA).

Lexam should tie the mine grids into the Universal Transverse Mercator (UTM) grid system and convert all units to metric.

26.2.2 Resource Model

<u>Fuller</u>

The block model is sufficiently reliable for estimation of Mineral Resources. In order to support mine planning, however, areas of the Fuller block models constructed by RPA for underground and open pit resources may require refinements in the future aimed at improving mineralization continuity. Consequently, and RPA recommends adding trend lines and surfaces locally to better align the orientation of the search ellipse.

Paymaster

Future refinements could include adding more search domains to better align the grade interpolation in some areas.

26.2.3 QA/QC

Fuller & Paymaster

- Begin a program of regular submission of blank material with regular drill core samples.
- Use certified commercial standards, with certified gold values to monitor the laboratory performance. RPA recommends the inclusion of at least three different grades of CRMs into the sampling stream: low grade (near cut-of), average grade and high grade.
- Begin to incorporate duplicate samples (field, pulp and coarse reject material) into the Fuller and Paymaster Project QA/QC protocol for drill programs.
- Implement a QA monitoring system used to detect failed batches, and in turn, identify sample batches for reanalysis.
- Lexam should use an accredited secondary laboratory to assess the assay accuracy of the primary laboratory

Paymaster

Although Lexam has implemented a fairly rigorous check sample program, it is RPA's opinion that the expected grades of the check samples should be representative of the deposit grade distribution.

26.2.4 Drilling

Fuller

Mineral resources can be increased by investigating gold mineralization located on the periphery of the current geological model, particularly in areas west of the Fuller Syncline. This would improve resource classification by upgrading areas not classified to Inferred and Inferred blocks to Indicated. It is RPA's opinion that additional drilling would provide valuable information on the continuity of grade and allow better local orientation of the search ellipse.

Furthermore, resource classification can be improved with infill drilling along the more widely spaced drilling areas. Within the Open Pit resource, RPA recommends a spacing of 100 x 100 feet along the east and west limbs of the Fuller Syncline.

RPA recommends Lexam begin preliminary rock geotechnical investigations. Lexam should routinely collect geotechnical data when logging drill core.

Paymaster

Resource classification for the underground could benefit from infill drilling, while it would also bring useful information for the open pit resource

RPA recommends Lexam begin preliminary rock geotechnical investigations. Lexam should routinely collect geotechnical data when logging drill core.

26.2.5 Density

Fuller & Paymaster

RPA recommends that the lithological model be updated and supplemented with additional bulk density determinations throughout the Fuller and Paymaster deposits in order to decrease density uncertainties for the mineralized and waste rock.

26.3 RECOMMENDATIONS AND PROPOSED BUDGET

Recommendations include limited diamond drilling and a preliminary economic assessment (PEA).

26.3.1 Drilling

Buffalo Ankerite
An exploration programme is proposed for the property with emphasis on exploring the depth extent of the North Zone, in particular the area below 2000 feet below surface where historic records indicate a mineralized Quartz-Feldspar Porphyry (QFP) body with gold mineralization. The proposed programme will concentrate on the continuation of the exploration of the Buffalo Ankerite North Zone, following up on the success of the drilling to date. Emphasis will be placed on expanding the depth extent of the gold mineralization. Upon completion of the drill programme an examination of the data should be conducted in order to determine where to explore the porphyry at depth as information indicates that economic values have been obtained at the 2200 to 3500 foot levels.
No further drilling is necessary on the South Zone as the eastern portion has been drilled to the plunge line and in the western section the better potential at depth is in the North Zone area.

Fuller

Two targets at depth remain not adequately tested on the Fuller property:

- Follow-up on holes VG96-26 and 96-30
- Below the 1500 foot level.

VG-96-26 intersected 3.9 gpt Au/63.5m including 7.5 gpt Au/8.9m and 8.3 gpt Au/3.2m and 4.8 gpt Au/16.7m at 1300 feet below surface.

VG-96-30 intersected 5.0 gpt Au/10.9m and 9.6 gpt Au/8.8m at 1600 feet below surface.

Two holes were drilled in 2009 - VGF-09-111 and VGF-09-112 to test this target. They intersected respectively 70m west/90m below and 40m east/200m below the previous drilling. The best result was 09-111 with 8.2 gpt Au/2.0m corresponding to the previous intersections. These holes are excessively distant from the previous drilling to eliminate this target.

Three (3) holes are recommended to intersect closer to the previous drilling to verify the mineralization.

<u>Paymaster</u>

No drilling is recommended at this time at the Paymaster property

<u>Davidson Tisdale</u>

No drilling is recommended at this time at the Tisdale property

<u>Budget</u>

TABLE 26.1
PROPOSED BUDGET

Project	Activity	Units	$/Unit	Cost
Drilling				
Buffalo Ankerite	drilling	5,000	$160	$800,000
Fuller	drilling	1,000	$160	$160,000
PEA				$300,000
Subtotal				$1,260,000
Admin - 15%				$190,000
Total				**$1,450,000**

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28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

WAYNE D. EWERT, P.GEO.

I, Wayne D. Ewert, P. Geo., residing at 10 Langford Court, Brampton, Ontario, L6W 4K4, do hereby certify that:

1. I am a principal of P & E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada" (the "Technical Report"),with an effective of June 1, 2013.
3. I graduated with an Honours Bachelor of Science degree in Geology from the University of Waterloo in 1970 and with a PhD degree in Geology from Carleton University in 1977. I have worked as a geologist for a total of 42 years since obtaining my B.Sc. degree. I am a P. Geo., registered in the Province of Saskatchewan (APEGS No. 16217), the Province of British Columbia (APEGBC No. 18965), the Province of Ontario (APGO No. 0866) and the Province of Newfoundland and Labrador (PEG No. 06005).

   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:

   - Principal, P&E Mining Consultants Inc. ....2004 – Present
   - Vice-President, A.C.A. Howe International Limited... 1992 – 2004
   - Canadian Manager, New Projects, Gold Fields Canadian Mining Limited............................ 1987 – 1992
   - Regional Manager, Gold Fields Canadian Mining Limited.. 1986 – 1987
   - Supervising Project Geologist, Getty Mines Ltd. .... 1982 – 1986
   - Supervising Project Geologist III, Cominco Ltd. .... 1976 – 1982
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 2-10, 23, 24 and 27 of this Technical Report
6. I am independent of the Issuer applying all of the tests in section 1.5 of National Instrument 43-101.
7. I have not had prior involvement with the project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 1, 2013
Signed Date: June 21, 2013

{SIGNED AND SEALED}
[Wayne Ewert]

Dr. Wayne D. Ewert P.Geo.

CERTIFICATE OF QUALIFIED PERSON

EUGENE J. PURITCH, P. ENG.

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

1. I am an independent mining consultant and President of P & E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada" (the "Technical Report"), with an effective date of June 1, 2013.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for Bachelor's Degree in Engineering Equivalency. I am a mining consultant currently licensed by the Professional Engineers of Ontario (License No. 100014010) and registered with the Ontario Association of Certified Engineering Technicians and Technologists as a Senior Engineering Technologist. I am also a member of the National and Toronto Canadian Institute of Mining and Metallurgy.

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

   I have practiced my profession continuously since 1978. My summarized career experience is as follows:
   - Mining Technologist - H.B.M.& S. and Inco Ltd.,...1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd.,..1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine,..1984-1986
   - Self-Employed Mining Consultant – Timmins Area,..1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, ...1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator,..1995-2004
   - President – P & E Mining Consultants Inc,..2004-Present
4. I have visited not the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 13 and co-authoring Section 14.4 of the Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 1, 2013
Signed Date: June 21, 2013

{SIGNED AND SEALED}
[Eugene Puritch]

____
Eugene J. Puritch, P.Eng.

CERTIFICATE of AUTHOR

TRACY J. ARMSTRONG, P.GEO.

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

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc. and have worked as a geologist continuously since my graduation from university in 1982.
2. This certificate applies to the technical report titled “Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada” (the “Technical Report”), with an effective date of June 1, 2013.
3. I am a graduate of Queen’s University at Kingston, Ontario with a B.Sc. (HONS) in Geological Sciences (1982). I am a geological consultant currently licensed by the Order of Geologists of Québec (License 566), the Association of Professional Geoscientists of Ontario (License 1204) and the Association of Professional Engineers and Geoscientists of British Columbia, (Licence No. 34720).

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. This report is based on my personal review of information provided by the Issuer and on discussions with the Issuer’s representatives. My relevant experience for the purpose of the Technical Report is:

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

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring of Sections 11.1 and 11.4, and co-authoring Sections 12.1 and 12.4 of this Technical Report.
6. I am independent of issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 1, 2013
Signing Date: June 21, 2013

{SIGNED AND SEALED}

[Tracy J. Armstrong]

____
Tracy J. Armstrong, P.Geo.

CERTIFICATE OF AUTHOR

YUNGANG WU, P.GEO.

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

1. I am an independent consulting geologist contracted by P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada" (the "Technical Report"), with an effective date of June 1, 2013.
3. I am a graduate of Jilin University, China with a Master Degree in Mineral Deposits (1992). I am a geological consultant and a registered practising member of the Association of Professional Geoscientist of Ontario (Registration No. 1681). I am also a member of the Ontario Prospectors Association.

I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
My relevant experience for the purpose of the Technical Report is as follows:

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

4. I have not visited the property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 25.4 and co-authoring Sections 14.4, 26.1 and 26.3 along with the Appendix of the Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective date: June 1 2013
Signing Date: June 21, 2013

{SIGNED AND SEALED}

[Yungang Wu]

Yungang Wu, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

ANTOINE R. YASSA, P. GEO.

I, Antoine R. Yassa, P. Geo., residing at 3602 Rang des Cavaliers, Rouyn-Noranda, Quebec, J0Z 1Y2, do hereby certify that:

I am an independent geological consultant contracted by P&E Mining Consultants Inc.

1. This certificate applies to the technical report titled "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada" (the "Technical Report"), with an effective date of June 1, 2013.

2. I am a graduate of Ottawa University at Ottawa, Ontario with a B.Sc (HONS) in Geological Sciences (1977). I have worked as a geologist for about 30 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Order of Geologists of Québec (License No 224) and a practising member of the APGO (Registration Number 1890).

   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:

   - Minex Geologist (Val d'Or), 3D Modeling (Timmins), Placer Dome 1993-1995
   - Database Manager, Senior Geologist, West Africa, PDX 1996-1998
   - Senior Geologist, Database Manager, McWatters Mine 1998-2000
   - Database Manager, Gemcom modeling and Resources Evaluation (Kiena Mine) QAQC Manager (Sigma Open pit), McWatters Mines... 2001-2003
   - Database Manager and Resources Evaluation at Julietta Mine, Far-East Russia, Bema Gold Corporation 2003-2006
   - Consulting Geologist .... since 2006

3. I have visited the Buffalo Ankerite and Davidson Tisdale Properties on November 3, 4, and 6, 2012.

4. I am responsible for co-authoring Sections 12.1, 12.4 and 14.4 of the Technical Report.

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

6. I have had no prior involvement with the Property that is the subject of this Technical Report.

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

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

Effective Date: June 1, 2013
Signed Date: June 21, 2013

{SIGNED AND SEALED}
[Antoine Yassa]

____
Antoine R. Yassa, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

RICHARD E. ROUTLEDGE, P.GEO.

I, Richard E. Routledge, P.Geo., residing at 82 Oriole Drive, Holland Landing, Ontario, L9N 1H3, do hereby certify that:

1. I am an independent Consulting Geologist who has been contracted by P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada" (the "Technical Report"), with an effective date of June 1, 2013.
3. I graduated with a Bachelor of Science degree, Major in Geology, from Sir George Williams (Concordia) University in 1971 and with a Masters degree in Applied Exploration Geology from McGill University in 1973. I have worked as a geologist for a total of 38 years since post-graduation. I am a Professional Geologist registered in the Province of Ontario (APGO No. 1354) and licensed by the Northwest Territories (NAPEGG No. L744).

   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:

   - Independent Consulting Geologist. ....2011 – Present
   - Roscoe Postle Associates Inc., Consulting Geologist .... 1998 – 2011
   - Independent Consulting Geologist.. 1994 – 1997
   - Vice President Exploration, Greater Lenora Resources Corp. .... 1993 – 1994
   - Teck Explorations Ltd, Evaluations and Mineral Commodities Geologist. .... 1985 – 1992
   - Derry, Michener, Booth & Wahl, Exploration and Consulting Geologist. .... 1973 – 1985

4. I have not visited the property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 25.1 and co-authoring Sections 14.1, 26.1 and 26.3 of the Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 1, 2013
Signed Date: June 21, 2013

{SIGNED AND SEALED}
[Richard E. Routledge]

____
Richard E. Routledge, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

KATHARINE M. MASUN, M.Sc., MSA, P.GEO.

I, Katharine M. Masun, P.Geo., as an author of this report entitled "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada" prepared for Lexam VG Gold Inc. with an effective date of June 1, 2013, do hereby certify that:

1. I am a Senior Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.
2. I am a graduate of Lakehead University, Thunder Bay, Ontario, Canada, in 1997 with an Honours Bachelor of Science degree in Geology and in 1999 with a Master of Science degree in Geology. I am also a graduate Ryerson University in Toronto, Ontario, Canada, in 2010 with a Master of Spatial Analysis.
3. I am registered as a Professional Geologist in the Province of Ontario (Reg. #1583). I have worked as a geologist for a total of 15 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Review and report as a professional geologist on several exploration projects in for regulatory requirements
   - Project Geologist on numerous field and drilling programs in North American, South American, Asia and Australia
   - Experience with Gemcom block modelling

4. 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.
5. I visited the Fuller Project on November 20-21, 2012.
6. I am responsible for authoring Sections 11.2, 12.2, 14.2, 25.2, and co-authoring Sections 26.2 and 26.3 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. To the best of my knowledge, information, and belief, the Sections 1, 11, 12, 14 and 25-27 for which I am responsible in the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: June 1, 2013
Signed Date: June 21, 2013

{SIGNED AND SEALED}
[Katharine M. Masun]

____
Katharine M. Masun, M.Sc.,MSA, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

TUDOREL CIUCULESCU, M.Sc., P.GEO.

I, Tudorel Ciuculescu, M.Sc., P.Geo., as an author of this report entitled "Technical Report and Updated Resource Estimate on the Buffalo Ankerite, Fuller, Paymaster, and Davidson Tisdale Gold Deposits Porcupine Mining Division North-Eastern Ontario, Canada" prepared for Lexam VG Gold Inc. with an effective date of June 1, 2013, do hereby certify that:

1. I am Senior Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.
2. I am a graduate of University of Bucharest with a B.Sc. degree in Geology in 2000 and University of Toronto with a M.Sc. degree in Geology in 2003.
3. I am registered as a Professional Geologist in the Province of Ontario (Reg. #1882). I have worked as a geologist for a total of 12 years since my graduation. My relevant experience for the purpose of the Technical Report is:

    - Preparation of Mineral Resource estimates.
    - Over 5 years of exploration experience in Canada and Chile.

4. 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 NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Paymaster Project on November 29-30, 2012.
6. I am responsible for authoring Sections 11.3, 12.3, 14.3, 25.3 and co-authoring Sections 26.2, and 26.3 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. To the best of my knowledge, information, and belief, the Sections 1, 11, 12, 14 and 25-27 for which I am responsible in the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

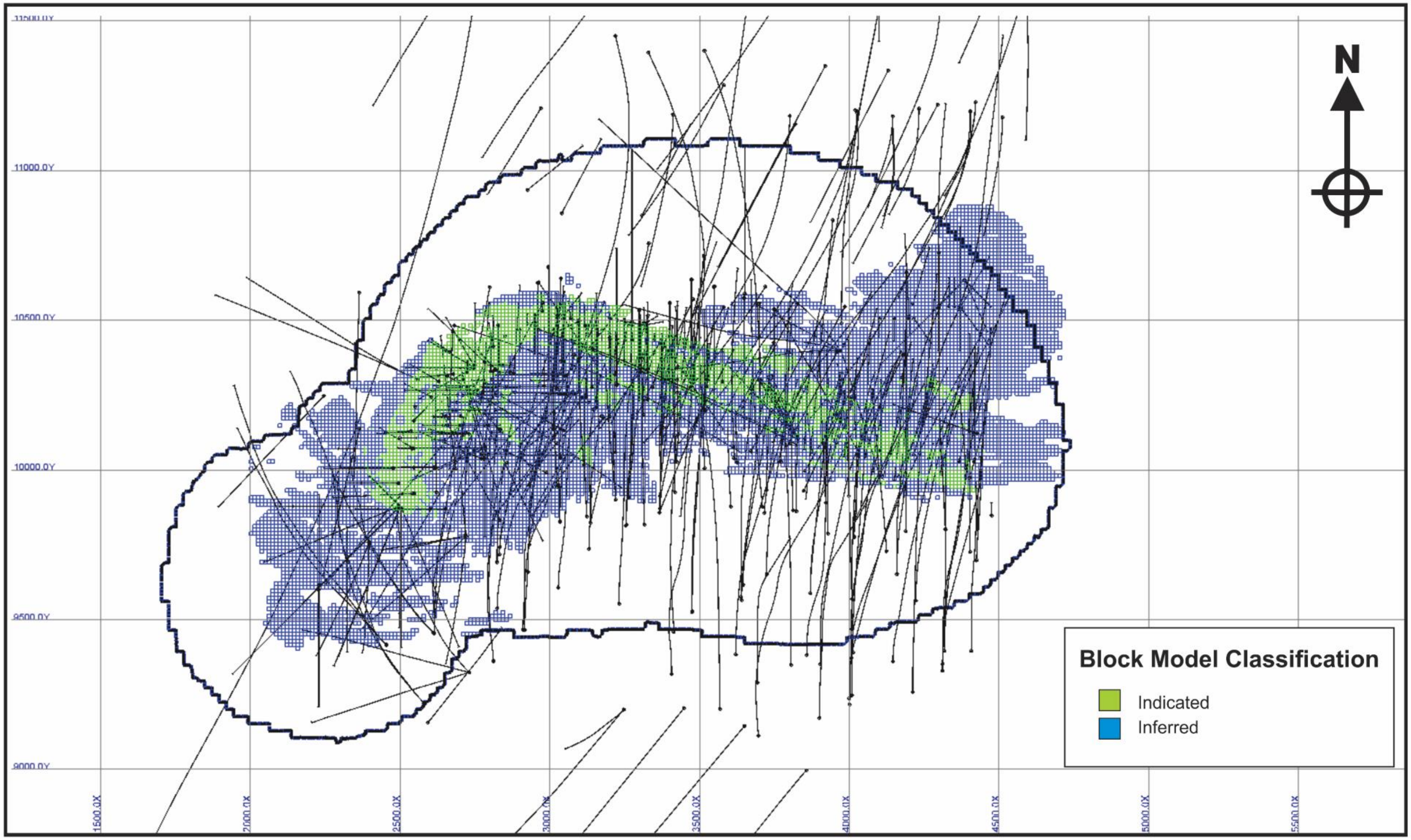
Effective Date: June 1, 2013
Signed Date: June 21, 2013

{SIGNED AND SEALED}
[Tudorel Ciuculescu]

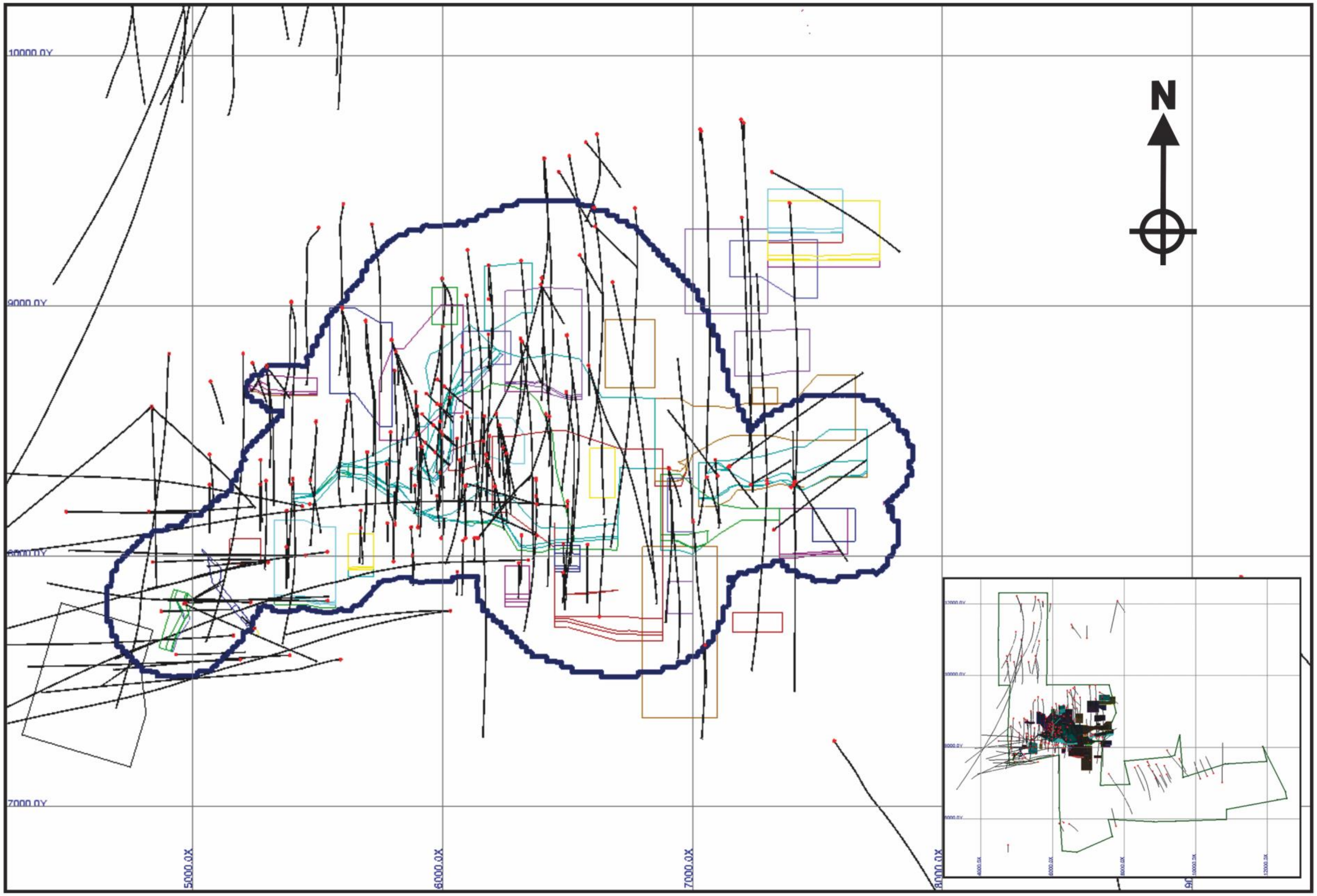
____
Tudorel Ciuculescu, M.Sc., P.Geo.

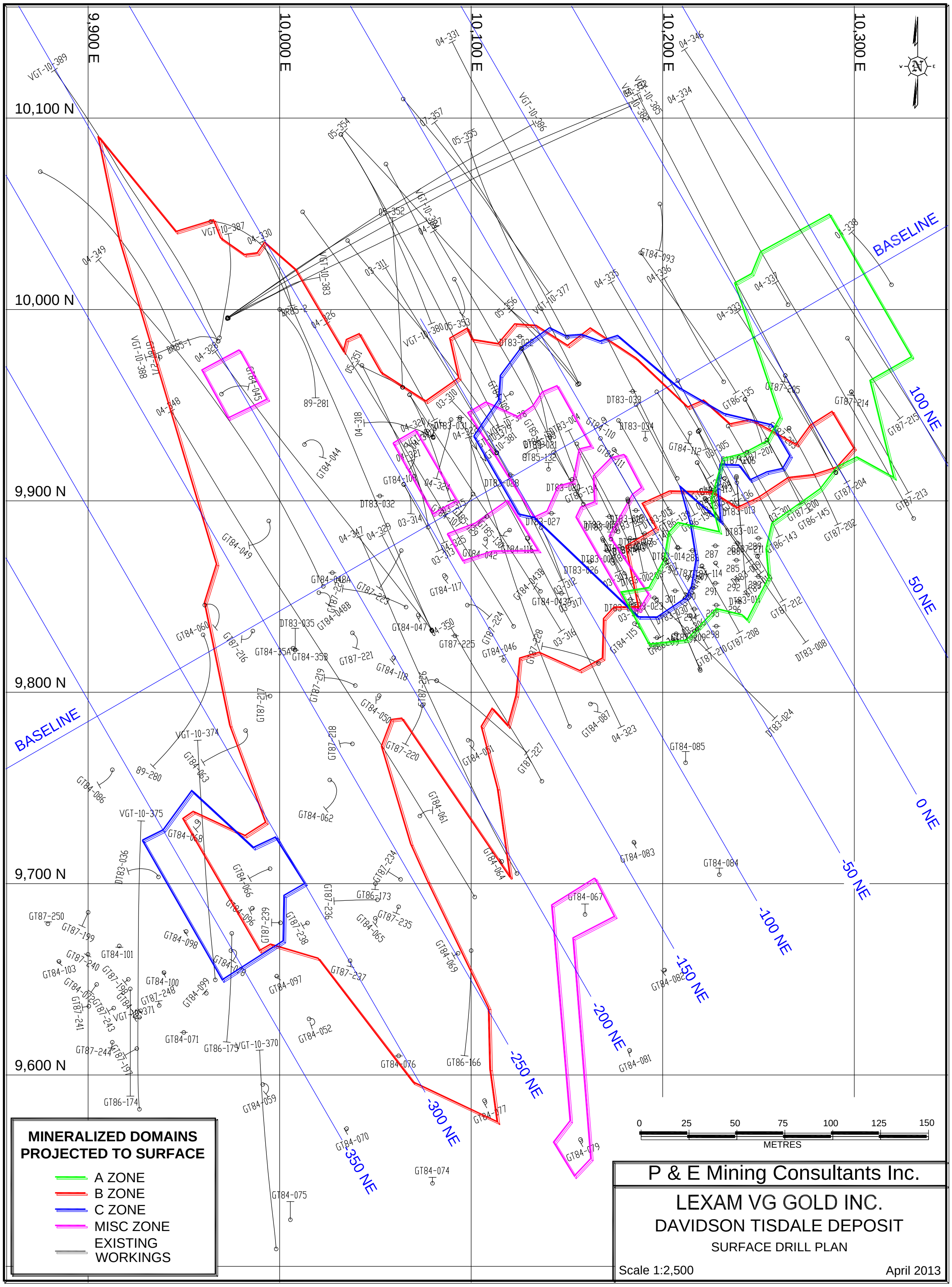
APPENDIX I. DRILL HOLE PLANS

Fuller Project - Plan View of Drill Hole Traces

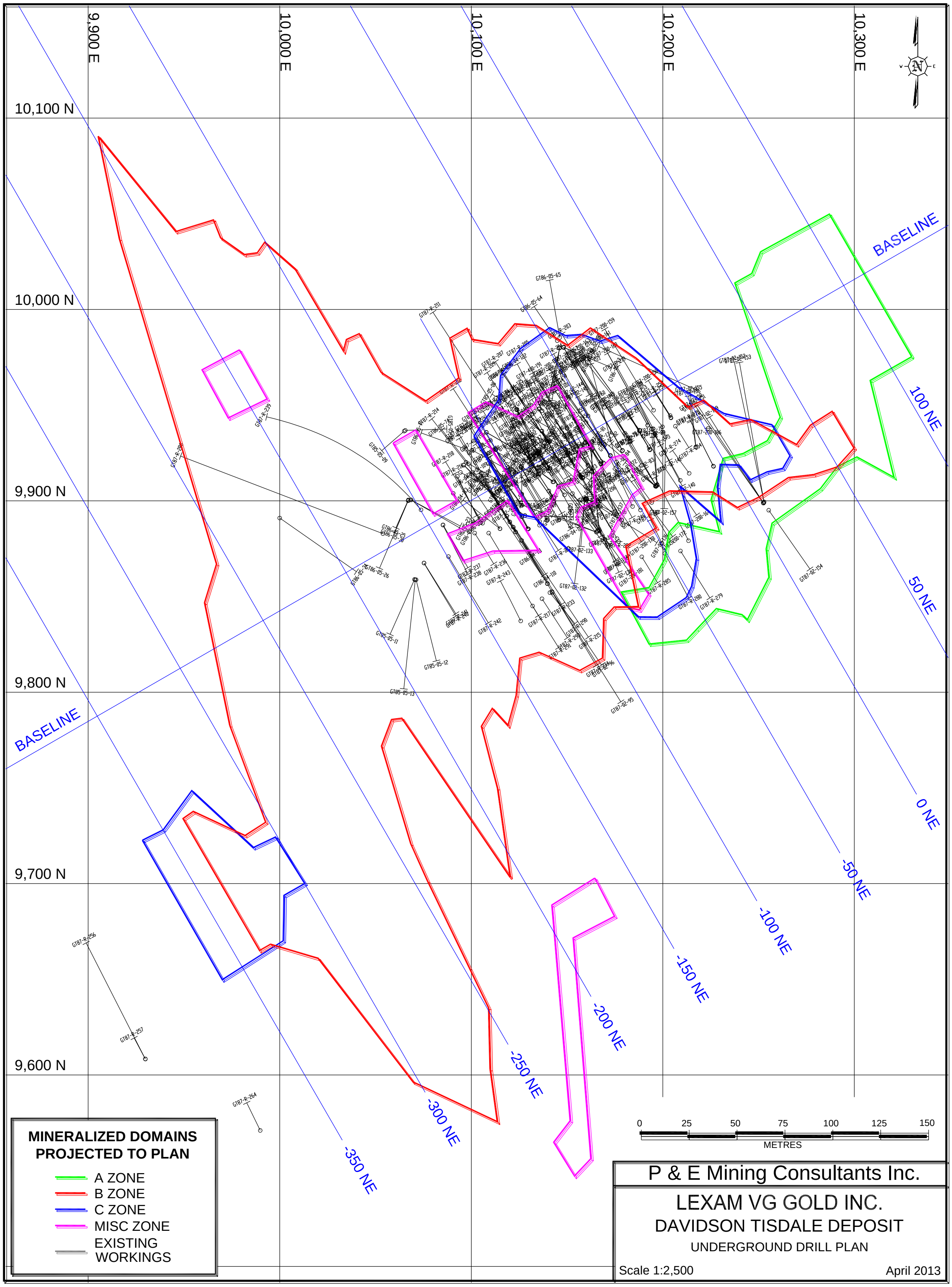


Paymaster Project - Plan View of Drill Hole Traces





MINERALIZED DOMAINS PROJECTED TO SURFACE
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
SURFACE DRILL PLAN
Scale 1:2,500
April 2013
0 25 50 75 100 125 150
METRES
9,900 E
10,000 E
10,100 E
10,200 E
10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
9,600 N
BASELINE
BASELINE
100 NE
50 NE
0 NE
-50 NE
-100 NE
-150 NE
-200 NE
-250 NE
-300 NE
-350 NE


9,900 E
10,000 E
10,100 E
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10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
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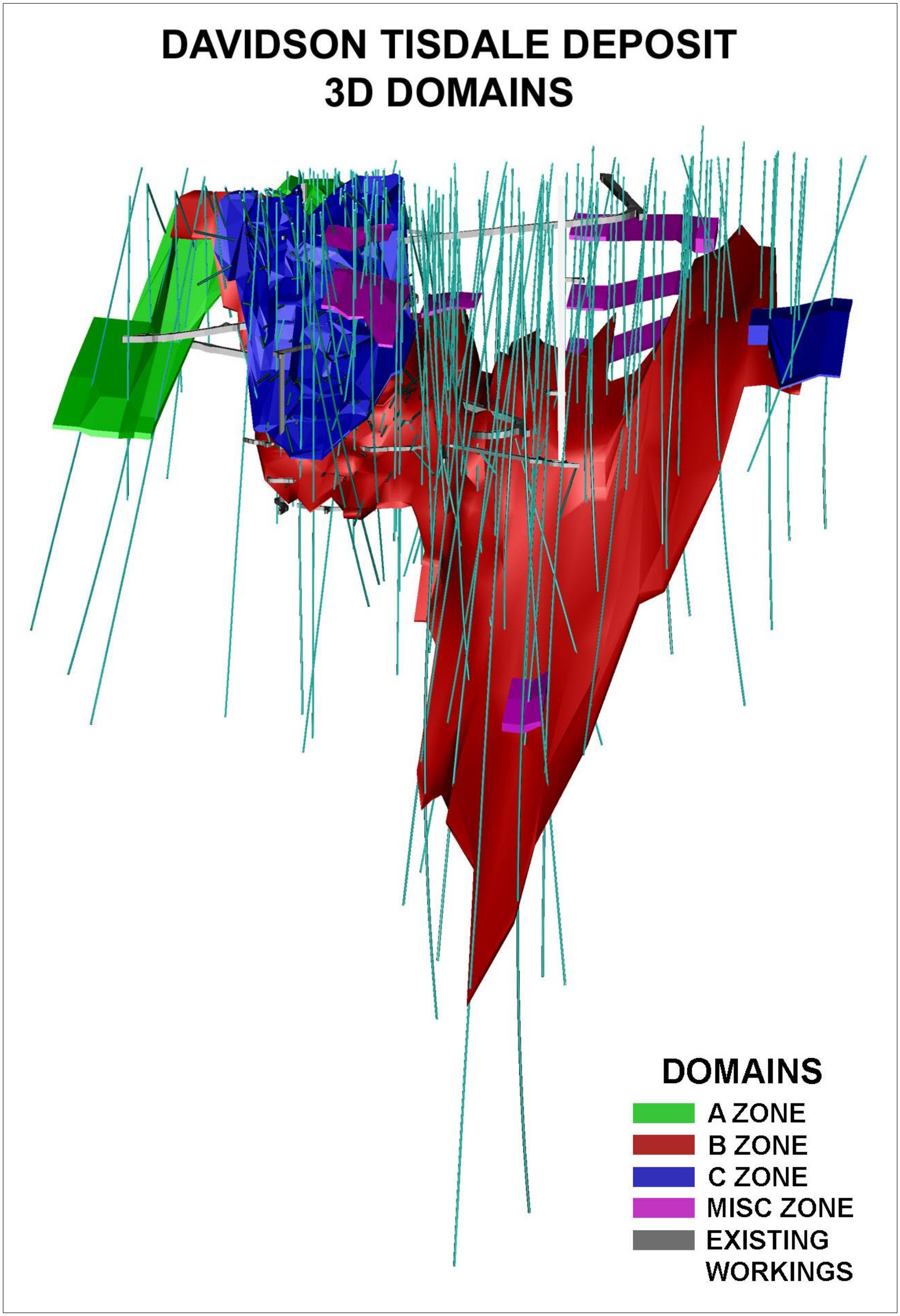
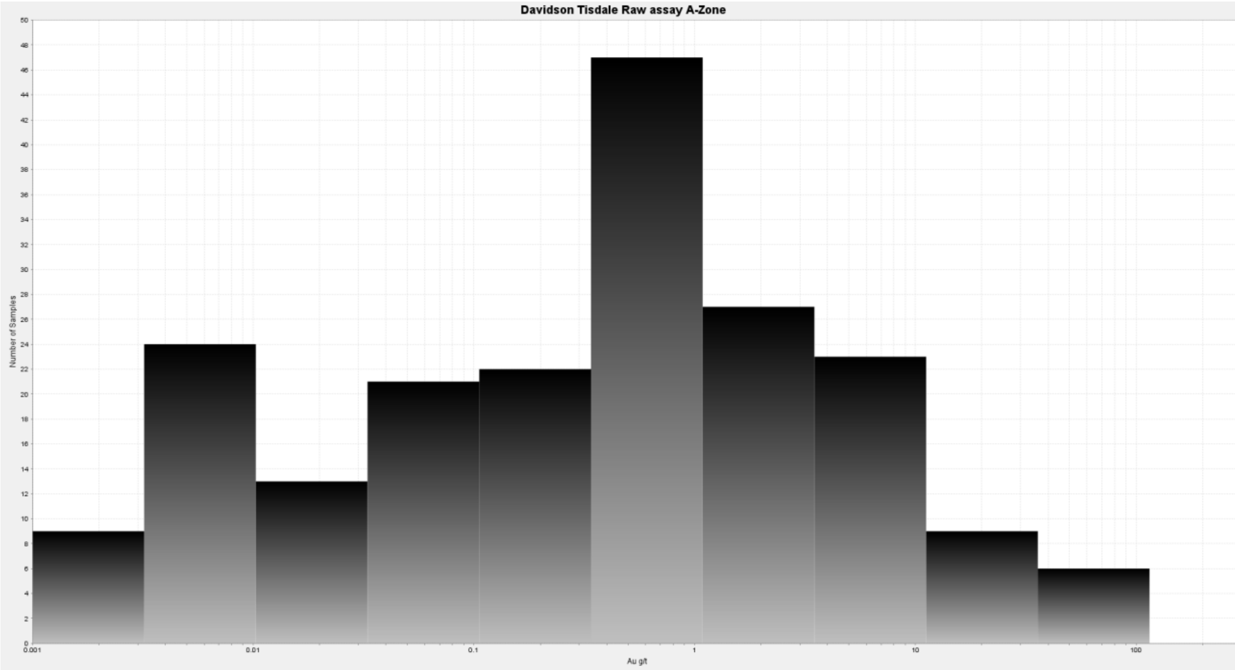
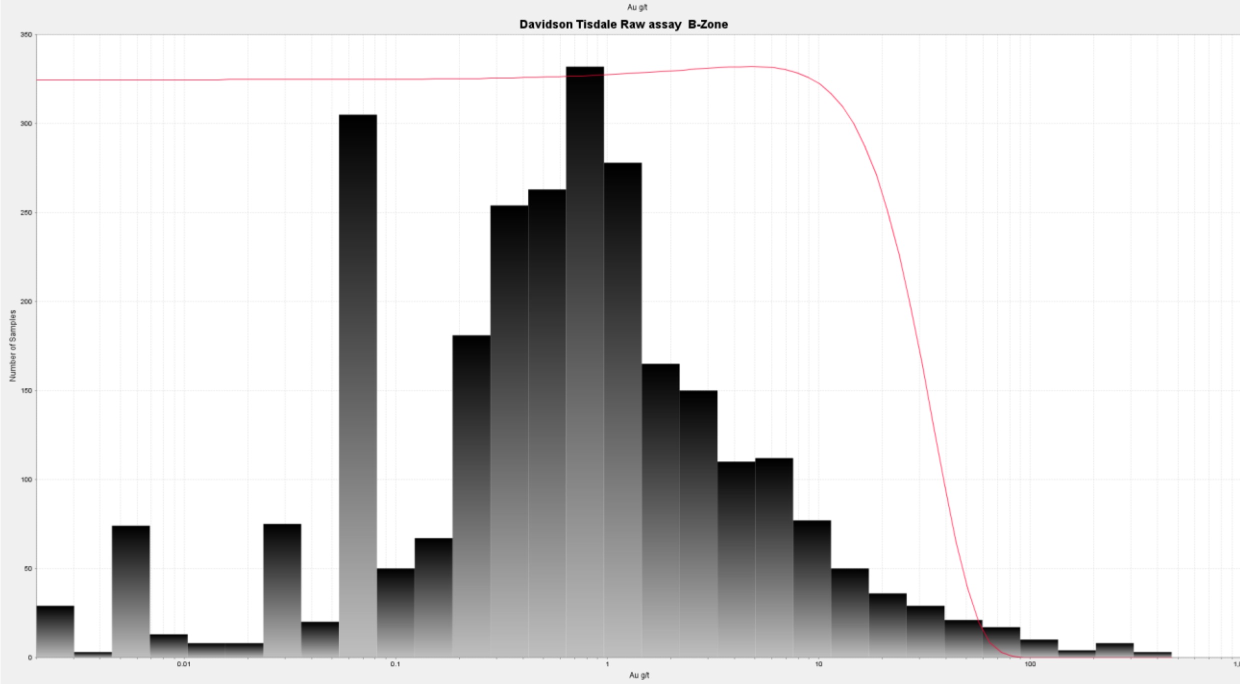
BASELINE
BASELINE
100 NE
50 NE
0 NE
-50 NE
-100 NE
-150 NE
-200 NE
-250 NE
-300 NE
-350 NE
MINERALIZED DOMAINS
PROJECTED TO PLAN
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING
WORKINGS
0
25
50
75
100
125
150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
UNDERGROUND DRILL PLAN
Scale 1:2,500
April 2013

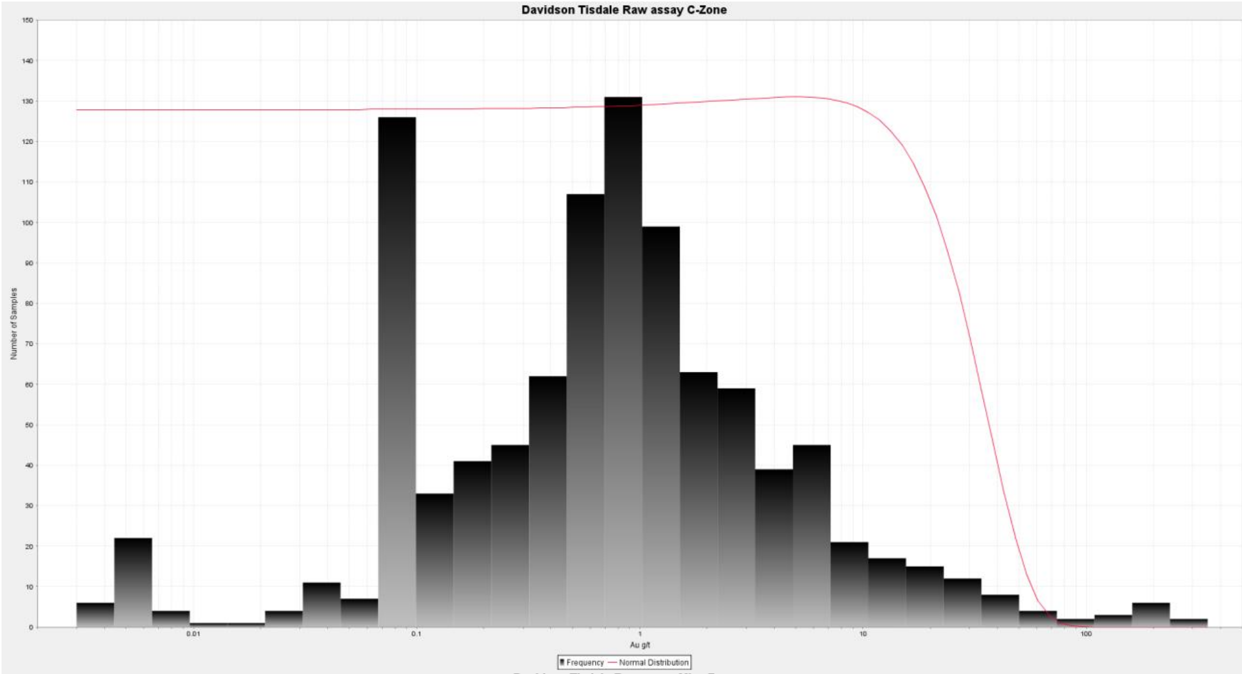
APPENDIX II. 3D DOMAINS

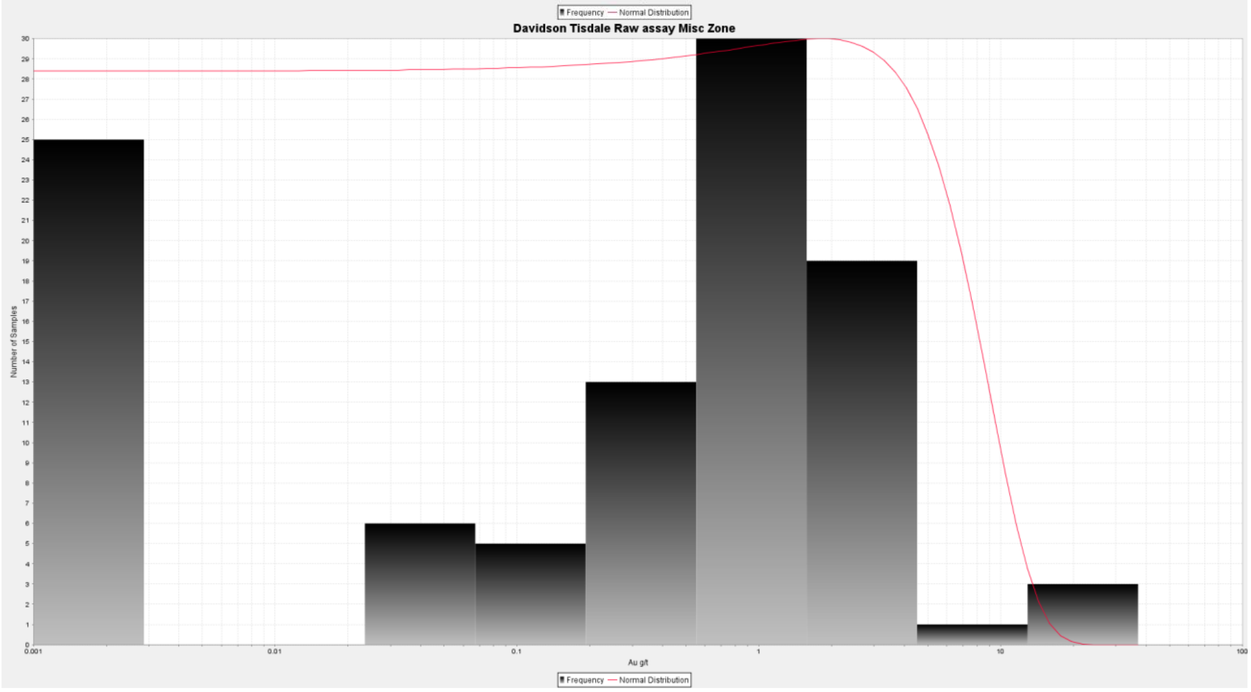

DAVIDSON TISDALE DEPOSIT
3D DOMAINS
DOMAINS
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING
WORKINGS

APPENDIX III. LOG NORMAL HISTOGRAMS


Davidson Tisdale Raw assay A-Zone
Number of Samples
Au g/t


Davidson Tisdale Raw assay B-Zone
Number of Samples
Au g/t

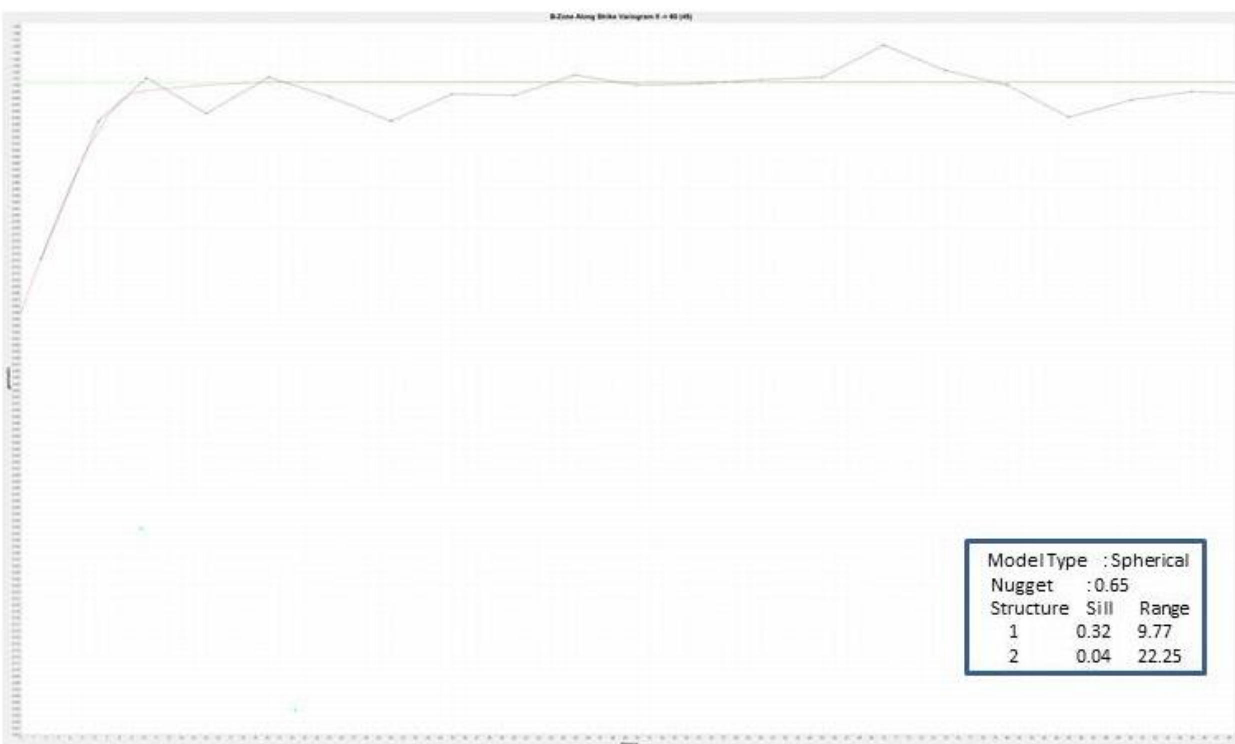

Davidson Tisdale Raw assay C-Zone
Number of Samples
Au g/t
Frequency
Normal Distribution


Davidson Tisdale Raw assay Misc Zone
Number of Samples
Au g/t
Frequency
Normal Distribution

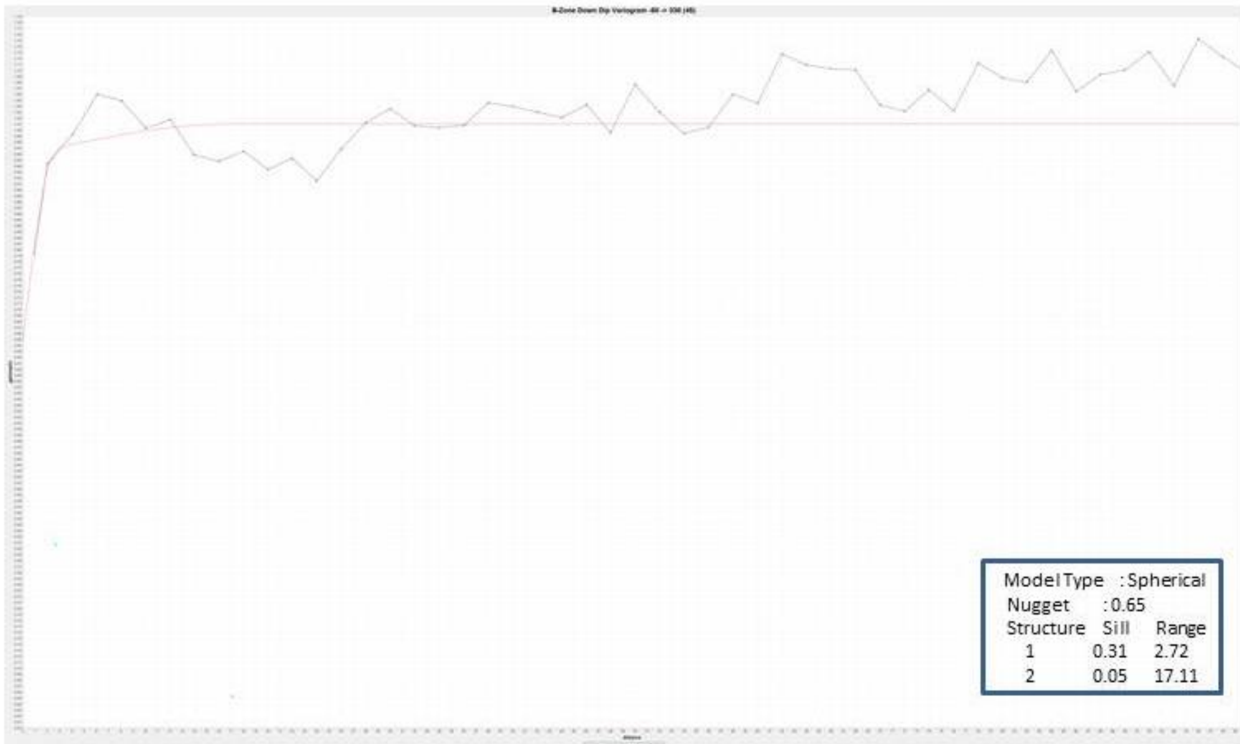
APPENDIX IV. VARIOGRAMS

Davidson Tisdale B-Zone Omni Variogram

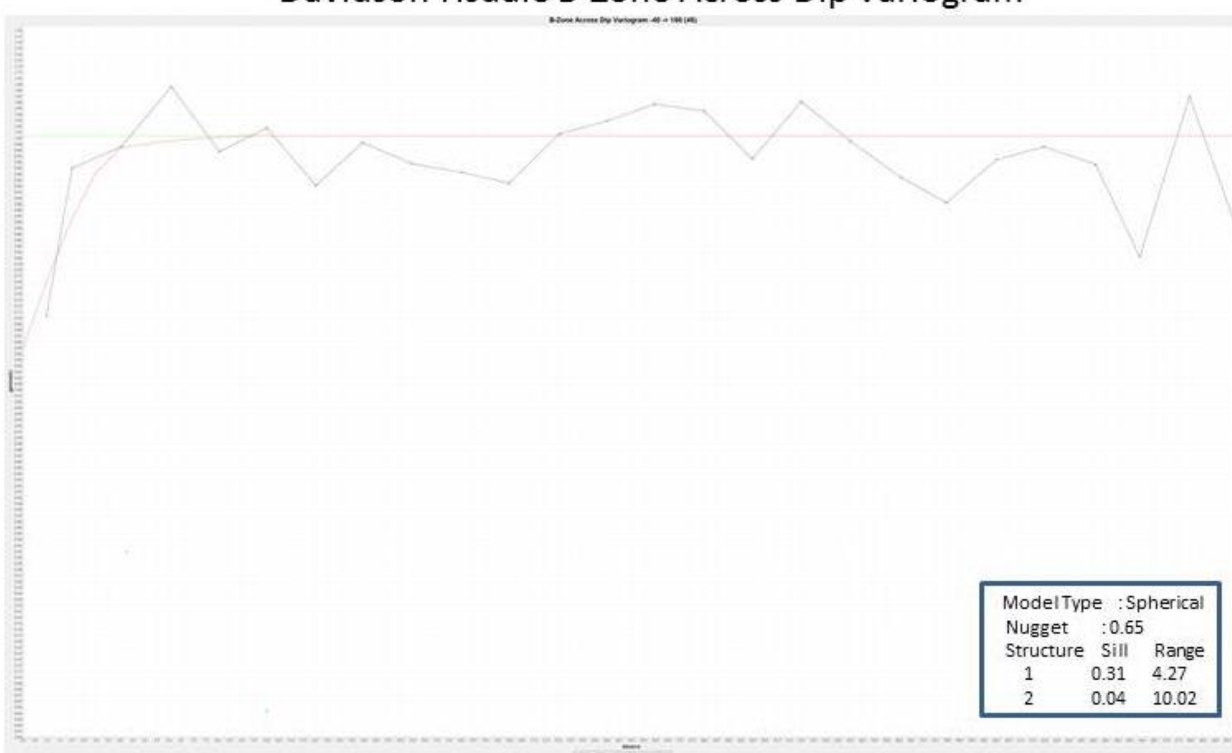
Davidson Tisdale B-Zone Along Strike Variogram



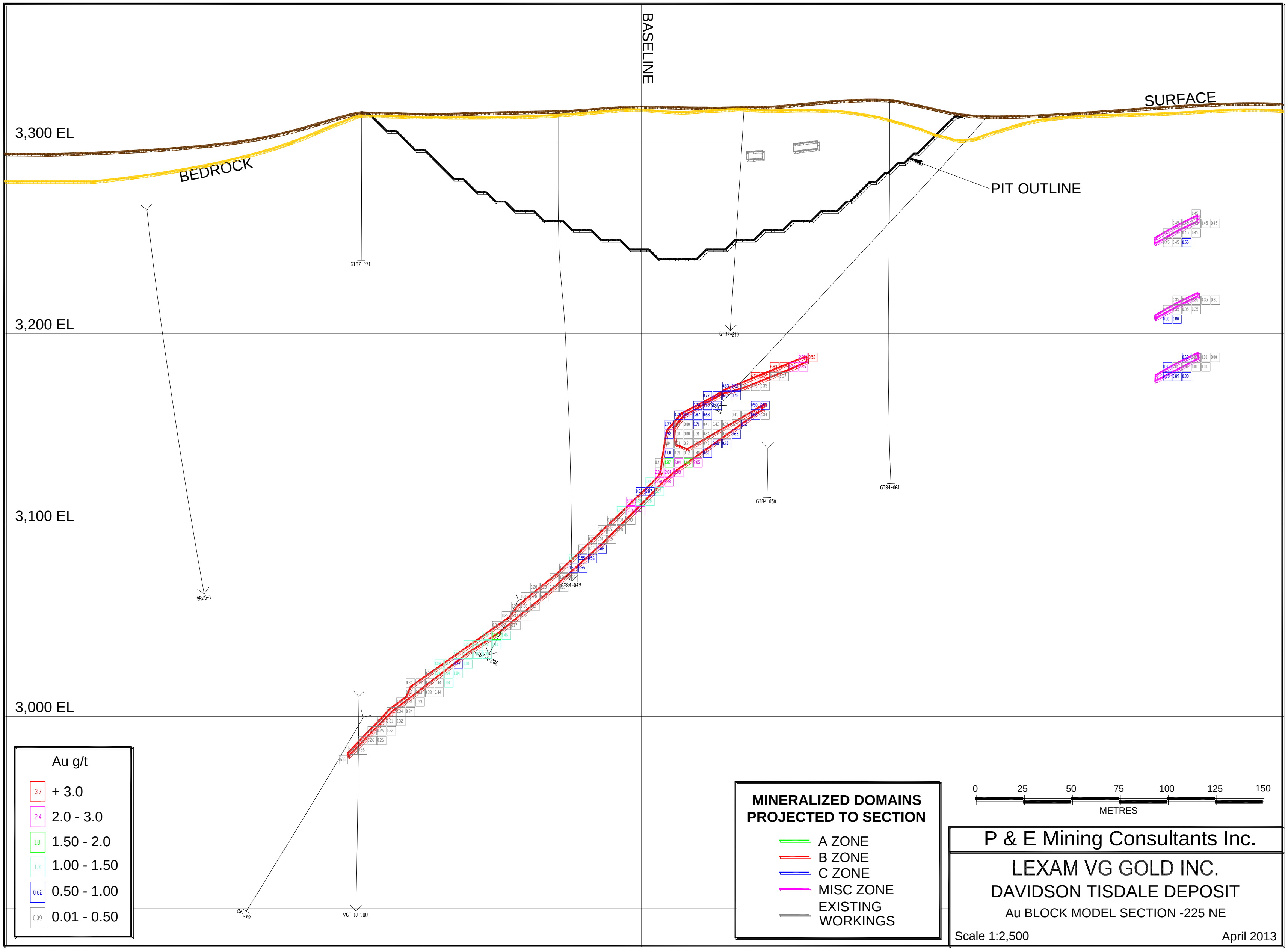
Davidson Tisdale B-Zone Down Dip Variogram

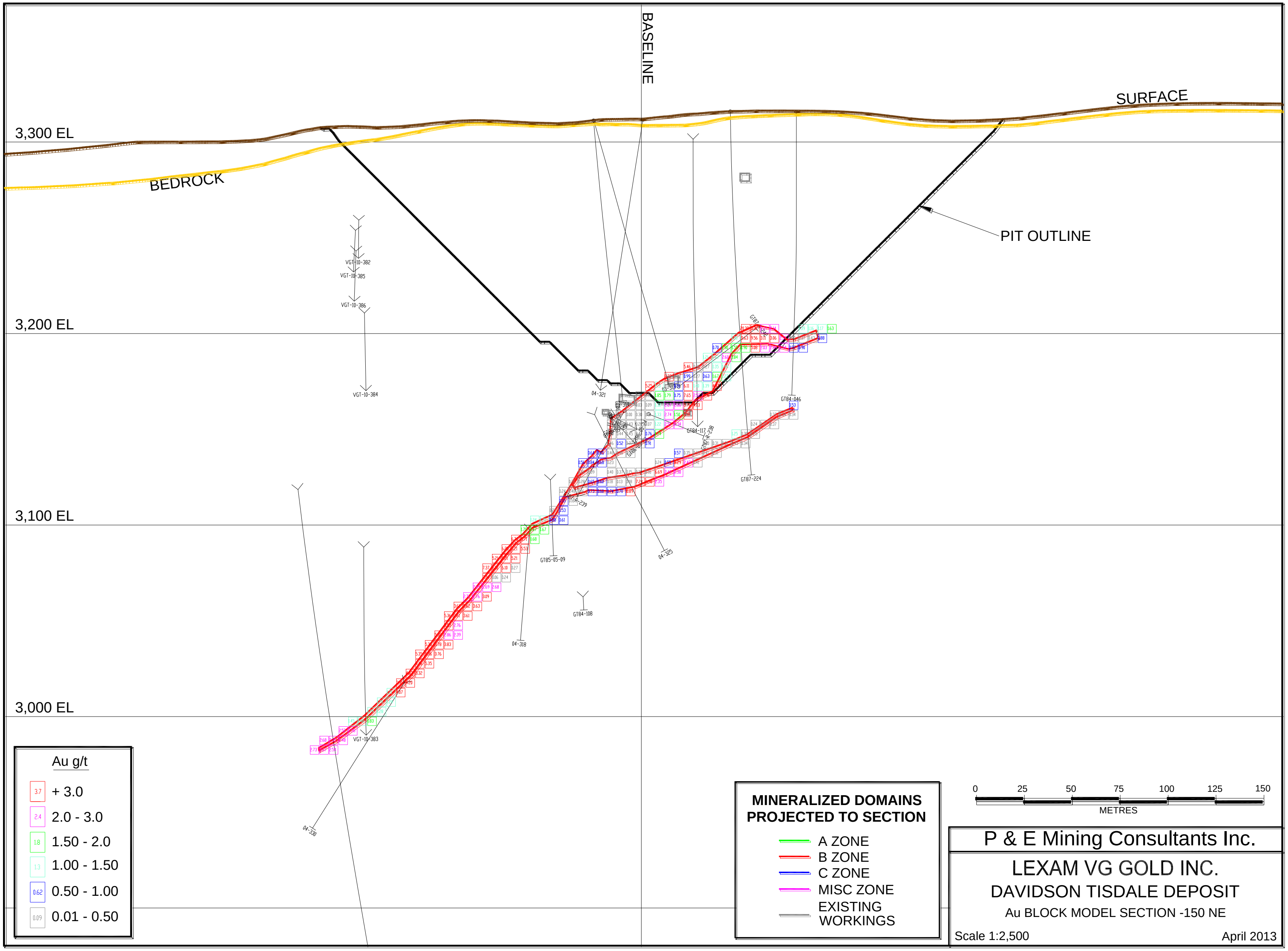


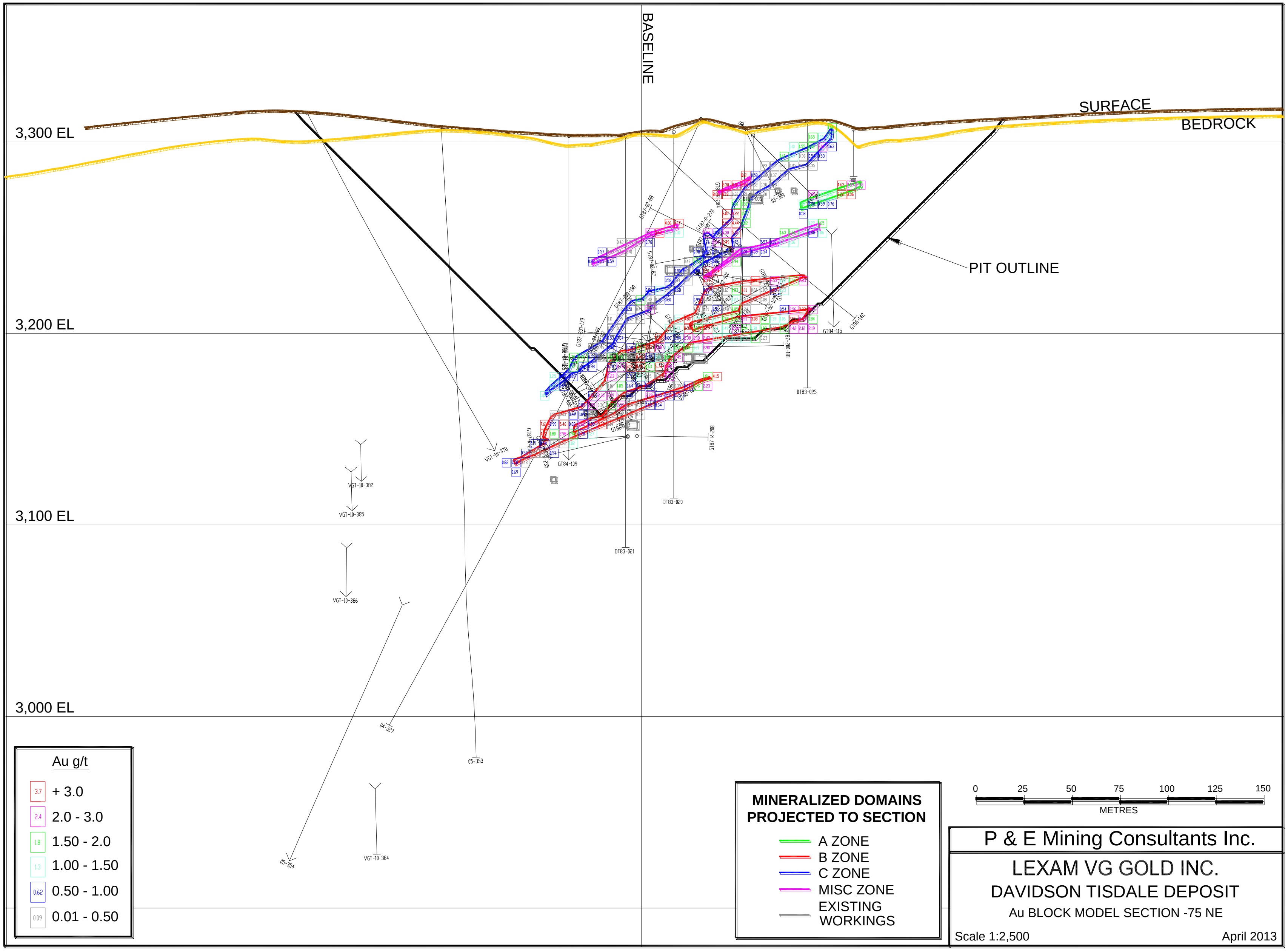
Davidson Tisdale B-Zone Across Dip Variogram

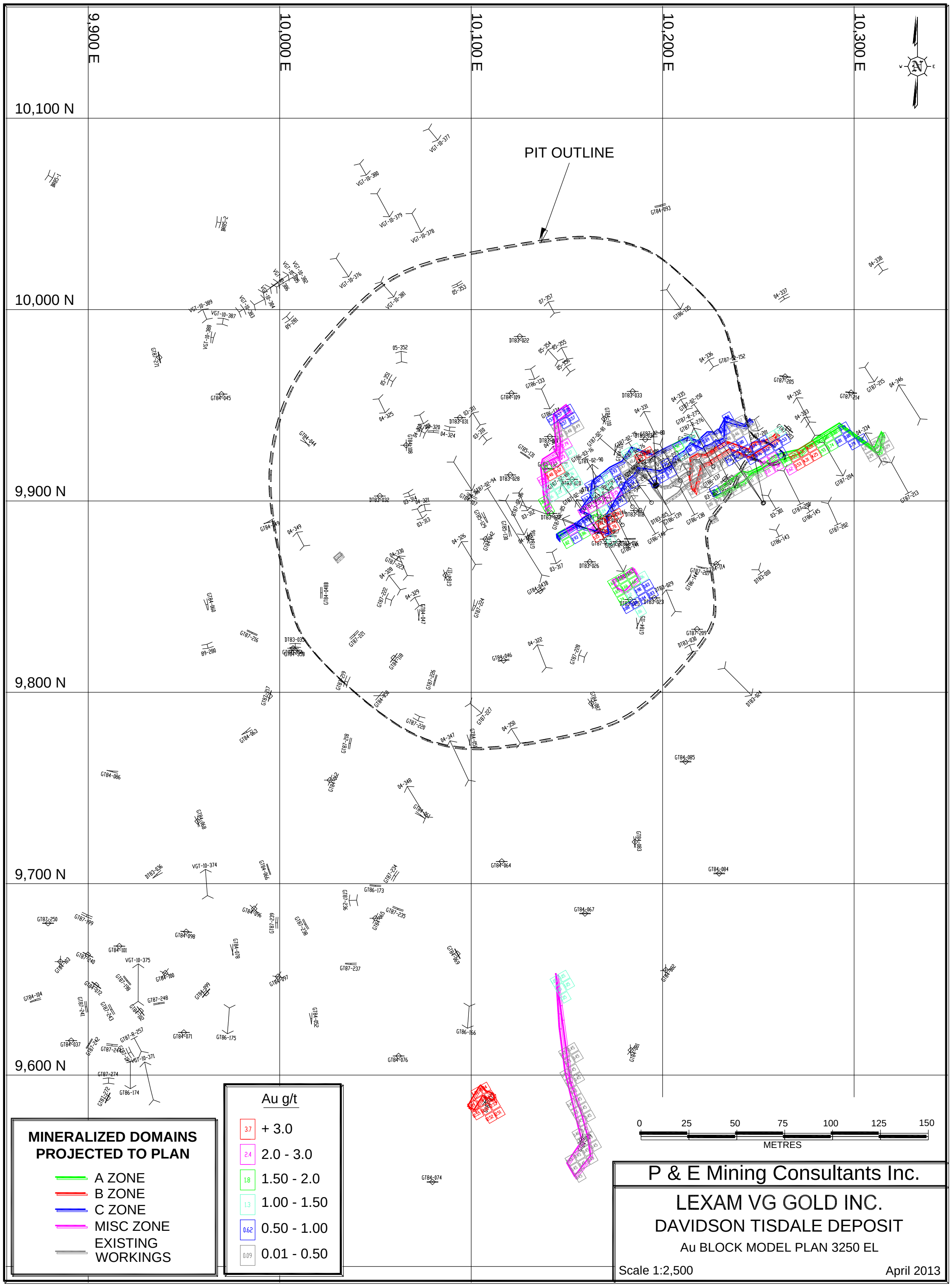


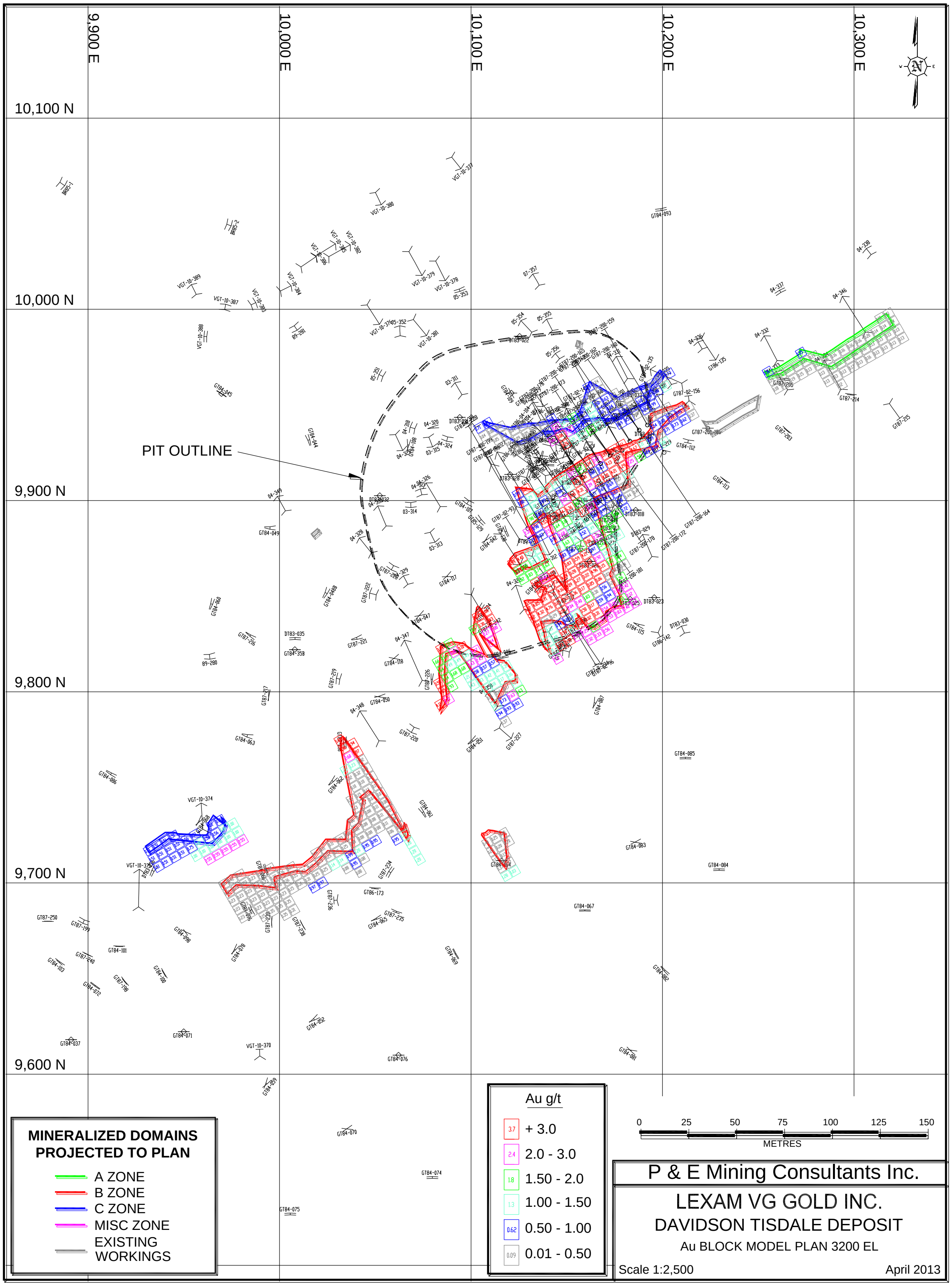
APPENDIX V. AU BLOCK MODEL CROSS SECTIONS AND PLANS

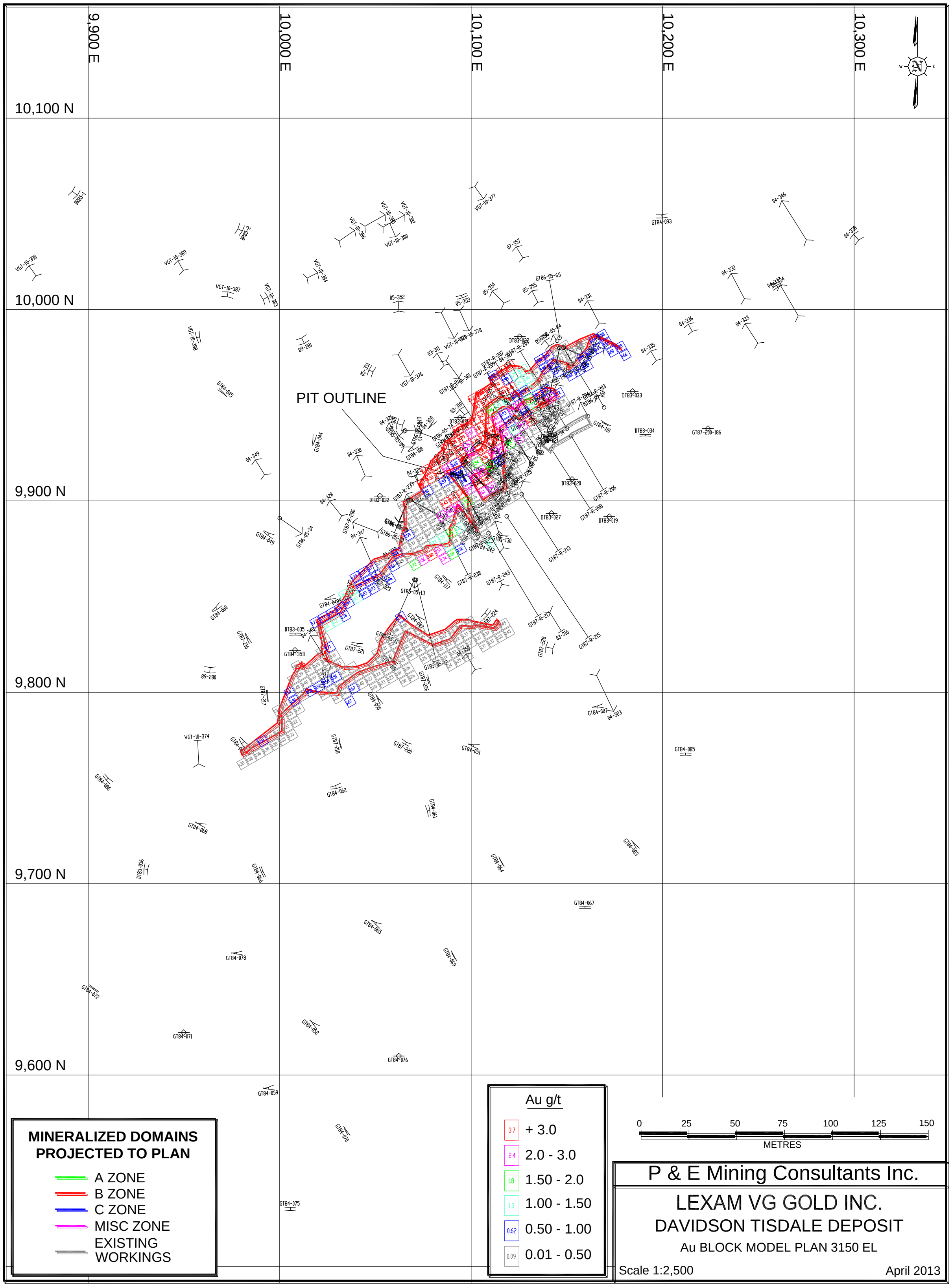

BASELINE
SURFACE
3,300 EL
BEDROCK
PIT OUTLINE
3,200 EL
3,100 EL
3,000 EL
Au g/t
+ 3.0
2.0 - 3.0
1.50 - 2.0
1.00 - 1.50
0.50 - 1.00
0.01 - 0.50
MINERALIZED DOMAINS PROJECTED TO SECTION
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
Au BLOCK MODEL SECTION -225 NE
Scale 1:2,500
April 2013


BASELINE
SURFACE
BEDROCK
PIT OUTLINE
3,300 EL
3,200 EL
3,100 EL
3,000 EL
Au g/t
+ 3.0
2.0 - 3.0
1.50 - 2.0
1.00 - 1.50
0.50 - 1.00
0.01 - 0.50
MINERALIZED DOMAINS PROJECTED TO SECTION
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
Au BLOCK MODEL SECTION -150 NE
Scale 1:2,500
April 2013

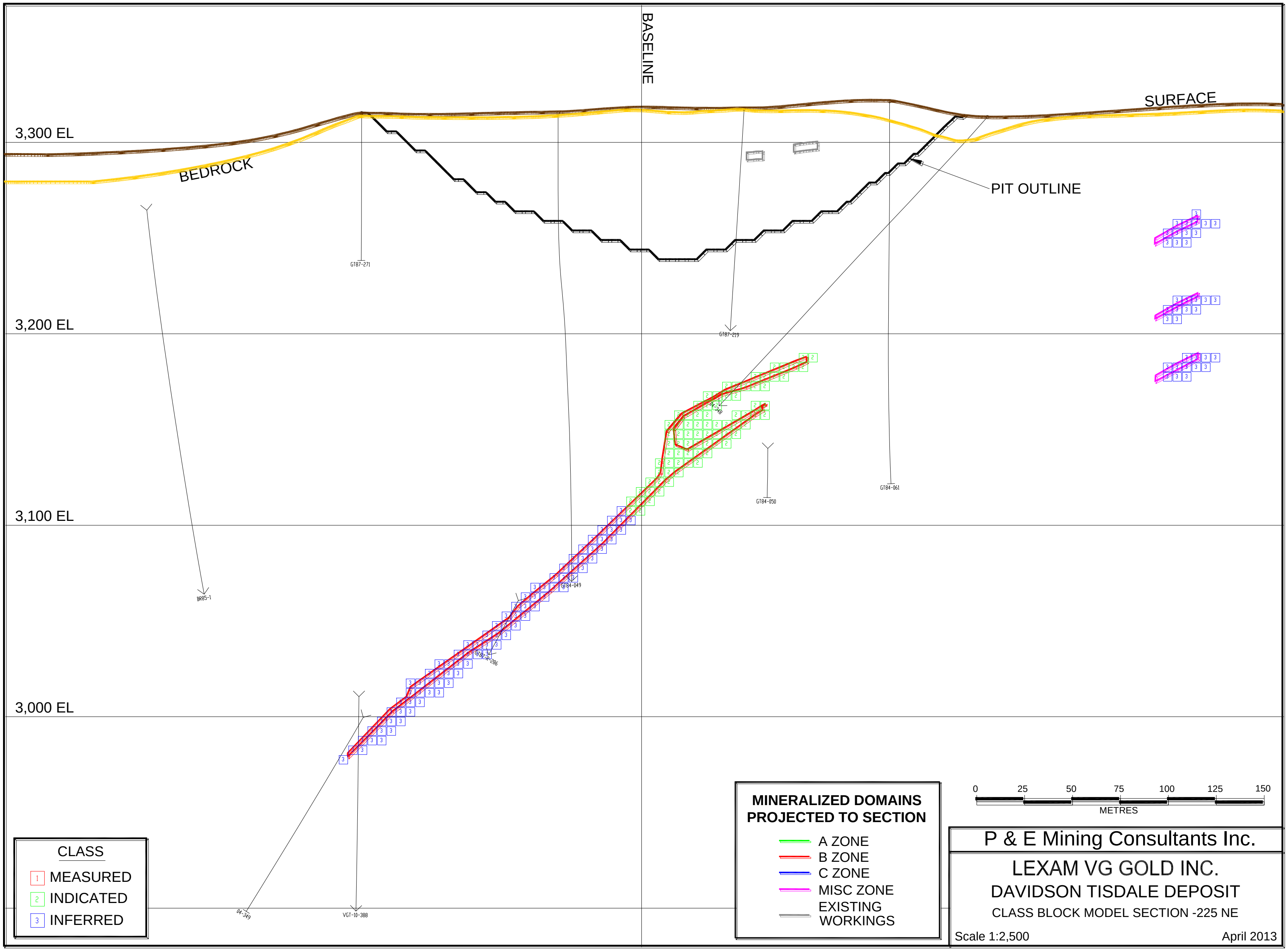

BASELINE
SURFACE
BEDROCK
PIT OUTLINE
3,300 EL
3,200 EL
3,100 EL
3,000 EL
Au g/t
+ 3.0
2.0 - 3.0
1.50 - 2.0
1.00 - 1.50
0.50 - 1.00
0.01 - 0.50
MINERALIZED DOMAINS PROJECTED TO SECTION
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
Au BLOCK MODEL SECTION -75 NE
Scale 1:2,500
April 2013


9,900 E
10,000 E
10,100 E
10,200 E
10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
9,600 N
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO PLAN
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
Au g/t
+ 3.0
2.0 - 3.0
1.50 - 2.0
1.00 - 1.50
0.50 - 1.00
0.01 - 0.50
0
25
50
75
100
125
150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
Au BLOCK MODEL PLAN 3250 EL
Scale 1:2,500
April 2013

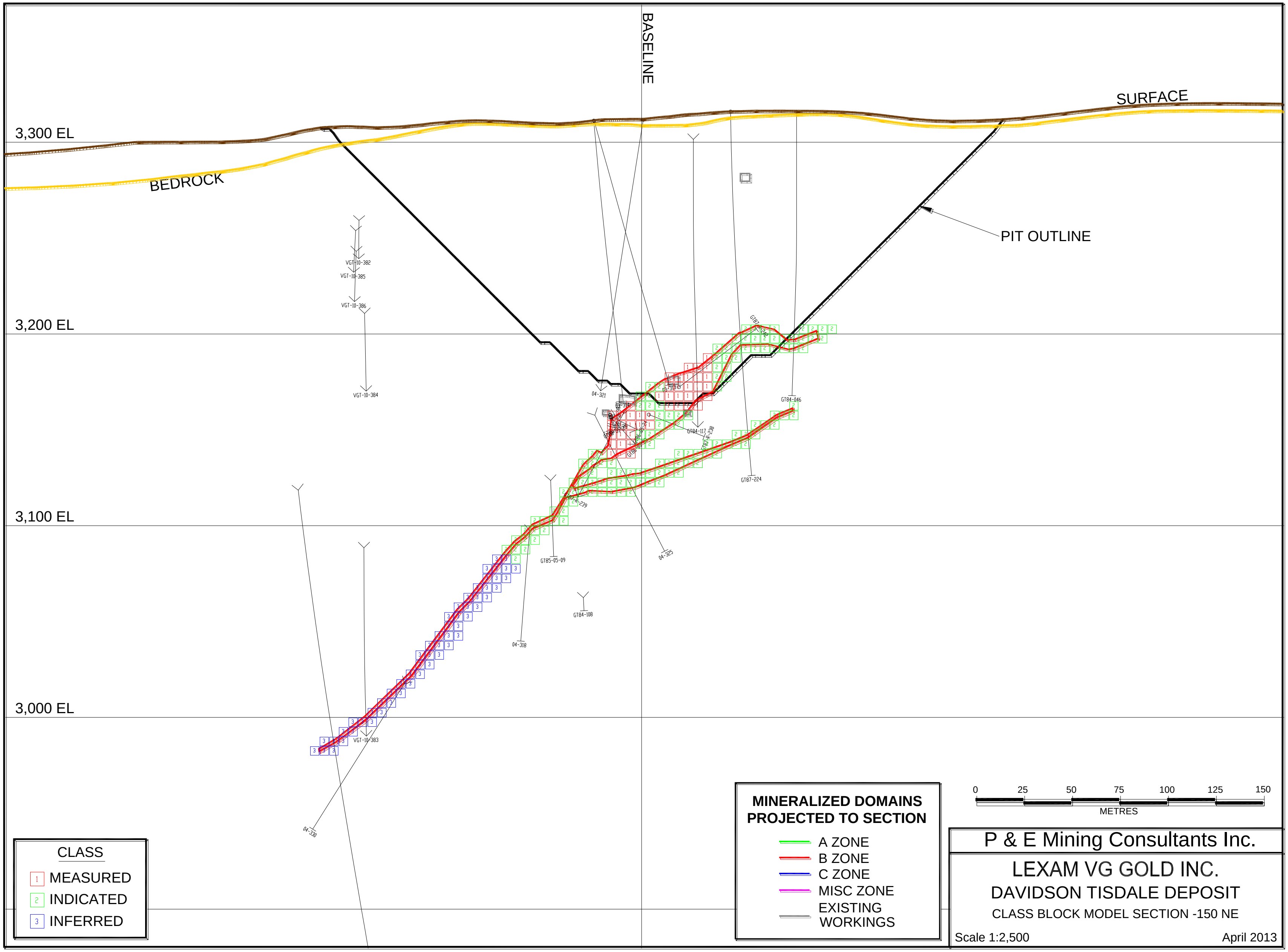

9,900 E
10,000 E
10,100 E
10,200 E
10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
9,600 N
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO PLAN
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
Au g/t
+ 3.0
2.0 - 3.0
1.50 - 2.0
1.00 - 1.50
0.50 - 1.00
0.01 - 0.50
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
Au BLOCK MODEL PLAN 3200 EL
Scale 1:2,500
April 2013


9,900 E
10,000 E
10,100 E
10,200 E
10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
9,600 N
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO PLAN
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
Au g/t
+ 3.0
2.0 - 3.0
1.50 - 2.0
1.00 - 1.50
0.50 - 1.00
0.01 - 0.50
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
Au BLOCK MODEL PLAN 3150 EL
Scale 1:2,500
April 2013

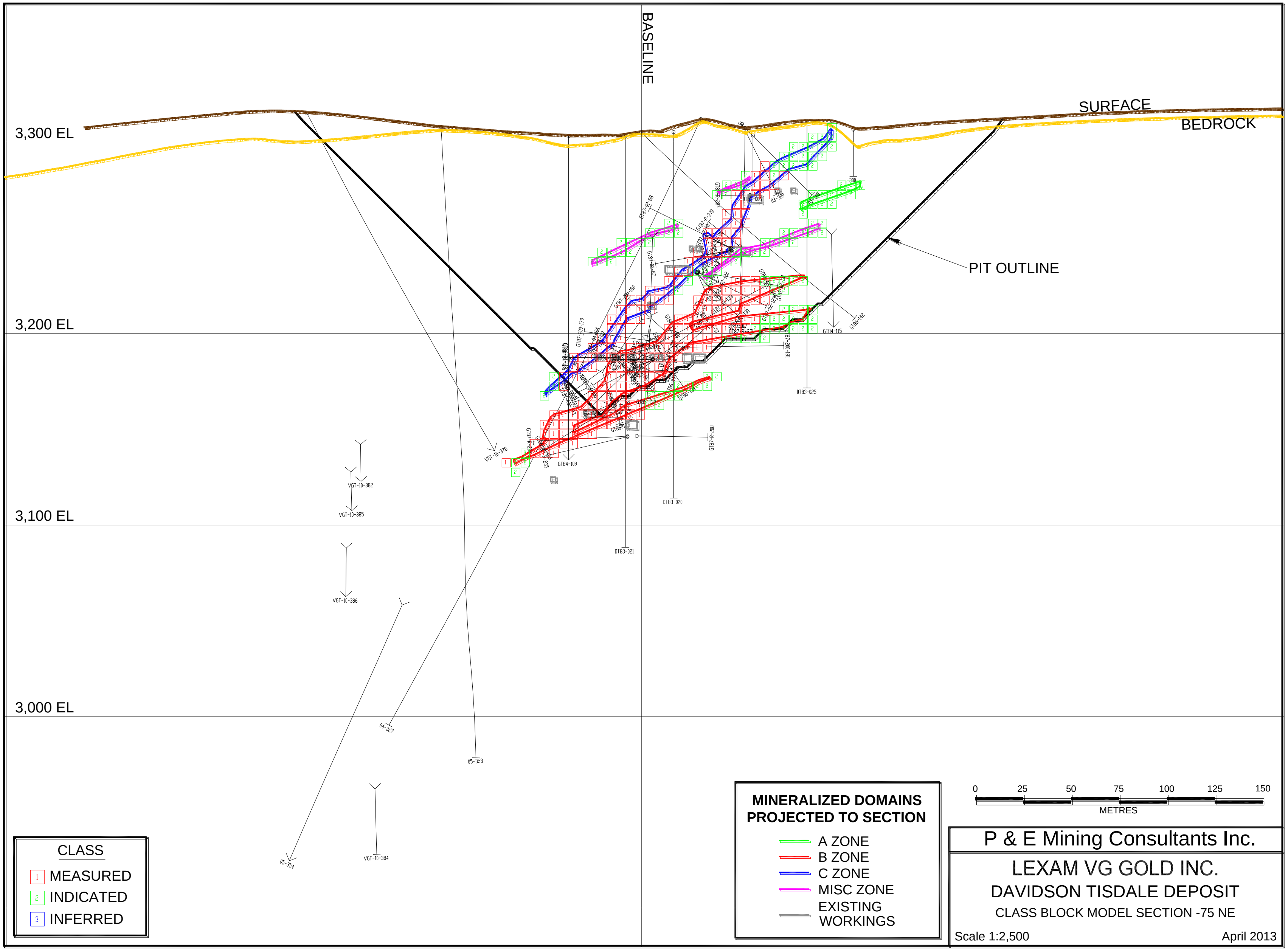
APPENDIX VI. CLASSIFICATION BLOCK MODEL CROSS SECTIONS AND PLANS

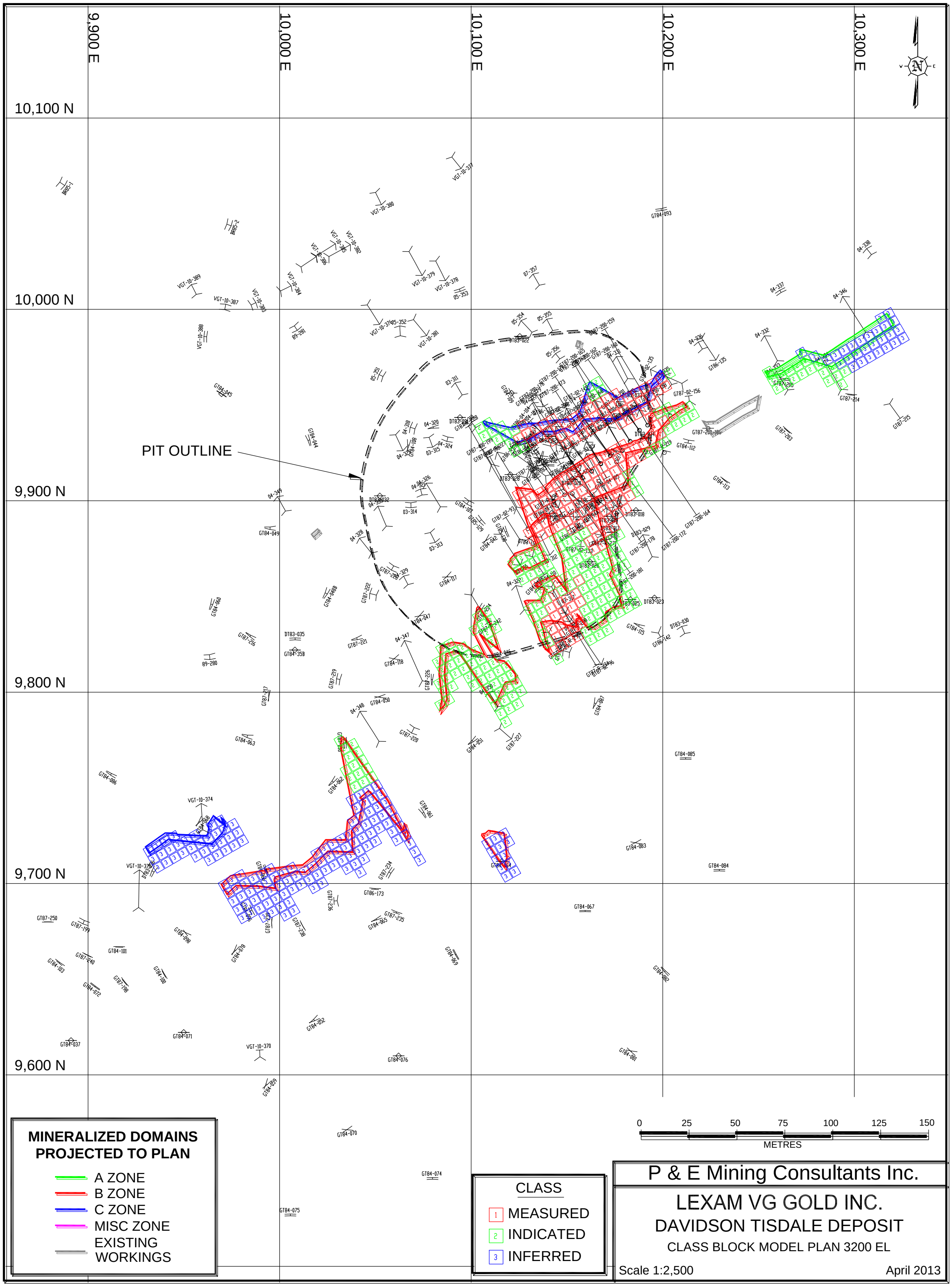


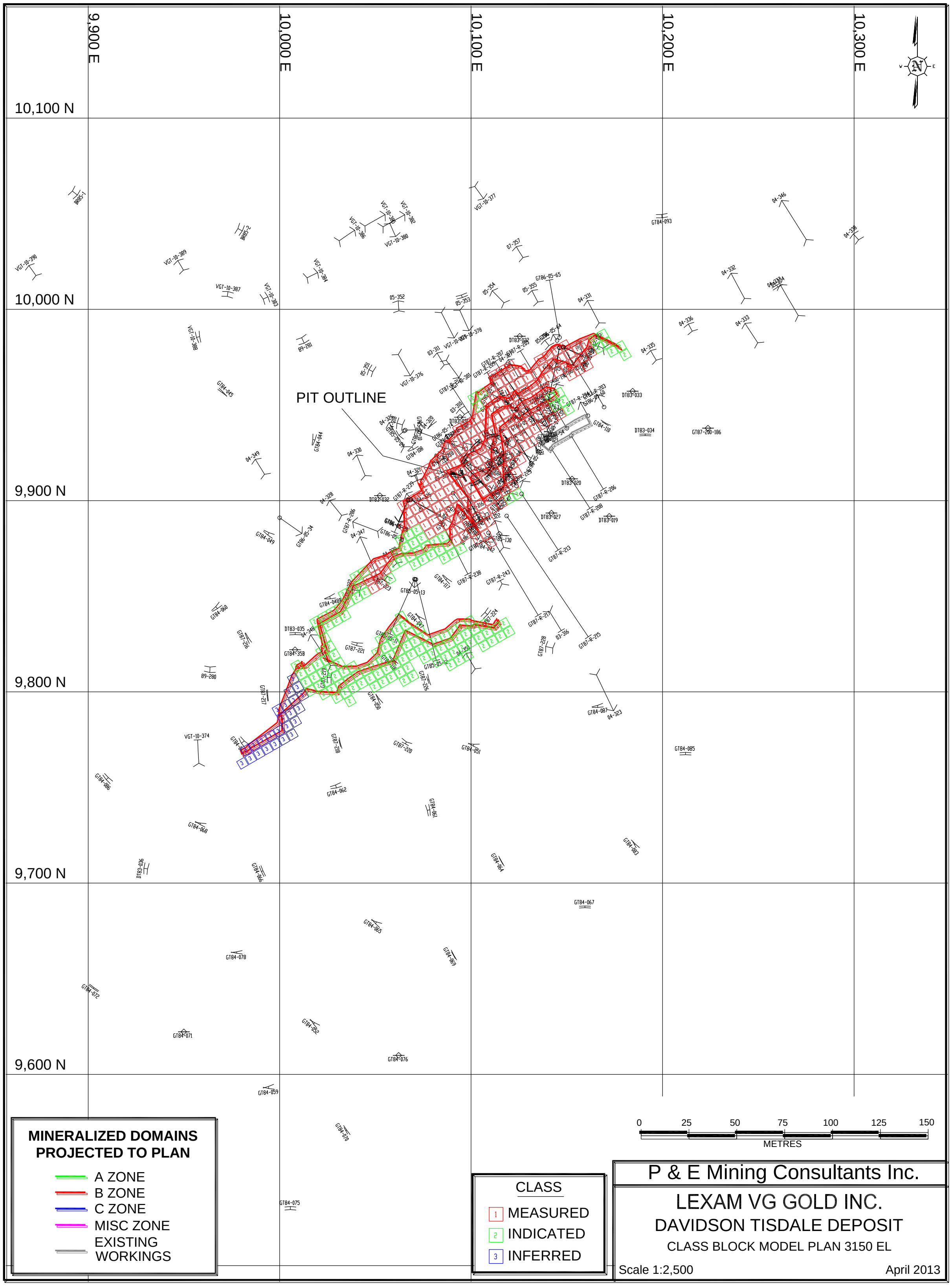
BASELINE
SURFACE
BEDROCK
PIT OUTLINE
3,300 EL
3,200 EL
3,100 EL
3,000 EL
CLASS
MEASURED
INDICATED
INFERRED
MINERALIZED DOMAINS PROJECTED TO SECTION
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
0
25
50
75
100
125
150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
CLASS BLOCK MODEL SECTION -225 NE
Scale 1:2,500
April 2013

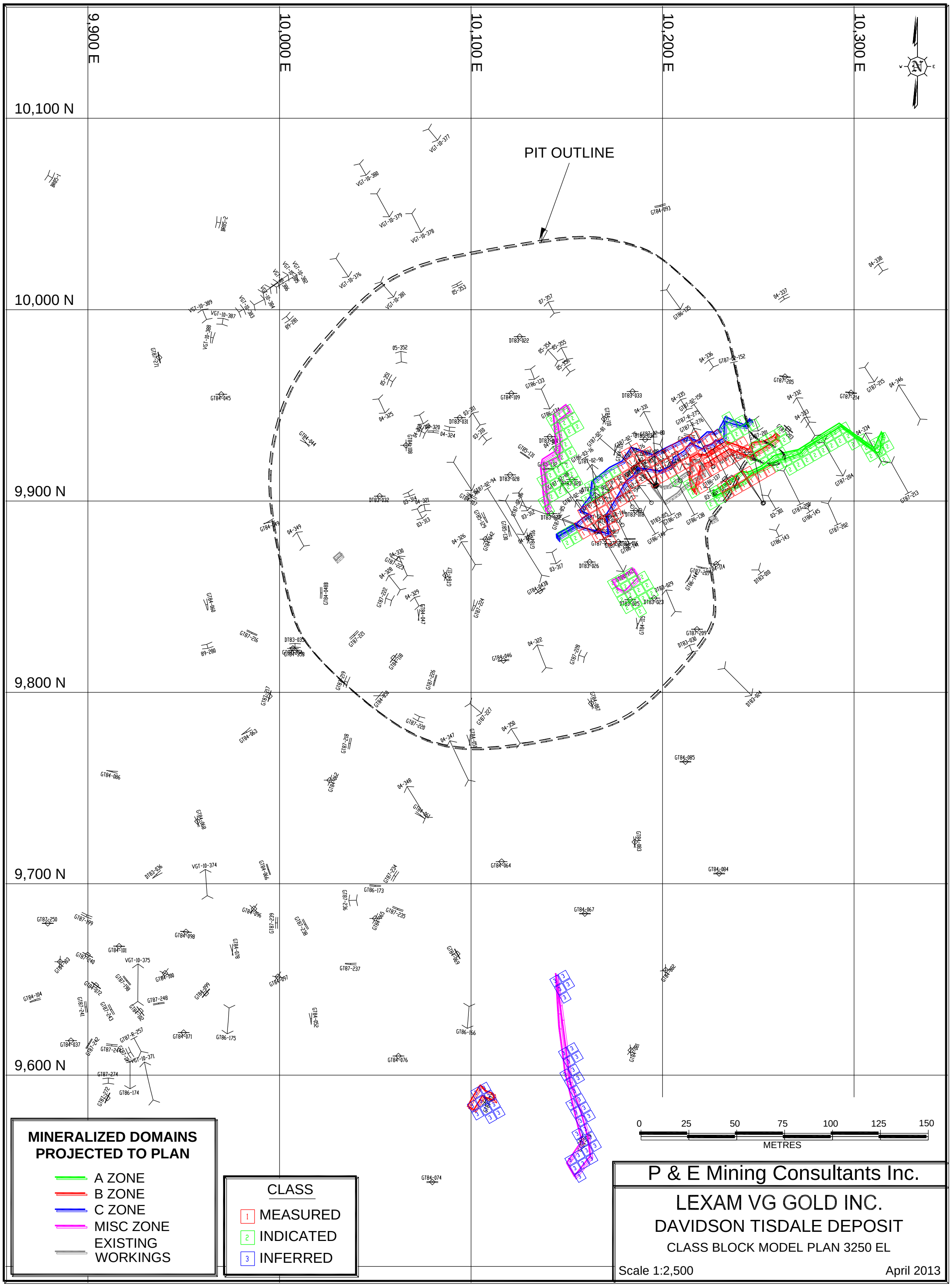


BASELINE
SURFACE
3,300 EL
BEDROCK
PIT OUTLINE
3,200 EL
3,100 EL
3,000 EL
CLASS
MEASURED
INDICATED
INFERRED
MINERALIZED DOMAINS PROJECTED TO SECTION
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
CLASS BLOCK MODEL SECTION -150 NE
Scale 1:2,500
April 2013


BASELINE
SURFACE
BEDROCK
3,300 EL
3,200 EL
3,100 EL
3,000 EL
PIT OUTLINE
CLASS
MEASURED
INDICATED
INFERRED
MINERALIZED DOMAINS
PROJECTED TO SECTION
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING
WORKINGS
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
CLASS BLOCK MODEL SECTION -75 NE
Scale 1:2,500
April 2013


9,900 E
10,000 E
10,100 E
10,200 E
10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
9,600 N
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO PLAN
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
CLASS
MEASURED
INDICATED
INFERRED
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
CLASS BLOCK MODEL PLAN 3200 EL
Scale 1:2,500
April 2013

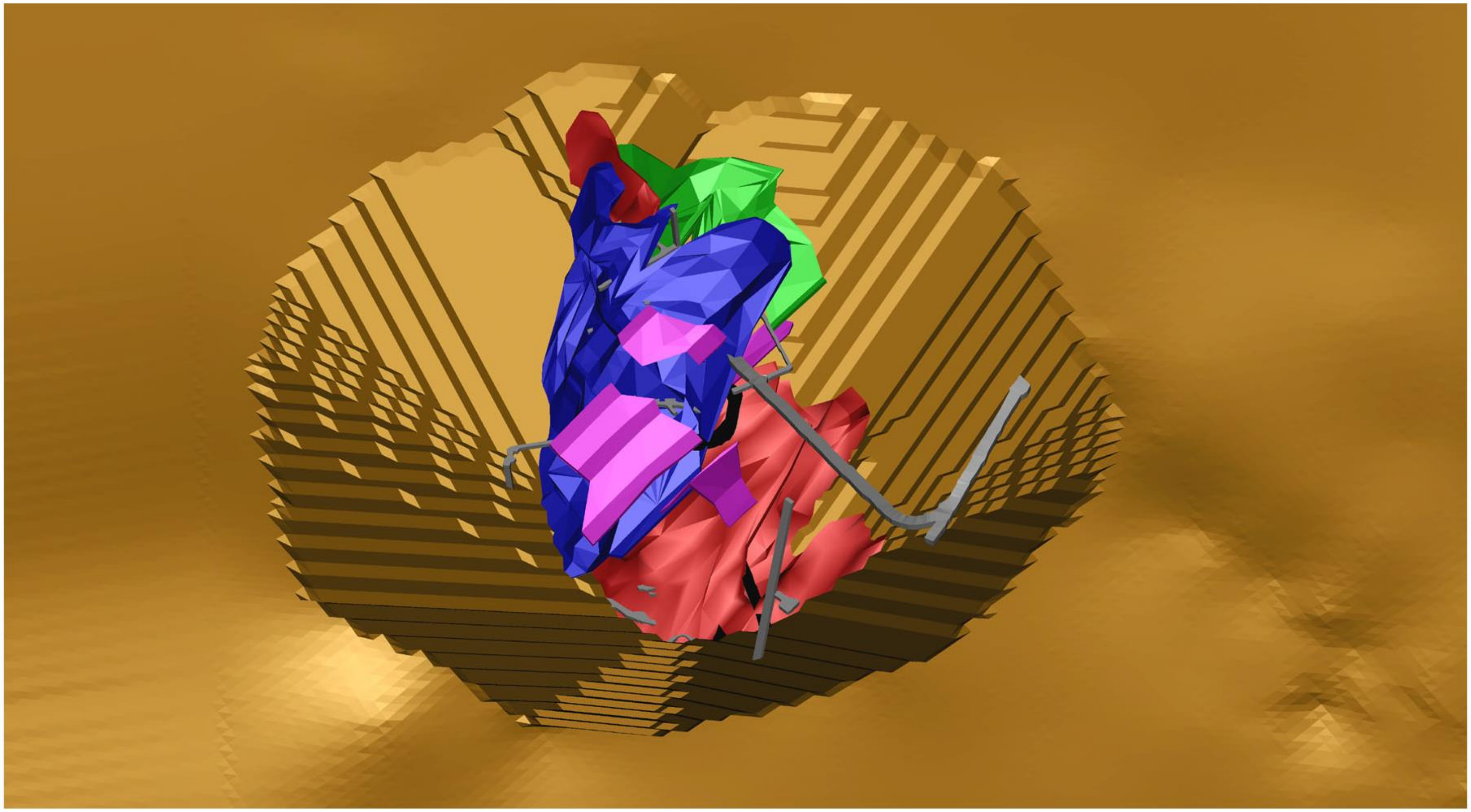

9,900 E
10,000 E
10,100 E
10,200 E
10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
9,600 N
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO PLAN
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
CLASS
MEASURED
INDICATED
INFERRED
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
CLASS BLOCK MODEL PLAN 3150 EL
Scale 1:2,500
April 2013



9,900 E
10,000 E
10,100 E
10,200 E
10,300 E
10,100 N
10,000 N
9,900 N
9,800 N
9,700 N
9,600 N
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO PLAN
A ZONE
B ZONE
C ZONE
MISC ZONE
EXISTING WORKINGS
CLASS
MEASURED
INDICATED
INFERRED
0 25 50 75 100 125 150
METRES
P & E Mining Consultants Inc.
LEXAM VG GOLD INC.
DAVIDSON TISDALE DEPOSIT
CLASS BLOCK MODEL PLAN 3250 EL
Scale 1:2,500
April 2013

APPENDIX VII. OPTIMIZED PIT SHELL

DAVIDSON TISDALE DEPOSIT - OPTIMIZED PIT SHELL



DOMAINS

A ZONE	MISC ZONE
B ZONE	EXISTING WORKINGS
C ZONE	