NOTICE TO READER

The attached Technical Report is being re-filed due to amendments made to Section 15. The amendments include a summary discussion and tabulation of the current mineral reserves at North American Palladium Ltd.'s Lac des Iles mine. No new reserves are estimated in the current report. The addition of a summary discussion of the current reserves for the property is included to provide a direct reference for other sections in the report in which mineral reserves are considered. No other changes have been made to the original report.

Amended and Restated
NI 43-101 Technical Report for Lac des Iles Mine, Ontario Incorporating
A Preliminary Economic Assessment of
the Mine Expansion Plan

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Effective date of the report: February 18, 2015
Signing date of the report: April 20, 2015

HATCH

North American Palladium Ltd.

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for North American Palladium Ltd. for the purpose of allowing North American Palladium Ltd. to reach informed decisions respecting the development of the Lac des Iles Mine. This report is to be used by North American Palladium Ltd. subject to the terms and conditions of its agreements with its service providers. The quality of information, conclusions and estimates contained herein are consistent with the level of effort involved in the authors' services based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions and qualifications set forth in this report. This report is intended to be read as a whole, and sections should not be read or relied upon out of context.

This report was authored by the qualified persons (each a "QP" and collectively, the "QPs") listed in Table 2-1. Each QP only assumes responsibility for those sections or areas of the report that are referenced opposite their name in Table 2-1. None of such QPs, however, accept any responsibility or liability for the sections or areas of this report that were prepared by other QPs.

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Sections of the report contain estimates, projections and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP's opinion at the time that they are made but in most cases involve significant risk and uncertainty. Although the responsible QP has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated or projected. There can be no assurance that forward looking information in this section of the report will prove to be accurate in such statements or information. Accordingly, readers should not place undue reliance on forward-looking information.

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Appendices B to I are available from NAP

Glossary

Units of Measure

above mean sea level ....amsl
acre ....ac
ampere ....A
annum (year) ....a
billion ....B
billion tonnes ....Bt
billion years ago ....a
British thermal unit ....BTU
centimetre ....cm
cubic centimetre cm3
cubic feet per minute ....cfm
cubic feet per second ft3/s
cubic foot ft3
cubic inch in3
cubic metre m3
cubic yard yd3
Coefficients of Variation ....CVs
day ....d
days per week ....d/wk
days per year (annum) ....d/a
dead weight tonnes ....DWT
decibel adjusted ....dBa
decibel ....dB
degree ....°
degrees Celsius ....°C
diameter ....ø
dollar (American) ....US$
dollar (Canadian) ....Cdn$
dry metric ton ....dmt
foot ....ft
gallon ....gal
gallons per minute (US) ....gpm
Gigajoule ....GJ
gigapascal ....GPa
gigawatt ....GW
gram ....g
grams per litre ....g/L
grams per tonne ....g/t
greater than ....>
hectare (10,000 m2) ....ha
hertz ....Hz
horsepower ....hp
hour ....h
hours per day ....h/d
hours per week..h/wk
hours per year ....h/a
inch ....in
kilo (thousand) ....k

kilogram ....kg
kilograms per cubic metre kg/m3
kilograms per hour ....kg/h
kilograms per square metre kg/m2
kilometre...km
kilometres per hour ....km/h
kilopascal ....kPa
kilotonne ....kt
kilovolt ....kV
kilovolt-ampere...kVA
kilovolts ....kV
kilowatt ....kW
kilowatt hour ....kWh
kilowatt hours per tonne ....kWh/t
kilowatt hours per year ....kWh/a
less than .... <
litre L
litres per minute ....L/m
megabytes per second ....Mb/s
megapascal ....MPa
megavolt-ampere ....MVA
megawatt ....MW
metre ....m
metres above sea level ....masl
metres Baltic sea level ....mbsl
metres per minute ....m/min
metres per second ....m/s
microns ....µm
milligram ....mg
milligrams per litre ....mg/L
millilitre ....mL
millimetre..mm
million ....M
million bank cubic metres Mbm3
million bank cubic metres per annum Mbm3/a
million tonnes ....Mt
minute (plane angle) ....'
minute (time) ....min
month ....mo
ounce ....oz
pascal ....Pa
centipoise ....mPa·s
parts per million ....ppm
parts per billion ....ppb
percent ....%
pound(s) ....lb
pounds per square inch ....psi
revolutions per minute ....rpm
second (plane angle) ...."
second (time) ....s
short ton (2,000 lb) ....st
short tons per day ....st/d

short tons per year ....st/y
specific gravity ....SG
square centimetre cm2
square foot ft2
square inch in2
square kilometre km2
square metre m2
three-dimensional 3D
tonne (1,000 kg) (metric ton) ....t
tonnes per day ....t/d
tonnes per hour ....t/h
tonnes per year ....t/a
tonnes seconds per hour metre cubed ts/hm3
volt V
week ....wk
weight/weight ....w/w
wet metric ton ....wmt

Abbreviations and Acronyms

A.R. MacPherson Consultants Ltd. ....ARMC
Accurassay Laboratories ....Accurassay
Activation Laboratories Ltd. ....ActLabs
alternating current ....AC
ammonium nitrate fuel oil ....ANFO
autogenous/ball mill/crushing ....ABC
Bond ball mill work index ....BWi
by inductively coupled plasma ....ICP
Canadian Institute of Mining, Metallurgy and Petroleum ....CIM
CDN Resources Laboratories ....CDN
Certificate of Approval ....CofA
close-circuit fully-autogenous grinding milling ....FAC
Commonwealth Scientific and Industrial Research Organisation ....CSIRO
Communities of Interest ....COI
counterweight ....CWT
Electron Probe Micro Analysis ....EPMA
equigranular gabbro ....EGAB
equivalent grinding length ....EGL
exchange traded funds ....ETF
Footwall ....FW
fully-autgenous grinding milling, ball milling and pebble crushing ....FABC
fully-autogenous grinding milling and ball milling ....FAB
fully-autogenous grinding milling, pebble milling and pebble crushing ....FAPC
fully-autogenous grinding ....FAG
fully-autogenous gringing milling and pebble milling ....FAP
Fusion Data Management ....Fusion

Gabbronorite .... GN
Gemcom GEMS™ .... GEMS™
Gemcom Whittle™ .... Whittle™
global positioning system .... GPS
ground support index .... GSI
Hangingwall .... HW
Heterolithic Gabbro Breccia .... HGABBX
high-density polyethylene .... HDPE
Imdex Limited .... Imdex
internal rate of return .... IRR
International Electrotechnical Commission .... IEC
International Organization for Standardization .... ISO
in-the-hole .... ITH
inverse distance squared ID2
Itasca Consulting Canada Inc. ICCI
John Patrick Sheridan .... Sheridan
Knelson Research and Technology Centre .... Knelson
laboratory information management system .... LIMS
Lac des Iles Intrusive Complex........ LDI-IC
Lac des Iles Mines Ltd. .... LDIM
Lac des Iles Property .... the Property
Lerchs-Grossman .... LG
life-of-mine .... LOM
load-haul-dump .... LHD
Local Citizens Committee LCC
locked-cycle test .... LCT
loss-on-ignition .... LOI
methyl isobutyl carbinol .... MIBC
Mine Block Intrusion .... MBI
Mine Design Engineering .... MDE
Ministry of Northern Development and Mines .... MNDM
motor control centre .... MCC
National Instrument 43-101 .... NI 43-101
nearest neighbour .... NN
net present value .... NPV
net smelter royalty .... NSR
New York Stock Exchange .... NYSE
North American Palladium Ltd. .... NAP
North Lac des Iles Intrusion .... NLDI
optical emission spectrometry .... OES
ordinary kriging .... OK
palladium equivalent .... PdEq
platinum group element .... PGE

platinum group metal ....PGM
potassium amyl xanthate ....PAX
preliminary economic assessment ....PEA
present value ....PV
programmable logical controller ....PLC
pyroxenite unit ....PYXT
Qualified Persons ....QPs
Qualitative Evaluation of Materials by Scanning Electron Microscope ....QEMSCAN™
quality assurance ....QA
quality control ....QC
Real Time Kinematic...RTK
regular grade ore ....RGO
rock quality designation ....RQD
run-of-mine ....ROM
semi-autogenous grinding milling, pebble milling and pebble crushing ....SAPC
semi-autogenous ....SAG
semi-autogenous/ball mill/crushing ....SABC
SGS Lakefield Research Ltd. ....SGS
Sheridan Platinum Group Ltd. ....SPG
Society for Mining, Metallurgy, and Exploration ....SME
South Lac des Iles Intrusion ....SLDI
Standards Council of Canada ....SCC
tailings management facility ....TMF
Toronto Stock Exchange ....TSX
unconfined compressive strength ....UCS
Unconsolidated rock backfill ....UCF
US Securities and Exchange Commission ....SEC
Vale Canada Limited ....Vale
varitextured gabbro ....VGAB
varitextured ....VT
versatile time-domain electromagnetic ....VTEM
volt ampere reactive ....VAR
volt direct current ....VDC
x-ray fluorescence ....XRF
Xstrata Process Support ....XPS

1. Summary

1.1 Introduction

The Lac des Iles (LDI) Mine is owned and operated by Lac des Iles Mines Ltd. (LDIM), a wholly owned subsidiary of North American Palladium Ltd. (NAP). The LDI mine site is located 90 km northwest of Thunder Bay, Ontario, and is accessed by provincial highway 527 (Figure 1-1). The LDI Mine has been in production since 1993, beginning with open pit mining of the Roby Zone and adding underground production in 2006 via ramp access to the Roby High Grade Zone.

Figure 1-1: Property Location Map

Production was continuous from 1993 through to late 2008, when declining metal prices forced NAP to place LDI on care and maintenance. Exploration drilling continued during the shutdown and delineated the first significant underground resource for the Offset Zone. The Offset Zone is considered an extension of the Roby Zone; however, it has been displaced by a major fault (the Offset Fault). In 2010, production resumed at LDI and the exploration drilling campaign continued. Also in 2010, a major expansion to the underground mine began with a shaft being sunk to access the Offset Zone which was deemed to be too deep to be economically mined using the original truck haulage ramp access. The shaft was completed to the 825 m level and the ore hoisting system commissioned at the end of 2013. The LDI site also consists of an operating concentrator with a nameplate capacity of 15,000 tonnes per day (tpd) and an active tailings management facility that is in the process of being re-permitted for expansion.

In 2014, LDI processed 2.6 million tonnes (Mt) of ore at a head grade of 2.7 g/t of palladium which came from a mix of underground ore and low grade surface stockpiles. A similar production profile is planned by NAP for 2015.

A technical study, titled “Technical Report for Lac des Iles Mine, Ontario Incorporating Prefeasibility Study of Life of Mine Plan” (2014 Technical Report) and dated March 21, 2014, confirmed mineral reserves down to the 1065 metre level (1065L) with additional mineral resources extending to 1,600 m depth. The mineral reserves calculated in the 2014 Technical Report totalled 15.0 Mt grading 2.77 g/t Pd and showed a mine life to 2019.

In February of 2015, NAP updated its mineral reserve and mineral resource estimates. The mineral reserve estimate was based on the 2014 Technical Report, updated to reflect material mined through December 31, 2014 and the conversion of additional Regular Grade Ore (RGO) stockpile to reserves. These mineral reserves, augmented with other mineral resources above the 1065L, are the basis for the LDI mine production plan (Current Mine Plan). As an operating facility considering an expansion, it is necessary to include the current production plan (and associated reserves) in the evaluation to properly assess the potential economics of any expansion, as this represents the alternative to developing the new resources (i.e. the opportunity cost of the investment decision).

Of significant interest in the 2014 Technical Report was the large mineral resource inventory for the LDI property, which includes material at and near surface, an expanded Offset hangingwall (HW) zone and a mineral resource estimate of the Offset and Roby footwall (FW) zones. In addition, a major exploration program was conducted throughout 2014 which continued to demonstrate the potential for more resources.

In the 4th quarter of 2014, LDI operated the concentrator full time versus the previous 2 weeks per month campaign basis, supplementing the higher grade underground ore with the lower grade stockpile from surface. This resulted in lower unit costs, improved cash flows and demonstrated the economic viability of processing material as low as 1.0 g/t Pd.

The price for palladium and the US/CAN currency exchange rate both moved significantly in a favourable direction in 2014 with long-term analyst consensus that they would stay favourable.

Given the large resource base, the positive results from the 2014 mineral exploration program and the improved economic factors as noted above, NAP commissioned a new technical study to take a step back and re-scope the economic potential of the mineral resource at LDI. Hatch was retained to evaluate this expansion potential. Since much of the material would only be drilled to an inferred resource category, this study was undertaken at a Preliminary Economic Assessment (PEA) level.

The results of the study are disclosed in this Technical Report that incorporates a Preliminary Economic Assessment (PEA) of the mine expansion plan, which was prepared in accordance with National Instrument 43-101.

This PEA is at a scoping level and is preliminary in nature. One of the expansion scenarios includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. There are no Inferred mineral resources contained within the Current Mine Plan.

Key differences from the 2014 Technical Report include:

- The inclusion of the results of the 2014 exploration program in this PEA, which was primarily focused on targets below the 1065L.
- The evaluation of an open pit expansion that does not include any Inferred mineral resources.
- The impact of operating the mill on a continuous full-time basis versus the previous batch process.
- The inclusion of updated economic parameters, including changes to foreign exchange rates and metal prices.
- The application of a lower cut-off grade as a result of improved metal prices and reduced operating costs resulting from operating the LDI mill on a full time basis and other realized cost improvements.
- The inclusion of some previously uneconomic footwall material into the mine plan.
- The evaluation of a potential shaft extension to mine mineral resources below the current limit of the Phase 1 mine plan (that contain Inferred mineral resources).

The PEA has no impact on the existing mineral reserves and the mineral reserve estimates are still current and valid in light of the PEA.

The effective date of this report is February 18, 2015. The effective dates for the mineral reserve and mineral resource estimates are December 31, 2014 and February 2, 2015, respectively.

For the purposes of this report, units of measurement are in metric and all currencies are expressed in Canadian dollars, unless otherwise specified.

1.2 Geology, Mineralization and Exploration

A majority of the past and current resources are contained in the Roby and Offset zone deposits. The resources contained in the Roby and Offset zone deposits have been separated into discrete HW and FW zones. The Roby Zone deposit extends to a minimum depth of 650 m below surface. It includes a thick FW zone in the west (tens to hundreds of metres wide) and a thinner, approximately 5 to 20 m thick HW zone in the east. The FW zone consists of vari-textured gabbro and local heterolithic gabbro breccia having typical palladium grades of approximately 1 to 3 g/t. The HW zone has typical palladium grades of 3 to 10 g/t and is hosted by a sheared, melanocratic norite ("pyroxenite"). The Offset Zone deposit shows similar grade distributions to the Roby Zone. The former is believed to represent the along strike continuation of the Roby Zone deposit but having been displaced from the latter along the east-striking and north dipping Offset Fault. The Offset Zone deposit remains open toward surface, to the southeast and at depth (greater than 1,500 m below surface). The Offset Zone is host to a majority of the higher-grade palladium resources on the Property (refer to Section 14) and is the focus for current underground mine production, development and exploration drilling. Exploration in 2014 confirmed the continuation of the thickest part of the Offset Zone below the current limit of the Phase 1 mine plan (1065 m) to a minimum depth of 1450 m, providing support for a potential mine expansion at depth. Additional drilling is required to increase the confidence of the mineral resources below the Phase 1 depth limit and to determine the full extent of potentially mineable palladium mineralization in the lower part of the Offset Zone deposit. Drilling completed in 2014 also delineated an inferred resource that extends along an easterly trend from the upper part of the Offset Zone and is referred to as the Upper Offset southeast extension.

Figure 1-2 consists of a typical section that illustrates the principal mineralized zones at LDI.

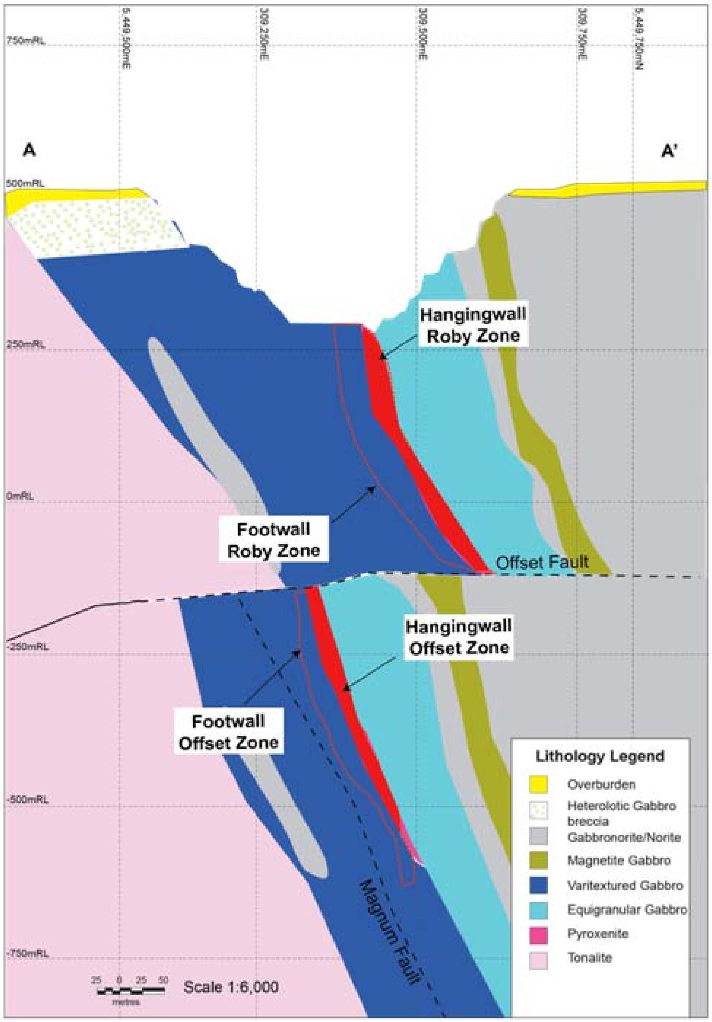


Figure 1-2: LDI Mineralization – Cross Section, Looking North

Several other mineralized zones occur on the Property including the North Varitextured Rim (North VT Rim) Zone and the Sheriff Zone. The North VT Rim Zone is a 2 km long, east- to northeast-striking mineralized zone consisting of sheared and altered varitextured gabbro and subordinate norite. The Sheriff Zone is a combination of the former Southeast Roby, Twilight and South Pit zones connected through additional drilling. Surface drilling in 2013 and 2014 delineated a high-grade resource that partly overlaps with the Sheriff Zone that is referred to as the Powerline Zone.

1.3 Mineral Resource Estimate

The updated mineral resource estimate has been prepared by NAP and is shown in Table 1-1 and in Section 14. It includes results from exploration drilling completed in 2014 and an update of the resource block model and mineral resource estimate for the Offset Zone. The major changes to the mineral resource estimate for the Property are summarized below:

- Updated Offset Zone resource estimate including expanded Lower Offset Zone Indicated resources in support of a potential mine expansion at depth.
- Increase to Roby Zone near surface resources based on a revised, preliminary pit expansion shell and an updated cut-off grade of 0.6 g/t Pd.
- Initial resource estimates for the Powerline and Upper Offset Southeast Extension zones.
- Revision to the Sheriff Zone near surface resource estimate based on a lowering of the Pd cut-off grade and the disclosure of an initial mineral resource in the higher grade Powerline Zone, part of which overlaps with the Sheriff Zone.

The current mineral resource estimate does not include any RGO stockpile material. The Company determined that all of its remaining RGO resources (McKinnon et al., 2014) should be converted to a Proven mineral reserve based on positive results from the initial mining of the RGO stockpile in 2014.

Table 1-1: Mineral Resource Estimates for the Lac des Iles Mine Property effective February 2, 2015 (see Notes)
(Mineral Resources are Exclusive of Mineral Reserves)

Near-Surface Resources								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Measured	(g/t)	(000's)	(g/t)	(g/t)	(g/t)	(%)	(%)	(000's oz)
Roby Zone Pit Expansion	0.60	20,108	1.23	0.17	0.10	0.07	0.05	795
Powerline Zone	1.00	251	3.04	0.22	0.12	0.06	0.04	25
Sheriff Zone	1.00	4,858	1.45	0.15	0.09	0.06	0.06	227
North VT Rim	1.00	437	2.03	0.12	0.03	0.03	0.01	29
Total Measured	-	25,654	1.30	0.17	0.10	0.06	0.05	1,076
Indicated								
Roby Zone Pit Expansion	0.60	9,634	1.20	0.17	0.10	0.07	0.05	372
Powerline Zone	1.00	65	2.48	0.20	0.08	0.05	0.04	5
Sheriff Zone	1.00	220	1.29	0.16	0.09	0.06	0.06	9
North VT Rim	1.00	14	1.75	0.12	0.03	0.03	0.01	1
Total Indicated	-	9,932	1.21	0.17	0.10	0.07	0.05	387
Total Measured + Indicated	-	35,586	1.28	0.17	0.10	0.07	0.05	1,463
Inferred								
Powerline Zone	1.00	51	3.10	0.28	0.09	0.05	0.05	
Sheriff Zone	1.00	479	1.50	0.21	0.10	0.07	0.07	
Total Inferred	1.00	530	1.66	0.21	0.10	0.07	0.07	

Underground Resources - Hangingwall Zones								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Measured	(g/t)	(000's)	(g/t)	(g/t)	(g/t)	(%)	(%)	(000's oz)
Offset Hangingwall Zone	2.50	2,683	4.36	0.31	0.29	0.11	0.08	376
Indicated								
Offset Hangingwall Zone	2.50	4,952	4.36	0.31	0.30	0.12	0.09	694
Total Measured + Indicated	2.50	7,635	4.36	0.31	0.30	0.12	0.09	1,070
Inferred								
Offset Hangingwall Zone	2.50	4,581	3.90	0.27	0.25	0.10	0.09	
Upper Offset Southeast Extension	TBD	827	3.20	0.28	0.24	0.09	0.08	
Total Inferred	-	5,407	3.80	0.27	0.25	0.10	0.08	

Underground Resources - Footwall Zones								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Measured	(g/t)	(000's)	(g/t)	(g/t)	(g/t)	(%)	(%)	(000's oz)
Offset Footwall Zone	1.50	10,584	2.22	0.23	0.16	0.08	0.06	755
Roby Footwall Zone	1.50	4,159	2.43	0.21	0.18	0.06	0.05	325
Total Measured	1.50	14,743	2.28	0.22	0.17	0.08	0.06	1,080
Indicated								
Offset Footwall Zone	1.50	11,163	2.10	0.21	0.15	0.08	0.07	754
Roby Footwall Zone	1.50	2,341	2.34	0.20	0.17	0.06	0.05	176
Total Indicated	1.50	13,504	2.14	0.21	0.16	0.08	0.06	930
Total Measured + Indicated	1.50	28,247	2.21	0.21	0.16	0.08	0.06	2,010
Inferred								
Offset Footwall Zone	1.50	8,853	2.05	0.16	0.12	0.07	0.06	
Roby Footwall Zone	1.50	248	2.43	0.18	0.08	0.03	0.02	
Total Inferred	1.50	9,101	2.06	0.17	0.12	0.07	0.06	

Combined Resources - All Sources								
Category/Source		Tonnes	Pd	Pt	Au	Ni	Cu	Pd Contained
Sub-Category		(000's)	(g/t)	(g/t)	(g/t)	(%)	(%)	(000's oz)
Total Measured + Indicated Resources		71,468	1.98	0.20	0.14	0.07	0.06	4,543
Total Inferred Resources		15,038	2.70	0.20	0.17	0.08	0.07	

Mineral Resource Estimate Notes:

1. The mineral resource for the Offset hangingwall and footwall zones and the Upper Offset southeast extension zone were estimated as of December 31, 2014 by Denis Decharte, P.Eng. and QP. Mr. Descharte is an employee of the Company.
2. The mineral resource for the Powerline Zone was estimated as of February 2, 2015 by Chris Roney, P.Geo. and QP. Mr. Roney is a private consultant to the Company.
3. The mineral resource for the Roby Zone pit expansion was estimated as of December 31, 2014 by David N. Penna, P.Geo. and QP. Mr. Penna is an employee of the Company.
4. Measured resources are based on an average idealized drill spacing of 12.5 to 20 m. Indicated resources are based on an average idealized drill spacing of 30 to 40 m. Inferred resources are based on an average idealized drill spacing of between 60 to 80 m.
5. Resources have been adjusted to account for depletion from mining activities to December 31, 2014.
6. Cut off grades for all resource estimates described in this report are based exclusively on palladium grades and are derived or modified from those described in McKinnon et al. (2014).
7. Metal price and currency exchange rate basis is unchanged from McKinnon et al. (2014): US$700/oz palladium, US$1,453/oz platinum, US$1,320/oz gold, US$6.47/lb nickel, US$3.26/lb copper and a CDN$/US$ exchange rate of CDN$1.00 = US$0.95.
8. Density for all mineralized zones is estimated at 2.89 g/cm3.
9. Drill hole assays are generally based on one metre sample interval lengths. Grade capping was applied where necessary.
10. Assays values used in the resource estimates are assumed to represent actual total abundances with the exception of nickel, which is given as a near-total abundance.
11. Totals may not add due to rounding.
12. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.4 Mining

The mine planning exercise evaluated an updated block model that incorporated all exploration data prior to December 31, 2014. Refer to Figure 1-3. Mining zones were evaluated individually using a break even cut-off grade methodology where operating cost equals the net smelter return (NSR) value of the mineralization. The LDI 2015 budget operating costs were used with cost estimates developed from first principles where operational costs were not available. The cut-off grade is also reported as the Pd grade for the given NSR value, assuming a constant ratio between grades for all other metal values.

In the case of the open pit, historic mining information at LDI was deemed outdated, so reasonable current operating cost estimates were used from similar sized operations. Cut-off grade estimates were made for the open pit, Roby FW, Offset HW (above 1065L), Offset FW (above 1065L), and the Offset Zone below the 1065L.

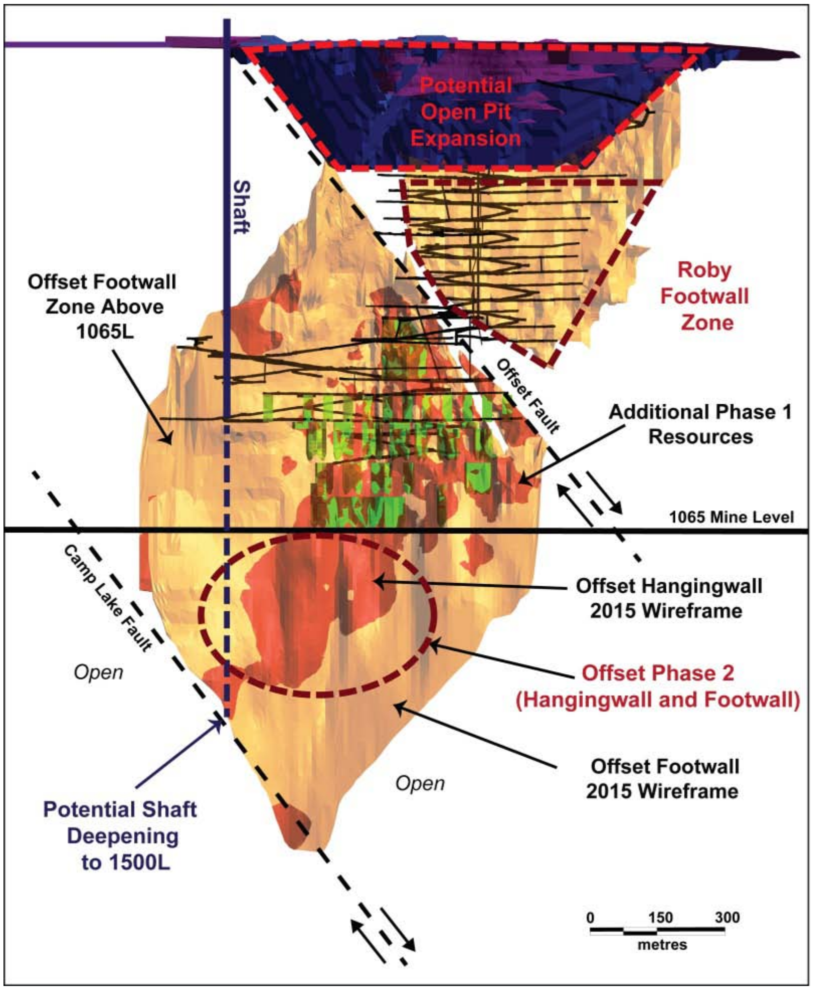


Figure 1-3: Overview of the Lac des Iles Mining Complex

The Current Mine Plan is based on the previously stated mineral reserves, augmented with the RGO stockpile and other mineral resources above the 1065L.This plan has a mine life through 2021, with a total tonnage mined of about 22.3 Mt. There are no Inferred mineral resources included in this plan. The underground mining is by blasthole methods using LDI employees.

The PEA examines two scenarios developed to capture potentially mineable material by region in an effort to isolate them for individual economic evaluations. Each mine plan targeted production rates to maintain a nominal mill throughput of 12,500 tpd. Production in 2015 is common to all scenarios and 2016 is the start year for these comparisons.

Open Pit Expansion: This scenario is the expansion of the currently idle open pit mine. Several near surface resource models were evaluated together to establish a potential ultimate pit mining shape. Due to the duration of the open pit, this scenario adds tonnage from the Offset FW above the 1065L which becomes potentially economic to supplement the open pit mill feed. The Open Pit Expansion scenario could add approximately 38.5 Mt of mill feed grading 1.27 g/t Pd, and when combined with the Current Mine Plan, extends the mine life to 2029. There are no Inferred mineral resources included in this plan. The mining is proposed to be by loader-truck using LDI employees.

Phase 2 Expansion at Depth: This scenario consists of deepening the shaft from the current 825L to 1500L to access the lower Offset Zone material that was drilled extensively in 2014 and will be further delineated by the 2015 exploration program. The Phase 2 Expansion includes the addition of a paste backfill plant, a material handling system and supporting infrastructure. Using an overhand blasthole method in a primary - secondary stoping sequence, the Phase 2 Expansion has the potential to add 5.1 Mt grading 3.6 g/t Pd of Measured and Indicated and 5.8 Mt grading 3.0 g/t Pd of Inferred mineral resources to the production profile.

The base case scenario for the PEA is the Current Mine Plan with the Open Pit Expansion (Base Case) and does not include the Phase 2 Expansion. The production plan for the Base Case is illustrated in Figure 1-4 and highlights production sequencing by zone.

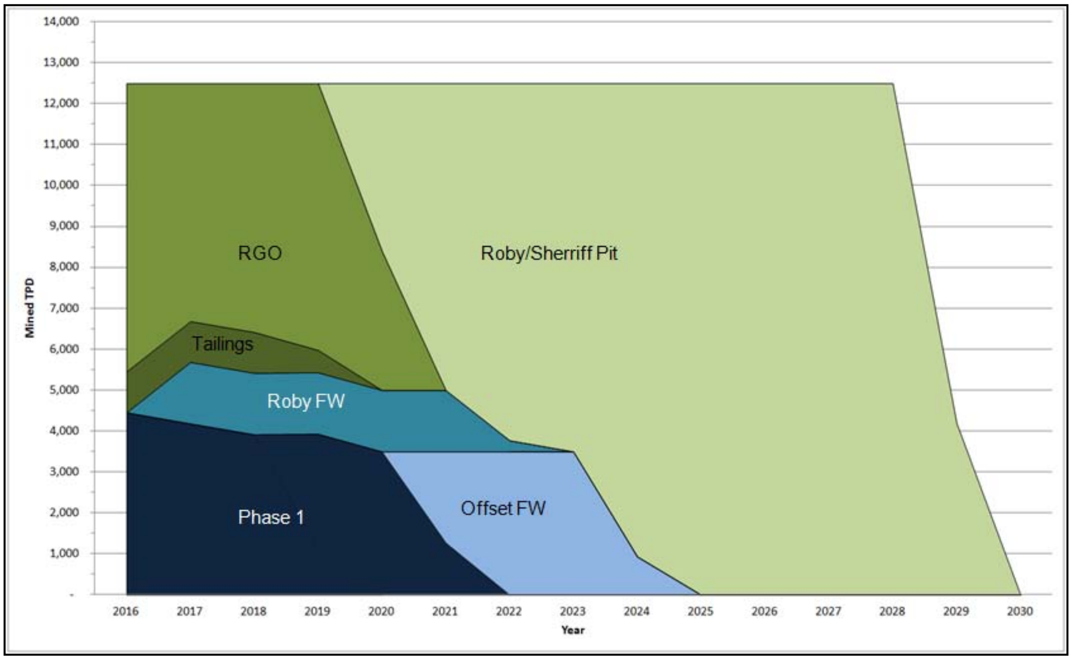


Figure 1-4: Production Plan for Base Case

1.5 Mineral Processing and Metallurgical Testwork

The LDI concentrator has a milling capacity of up to 15,000 tpd, depending on head grade, utilizing primary and secondary crushing for run of mine material, followed by SAG, Ball Milling and ultrafine grinding in Vertimills®. The mill is operated 365 days per year. Mill feed is a combination of low grade from the RGO stockpile (about 1.0 g/t Pd) and higher grade material from the underground mine (average grade of 4.4 g/t Pd in 2014). Palladium recovery is most sensitive to particle size and head grade. Tertiary grinding is performed by three Vertimills® (two in service) reducing particle size to a P_{80} of 50 microns.

Opportunity exists to further increase the Pd recovery from flotation by achieving a P_{80} of 38 microns. At this particle size, laboratory testwork performed by Xstrata Processing Services (XPS), without flash flotation, indicated that recovery of 86% Pd is achievable.

The LDI mill installed a flash flotation cell in late 2014 which has the potential of realizing additional incremental palladium recovery, above the current baseline. The flow sheet is shown in Figure 1-5.

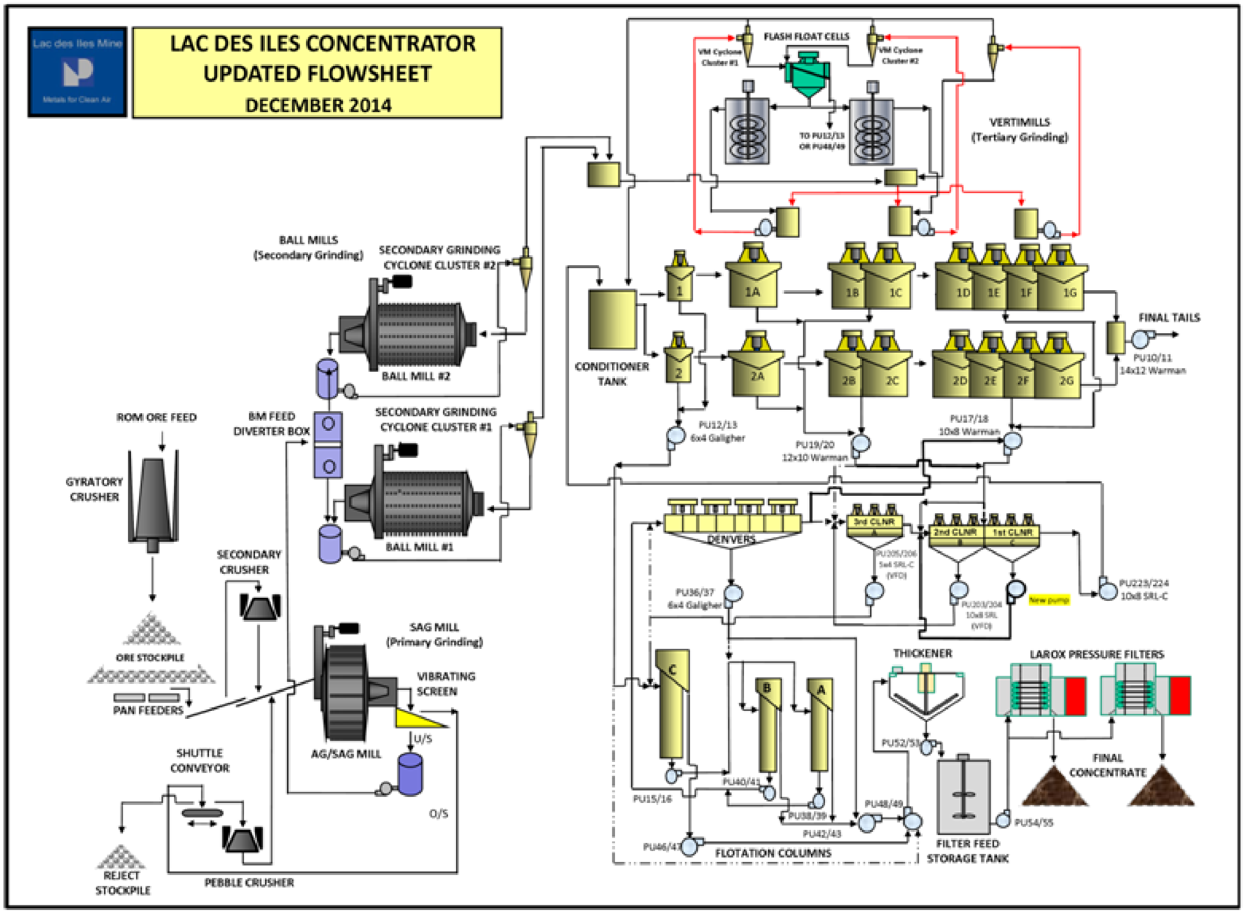


Figure 1-5: Lac des Iles Concentrator Updated Flow sheet

Production data for palladium for 2014 is shown in Figure 1-6. The black curve is a logarithmic fit of the data. With the commissioning of the flash cell, an increase in recovery of two percentage points is expected based on current operational results at LDI and industry experience. This is represented by the red curve which has been used to estimate palladium recovery in the financial section of this report.

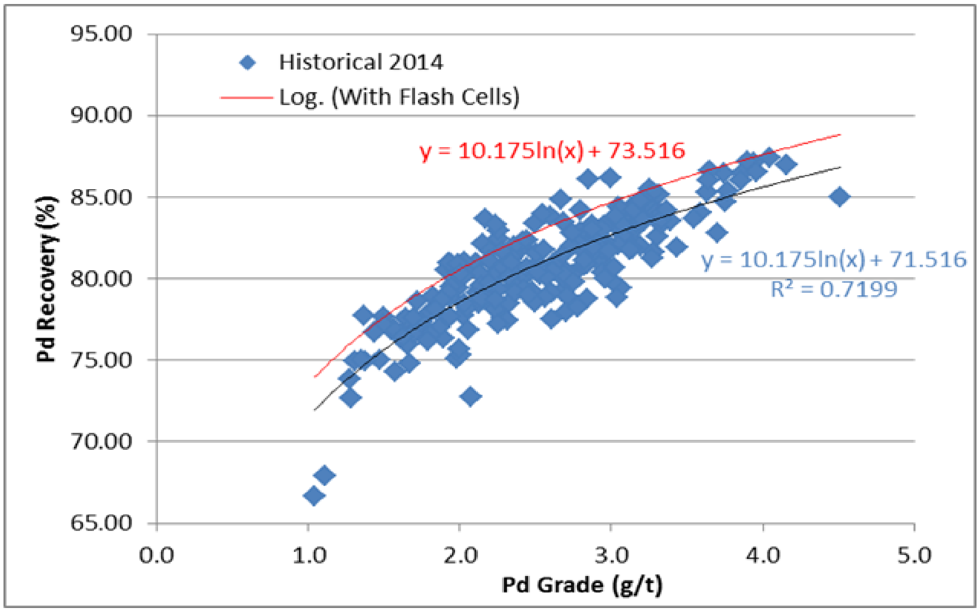


Figure 1-6: Grade/Recovery Data for Palladium

1.6 Capital and Operating Costs

1.6.1 Capital Cost

A project capital expenditure forecast was developed for the two expansion projects and is summarized in Table 1-2.

Table 1-2: Annual Project Capital Expenditures

$Cnd x 1 Million	SUM	2016	2017	2018	2019	2020	2021
Current Mine Plan	7.5	7.5					
Open Pit Expansion	51.1	0.0	0.0	8.5	22.0	20.3	0.2
Base Case	58.6	7.5	0.0	8.5	22.0	20.3	0.2
Phase 2 Expansion	242.2	35.2	76.7	99.7	29.3	1.3	

Table 1-3 is a detailed tabulation of the capital cost schedule for the Base Case scenario, which includes the Open Pit Expansion and selective mining of the FW zone in addition to the Current Mine Plan. The pre-stripping capital was treated as sustaining capital until the year in which production exceeded 60% of full commercial production.

Table 1-3: Annual Capital Expenditures for the Base Case

$Cnd x 1 Million	SUM	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Phase 1																	
Sustaining Mobile Equipment	16.5	3.7	3.7	3.1	3.0	3.0											
Sustaining Capital Projects	15.5	4.1	3.7	3.7	2.0	2.0											
UG Capital Development	20.5	9.3	7.2	2.0	2.0												
TMF Upgrade	9.0	9.0															
Sustaining TMF Expansion	19.6	4.6	4.0	4.0	4.0	2.5	0.5										
Subtotal Phase 1	**81.1**	**30.6**	**18.6**	**12.8**	**11.0**	**7.5**	**0.5**	-	-	-	-	-	-	-	-	-	-
Roby Footwall	-																
Mobile Equipment	10.5	7.5					1.5	1.5									
Sustaining TMF Expansion	2.8		0.5	0.5	0.5	0.5	0.5	0.1	-	-	-	-	-	-	-	-	-
Subtotal Roby FW	**13.3**	**7.5**	**0.5**	**0.5**	**0.5**	**0.5**	**2.0**	**1.6**	-	-	-	-	-	-	-	-	-
Studies	**1.0**	**1.0**															
Open Pit																	
Prestripping - Project	33.7				13.5	20.1				-							
Prestripping - Sustaining	50.8						20.5	19.3	11.0								
Sustaining Capital	20.6						1.5	1.5	1.5	1.5	1.5	5.8	5.8	1.5			
Sustaining TMF Expansion	34.7	-	-	-		1.5	2.7	3.1	3.3	3.3	4.5	4.6	4.6	4.6	2.6	-	-
Equipment	17.4			8.5	8.5	0.2	0.2										
Infrastructure Relocation	19.7						2.6		3.6	10.9	2.6						
Total Open Pit	**176.9**	-	-	**8.5**	**22.0**	**21.8**	**27.5**	**23.9**	**19.4**	**15.7**	**8.6**	**10.4**	**10.4**	**6.1**	**2.6**	-	-
Offset Footwall																	
Sustaining Mobile Equipment	6.0						1.5	1.5	1.5	1.5							
Sustaining Capital	3.0								1.5	1.5							
Sustaining TMF Expansion	3.7	-	-	-	-	-	0.8	1.3	1.3	0.3							
Subtotal Offset FW	**12.7**	-	-	-	-	-	**2.3**	**2.8**	**4.3**	**3.3**	-	-	-	-	-	-	-
Base Case Capital Expenditures	**285.0**	**39.1**	**19.2**	**21.9**	**33.6**	**29.9**	**32.4**	**28.4**	**23.7**	**19.9**	**8.7**	**10.4**	**10.4**	**6.1**	**1.5**	**0.0**	**0.0**
Project Capital	58.6	7.5	0.0	8.5	22.0	20.3	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Capital	226.4	31.6	19.2	13.4	11.6	9.5	32.2	28.4	23.7	19.9	8.7	10.4	10.4	6.1	1.5	0.0	0.0

1.6.2 Operating Cost

Many of the operating costs were derived based on LDI's 2015 operating cost plan. These were adjusted to forecast the operating cost for varying production scenarios. These adjusted costs were used to form the basis for cut-off grade calculations described previously in this summary and in more detail in Section 21.

In the case of the Open Pit Expansion, historic mining at LDI was deemed outdated, so reasonable current estimates were used from similar size operations.

Table 1-4 displays the operating cost summary for all zones associated with the Base Case plan.

Table 1-4: Operating Cost Summary for the Base Case Scenario

Cost per Tonne Milled					
Base Case	**Tailings / RGO**	**Open Pit**	**Offset Phase 1**	**Roby FW**	**Offset FW**
	$18.34	$20.01	$55.39	$47.35	$55.43

1.7 Financial Evaluation

1.7.1 Introduction

To evaluate the potential economic benefits of the required capital investment for each scenario, the Open Pit and Phase 2 expansion scenarios were evaluated independently and were structured to incrementally assess the overall economics with or without the expansion projects. The key model assumptions and financial results, project returns and cash flows are presented herein.

All economic assessments are calculated at the LDI mine level, and therefore, do not include certain costs including corporate office, interest, financing and exploration expenses.

Given the large lower grade resource at the LDI site and the potential to convert this material into a viable mining plan with a life of 15 years or greater, NAP felt it was important to use generally accepted long-term price forecasts. The company compiled, with the aid of its financial partners, a commodity price forecast from an extensive list of financial analysts. These prices are summarized in Table 1-5 and are used in the Base Case analyses.

Table 1-5: Consensus Commodity Forecast Price and Foreign Exchange Assumptions

Price	Unit	2016	2017	2018	2019+
Palladium	USD$/oz	901	935	948	855
Platinum	USD$/oz	1,440	1,543	1,600	1,611
Gold	USD$/oz	1,265	1,253	1,246	1,275
Nickel	USD$/lb	9.31	9.53	10.11	8.87
Copper	USD$/lb	3.11	3.29	3.39	3.01
Exchange Rate	CAD per USD	1.15	1.12	1.11	1.11

Source: Average of recent analyst forecasts as of January 28, 2015

1.7.2 Financial Results

A summary of the financial results is provided in Table 1-6, Table 1-7 and Table 1-8.

Table 1-6: Financial Results for Current Mine Plan

LOM Totals	Units	Current Mine Plan
Production	kt	22,329
Pd (g/t)	g/t	2.06
Pd	%	80.9%
Pd Recovered	oz	1,197,516
Pd Payable	oz	1,084,414
Net Smelter Return less Royalties	CAD $M	1,326
Total OPEX	CAD $M	(776)
EBITDA	CAD $M	550
Project CAPEX	CAD $M	(8)
Sustaining CAPEX	CAD $M	(84)
Change in Working Capital	CAD $M	70
Closure Costs	CAD $M	(15)
Pre Tax Cash Flow	CAD $M	513
Taxes	CAD $M	(21)
After-tax Cash Flow	CAD $M	493
After-tax NPV @5%	CAD $M	435

The Current Mine Plan is based on the previously stated mineral reserves, augmented with the RGO stockpile and other mineral resources above the 1065L.

The economic analysis on the mine expansion plans is at a scoping level and is preliminary in nature. The Phase 2 Expansion scenario includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. There are no Inferred mineral resources in the Current Mine Plan or the Open Pit Expansion.

Table 1-7: Financial Results for Open Pit and Phase 2 Expansion Scenarios

LOM Totals	Units	Open Pit Expansion	Phase 2 Expansion
Production	kt	38,514	10,857
Pd (g/t)	g/t	1.27	3.27
Pd Recovery	%	75.9%	84.5%
Pd Recovered	oz	1,194,222	963,102
Pd Payable	oz	1,080,105	871,476
Net Smelter Return less Royalties	CAD $M	1,438	980
Total OPEX	CAD $M	(961)	(538)
EBITDA	CAD $M	477	442
Project CAPEX	CAD $M	(51)	(242)
Sustaining CAPEX	CAD $M	(143)	(50)
Change in Working Capital	CAD $M	-	-
Closure Costs	CAD $M	-	-
Pre Tax Cash Flow	CAD $M	283	151
Taxes	CAD $M	(35)	(33)
After-tax Cash Flow	CAD $M	248	117
After-tax NPV @5%	CAD $M	138	22
After-tax IRR	%	31%	7%

The PEA demonstrates that newly classified, near surface mineral resources, previously not considered economic, have the potential to be extracted via an Open Pit Expansion. A $194M total capital expenditure, including a project capital cost of $51M and sustaining capital of $143M, generates an after-tax internal rate of return of 31% and an after-tax $NPV_{5\%}$ of $138M using the Base Case economic assumptions.

The mineral resources identified below the 1065L in the 2014 exploration drill program were used to evaluate the Phase 2 Expansion project. When considering the costs of deepening the mine shaft, constructing a paste backfill plant and other infrastructure, the project capital for the Phase 2 Expansion is estimated at $242M. This contributes to an after-tax rate of return of 7% and an after-tax $NPV_{5\%}$ of $22M using the Base Case economic assumptions. Due to the amount of inferred mineral resource and economic sensitivity, the inclusion of Phase 2 into the Base Case mine plan is not considered until the mineral resource is further upgraded and expanded.

The Base Case scenario developed in the PEA includes the Current Mine Plan, as stated previously, with the addition of the Open Pit Expansion. This combined scenario includes 61 Mt of mineralized material grading 1.6 g/t Pd that can be extracted during a mine life that extends to 2029. The Base Case scenario requires $59M of project capital expenditures and generates an estimated after-tax $NPV_{5\%}$ of $573M with no negative cash flow throughout the mine life. The financial results for the Base Case are depicted in Table 1-8 and Figure 1-7.

Table 1-8: Financial Results for the Base Case

LOM Totals	Units	Base Case Scenario
Production	kt	60,843
Pd (g/t)	g/t	1.56
Pd Recovery	%	78.3%
Pd Recovered	oz	2,391,013
Pd Payable	oz	2,164,520
Net Smelter Return less Royalties	CAD $M	2,765
Total OPEX	CAD $M	(1,738)
EBITDA	CAD $M	1,027
Project CAPEX	CAD $M	(59)
Sustaining CAPEX	CAD $M	(226)
Change in Working Capital	CAD $M	70
Closure Costs	CAD $M	(15)
Pre Tax Cash Flow	CAD $M	797
Taxes	CAD $M	(56)
After-tax Cash Flow	CAD $M	741
After-tax NPV @5%	CAD $M	573

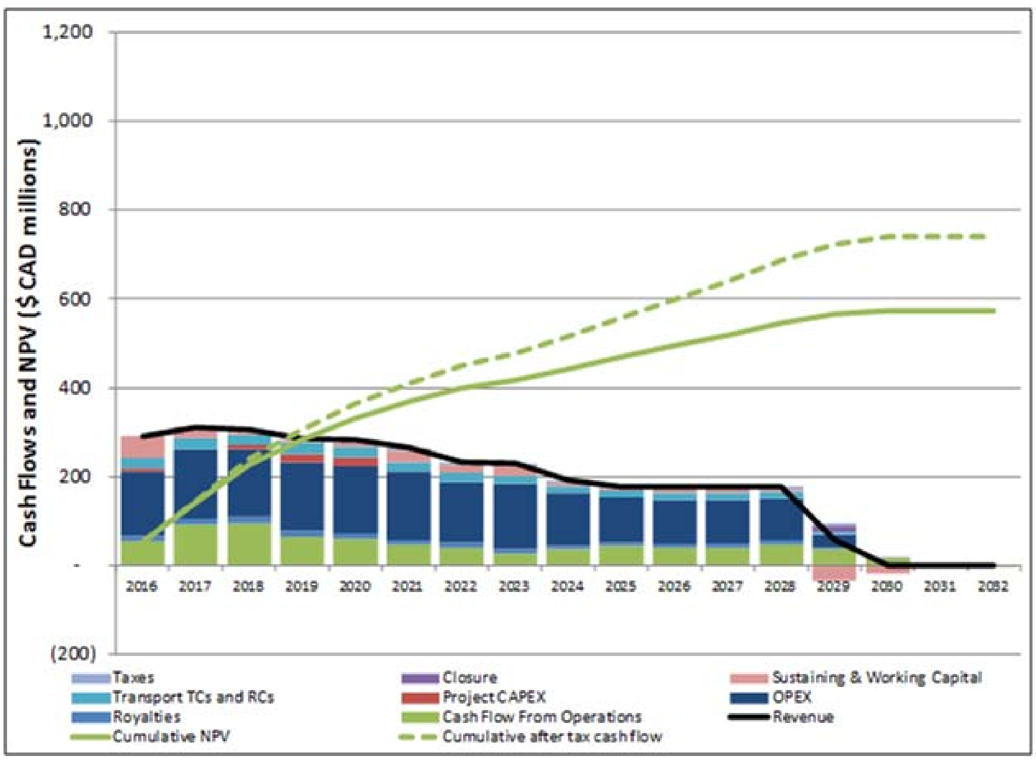


Figure 1-7: Base Case Plan After-tax Cash Flow

Table 1-9 shows sensitivities to palladium price and exchange rate and highlights consensus, spot and three year trailing pricing.

Table 1-9: Palladium Price and Exchange Rate Sensitivity – Base Case

NPV@ 5% CAD $ millions		Palladium Price (USD $/oz)																	
			3 yr Avg		Spot			Consensus											
		700	725	750	772	800	825	855	875	900	925	950	975	1,000	1,100	1,200	1,300	1,400	1,500
Exchange Rate CAD per USD	0.95	104	154	204	246	292	340	393	428	472	516	561	605	645	781	914	1,045	1,175	1,305
	1.00	151	203	254	295	343	394	449	487	533	579	626	663	700	841	980	1,117	1,255	1,392
	1.05	200	254	303	347	397	450	509	548	597	643	681	719	757	904	1,050	1,194	1,339	1,483
	1.10	243	295	347	392	445	501	562	603	649	689	729	768	807	960	1,111	1,263	1,413	1,564
	1.11	252	304	357	402	456	511	**573**	615	659	699	739	779	818	972	1,125	1,277	1,429	1,581
	1.15	285	340	394	441	496	554	618	655	697	738	779	819	860	1,019	1,177	1,335	1,493	1,650
	1.16	294	349	403	451	507	565	630	664	706	748	789	830	870	1,031	1,190	1,349	1,508	1,668
	1.20	327	384	440	489	547	608	666	701	744	786	828	870	912	1,078	1,243	1,407	1,572	1,737
	1.25	366	425	483	534	595	652	706	742	786	830	874	917	960	1,132	1,303	1,474	1,645	1,816
	1.30	411	472	533	587	646	697	753	790	836	881	926	971	1,016	1,195	1,374	1,552	1,731	1,909
	1.35	453	516	580	634	686	739	796	834	881	928	975	1,021	1,068	1,254	1,439	1,624	1,810	1,995
	1.40	495	560	626	673	726	779	838	877	926	975	1,023	1,071	1,120	1,312	1,504	1,697	1,889	2,081

1.8 Risks

The major risks are:

- If the contract for concentrate smelting is not renewed by its June 30, 2015 expiration or alternative options are not arranged, there could be a negative impact on company cash flows.
- The Base Case is most sensitive to mineral grade, metal recoveries, foreign exchange rate and palladium price. Economic results may change if the factors estimated in this report are not realized.
- The project will require additional environmental permitting and there is a risk that if these permits are delayed, project timelines will lengthen.

The execution risk for the development of the proposed Base Case mining plan is less than that of a greenfield project because:

- It is in a mining friendly jurisdiction and is an extension of an existing mine.
- It has an operating mill with known recoveries and a permitted tailings facility with long-term expansion capability.
- It has experienced personnel.
- It has long established and positive relations with the regional First Nation’s communities.

As in any mining project, there are risks associated with geological interpretation and geomechanics. These will require further analysis at the next level of study.

1.9 Conclusions

Based on long-term consensus pricing and exchange rates and the technical work completed at a scoping level in this PEA, opportunities exist at LDI for:

- An Open Pit Expansion project with an estimated project capital cost of $51M, an after-tax $NPV_{5\%}$ of $138M and an after-tax IRR of 31%.
- A proposed Base Case of adding an Open Pit Expansion to the Current Mine Plan that produces an after-tax $NPV_{5\%}$ of $573M and extends the mine life from 2021 to 2029.
- Selective mining of the large lower grade FW resource in both the Roby and Offset Zones.
- Further investigation of the Phase 2 Expansion project. With the current resource, this project has a potential after-tax $NPV_{5\%}$ of $22M, a project capital cost of $242M and an after-tax IRR of 7%.

The Phase 2 project is sensitive to changes in exchange rate and palladium price. A significant portion of the resource is still inferred and therefore the Phase 2 Expansion scenario has not been included in the Base Case.

1.10 Recommendations

Based on the results of this PEA study, it is recommended that NAP:

- Commence a Preliminary Feasibility Study (PFS) on the Base Case mine plan, which includes the Open Pit Expansion and selective mining of the FW zone. The estimated cost of this is approximately $1.5M.
- Investigate opportunities of accelerating the development of some of the higher grade open pit options that exist within the shell of the major pit expansion. The estimated cost of this is approximately $0.1M.
- Continue exploration and definition drilling at depth to upgrade and expand the Offset Zone resource to support future evaluation of the Phase 2 Expansion scenario. The estimated cost of this is approximately $7M.
- Perform engineering studies investigating opportunities to improve the business case for the Phase 2 Expansion. The estimated cost of this is approximately $0.4M.
- To perform an engineering study to investigate the benefits of adding paste backfill capabilities to support the Base Case mine plan. The estimated cost of this is approximately $0.1M.

Per NAP, these expenditures are provided in the LDI 2015 budget.

2. Introduction

NAP retained Hatch to prepare certain sections of this PEA on the potential viability of the resources at LDI. This resource includes measured, indicated and inferred categories. As part of this work, Hatch assisted NAP with the development of a strategic Life of Mine (LOM) plan with the purpose of identifying opportunities for mine expansion and extending the mine life.

A technical study completed by another firm in early 2013 and updated in 2014, confirmed mineral reserves down to the 1065 m level with additional mineral resources, mainly inferred, extending to 1,600 m. The mineral reserves calculated in the 2014 Technical Report totalled 15.0 Mt grading 2.77 g/t Pd and showed a mine life to 2019. A major exploration program was conducted throughout 2014 with the primary focus being the continued delineation of these mineral resources below the 1065 level. The results of that program are included in this report.

In light of improvements in spot and long term forecasts for palladium price and foreign exchange rate and the results of the 2014 exploration drilling program, this report has been prepared at the request of NAP, and is in accordance with the guidelines provided in NI 43-101 Standards of Disclosure for Mineral Projects.

The following information contained in this report was supplied by NAP:

- Current operating and capital budgets.
- Current production plan, including stope geometry, future development design and current openings.
- Geological block models and wireframes.
- Historical production data.

NAP is a Toronto, Ontario-based company, trading on the Toronto Stock Exchange (TSX) under the symbol PDL and on the New York Stock Exchange (NYSE) under the symbol PAL.

The LDI mine is located approximately 90 km northwest of Thunder Bay, Ontario.

The effective date of this report is February 18th, 2015. The effective date of the mineral resource is February 2, 2015.

The following QPs completed a site visit of the Property:

- Brian Young, P.Eng. of Hatch Ltd. visited the site from October 28 to October 29, 2014 for 2 days.
- David C. Peck, P.Geo. is a NAP employee and visited the site on several days in 2014.
- David N. Penna, P.Geo. is a LDI site-based employee and worked at site on a regular basis in 2014.

- Denis Decharte, P.Eng. is a NAP and partly site-based employee and worked at the site for parts of several of weeks in 2014.
- Chris Roney is a private consultant to LDI and worked at site for several days in 2014.

A summary of Qualified Persons (QPs) responsible for each section of this report is provided in Table 2-1.

Table 2-1: Summary of Qualified Persons

Section	Report Section	QP
1	**Summary**	Sign-off by Section
2	**Introduction**	Brian W. Young, P. Eng
3	**Reliance on Other Experts**	Brian W. Young, P. Eng
4	**Property Description and Location**	David C. Peck, P. Geo.
5	**Accessibility, Climate, Local Resources, Infrastructure and Physiography**	Brian W. Young, P. Eng
6	**History**	David C. Peck, P. Geo.
7	**Geological Setting and Mineralization**	David C. Peck, P. Geo.
8	**Deposit Types**	David C. Peck, P. Geo.
9	**Exploration**	David C. Peck, P. Geo
10	**Drilling**	David C. Peck, P. Geo.
11	**Sample Preparation, Analyses and Security**	David C. Peck, P. Geo.
12	**Data Verification**	David C. Peck, P. Geo.
13	**Mineral Processing and Metallurgical Testing**	Babak Houdeh, P. Eng
14	**Mineral Resource Estimates**	Dave Penna, P. Geo., Denis Decharte, P. Eng., Chris Roney, P. Geo., David C. Peck, P. Geo
15	**Mineral Reserve Estimates**	David C. Peck, P. Geo.
16	**Mining Methods**	Brian W. Young, P. Eng
17	**Recovery Methods**	Babak Houdeh, P. Eng
18	**Project Infrastructure**	Brian W. Young, P. Eng
19	**Market Studies and Contracts**	Rob Duinker, P. Eng
20	**Environmental Studies, Permitting and Social or Community Impact**	Brian W. Young, P. Eng
21	**Capital and Operating Costs**	Brian W. Young, P. Eng
22	**Economic Analysis**	Rob Duinker, P. Eng
23	**Adjacent Properties**	David C. Peck, P. Geo.
24	**Other Relevant Data and Information**	Brian W. Young, P. Eng
25	**Interpretation and Conclusions**	Sign-off by Section
26	**Recommendations**	Sign-off by Section
27	**References**	Brian W. Young, P. Eng

All units of measurement used in this technical report are in metric unless otherwise specified.

All currencies are expressed in Canadian dollars, unless otherwise specified.

3. Reliance on Other Experts

This report has been prepared by the QPs referred to in Table 2-1 for NAP. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the QPs at the time of preparation of this report.
- Assumptions, conditions, and qualifications as set forth in this report.
- Data, reports, and other information supplied by NAP.

For the purpose of this report, the QPs have relied on property ownership information provided by NAP. Hatch has not researched property title or mineral rights for the Lac des Iles mine and property and expresses no opinion as to the ownership status of the property.

- Brian Young, P. Eng., relied upon Mike Wanecki, Environment and Community Relations Superintendent of the Lac des Iles Mine for matters found in Section 20.
- Robert Duinker, P. Eng., relied upon David Langille, Chief Financial Officer for NAP for matters relating to taxes found in Section 22.

4. Property Description and Location

4.1 Description

The Property is located at latitude 49°10' north, longitude 89°37' west, 90 km northwest of the City of Thunder Bay in north-western Ontario (Figure 4-1). The Property comprises approximately 86.2 km^2 (8,623 ha) of mineral claims and leases. LDIM, a wholly-owned subsidiary of NAP, holds title to the leases and claims.

Figure 4-1: Property Location Map

4.2 Land Tenure

LDIM holds 100% interest in six mining leases, land registry parcel numbers 2982, 2983, 2984, 2985, 2531, and 2532 comprising 3,513 ha (Table 4-1). Contiguous with these leases are 54 mineral claims held 100% by LDIM (consisting of 331 claim units) covering 5,110 ha, for a total area of 8,623 ha (Table 4-2, Figure 4-2).

As of the effective date of this report, all claims and leases on the Property are in good standing.

Table 4-1: Lease Summary

Claim No.	Parcel	Area (ha)	Lease No.	Due Date	Annual Taxes ($)	Comments
CLM251	2982LTB	235.0	107910	2027-Aug-31	705	Surface and Mining Rights
CLM252	2983LTB	341.4	107911	2027-Aug-31	1,024	Surface and Mining Rights
CLM253	2985LTB	395.7	107909	2027-Aug-31	1,187	Surface and Mining Rights
CLM254	2984LTB	497.4	107908	2027-Aug-31	1,492	Mining Rights Only
CLM430	2531LTB	348.4	108139	2027-Sep-30	1,045	Surface and Mining Rights
CLM431	2532LTB	1,695.3	108138	2027-Sep-30	5,086	Surface and Mining Rights
Total	**6**	**3,513.2**	**-**	**-**	**10,539**	**-**

Table 4-2: Claim Summary

Township/Area	Claim Number	Unit	Area (ha)	Recording Date	Claim Due Date	Work Required ($)
Lac des Iles	845318	1	9.0	1985-Dec-04	2016-Dec-04	400
Lac des Iles	864416	1	12.9	1985-Nov-19	2016-Nov-19	400
Lac des Iles	864417	1	16.0	1985-Nov-19	2016-Nov-19	400
Lac des Iles	864418	1	12.6	1985-Nov-19	2016-Nov-19	400
Lac des Iles	864419	1	12.2	1985-Nov-19	2016-Nov-19	400
Lac des Iles	864420	1	13.1	1985-Nov-19	2016-Nov-19	400
Lac des Iles	864421	1	13.0	1985-Nov-19	2016-Nov-19	400
Lac des Iles	873576	1	16.7	1986-May-05	2016-May-05	400
Lac des Iles	873577	1	13.2	1986-May-05	2016-May-05	400
Lac des Iles	873578	1	16.5	1986-May-05	2016-May-05	400
Lac des Iles	873579	1	17.7	1986-May-05	2016-May-05	400
Lac des Iles	873580	1	14.1	1986-May-05	2016-May-05	400
Lac des Iles	873581	1	16.6	1986-May-05	2016-May-05	400
Lac des Iles	909816	1	11.2	1986-May-16	2016-May-16	400
Lac des Iles	1165555	12	189.0	1992-Mar-06	2016-Mar-06	4,800
Lac des Iles	1165557	3	55.5	1992-Mar-06	2016-Mar-06	1,200
Lac des Iles	1165558	8	134.9	1992-Mar-06	2016-Mar-06	3,200
Lac des Iles	1191463	6	99.3	1993-Aug-23	2016-Aug-23	2,400
Lac des Iles	1191464	9	127.3	1993-Aug-23	2016-Aug-23	3,600

Township/Area	Claim Number	Unit	Area (ha)	Recording Date	Claim Due Date	Work Required ($)
Lac des Iles	1191467	4	59.8	1994-Mar-25	2016-Mar-25	1,600
Lac des Iles	1194309	4	51.0	1991-Sep-09	2016-Sep-09	1,600
Lac des Iles	1200770	11	162.1	1994-Dec-02	2016-Dec-02	4,400
Lac des Iles	1205064	12	191.0	1999-Jul-20	2016-Jul-20	4,800
Lac des Iles	1207892	6	96.3	1995-Feb-03	2016-Feb-03	2,400
Lac des Iles	1207893	6	103.8	1995-Feb-03	2016-Feb-03	2,400
Lac des Iles	1215285	12	206.5	1996-Jun-17	2016-Jun-17	4,800
Lac des Iles	1215286	1	8.4	1996-Jun-17	2016-Jun-17	400
Lac des Iles	1215287	1	9.0	1996-Jun-17	2016-Jun-17	400
Lac des Iles	1215288	1	16.7	1996-Jun-17	2016-Jun-17	400
Lac des Iles	1215289	15	260.6	1996-Jun-17	2016-Jun-17	6,000
Lac des Iles	1215290	15	223.2	1996-Jun-17	2016-Jun-17	6,000
Lac des Iles	1215291	15	218.4	1996-Jun-17	2016-Jun-17	6,000
Lac des Iles	1215292	3	24.5	1996-Jun-17	2016-Jun-17	1,200
Lac des Iles	1215294	3	35.7	1996-Jun-17	2016-Jun-17	1,200
Lac des Iles	1217213	6	99.0	1997-Feb-21	2016-Feb-21	2,400
Lac des Iles	1217347	1	12.5	1998-Apr-14	2016-Apr-14	400
Lac des Iles	1232007	6	98.6	1998-Feb-05	2016-Feb-05	2,400
Lac des Iles	1232008	2	39.0	1998-Feb-06	2016-Feb-06	800
Lac des Iles	1232009	1	8.4	1998-Apr-14	2016-Apr-14	400
Lac des Iles	1232010	2	34.4	1998-Apr-14	2016-Apr-14	800
Lac des Iles	1232011	2	31.6	1998-Apr-14	2016-Apr-14	800
Lac des Iles	1232619	8	151.3	1998-May-07	2016-May-07	3,200
Lac des Iles	1232620	8	128.8	1998-May-07	2016-May-07	3,200
Lac des Iles	1232742	4	69.7	1998-Apr-21	2016-Apr-21	1,600
Lac des Iles	1232962	12	212.5	1999-Jun-29	2016-Jun-29	4,800
Shelby Lake	1238057	16	256.0	1999-Jun-29	2016-Jun-29	6,400
Shelby Lake	1238058	16	257.6	1999-Jun-29	2016-Jun-29	6,400
Shelby Lake	1238059	16	252.9	1999-Jun-29	2016-Jun-29	6,400
Lac des Iles	1238060	6	77.3	1999-Jun-29	2016-Jun-29	2,400
Lac des Iles	1238061	15	199.6	1999-Jun-29	2016-Jun-29	6,000
Lac des Iles	1238062	15	174.0	1999-Jun-29	2016-Jun-29	6,000
Heaven Lake	1245678	15	234.3	2000-Dec-08	2016-Dec-08	6,000
Heaven Lake	1245679	15	246.5	2000-Dec-08	2016-Dec-08	6,000
Total	-	331	5109.8	-	-	132,400

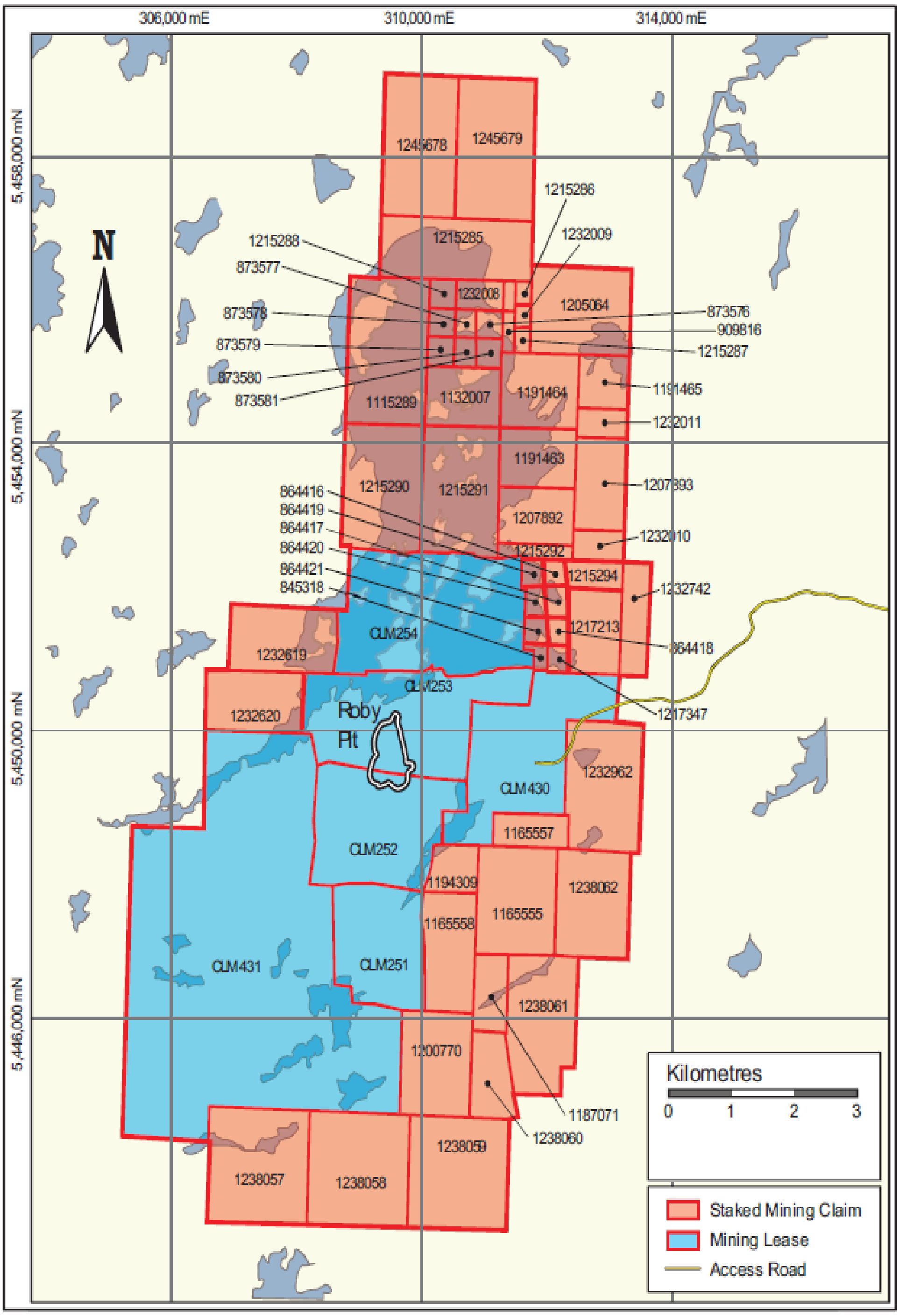


Figure 4-2: Land Tenure of the Property

4.3 Royalties

A 5% net smelter royalty (NSR) is paid to Sheridan Platinum Group Ltd. (SPG) from Toronto as a result of a 1994 settlement. The terms of the settlement saw NAP buy SPGs interest in the mine in exchange for cash, stock, and a reserved net interest of 3%, increasing to 5% after the year 2000. SPG also received $10 million and 2 million NAP shares.

4.4 Liabilities and Permits

All environmental liabilities are disclosed in Section 20, which covers the mine closure plan. All permits required to conduct work are also disclosed in Section 20.

4.5 Royalty Obligations

On August 31, 1994, LDIM, SPG and John Patrick Sheridan (Sheridan) entered into a royalty agreement pursuant to which SPG and Sheridan transferred certain LDIM claims and leases to LDIM in exchange for a NSR.

NAP is required to pay SPG and Sheridan a 5% NSR at LDIM until the expiration of the LDIM leases or any renewal thereof. The term NSR is defined in the royalty agreement as the net proceeds receivable by LDIM from the production and sale of concentrates from LDIM after deducting: the costs of sampling, assaying, transportation and insuring of concentrate, smelting, processing, and refining charges and penalties (excluding LDIM milling costs). All mining operations at LDIM are on the mining leases covered by the royalty.

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Site Topography, Elevation, and Vegetation

The Property is located in northwestern Ontario within the Superior Province of the Canadian Shield. The Property is situated in a boreal forest region typified by uplands forested mostly by black spruce, birch, poplar, and jack pine, as well as low-lying areas comprising lakes, streams, and bogs. Drainage is poorly integrated and generally runs to the South toward Lake Superior. The topography of the site is favourable for the placement of facilities, being generally of low relief. Elevations on the Property range from 418 to 550 m above sea level, excluding mining operations.

5.2 Access

The Property is located approximately 90 km northwest of the city of Thunder Bay in northwestern Ontario (Figure 5-1). Access to site is provided by a year-round gravel access road that connects to Provincial Highway 527 at a point 16 km east of the mine. Rail access is not available at or near the site. The nearest access to rail transport is at Thunder Bay, to the south, or at Armstrong, 133 km to the north. Air access is available at Thunder Bay and Armstrong, Ontario. Thunder Bay has an international airport which is serviced by several major airlines (Figure 5-1).

5.3 Climate

The Property experiences hot summers and cold winters, typical of this part of the Canadian boreal forest region. Maximum and minimum temperatures range from an extreme low of -30°C in the winter months to an extreme high of 38°C in the summer months. The average summer temperature is 14.8°C, and the average winter temperature is -13.0°C. The area is snow covered for 5.5 months per year, with the annual snowfall averaging approximately 150 cm. The precipitation in the form of rain averages 454 mm for the year. Prevailing winds on the Property are from the northwest. The relative humidity ranges from 50 to 77%. Weather conditions are rarely severe enough to halt mining operations and generally the only issue is related to safe traction on the access roads and ramps within the open pit.

Mill operations are enclosed and are not exposed to the weather other than feed inputs. LDI does not budget for weather-related shutdowns.

5.4 Infrastructure and Local Resource

With the proximity to Thunder Bay, there is access to experienced staff and personnel with good mining and processing experience. Mine and mill consumables including fuel, propane and cement are readily available.

Electrical power is supplied by Hydro One and LDI has approval to draw up to 47 MW of power. LDI has three Permits to Take Water that total approximately 62 million litres per day.

Ore is treated on site at the LDI Mill. The Tailings Management Facility is currently being expanded and will have a nominal capacity of 60 Mt. Waste rock is stockpiled on surface and is being removed as required for backfill underground. Re-activation of the open pit operation will result in additional waste rock being generated and may require revisions to the current permits and closure plan.

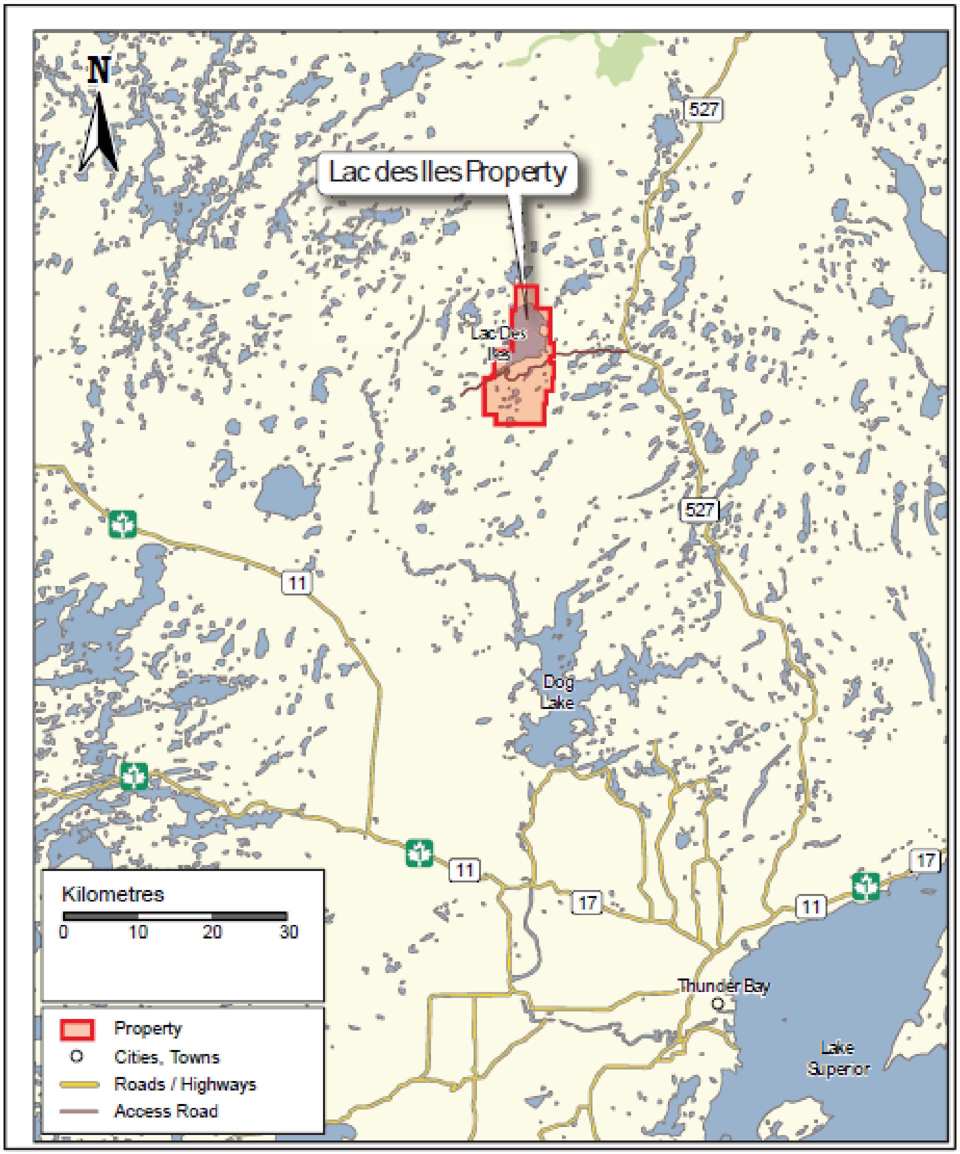


Figure 5-1: Access to Lac des Iles Mine

6. History

Geological investigations in the area began with reconnaissance mapping in the early 1930s, and again in the late 1960s, sparked by the discovery of aeromagnetic anomalies in the late 1950s. Various exploration programs were undertaken over the next 25 years by a number of companies.

In 1992, LDIM acquired the Property, with open pit production commencing in 1993. Until 2008, LDIM operated a combined open pit and underground mine from the Roby Zone and a 14,000 tpd processing plant, producing a bulk nickel-copper-PGE concentrate with gold credits. On October 21, 2008, NAP announced that it was temporarily placing LDI on care and maintenance effective October 29, 2008, in response to a significant decline in the palladium price. The LDI produced 212,046 oz of palladium in 2008 prior to going on care and maintenance.

When palladium prices began to recover in December 2009, NAP announced that it would restart operations exclusively from underground mineral reserves. On April 14, 2010, NAP announced that the Roby underground mine was back in production with a potential two to three year mine life.

In 2010, LDI produced 95,057 oz of palladium. Mine expansion commenced in early 2010 in order to access the Offset Zone. The Offset Zone is believed to be the fault-offset, down-plunge extension of the Roby Zone, a disseminated magmatic nickel-copper-PGE deposit. Mine expansion plans consisted of shaft sinking and extension of the ramp from the Roby Zone to the Offset Zone.

In 2011, LDI produced 146,624 oz of palladium from the blending of underground ore with lower-grade surface sources.

In 2012, LDI produced 163,980 oz of palladium from both underground and surface sources. Mining of the first Offset Zone stope was started.

In 2013, LDI produced 135,158 oz of palladium primarily sourced from the Offset Zone, low-grade surface stockpiles and residual ore from the Roby Zone.

In 2014, LDI produced 174,195 oz of palladium primarily sourced from the Offset Zone, low-grade surface stockpiles and residual ore from the Roby Zone.

The significant milestones of the operation are listed in Table 6-1.

Table 6-1: Property History

Year	Company	Description
1930-1960	-	Reconnaissance mapping, sparked by discovery of aeromagnetic anomalies in late 1950s.
1958	F.H. Jowsey Limited	Acquired two groups of 80 claims. Five (5) diamond drillholes.
1963	Geological Survey of Canada & ODM	Released aeromagnetic maps covering Lac des Iles area.
1963-1964	Gunnex	Optioned the Property from prospectors. Eleven (11) diamond drillholes totaling 1,516 m.
1966-1968	Anaconda American Brass Limited	Optioned the Property. Mapping and geophysical surveys. Thirteen (13) diamond drillholes totaling 1,900 m.
1974	Boston Bay	Optioned the Property. One hundred eleven (111) diamond drillholes totaling 18,009 m.
1986	Madeleine Mines	Nine (9) diamond drillholes totaling 3,007 m.
1987	Madeleine Mines	Two (2) diamond drillholes totaling 609 m.
1988	Madeleine Mines	Six (6) diamond drillholes totaling 1,081 m.
1989	Madeleine Mines	Four (4) diamond drillholes totaling 609 m.
1992	NAP	Twenty two (22) diamond drillholes totaling 1,177 m.
1993	NAP	Commercial open pit production of the Roby Zone commences.
1995	NAP	Fifty six (56) diamond drillholes totaling 7,802 m.
1992-1995	NAP	Detailed mapping in Roby Zone. Eastern part of the Roby Zone was stripped, mapped, and sampled. Mapping continued over the South Roby area.
1997	NAP	Nineteen (19) diamond drillholes totaling 4,243 m.
1998	NAP	One hundred fifty nine (159) diamond drillholes totaling 10,863 m.
1998	NAP	Surface exploration over Baker Zone, Moore Zone, North Varitextured Rim Zone, and Creek Zone. Extensive prospecting, stripping/trenching, channel sampling, geological mapping, geophysical surveys.
1999	NAP	Two hundred fifty four (254) diamond drillholes totaling 54,758 m. Sixteen (16) diamond drillholes tested the Baker Zone. Eleven (11) diamond drillholes tested Creek Zone. Limited diamond drillholes drilled at Moore Zone.
2000	NAP	Two hundred fifty three (253) diamond drillholes totaling 117,324 m including 11 diamond drillholes totaling 11,459 m that targeted the Offset Zone.
2001	NAP	Thirty six (36) diamond drillholes totaling 26,792 m including 11 diamond drillholes totaling 11,920 that targeted the Offset Zone.
2002	NAP	Eighty one (81) diamond drillholes totaling 47,602 m.

Year	Company	Description
2003	NAP	Twenty five (25) diamond drillholes totaling 10,211 m including 2 diamond drillholes totaling 1,773 m that targeted the Offset Zone.
2004	NAP	Underground development at High Grade Zone below Roby pit commenced. Four (4) diamond drillholes totaling 2,546 m including 1 diamond drillhole totaling 1,811 m that targeted the Offset Zone.
2005	NAP	Thirty eight (38) diamond drillholes totaling 22,623 m including twenty two (22) diamond drillholes totaling 27,698 m that targeted the Offset Zone.
2006	NAP	Underground commercial producing commenced from ramp access. Eight (8) diamond drillholes totaling 9,619 m.
2007	NAP	Forty five (45) diamond drillholes totaling 22,195 m all targeting the Offset Zone.
2008	NAP	LDI is placed on care and maintenance on October 29, 2008 due to declining metal prices. Thirty one (31) diamond drillholes totaling 18,988 m.
2009	NAP	Eighty six (86) diamond drillholes totaling 41,590 m all targeted the Offset Zone.
2010	NAP	Mining resumed in underground Roby High Grade Zone. Commenced a major expansion to mine via shaft to access the Offset Zone deposit. Two hundred forty two (242) diamond drillholes totaling 82,131 m.
2011	NAP	Offset Mine development underway. Two hundred thirty four (234) diamond drillholes totaling 79,991 m.
2012	NAP	Offset Mine development continues. 50,148 m of diamond drilling.
2013	NAP	Offset Mine Phase 1 completed, shaft commissioned. Two hundred twelve (212) diamond drillholes totaling 54,532 m.
2014	NAP	Ramp up to full production rate for Offset Zone. Sixty eight (68) diamond drillholes totalling 37,092 m.

Key highlights are:

- 1993: Began mining from the Roby Zone using open pit mining methods.
- 2001: Major expansion of the Roby open pit mine.
- 2002: Major increase in Roby Zone pit mineral reserves and mineral resources. Commissioning of a new 15,000 tpd mill. Record ore tonnage mined from Roby pit (7.25 million tonnes).
- 2003: Major write down of Roby open pit mineral reserves and mineral resources based on falling metal prices.

- 2006: Began mining underground from the Roby Zone via ramp access, while concurrently mining from the open pit.
- 2008: Mine was temporarily placed on care and maintenance, effective October 29, 2008, due to declining metal prices. On December 8, 2009, NAP announced its intention to restart operations at the mine.
- 2010: Mine was successfully restarted, ahead of schedule and under budget, in May 2010. Mining resumed from underground only (Roby Zone) as the open pit had been predominantly mined out. At the start of 2010, NAP also commenced a significant mine expansion to prepare for production from the Offset Zone, currently in progress.
- 2011: Underground Roby Zone mining was augmented with Offset Zone development material and available surface stockpiles. The mine expansion was significantly advanced.
- 2012: The Roby Zone open pit was restarted and blended with underground production from the Roby Zone and Offset Zone. The mine expansion progressed substantially, with most of the surface infrastructure completed.
- 2013: Shaft sinking was completed as was surface and underground infrastructure construction. Production from the shaft commenced in Q4.
- 2014: Ramp up to planned production for Phase 1 of the Offset Zone underground mine. No production from open pit.

Table 6-2 summarizes the annual production from the mine.

Table 6-2: Production History

	Totals	2014	2013	2012	2011	2010	2009	2008	2007	2006	2005	2004
Pd (oz)	3,039,719	174,194	135,158	163,980	146,624	95,057	0	195,083	286,334	237,338	177,167	308,931
Pt (oz)	194,528	13,072	10,222	11,187	9,143	4,894	0	16,311	0	0	18,833	25,128
Au (oz)	212,742	11,607	10,423	11,106	21,890	21,718	0	15,921	0	0	14,308	25,679
Cu (lb)	51,058,861	3,029,522	2,828,271	2,592,748	1,596,185	658,013	0	4,623,278	0	0	5,514,670	7,836,183
Ni (lb)	29,247,846	1,677,820	1,437,311	1,348,179	816,037	395,622	0	2,503,902	0	0	2,353,227	4,320,970
Pd Head Grade (g/t)	-	2.7	2.8	3.4	3.7	6.1	-	2.3	2.4	2.2	1.7	2.4
Pd Recovery (%)	-	82.4	80.7	78.4	78.3	80.8	-	75.3	74.7	74.0	69.6	75.2
Tonnes milled (t)	50,889,485	2,684,782	2,048,082	2,051,563	1,689,781	649,649	0	3,722,732	5,006,383	4,570,926	4,780,599	5,298,544
Tonnes of Ore Mined (t)	**56,380,092**	**2,637,023**	**2,093,669**	**2,063,260**	**1,830,234**	**693,078**	**-**	**3,676,418**	**5,143,066**	**4,648,090**	**3,705,555**	**4,574,134**
Underground (t)	6,605,931	1,225,547	816,705	853,600	988,503	615,926	-	615,630	768,841	721,179	-	-
Open Pit (t)	48,362,685		1,276,964	1,209,660	841,731	77,152	-	3,060,788	4,374,225	3,926,911	3,705,555	4,574,134
RGO	1,411,476	1,411,476										
Year		**2003**	**2002**	**2001**	**2000**	**1999**	**1998**	**1997**	**1996**	**1995**	**1994**	**1993**
Pd (oz)		288,703	219,325	123,281	95,116	64,441	73,390	59,477	60,426	76,668	59,026	-
Pt (oz)		23,742	19,180	10,073	6,074	4,744	4,770	4,083	4,285	5,013	3,774	-
Au (oz)		23,536	16,030	9,603	6,035	4,888	4,179	3,675	3,811	4,870	3,463	-
Cu (lb)		7,142,674	5,295,486	3,123,763	1,362,266	1,377,464	1,010,252	963,530	941,105	1,162,677	744	-
Ni (lb)		4,070,785	2,763,654	1,595,179	1,035,485	973,817	795,131	777,684	740,689	978,857	663,497	-
Pd Head Grade (/t)		2.3	1.9	2.1	4.5	-	-	-	-	-	-	-
Pd Recovery (%)		75.5	73.8	67.4	74.0	-	-	-	-	-	-	-
Tonnes milled (t)		5,159,730	4,851,621	2,662,240	893,017	894,168	963,332	803,189	757,248	743,580	607,390	50,929
Tonnes of Ore Mined (t)		**4,396,847**	**7,250,963**	**5,768,157**	**2,689,634**	**1,271,816**	**963,332**	**803,189**	**757,248**	**789,969**	**573,481**	**50,929**
Underground (t)		-	-	-	-	-	-	-	-	-	-	-
Open Pit (t)		4,396,847	7,250,963	5,768,157	2,689,634	1,271,816	963,332	803,189	757,248	789,969	573,481	50,929

7. Geological Setting and Mineralization

In addition to unpublished internal investigations, the information presented in this section is partly sourced from previous NI 43-101 technical reports (McKinnon et al., 2014; Clow and Rennie 2004; McCombe et al. 2009; Blakely 2009; Routledge 2010; McCracken et al. 2013) and partly from selected published accounts of the geology and PGE copper-nickel mineralization of the LDIM block property including Watkinson (1979), Sutcliffe and Sweeny (1986), MacDonald and Lawson (1987), MacDonald (1988), Sutcliffe (1989), Brugmann et al. (1989), Edgar and Sweeny (1991), Lavigne and Michaud (2001), Michaud and Lavigne (2003), Hinchey et al. (2005), Lavigne and Michaud (2005), and Gomwe (2008).

7.1 Regional Geology

The Property is underlain by mafic to ultramafic rocks of the LDIM intrusive complex (LDI-IC; Figure 7-1). The LDI-IC is the best documentation of a suite of Neoarchean mafic to ultramafic intrusive bodies occurring within a sub-circular area of approximately 35 km by 40 km in the Wabigoon Subprovince of the Canadian Shield. The intrusions are located immediately to the north of the Quetico Subprovince and directly west of the Nipigon embayment of the Mid-continent Rift System. At the time of writing, NAP owned or had options to acquire a majority interest in most of the known LDIM suite intrusions including parts or all of the following bodies: LDI-IC; Legris Lake intrusion; Wakinoo Lake intrusion; Demars Lake intrusion; Taman Lake intrusion; Dog River intrusion; Buck Lake intrusion; and Tib Lake intrusive complex (Figure 7-1). The easternmost bodies of the LDIM suite of intrusions are the LDI-IC and the Legris Lake complex. Both the LDI-IC and the Legris Lake intrusion appear to have been emplaced along a pre-existing northeast-trending splay structure (Shelby Lake fault) emanating from the east to northeast trending collisional structural boundary zone between the Quetico and Wabigoon subprovinces during the Shebandowanian orogeny at approximately 2,695 Ma (Corfu and Stott 1986). Similarly, many of the LDIM suite intrusions located in the western part of the LDIM area are spatially associated with northeast to north striking faults that splay off this collisional boundary (e.g., Dog River fault, Figure 7-1).

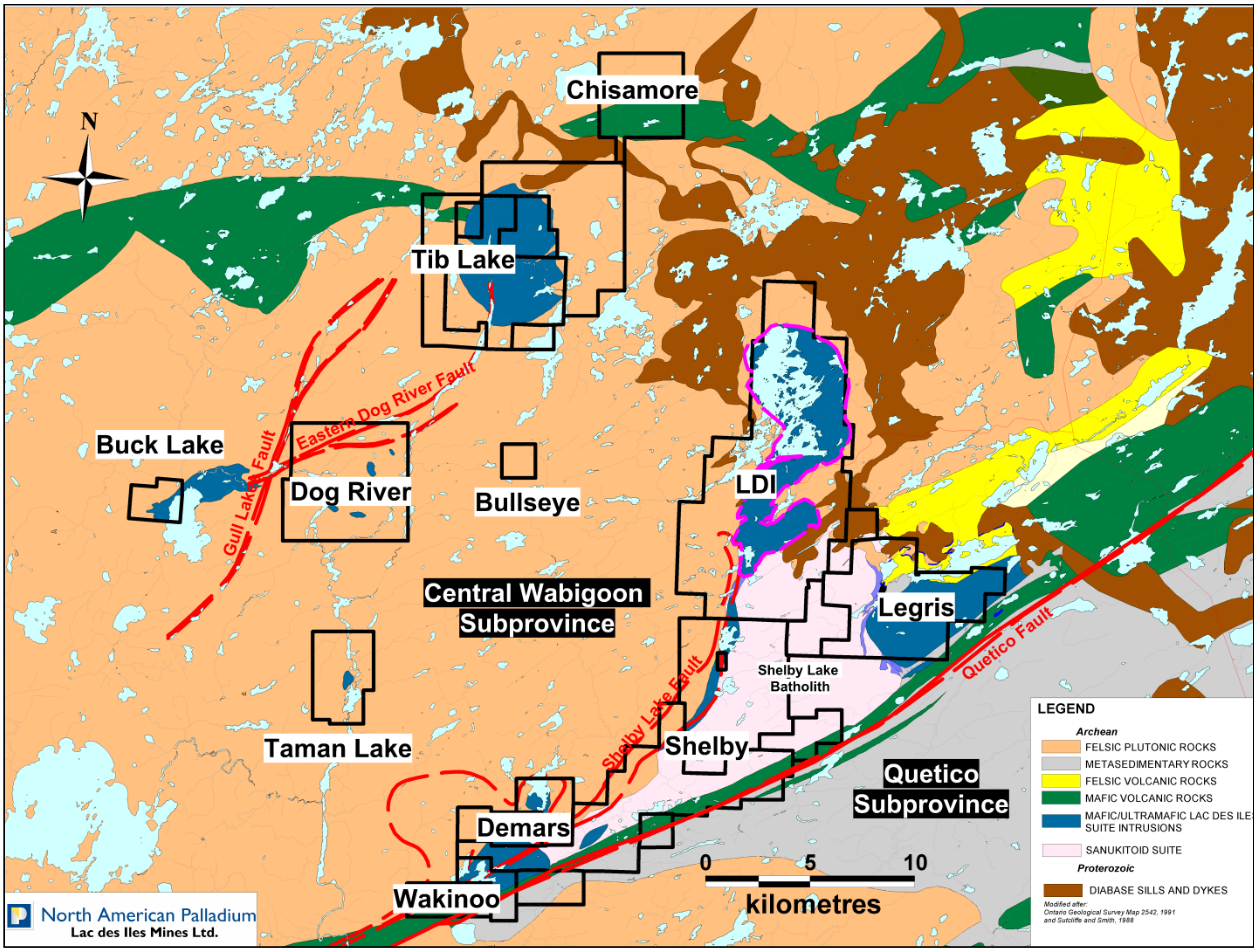


Figure 7-1: Simplified Regional Geology of the Lac des Iles Suite Intrusions

General descriptions of the regional geology of the area covered by the LDIM suite intrusions can be found in Pye (1968), Sutcliffe et al. (1986), Stone et al. (2003), and Stone and Davis (2006). The LDIM suite of intrusions is predominantly mafic in composition although some complexes, including the North LDIM intrusion, contain more ultramafic than mafic rocks. The uranium-lead dates on zircon in the mafic rocks showed that the LDIM suite intrusions were likely emplaced between 2,699 and 2,686 Ma (Stone and Davis 2006) during the same time that sanukitoid magmatism (Stern et al. 1989) and associated copper-PGE mineralization was forming in more evolved diorite-monzonite-granodiorite intrusions in both the Wabigoon Terrane and the Quetico Subprovince. The close spatial association between some of the LDIM suite intrusions and sanukitoid bodies (e.g., along Shelby Lake fault, Figure 7-1) suggests that a genetic relationship exists between these contemporaneous magmatic events.

The LDIM suite of intrusions typically has a characteristic sub-circular morphology and commonly exhibit igneous layering dipping inwards from the intrusive margins. The intrusions were emplaced into 3.01 to 2.89 Ga granite-greenstone basement rocks recently designated as the Marmion Terrane and representing an older slice of magmatic arc-related crustal rocks within the former Wabigoon Subprovince (Stone and Davis 2006). The Marmion Terrane, formerly included as part of the south-central Wabigoon Subprovince, comprises intermediate to felsic orthogneiss and infolded, metamorphosed supracrustal rocks including mafic-felsic volcanics and clastic and chemical sedimentary rocks. The mafic components of the LDIM suite intrusions are commonly dominated by norite units with gabbro only being developed in the most evolved units that typically occupy the interior/upper portions of the bodies. Ultramafic components occurring in some of the LDIM suite intrusions can include dunite, peridotite and pyroxenite. These units can have orthopyroxene or clinopyroxene as the dominant pyroxene. Most of the known LDIM suite intrusions host economically interesting (e.g., greater than 1 g/t combined palladium + platinum) PGE ± copper-nickel sulphide mineralization in the form of surface showings and/or shallow drilling intersections. The best documented member of the suite is the LDI-IC that remains the only member of the suite in which significant PGE resources have been defined in accordance with NI 43-101.

7.2 Property Geology

The Property captures the known extent of the LDI-IC, an irregularly-shaped Neoarchean-age mafic-ultramafic intrusive body having maximum dimensions of approximately 9 km in the north-south direction and approximately 4 km in the east-west direction (Figure 7-2). The complex is currently interpreted to be made up of three discrete intrusive bodies:

- The North LDIM intrusion (NLDI) characterized by a series of relatively flat-lying and nested ultramafic bodies with subordinate mafic rocks.
- The Mine Block intrusion (MBI), described in detail in Section 7.2.1.
- The South LDIM intrusion (SLDI), a predominantly mafic (gabbroic) intrusion having many similarities to the western part of the MBI in terms of rock types and textures.

In addition, a poorly exposed/documented gabbroic intrusion, the Camp Lake intrusion, occurs in the southwestern part of the Property. The Camp Lake intrusion is not currently considered to be a priority target for PGE exploration (Figure 7-2).

The principal rock types in and adjacent to the LDI-IC are discussed in Sections 7.2.1to 7.2.3 with reference to the host intrusion and the Property geology map (Figure 7-2). The term gabbro or gabbroic is applied as a general indicator of any mafic intrusive rock having a mineral assemblage dominated by plagioclase and pyroxene (either orthopyroxene or clinopyroxene).

To date, NAP’s exploration activities have been focused on the MBI with lesser, intermittent activity on both the NLDI and SLDI.

7.2.1 North LDIM Intrusion

The geology of the NLDI is currently being re-assessed as part of a Ph.D. thesis project being carried out by L. Djon at Queens University, Kingston, Ontario. Initial findings from this study, coupled with historical exploration activities and recent lithomagnetic interpretations derived from the Company's 2012 versatile time-domain electromagnetic (VTEM) survey over most of the Property, suggests that:

- The NLDI is a complex polyphase intrusive body likely consisting of a series of nested to locally cross-cutting intrusions.
- The NLDI was approximately coeval with the MBI although rarely observed intrusive contacts between the two bodies suggest that the NLDI was younger.
- Most of the NLDI is composed of ultramafic rocks including olivine-rich to orthopyroxene-rich rock types (dunite, harzburgite, orthopyroxenites, and websterite) and clinopyroxene-rich rock types (clinopyroxenite, websterite, lherzolite and wehrlite) and featuring cyclic igneous differentiation and local well-developed igneous layering.
- Evidence of metamorphism and hydrothermal alteration are minimal throughout the intrusion; however local strong alteration to massive serpentinite is commonly observed in olivine-rich ultramafic units.
- Historical surface prospecting, mapping and limited trenching and diamond drilling has identified several areas in the NLDI where PGE grades exceed 1 g/t in grab or short (approximately 1 m) interval sampling.
- The NLDI PGE occurrences have palladium: platinum ratios of approximately 3:1, in contrast to the characteristic approximate 10:1 palladium: platinum ratios observed in the mineralized zones of the MBI.
- Disseminated chromite mineralization is locally developed in association with olivine pyroxene cumulate rocks in the NLDI with maximum chromium contents of 4.8% having been reported to date.

A revised geological interpretation including the distribution of anomalous PGE occurrences (greater than 1 g/t palladium + platinum + gold) for the NLDI is shown in Figure 7-3.

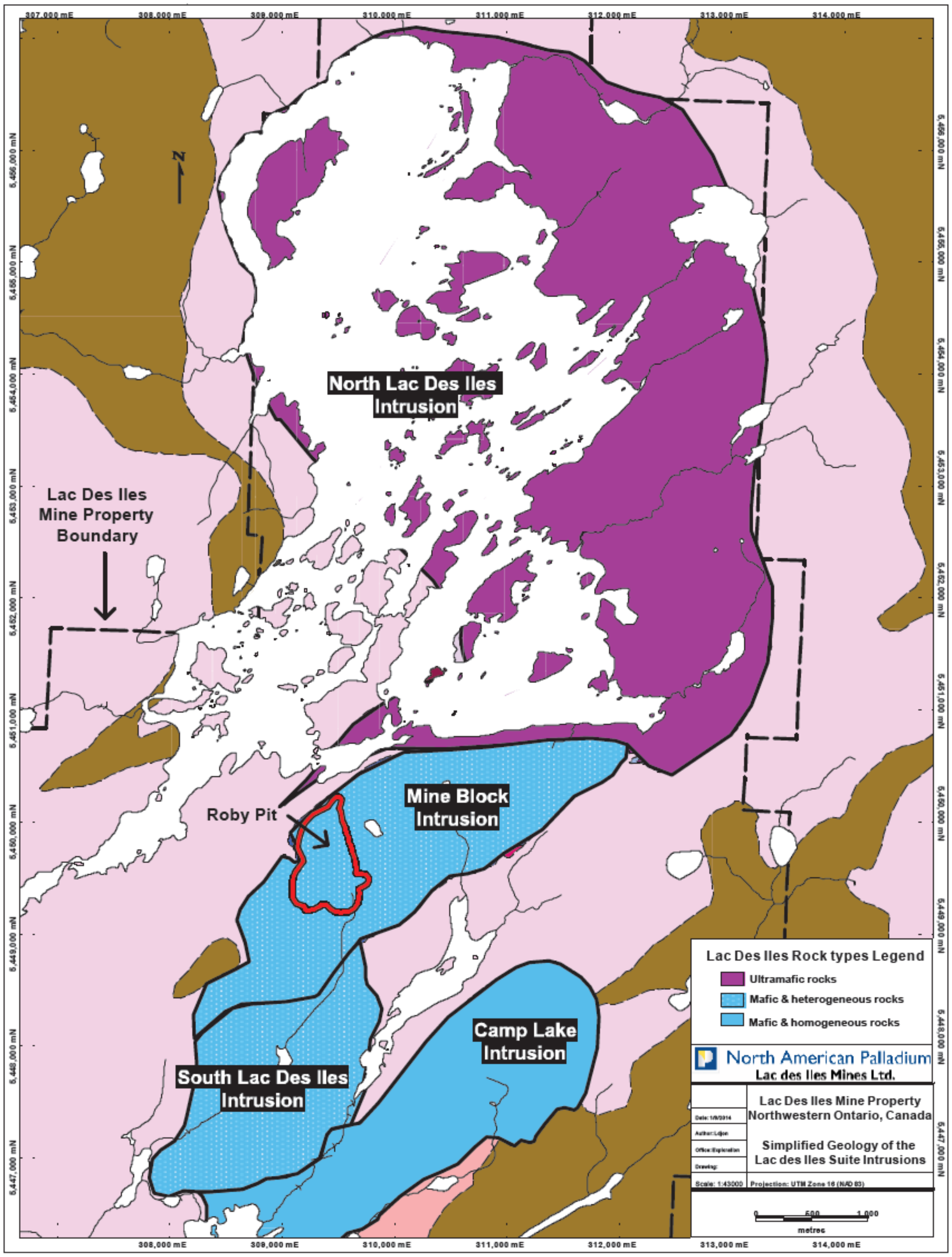


Figure 7-2: Simplified Local Geology of the Lac des Iles Suite Intrusions

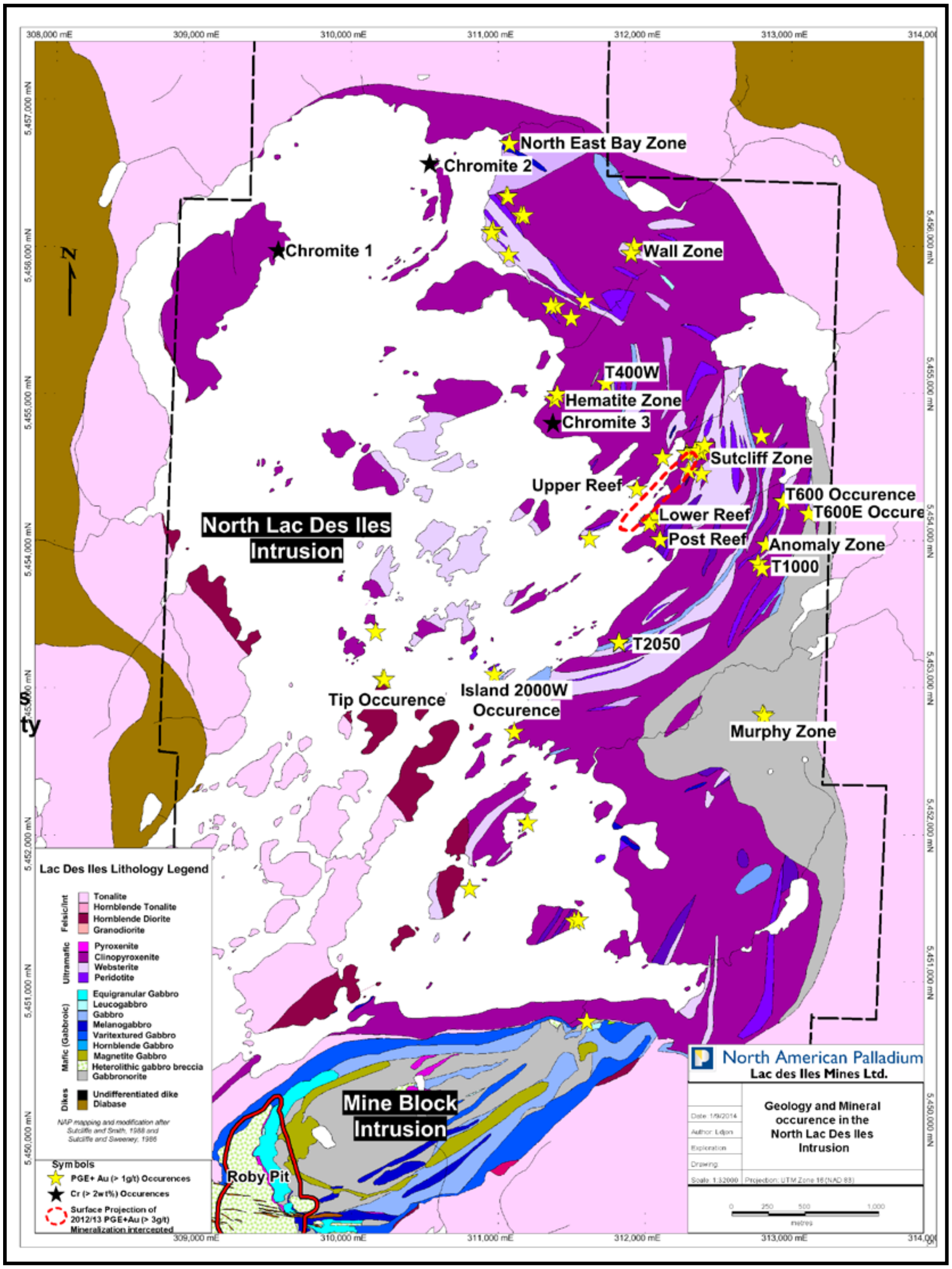


Figure 7-3: Geology and Mineral Occurrences in the NLDI

7.2.2 Mine Block Intrusion

The MBI is a small, teardrop-shaped mafic complex with maximum dimensions of 3 km by 1.5 km with an elongation in an east-northeast direction (Figure 7-4). The MBI consists of gabbroic (noritic) rocks and metamorphosed and/or hydrothermally altered equivalents with highly variable plagioclase-pyroxene proportions, textures and structures. Accessory igneous minerals include magnetite and titanium-rich magnetite, ilmenite, and quartz-feldspar granophyre. The MBI was emplaced into predominantly intermediate composition orthogneiss basement rocks. The emplacement age of the MBI has been established by precise uranium-lead zircon methods as 2,689 to 2,693 Ma (Stone and Davis 2006 and references contained therein). The MBI geology is dominated by gabbroic, melanogabbroic and leucogabbroic rock types. The common reference to gabbroic rather than noritic rocks in the many historical reports on the geology of the MBI is a reflection of the continued difficulty in distinguishing the composition of igneous pyroxenes in both outcrop and drill core. This difficulty has resulted in a mixed lithological nomenclature for the MBI in which gabbro, norite, and gabbronorite rock names have been somewhat interchangeably used. However, recent internal and external research has shown that the majority of the mafic rocks in the MBI, especially those associated with palladium mineralization, have clear noritic affinities such that orthopyroxene (as opposed to clinopyroxene) is the earliest-formed and generally most abundant igneous pyroxene in the rocks. In this way the MBI has affinities to the mafic portions of better documented giant mafic-ultramafic complexes such as the Bushveld, Great Dyke, and Stillwater complexes. In terms of its rock types, textures, and mineralization styles the LDI-IC is generally analogous to the Platreef Deposit of the northern lobe of the Bushveld Igneous Complex in South Africa (Kinnaird and MacDonald 2005; Kinnaird et al. 2005).

The elliptical-shaped distribution of major rock units within the MBI is depicted in Figure 7-4. Mapping and drilling have shown that most of the units exposed along the northern and southern margins dip moderately to steeply to the south but the marginal units at the western end of the MBI that host the majority of the palladium mineralization on The Property display steep easterly dips. The intrusion is dissected by a series of brittle to ductile faults and shear zones, some of which appear to control the distribution of higher-grade palladium mineralization. One important fault, the Offset fault, separates the Roby Zone and Offset Zone deposits and is part of a series of mapped west-southwest trending faults and shear zones on the Property. A major north-trending shear zone appears to have cut the western end of the MBI and is spatially associated with the development of high-grade palladium mineralization in both the Roby Zone and Offset Zone deposits. It is likely that this north-trending shear zone represents an extension to the Shelby Lake fault system that is clearly defined to the south of the Property (Figure 7-1). Other important fault systems include northwest dipping sinistral shear zones (e.g., northern margin of MBI) and early northwest striking faults that, in part, appear to control the distribution of the LDI suite of intrusions.

Textural and mineralogical variability is greatest in the outer margins of the MBI, especially along the well documented western and northern margins that host most of the known palladium resources and palladium-rich mineralized zones on the Property.

Commonly observed textures in the noritic marginal units of the MBI include equigranular, fine- to coarse-grained (seriate textured), porphyritic, pegmatitic and varitextured. The interior portions of the MBI consist of more regularly-textured and evolved rock types including magnetite gabbro and leucogabbro (Figure 7-4).

Varitextured gabbroic units in the northern and western margins locally include heterolithic gabbro breccia zones consisting of cognate mafic to ultramafic xenoliths of highly variable form and size within a matrix of varitextured gabbro. Internal to the varitextured rim of the western and northern MBI is a foliated medium-grained gabbro referred to as equigranular gabbro (EGAB; formerly named “East Gabbro”). In the westernmost part of the MBI an improperly named pyroxenite unit (PYXT) is commonly developed along the contact between the varitextured gabbro unit (footwall side), the EGAB unit and the varitextured rim. Recent research has demonstrated that the PYXT unit is a highly sheared, schistose and recrystallized norite to melanorite originally featuring cumulus orthopyroxene and intercumulus plagioclase and minor clinopyroxene.

7.2.3 South LDIM Intrusion

The geology of the western MBI and SLDI are broadly equivalent. Historical airborne magnetic survey data and very recent (2013) structural mapping and 3D structural modelling suggest that both bodies were emplaced as broadly ring-shaped, dike-like intrusive complexes with magmatism focused along the northern extension of the near-vertical, north-trending Shelby Lake fault. The SLDI features similar varitextured gabbro (VGAB) and gabbronorite units that are observed in the western MBI and anomalous PGE abundances in excess of 1 g/t palladium + gold are also recognized from historical surface sampling and limited drilling. The boundary between the northern end of the SLDI and the southwestern part of the MBI is not well defined but has been inferred from sharp gradients in the available airborne magnetic survey imagery (Figure 7-2). Recent geological insights suggest that the western MBI and much of the SLDI are part of a single, vertically-oriented, north-trending dyke-like intrusion that formed subsequent to the emplacement of the main mass of the MBI.

7.3 Deposit Geology

The geology of the Offset Zone deposit is illustrated and described with reference to an interpreted type geological cross section through the central part of the deposit (Figure 7-5).

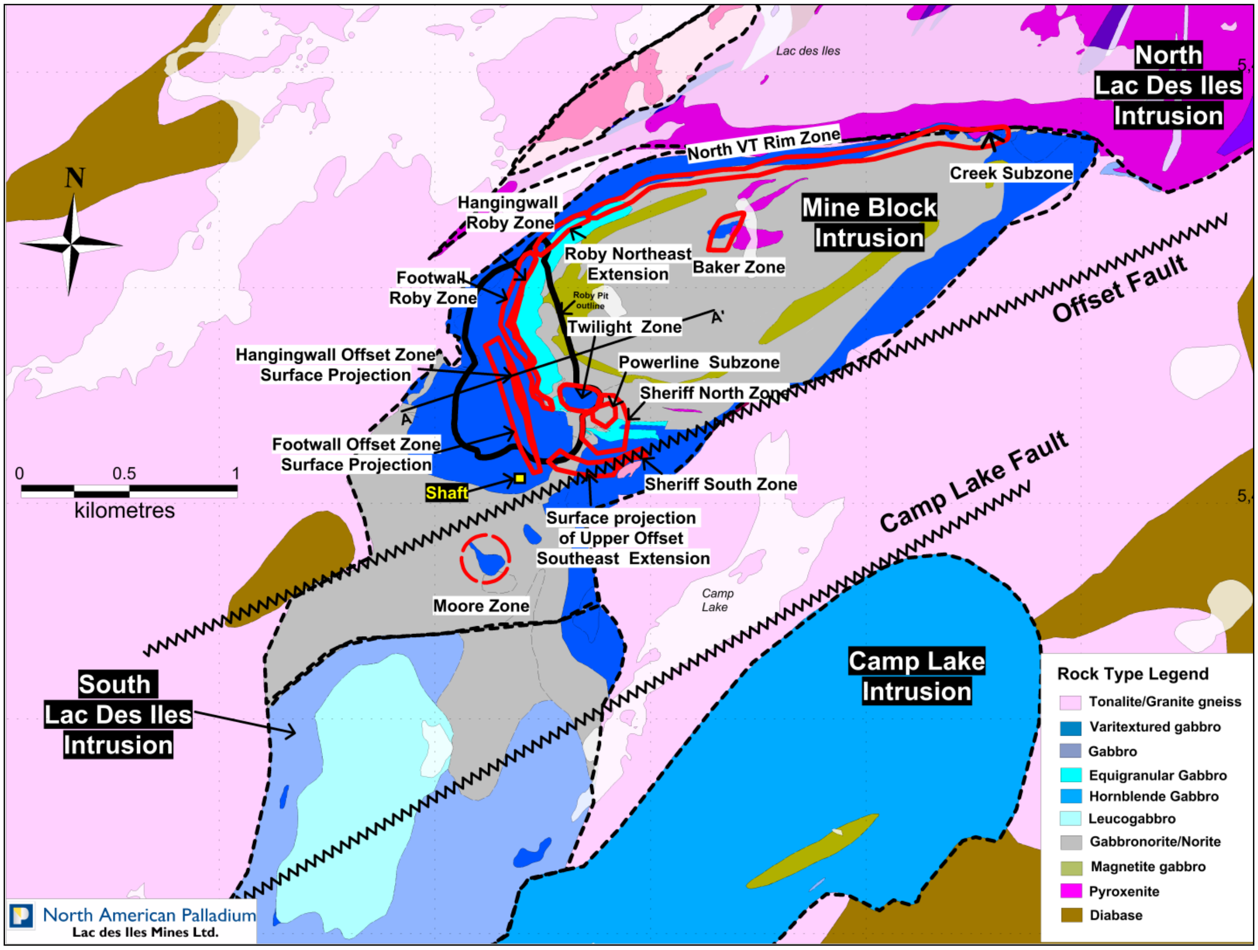


Figure 7-4: Geology and Palladium Mineralization of the Mine Block Intrusion

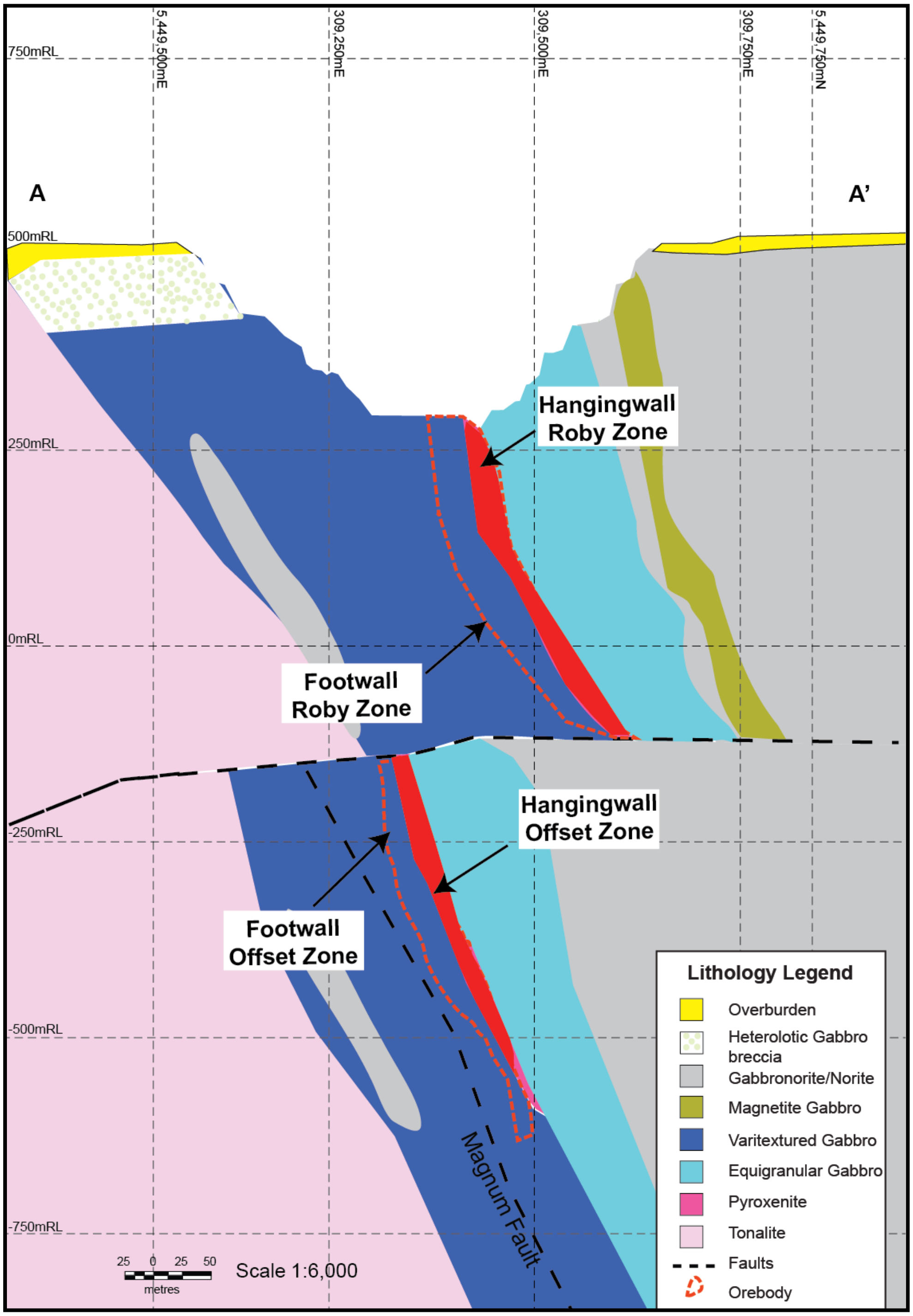


Figure 7-5: Type Geological Section through the Offset Zone Deposit (Looking North Through the Central Portion of the Deposit)

7.3.1 Structure

Three major faults and one major shear zone have been interpreted to influence the geology and PGE-copper-nickel grade distributions in the Offset Zone. A north-striking and steeply east-dipping zone of intensive schistosity development, local mylonitization and pervasive recrystallization (destruction of original plagioclase + pyroxene assemblages by amphibole ± chlorite ± talc ± quartz) has apparently localized development of the PYXT unit and the attendant high-grade palladium mineralization in the hangingwall portion of the Offset Zone deposit (Figure 7-5). This zone of intensive recrystallization and fabric development is currently interpreted to represent the northern continuation of the regionally important Shelby Lake fault. Three younger brittle-ductile faults also impact the geology of the Offset Zone deposit (Figure 7-6). The most important of these, the Offset fault, is defined as an east-northeast (075°) striking, oblique-slip fault that dips approximately 40° to the northwest. In the underground workings and in drill core, the Offset fault is commonly marked by extensive fault gouge, fracturing and alteration of adjacent country rock and local infilling by fault-parallel mafic dikes. Drilling conducted in 2014 also defined a parallel structure to the Offset fault which is herein referred to as the Camp Lake fault. The Camp Lake fault appears to interrupt the down plunge southward extension of the Offset deposit (Figure 7-6). Drilling planned for 2015 will test for the continuation of the Offset deposit south of the Camp Lake fault. The B2 fault was interpreted from the underground Offset Zone diamond drilling. It lies approximately 20 to 40 m below and parallel to the north-striking and shallow west-dipping Baker fault and is marked by narrow intersections of fault gouge, fracturing, and late mafic dikes. The Offset fault combined with the B2 fault could have been responsible for the displacement of the Offset Zone both down dip (magnitude of displacement not established) and along strike (measured right-lateral displacement of Roby and Offset zones is approximately 300 m). The B2 fault may also have played a role in the localization of Pd mineralization in the recently defined Upper Offset southeast extension zone. The Magnum fault is defined as a right-lateral southeast (130°) trending strike-slip fault that dips approximately 70° to the northeast. The structure occurs below the Offset fault and is marked by narrow intersections of fault gouge, fracturing and late mafic dikes. The Magnum fault was interpreted to influence the stratigraphy of the lithological sequences within the Offset Zone. There is no evidence for major folding of the Offset Zone deposit.

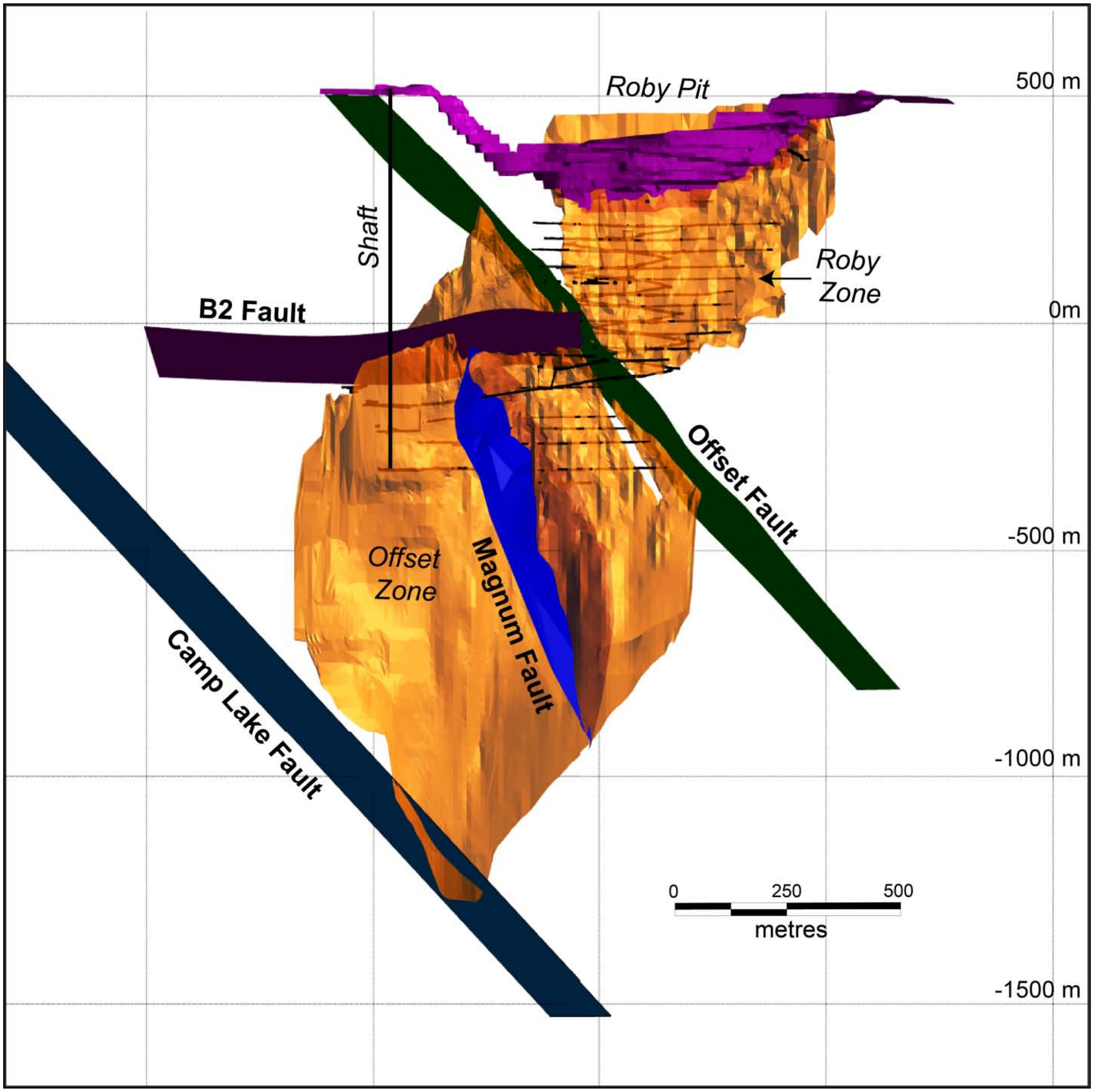


Figure 7-6: Idealized Longitudinal Projection (Looking West) for the Western Mine Block Intrusion Showing the Relative Position of the Offset Zone and Roby Zone Deposits and the Known Major Faults

7.3.2 Rock Types

The principal rock types in and adjacent to the Offset Zone deposit are discussed below. As previously discussed, the use of the term “gabbro” or “gabbroic” is not strictly correct because most of the mafic intrusive units in the MBI are now interpreted to be variably altered and recrystallized products of original noritic or gabbronoritic rock types (Boudreau et al. 2014).

Equigranular Gabbro (EGAB): Historically named “East Gabbro”, EGAB is a well-known gabbro marker unit that is characteristically uniform and compositionally homogeneous. EGAB has no significant associated PGE or copper-nickel sulphide mineralization and bounds the Offset Zone to the east. As such, EGAB is commonly used as a definitive geological hangingwall unit that can be recognized in the underground workings and in drill core. Whole-rock geochemical analyses indicate that EGAB is a more evolved, plagioclase-rich unit than the adjacent mineralized marginal units of the MBI and likely formed from an earlier-emplaced and more fractionated magma.

Varitextured Gabbro (VGAB): VGAB is ubiquitous through the lower-grade, thicker footwall side (west side) of the Offset Zone deposit. VGAB varies from leucocratic gabbro to pyroxenitic, with grain sizes showing erratic variation on all scales from fine to coarse to pegmatitic. The coarser grained units form irregular patches and veins within finer-grained counterparts.

Heterolithic Gabbro Breccia (HGABBX): Despite being a major component of the upper part of the Roby Zone deposit (Figure 7-5), HGABBX has only rarely been definitively recognized within the Offset Zone deposit. Where present, it consists of a melanogabbro to gabbro matrix with variable clast composition, ranging from leucogabbro to pyroxenite.

Pyroxenite (PYXT): PYXT is a steeply dipping, schistose, amphibolitized and chloritized melanocratic noritic unit that is typically situated along the contact between the VGAB and EGAB. It hosts the highest-grade palladium mineralization in both the Offset Zone and Roby Zone deposits and is the major rock type in the hangingwall zones in both deposits. PYXT locally grades into unaltered, unrecrystallized norite that appears to become more prevalent in the southern end of the Offset Zone deposit.

Gabbronorite (GN): Relatively equigranular GN is locally developed between EGAB and PYXT. It is locally host to low-grade palladium mineralization in the Offset Zone deposit, although to a much lesser degree than the VGAB unit.

Mafic to Intermediate Dikes: Late, post-mineralization mafic dikes vary from small (less than 1 m thick), discrete bodies that occupy space within the Roby and Offset Zone resource shells to large bodies (greater than 10 m thick) that control the northern termination of the Offset Zone. A dike swarm approximately 30 m wide and showing an easterly strike and steep dips was mapped at the southern extent of the Roby Zone along the Offset Fault.

Basement Orthogneiss (Tonalite): Recent deep drilling into the basement rocks beyond the western limit to the Offset Zone suggest a very complicated intrusive contact exists in this part of the MBI featuring interleaving of vertically-oriented and sulphide-bearing VGAB, isolated fragments and larger remnants of basement orthogneiss, discrete veins of felsic partial melts of the basement gneiss and associated hybrid rock types. The limited information available for this structurally and petrologically complex western margin to the MBI in the footwall to the Offset Zone deposit suggests that the true limit to the MBI may extend tens to hundreds of metres further west than the previously interpreted.

7.4 Mineralization

7.4.1 General Characteristics

Previous published accounts of the PGE and base metal sulphide mineralization in the MBI include key papers by Macdonald et al. (1987), Sutcliffe et al. (1989), Edgar and Sweeny (1991), Lavigne et al. (2001), Michaud et al. (2003), Hinchey et al. (2005), Lavigne et al. (2005), Gomwe (2008), Barnes and Gomwe (2011), Djon and Barnes (2012) and Boudreau et al. (2014). The results of academic and internal research into the characteristics of PGE-copper-nickel mineralization in the MBI provide the basis for an updated deposit and genetic model. In summary, PGE mineralization in the MBI is found in a variety of structural and geological settings but in general is characterized by the presence of small amounts (less than 1 to 3%) of fine- to medium-grained disseminated iron-copper-nickel sulphides within broadly stratabound zones of PGE and gold enrichment associated with VGAB rocks, coarse-grained noritic rocks and local, intensive zones of amphibolitization, chloritization and shearing. An important, distinguishing characteristic of the MBI mineralization relative to other PGE deposits is the consistently high palladium: platinum ratio, commonly averaging 10:1 or more in most of the known zones. Although many ideas have been advanced to account for the high palladium: platinum ratios, it remains equivocal what specific magmatic and/or post magmatic processes have led to the extreme enrichment in palladium over platinum in the MBI. Sulphide mineralogy is dominated by pyrite with lesser pyrrhotite, chalcopyrite, pentlandite and millerite. A large proportion of the nickel cited in published resources for the Property (see Section 14) appears to be hosted by silicate minerals (chlorite, amphibole). If correct, this explains the low nickel recoveries (generally less than 40%) reported by the LDIM mill.

Nearly all of the known mineralized zones in the MBI occur in the marginal units of the intrusion. In recent years, a much clearer picture of the relationship between previously interpreted disparate mineralized zones has emerged based on over two decades of intensive exploration and development work on the Property. This picture can be summarized as follows:

- The majority of the currently defined palladium-platinum-gold-copper-nickel mineral resources (see Section 14) is restricted to the western end of the MBI and consists of a thicker but lower-grade footwall zone and a thinner, higher-grade hangingwall zone.
- The footwall portion of the Roby Zone has only been mined from the Roby open pit and is typically hosted by VGAB and HGABBX. This style of lower-grade palladium mineralization extends to the full strike and depth extent of both the Offset Zone and Roby Zone deposits and is also the dominant style of mineralization in the Twilight Zone, Sheriff Zone and Baker Zone.
- The higher-grade hangingwall zone in both the Roby and Offset zone deposits is typically restricted to a schistose PYXT unit that is currently interpreted by NAP to represent an intensely hydrated and recrystallized (magmatic fluid alteration and metasomatism, Boudreau et al. 2014) melanocratic norite with schistosity and alteration having been focused along a steeply east-dipping and generally north-striking shear zone.
- The Roby Zone and Offset Zone deposits were likely part of a single contiguous deposit that was subsequently pulled apart by predominantly dextral displacement along the east-northeast striking and north dipping Offset fault.
- Satellite zones of palladium mineralization such as the Moore, Sheriff, and Baker zones appear to represent localized development of similar styles of deformation, recrystallization, and textural and lithological variability that characterize the footwall portions of both the Roby Zone and Offset Zone deposits.
- The North Varitextured (VT) Rim Zone is currently interpreted to represent the along strike extension of the Roby Zone with palladium mineralization becoming telescoped into a less than one- to several-metre thick mineralized VGAB containing discontinuous decimetre- to metre-thick bands of more melanocratic and variably recrystallized norite. Palladium grades in the North VT Rim Zone show much more local-scale variability than is generally observed in the Roby Zone or Offset Zone deposits, including discontinuous bonanza grades of up to 63 g/t palladium over 1 m (see the NAP website for an exploration press release dated September 5th, 2013). The mineralization in part follows and in part is cut by northeast-striking ductile shear zones that are sub-parallel to the strike of the northern margin of the MBI. A distinctive feature of the North VT Rim Zone palladium mineralization is the absence of visible sulphides and very low copper and nickel abundances (see Section 14).

- The primary controls on the localization of economic grade and width palladium mineralization on the Property are currently interpreted as a combination of two contemporaneous and high-temperature magmatic processes, viz.:
    - A volatile-enriched magmatic fluid focused high-grade palladium mineralization and subordinate platinum, gold, copper, and nickel within actively deforming major structures (i.e., magma-fluid pathways); this led to the development of the hangingwall zones of the Roby Zone and Offset Zone deposits that are hosted by schistose and filter-pressed melanocratic norite.
    - Accumulations (rarely exceeding one weight percent) of moderately PGE-enriched magmatic sulphides in feeder faults and within the texturally-heterogeneous marginal units of the MBI (e.g., VGAB-hosted footwall zones of the Roby and Offset deposits).
- The palladium: platinum ratio increases systematically with palladium grade suggesting preferential fixing of palladium relative to platinum in the known mineralized zones and especially in the schistose hangingwall portions of the Roby Zone and Offset Zone deposits. This is apparently a fairly unique, high-temperature feature of the mineralizing processes at LDIM relative to other well-documented PGE (copper, nickel) deposits.
- Well-defined greenschist to amphibolite facies hydration reactions affected all of the mineralized units in the MBI and are believed to be responsible for local redistribution of precious and base metals, but only on the grain-size scale to metre scale. These reactions do not appear to be the primary agents for development of high-grade palladium mineralization in the MBI (Boudreau et al. 2014).

Ongoing data interpretation from outcrop, geophysics, geochemistry and drilling in 2014 have added further details to the understanding of the distribution and controls on PGE mineralization at LDIM. It is currently believed that the majority of the MBI was emplaced in its present configuration prior to the emplacement of the pyroxenite and vari-textured gabbro that host the majority of the known mineral resources. The generally inward dipping MBI stratigraphy was derived from an early pulse of relatively fractionated mafic magma that produced the plagioclase- and magnetite-rich "hangingwall units" to the Roby and Footwall Zone deposits. These units were subsequently intruded by more primitive palladium-bearing magmas that exploited intersections of orthogonal fracture zones. These fracture zones include northerly-trending structures such as the regionally important Shelby Lake fault zone and the east-southeast striking fractures that host late dykes. A current geological working model has the sub-vertical and approximately tabular Roby and Offset hangingwall zones, which have a combined strike length of ~1.1 km and average thickness of 5-20m, as representing the first pulse of a more primitive, pyroxenitic Pd-rich magma into the western MBI between tonalitic basement rocks in the west and the main mass of the MBI in the east.

A generally coeval vari-textured gabbro intrusion, host to the Roby and Offset footwall zones, was emplaced between the tonalitic orthogneiss country rock in the west and the pyroxenitic Roby and Offset hangingwall zones. Pyroxenitic and vari-textured gabbroic magmas were locally complexly intermixed leading to the formation of nested, pipe-like intrusions having variable rock type associations, metal grade characteristics, sizes and shapes. Examples include the triangular cross-sectioned central Offset Zone (thickest part of the Offset Zone deposit), the SW Breccia unit in the mined out Roby open pit, the C-Zone and the Twilight Zone. Additional examples include the Roby Footwall Zone, the Powerline subzone, part of the Sheriff South Zone, and parts of the North VT Rim Zone (Figure 7-4). If this interpretation is correct, it opens up the potential for discovering additional high-grade palladium mineralization along under-explored portions of any of the pre- to syn-magmatic fracture zones mapped on the Property.

PGE grades in the MBI show varying degrees of correlation with nickel and copper concentrations. Many zones display a general moderate to strong positive correlation between palladium, copper, and nickel but others (e.g., North VT Rim Zone) show no significant, positive correlation between these metals. Several applied mineralogical studies of the PGE and base metal sulphide mineralization in the MBI have been conducted. These include academic publications (e.g., Talkington and Watkinson 1984; Sweeny 1989; Lavigne and Michaud 2001; Djon and Barnes 2012) and unpublished NAP reports related to process mineralogy test work conducted at commercial laboratories. The majority of platinum-group minerals in the Roby Zone and Offset Zone deposits occur either interstitially to sulphides in association with gangue minerals such as plagioclase, amphibole, chlorite, orthopyroxene and talc, or at sulphide-silicate boundaries (Yu et al. 2010; Huminicki 2013). At the present time the average relative proportions of the identified PGE minerals in the known mineralized zones in the MBI have not been quantified. However, the relative abundance of PGE-bearing minerals in the mill feed and concentrates from the recently mined zones (Roby, Offset) on the Property are: palladium tellurides > palladium antimonides > palladium sulphides > sperrylite (platinum arsenide) > gold-silver alloys (Yu et al. 2010; Huminicki 2013). Another important characteristic of PGE mineralization in the MBI is the small average grain size at typically less than 10 µm (Yu et al. 2010; Huminicki 2013).

Nomenclature for the many recognized mineralized zones in the MBI has evolved over the more than two decades of systematic exploration on the Property. The currently favoured zone nomenclature includes some simplifications to the internal subdivisions of the main mineralized zones in an attempt to better reflect the most recent resource models and geological interpretations.

7.4.2 Roby Zone

7.4.2.1 Main Roby Zone

The main part of the Roby Zone is a palladium-enriched disseminated sulphide deposit with a strike length of approximately 1,000 m and a maximum width of approximately 800 m. The Roby Zone deposit extends to a minimum depth of 650 m below surface. The deposit includes a thick footwall zone in the west (tens to hundreds of metres wide) and a thinner, approximately 5 to 20 m-thick hangingwall zone in the east. The footwall zone consists of VGAB-±HGBBX-hosted PGE-copper-nickel mineralization with typical palladium grades of approximately 1.0 to 2.5 g/t. The hangingwall zone has typical palladium grades of 3.0 to 10 g/t and is hosted by PYXT and subordinate norite, VGAB, and GN. In addition to having distinctive footwall and hangingwall zones, the Roby Zone deposit also features vertical zonation including:

- A near surface portion that was the principal ore source during mining of the Roby open pit and having a higher proportion of footwall zone and HGBBX.
- An underground portion sharing many similarities to the near surface portion but lacking clear evidence of brecciation in the footwall zone.

The open pit portion of the Roby Zone deposit has been well characterized based on extensive historical drilling, logging, and pit mapping. It consists of three distinct ore types:

- The footwall zone (formerly referred to as the Breccia zone) representing approximately 87% of the deposit volume and having typical grades of 1.5 to 2.5 g/t palladium.
- The hangingwall zone (formerly referred to as the high-grade zone), representing approximately 8% of the deposit volume and having typical grades of approximately 4 to 6 g/t palladium.
- A narrow low- to high-grade North subzone (approximately 5% of volume) likely representing the amalgamation and telescoping of the main footwall and hangingwall zones.

Most of the historical published resources defined within the original Roby pit shell have been mined out. Residual lower-grade palladium mineralization still exists along the western and southern parts of the Roby Zone deposit, beyond the current limits of the Roby open pit (see Section 14).

Underground mining of the Roby Zone commenced in 2006 and has been confined to the higher-grade part of the deposit, the hangingwall zone. By the end of 2012 most of the underground reserves in the Roby hangingwall zone had been mined out. However, most of the lower-grade footwall mineralization beneath the current floor of the Roby open pit is undeveloped.

The underground portion of the Roby hangingwall zone is typically bounded by the barren EGAB to the east and changes abruptly into the VGAB-hosted footwall zone in the west. The hangingwall zone is primarily confined to a 400 m long segment of PYXT although it does extend northward into a gabbronorite unit.

The hangingwall zone, striking north-northwest to north-northeast, dips almost vertically near surface and flattens to nearly 45° at the currently defined bottom of the Roby Zone deposit. The zone is terminated to the south along the Offset fault. The underground portion of the Roby footwall zone is distinguished from equivalent VGAB-hosted mineralization in the open pit portion of the deposit by an apparent lack of significant brecciation.

7.4.2.2 Roby Zone Northeast Extension

Historical and recent exploration and infill drilling on the northern end of the Roby Zone has shown that the Roby hangingwall zone extends, at least intermittently, along an apparent curvilinear trend toward the northeast and possibly connects to the western end of the North VT Rim Zone (see Section 7.4.3). Additional infill and extension drilling will be required to assess whether economic grade-width palladium mineralization is developed along the Roby northeast extension.

7.4.2.3 Roby Zone Southeast Extension

An internal review of results obtained from historical drilling on the southern Roby Zone confirmed that the hangingwall zone extends beyond the southern limit of the historical resource shell until it reaches the Offset fault where it is abruptly terminated. As observed in the northeast extension, high-grade palladium mineralization tends to become narrower and more erratically distributed moving beyond the historical limits to the hangingwall zone resource shell. Nonetheless, the Roby Zone southeast extension remains a potential longer-term development opportunity that will assessed through future engineering studies and economic analyses.

7.4.3 Offset Zone

As previously stated, the Offset Zone is believed to represent the along strike continuation of the Roby Zone, having been displaced from the latter along the east-striking Offset fault. Recent 3D modelling suggests that the northern side of the fault (Roby block) moved down and east relative to the south side of the fault (Offset block) such that the Offset fault is an oblique slip fault with a normal sense of vertical displacement and a dextral sense of lateral displacement. The magnitude of the displacement is estimated to be approximately 200m in the east-west direction and ~50-100m vertically. Drilling completed in 2014 has provided additional constrains on the size and shape of the Offset Zone deposit, particularly below the lower limit of the Phase 1 reserves (i.e., below the 1065 m level in the Offset Mine). Specifically, the down plunge southerly extension of the deposit appears to be truncated by the newly recognized Camp Lake fault (Figure 7-6). The latter structures are currently believed to have a similar orientation and sense and magnitude of lateral displacement as the Offset fault.

This interpretation will be tested as part of the 2015 exploration program. The Offset Zone deposit remains open toward surface, to the south (south of the Camp Lake fault) and to a lesser extent in the lower, northern portion below the 1065 m level and near the Offset fault. The updated Offset Zone mineral resources described in Section 14 of this report comprise a relatively thick footwall zone and a relatively thin hangingwall zone (Figure 7-5). The hangingwall zone in both the Roby Zone and Offset Zone deposits are believed to be exact equivalents and form useful visual marker horizons.

The Offset Zone was previously divided into three grade-and geology-based subzones: the high grade subzone, the mid subzone, and the footwall subzone (McCracken et al. 2013). However, resource modeling completed in 2013 and 2014 (McKinnon et al., 2014 and Section 14, this report) indicates that this subdivision is arbitrary and that from both a mining and geological perspective, the footwall subzone and mid subzone together with the former Cowboy and Outlaw zones are more appropriately considered as a single unit of lower-grade mineralization displaying significant local grade variability that is referred to as the Offset footwall zone.

A similar grade distribution involving a thick envelope of lower-grade PGE and disseminated copper-nickel sulphide mineralization featuring multiple, narrow and discontinuous high-grade PGE “reefs” is present in the Platreef Deposit, South Africa (Kinnaird and MacDonald 2005).

7.4.3.1 Offset Hangingwall Zone

The Offset hangingwall zone is a broadly stratabound unit of higher-grade Pd mineralization developed along the north-striking and steeply (east) dipping contact between the EGAB in the east and the VGAB-hosted footwall zone in the west. Its width varies from 4 to 30 m, with an average of 15 m. Approximately 2% of the Offset hangingwall zone is occupied by late dikes (dilution) and approximately 1% is occupied by shears and faults. The southern part of the Offset hangingwall zone appears to gradually disappear over a distance of 50 to 100 m along strike and immediately to the south of the Offset mine shaft. PYXT is the principal host to the hangingwall zone but portions are hosted by relatively unaltered norite. Underground mining of the Offset hangingwall zone commenced in 2013 (Phase 1) in the upper part of the currently defined resources and reserves (see Sections 14 and 15).

7.4.3.2 Offset Footwall Zone

The lower-grade Offset footwall zone is equivalent to the Roby footwall zone. It is a north-striking mineralized package having typical widths of several tens of metres with its western boundary, a gradational one, being defined by the western limit of a 1 g/t Pd grade shell (see Section 14). It has a sharp contact with the hangingwall zone on its eastern margin. It displays highly variable grades but has average grades in the range 1 to 3 g/t palladium. Local higher-grade intervals (> 3 g/t palladium; e.g., locally present in the former Cowboy and Outlaw zones) have been recognized but appear to lack lateral continuity. VGAB is the dominant host lithology.

The exact western limit to the Offset footwall zone remains poorly constrained. Historical deep drilling into the basement orthogneiss locally intersected VGAB hosted low-grade palladium mineralization over narrow intervals. Additional exploration drilling will be required to adequately define the position and form of the basement-MBI contact and the full extent and quality of palladium mineralization on the footwall side of the Offset Zone deposit.

7.4.3.3 Upper Offset Zone Southeast Extension

Recent exploration drilling (2013, 2014) has shown that palladium mineralization having similar grades to the Offset hangingwall zone is locally present along a postulated southeast extension branch or "pantleg" to the main Offset Zone deposit that was interpreted from a 3D model of the EGAB unit in 2013. Drilling completed in 2014 has defined an east-west striking, moderately dipping zone having widths of 10 to 30 m with its lower and upper boundaries located at the B2 and Offset faults, respectively. The zone has average grades of approximately 3-4 g/t palladium and remains open to the east (see Section 14 for resource estimate). The geology of the Upper Offset southeast extension features footwall stratigraphy that is different to the typical Offset footwall zone. The highest grade palladium mineralization occurs in altered noritic rock just below the EGAB footwall contact.

It is spatially associated with disseminated sulfide mineralization with the dominant sulfides being pyrrhotite and chalcopyrite.

7.4.4 North VT Rim Zone

The North VT Rim Zone is a >2 km long, east- to northeast-striking mineralized zone consisting of sheared and altered VGAB and HGABBX and subordinate, boudinaged melanorite layers, mafic dikes, aplitic to pegmatitic granitic veins and quartz veins. Palladium grades are extremely variable within the North VT Rim Zone and to date no mineral reserves have been declared. Exploration has largely been restricted to trenching and shallow drilling particularly in the westernmost end of the North VT Rim subzone where a small near-surface mineral resource was defined in 2013 (McKinnon et al., 2014). The average grade and width of this initial resource is approximately 2 g/t of palladium and 3 to 8 m, respectively. Palladium mineralization is generally developed within a few metres to tens of metres to the north of the EGAB unit and tens of metres to the south of the northern margin of the MBI. A set of conjugate faults and shear zones has modified locally developed primary magmatic layering within the North VT Rim Zone. Extreme grade variability characterizes most of the documented portions of the zone with grades ranging from less than 1 g/t to 63 g/t palladium. High-grade palladium mineralization is commonly associated with both northwest-striking and east-northeast striking shear zones. In addition, some of the higher-grade assays reported to date are hosted by narrow (less than 1 m thick) melanorite bands. In contrast to the Roby and Offset zones, there is no significant correlation between copper and palladium grades in the North VT Rim Zone. Also, the zone appears to have a much higher proportion of braggite (PdS) that is also much coarser grained (tens to hundreds of microns in length) than the PGMs observed in both the Roby and Offset zones (Huminicki, 2013).

7.4.4.1 Creek Subzone

The Creek subzone is located approximately 2 km northeast of the Roby pit at the eastern end of the MBI, near the contact with the north LDI-IC. Surface trenching has exposed the main portion of the Creek subzone in an area 90 m long by 10 to 40 m wide. It is dominated by low-sulphide breccias that have intruded the VGAB rim of the MBI. The breccias consist of approximately 90% GBNR clasts and only approximately 10% MGAB matrix. Palladium mineralization in the Creek subzone typically occurs within pegmatitic gabbronorite. No resource has been estimated for the Creek subzone but additional exploration and resource definition drilling is planned.

7.4.5 Sheriff Zone

The Sheriff Zone is a combination of the former Southeast Roby, Twilight and South Pit zones connected through additional drilling and with some material recently moved into a NI 43-101 compliant resource (NAP Press Release, September 5, 2013). Definition drilling has identified a large-tonnage, low-grade resource that appears to track along an apparent north-striking structure (not yet recognized from surface mapping) that is potentially amenable to surface mining under a high palladium price environment based on its proximity to the LDIM mill. The Sheriff Zone has been subdivided into North and South zones based on their location relative to the Offset fault.

The Sheriff Zone is generally poorly exposed. It appears to comprise abundant VGAB and local PYXT and HGABBX with the best palladium grades appearing to follow along east-northeast striking shear zones (parallel to the Offset fault) and north-trending faults and high-strain zones.

7.4.5.1 Powerline Subzone

Recent (2013, 2014) surface exploration drilling in the Sheriff North zone has identified an area of atypically high-grade palladium mineralization located within 60 m of surface. This mineralization is referred to as the Powerline subzone that occurs immediately south of the southern edge of the Twilight Zone pit (Figure 7-4). Surface drilling targeting the Powerline subzone returned several intersections of >4 g/t of palladium over core lengths of >20m including an intersection of 18 g/t of Pd over a core length of 12 m in drill hole 14-125 (see NAP press releases dated October 16, 2014 and February 10, 2015). The Powerline subzone occurs in an area with abundant, narrow mafic and intermediate dykes. The highest grade palladium mineralization encountered to date in the Powerline Zone generally occurs in VGAB or PYXT containing trace to minor disseminated sulphide. An initial resource estimate for the Powerline subzone is presented in Section 14.

7.4.6 Baker Zone

The Baker Zone is located approximately 500 m to the south of the North VT Rim Zone in the central part of the MBI (Figure 7-4). Surface exploration has exposed the Baker Zone over a 150 m by 55 m area. The zone consists of a thin (less than 3 m) and relatively flat-lying unit of low-grade PGE-copper-nickel mineralization associated with heterolithic melanogabbro breccia and lesser melanogabbro, leucogabbro breccia, varitextured gabbro, anorthosite, and minor pyroxenite. Typical grades in the Baker Zone range from less than 1 to 2 g/t palladium. Sulphide mineralization appears to have a strong correlation to palladium grades in the Baker Zone and consists of disseminated iron-copper-nickel sulphide mineralization and rare, local, semi-massive sulphide bands. The north-trending, shallow west-dipping Baker fault appears to run sub-parallel to shallow-dipping igneous layering in the Baker Zone but truncates the Baker Zone mineralization at depth suggesting it may have had some control on the localization of the PGE. In 2013, a 10 m intersection of 3.44 g/t palladium was encountered in varitextured gabbroic rocks at approximately 250 m below the surface and directly beneath the Baker Zone (see NAP exploration press release dated September 5, 2013). This relatively high-grade palladium mineralization remains a priority, future exploration target.

7.4.7 Moore Zone

The Moore Zone is a low-grade, presently sub-economic mineralized zone located approximately 500 m south of the current Roby open pit with similar lithologies and textures to other MBI breccias. The central area of interest is a small breccia pod measuring approximately 200 m long and varying from approximately 15 to 115 m wide. This pod occurs within massive, medium-grained gabbronorite that is commonly observed near the southern margin of the MBI.

The highest-grade palladium mineralization thus far encountered in the Moore Zone is located in the eastern portion of the breccia pod where palladium enrichment occurs in a 5 to 25 m wide interval that appears to be structurally controlled trending north-north-easterly and dipping approximately 70° to the east. Significant prior prospecting, mapping, trenching, sampling, and limited diamond drilling of the Moore Zone indicated very limited economic potential near surface. However, recent insights into the relationship between structure and palladium grade have elevated the prospectivity of the Moore Zone and the immediately adjacent area.

8. Deposit Types

8.1 Introduction

Mineralization in the MBI includes several distinctive styles. In summary, these include:

- Roby and Offset zone-style mineralization, comprising:
    - A relatively thick and lower-grade footwall zone comprising minor <1-2% disseminated Fe-Cu-Ni sulphide hosted by VGAB and local heterolithic gabbro breccia.
    - A thinner and generally higher-grade, tabular hangingwall zone typically hosted by a schistose and recrystallized norite to melanorite unit (PYXT) containing <1-2% disseminated Fe-Cu-Ni sulphide.
- North VT Rim Zone-style mineralization, featuring low-sulphide palladium mineralization with local bonanza grades and negligible base metal content that is associated with a narrow (approximately 1.0 to 10m thick), sheared package of VGAB, heterolithic gabbro breccia, mafic dikes and granitic veins featuring highly variable grades on all scales.
- VGAB-hosted and relatively low-grade palladium and base metal sulphide mineralization associated with north-trending faults and shear zones (e.g., Baker Zone, Sheriff Zone).

All of these mineralization styles can be classified as belonging to the magmatic sulphide deposit class. Like lode gold deposits, the palladium mineralization at LDI is now clearly understood as being structurally controlled. However, unlike lode gold deposits, it was volatile-rich noritic magmas and not lower temperature hydrothermal fluids that deposited the PGE, Au, Ni and Cu at LDI.

8.2 Deposit Model

A plausible deposit model for LDI must explain the following critical characteristics:

- The two most important structural trends on the MBI in terms of spatial association with high-grade palladium mineralization are: 1) north-trending and vertically-oriented shear zones representing the northern continuation of the Shelby Lake fault system; and, 2) the Offset fault and sub-parallel east-northeast trending structures including the Camp Lake fault. In the North VT Rim Zone, east-southeast striking and south dipping shear zones and thrust faults appear to have a major control on palladium grade. In addition, there is some evidence that pre-MBI northwest-striking regional-scale faults may also have focused palladium mineralization at LDI.

- The 3D form of the well-defined Roby and Offset Zone deposits features a roughly planar and abrupt hangingwall contact and an irregular and diffuse footwall contact. Deposit thickness in both the Offset and Roby structural blocks shows tremendous variability along strike and down-dip, from <1m to >100m. The footwall contact locally deviates along planar to curvilinear surfaces for up to several tens of metres to the west, producing cross-sectional forms in plan view ranging from crescent-shaped to triangular-shaped. These channel-like features extend for a few metres to tens of metres along strike, tens of metres to several hundred metres down dip and a few tens of metres into the footwall. At LDI the largest channel-like mineralized zones discovered to date include the Central Offset Zone and it's largely mined out equivalent in the Roby Zone. In addition to hosting most of the volume of the Roby and Offset Zone deposits, these channel-like features also contain the best footwall zone grades and possess the highest base metal sulphide contents.
- The Roby and Offset Zone deposits were originally part of a continuous north-striking and steeply east-dipping orebody that was disrupted by post-magmatic deformation to produce the Offset and Roby structural blocks. This dislocation involved oblique-slip displacement along the north-dipping Offset fault such that the Offset block moved up and west relative to the Roby block. A parallel fault zone referred to as the Camp Lake fault may have separated the Offset Zone deposit from an as yet to be defined southern continuation in the poorly documented Camp Lake structural block. In detail, these north-dipping oblique slip faults comprise a series of parallel brittle-ductile fault surfaces having a periodicity of ~200m. Numerous parallel but minor fault planes occur between these regularly-spaced major fault surfaces.
- There were repeated injections of magma into the known mineralized zones. Each injection caused the fragmentation of the rocks already in place.
- Mineralization in the Offset and Roby hangingwall zones occurs in orthopyroxene-rich cumulates (PYXT) having very low amounts of trapped intercumulus liquid. Prior to consolidation, these melanocratic noritic rocks were altered and deformed to produce chlorite-amphibole ± talc schist during high-temperature syn-magmatic compressive deformation and coincident hydrous magmatic fluid alteration. During compression filter pressing expelled most of the trapped liquid that was originally deposited in the mineralized PYXT unit.
- The Roby and Offset footwall zones and most of the other known mineralized zones feature abundant VGAB and locally abundant gabbroic pegmatite that are believed to have crystallized in the presence of a high-temperature magmatic hydrous fluid.
- The main mass of the MBI comprises generally inward- and moderate- to shallow-dipping plagioclase-rich rocks that appear to have formed prior to emplacement of the sub-vertical, mineralized VGAB and PYXT units.
- Pre-concentration of the PGE, Au, Ni and Cu by an immiscible magmatic sulphide liquid can explain much of the observed variation in metal contents in both the Roby Zone and

Offset Zone deposits. However, the partition coefficients of the PGE into sulphide liquid are thought to be similar for all of the PGE (Peach et al. 1990; Fleet et al. 1991). Accordingly, the collection of the PGE by a sulphide liquid is unlikely to explain the extremely high palladium: platinum ratios observed in most of the mineralized zones in the MBI. The palladium: platinum ratio in both the Roby Zone and Offset Zone deposits increases systematically with increasing palladium grades suggesting that these high ratios were developed *in situ* and are a primary characteristic of the local, high-temperature mineralizing process(es) that favoured the nucleation (or stabilization) of Pd-rich minerals relative to Pt-bearing minerals.

- Sulphide mineral assemblages in the known mineralized zones show progressive changes across the strike of the zones (Barnes and Gomwe 2011). Sulphide abundances are typically 1-2% in the main palladium-rich mineralized zones on the MBI but do reach much higher values as observed in local net-textured and semi-massive sulphide occurrences that generally do not exceed a metre in thickness.

Recent deposit models for the MBI palladium mineralization presented in Barnes and Gomwe (2011) and discussed in McCracken et al. (2013) explain most, but not all, of these characteristics. Lingering uncertainties include:

- Relative timing of the formation of the main magma-fluid pathways (ductile to brittle shear zones and faults) and the emplacement of the parental magmas to the mineralized zones in the MBI.
- Genetic relationship between the compositionally distinctive magma pulses recognized in the known mineralized zones, each with its unique physicochemical and mineralogical characteristics and propensity for hosting high-grade palladium mineralization.
- Specific factors controlling the localization of base metal-enriched magmatic sulphides.
- Processes responsible for the ubiquitous, distinctive selective enrichment in palladium over platinum.

NAP geologists have recently revised the working deposit model for the MBI, which can be summarized as follows:

- Emplacement of the lopolith-shaped main mass of the MBI at a regional structural intersection between the north-striking Shelby Lake fault and northwest-striking faults concurrent with the development of similar gabbroic units in the other LDI suite intrusions.
- Cooling and crystallization of generally unmineralized gabbroic to ferrogabbro/diorite units of the main mass of the MBI.
- Later sub-vertical injection of more primitive sulphide-bearing and PGE-enriched noritic magmas along the N-S striking Shelby Lake fault zone.
- In the western MBI, the Roby and Offset Zone deposits developed along one branch of the north-striking and sub-vertical Shelby Lake fault extension.

- The orebody thickness in the Roby and Offset zones reaches a maximum within vertically-oriented magmatic conduits that appear to have developed at the intersection point of two to three faults and/or shear zones.
- These mineralized conduits developed a planar hangingwall contact and an irregular footwall contact as a consequence of magma flow focused within originally triangular-shaped fault intersections.
- The final form of the conduits was dictated by flow-related scouring and erosion along the footwall contact to produce channel-like high-grade zones having limited strike length but greater down-plunge extent.
- The controlling structures are magma feeder faults and the conduits likely developed in areas of maximum dilation (structural intersections) where magma flow may have been reduced, thereby allowing sulphides and cumulus orthopyroxene to co-accumulate.
- The hangingwall zone PYXT unit formed in response to magmatic fluid-enhanced chemical exchange between the consolidated main mass of the MBI and the metal-rich late noritic magmas through high-temperature metasomatic reactions.
- Syn-depositional compressional deformation promoted fabric development and filter pressing of trapped intercumulus liquid to further modify the texture, mineralogy and geochemistry of the hangingwall zone PYXT.
- The footwall zone formed by repeated injections of sulphide-bearing, metal-rich and volatile-rich noritic magma that generated the widespread magmatic breccia and varitextured gabbro units developed along the western and northern margin of the MBI.
- Much of the South LDI appears to have formed along the same north-south structure that fed the Roby and Offset Zone deposits.
- High-temperature magmatic fluids likely had some effect on the final distribution of PGE and base metals and on the ore and gangue mineralogy by promoting hydration reactions involving orthopyroxene and some of the original plagioclase – these reactions are best preserved in the PYXT unit.
- The final position of high-grade palladium mineralization on the Property was significantly impacted by oblique slip movement on the north-dipping and east-northeast striking Offset and Camp Lake fault system (e.g., in the Offset and Roby zone deposits) and on east-northeast striking south-dipping shear zones (e.g., in the North VT Rim zone).

Despite evidence for locally pervasive high-temperature hydration reactions there is no compelling evidence for magmatic fluids as being the primary agent for high-grade palladium mineralization on the Property. As previously discussed, the weight of evidence from all historical assays (core, surface and underground sampling) shows a strong correlation between palladium grade and both nickel and copper contents that is consistent with a critical role for magmatic sulfide in localizing palladium in the MBI.

Furthermore, the extreme Pd: Pt ratio (locally >20:1 and typically >10:1) is a pervasive and unique feature in the MBI such that localized magmatic fluid-related alteration reactions as seen in the PYXT unit are unlikely to explain the strong palladium enrichment evident in all of the mineralized zones. The current working model for the LDI palladium deposits will continue to be refined as new geological, geochemical and mineralogical evidence comes to light. The coincidence of strike-limited, vertically-oriented, large tonnage palladium-rich magma conduits along regionally extensive north-trending vertical fault zones is an important guideline for future exploration at LDI and on coeval intrusions on the Company's greenfields properties. It is not known if local deflections of the generally north-striking high-grade palladium mineralization in the MBI along east to southeast-striking sub-vertical faults (e.g., Powerline Zone, Upper Offset southeast-extension) reflects syn-magmatic or post-magmatic processes. It is recommended that all pre- to syn-magmatic faults in the region and especially vertically-oriented faults be considered as potential feeder structures capable of localizing high-grade palladium mineralization.

8.3 Analogous Deposits

The Roby Zone and Offset Zone deposits share many similarities with relatively thick contact-type or marginal series-related PGE-copper-nickel deposits (Iljina and Lee 2005). They share fewer but still important similarities with much narrower, stratiform, reef-type PGE deposits associated with noritic magmas including the Merensky Reek in the Bushveld Complex, South Africa (Cawthorn 2005) and the JM Reef in the Stillwater Complex, Montana (Todd et al., 1982).

The closest known analogue for the western MBI palladium deposits is the Platreef deposit in the northern lobe of the Bushveld Complex (Kinnaird and McDonald, 2005) that falls into the general class of contact-type PGE deposits. A large number of common geological, mineralogical and geochemical features can be cited from both the MBI deposits and the Platreef deposit, including:

- Similarity in total PGE + gold grades and copper + nickel sulfide abundances.
- Significant width of lower-grade subzones (tens to hundreds of metres).

- Presence of multiple, narrower higher-grade subzones – in the Platreef these are referred to as the A, B and C reefs; in the Roby and Offset zones, they include the hangingwall zone and as well as the former Cowboy and Outlaw zones (now included in the footwall zone).
- General stratabound nature with adherence of the mineralized package to an irregular footwall contact with the local basement rocks and a sharp, upper contact with a massive to foliate more fractionated gabbroic unit (Main Zone gabbro in the Platreef; EGAB in the MBI) that locally grades into magnetite gabbro.
- Presence of abundant heterolithic breccia, contact metamorphic aureoles, zones of partial melting of the basement rocks and development of hybrid rock types interpreted as the product of mixing of basement-derived partial melts and parental mafic magmas.
- Preponderance of cumulus orthopyroxene and lower-temperature cumulus to intercumulus plagioclase.
- Strong evidence for syn-magmatic deformation and channeling of hydrous magmatic fluids through the orebodies with concomitant recrystallization of original gangue mineralogy, PGM and sulphide minerals in areas of intensive recrystallization and metasomatism.

As is the case with the Platreef, mineralization at LDI appears to be focused on one or more regional-scale structures that acted as feeders to the MBI including the late pulses of fertile, noritic magma that produced all of the known zones of palladium mineralization.

9. Exploration

The LDIM Property has had more than 50 years of exploration by former explorers and LDIM. The following sub-sections briefly describe the nature and extent of all relevant exploration work other than drilling (see Section 10) conducted by or on behalf of LDIM.

9.1 Research, Compilation, Review, Recording

LDIM has an Exploration Department that operates from a Thunder Bay Office and facilities on the mine site. Historical data from decades of exploration programs is stored in filing cabinets, map cabinets and digitally on a network server and company computers. A master database, securely maintained on the server, records all drill hole, surface sample and trench sample metadata and analytical results from exploration programs. Data is flagged as non-compliant for NI43-101 purposes when historical records are not to standard or there are data quality issues that are unresolved. Geospatially located data is commonly reviewed by company geoscientists using a MapInfo-based GIS with 3D capabilities. Various software is used to review different types of geochemical data.

9.2 Surface Mapping, Trenching, Sampling

Direct geological observations of outcrops on the LDIM property is an extension of the prospecting that led to the discovery of palladium occurrences in the area over half a century ago. Initial prospecting samples were submitted for gold analysis only, but the assayer recognized the potential for platinum group metals by the color of the assay bead. The prospectors heeded the assayer's advice and the subsequent assay results indeed demonstrated the occurrence of palladium and platinum. It has been advised repeatedly over the years, by all geoscientists familiar with the LDIM property, that elevated and anomalous palladium commonly occur around the Property in rocks that bear no indication of being mineralized. Grab samples; saw cut samples and sawn channel samples have all been used to geochemically characterize outcrop exposures. Where favourable results have been found in areas overlain by thin overburden, excavators have been used to remove it to "make" new outcrops. Hydraulic outcrop washing and subsequent sampling precisely located by mapping and/or GPS have extended mineralization beyond pit limits. It is not surprising that palladium, platinum, gold, copper and nickel results are the most relevant geochemical data from surface sampling.

Since 1998, Lac des Iles' Exploration Department, working in a 6km by 6 km area surrounding the LDIM (NAD83 UTM Z16 5445000N to 5451000N, 307000E to 313000E), has collected 21,802 precisely located surface samples. These were mainly analysed using commercial multi-element packages. All 21,802 samples have palladium; platinum and gold results and 16,436 samples also have copper and nickel results. Figure 9-1 illustrates the location of these samples which are concentrated on the Mine Block Intrusion and within mine leases CLM251, CLM252 and CLM253.

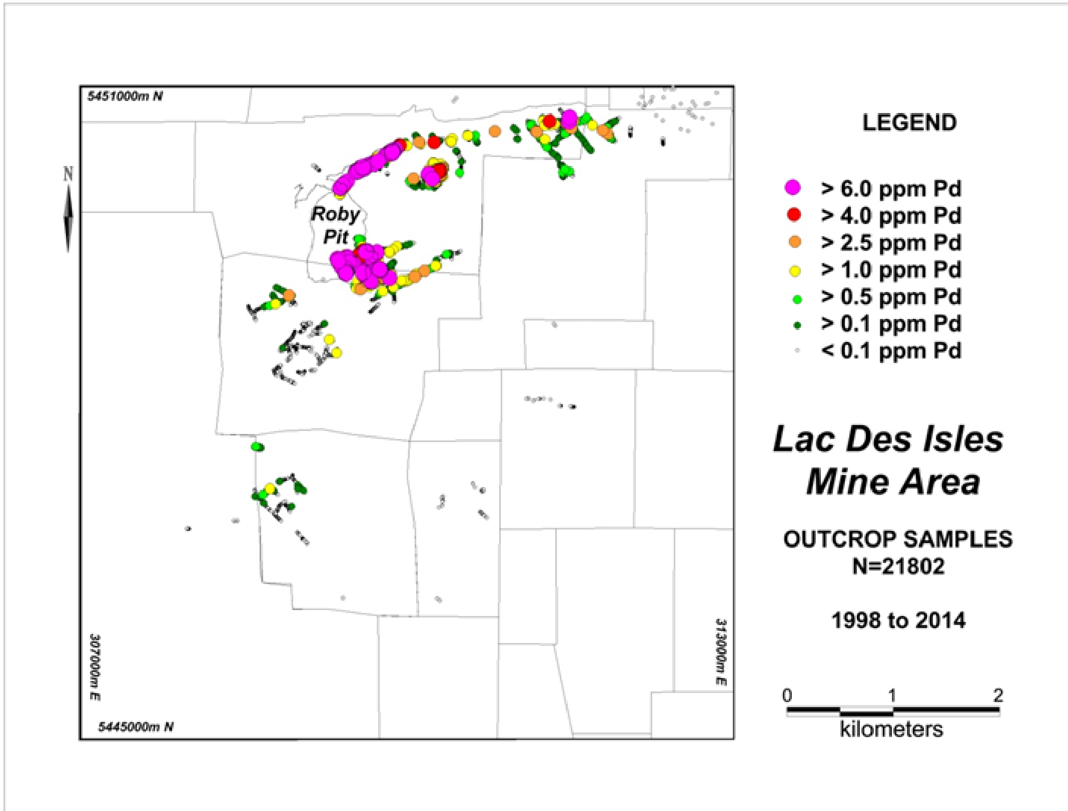


Figure 9-1: Palladium Results In Outcrop Samples in the Lac Des Isles Mine Area

High-grade palladium samples (>4ppm Pd) occur in vari-textured gabbro around the Mine Block Intrusion along the North VT Rim and South VT Rim and centrally at the Baker Zone. Elevated palladium occurs south and southwest of Roby Pit in noritic intrusions that were mapped by Anaconda in the 1960's and by Texasgulf in the 1970's as containing pyrrhotite and chalcopyrite disseminations. Researching sources of old data and digitally reviewing it with what is known now provides accelerated testing and implementation of new targeting ideas. In this case it is invaluable as mine tailings covers some of these outcrops requiring geological modelling, geophysical surveys and targeted drilling to fully advance these targets.

9.3 Integration of Blasthole Sampling, Geophysical and Fracture Surveys

The LDIM Exploration Department is using historic blasthole palladium geochemical data from the Roby Open pit to define and characterize Pd-enriched geological units within the orebody. This provides excellent location and analytical accuracy and precision that is commonly lost in ore deposit summaries. At present the Company is reviewing the blasthole data on 25 m levels to derive patterns that can be applied to locating these nearer surface zones below the Offset Fault.

A significant observation in 2014 was the Roby High-Grade (now referred to as the Roby hangingwall zone) and Roby Central zones seen in the Roby Block above the Offset Fault have a similar pattern as the Offset Hangingwall and Central zones in the Offset Block below the Offset Fault.

Another significant observation in 2014 was the highest blasthole Pd values on the 490 m level (about 10 m below surface) in the Roby High Grade and Central Zones matches an elevated magnetic pattern in historical ground Total Field Magnetics data (Figure 9-2). This observation is consistent with magnetic susceptibility results consistently taken at one metre intervals from the 2014 drilling program and confirms that certain magnetic responses may be related to palladium-enriched intrusions.

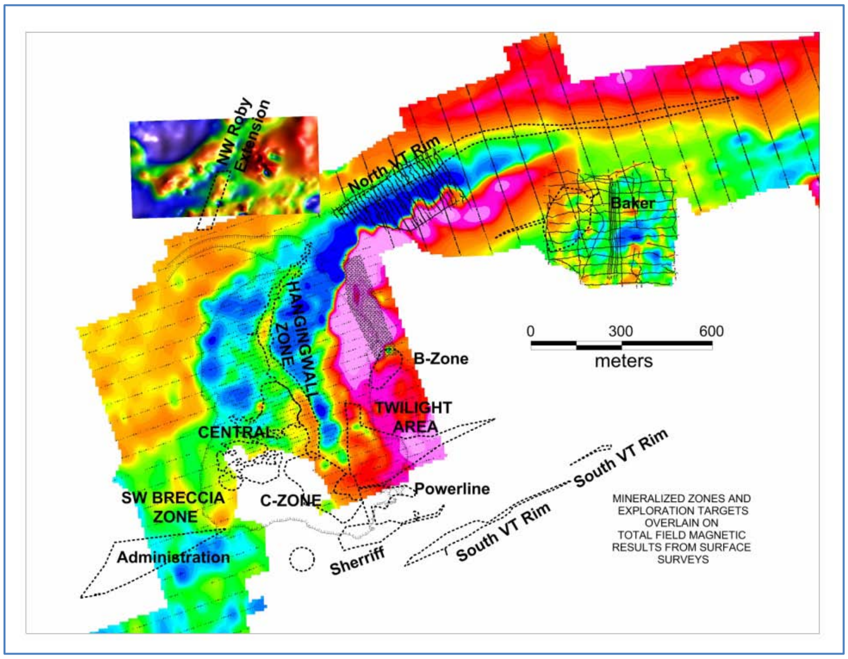


Figure 9-2: Mineralized Zones and Exploration Targets Overlain on Total Magnetic Field Results from Surface Surveys

Accordingly, magnetic properties can now be used with confidence as a direct means to prioritize exploration targets in favourable geological and geochemical settings. This extends to subsurface exploration where in hole magnetic susceptibility data and downhole magnetic field strength, bearing and inclination survey data can be used to vector to offhole magnetic anomalies. Airborne and ground geophysical surveys on the property include magnetometer, EM, IP, VLF and magnetotelluric surveys.

Periodic comparison of old and recent geophysical surveys with current geological models has resulted in refinements to structural geological interpretations of faults including revised estimates of the magnitude and sense of the displacement. This directly assists targeting poorly documented areas below a fault by applying the estimated displacement from a known mineralized zone above that fault.

As discussed in Section 8, this approach is being use to predict the position of the modeled down-plunge continuation of the Lower-Central Offset Zone.

Targeted surveying of fractures in the Roby Pit walls is being integrated into 3D geological models that characterize the minor displacements to mineralized zones that we see in the Roby block model. Use of drill log descriptions of fracture zones and Rock Quality Designation (RQD) measurements also help identify the major structures that regularly offset mineralization at LDI. Combining geological, geophysical and geochemical results has resulted in the current exploration targets on the Mine Block property shown on Figure 9-3.

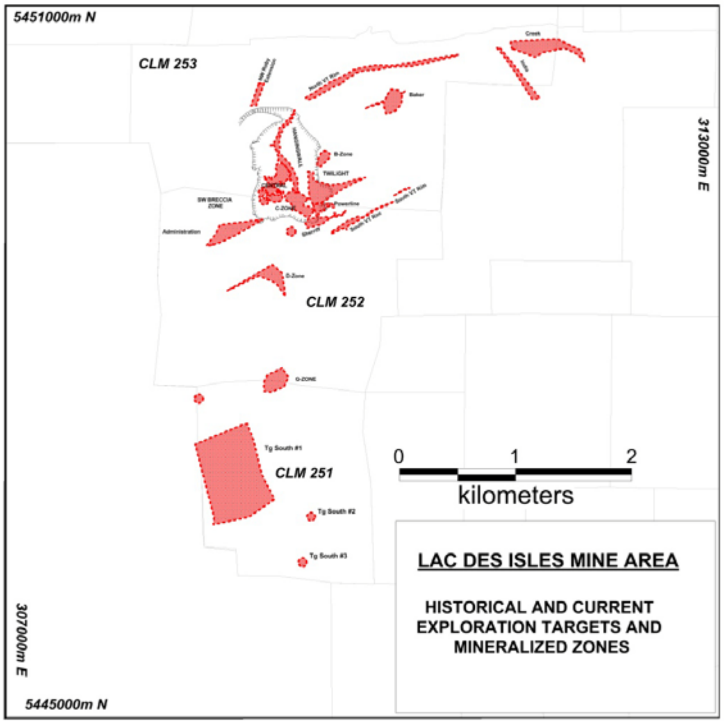


Figure 9-3: Priority Surface Exploration Targets on the Lac des Iles Mining Property

10. Drilling

10.1 Historical Database

The LDIM Exploration Department database contains information on all known surface and underground diamond drilling on the Property. The current database content is summarized in Table 10-1. Drilling statistics are not included here for other Company controlled LDIM Suite intrusions (see Section 7 and 23). Prior owners drilled 143 holes totalling 22,862 m on the Property between 1964 and 1976. There was no drilling from 1977 to 1985.

10.2 North American Palladium Ltd

North American Palladium Ltd. and its predecessor and subsidiary companies have drilled 2,100 holes totaling 741,930 m on the LDIM property since 1986 (Table 10-1).

10.2.1 Surveying

10.2.1.1 Collar Surveys

Diamond drillhole collars use mine grid coordinates and bearings. The LDIM grid north bearing is at a True North azimuth of 359.725 degrees.

10.2.1.2 Downhole Surveys

Since 1995, all holes have been routinely surveyed along-the-hole using a variety of tools. These have included magnetic-based tools (Tropari, Reflex™ Multishots), non-magnetic light logging tools (Reflex™ Maxibor), north-seeking gyros (Halliburton™) and incremental gyros (Icefield Tools, Reflex™, Halliburton™).

Table 10-1: Lac des Iles Mine Area Drilling Summary

Year	# Holes	Metres	Company	Hole ID From	Hole ID To
1964	11	1,516	Gunnex	G64-1	G64-11
1966	14	2,114	Anaconda	A66-12	A66-25
1974	34	4,586	Boston Bay Mines	P001	P034
1975	72	12,563	Texasgulf Inc.	P035	P107
1976	12	2,083	Texasgulf Inc.	P108	P119
1986	29	9,169	Madeleine Mines Ltd	86-01	86-34
1987	22	4,424	Madeleine Mines Ltd	87-01	87-51
1988	8	1,452	Madeleine Mines Ltd	88-1	88-8
1989	4	609	Madeleine Mines Ltd	89-1	89-4
1992	22	1,177	Lac des Iles Mines	92-01	92-22
1995	56	7,802	Lac des Iles Mines	95-01	95-57
1997	19	4,243	Lac des Iles Mines	97-01	97-19
1998	51	7,591	Lac des Iles Mines	98-001	98-053
1999	179	52,464	Lac des Iles Mines	99-001	99-191
2000	251	117,994	Lac des Iles Mines	00-001	00-339
2001	36	26,792	Lac des Iles Mines	01-001	01-086
2002	82	46,966	Lac des Iles Mines	00-149W	02-094
2003	25	10,211	Lac des Iles Mines	03-001	03-029
2004	7	3,276	Lac des Iles Mines	04-001	04-004C
2005	39	22,308	Lac des Iles Mines	05-001	05-052
2006	9	5,720	Lac des Iles Mines	05-006W1	05-016W4
2007	47	24,253	Lac des Iles Mines	07-001	07-052
2008	30	17,200	Lac des Iles Mines	07-053	08-108
2009	135	54,916	Lac des Iles Mines	09-001	09-956
2010	225	78,830	Lac des Iles Mines	09-957	10-999
2011	263	85,001	Lac des Iles Mines	10-510	UG050
2012	229	50,191	Lac des Iles Mines	11-249	12-850
2013	259	73,069	Lac des Iles Mines	13-001	14-906
2014	73	36,272	Lac des Iles Mines	14-08-001W1	14-974
TOTAL	2,243	764,792			

10.2.2 Core Delivery and Chain of Custody

Core boxes are delivered from the drill site to the core logging facilities on the Property by the drill contractor as soon as practicable at the end of each shift. For drill core and its samples, a continuous chain of custody tracking system is maintained from receipt of sealed core boxes from the drillers through logging/sampling, sample shipping, analysis, reject and pulp storage at the lab, reject and pulp return delivery to LDIM and finally core and sample storage on the Property.

10.2.3 Core Preparation

These are the standardized steps used to prepare and verify drill core prior to geological logging and sampling:

1. Core boxes are loaded on logging benches in numerical order for both box numbers and metre tags.
2. Core is manually tightly packed by removing gaps between pieces and checking for misplaced core. China markers are used to make fiducial marks every metre consistent with the driller's tags that nominally mark the end of each 3 m run. Drillers measure along-the-hole distance from the face in underground holes and from the top of casing at surface.
3. Foliated rocks are turned and tight-packed to present the maximum core axis angle for logging and the photographic record. Tightly packed core is fundamental to ensure the same half of the core is sampled between core pieces.
4. Core recovery, fracture density per metre or rock quality measurements (a percentage of interval with core pieces greater than 10 cm), and box "from" and "to" intervals are recorded. Aluminum labels are stapled to core box ends for long-term identification in core racks or pallet storage.
5. Magnetic susceptibility readings every metre provide data used to consider logging unit boundaries and Multishot survey data quality.
6. Specific gravity measurements are recorded for a selection of rock types from drillholes by the logging geologist or technician.

10.2.4 Core Logging

Since 2008, the Company's Exploration Office in Thunder Bay, Ontario has used the Fusion Data Management (Fusion) suite of software by CAE Mining to manage borehole data including analytical results. Geologists and technicians enter collar coordinates, downhole survey data, geological descriptions, rock quality designation (RQD), magnetic susceptibility, specific gravity and lists for sample analysis. Commercial labs report analytical results digitally in a format that is imported directly into the Fusion software. Digital copies of analytical result certificates are catalogued by certificate number on the exploration file server.

As the core is being prepared (as described in Section 10.2.3), a geologist reviews the core to affirm drilling quality and compare general lithologies encountered to the proposed hole's objectives. Actionable items are then addressed in a timely manner with the Senior Geologist.

Currently, available core is systematically logged by a geologist recording lithology, alteration, mineralization, veining and structure. Observations are recorded directly onto a laptop computer that is regularly uploaded to the database server.

10.2.5 Sample Layout

Following logging, samples are laid out incorporating unique and sequentially numbered identifiers. Sequentially embedded control samples including blank siliceous dolomite coarse reject and commercial standards are set within each batch. About 5% of all half-core samples are sawn into two quarter core duplicates. A mass balanced average assay for the half-core duplicate interval is calculated by the software.

Sample lengths depend on lithologic contacts but the most common sample has a 1 m sample length starting and ending on the metre-spaced fiducial marks. In rare cases, such as massive sulphide intersections, a 20 cm minimum sample length may be used. Sample intervals greater than 1 m are generally restricted to rare sections with poor recovery and in these cases should comprise up to 1 m of re-composited core. Sample limits are adjusted off the "integer metre marks" to coincide with lithological boundaries (i.e., late dikes, major units such as sulphide-rich material, pyroxenite, equigranular gabbro, magnetite gabbro, tonalite, etc.). Sample start and ends are clearly marked on the core. The last three digits of the unique identifier are written on the core at the sample start and two tags are inserted at the end of each sample. One tag goes into the labelled plastic sample bag when the core is sawn and the other is stapled to the box bottom at the end of the sample. Control sample tags are also inserted at the end of the sample they numerically follow and one tag goes into the plastic bag with the control sample and the second tag is stapled in the box.

10.2.6 Core Photographs

Following sample layout, each set of four to five core boxes are photographed on the core benches and the image named using the hole name and depth. Core is commonly photographed dry then again with a wet surface. Core photos are loaded to the server by a geologist or technician.

10.2.7 Core Sampling

From the logging benches, core is transferred to pallets, strapped for secure transport, and then moved to a core sawing facility on site where it is offloaded onto core racks. A sample list is prepared and printed for use in continuous monitoring of sample progress and proper incorporation of control samples.

One box of core is loaded onto a bench adjacent to a Vancon core saw. The half-core retained is tightly packed in the core box retaining the original sequence. The half-core for analysis is placed in a new clear plastic sample bag which has been previously labeled with a machine-printed tag bearing the unique sample identifier. One machine-printed sample tag placed at the end of the sample during sample layout is verified against the sample bag number and placed inside of the bag with the core sample. The other tag is placed in the bottom of the channel in the core box at the end of the sample and covered by a piece of split core. Once all the samples have been split from a core box; it is returned to the core rack and the next box is loaded onto the splitter bench.

Control samples without source identification are inserted into the same new clear plastic sample bags having a machine-printed tag bearing the unique sample identifier and added to the rice bags in numerical order. The commercial supplier's identification label is peeled off the Kraft paper bag containing the standard or blank pulp and this label affixed to the sample list next to its corresponding sample number for QA/QC assurance (see Section 11 for more information on QA/QC procedures).

Each core sample or control sample bag is sealed with a plastic tie strap and placed in hand-labelled rice bags for efficient storage, shipping and handling. Rice bags are sealed using zip-ties. Saws and water recirculation tanks are cleaned regularly.

Core boxes of half core are transferred from the sawing facility core rack to pallets and strapped for transport and short- or long-term storage. Core is kept outdoors on strapped pallets or in outdoor covered racks for future reference.

While on site, all rice bagged samples are stored under cover until loaded onto NAP pickup trucks and delivered directly to the Commercial Lab's prep facility in Thunder Bay, Ontario by LDIM personnel.

10.2.8 Sample Shipping

Drill core is currently sent for analysis in batches of 78 samples containing certified standards, duplicates and blanks.

Current shipping protocol for core samples in rice bags is to be loaded onto NAP pickup trucks and delivered directly to the Commercial Labs prep facility in Thunder Bay, Ontario by NAP personnel.

The lab stores rejects in sample bags within rice bags on plastic wrapped pallets. The lab stores pulps in Kraft paper bags in cardboard boxes in totes. Periodically, rejects or pulps are returned to NAP for storage in unheated sea shipping containers at the mine site.

11. Sample Preparation, Analyses and Security

11.1 Sample Preparation, Analyses and Security 2003 to 2008

For 2003 to mid-2008, drillhole samples were prepared and analyzed by Accurassay Laboratories (Accurassay), a division of Assay Laboratory Services Inc., in Thunder Bay, Ontario. Accurassay is accredited by the Standards Council of Canada (SCC) under CAN P-4E, the International Organization for Standardization (ISO), International Electrotechnical Commission (IEC) 17025 and CAN-P-1579 guidelines for PGE, copper, nickel, and cobalt analysis by atomic absorption spectroscopy (www.accurassay.com). Core samples were secured in the logging/sampling geology facility at the mine site. The mine itself has a gate house and barriers to restrict public access. Samples in the NAP Thunder Bay office and core facility are secured indoors. Core samples were trucked by exploration staff or by Courtesy Courier to Accurassay in Thunder Bay.

11.1.1 Sample Preparation

The sample preparation and assay procedures used by Accurassay are as follows:

- Core sample numbers are entered into the local laboratory information management system (LIMS).
- Samples are dried, if necessary.
- Samples are jaw crushed to -8 mesh (2.36 mm).
- A 250 to 400 g cut is taken by riffle splitting, with the balance stored as coarse reject.
- The cut is plate pulverized to 90% -150 mesh (106 µm) and then matted to ensure homogeneity.
- Silica sand is used to clean out the pulverizing dishes between each sample to prevent cross-contamination.

11.1.2 Sample Analyses

For precious metal assay, a one-assay ton pulp split (±30 g) is mixed with a lead-based flux and fused in a muffle oven. The resulting lead button is placed in a cupelling furnace where all of the lead is absorbed by the cupel, and a silver bead, which contains any gold, platinum, and palladium, is left in the cupel. Once the cupel has been removed from the furnace and cooled, the silver bead is placed in a small, labelled test tube and digested using a 1:3 ratio of nitric acid to hydrochloric acid. The samples are bulked up with 1.0 mL of distilled de-ionized water and 1.0 mL of 1% digested lanthanum solution for a total volume of 3.0 mL. The solution is cooled and vortexed and then allowed to settle. Analysis for gold, platinum, and palladium is then completed using atomic absorption. The atomic absorption unit is calibrated for each element using the appropriate ISO 9002 certified standards in an air-acetylene flame.

For a base metal assay, pulps are digested using a multi-acid digest (nitric acid, hydrogen fluoride, hydrochloric acid). The samples are bulked up with 2.0 mL of hydrochloric acid and brought to a final volume of 10 mL with distilled de-ionized water.

The samples are vortexed and allowed to settle and then analyzed for copper, nickel, and cobalt using atomic absorption. The atomic absorption results are checked by the technician, forwarded to data entry by electronic transfer, and a certificate is produced. The Laboratory Manager checks the data and validates them if they are error-free. The results are then forwarded to LDIM by email and hardcopy in the mail.

11.2 Sample Preparation, Analyses and Security 2009

For the 2009 series drilling, NAP used ActLabs of Thunder Bay, Ontario for core sample preparation and analysis. ActLabs is accredited ISO/IEC 17025 and CAN-P-1579 (www.actlabs.com). Core samples were secured in the logging/sampling geology facility at the mine site and in a secure administration building/core facility in Thunder Bay. Core samples were trucked by Courtesy Courier to ActLabs in Thunder Bay.

11.2.1 Sample Preparation

Sample preparation at ActLabs was as follows:

- Drying at 60°C.
- Size reduction in a TM Engineering Ltd. Terminator jaw crusher to 90% passing - 8 mesh (2 mm).
- Riffle split to 250 g.
- Grind in a TM Engineering Ltd. Max 2 pulveriser with mild steel bowls to 95% passing - 150 mesh (105 µm).

11.2.2 Sample Analyses

11.2.2.1 Palladium-Platinum-Gold

ActLabs used a standard fire assay fusion on a 30 g aliquot with silver inquart. Furnace fusion is for 60 minutes at 850 to 1,060°C. The resulting lead button is cupelled at 950°C and the resulting silver-doré bead is digested in hot nitric acid and hydrochloric acid, cooled, and the solution analyzed by inductively coupled plasma (ICP)-optical emission spectrometry (OES).

11.2.2.2 Nickel-Copper-Cobalt-Silver

ActLabs used a four acid, near total, digestion and ICP-OES finish.

11.3 Sample Preparation, Analyses and Security 2011 to 2013

For the 2011 through 2013 series drilling, NAP used ActLabs of Thunder Bay, Ontario for core sample preparation and analysis.

11.3.1 Sample Preparation

- Sample preparation at ActLabs was as follows:
  - Samples are sorted, opened, and dried at 60°C.
  - Logged into LIMS database.
  - Samples are crushed to 80% -2 mm.
  - Samples are split to 250 g using a Jones Riffle.
  - That 250 g portion is pulverized to 95% -100 µm.
  - There are pulp duplicates taken every 30 samples.
  - There are fine crush duplicates taken every 50 samples.
  - Crusher jaws and work stations are cleaned between each sample with compressed air.
  - The grinding bowls are cleaned between each sample with abrasive cleaning sand and then cleaned with compressed air.
  - Once QA/QC checks were completed the pulps and rejects were then shipped back to the mine where they were stored in sea-can shipping containers.

11.3.2 Sample Analyses

11.3.2.1 Palladium-Platinum-Gold (ActLabs Schedule of Services 1C-OES Fire Assay Analysis)

ActLabs used a standard fire assay fusion on a 30 g aliquot with silver inquart. Furnace fusion is for 60 minutes at 850 to 1,060°C. The resulting lead button is cupelled at 950°C and the resulting silver-doré bead is digested in hot nitric acid and hydrochloric acid, cooled, and the solution analyzed by ICP-OES using a Varian 735 ICP. A blank and digested standard are run every 15 samples. Synthetically prepped standards monitor QC and instrument drift.

11.3.2.2 Nickel-Copper-Cobalt-Silver and Multi-Element (ActLabs Schedule of Services 1F2 - Total Digestion ICP Analysis)

A 0.25 g sample is digested with four acids (hydrogen fluoride, nitric acid, perchloric acid, and hydrochloric acid) and heated in several ramping and holding cycles taking the samples to incipient dryness after which samples are brought back into solution using aqua regia. Samples are analyzed using a Varian ICP. Each digestion batch has 14% QC including 5 method reagent blanks, 10 in-house controls, 10 sample duplicates, and 8 certified reference materials. Instrumental analysis undergoes and additional 13% QC to ensure control of instrument drift.

11.3.2.3 *Whole Rock and Oxide Analytes - 4C - XRF Fusion – XRF*

To minimize the matrix effects of the samples, the heavy absorber fusion technique of Norrish and Hutton (1969) are used for major element (oxide) analysis. Prior to fusion, the loss-on-ignition (LOI), which includes water (H_2O) + carbon dioxide (CO_2), sulphur, and other volatiles can be determined from the weight loss after roasting the sample at 1,050°C for two hours. The fusion disk is made by mixing a 0.5 g equivalent of the roasted sample with 6.5 g of a combination of lithium metaborate and lithium tetraborate with lithium bromide as a releasing agent. Samples are fused in platinum crucibles using an automated crucible fluxer and automatically poured into platinum molds for casting. Samples are analyzed on a Panalytical Axios Advanced wavelength dispersive x-ray fluorescence (XRF) unit.

The intensities are then measured and the concentrations are calculated against the standard G-16 provided by Dr. K. Norrish of the Commonwealth Scientific and Industrial Research Organisation (CSIRO), Australia. Matrix corrections were done by using the oxide alpha; influence coefficients provided also by K. Norrish. In general, the limit of detection is about 0.01 wt% for most of the elements.

11.4 Sample Preparation, Analyses and Security 2014

All of the 2014 LDIM program core samples were processed and secured in the Exploration Department’s logging and core sawing facilities at the LDIM mine site. Selected drill core intervals were sawn in half using Vancon core saws. One half of the sample was submitted for analysis while the other half sample was returned to the corebox. After completion of sampling, all coreboxes were palletized, band strapped and stored on the LDIM site. The samples for analysis were packaged in rice bags with the tops secured by zip ties. For the 2014 series drilling, NAP used ALS Minerals (ALS) which has preparation facilities in Thunder Bay, Ontario and analytical facilities in North Vancouver, British Columbia. The North American ALS analytical laboratories are accredited by the Standards Council of Canada (SCC) for specific tests listed in their Scopes of Accreditation which conforms with CAN-P-1579: Requirements for the Accreditation of Mineral Analysis Testing Laboratories and CAN-P-4E ISO/IEC 17025: General Requirements for the Competence of Testing and Calibration Laboratories.

11.4.1 *Sample Preparation*

Sample preparation at ALS was as follows: All samples are submitted to ALS with barcode tags. ALS scans barcodes, weighs samples and logs-in samples to LIMS. Received samples are compared to submitted list on request for analysis. Samples are crushed to 90% passing -2mm. A riffle split of 500g is taken and then pulverized to >95% passing 106 microns. Analytical duplicates from pulps are taken within each analytical run at the end of the batch. The minimum number of duplicates required is based on the rack size specific to the method.

For PGM-ICP methods, a minimum of 3 duplicates are taken and for ICP-AES methods, a minimum of 1 duplicate is taken. For every 50 samples prepared, an additional split is taken from the crushed material to create a pulverizing duplicate. This additional split is processed and analyzed in a similar manner to other samples in the submission.

Crusher jaws and work stations are cleaned before the first sample of every new work order with barren material and compressed air and cleaned between each subsequent sample with compressed air. The grinding bowls are cleaned before the first sample of every new work order with silica and compressed air and between each subsequent sample with compressed air. A 100-150 gram split of each pulp sample is packaged, and then shipped to the analytical facilities in North Vancouver via FedEx Express. Pulps and coarse rejects were retained at ALS' Thunder Bay location. Once QA/QC checks have been completed the pulps and rejects are then transported by Exploration department personnel back to the LDIM, where they are stored in shipping containers.

11.4.2 Sample Analyses

All 2014 drill core samples were analyzed by ALS using the PGM-ICP23 (Pd, Pt, Au) and ME-ICP61 (Cu, Ni, Co, Mg, Ag) procedures. Where necessary, over-limit procedures PGM-ICP27 and ME-OG62 were used. Received results are imported into the database as ALS2014 lab package.

11.4.2.1 PGM-ICP23

Sample Decomposition: Fire Assay Fusion (FA-FUSPG1, FA-FUSPG2). Analytical Method: Inductively Coupled Plasma – Atomic Emission Spectrometry (ICP-AES). A prepared sample (30 g) is fused with a mixture of lead oxide, sodium carbonate, borax and silica, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested for 2 minutes at high power by microwave in dilute nitric acid. The solution is cooled and hydrochloric acid is added. The solution is digested for an additional 2 minutes at half power by microwave. The digested solution is then cooled, diluted to 4 mL with 2 % hydrochloric acid, homogenized and then analyzed for gold, platinum and palladium by inductively coupled plasma – atomic emission spectrometry.

Table 11-1: Detection Limits for ALS PGM-ICP23

Element	Detection Limit	Upper Limit
Pd	1 ppb	10,000 ppb
Pt	5 ppb	10,000 ppb
Au	1 ppb	10,000 ppb

11.4.2.2 *ME-ICP61*

Sample Decomposition: HNO_3 -$HClO_4$ -HF-HCl digestion, HCl Leach (GEO -4ACID). Analytical Method: Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP - AES). A prepared sample (0.25 g) is digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed by inductively coupled plasma-atomic emission spectrometry. Results are corrected for spectral interelement interferences.

Table 11-2: Detection Limits for ALS ME-ICP61

Element	Detection Limit	Upper Limit
Cu	1 ppm	10,000 ppm
Ni	1 ppm	10,000 ppm
Co	1 ppm	10,000 ppm
Mg	0.01%	50%
Ag	0.5 ppm	100 ppm

11.4.2.3 *PGM-ICP27*

Sample Decomposition: Fire Assay Fusion (FA-FUSPG3). Analytical Method: Inductively Coupled Plasma – Atomic Emission Spectrometry (ICP-AES). A 30 g prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax and silica, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested for 2 minutes at high power by microwave in dilute nitric acid. The solution is cooled and hydrochloric acid is added. The solution is digested for an additional 2 minutes at half power by microwave. The digested solution is then cooled, diluted to 4 mL with 2 % hydrochloric acid, homogenized and then analyzed for gold, platinum and palladium by inductively coupled plasma – atomic emission spectrometry.

Table 11-3: Detection Limits for ALS PGM-ICP27

Element	Detection Limit	Upper Limit
Pd	0.03 ppm	100 ppm
Pt	0.03 ppm	100 ppm
Au	0.03 ppm	100 ppm

11.4.2.4 *ME-OG62*

Sample Decomposition: HNO_3 -$HClO_4$ -HF-HCl Digestion (AS Y-4A01). Analytical Method: Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP - AES). A prepared sample is digested with nitric, perchloric, hydrofluoric, and hydrochloric acids, and then evaporated to incipient dryness. Hydrochloric acid and de-ionized water is added for further digestion, and the sample is heated for an additional allotted time. The sample is cooled to room temperature and transferred to a volumetric flask (100 mL). The resulting solution is diluted to volume with de-ionized water, homogenized and the solution is analyzed by inductively coupled plasma - atomic emission spectroscopy or by atomic absorption spectrometry.

Table 11-4: Detection Limits for ALS ME-OG62

Element	Detection Limit	Upper Limit
Cu	0.001%	40%
Ni	0.001%	30%
Co	0.001%	20%
Mg	0.01%	50%
Ag	1 ppm	1,500 ppm

11.4.3 Check Sample Analyses at ActLabs

An approximate 100g pulp split for requested samples is generated by ALS during the initial sample preparation. Exploration department personnel retrieve the pulp splits from ALS, insert QC samples (blank and standard reference materials) at a rate 3 per batch of 34 samples and then deliver to Activation Laboratories (ActLabs) in Thunder Bay for analysis. ActLabs is accredited to international quality standards through the International Organization for Standardization /International Electrotechnical Commission (ISO/IEC) 17025 (ISO/IEC 17025 includes ISO 9001 and ISO 9002 specifications) with CAN-P-1578 (Forensics), CAN-P-1579 (Mineral Analysis) and CAN-P-1585 (Environmental) for specific registered tests by the SCC.

11.4.3.1 1COES-Fire Assay ICPOES

A sample size of 5 to 50 grams can be used but the routine size is 30 g for rock pulps, soils or sediments (exploration samples). The sample is mixed with fire assay fluxes (borax, soda ash, silica, litharge) and with Ag added as a collector and the mixture is placed in a fire clay crucible. The mixture is then preheated at 850°C, intermediate 950°C and finish 1060°C with the entire fusion process lasting 60 minutes. The crucibles are then removed from the assay furnace and the molten slag (lighter material) is carefully poured from the crucible into a mould, leaving a lead button at the base of the mould. The lead button is then placed in a preheated cupel which absorbs the lead when cupelled at 950°C to recover the Ag (doré bead) + Au, Pt and Pd. The Ag doré bead is digested in hot (95°C) HNO 3 + HCl.

After cooling for 2 hours the sample solution is analyzed for Au, Pt, Pd by ICP/OES using a Varian 735 ICP. The instrument is recalibrated every 45 samples. On each tray of 42 samples there are two method blanks, three sample duplicates, and 2 certified reference materials. If values exceed upper limits, reanalysis by fire assay Au, Pt, Pd (Code 8) is recommended.

Table 11-5: Detection Limits for ActLabs 1COES

Element	Detection Limit	Package Upper Limit	LDIM Upper Limit
Pd	5 ppb	30,000 ppb	10,000 ppb
Pt	5 ppb	30,000 ppb	10,000 ppb
Au	2 ppb	30,000 ppb	10,000 ppb

11.4.3.2 Code 8 Au Pd Pt - Fire Assay - ICP

A 30 gram sample is mixed with fire assay fluxes (borax, soda ash, silica, litharge) and with Ag added as a collector and the mixture is placed in a fire clay crucible, the mixture is preheated at 850 °C, intermediate 950 °C and finish 1060 °C, the entire fusion process should last 60 minutes. After cooling, the lead button is separated from the slag and cupelled at 950 °C to recover the Ag (doré bead) + Au, Pt, Pd.

The Ag doré bead is digested in hot (95 °C) HNO3 + HCl. After cooling for 2 hours the sample solution is analyzed for Au, Pt, Pd by ICP/OES using a Varian 735 ICP. A blank and a digested standard are run every 15 samples. Instrument is recalibrated every 45 samples. All samples are analyzed in duplicate.

Table 11-6: Detection Limits for ActLabs Code 8

Element	Detection Limit	Upper Limit
Au	0.001 ppm	1,000 ppm
Pt	0.001 ppm	1,000 ppm
Pd	0.001 ppm	1,000 ppm

11.4.3.3 1F2-Total Digestion ICP

A 0.25 g sample is digested with four acids beginning with hydrofluoric, followed by a mixture of nitric and perchloric acids. This is then heated using precise programmer controlled heating in several ramping and holding cycles which takes the samples to incipient dryness. After incipient dryness is attained, samples are brought back into solution using aqua regia. With this digestion, certain phases may be only partially solubilized. These phases include zircon, monazite, sphene, gahnite, chromite, cassiterite, rutile and barite. Ag greater than 100 ppm and Pb greater than 5000 ppm should be assayed as high levels may not be solubilized. Only sulphide sulfur will be solubilized. The samples are then analyzed using an Agilent 735 ICP. QC for the digestion is 14% for each batch, 5 method reagent blanks, 10 in-house controls, 10 samples duplicates, and 8 certified reference materials. An additional 13% QC is performed as part of the instrumental analysis to ensure quality in the areas of instrumental drift.

Table 11-7: Detection Limits for ActLabs Code 1F2

Element	Detection Limit	Upper Limit	Element	Detection Limit	Upper Limit	Element	Detection Limit	Upper Limit
Ag	0.3	100	Ga	1	10,000	Sb	5	10,000
Al*	0.01%	50%	Hg	1	1000	Sc	4	10,000
As*	3	5,000	K	0.01%	10%	Sr	1	10,000
Ba*	7	1,000	Li	1	10,000	Te	2	10,000
Be	1	10,000	Mg	0.01%	50%	Ti	0.01%	10%
Bi	2	10,000	Mn	1	100,000	Tl	5	10,000
Ca	0.01%	70%	Mo	1	10,000	U	10	10,000
Cd	0.3	2,000	Na	0.01%	10%	V	2	10,000
Co	1	10,000	Ni	1	10,000	W*	5	10,000
Cr*	1	10,000	P	0.001%	10%	Y*	1	1000
Cu	1	10,000	Pb	3	5,000	Zn	1	10,000
Fe*	0.01%	-	S	0.01%	20%	Zr*	5	10,000

Notes: * Element may only be partially extracted.
+ Only sulphide sulphur is extracted. Assays are recommended for values which exceed the upper limits.

12. Data Verification

NAP exploration department has data verification procedures for all diamond drill programs. These procedures include but are not limited to verification of coordinate data, survey data, assay data, duplicate assay sampling, check assay sampling and data management. In 2014, primary assays were obtained from ALS Global and check sample assays were obtained from Actlabs. Both labs are accredited to international quality standards and are ISO certified, as described in Section 11.

12.1 Drill Hole Data Capture

Drill hole related data including samples and analytical results are managed in CAE Mining's SQL server based Fusion software. Logging of drill hole data is done on laptop computers directly into the DHLogger component. This includes data capture verification such as unique drill hole number, unique sample number and prevention of gaps or overlaps in major lithology units. Completed drill holes are 'checked-in' to the SQL server central database.

Down hole orientation surveys are imported as tables into this database. All 2014 down hole surveys were conducted using the Reflex Gyro and included in the 2014 resource estimate. All drillholes used in the resource estimation are reviewed for atypical survey profiles. 35 of the total 1,726 surface and underground drillholes were deemed to have non-compliant surveys and are not included in the current resource estimate.

Analytical results are reported by the laboratories in a specific .csv template format which allows for direct import into the Fusion software. For import to occur, sample numbers listed in the analytical certificate must match a corresponding sample entered during the drill hole logging process. All analytical data handling is done by the software including conversion of reported values at the lower detection limit ('<' values) to half of the lower detection limit and presentation of results converted to a common unit of measure. Software end-users cannot access or modify imported analytical results through the DHLogger interface.

12.2 QA/QC

2014 diamond drilling QA/QC included quarter core duplicate analyses, check sample analyses and insertion of blank material and standard reference material into the sampling series.

12.2.1 Quarter Cut Duplicate Analyses

Quarter cut drill core sample pairs from the Mineblock Intrusion (MBI) DDH were taken in 2014 for duplicate result comparison. Approximately 5% of sampled intervals were visually selected for quarter cut duplicate sampling. From the half core normally submitted for analysis, the selected intervals were cut into two quarter core pieces and submitted as separate sample numbers. A custom trigger in the Fusion software provided by CAE performs a weighted average of the two quarter cut sample results as the received certificate is imported. This weighted average is used in all data extracts for that sampled interval.

A review of the quarter cut duplicate analyses for the 2014 drill core samples was still in progress as of the effective date of this report. To date, the 1257 quarter cut duplicate results have been received from drilling completed in 2014 on the MBI, including samples from the Offset Zone. Quarter cut duplicate analyses were performed by ALS as part of the normal drill core sampling series.

Plots of quarter cut duplicate pairs assay results for palladium, platinum, and gold are displayed in Figure 12-1 to Figure 12-3. These plots display the final values for each quarter cut duplicate sample pair (over limits used for one or both samples if applicable; re-assays due to inserted QC failure if applicable).

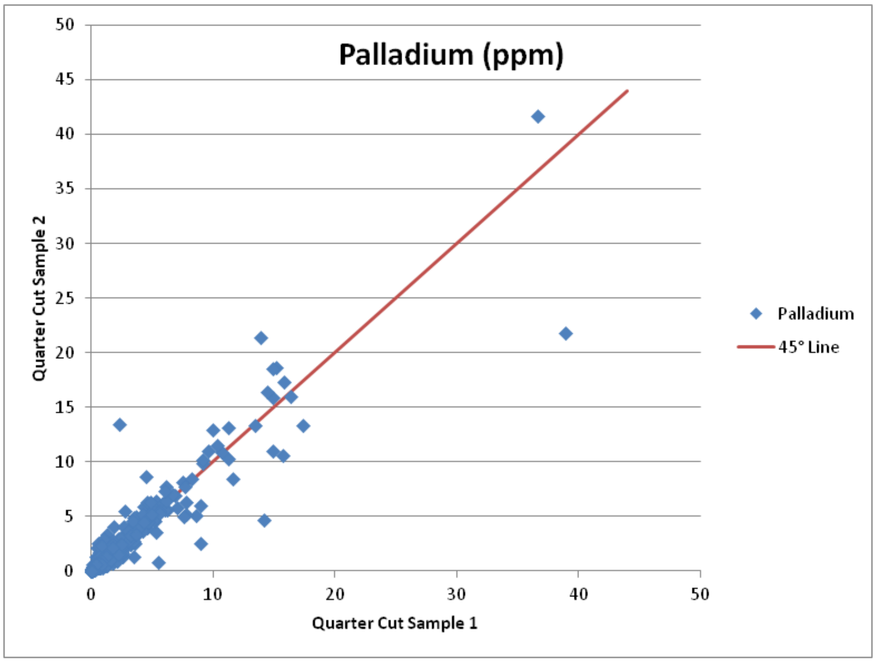


Figure 12-1: Comparison of Quarter-cut Core Duplicate Assays for Palladium from the 2014 Exploration Program on the Offset Zone

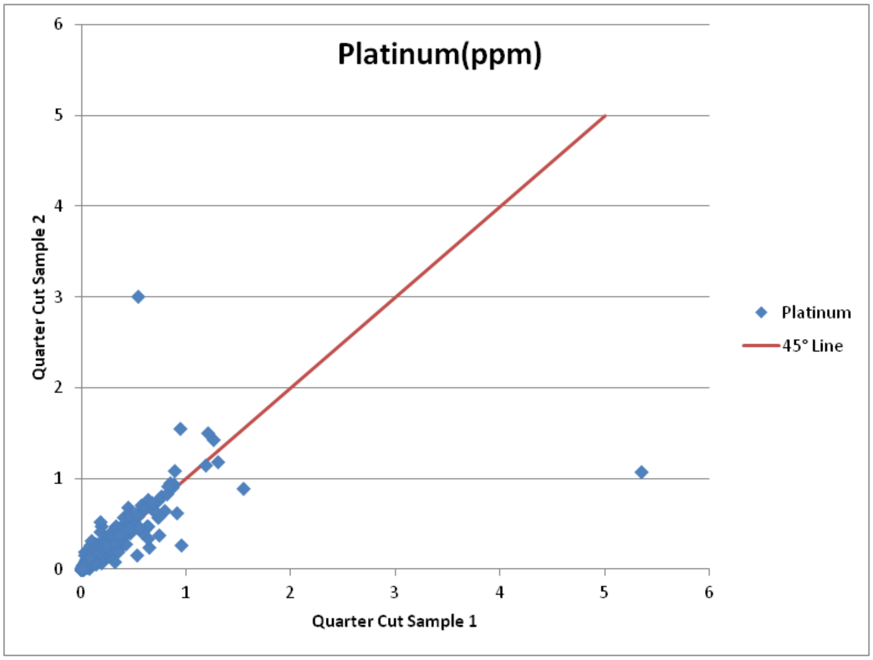


Figure 12-2: Comparison of Quarter-Cut Core Duplicate Assays for Platinum from the 2014 Exploration Program on the Offset Zone

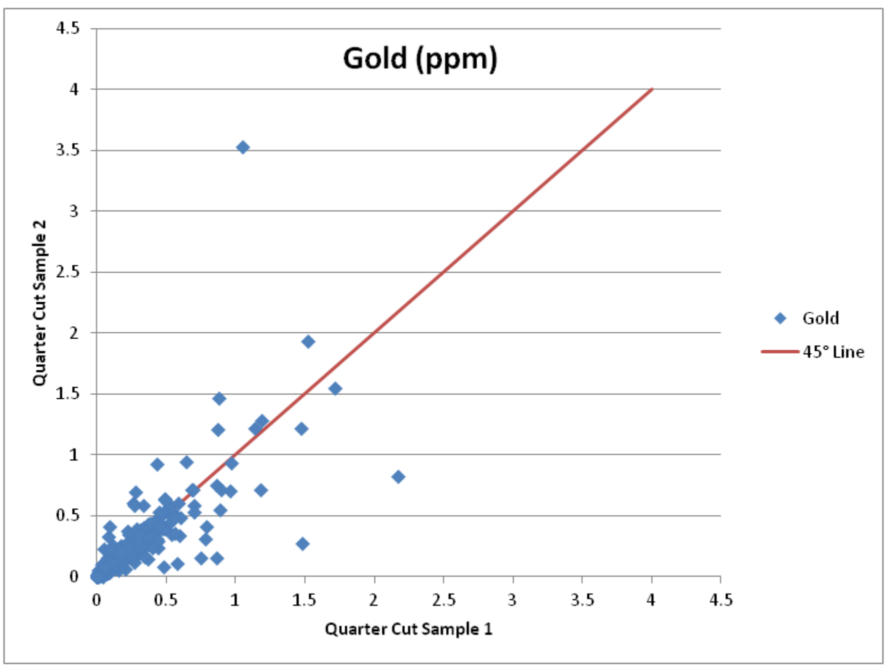


Figure 12-3: Comparison of Quarter-Cut Core Duplicate Assays for Gold from the 2014 Exploration Program on the Offset Zone

12.2.2 Check Sample Analyses

Secondary laboratory check sample analyses at ActLabs were still in progress at the time of writing. To date, 1221 results have been received from drill core samples obtained from the 2014 exploration program on the MBI. Samples submitted for check sampling analysis were dominantly taken from one of each quarter cut duplicate pairs for approximately 5% of the sampled intervals.

Plots of original assays against check assays for palladium, platinum, and gold are displayed in Figure 12-4 to Figure 12-6.

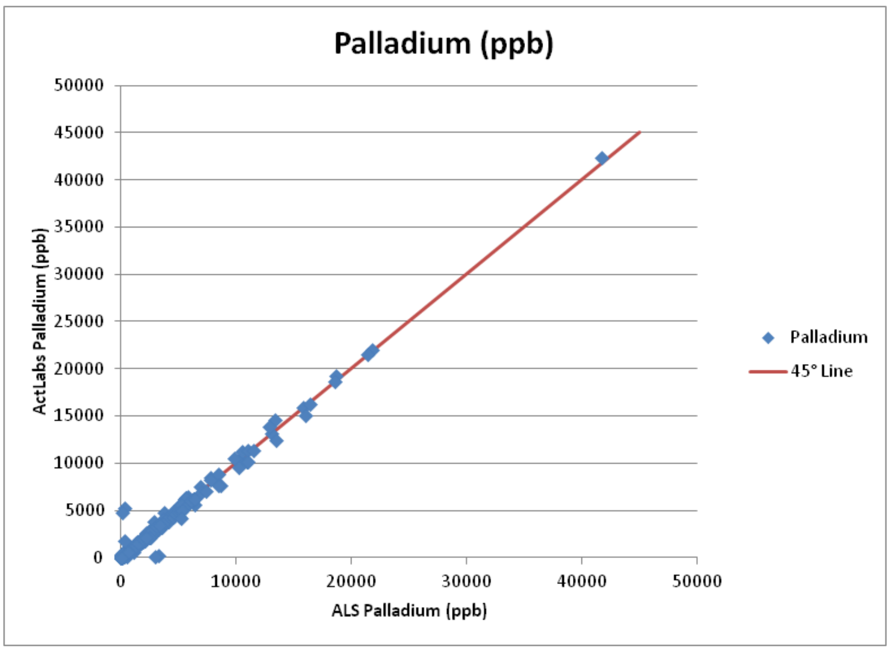


Figure 12-4: Palladium Validation Plot from the 2014 Exploration Program on the Offset Zone

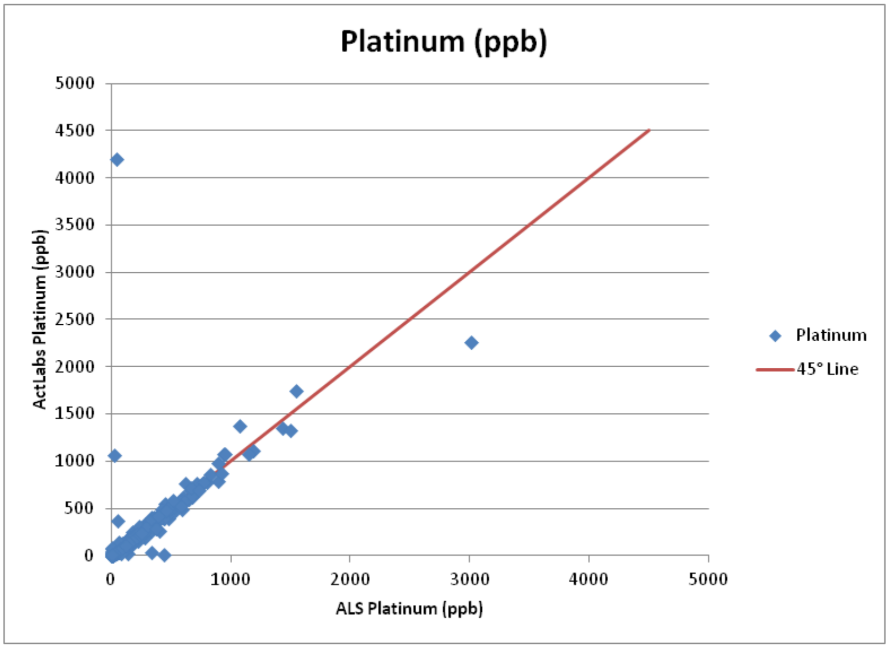


Figure 12-5: Platinum Validation Plot from the 2014 Exploration Program on the Offset Zone

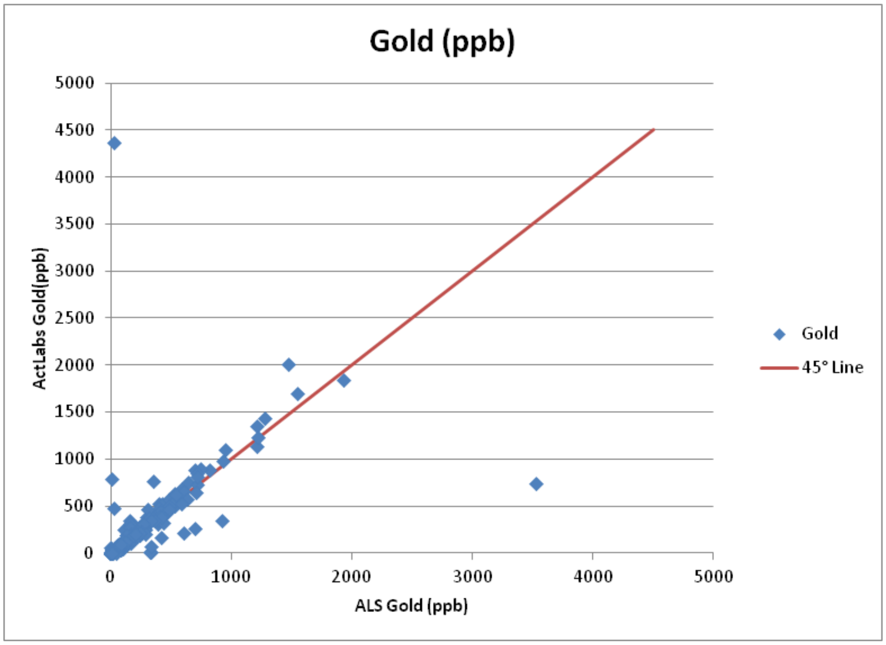


Figure 12-6: Gold Validation Plot from the 2014 Exploration Program on the Offset Zone

12.2.3 Blanks and Standards

Coarse crush blank material and standard reference materials were inserted into the drill core continuous sampling series at regular intervals during the 2014 drill program. Samples were mostly submitted to ALS in batches of 78 samples including the NAP inserted quality control (QC) samples. Batches covering the end of one hole would then include the start of the next sampled hole to total 78 samples. Batches in 2014 contained 6 NAP inserted QC samples spaced at regular intervals through the batch. The analytical results for the inserted QC samples are assessed by the Fusion software during certificate import.

Table 12-1: Sources and Names for Standards and Blanks Used in the 2014 QA/QC Program

Material	Source	Total # Used in 2014
CDN-PGMS-24	CDN Resources	636
CDN-PGMS-25	CDN Resources	647
Blank Crush	Lockstone Yard and Patio Centre, Thunder Bay	646

The blank crush samples are considered failed if one or more of the palladium, platinum, or gold results exceed the maximum allowable upper limit determined by 10X the lower detection limit of the analytical method. Once a failure is detected, the lab is instructed to re-assay from pulps all samples from the preceding blank or standard that passed to the following blank or standard that passed. These pulp re-assays are then used in the drillhole database.

The CDN standard samples results for palladium, platinum, and gold are assessed based on their recommended values and standard deviations reported on the CDN certificates. A standard sample fails if any of the three elements exceed three standard deviations. Additionally, a warning is issued if a palladium result falls between the 2 and 3 standard deviations and the adjacent results reviewed. In these cases, a standard would be considered failed if two or more palladium results in a row fall between the 2 and 3 standard deviations on the same side. Where a failure is detected, the lab is instructed to re-assay from pulps all samples from the preceding blank or standard that passed to the following blank or standard that passed. These re-assays are then used in the drillhole database.

12.3 QP's Opinion

NAP continues to keep to rigorous validation standards for its drill hole assay data. The assay data used for resource estimation described in Section 14 of this report is considered to be valid and acceptable for use for this purpose. Information for the validation of historical assay data used in this most recent resource estimation is available in previously filed technical reports.

13. Mineral Processing and Metallurgical Testing

13.1 History and Previous Reports

The present processing plant, which has a nominal capacity of 15,000 tpd, will process all of the material hauled from the open pit and underground mines. The plant runs on a continuous 365 days per year schedule.

A grinding study was completed by SGS Lakefield Research Ltd. (SGS) in 2008 (SGS 2008) to evaluate different grinding circuit scenarios based on the equipment currently available on site. The evaluations were made using JKSimMet simulations. These scenarios were based on a primary grind size that was too large to achieve an optimum palladium flotation recovery. These scenarios were also based on lower mill throughputs than what is currently fed to the mill.

A subsequent metallurgical investigation of samples from the Roby and Offset zones by Xstrata Process Support (XPS) in 2010 (XPS 2010) determined that a grind size P_{80} of 38 µm would achieve the optimum palladium flotation recovery. A semi-autogenous (SAG)/ball mill/crushing (SABC) grinding circuit arrangement with two Vertimills® for tertiary grinding is used to reach this optimum grind size. The investigation by XPS also determined that the Roby and Offset zones have the same flotation metallurgical response.

In 2012, an additional test program was undertaken by NAP staff at the LDI site to evaluate the feasibility of introducing flash flotation to the tertiary grinding circuit. The testing showed that very promising recoveries could be obtained with flash flotation on the Vertimill® hydrocyclones underflows.

The assays and metallurgical data in this report are received from LDI/NAP and they are accepted to be correct. Hatch did not verify the accuracy of these data via independent third party laboratories.

13.2 SGS Grinding Circuit Evaluation

13.2.1 Grinding Test Work

At the time of the 2008 SGS test works the LDI operation was running on an intermittent operating schedule as the mill was operating below capacity. In an effort to develop process alternatives at this reduced throughput, a number of grinding circuit simulations utilizing existing equipment were completed by SGS.

The circuit simulations considered are listed below. The simulations were completed using JKSimMet software and verified using Bond's third theory of comminution. Emphasis was placed on the newer equipment, but some options using equipment in the 2,400 tpd mill were also considered:

- FAB (fully-autogenous grinding (FAG) milling and ball milling).
- FABC (FAG milling, ball milling and pebble crushing).
- FAPC (FAG milling, pebble milling and pebble crushing).
- SAPC (SAG milling, pebble milling and pebble crushing).
- FAP (FAG milling and pebble milling).
- FAC (close-circuit FAG milling).
- Rod milling and ball milling at 3,500 tpd.
- Rod milling, ball milling and Vertimilling® at 3500 tpd.

The breccia grindability test results are summarized in Table 13-1, which were reported in A.R. MacPherson Consultants Ltd. (ARMC) project 9876 (as referenced in XPS 2010). These grindability results were used for all simulations.

Table 13-1: Breccia Grind-ability Test Results

Sample Name	Relative Density	Relative Density JK Drop Test Parameters		BWi (kWh/t)
		A x b	t_a	
Breccia	3.06	29.8	0.24	19.2

Note: BWi – Bond ball mill work index, A – maximum breakage, b – relation energy versus impact breakage, ta – abrasion parameter
Source: SGS (2008)

The ultimate goal of these simulations was to determine a feasible flow sheet that would minimize capital and operating costs. According to JKSimMet simulations run by SGS, pebble milling would be an excellent option, but the complexity of adding the circuit into the current infrastructure made it a non-practical option. Refurbishing the equipment in the 2,400 tpd mill also could not meet the long-term tonnage requirement.

Given these constraints, the preferred option of LDI was a SABC circuit using installed Vertimills® as tertiary grind to a final product of P_{80} -38 µm (current P_{80} is - 58 µm pending optimization). The P_{80} is defined as 80% of the grind product passing through a sieve of a defined size. This particular circuit was not one of the configurations simulated in the 2008 SGS study. It is the configuration currently used at Lac des Iles Mine with a 13,000 tpd throughput.

13.3 XPS Roby and Offset Zone Mineralized Material Characteristics

XPS was retained for characterization of Roby and Offset Zone mineralized material. Drill core composite samples from each zone were submitted to XPS by LDI in February 2010 for this purpose. XPS also assisted with the sample selection. The Roby Zone samples were chosen so that they were representative of the entire zone, including the mined out areas. The Offset Zone samples were limited to the upper phase of the Offset Zone, although samples were selected from different levels and lithologies of the upper phase to ensure the sample was representative.

The scope of work for the test program included the following:

- Modal mineralogy of mill feed composites and one locked-cycle test concentrate sample for Qualitative Evaluation of Materials by Scanning Electron Microscope (QEMSCAN™).
- Flowsheet development investigating primary grind size, reagent screening, staged grind/float evaluation, cleaning and magnesium oxide (MgO) depressant tests.
- Locked-cycle test for metallurgical performance of the Roby and Offset zones.
- Bond's ball mill work index for each zone.
- Gravity recovery tests.

13.3.1 Modal Mineralogy

The modal mineralogy of the Roby and Offset zones are shown in Figure 13-1. These mineralogy studies were carried out by QEMSCAN™ and Electron Probe Micro Analysis (EPMA). As illustrated in Figure 13-1, the modal mineralogy of both zones was very similar, with slight variations in the amount of sulphides, quartz, micas, orthopyroxene, talc, and chlorites.

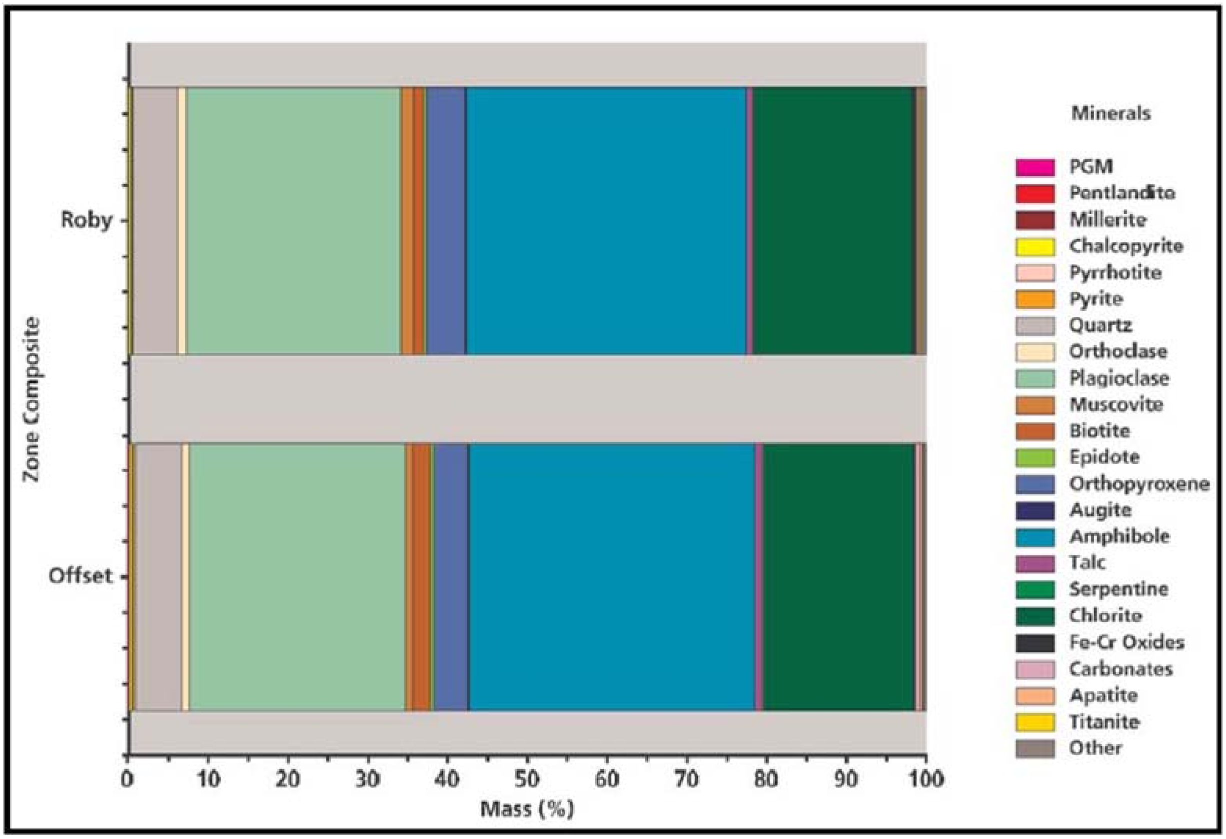


Figure 13-1: Mineralogy of Roby and Offset Zones - Source: XPS (2010)

The predominant difficulty in processing these mineralized material types is in dealing with floatable gangue. Mineralized material variability, in terms of talc content, is expected to occur in localized regions.

The sulphides make up less than 1% (w/w) of the mineralogy. The sulphide mineralogy of both zones is shown in Figure 13-2. There are a greater amount of sulphides contained in the Offset Zone with proportionately more pentlandite, pyrrhotite, and pyrite with less millerite and chalcopyrite. The greater amounts of sulphides will not have any adverse effects on the process. The main effect of the greater amount of iron sulphides (i.e. pyrrhotite and pyrite) will be to reduce the concentrate grade, although this is not expected to be a significant reduction.

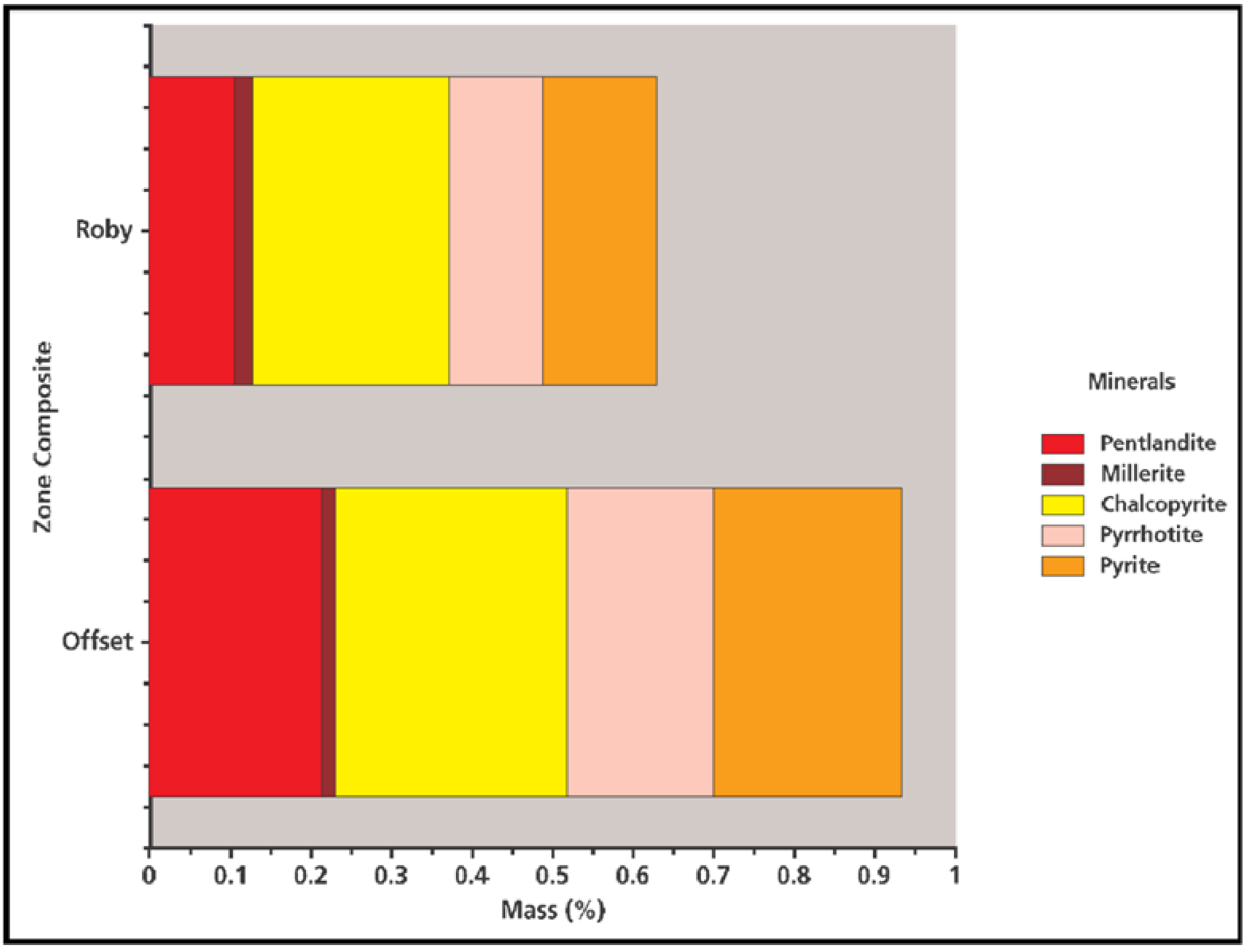


Figure 13-2: Sulphide Mineralogy of Roby and Offset Zones - Source: XPS (2010)

13.3.2 Nickel and Copper Sulphide Liberation

The liberation of both zones composites has been evaluated at the present 75 µm grind size. The liberation of the nickel and copper sulphides is represented by the bars in Figure 13-3.

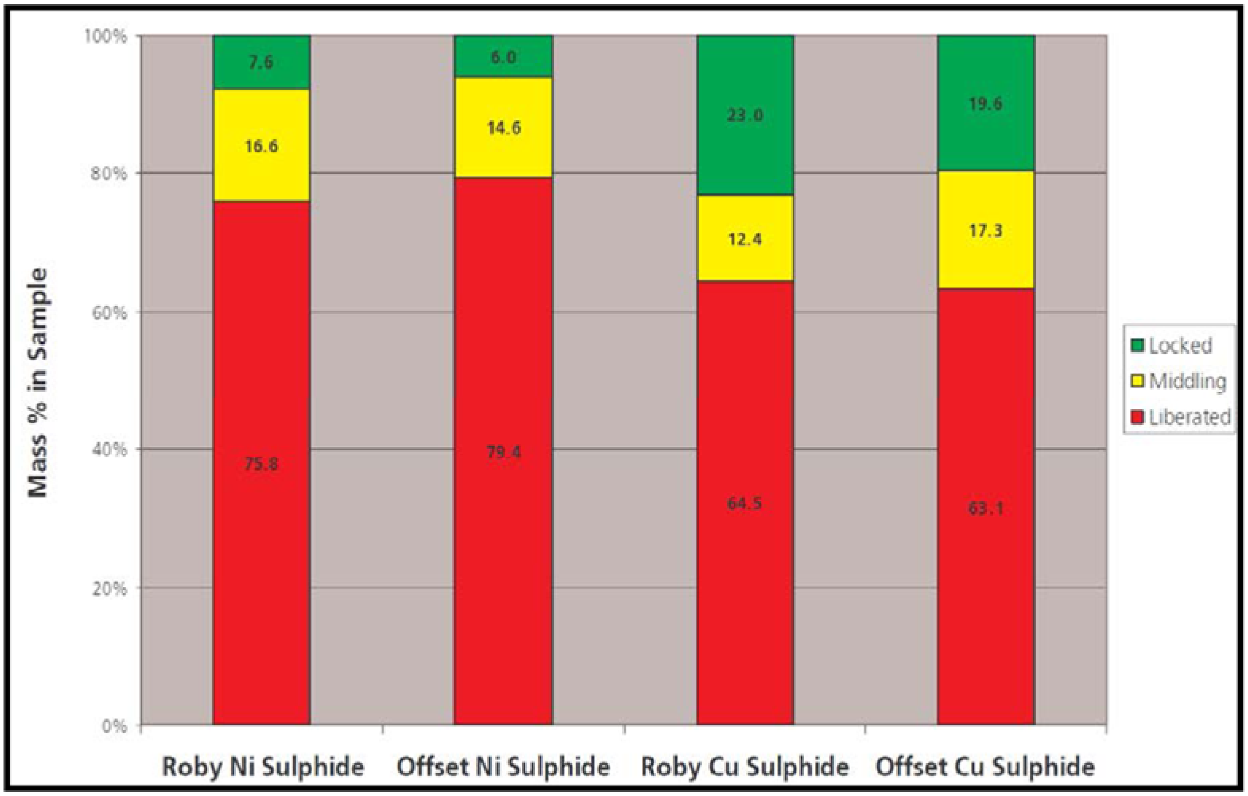


Figure 13-3: Total Liberation of Ni and Cu Sulphide Roby and Offset Zone - Source: XPS (2010)

The nickel sulphides are moderately well liberated while the copper sulphides are contained in more locked particles. This result was surprising, given LDI history of poor nickel recoveries but excellent copper recoveries.

13.3.3 Nickel Deportment and Recovery Issues

In an attempt to explain the historically poor nickel recoveries at LDI a number of other mineralogical investigations were evaluated. Even though the nickel sulphides are shown to be well liberated, an evaluation of the middling and locked particle associations was done and results are shown in Figure 13-4. The graph indicates that there is a much greater proportion of nickel associated with non-sulphide gangue for the Offset Zone and may result in poorer recovery than for the Roby Zone.

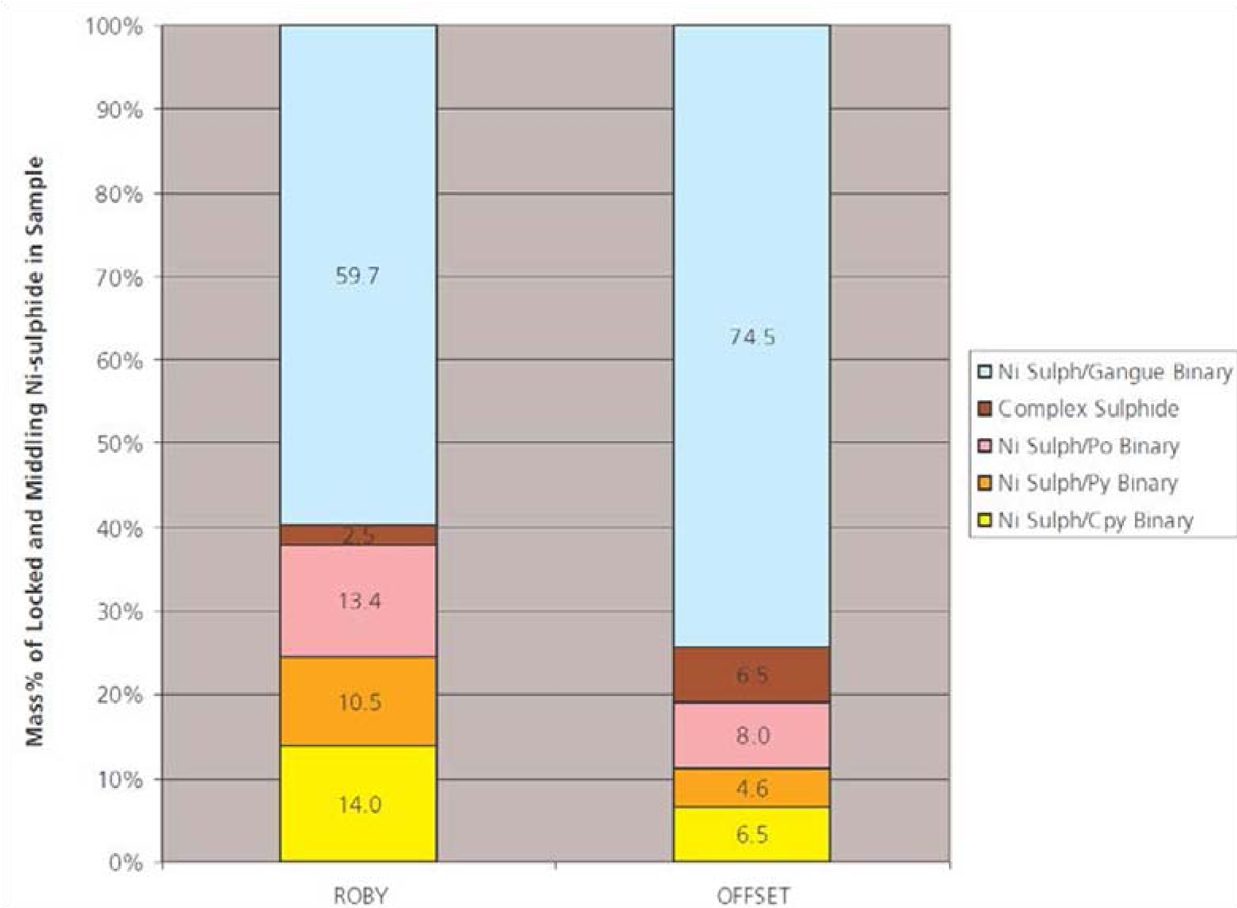


Figure 13-4: Nickel Sulphide Locking Characteristics - Source: XPS (2010)

Figure 13-5 shows that 52% and 60% of the nickel is contained in sulphide minerals for the Roby and Offset zones respectively.

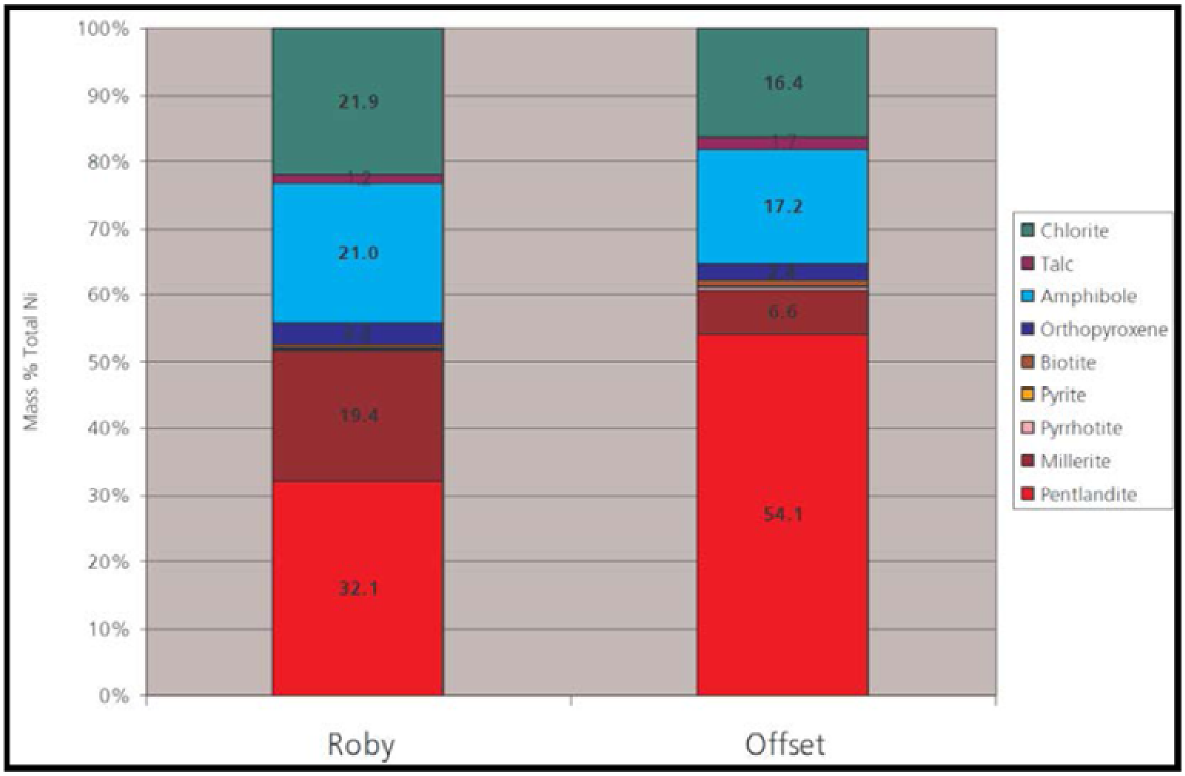


Figure 13-5: Nickel Deportment by Mineral Species - Source: XPS (2010)

There is a considerable amount of nickel associated with gangue minerals which is lost to tailings. The historical nickel recovery has been in the order of 30%.-It is understood that previous student studies have investigated the tailings mineralogy and have indicated limited opportunity to improve nickel recovery.

13.3.4 PGM Mineralogy

The predominant Pd minerals are sulphides (vysotskite), tellurides (kotulskite, merenskyite) and antimonides (mertieite) with minor amounts in Au/Ag species (electrum), arsenides and other Te/Bi minerals. By mass, the majority of PGMs are in the tellurides, antimonides, followed by sulphides.

The grain size of PGM minerals is shown in Figure 13-6. As can be seen, the PGM grain sizes have a bi-modal nature but all exclusively under 25 µm which precludes gravity concentration strategies.

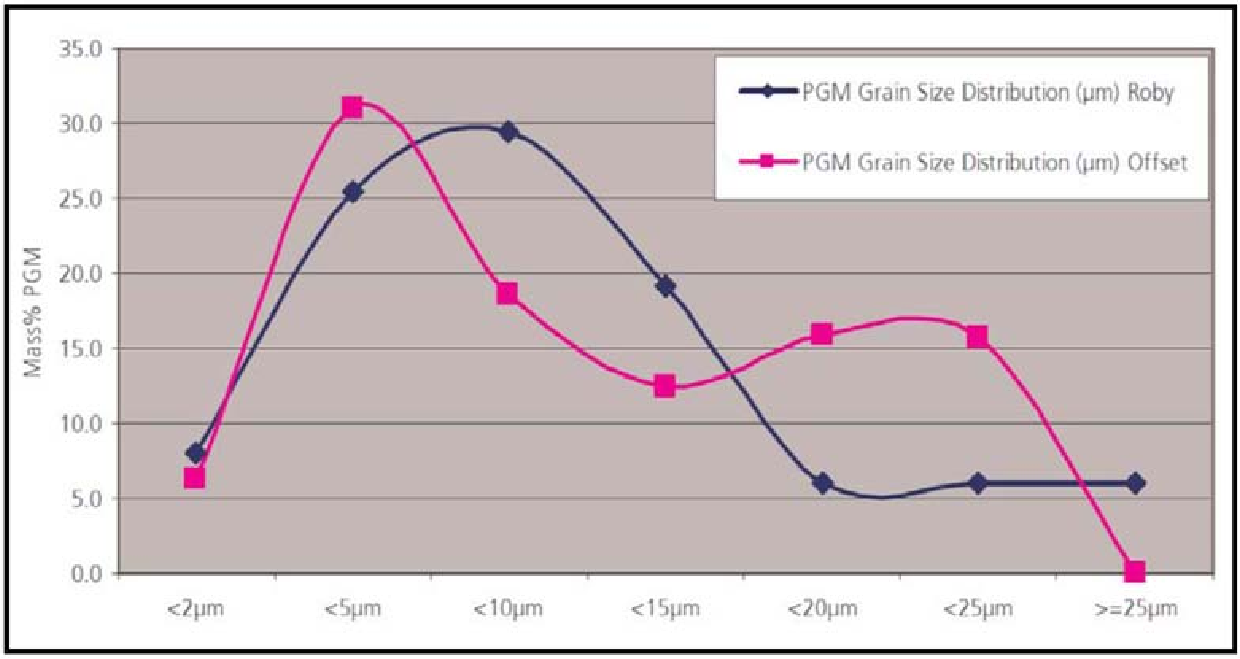


Figure 13-6: PGM Mineral Grain Size Distribution - Source: XPS (2010)

13.3.5 Flotation Test Work

A series of rougher-scavenger flotation tests were conducted at various grind and reagent conditions that concentrated on the Roby Zone and then evaluated on the Offset Zone. Variables evaluated include:

- Grinding between P_{80} value of 38 to 75 µm.
- Two levels of CMC addition and the use of Aero 3477 (a PGM promoter).
- Stage grinding.

The head grades of the feed to flotation for each sample are presented in Table 13-2.

Table 13-2: Head Grade of Roby and Offset Composites

Head Grade	Ni (%)	Cu (%)	S (%)	MgO (%)	Au (g/t)	Pt (g/t)	Pd (g/t)
Roby Zone	0.12	0.07	0.18	12.2	0.37	0.31	6.12
Offset Zone	0.14	0.1	0.28	11.6	0.29	0.35	5.52

Source: XPS (2010)

13.3.6 Rougher Scavenger – Grind Sensitivity

Rougher-scavenger flotation tests were carried out at 75, 53, and 38 µm grind sizes on the Roby Zone sample to determine the optimal flotation feed size. Figure 13-7 presents the results from these rougher tests.

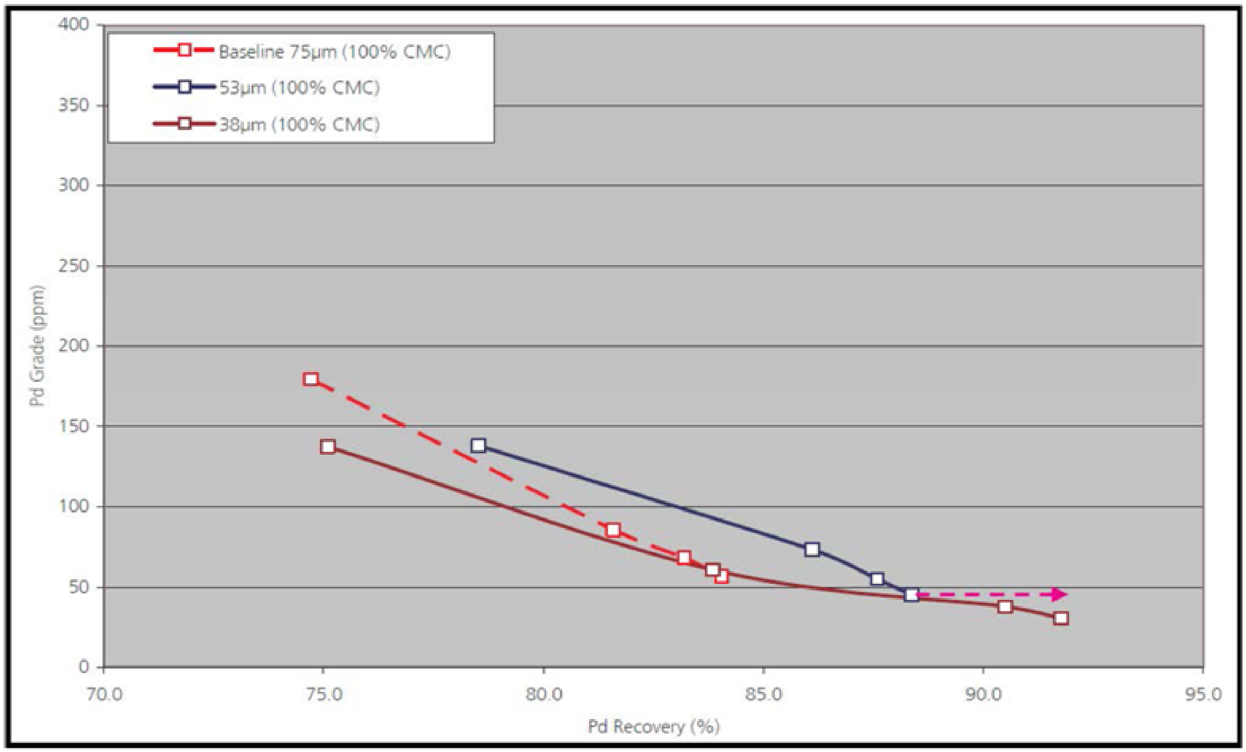


Figure 13-7: Roby Zone – Rougher Scavenger Palladium Grade/Recovery versus Primary Grind Size - Source: XPS (2010)

As can be seen in Figure 13-7, the finer the grind size the better the palladium recovery. This is consistent with the results found in the mineralogical work. Similarly, tests were also performed on the Offset Zone sample at grind sizes of 75 µm and 38 µm. Results from these tests are shown in Figure 13-8. The same conclusion can be drawn for these tests that the finer grind resulted in higher palladium recovery.

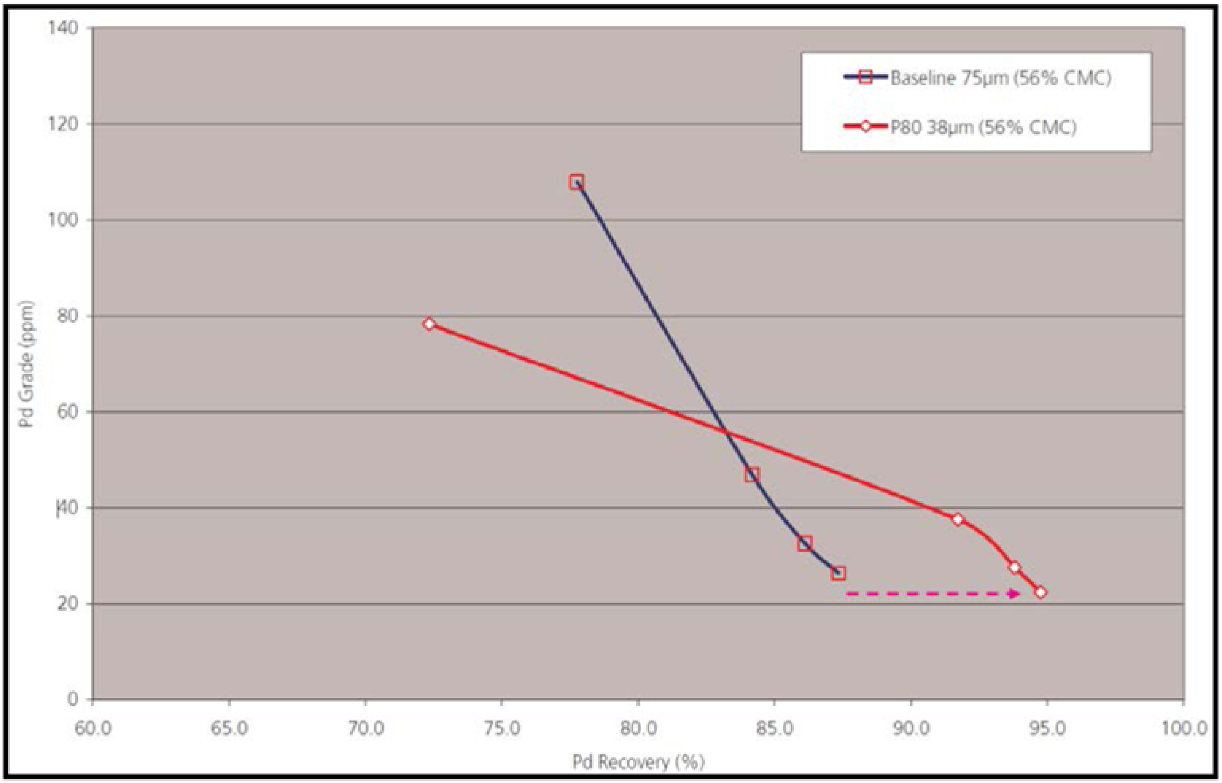


Figure 13-8: Offset Zone – Rougher Scavenger Palladium Grade/Recovery versus Primary Grind Size - Source: XPS (2010)

13.3.7 *Rougher Scavenger – CMC Dosage*

A series of rougher-scavenger tests using a primary grind of 75 μm were completed on the Roby Zone sample to determine the effect of CMC dosage on the palladium recovery. The results of these tests are presented in Figure 13-9. The recommended CMC dosage of 172 g/t was tested as well as 56% of the recommended dosage (i.e., 92 g/t). The test results illustrate that the higher CMC dosage has a detrimental effect on the palladium recovery. Figure 13-10 shows the results of similar tests completed on Roby Zone samples but at 53 μm. The same effect can be seen in these tests although the effect is not as pronounced at the 53 μm grind.

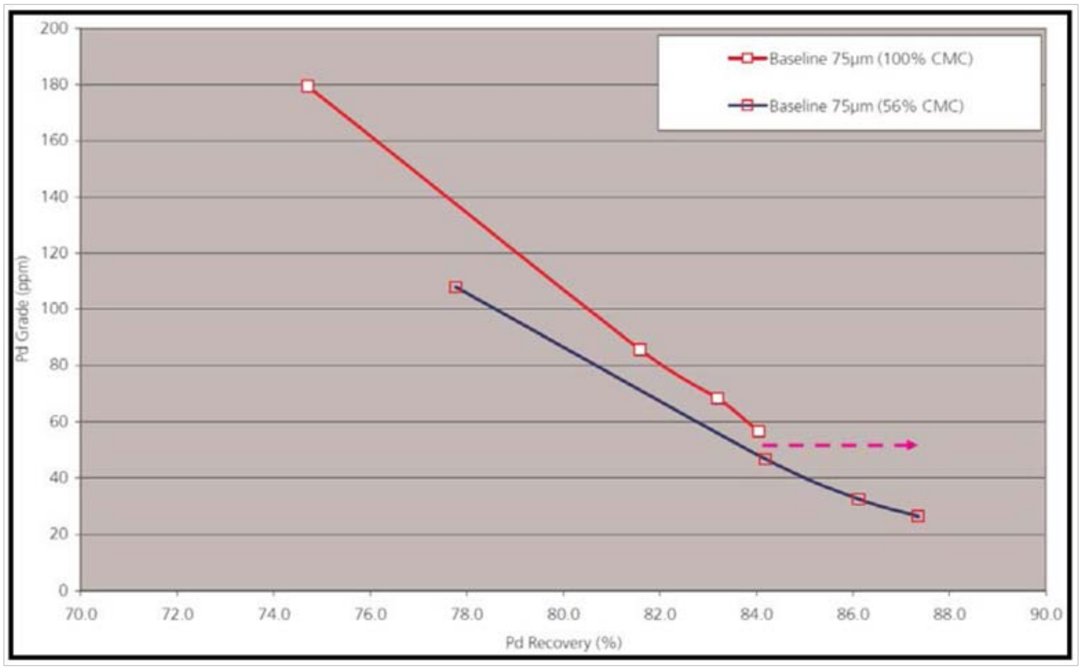


Figure 13-9: Roby Zone – Rougher Scavenger Palladium Grade/Recovery versus CMC Dosage Primary at Primary Grind of 75 µm - Source: XPS (2010)

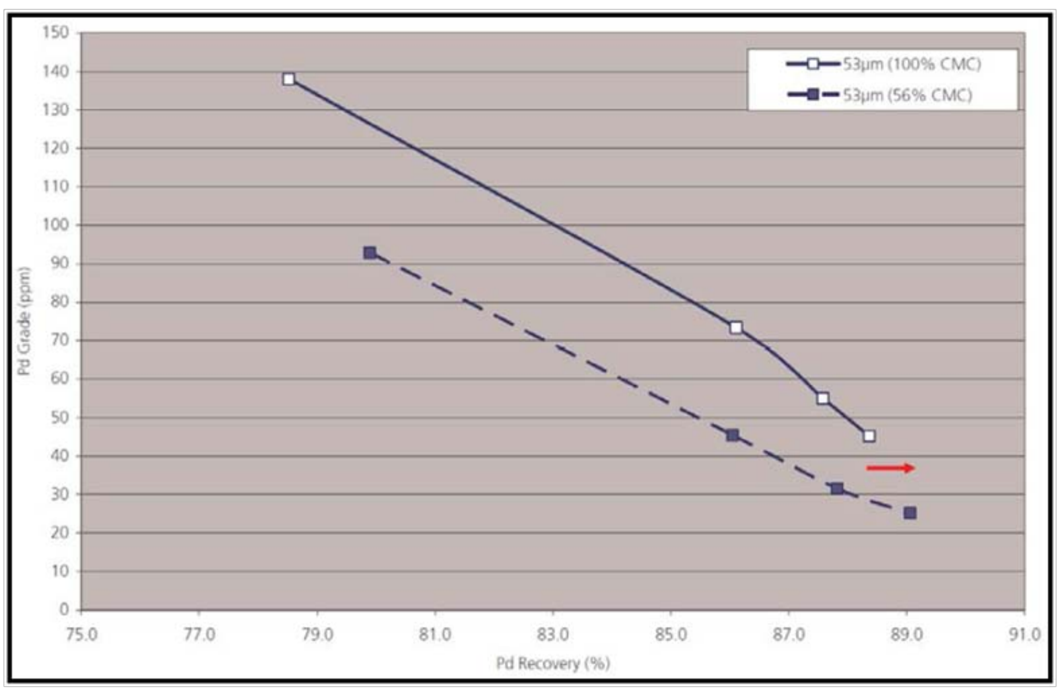


Figure 13-10: Roby Zone – Rougher Scavenger Palladium Grade/Recovery versus CMC Dosage Primary at Primary Grind of 53 µm - Source: XPS (2010)

Validation rougher-scavenger tests using the same dosage of CMC were carried out on the Offset Zone sample as well. Results from these tests are presented in Figure 13-11. The lower CMC dosage of 92 g/t resulted in higher palladium recoveries as it did in the Roby Zone sample tests.

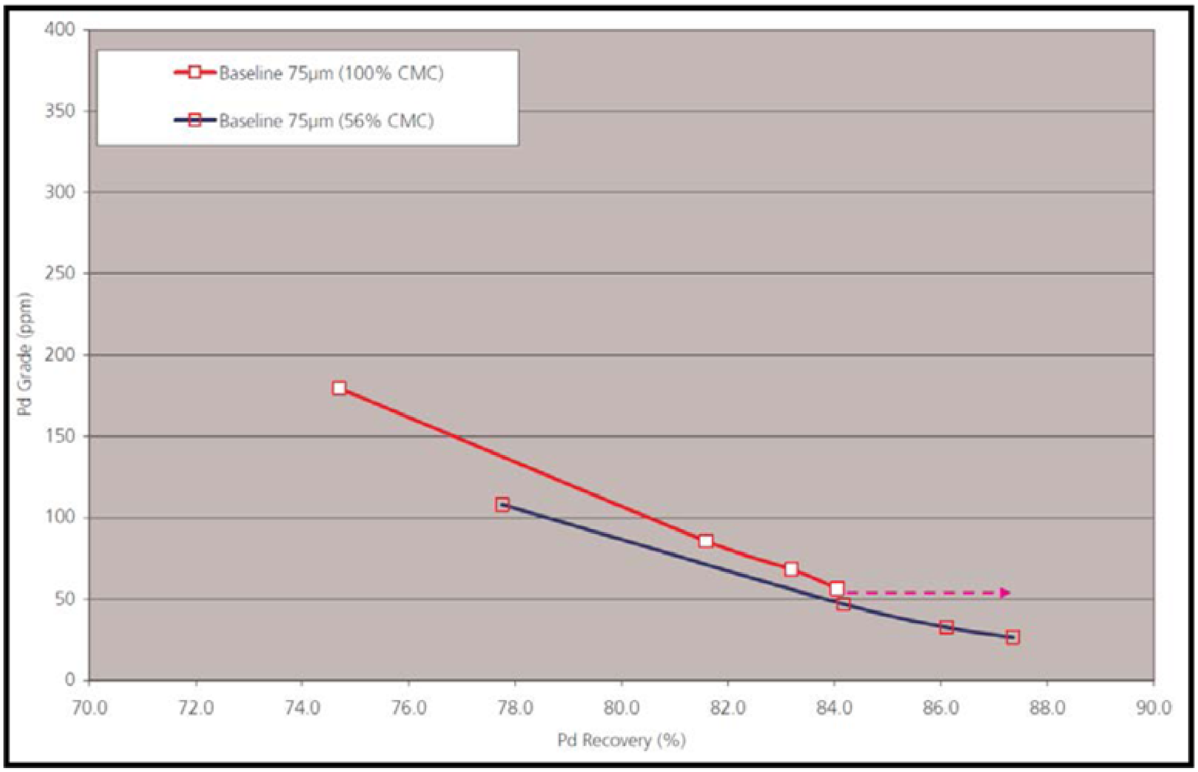


Figure 13-11: Offset Zone – Rougher Scavenger Palladium Grade/Recovery versus CMC Dosage Primary at Primary Grind of 75 µm - Source: XPS (2010)

13.3.8 Rougher Scavenger – Roby Zone Sample-Aero 3477

Rougher-scavenger tests were performed on Roby Zone samples to determine the effect of the PGM promoter Aero 3477 at a primary grind of 75 µm. The Aero 3477 was added at the prescribed dosage of 16 g/t (i.e., 8 g/t in the secondary rougher and 8 g/t in the scavenger). As shown in Figure 13-12, the addition of the PGM promoter resulted in a 2% improvement in overall palladium recovery.

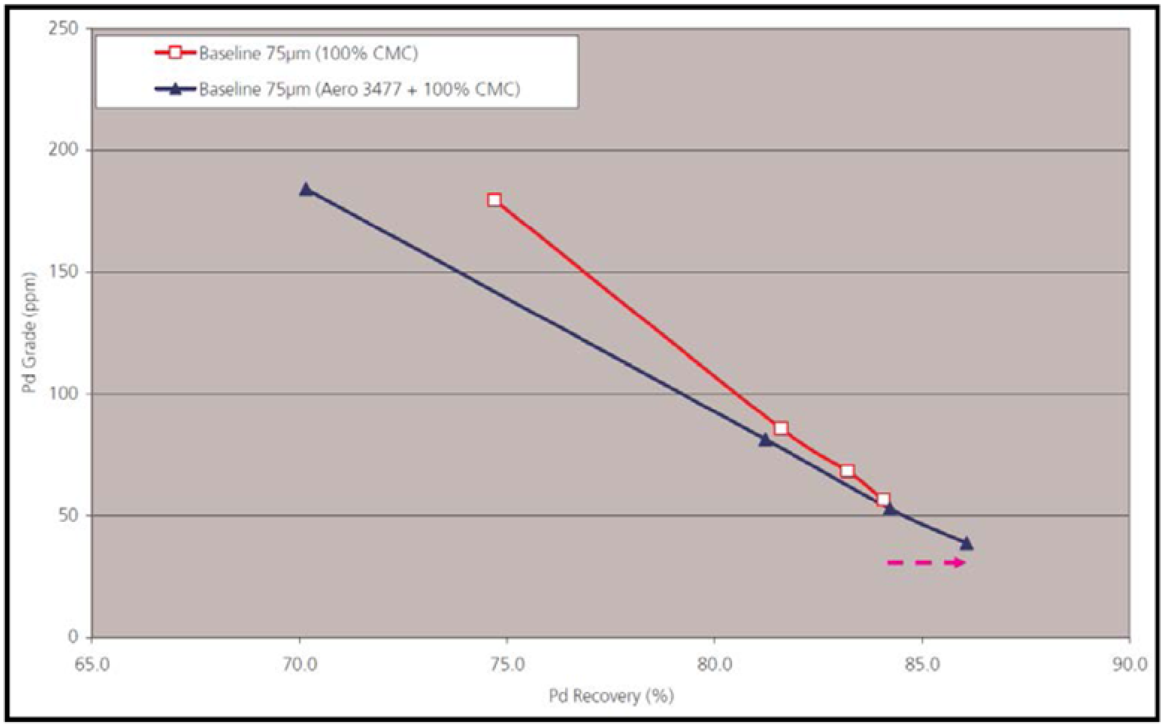


Figure 13-12: Rougher Scavenger Palladium Grade/Recovery versus Aero 3477 Dosage Primary at Primary Grind of 75 µm - Source: XPS (2010)

13.3.9 Rougher Scavenger – Roby Zone Sample – Fine Primary Grind versus Staged Grinding

A series of rougher-scavenger tests were completed on the Roby Zone sample to investigate the effect of a fine primary grind and of staged grinding on the palladium recovery. The final fine grinds chosen were 53 µm and 38 µm. For the staged grinding the primary stage grind of 75 µm was selected. Figure 13-13 contains the rougher - scavenger results for the 75 µm/53 µm-staged grinding versus the 53 µm single-stage grind, and Figure 13-14 contains the results for the 75 µm/38 µm-staged grinding versus the 38 µm single-stage grind.

From Figure 13-13 and Figure 13-14, it can be seen that the single-stage, fine grind resulted in higher palladium recoveries than the two-stage grinding.

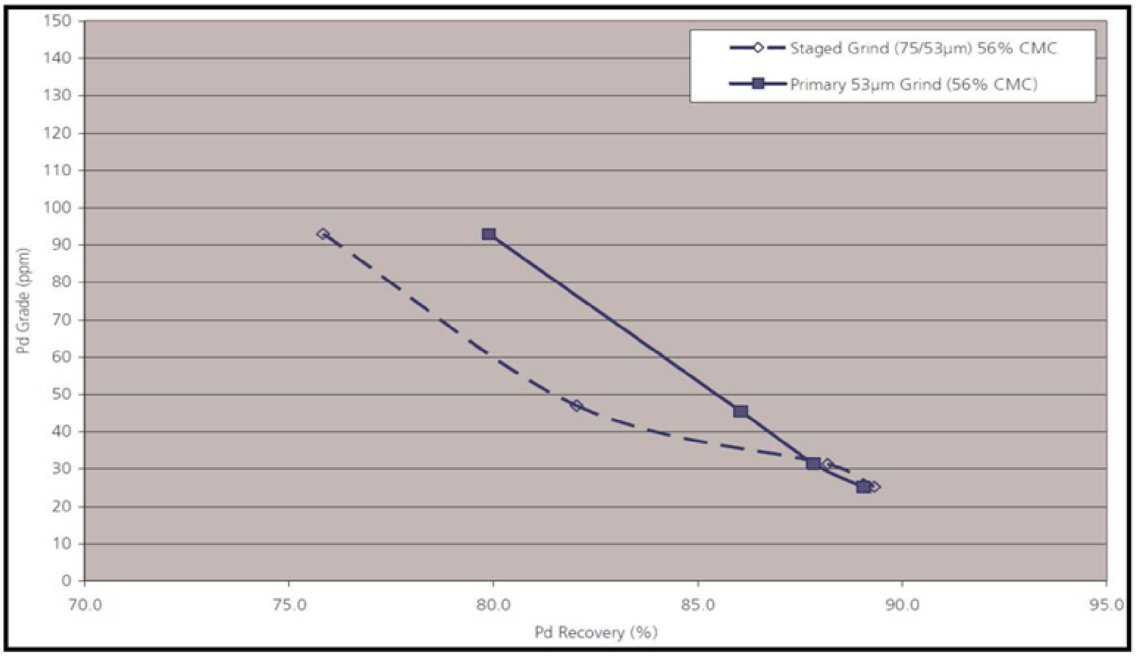


Figure 13-13: Rougher Scavenger Palladium Grade/Recovery at 53 µm Primary Grind versus Staged Grind - Source: XPS (2010)

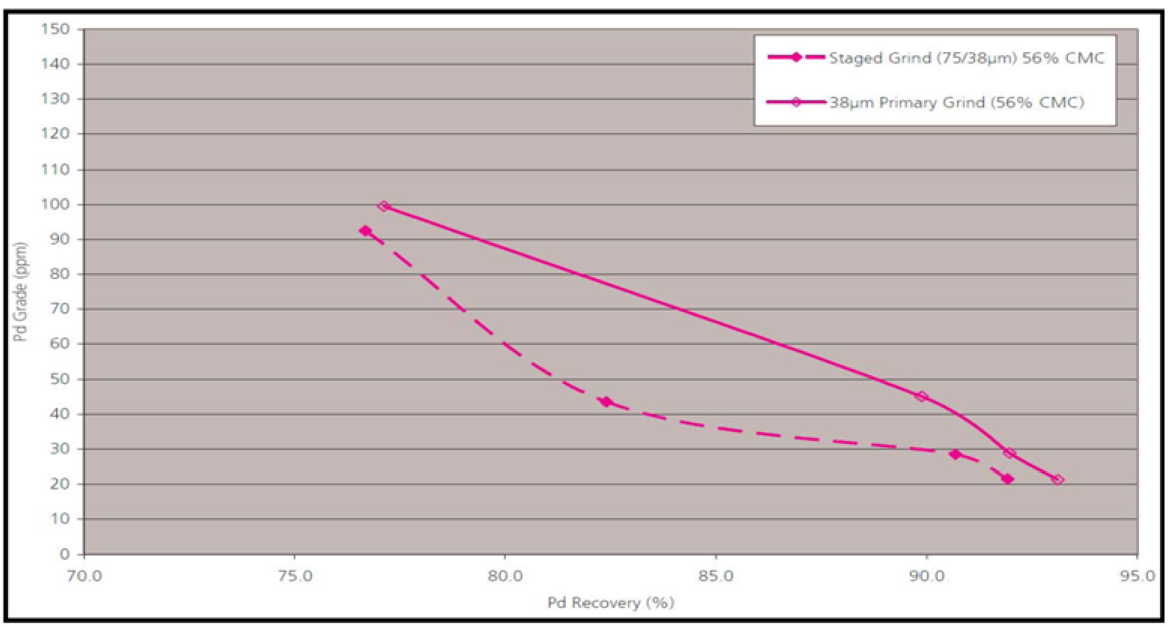


Figure 13-14: Rougher Scavenger Palladium Grade/Recovery at 38 µm Primary Grind versus Staged Grind - Source: XPS (2010)

13.3.10 *Open Circuit Cleaner Tests – Grind Sensitivity*

Cleaner flotation tests were conducted at grinds of 75 µm and 38 µm on both Roby and Offset samples. Figure 13-15 and Figure 13-16 show the results of the cleaner tests at the specified grind sizes for the Roby and Offset samples respectively. In both the Roby and Offset tests, the benefits of the finer grind to the palladium recovery can be seen.

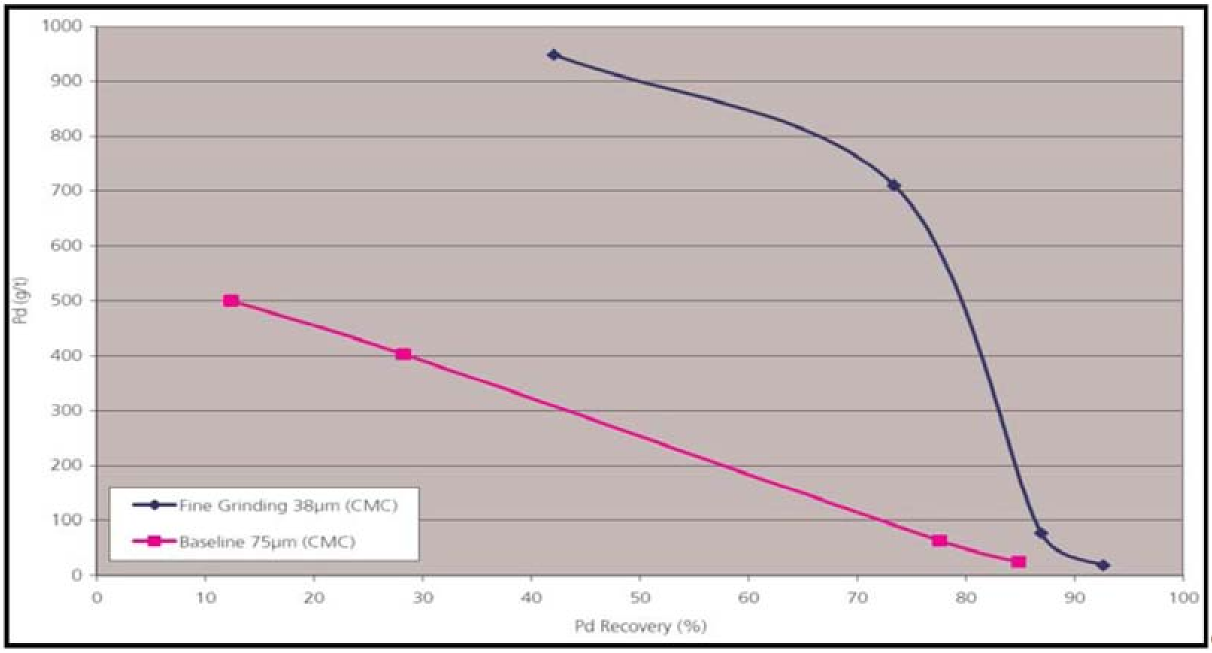


Figure 13-15: Roby Zone – Cleaner Palladium Grade/Recovery Comparing 38 µm and 75 µm Primary Grinds - Source: XPS (2010)

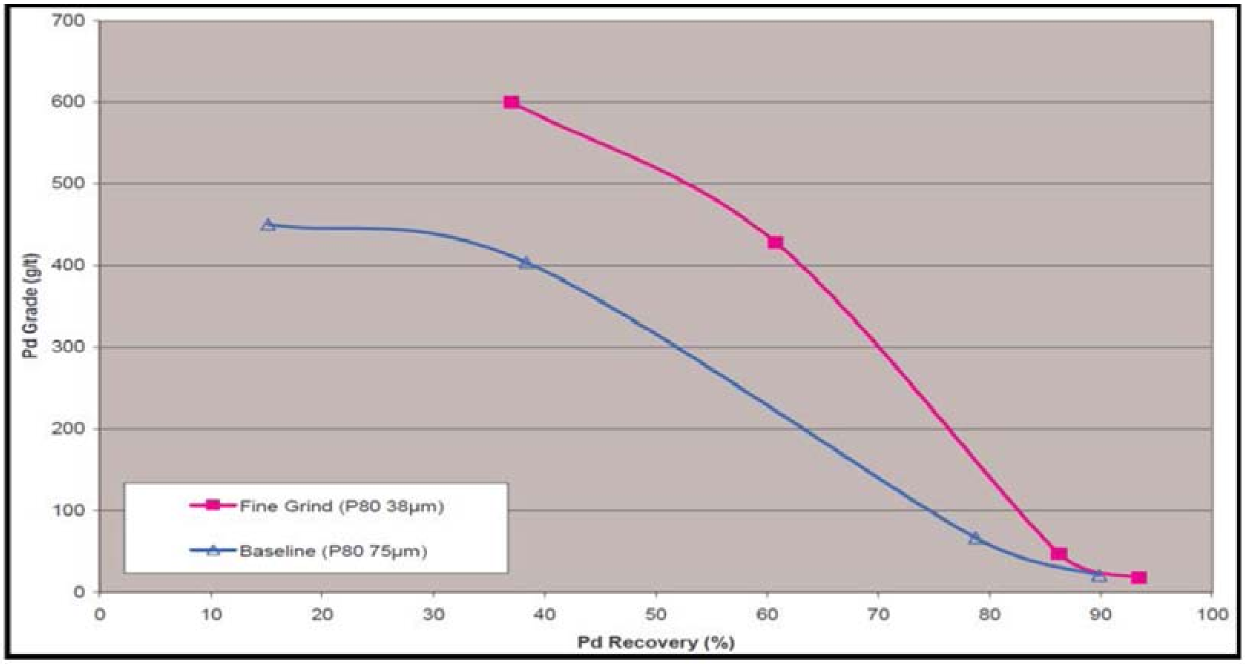


Figure 13-16: Offset Zone – Cleaner Palladium Grade/Recovery Comparing 38 µm and 75 µm Primary Grinds - Source: XPS (2010)

13.3.11 Open Circuit Cleaner Tests – Fine Grinding Versus Staged Grinding

A series of cleaner flotation tests were performed to determine the effect of fine grinding versus staged grinding as performed previously in the rougher-scavenger test work.

Figure 13-17 and Figure 13-18 show the cleaner flotation tests for the Roby and Offset samples respectively. The fine grind is at 38 µm, and the staged grinding is to 75 µm followed by 38 µm.

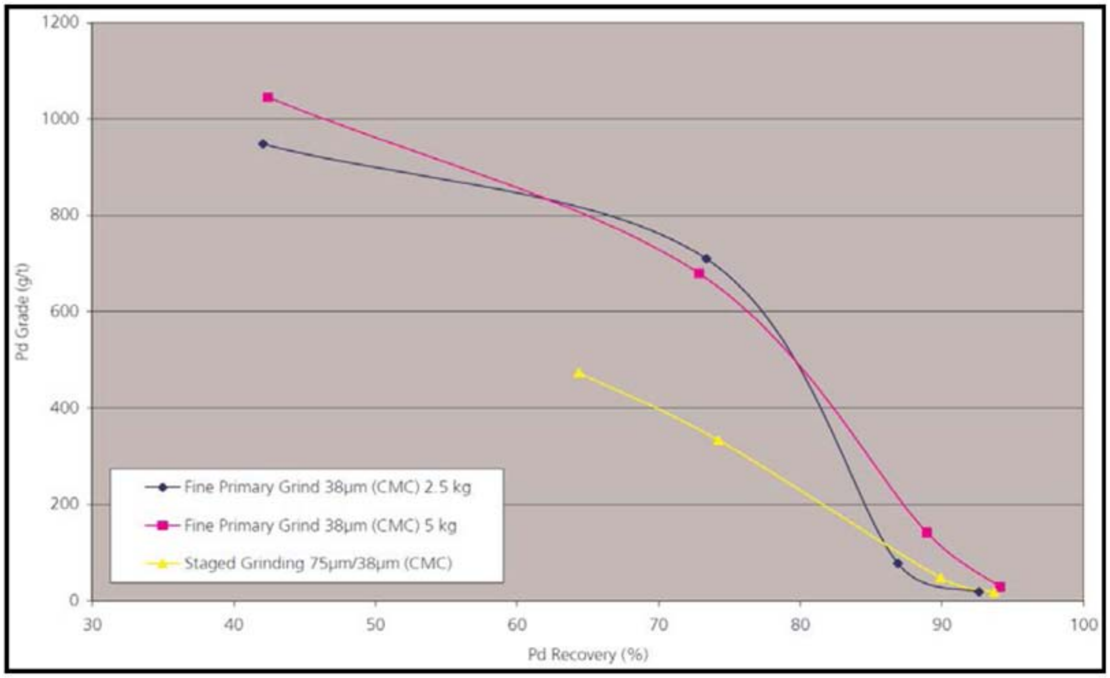


Figure 13-17: Roby Zone – Cleaner Palladium Grade/Recovery Comparing 38 µm Primary Grind versus Staged Grinding - Source: XPS (2010)

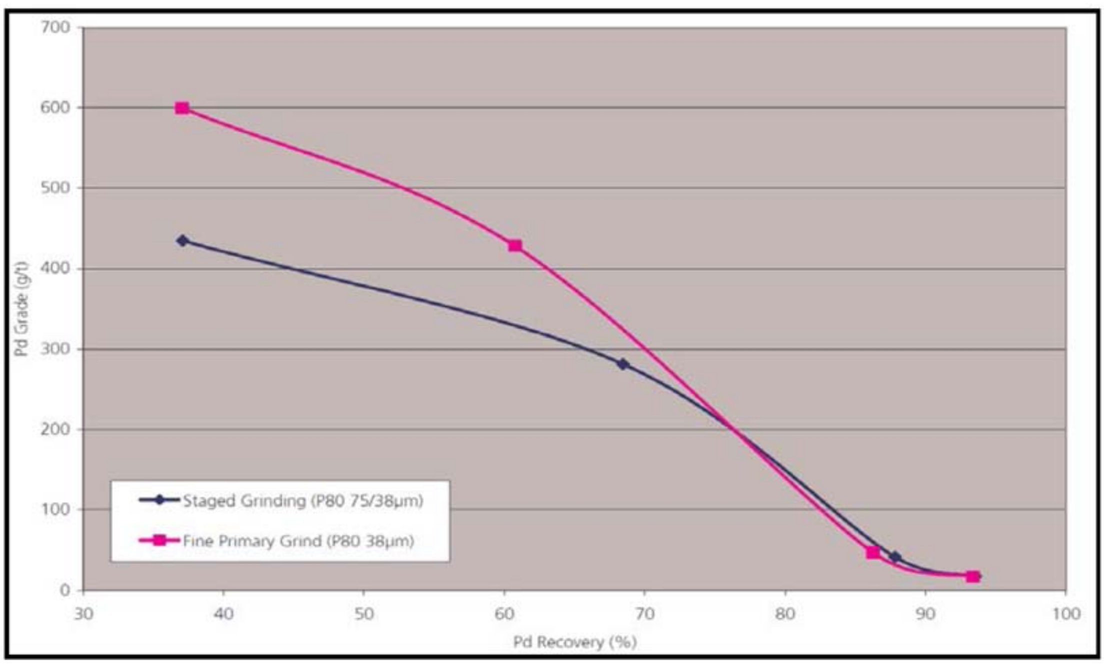


Figure 13-18: Offset Zone – Cleaner Palladium Grade/Recovery Comparing 38 µm Primary Grind versus Staged Grinding - Source: XPS (2010)

As previously shown in the rougher-scavenger tests, the staged grinding does not offer any advantage to the palladium recoveries for the open circuit cleaner flotation tests.

13.3.12 Open Circuit Cleaner – Roby Magnesium Oxide Depressant Tests

A series of cleaner tests were conducted using CMC, dextrin and guar gum as magnesium oxide depressants. The results of these tests are presented in Figure 13-19.

The CMC and dextrin work as magnesium oxide depressants without a large impact on the grade and recovery of the palladium.

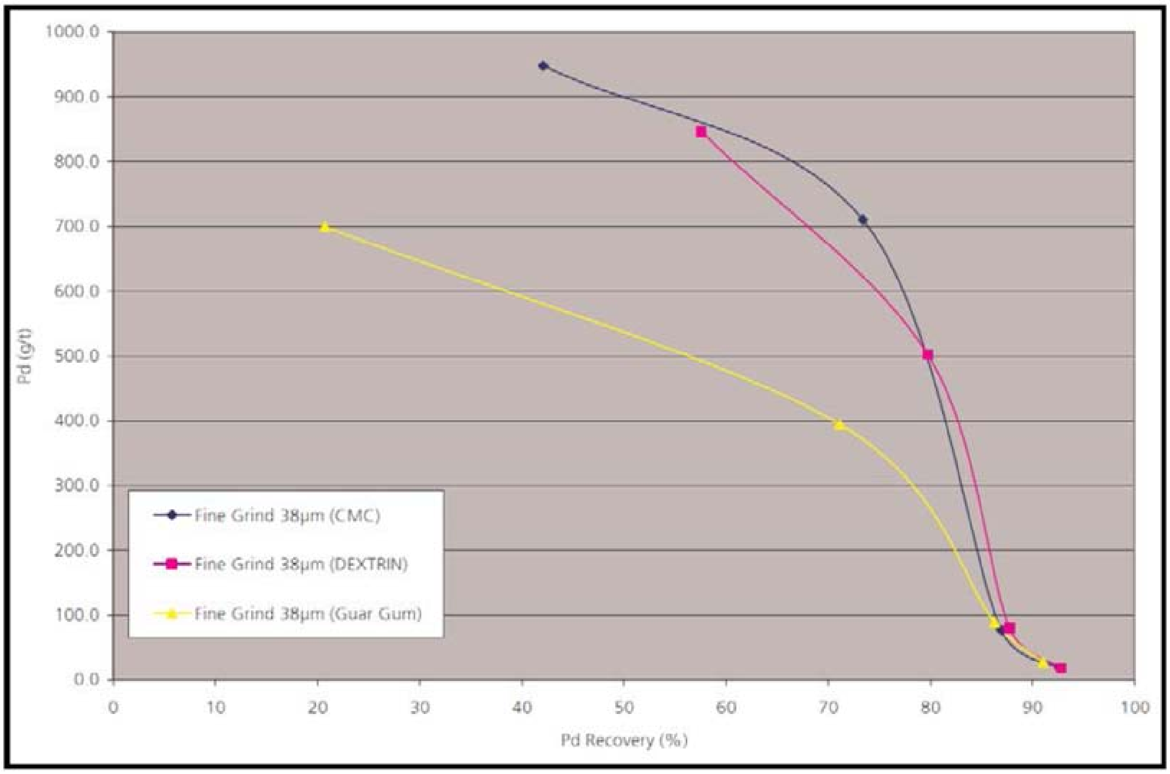


Figure 13-19: Roby Zone – Cleaner Palladium Grade/Recovery Comparing Magnesium Oxide Depressants - Source: XPS (2010)

13.3.13 Locked Cycle Tests

Using a 38 μm primary grind and the optimum reagent conditions developed in the rougher-scavenger and open circuit cleaner tests, LCTs were completed for both the Roby Zone and Offset Zone samples.

Table 13-3 shows the average projected results based on four methods of LCT analysis (i.e., two product formulas; the Society for Mining, Metallurgy, and Exploration (SME) handbook; concentrate production; and alternative). The LCT test for the Roby sample was not entirely stable but considered satisfactory. The LCT for the Offset sample was not stable and the results are unreliable.

Table 13-3: Results of LCT for Roby and Offset Zones Mineralized Material Samples

Ore Zone	Item	Pd	Pt	Au* (%)	Cu (%)	Ni (%)	MgO (%)
Roby	Concentrate Grade	664 g/t	32.8 g/t	-	7.32	5.15	8.58
	Recovery	86.2%	78.6%	87.5	86.1	35.9	0.57
Offset	Concentrate Grade	419 g/t	33.1 g/t	-	10.04	3.96	10.06
	Recovery	83.1%	86.8%	85.3	90.5	27.3	0.8

Note: *Gold assay on concentrate was not possible due to limited concentrate mass. Recovery estimated from head and tailings assay.
Source: XPS (2010)

13.3.14 Mineralized Material Hardness

Roby and Offset samples were sent to SGS for Bond Work Index (BWi) tests using standard procedures. Tests were carried out at 75 µm. The Roby and Offset samples were classified as "hard" as shown in Figure 13-20. The BWi for Roby was 17.9 kWh/t and for Offset 17.0 kWh/t.

Bond Ball Mill Grindability Test Summary

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
LDI Roby Zone Comp	150	2,285	84	1.07	17.9	80
LDI Offset Zone Comp	150	2,650	85	1.13	17.0	74

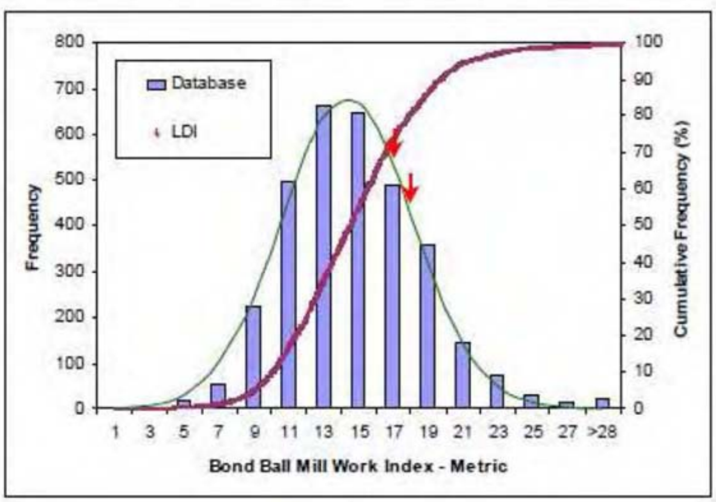


Figure 13-20: Roby and Offset BWi - Source: XPS (2010)

13.3.15 Gravity Recovery Test Work

Reject material from the mineralized material hardness testing (approximately 25 kg) of the Roby and Offset samples was sent to the Knelson Research and Technology Centre (Knelson) for gravity recoverable gold, silver, platinum, and palladium test work. The test work is accomplished by progressive particle size reduction of the material and gravity concentration at the different sizes. The test results have been summarized in Table 13-4.

Table 13-4: Gravity Recovery Test Results

Extended Gravity Recovery Test								
Grind Size (µm)	**Product**	**Mass (%)**	**Assay (g/t)**			**Recovery (%)**		
			Pd	Pt	Au	Pd	Pt	Au
P80 840 µm	Stage 1 Concentrate	0.4	183.0	12.9	47.9	11.5	13.1	33.8
P80 206 µm	Stage 2 Concentrate	0.4	185.2	11.7	13.5	11.3	11.6	9.3
P80 54 µm	Stage 3 Concentrate	0.2	152.5	11.2	18.9	6.0	7.2	8.4
Knelson Concentrate	Stage 1-3 Concentrate	1.0	176.5	12.0	0.54	28.8	31.9	51.5
Head		-	6.06	0.37	0.54	-	-	-

Source: XPS (2010)

The gravity recovery for palladium and platinum were 28.8% and 31.9%, respectively, due to the fine grain size of the PGM minerals shown in the mineralogical work (less than 25 µm). These poor gravity recoveries are as expected.

13.3.16 Flash Flotation Test Work

Test work was conducted by NAP technical staff to evaluate the feasibility of flash flotation in the grinding circuit of the LDI process plant. Flash flotation is an effective way of removing heavy minerals that have difficulty exiting a closed grinding circuit, minimizing over grinding, and alleviating the load of downstream equipment. A total of three sample sets of Vertimill® cyclone underflow were taken for each cyclone cluster that is considered for flash flotation. There are two sets of Vertimill® cyclone clusters that are being considered for flash flotation, therefore six sample sets were taken in total. The flash flotation tests were conducted with regular mill reagents.

Concentrate samples were taken at two and four minutes in order to achieve two points on the grade recovery curve. Potential recovery at 250 g/t palladium product grade could then be extrapolated. Mass recovery at the two minute mark was between 0.8 and 1.9% and at the 4 minute mark was between 1.6 and 3.4%. Mass pull to concentrate is traditionally 1 to 2%, so the test work is indicative of regular practice. The test results have been summarized in Table 13-5, Table 13-6, Table 13-7, and Table 13-8.

Table 13-5: Vertimill® Cyclone Underflow Cluster 1, Overall Recoveries

	Pd	Pt	Au	Cu	Ni	MgO	SiO_2	Al_2O_3	Fe_2O_3	S
Feed Grade	16.17 g/t	0.82 g/t	2.48 g/t	0.08%	0.13%	7.89%	44.0%	14.27%	8.9%	0.46%
Lowest Recovery	87%	82%	79%	71%	74%	18%	18%	17%	21%	100%
Highest Recovery	97%	95%	99%	91%	91%	73%	74%	74%	75%	100%
Average Recovery	94%	90%	92%	88%	79%	51%	51%	51%	53%	100%

Table 13-6: Vertimill® Cyclone Underflow Cluster 1, Individual Recoveries (%)

	Pd	Pt	Au	Cu	Ni	MgO	SiO_2	Al_2O_3	Fe_2O_3	S
Sample Set 1 Recoveries	94	91	96	91	85	64	64	64	66	100
Sample Set 2 Recoveries	93	89	93	83	75	34	34	34	37	100
Sample Set 3 Recoveries	94	90	86	89	76	50	50	49	52	100

Table 13-7: Vertimill® Cyclone Underflow Cluster 2, Overall Recoveries (%)

	Pd	Pt	Au	Cu	Ni	MgO	SiO_2	Al_2O_3	Fe_2O_3	S
Feed Grade	9.14 g/t	0.51 g/t	2.16 g/t	0.07 %	0.10 %	6.26 %	44.36 %	15.75 %	7.45 %	0.41 %
Lowest Recovery	87%	80%	93%	71%	50%	7%	4%	2%	9%	97%
Highest Recovery	95%	92%	97%	89%	79%	57%	57%	57%	60%	100%
Average Recovery	92%	88%	96%	84%	68%	32%	32%	32%	36%	100%

Table 13-8: Vertimill® Cyclone Underflow Cluster 2, Individual Recoveries (%)

	Pd	Pt	Au	Cu	Ni	MgO	SiO_2	Al2O_3	Fe_2O_3	S
Sample Set 1 Recoveries	91	87	95	84	68	27	27	27	30	100
Sample Set 2 Recoveries	91	85	96	79	62	25	25	25	29	100
Sample Set 3 Recoveries	93	91	96	87	73	42	41	41	45	99

The average palladium recovery at a concentrate grade of 250 g/t was determined to be 93%. This value ranged from 87 to 97% throughout the tests.

The recoveries of other selected minerals are:

- Platinum: 88 to 90%.
- Gold: 92 to 96%.
- Copper: 84 to 88%.
- Nickel: 68 to 79%.

These recoveries are promising for a flash flotation.

13.4 Representativeness

The test samples were taken from the Roby and Offset zones and were deemed representative of the mineralization in these two different deposits by LDI personnel. According to LDI, the samples were taken from different areas and at different elevations so that they would be representative of the deposit. The identification and selection of the samples was by others, and it was not in Hatch's scope to determine the representativeness of those samples to the overall ore body.

There were three sets of samples taken for each Vertimill® cyclone underflow for the flash flotation test work. The Vertimill® cyclone underflow cluster No. 1 samples were taken on July 14, 2012 at 2:00 p.m., July 15, 2012 at 8:00 a.m., and July 15, 2012 at 11:00 a.m. The Vertimill® cyclone underflow cluster No. 2 samples were taken on July 16, 2012 at 6:30 a.m., July 16, 2012 at 9:30 a.m., and July 16, 2012 at 2:30 p.m.

13.5 Processing Factors or Deleterious Elements

As shown in the test program completed by XPS, the major issue which could have a considerable effect on the concentrate grades is magnesium oxide flotation. This can be controlled through the use of magnesium oxide depressants such as CMC. The use of this depressant in operations was proven as being effective and the test work reinforced this.

14. Mineral Resource Estimates

14.1 Offset Zone

14.1.1 Introduction

NAP has updated the resource model for the Offset Zone at the LDI property. The new resource model described below has an effective date of December 31, 2014. The resource estimate was completed using GEOVIA GEMS™ 6.7 Desktop resource estimation software.

This resource model supersedes the previous resource models of the Offset Zone completed by Tetra Tech in February 2013 (effective date of March 31, 2012; McCracken et al., 2013), and by NAP in March 2014 (effective date of December 31, 2014; McKinnon et al., 2014).

Assumptions for the resource tables presented in this section are discussed in the footnotes for Table 1-1 and in the relevant, supporting documentation for each zone described in this section.

14.1.2 Database

NAP maintains all borehole data in a Century Systems Fusion Server®. All drillhole information, including the header, survey, assay, and lithology tables, are saved in the database. The database used for the current resource model was updated as of December 31, 2014.

The complete LDI database used in the current resource model contained data for 2,132 boreholes. The database includes all available surface and underground drillholes for the Roby underground and Offset Zones. All drillholes that were logged as non-compliant because the samples were analyzed at the mine laboratory were removed from the dataset. In 2013, NAP's Exploration Department identified 26 surface drill holes that were drilled in 2011 and surveyed using a specific tool, the Icefield tool, that have a different deviation profile than the majority of the surface drill holes on the Property. New survey tests were done and these showed a significant error in the initial Icefield tool survey data. Five of these holes intersected the Offset zone geological wireframe, two were resurveyed (11-019 and 11-024) and three were removed from the dataset (11-002, 11-003 and 11-023). As the same tool has been used for surveying underground holes, an internal study has been completed on all the holes surveyed using the Icefield tool. As a precaution, the Company has decided to remove from the dataset all holes longer than 450m and all holes that deviated more than 10m from a straight line oriented to the collar position. This led to the removal of 35 (14%) of the available 251 underground drill holes surveyed using the Icefield tool for the Offset zone, or 1.64% of the complete drill hole database comprising 2,132 drill holes. After these exclusions, the compliant database contains 1,732 boreholes, of which 655 intersect the Offset zone geological wireframe and were used for the resource estimate described below.

14.1.3 Specific Gravity

The LDI routinely collects specific gravity data during the logging of drill core. The methodology involves the weighing of dry core on a scale, and then weighing the core suspended in water. There is limited specific gravity data available on the LDI. A specific gravity of 2.89 was used for the resource estimate, which is the same number used in the previous estimate (McKinnon et al., 2014). A specific gravity of 2.89 is within the accepted range of a gabbro with low-sulphide content.

14.1.4 Geological Interpretation

For the previous (2013) resource estimation (McCracken et al., 2013), Tetra Tech developed two 3-dimensional (3D) wireframe models of mineralization using Datamine™ software. The basis for each wireframe included a minimum downhole width of 2.0 m, a minimum waste inclusion of 0.5 m downhole, and a minimum grade of 1.0 g/t palladium. The second wireframe was created within this 1.0 g/t palladium wireframe, where a minimum grade of 2.0 g/t palladium was captured.

The 1.0 g/t palladium wireframe was updated from the original contour lines provided by Tetra Tech. NAP imported these lines into GEOVIA GEMS™ 6.7 Desktop software, and the contour lines were updated based on the more recent drillhole database. A new 1.0 g/t wireframe solid was created using these contour lines.

Three other wireframe solids located within the 1.0 g/t palladium wireframe solid were created using Leapfrog® software. Palladium assays from the drillholes inside the 1.0 g/t palladium wireframe were used to create 1.75 g/t, 2.00 g/t and 2.25 g/t palladium wireframes (see Figure 14-1).

The 1.0 g/t wireframe has been split into five different domains. These domains are strictly geometric and are not based on any lithological information. Each domain has a general spatial orientation, and a unique “rock type number” has been assigned to each (see Figure 14-2).

The zones of mineralization interpreted for each area were generally contiguous however, due to the nature of the mineralization, there are portions of the wireframe that have grades less than 1.0 g/t palladium, yet are still within the mineralizing trend. The wireframes were trimmed to the Offset Fault in order to avoid an estimation of material above the fault plane – which is considered by NAP to represent a hard boundary between the Roby Zone deposit (Roby block) and the Offset Zone deposit (Offset block).

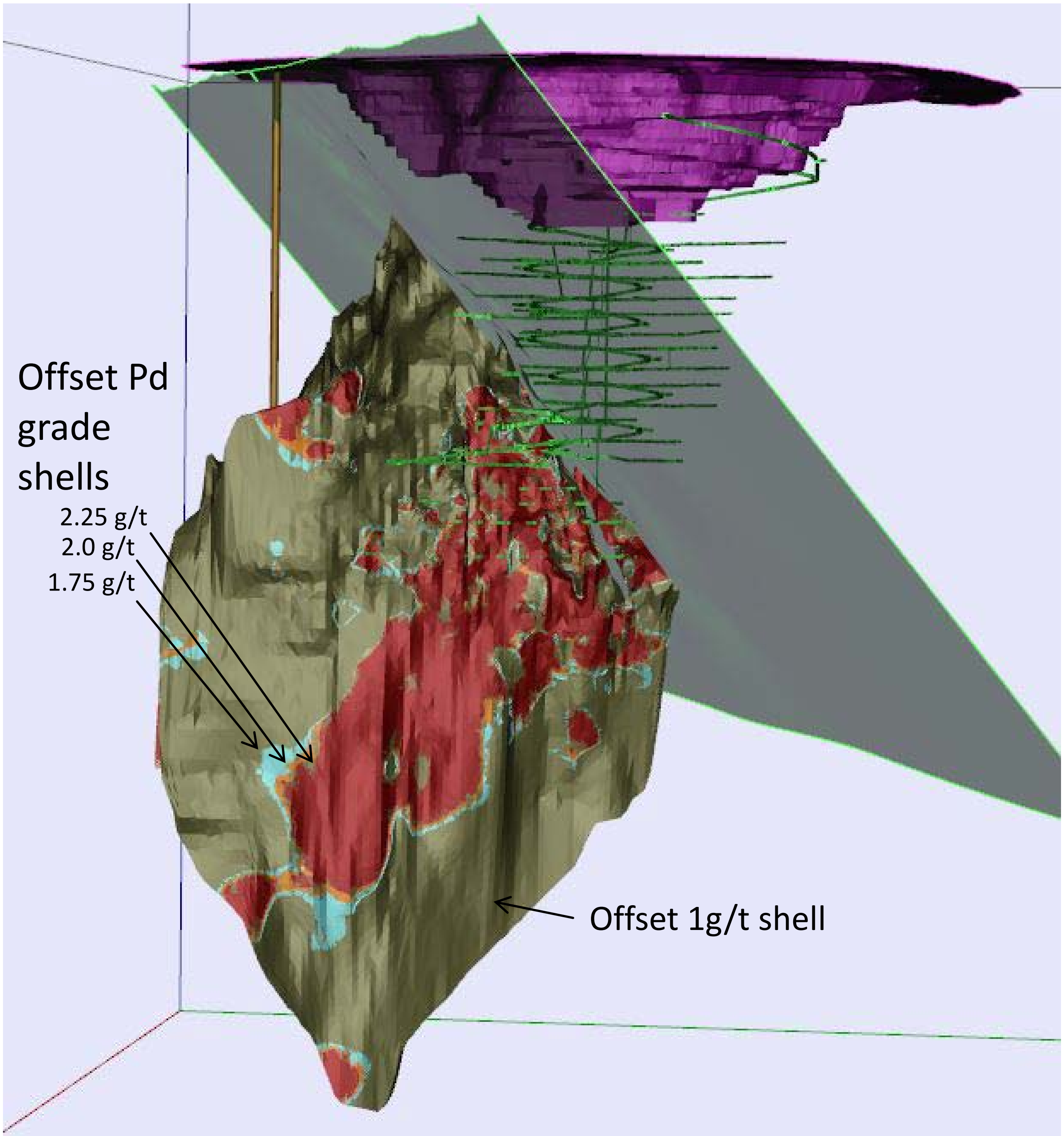


Figure 14-1: Oblique View (looking west) of the Offset Zone Palladium Grade Wireframes

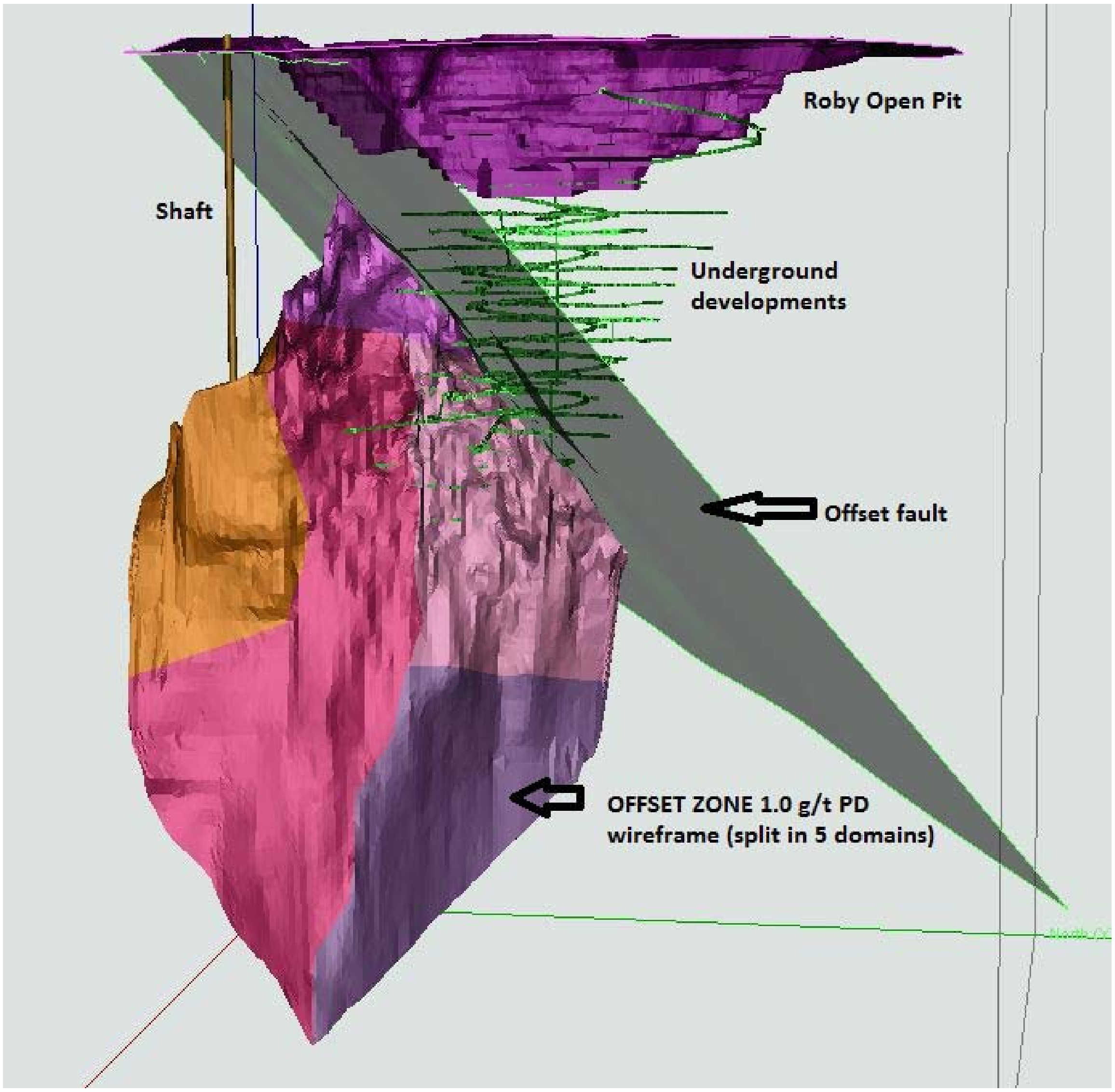


Figure 14-2: Oblique View (looking west) of Offset Zone 1.0 g/t Palladium Mineral Wireframe Split into Five Domains

14.1.5 Exploratory Data Analysis

14.1.5.1 Assays

The portion of the deposit included in the mineral resource was based on a total of 65,211 palladium assays. The assay intervals within each zone were captured using GEOVIA GEMS™ cross-table manipulation into individual borehole files. These borehole files were reviewed to ensure that all relevant assay intervals were captured. Table 14-1 summarizes

Table 14-1: Summary of Offset Zone Assay Basic Statistics

Variable	Length	PD	PT	AU	NI	CU	CO
Number of samples	65,240	65,211	65,211	65,211	65,206	65,209	65,209
Minimum value	0.00	0.00	0.00	0.00	0.0001	0.0001	0.0001
Maximum value	3.90	58.4	4.17	5.44	1.23	1.59	0.03
	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data	Ungrouped Data
Mean	1.01	1.92	0.18	0.14	0.08	0.05	0.006
Median	1.00	0.90	0.11	0.06	0.06	0.03	0.006
Geometric Mean	Not Calculated	Not Calculated	Not Calculated	Not Calculated	0.056	0.03	0.005
Variance	0.14	8.54	0.04	0.05	0.004	0.004	0.00001
Standard Deviation	0.38	2.92	0.20	0.23	0.06	0.07	0.002
Coefficient of variation	0.37	1.52	1.12	1.62	0.81	1.20	0.40
Moment 1 About Arithmetic Mean	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Moment 2 About Arithmetic Mean	0.14	8.53	0.04	0.05	0.004	0.004	0.00001
Moment 3 About Arithmetic Mean	0.13	98.0	0.02	0.06	0.0006	0.0008	0.00
Moment 4 About Arithmetic Mean	0.40	2120	0.03	0.15	0.0003	0.0004	0.00

the basic statistics for the assays in the Offset Zone for the 1 g/t wireframe. The non-assayed intervals were assigned void (-) values. NAP believes that non-assayed material should not be assigned a zero value, as this does not reflect the true value of the material.

14.1.5.2 Compositing

Compositing of all assay data was completed within the 1 g/t palladium wireframe. A composite interval of 1 m was used, the same composite interval that was used in previous resource estimates. The 1 m composite intervals were extracted to points and classified by grade shell wireframes. The different class of composites are: 1.00 to 1.75 g/t palladium domain, 1.75 g/t to 2.00 g/t palladium domain, 2.00 g/t to 2.25 g/t palladium domain and >2.25 g/t palladium domain. It is important to understand that the class of composites depends upon which wireframe (see Section 14.1.4) they occur within, and not the actual palladium grade of the composite.

14.1.5.3 Grade Capping

In 2014 (McKinnon et al., 2014), composite data for the 1 g/t palladium wireframe was examined individually to assess the amount of metal that is at risk from high-grade assays. The Parrish analysis (Parrish, 1997) was used to determine if grade capping was required for each element.

A maximum value of 30 g/t for palladium and 3.0 g/t for gold was assigned for composites with higher values. No capping has been applied on the other metals (Pt, Ni, Cu) considered in the current resource estimate.

In 2015, the addition of new assay values represents less than 5% of the total database. All palladium assay values, expect one value, are below 25 g/t. All 2015 gold assay values report below 2.7 g/t for Au. For these reasons, the criteria for grade capping used for the 2014 estimate (McKinnon et al., 2014) were considered to be adequate for the 2015 estimate.

14.1.6 Spatial Analysis

For the previous mineral resource estimate for the Offset Zone (McCracken et al., 2013), Tetra Tech completed a variography analysis using Datamine™ software on a global basis for each element for all the composited data. Downhole variograms were used to determine nugget effect and then correlograms were modelled to determine spatial continuity in the zones.

For the current resource estimate, all variograms were produced with the new composite data using GEOVIA GEMS™ software (see Figure 14-3 to Figure 14-7) by NAP As no significant change has been observed between the variography from the last mineral resource estimate and the variography for the current composite database, the variogram parameters for each element provided by Tetra Tech were considered as valid and up to date for the current mineral resource estimate.

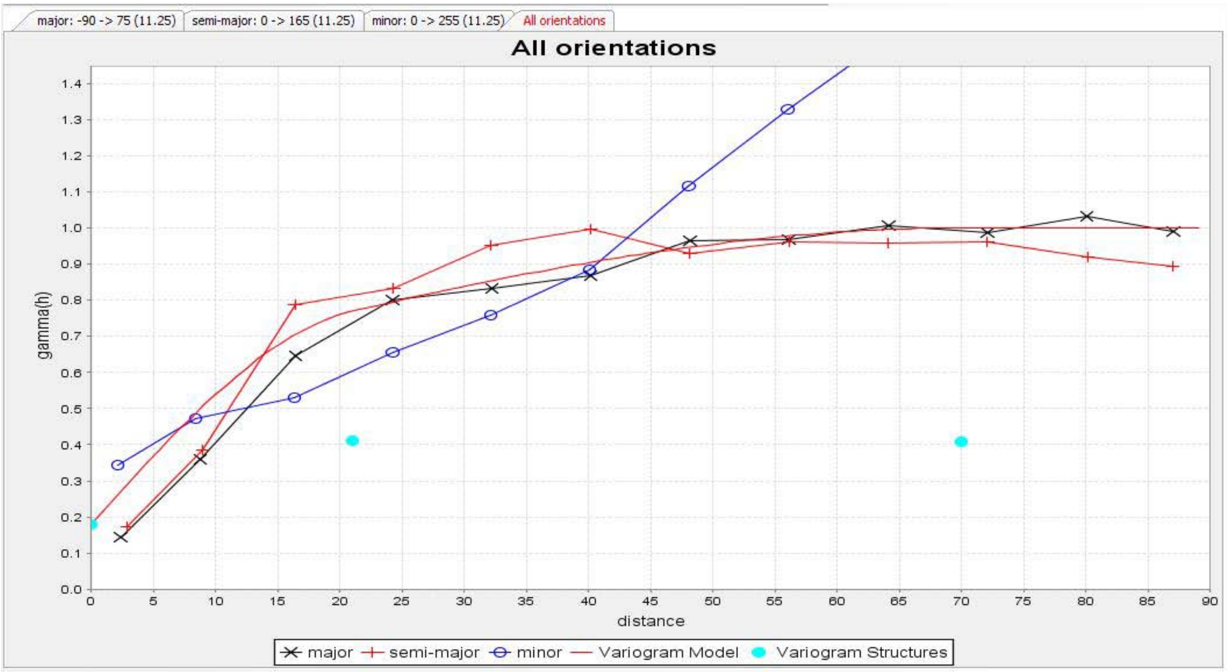


Figure 14-3: Palladium Variogram

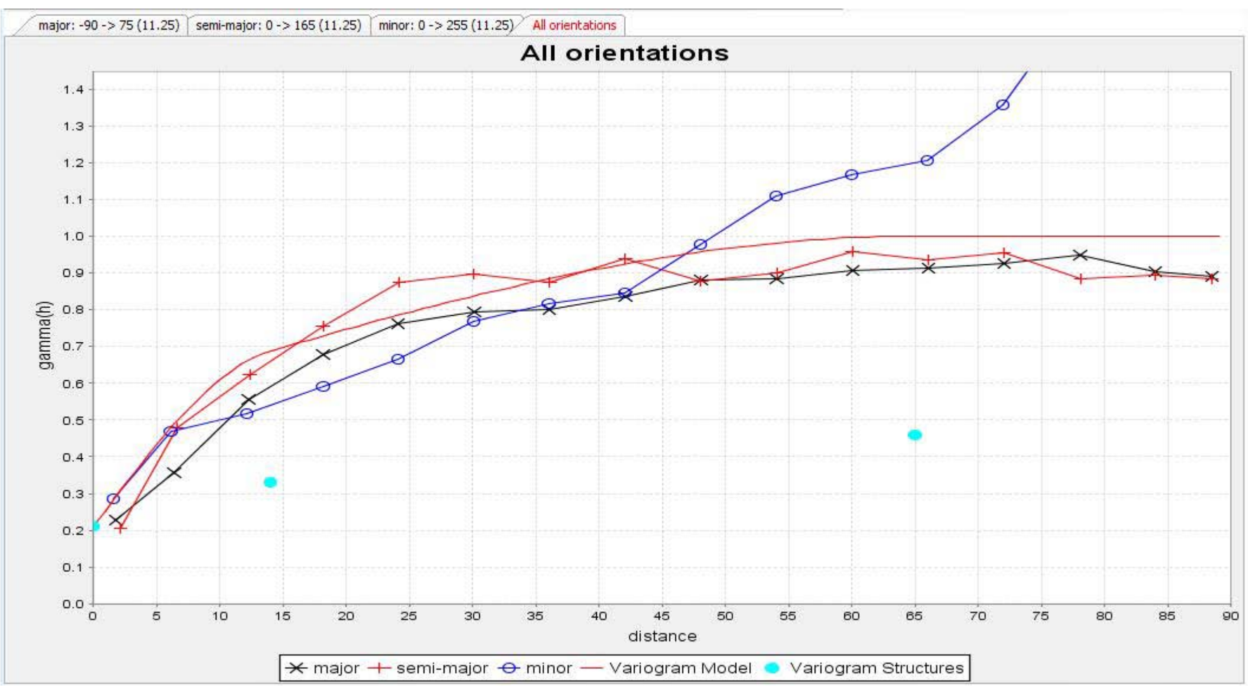


Figure 14-4: Platinum Variogram

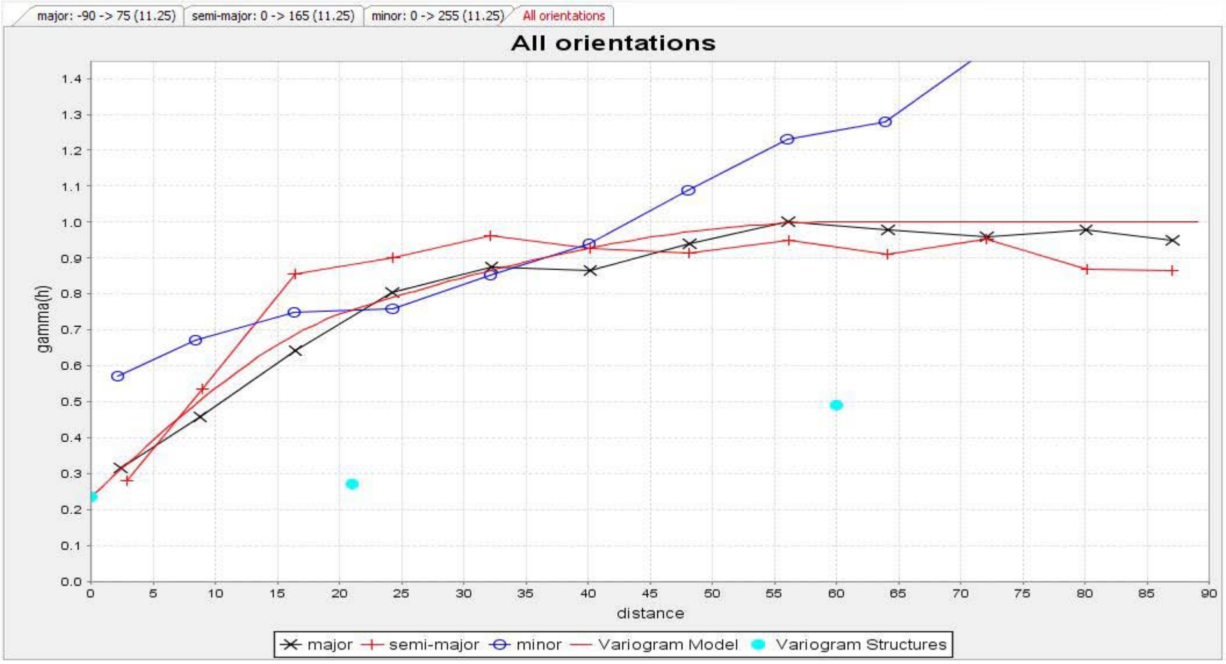


Figure 14-5: Gold Variogram

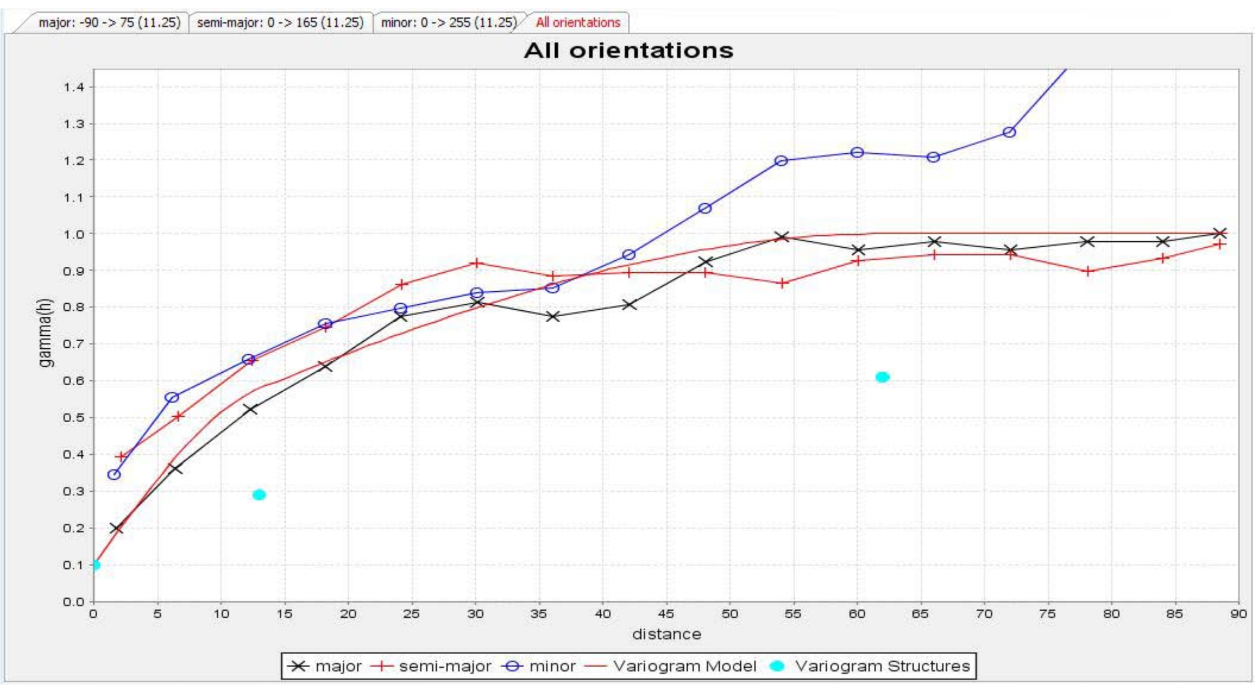


Figure 14-6: Nickel Variogram

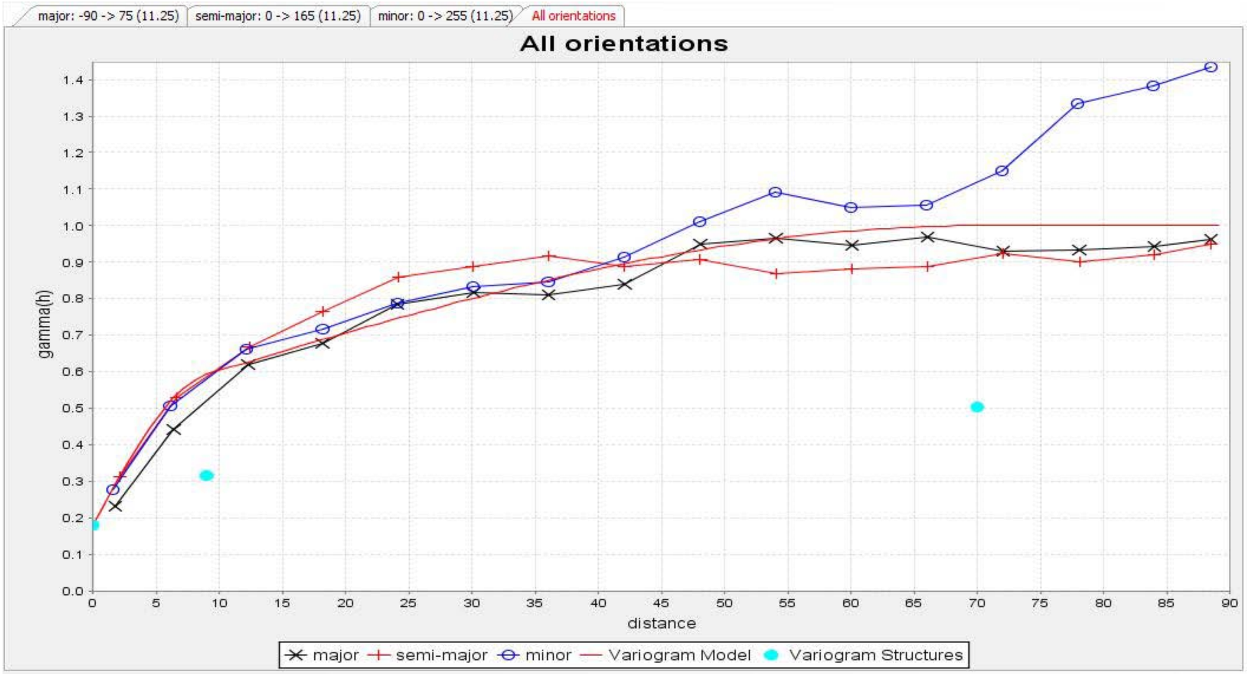


Figure 14-7: Copper Variogram

Table 14-2 shows the variogram parameters used for the previous estimate by Tetra Tech in Datamine™. Table 14-3 shows the variogram parameters as they were entered into GEOVIA GEMS™.

Table 14-2: Normalized Variogram Parameters from Datamine™ Tetra Tech Offset Model

Datamine	Metal				C0	C1				C2				Sill
	VDESC	VANGLE1	VANGLE2	VANGLE3	NUGGET	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4	
	pd	-15	0	180	0.18	8	16	21	0.411	30	50	70	0.409	1
	pt	-15	0	180	0.21	8	8	14	0.331	34	37	65	0.459	1
	au	-15	0	180	0.24	10	15	21	0.272	45	40	60	0.492	1
	cu	-15	0	180	0.18	8	9	9	0.316	40	45	70	0.504	1
	ni	-15	0	180	0.1	8	8	13	0.29	40	44	62	0.61	1
	co	-15	0	180	0.1	8	10	13	0.348	47	49	80	0.552	1
	ag	-15	0	180	0.57	14	3	17	0.035	40	11	40	0.395	1
	fe	-15	0	180	0.1	13	8	17	0.438	60	80	90	0.462	1
	s	-15	0	180	0.25	8	8	12	0.332	80	70	90	0.418	1

Table 14-3: Normalized Variogram Parameters used in GEOVIA GEMS™ for the Current Resource Estimate

Gems	Metal				C0	C1				C2				Sill
		Z	X	Z	Nugget	Range X	Range Y	Range Z	C1	Range X	Range Y	Range Z	C2	
	pd	15	0	180	0.18	8	16	21	0.411	30	50	70	0.409	1
	pt	15	0	180	0.21	8	8	14	0.331	34	37	65	0.459	1
	au	15	0	180	0.24	10	15	21	0.272	45	40	60	0.492	1
	cu	15	0	180	0.18	8	9	9	0.316	40	45	70	0.504	1
	ni	15	0	180	0.1	8	8	13	0.29	40	44	62	0.61	1

14.1.7 Resource Block Model

A single block model was created to cover the currently interpreted extent of the Offset Zone. Table 14-4 shows the GEOVIA GEMS™ coordinates for the block model origins. The coordinate system used is a local (mine) metric system for the Northing and Easting coordinates. However, elevations are in metres above sea level. A block size of 5m x 5m x 5m was used for block modeling and resource estimation in adherence to the previous block model (McKinnon et al., 2014).

Table 14-4: Block Coordinates for the Offset Block Model

	Minimum	Maximum	Number
Easting	31000	33000	400 columns
Northing	31000	33000	400 rows
Elevation	-1300	300	320 levels

Two different folders were created in the block model. The first folder is for the interpolation of blocks inside of the 1.00 g/t and 1.75 g/t palladium wireframe, exclusive of blocks inside the 2.00 g/t and 2.25 g/t palladium wireframe (the “1.0 g/t shell folder”). The second folder is for the interpolation of blocks inside the 2.00 g/t and 2.25 g/t palladium wireframe (the “2.00 g/t shell folder”). A third “standard” folder was also created in order to reflect the average grade for each block inside the whole mineralized wireframe.

14.1.7.1 Estimation and Search Parameters

The interpolations of the zones were completed using the following estimation methods: Ordinary Kriging (OK), Anisotropic Inverse Distance (ID2) and Nearest Neighbour (NN). The resource estimates reported in this section are from OK interpolation and where ID2 and NN were used in validation of the block model. The estimations were designed for three passes (Table 14-5). In each pass a minimum and maximum number of samples were required as well as a maximum number of samples from a borehole in order to satisfy the estimation criteria.

Table 14-5: Description of Interpolation Passes

	Min. Comps.	Max. Comps.	Maximum Samples per Drillhole	Min. No. of Drillholes	Max. No. of Drillholes	Minimum No. of Octants	Min. Samples per Octant	Max. Samples per Octant
Pass 1	10	50	5	2	10	2	1	10
Pass 2	8	40	5	2	8	2	1	10
Pass 3	6	40	5	2	8	2	1	10

Estimation runs were completed in two steps. The first step involved the estimation on the blocks inside the 1.0 g/t shell folder. For this block folder, the composite points used are those that are inside the 1.0 to 2.25 g/t shell domain (composites from the 2.25 g/t shell were excluded). The second step was the estimation of the blocks inside the 2.0 g/t shell folder. For this block folder, the composite points used are those that are inside the 1.75 g/t, 2.0 g/t and 2.25 g/t shells (composite from the 1.0 g/t shell that were not inside another shell were excluded). As a block could have different values for each folder, a percent attribute was added in each folder. The percent value represents the percentage of the block that is inside each shell domain. A consolidated model was made in the “Standard” folder, where the value for each grade is the average value for the grade in the 1.0 g/t shell folder and the 2.0 g/t shell folder balanced by their respective percentage. This allowed the higher-grade hanging wall domain to be preserved and eliminated the potential for grade smearing across strike.

A search ellipse direction has been defined for each of the 5 domains of the mineralized wireframe (defined in Section 14.1.4 and shown in Figure 14-2). Each block inside the mineralized wireframe has a rock type number assigned depending on which of the 5 domains it belongs to. This allows using the proper search ellipse direction for each block. Table 14-6 shows the different search ellipse directions for each domain.

Table 14-6: Search Ellipse Direction for Each Domain (Rock Type Number)

Rock Type Number	Rotation			Search
	Z	Y	Z	Type
1	22	0	180	Octant
2	2	0	180	Octant
3	12	-4	180	Octant
4	-7	-13	180	Octant
5	-16	-7	180	Octant

Table 14-7 shows the first pass search ellipse ranges for all metals. Pass 2 uses two times the range of the first pass and Pass 3 uses four times the range of the first pass.

Table 14-7: Search Ellipse Parameters for GEMS for Pass 1

Gems	Metal	Range X (m)	Range Y (m)	Range Z (m)	Search Type
Pass 1	Pd	9	25	35	Octant
Pass 1	Pt	10	18.5	32.5	Octant
Pass 1	Au	13.5	20	30	Octant
Pass 1	Cu	12	22.5	35	Octant
Pass 1	Ni	12	22	31	Octant

14.1.8 Resource Classification

Several factors are considered in the definition of a resource classification:

- NI 43-101 requirements.
- Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines.
- The author's experience with ultramafic-mafic hosted PGE-Cu-Ni deposits such as those present in the Mine Block intrusion on the LDI property (see Sections 7 and 8 in this report).
- Spatial continuity based on variography of the assays within the drillholes.
- Borehole spacing and estimation runs required to estimate the grades in a block.
- Observed mineralization in underground development.
- The number of samples and boreholes used in each of the block estimations.

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to NAP that may affect the estimate of mineral resources. As per NI 43-101 guidelines, mineral resources, which are not mineral reserves, have not demonstrated economic viability.

14.1.9 Mineral Resource Tabulation

In 2014, the Offset zone resource was reported in two different zones, the Offset Hangingwall zone and the Offset Footwall zone (McKinnon et al., 2014). In 2015, those two zones are still used for resource reporting. Each zone has been recently updated based on the new drillhole information (Figure 14-8).

The resource reported as of December 31, 2014 has been tabulated by palladium cut-off grade. Table 14-8 to Table 14-10 present the grade tonnage tables for the Offset Zone for each resource category and each zone. The resources are tabulated using various cut-off grades to demonstrate the robust nature of the resource. The resources presented here are inclusive of any reserve previously defined (McKinnon et al., 2014), and all mined-out material as of December 31, 2014 has been excluded.

Table 14-8: Offset Zone Measured Resource Grade-Tonnage Table

ZONE	Cut-off Grade	Tonnage	PD	PT	AU	NI	CU
		T x 1000	g/t	g/t	g/t	%	%
Offset Hangingwall	3.0 g/t PD	5,255	5.24	0.348	0.342	0.127	0.096
	2.9 g/t PD	5,440	5.16	0.345	0.339	0.126	0.095
	2.8 g/t PD	5,614	5.09	0.342	0.336	0.126	0.095
	2.7 g/t PD	5,802	5.01	0.338	0.332	0.125	0.094
	2.6 g/t PD	6,004	4.93	0.335	0.329	0.124	0.094
	2.5 g/t PD	6,186	4.86	0.332	0.326	0.124	0.093
	2.4 g/t PD	6,358	4.80	0.329	0.323	0.123	0.093
	2.3 g/t PD	6,540	4.73	0.326	0.320	0.122	0.092
	2.25 g/t PD	6,621	4.70	0.324	0.319	0.122	0.092
	2.2 g/t PD	6,706	4.67	0.323	0.317	0.122	0.092
	2.1 g/t PD	6,853	4.61	0.321	0.314	0.121	0.091
	2.0 g/t PD	7,001	4.56	0.318	0.312	0.121	0.091
	1.9 g/t PD	7,136	4.51	0.316	0.310	0.120	0.090
	1.8 g/t PD	7,261	4.46	0.314	0.307	0.119	0.090
	1.7 g/t PD	7,379	4.42	0.312	0.306	0.119	0.090
	1.6 g/t PD	7,487	4.38	0.310	0.303	0.119	0.089
	1.5 g/t PD	7,578	4.35	0.308	0.302	0.118	0.089
	1.4 g/t PD	7,660	4.32	0.306	0.300	0.118	0.089
	1.3 g/t PD	7,734	4.29	0.305	0.299	0.117	0.088
	1.2 g/t PD	7,795	4.26	0.303	0.297	0.117	0.088
	1.1 g/t PD	7,846	4.24	0.302	0.296	0.117	0.088
	1.0 g/t PD	7,891	4.23	0.301	0.295	0.116	0.088
	0 g/t PD	8,036	4.16	0.298	0.291	0.115	0.087
Offset Footwall	3.0 g/t PD	1,383	3.99	0.340	0.254	0.103	0.084
	2.9 g/t PD	1,566	3.86	0.333	0.248	0.102	0.083
	2.8 g/t PD	1,776	3.74	0.326	0.243	0.101	0.082
	2.7 g/t PD	2,037	3.62	0.318	0.236	0.100	0.081
	2.6 g/t PD	2,332	3.49	0.310	0.230	0.098	0.079
	2.5 g/t PD	2,696	3.37	0.303	0.223	0.097	0.078
	2.4 g/t PD	3,091	3.25	0.295	0.217	0.095	0.077
	2.3 g/t PD	3,565	3.13	0.288	0.211	0.094	0.076
	2.25 g/t PD	3,830	3.07	0.284	0.207	0.093	0.075
	2.2 g/t PD	4,086	3.02	0.281	0.204	0.093	0.074
	2.1 g/t PD	4,740	2.90	0.273	0.198	0.091	0.073
	2.0 g/t PD	5,469	2.78	0.266	0.192	0.089	0.072
	1.9 g/t PD	6,312	2.67	0.258	0.186	0.088	0.070

ZONE	Cut-off Grade	Tonnage	PD	PT	AU	NI	CU
		T x 1000	g/t	g/t	g/t	%	%
	1.8 g/t PD	7,254	2.57	0.251	0.181	0.087	0.069
	1.7 g/t PD	8,344	2.46	0.243	0.175	0.085	0.067
	1.6 g/t PD	9,599	2.35	0.236	0.169	0.084	0.066
	1.5 g/t PD	11,081	2.25	0.228	0.163	0.082	0.065
	1.4 g/t PD	12,743	2.14	0.220	0.158	0.081	0.063
	1.3 g/t PD	14,582	2.04	0.212	0.152	0.079	0.062
	1.2 g/t PD	16,653	1.94	0.205	0.147	0.078	0.060
	1.1 g/t PD	18,942	1.85	0.197	0.141	0.077	0.059
	1.0 g/t PD	21,366	1.76	0.190	0.136	0.075	0.057
	0 g/t PD	40,036	1.22	0.143	0.102	0.065	0.046

Table 14-9: Offset Zone Indicated Resource Grade Tonnage Table

ZONE	Cut-off Grade	Tonnage	PD	PT	AU	NI	CU
		T x 1000	g/t	g/t	g/t	%	%
Offset Hangingwall	3.0 g/t PD	7,147	4.86	0.336	0.329	0.124	0.104
	2.9 g/t PD	7,417	4.79	0.333	0.327	0.123	0.103
	2.8 g/t PD	7,677	4.73	0.330	0.326	0.123	0.103
	2.7 g/t PD	7,937	4.66	0.328	0.323	0.123	0.102
	2.6 g/t PD	8,198	4.60	0.325	0.322	0.122	0.102
	2.5 g/t PD	8,464	4.53	0.322	0.319	0.122	0.101
	2.4 g/t PD	8,679	4.48	0.320	0.318	0.121	0.101
	2.3 g/t PD	8,883	4.43	0.317	0.316	0.121	0.101
	2.25 g/t PD	8,975	4.41	0.316	0.316	0.121	0.101
	2.2 g/t PD	9,074	4.39	0.315	0.315	0.121	0.100
	2.1 g/t PD	9,250	4.34	0.313	0.313	0.120	0.100
	2.0 g/t PD	9,393	4.31	0.312	0.312	0.120	0.100
	1.9 g/t PD	9,525	4.28	0.310	0.311	0.120	0.099
	1.8 g/t PD	9,661	4.24	0.309	0.310	0.120	0.099
	1.7 g/t PD	9,772	4.21	0.307	0.309	0.119	0.099
	1.6 g/t PD	9,879	4.19	0.306	0.308	0.119	0.099
	1.5 g/t PD	9,971	4.16	0.305	0.307	0.119	0.098
	1.4 g/t PD	10,047	4.14	0.304	0.307	0.119	0.098
	1.3 g/t PD	10,111	4.12	0.303	0.306	0.119	0.098
	1.2 g/t PD	10,159	4.11	0.302	0.306	0.118	0.098
	1.1 g/t PD	10,204	4.10	0.301	0.305	0.118	0.098
	1.0 g/t PD	10,232	4.09	0.301	0.305	0.118	0.098
	0 g/t PD	10,304	4.07	0.300	0.303	0.118	0.097

ZONE	Cut-off Grade	Tonnage	PD	PT	AU	NI	CU
		T x 1000	g/t	g/t	g/t	%	%
Offset Footwall	3.0 g/t PD	891	3.71	0.321	0.238	0.105	0.094
	2.9 g/t PD	1,058	3.59	0.312	0.231	0.103	0.092
	2.8 g/t PD	1,243	3.48	0.305	0.226	0.101	0.091
	2.7 g/t PD	1,473	3.37	0.297	0.218	0.099	0.088
	2.6 g/t PD	1,765	3.25	0.288	0.211	0.097	0.086
	2.5 g/t PD	2,074	3.14	0.281	0.208	0.096	0.085
	2.4 g/t PD	2,457	3.04	0.274	0.204	0.094	0.083
	2.3 g/t PD	2,935	2.92	0.267	0.199	0.093	0.082
	2.25 g/t PD	3,202	2.87	0.263	0.196	0.092	0.081
	2.2 g/t PD	3,486	2.82	0.260	0.193	0.091	0.080
	2.1 g/t PD	4,142	2.71	0.252	0.187	0.089	0.078
	2.0 g/t PD	4,920	2.61	0.245	0.182	0.087	0.076
	1.9 g/t PD	5,798	2.51	0.238	0.176	0.086	0.074
	1.8 g/t PD	6,820	2.41	0.231	0.171	0.084	0.072
	1.7 g/t PD	8,115	2.30	0.224	0.165	0.082	0.070
	1.6 g/t PD	9,587	2.20	0.217	0.160	0.081	0.069
	1.5 g/t PD	11,272	2.11	0.210	0.155	0.079	0.067
	1.4 g/t PD	13,196	2.01	0.203	0.150	0.077	0.065
	1.3 g/t PD	15,418	1.91	0.196	0.145	0.076	0.063
	1.2 g/t PD	17,894	1.82	0.190	0.140	0.074	0.061
	1.1 g/t PD	20,523	1.74	0.183	0.135	0.073	0.059
	1.0 g/t PD	23,491	1.65	0.177	0.130	0.071	0.058
	0 g/t PD	44,593	1.17	0.136	0.102	0.063	0.047

Table 14-10: Offset Zone Inferred Resource Grade Tonnage Table

ZONE	Cut-off Grade	Tonnage	PD	PT	AU	NI	CU
		T x 1000	g/t	g/t	g/t	%	%
Offset Hangingwall	3.0 g/t PD	3,693	4.24	0.285	0.265	0.108	0.091
	2.9 g/t PD	3,899	4.17	0.282	0.263	0.108	0.091
	2.8 g/t PD	4,117	4.10	0.279	0.259	0.107	0.090
	2.7 g/t PD	4,342	4.03	0.275	0.256	0.106	0.089
	2.6 g/t PD	4,547	3.97	0.272	0.252	0.105	0.088
	2.5 g/t PD	4,696	3.93	0.271	0.251	0.105	0.088
	2.4 g/t PD	4,849	3.88	0.269	0.250	0.104	0.087
	2.3 g/t PD	4,977	3.84	0.267	0.249	0.104	0.087
	2.25 g/t PD	5,035	3.82	0.266	0.248	0.104	0.087
	2.2 g/t PD	5,094	3.80	0.266	0.248	0.104	0.087
	2.1 g/t PD	5,204	3.77	0.264	0.248	0.104	0.086
	2.0 g/t PD	5,297	3.74	0.263	0.247	0.103	0.086
	1.9 g/t PD	5,372	3.71	0.262	0.247	0.103	0.086
	1.8 g/t PD	5,439	3.69	0.261	0.246	0.103	0.086
	1.7 g/t PD	5,492	3.67	0.260	0.245	0.103	0.086
	1.6 g/t PD	5,538	3.66	0.259	0.245	0.103	0.085
	1.5 g/t PD	5,571	3.64	0.259	0.244	0.103	0.085
	1.4 g/t PD	5,600	3.63	0.258	0.244	0.103	0.085
	1.3 g/t PD	5,624	3.62	0.257	0.244	0.102	0.085
	1.2 g/t PD	5,648	3.61	0.257	0.243	0.102	0.085
	1.1 g/t PD	5,666	3.60	0.257	0.243	0.102	0.085
	1.0 g/t PD	5,681	3.60	0.256	0.243	0.102	0.085
	0 g/t PD	5,715	3.58	0.255	0.242	0.102	0.085
Offset Footwall	3.0 g/t PD	461	3.65	0.225	0.181	0.084	0.077
	2.9 g/t PD	558	3.53	0.219	0.176	0.082	0.075
	2.8 g/t PD	687	3.40	0.214	0.170	0.080	0.074
	2.7 g/t PD	866	3.27	0.208	0.164	0.079	0.071
	2.6 g/t PD	1,088	3.14	0.204	0.158	0.077	0.068
	2.5 g/t PD	1,491	2.98	0.212	0.145	0.078	0.064
	2.4 g/t PD	1,745	2.90	0.210	0.145	0.077	0.064
	2.3 g/t PD	2,078	2.81	0.205	0.143	0.076	0.063
	2.25 g/t PD	2,298	2.76	0.200	0.141	0.075	0.062
	2.2 g/t PD	2,524	2.71	0.197	0.140	0.075	0.062
	2.1 g/t PD	3,019	2.62	0.194	0.138	0.074	0.062
	2.0 g/t PD	3,614	2.53	0.191	0.137	0.073	0.061
	1.9 g/t PD	4,288.68	2.436	0.187	0.135	0.072	0.061
	1.8 g/t PD	5,162.87	2.336	0.181	0.132	0.071	0.060

ZONE	Cut-off Grade	Tonnage	PD	PT	AU	NI	CU
		T x 1000	g/t	g/t	g/t	%	%
	1.7 g/t PD	6,206.80	2.237	0.176	0.128	0.070	0.059
	1.6 g/t PD	7,478.41	2.137	0.170	0.125	0.069	0.058
	1.5 g/t PD	8,854.11	2.046	0.165	0.121	0.068	0.056
	1.4 g/t PD	10,478.93	1.953	0.160	0.117	0.066	0.055
	1.3 g/t PD	12,585.09	1.852	0.153	0.111	0.065	0.053
	1.2 g/t PD	14,683.86	1.765	0.148	0.107	0.064	0.052
	1.1 g/t PD	17,125.80	1.677	0.143	0.103	0.063	0.050
	1.0 g/t PD	19,928.88	1.589	0.138	0.099	0.062	0.049
	0 g/t PD	35,956.16	1.178	0.110	0.081	0.057	0.043

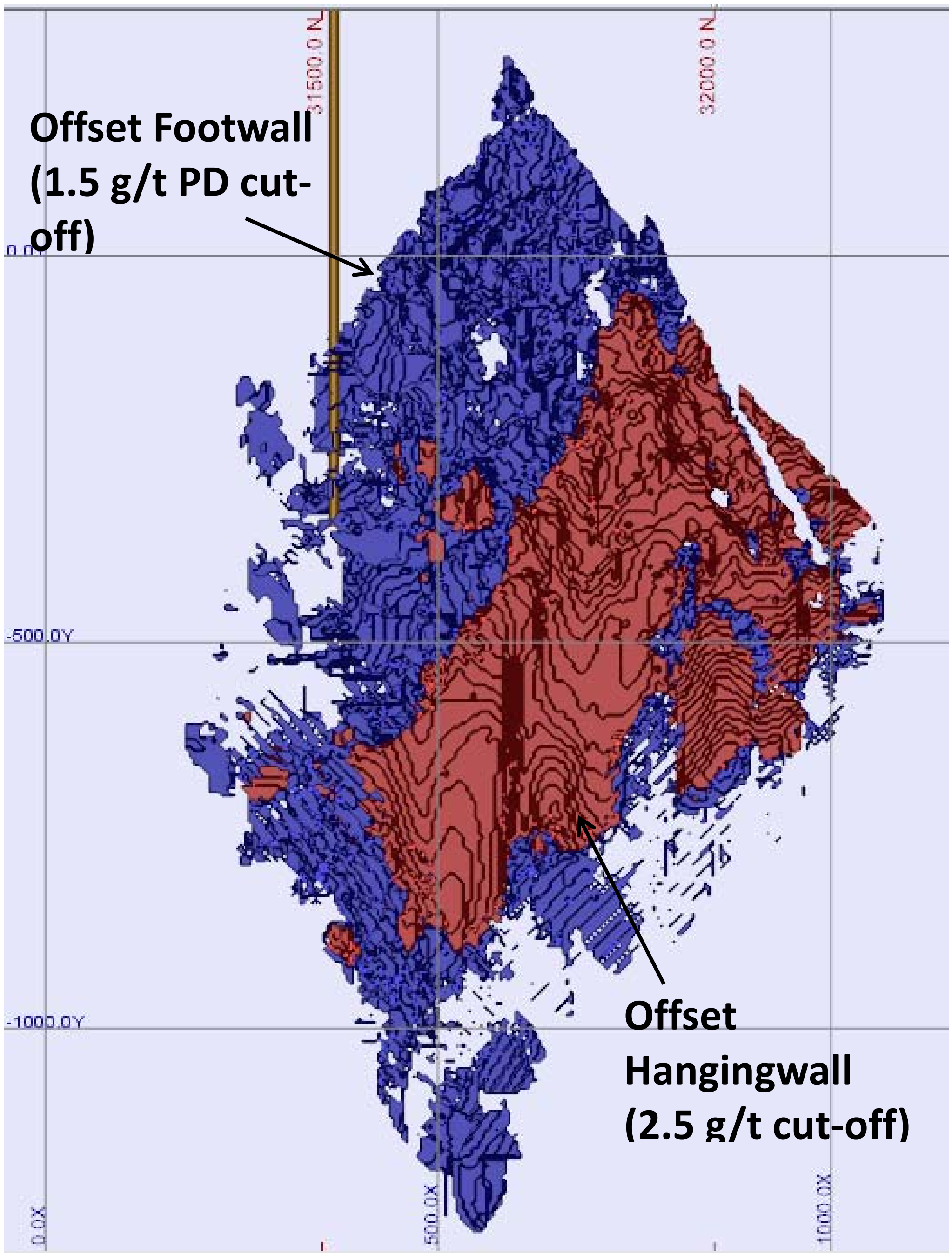


Figure 14-8: Offset Hangingwall Zone and Offset Footwall Zone Resources Reported in the Current Study and Based on Different Pd Cut-Off Grades (as shown)

Table 14-11: Mineral Resource Estimate for the Offset Hangingwall Zone and the Offset Footwall Zone

ZONE	Resource Category	Cut-off	Tonnage	PD	PT	AU	NI	CU	PD
		g/t	Tonnes	g/t	g/t	g/t	%	%	Oz
Offset Hangingwall Zone	Measured	2.5	6,186,000	4.86	0.332	0.326	0.124	0.093	967,000
	Indicated	2.5	8,464,000	4.53	0.322	0.319	0.122	0.101	1,234,000
	Measured + Indicated	2.5	14,650,000	4.67	0.326	0.322	0.122	0.098	2,200,000
	Inferred	2.5	4,696,000	3.93	0.271	0.251	0.105	0.088	-
Offset Footwall Zone	Measured	1.5	11,080,000	2.25	0.228	0.163	0.082	0.065	800,000
	Indicated	1.5	11,270,000	2.11	0.21	0.155	0.079	0.067	762,800
	Measured + Indicated	1.5	22,350,000	2.18	0.219	0.159	0.081	0.066	1,563,000
	Inferred	1.5	8,854,000	2.05	0.165	0.121	0.068	0.056	-

14.1.10 Validation

The Offset models were validated by three methods:

1. Visual comparison of colour-coded block model grades with composite grades on section and plan.
2. Comparison of the global mean block grades for OK, ID2, NN and composites.
3. Swath plots of the block models in both level, row and column directions.

14.1.10.1 Visual Validation

The visual comparisons of block model grades with composite grades for each of the zones show a reasonable correlation between the values. No significant discrepancies were apparent from the sections reviewed. Figure 14-9 and Figure 14-10 show examples of a cross-section and a plan view image with the same colour code for composite points and blocks.

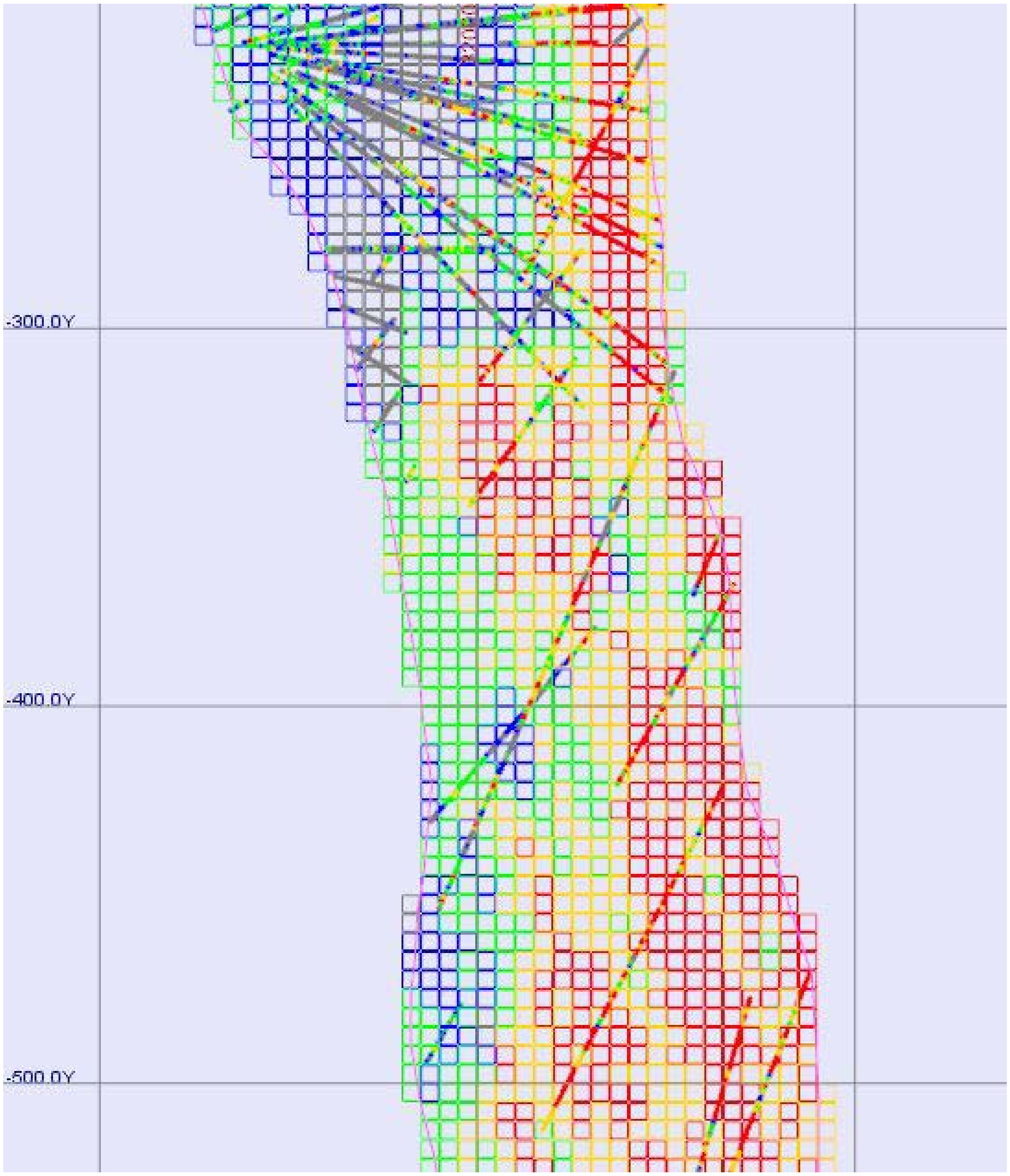


Figure 14-9: Composite Data and Block Model, Section 31865N, Looking North

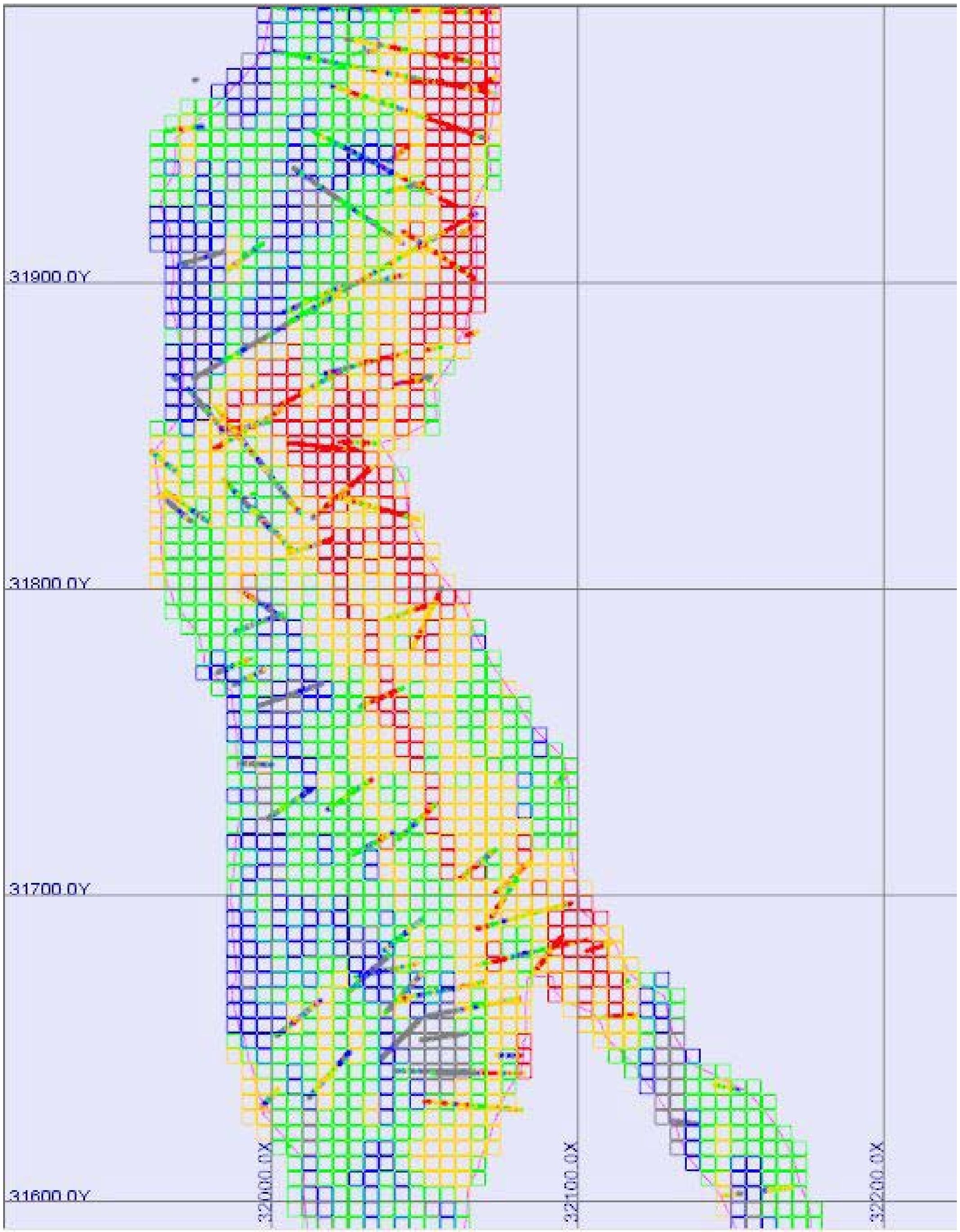


Figure 14-10: Composite Data and Block Model, Plan View, Elevation -325 (mine level 825)

14.1.10.2 Global Comparison

The global block model statistics for the OK model were compared to the global ID2 and NN model values as well as the composite capped drillhole data (Table 14-12). In general, there is agreement between the OK model and the ID2 model. Larger discrepancies are reflected as a result of lower drill density in some portions of the model. There is a degree of smoothing apparent when compared to the NN model and the composite data. Comparisons were made using all blocks at a 0% Pd cut-off.

Table 14-12: Offset Zone Global Statistics Comparison

Element	Attribute	Min Value	Max Value	Average	Variance	Standard Deviation
Palladium	PD_OK	0.000	21.4	1.694	2.323	1.524
	PD_ID2	0.001	26.0	1.708	2.683	1.638
	PD_NN	0.000	28.6	1.226	2.156	1.468
	PD_Comp	0.000	30.0	1.894	7.156	2.675
Platinum	PT_OK	0.000	2.05	0.166	0.011	0.104
	PT_ID2	0.000	1.63	0.172	0.013	0.114
	PT_NN	0.000	3.95	0.165	0.029	0.171
	PT_Comp	0.000	3.95	0.174	0.033	0.181
Gold	AU_OK	0.000	1.73	0.132	0.015	0.122
	AU_ID2	0.000	2.18	0.133	0.017	0.130
	AU_NN	0.000	2.97	0.131	0.037	0.194
	AU_Comp	0.000	3.00	0.141	0.043	0.207
Nickel	NI_OK	0.000	0.47	0.072	0.001	0.035
	NI_ID2	0.000	0.56	0.072	0.001	0.037
	NI_NN	0.000	1.25	0.072	0.003	0.053
	NI_Comp	0.000	1.25	0.075	0.003	0.058
Copper	CU_OK	0.000	0.66	0.054	0.001	0.036
	CU_ID2	0.000	0.66	0.054	0.002	0.039
	CU_NN	0.000	1.52	0.054	0.003	0.058
	CU_Comp	0.000	1.52	0.054	0.004	0.061

14.1.10.3 Swath Plots

Swath plots of levels, columns and rows have been generated for both elements. These plots compare the OK estimates with the NN and ID2 estimates for blocks into the measured and indicated resource category, and are illustrated Figure 14-11 to Figure 14-25.

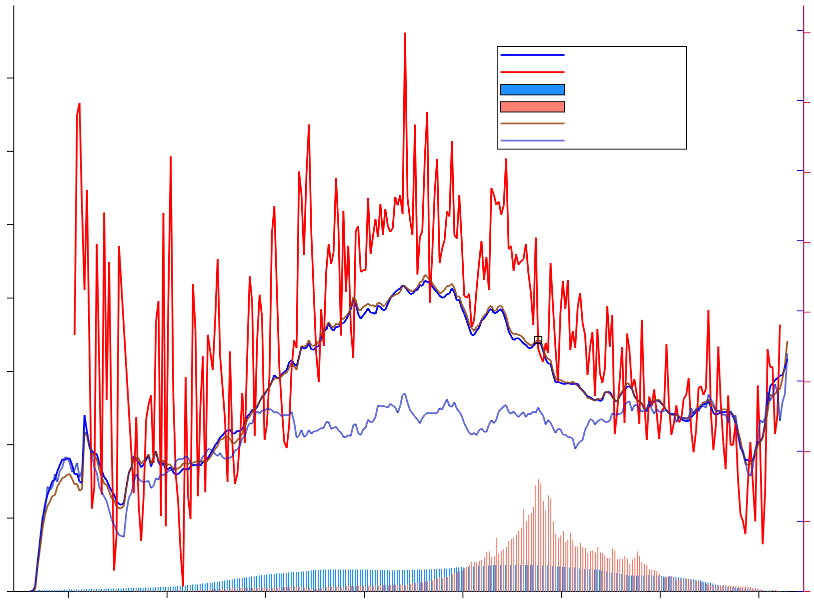


Figure 14-11: Swath Plot of Levels for Palladium

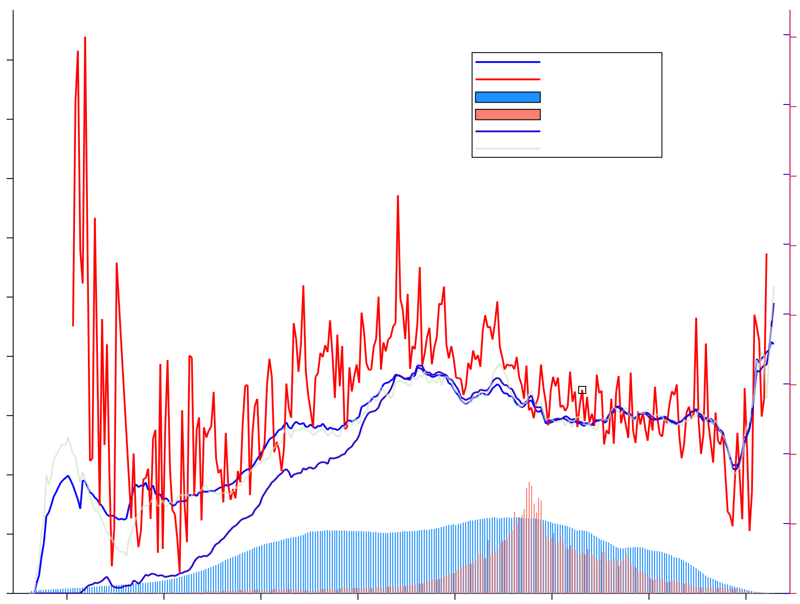


Figure 14-12: Swath Plot of Levels for Platinum

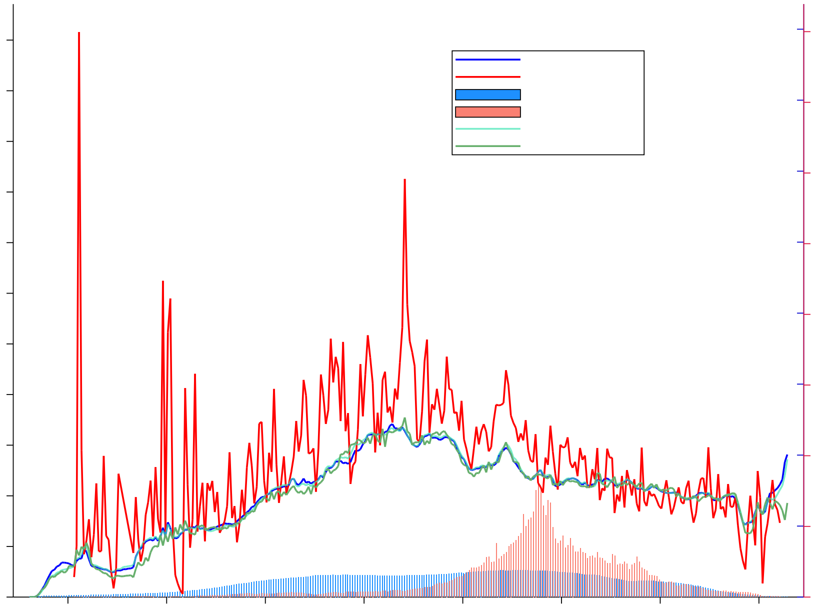


Figure 14-13: Swath Plot of Levels for Gold

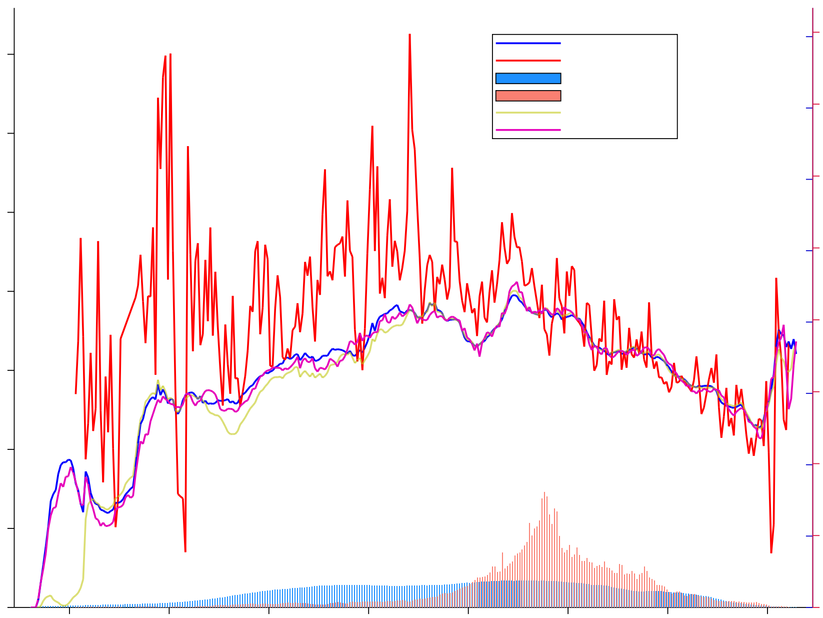


Figure 14-14: Swath Plot of Levels for Nickel

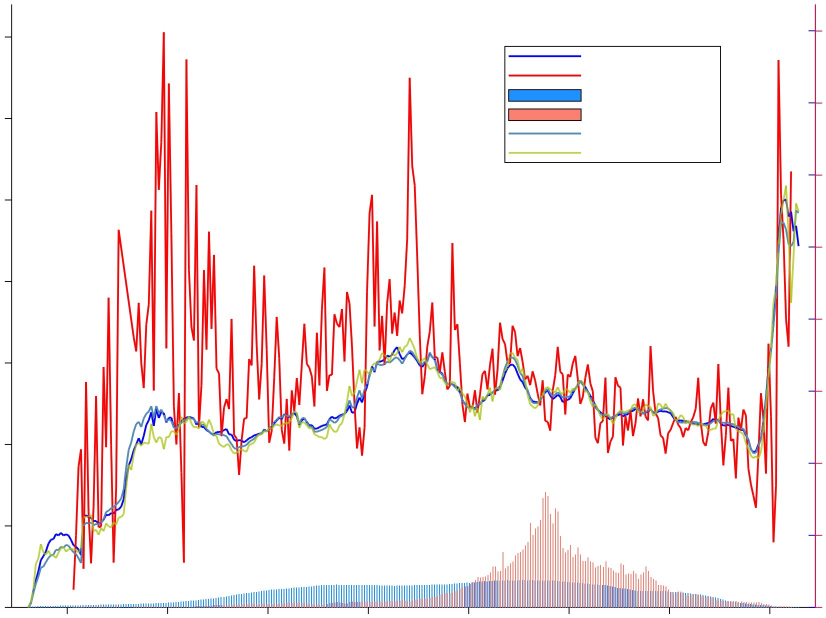


Figure 14-15: Swath Plot of Levels for Copper

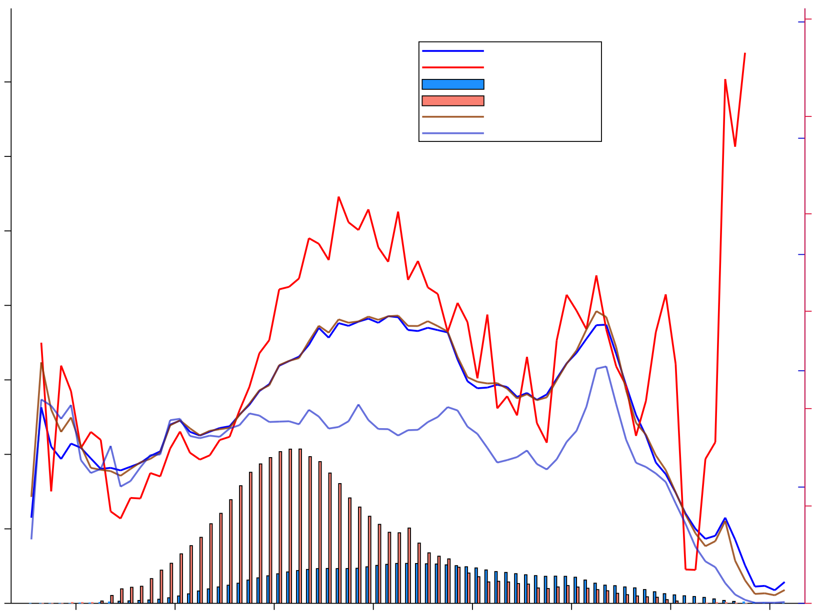


Figure 14-16: Swath Plot of Columns for Palladium

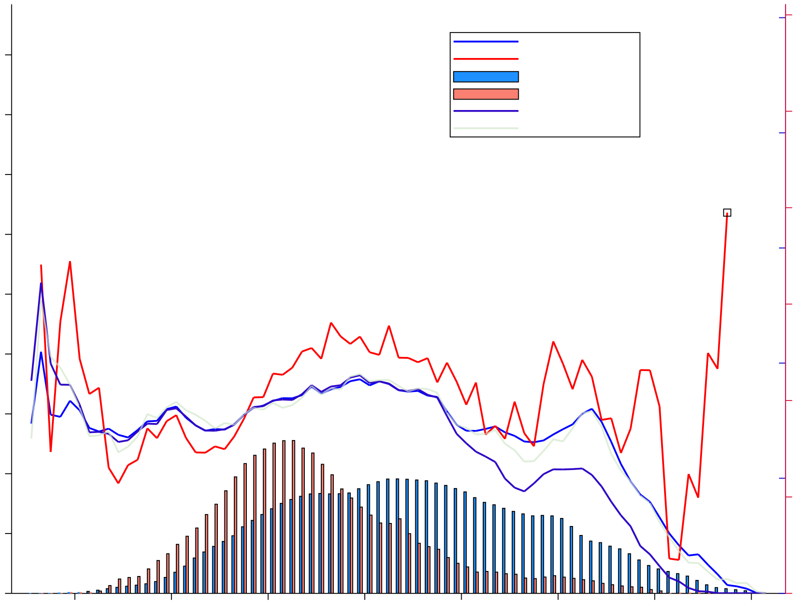


Figure 14-17: Swath Plot of Columns for Platinum

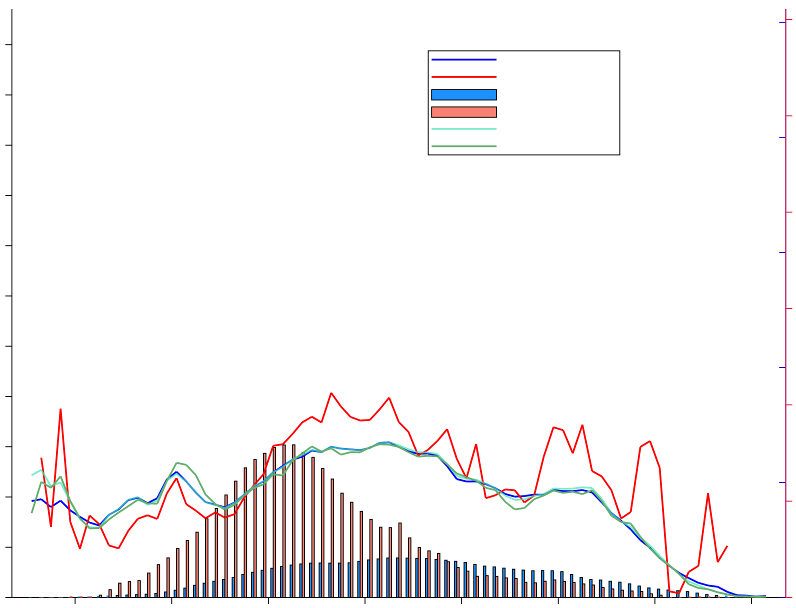


Figure 14-18: Swath Plot of Columns for Gold

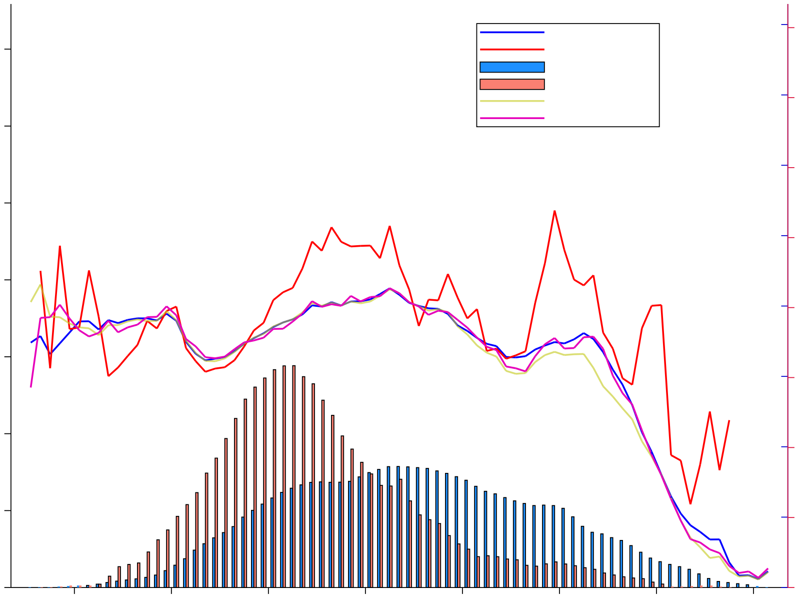


Figure 14-19: Swath Plot of Columns for Nickel

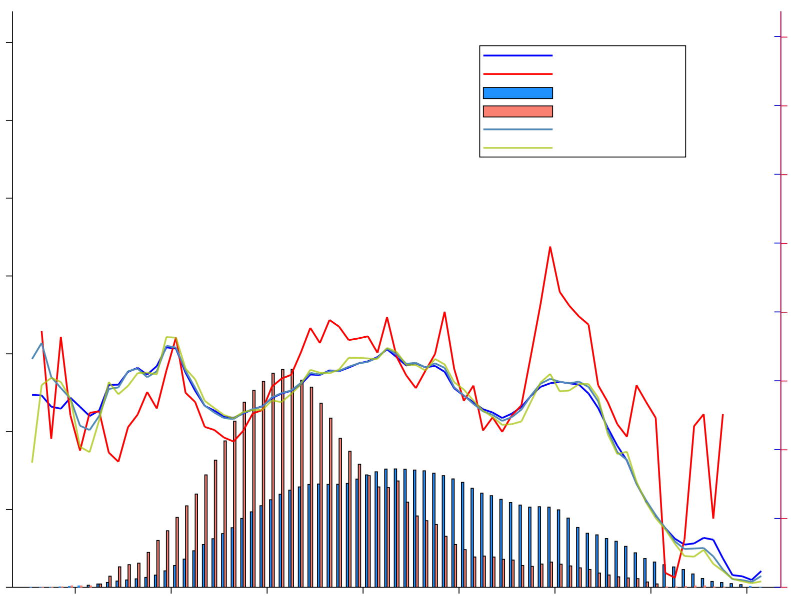


Figure 14-20: Swath Plot of Columns for Copper

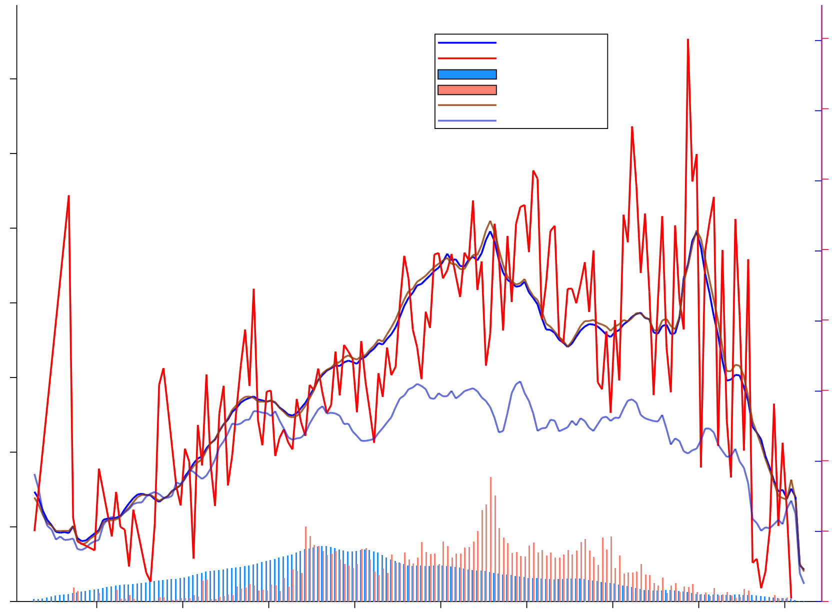


Figure 14-21: Swath Plot of Rows for Palladium

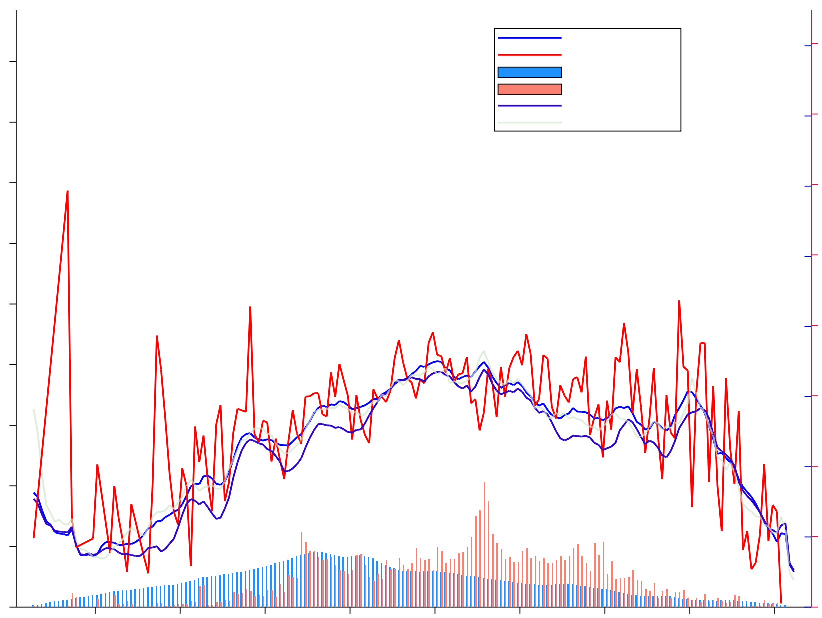


Figure 14-22: Swath Plot of Rows for Platinum

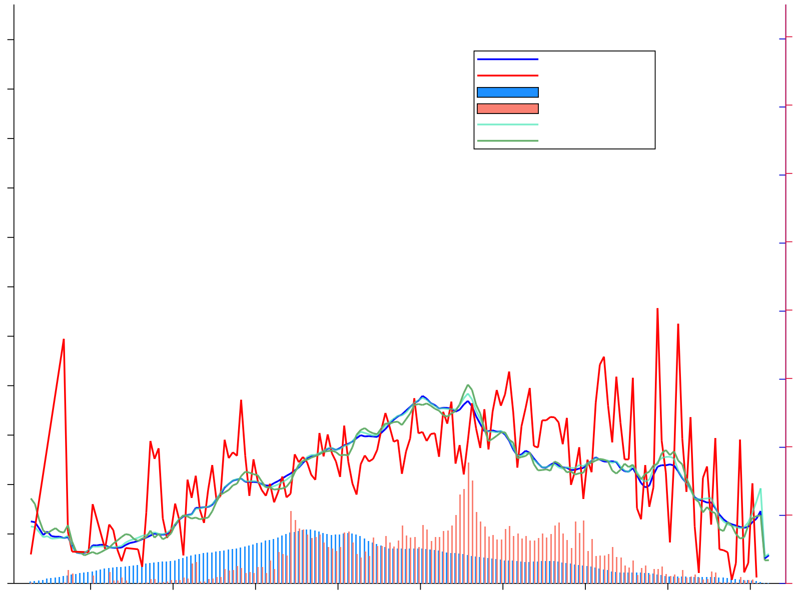


Figure 14-23: Swath Plot of Rows for Gold

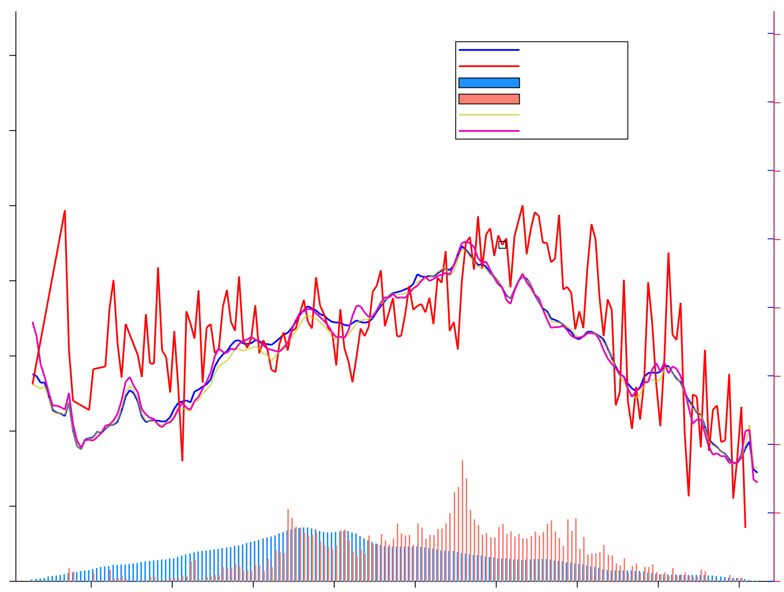


Figure 14-24: Swath Plot of Rows for Nickel

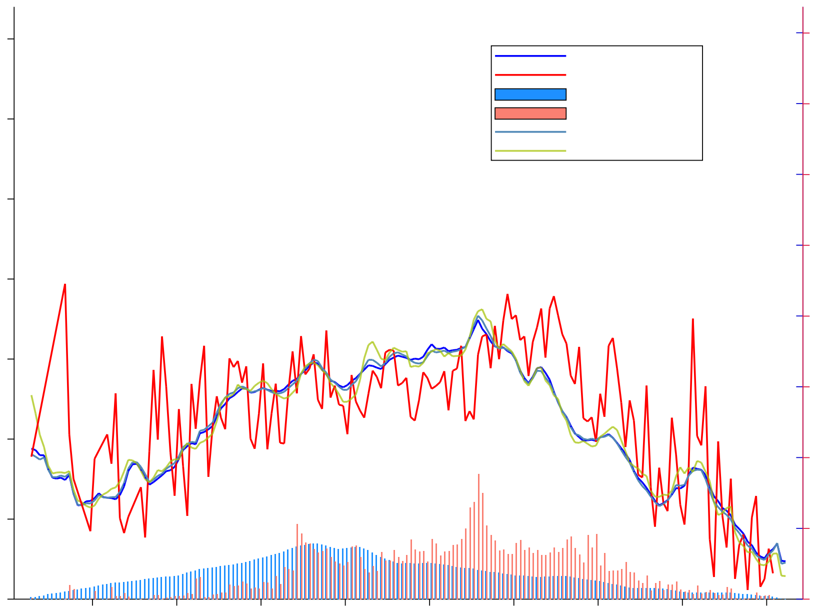


Figure 14-25: Swath Plot of Rows for Copper

14.1.11 Previous Estimates

A comparison with the previous (2010 and 2012) Offset model provided by external consultants is presented in McKinnon et al., 2014.

Table 14-13 illustrates the differences in the 2014 resource estimate (McKinnon et al., 2014), and the current NI 43-101 compliant resource as of December 31, 2014. The resource estimates in this table are inclusive of any defined mineral reserve.

Table 14-13: Offset Zone Global Statistics Comparison

ZONE	Resource Category	Cut-off	Tonnage	PD	PT	AU	NI	CU	PD
		g/t	Tonnes	g/t	g/t	g/t	%	%	Oz
NAP Offset resource model (effective date Dec 31,2013)									
Offset Hangingwall Zone	Measured	2.5	6,735,000	4.97	0.34	0.33	0.13	0.09	1,077,000
	Indicated	2.5	7,098,000	4.51	0.32	0.31	0.12	0.1	1,030,000
	Inferred	2.5	6,166,000	3.79	0.23	0.23	0.1	0.09	-
Offset Footwall Zone	Measured	1.5	11,620,000	2.26	0.23	0.16	0.08	0.06	845,500
	Indicated	1.5	8,862,000	2.13	0.22	0.16	0.08	0.07	605,600
	Inferred	1.5	9,534,000	2.16	0.15	0.12	0.06	0.05	-
NAP Offset resource model (effective date Dec 31,2014)									
Offset Hangingwall Zone	Measured	2.5	6,186,000	4.86	0.33	0.33	0.12	0.09	967,000
	Indicated	2.5	8,464,000	4.53	0.32	0.32	0.12	0.1	1,234,000
	Inferred	2.5	4,696,000	3.93	0.27	0.25	0.1	0.09	-
Offset Footwall Zone	Measured	1.5	11,080,000	2.25	0.23	0.16	0.08	0.06	800,000
	Indicated	1.5	11,270,000	2.10	0.21	0.16	0.08	0.07	762,800
	Inferred	1.5	8,854,000	2.05	0.16	0.12	0.07	0.06	-

The differences between estimates are due to:

- The mining depletion which occurred in 2014.
- The removal of some material in rib pillars which, in the author's opinion, do not demonstrate reasonable potential for eventual economic extraction.
- The removal of some material pinched inside a dike swarm which, in the author's opinion, do not demonstrate reasonable potential for eventual economic extraction.
- Definition of inferred resource and conversion of previously defined inferred and indicated resource into indicated and measured resource respectively, due to the 2014 drillhole campaign results.

Globally, the author considers that there is no material change between the 2014 and 2015 estimates, at the scale of the whole deposit.

14.2 Upper Offset Southeast Extension

14.2.1 Introduction

In 2014, NAP drilled a zone situated at the Upper South extremity of the Offset fault. This zone was previously reported in the Offset Footwall, but 2014 drillhole allowed identifying a specific zone with higher grade and different mineralized direction than the general Offset 1g/t palladium wireframe described in Section 14.1.

14.2.2 Database

The database used for the Upper Offset Southeast extension is the same as described in Section 14.1.2.

14.2.3 Specific Gravity

The specific gravity used for this estimate is 2.89, as described in Section 14.1.3.

14.2.4 Geological Interpretation

The zone of interest is limited by different geological structures (described in Section 7.3): The upper limit consists of the Offset fault and the lower limit consists of the B2 fault. The eastern contact consists of the EGAB contact (Section 7.2.1).

Drillhole assays samples situated in the zone described above were imported into Leapfrog™ in order to build a 2 g/t Pd wireframe. A plane of continuity (azimuth N 87°, dip -57° S) for the mineralisation was identify and used to create an anisotropic interpolant based on palladium grade in assay points. The 2 g/t Pd wireframe was extracted and imported into GEOVIA GEMS 6.7 ™, and was used to create a more regular and consistent 2 g/t Pd wireframe using parallel rings and tie lines.

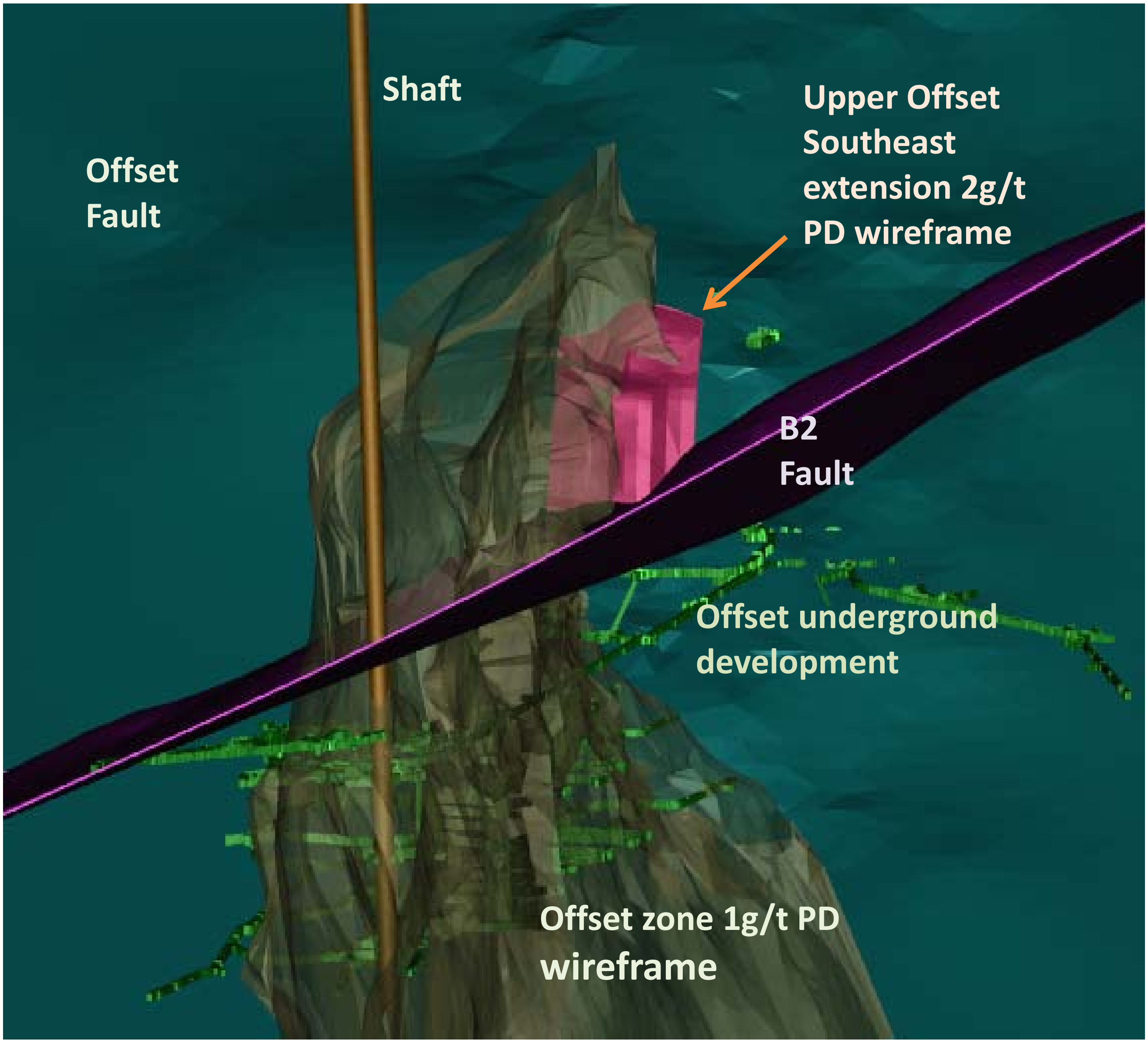


Figure 14-26: 3D View of the Upper Offset Southeast Extension Wireframe, Looking North

14.2.5 Exploratory Data Analysis

14.2.5.1 Assays

The compliant drillholes which intersect the Upper Offset Southeast extension wireframe described above are the following: 11-055, 13-717, 14-771, 14-772, 14-773, 14-776 and 14-777.

The portion of the deposit included in the mineral resource was based on a total of 252 palladium assays. The assay intervals within each zone were captured using GEOVIA GEMS™ cross-table manipulation into individual drillhole files. These drillhole files were reviewed to ensure that all relevant assay intervals were captured.

14.2.5.2 Compositing

Compositing of all assay data was completed into the wireframe defined in 14.2.4. A 1m composite length was chosen, however, the backstitching process was used in the composite routine to ensure all captured sample material was included. The backstitching process adjusts the composite lengths for each individual drillhole in order to compensate for the last sample interval.

14.2.5.3 Grade Capping

The raw assay data for the 2 g/t palladium wireframe was examined to assess the amount of metal that is at risk from high-grade assays. Histogram plots were used to determine if grade capping was required for each element. Figure 14-27 to Figure 14-31 show the results of the Histograms plots. Grade capping was determined not to be necessary.

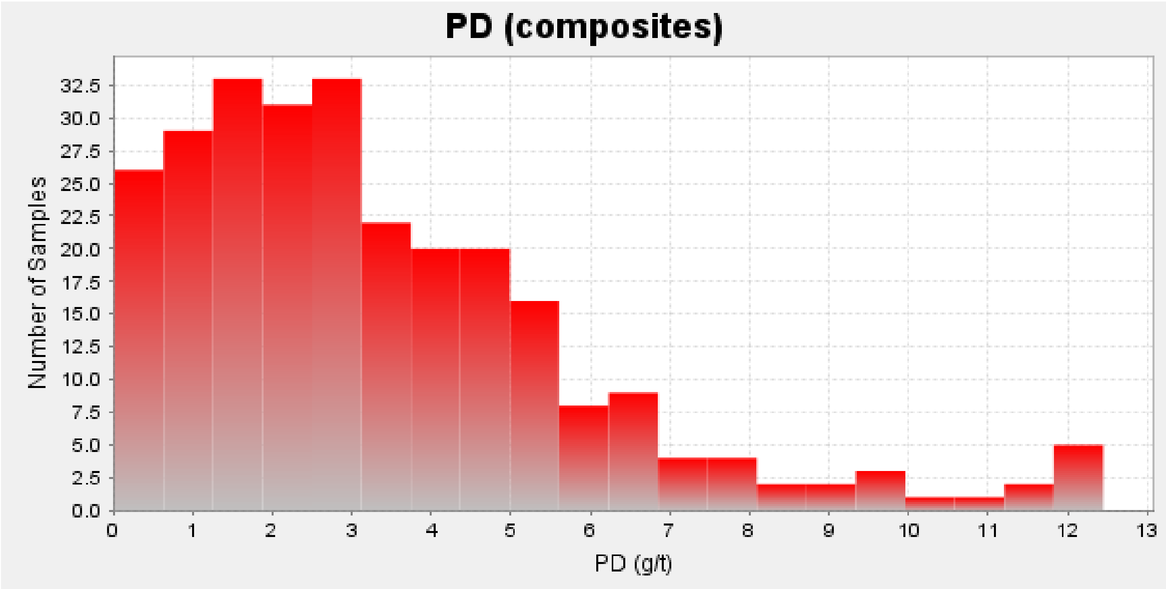


Figure 14-27: Palladium Histogram

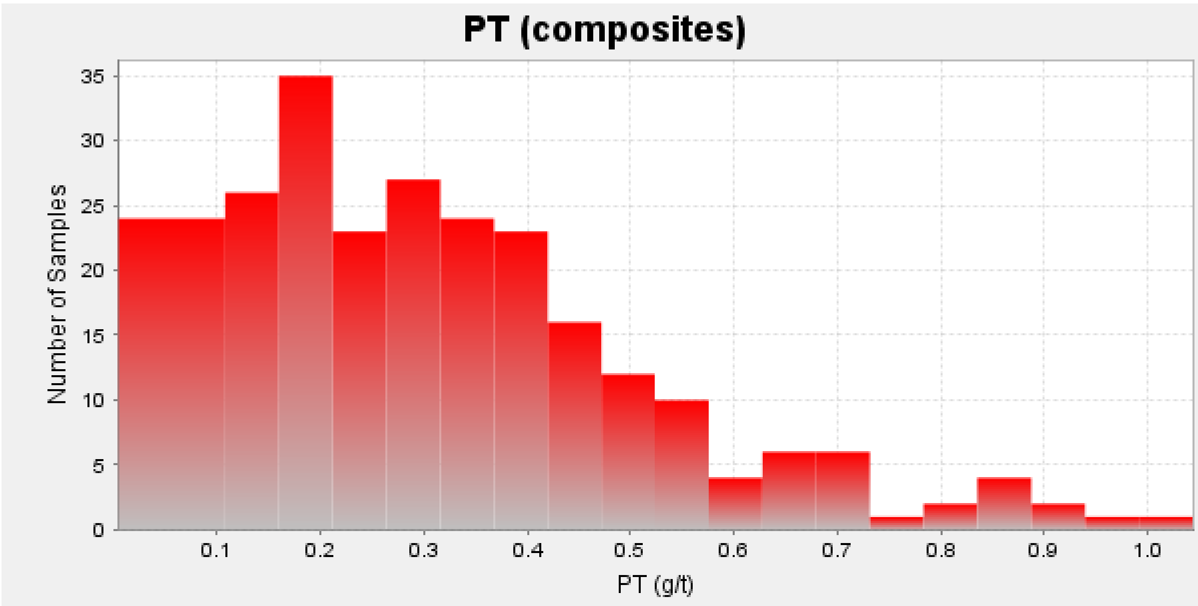


Figure 14-28: Platinum Histogram

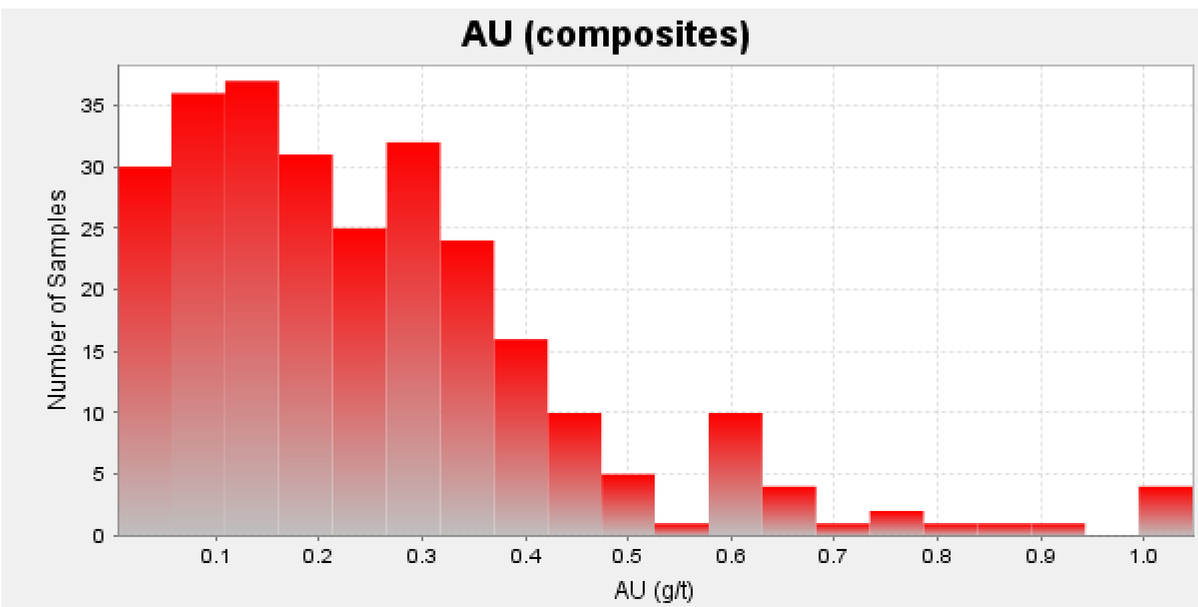


Figure 14-29: Gold Histogram

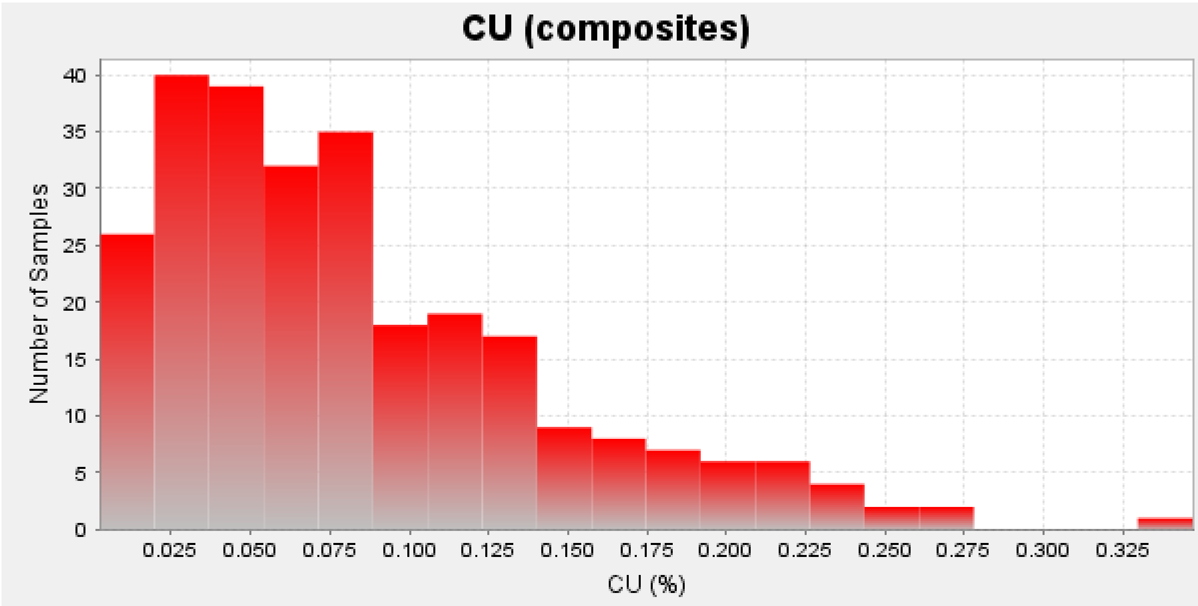


Figure 14-30: Copper Histogram

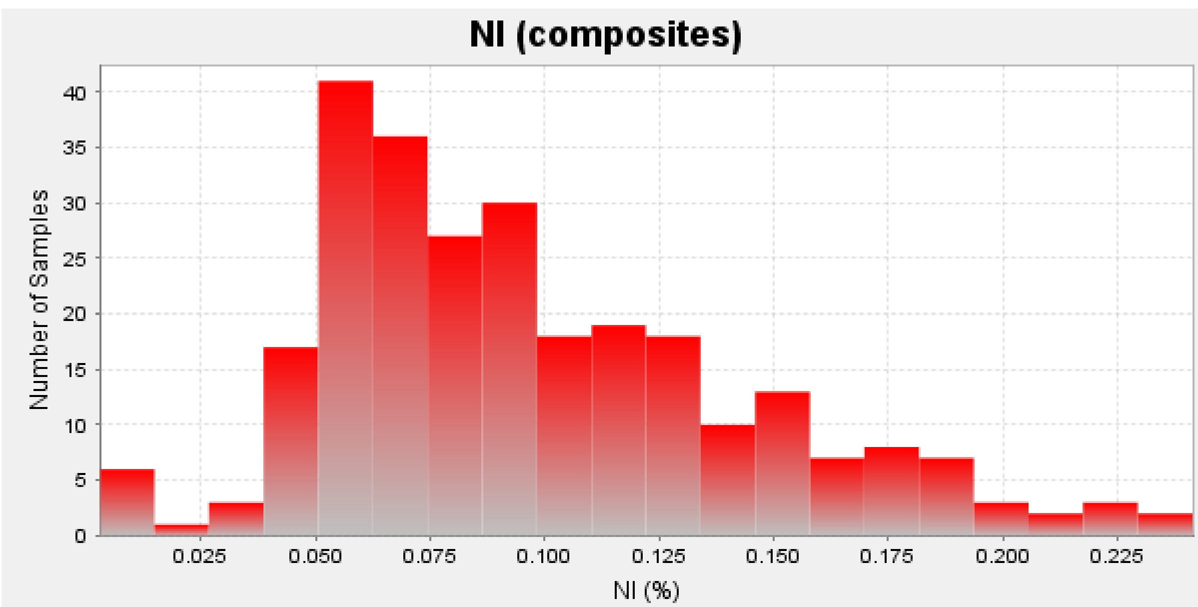


Figure 14-31: Nickel Histogram

14.2.6 *Spatial Analysis*

Variography using GEOVIA GEMS™ software was attempted for each element globally for all the composited data. However, the limited number of samples available in the 2 g/t wireframe does not allow the creation of variograms. Accordingly, an isotropic inverse distance ID^2 resource model was used in the estimation of resources for the Upper Offset Southeast Extension zone.

14.2.7 *Resource Block Model*

The same block model as described in Section 14.1.7 was used for the estimation. However, to keep the resources separated, a new folder strictly made for the Upper Offset Southeast extension was created.

14.2.7.1 *Estimation and Search Parameters*

The interpolations of the zones were completed using the estimation method inverse distance squared ID2. The estimations were designed for three passes. In each pass a minimum and maximum number of samples were required as well as a maximum number of samples from a drillhole in order to satisfy the estimation criteria.

Table 14-14: Description of Interpolation Passes

	Min. No. of Composites	Max. No. Of Composites	Maximum Samples per Drillhole	Min. No. of Drillholes	Max. No. of Drillholes	Maximum Search Radius
Pass 1	10	50	5	2	10	20
Pass 2	8	40	5	2	8	40
Pass 3	6	40	5	2	8	80

14.2.8 *Resource Classification*

Several factors are considered in the definition of a resource classification:

- NI 43-101 requirements.
- Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines.
- The author's experience with ultramafic-mafic hosted PGE-Cu-Ni deposits such as those present in the Mine Block intrusion on the LDI property (see Sections 7 and 8 in this report).
- Borehole spacing and estimation runs required to estimate the grades in a block.
- Observed mineralization in underground development.
- The number of samples and boreholes used in each of the block estimations.

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to NAP that may affect the estimate of mineral resources. Mineral reserves can only be estimated on the basis of an economic evaluation that is used in a preliminary feasibility study or a feasibility study of a mineral project; thus, no reserves have been estimated. As per NI 43-101 guidelines, mineral resources, which are not mineral reserves, do not have to demonstrate economic viability.

Given the limited number of sample available for this zone, the fact that it is not possible to complete a proper variography study and the different nature and orientation of the zone compared to the Offset zone described in Section 14.1, it is the author's opinion that the Upper Offset Southeast extension resources have to be classified entirely as inferred resources. More drillholes are required to define an indicated resource category.

14.2.9 Mineral Resource Tabulation

The resources cited in Table 14-15 are reported with an effective date of December 31, 2014.

Table 14-15: Upper Offset Southeast Extension Mineral Resource Tabulation

Resource Category	Volume (M**3)	Density (T per M**3)	Tonnage (Tonnes)	PD (g/t)	PT (g/t)	AU (g/t)	NI (%)	CU (%)
Inferred	286,100	2.890	826,739	3.20	0.284	0.238	0.091	0.076

14.2.10 Validation

The Upper Offset Southeast Extension models were validated by two methods:

1. Visual comparison of colour-coded block model grades with composite grades on sections.
2. Comparison of the global mean block grades for block model and composites.

14.2.10.1 Visual Validation

The visual comparisons of block model grades with composite grades show a reasonable correlation between the values. No significant discrepancies were apparent from the sections reviewed. Figure 14-32 and Figure 14-33 show examples of cross-section images with the same colour code for composite points and blocks.

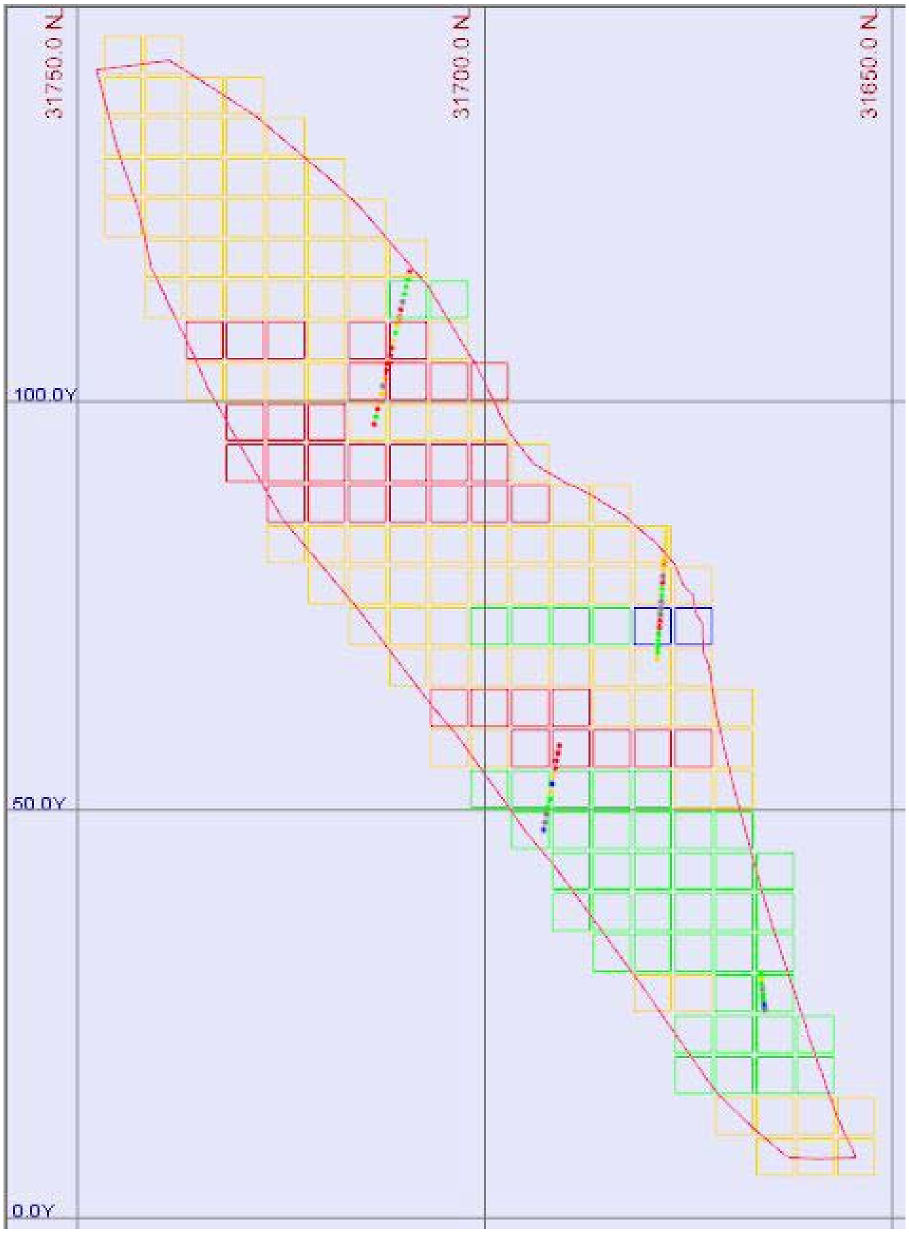


Figure 14-32: Composite Data and Block Model, Section 32187, Looking East

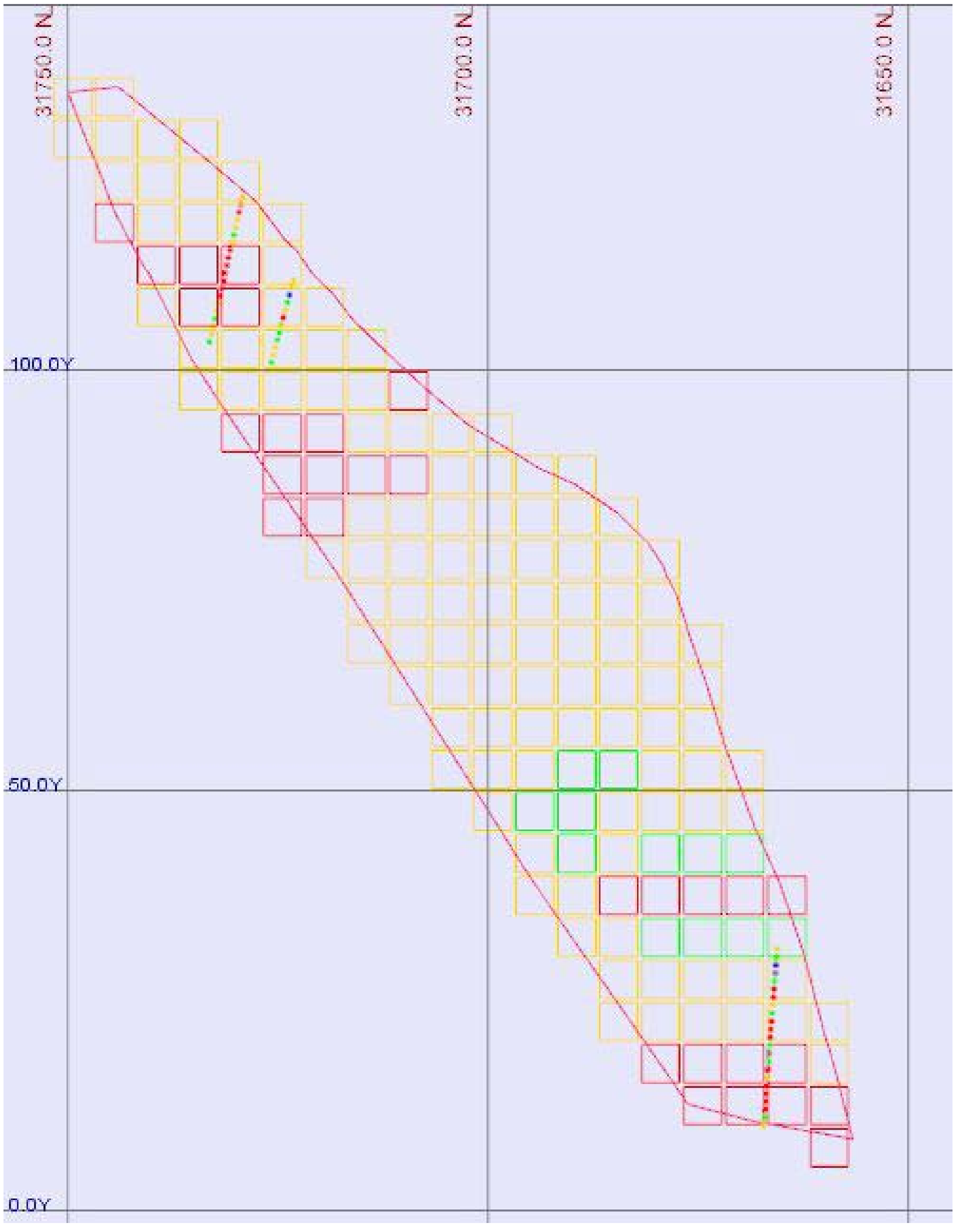


Figure 14-33: Composite Data and Block Model, Section 32217, Looking East

14.2.10.2 Global Comparison

The global block model statistics for the block model were compared to the composite drillhole data (Table 14-16). The average grades from the block model are generally slightly lower than the ones from the composites (with a difference lower than 10%). No significant discrepancies are shown in the global comparison.

Table 14-16: Upper Offset Southeast Extension Global Statistics Comparison

Element	Attribute	Average	Variance	Standard Deviation
PD	Block model	3.187	1.460	1.208
	Composites	3.378	6.752	2.598
PT	Block model	0.284	0.009	0.094
	Composites	0.302	0.043	0.208
AU	Block model	0.239	0.009	0.093
	Composites	0.256	0.040	0.199
NI	Block model	0.091	0.001	0.024
	Composites	0.097	0.002	0.046
CU	Block model	0.077	0.001	0.034
	Composites	0.085	0.004	0.062

14.2.11 Previous Estimates

This is the first estimate for the Upper Offset Southeast extension as a newly identified zone, separated from the Offset zone described in Section 14.1.

Some of this material may have been previously reported in the main Offset zone. It was a portion of the Offset footwall zone in the previous resource estimate (McKinnon et al., 2014). All material of the Upper Offset Southeast zone has been removed from the main Offset zone described in Section 14.1

14.3 Underground Roby Zone

No changes are reported from the previous estimate (McKinnon et al., 2014).

14.4 Roby Zone Open Pit and Pit Expansion Resources

14.4.1 Introduction

The most recent, detailed resource estimates for the near-surface portion of the Roby Zone deposit are discussed in Buck et al. (2010). Since the release of that report, the Roby Zone open pit resources have been depleted (no additions) using a simple arithmetic depletion method by David N. Penna, P.Geo., an employee of NAP.

As part of the current report, a new resource estimate based on previous NI 43-101 compliant resource estimates and block models for the open pit portion of the Roby Zone deposit has been prepared. This new resource estimate reflects mining depletion to December 31, 2013 (no mining occurred in the Roby open pit in 2014) and includes material largely occurring outside the current pit shell as defined in Pincock, Allen and Holt (2003). The new resource estimate for the Roby pit expansion also reflects changes in the Company's long-term metal price and foreign exchange assumptions that support the application of a lower cut-off grade for this specific resource. The new resources estimates listed in Section 14.4.9 are referred to as the Roby Zone open pit expansion resources.

14.4.2 Database

Not applicable – refer to Pincock, Allen and Holt (2003).

14.4.3 Specific Gravity

Refer to Section 14.1.3.

14.4.3.1 Geological Interpretation

Not applicable – refer to Pincock, Allen and Holt (2003).

14.4.4 Exploratory Data Analysis

Not applicable – refer to Pincock, Allen and Holt (2003).

14.4.5 Spatial Analysis

Not applicable – refer to Pincock, Allen and Holt (2003).

14.4.6 Resource Block Model

Not applicable – refer to Buck et al. (2010). There has been no new drilling that would affect the 2003 open pit block model described in Pincock, Allen and Holt (2003). The revised resource estimate presented in Section 14.4.9 is based on the 2003 block model that was provided as a Gemcom GEMS™ block model by Dave Penna, a QP under NI 43-101 and an employee of LDIM in order to generate the updated mineral resource.

14.4.7 Resource Classification

Not applicable – refer to Pincock, Allen and Holt (2003) and Buck et al. (2010).

14.4.8 Mineral Resource Tabulation

NAP has reviewed the Roby Zone open pit and pit expansion mineral resources after accounting for mining depletion to December 31, 2013.

A Gemcom GEMS™ block model for historical estimates of the Roby Zone open pit resources was provided by Dave Penna, a QP under NI 43-101 and an employee of LDIM in order to generate the updated mineral resource.

The following constraints have been applied for the mineral resource reporting:

- A block cut-off grade of 0.6 g/t palladium was applied.
- A bottom limit of the resource at 235 m (level 265) has been applied (limit between Roby open pit zone and Roby underground zone).
- The actual pit surface was used as a top limit for the resource (to reflect mining depletion in the open pit).
- Major mined-out ramps inside the resource zone were removed.
  - The Roby open pit reserve volume, defined in McKinnon et al. (2014), has been excluded.

Table 14-17 presents the mineral resource table for the Roby Zone open pit expansion resources for the Mineral resource presented here are exclusive of mineral reserves.

Table 14-17: Roby Zone Open Pit Expansion Resources at a 0.6 g/t Palladium Cut-off Grade

Resource Category	Volume (000's m^3)	Density	Tonnes (000's)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Contained Pd (000's oz)
Measured	6,957	2.89	20,108	1.23	0.17	0.10	0.07	0.05	795
Indicated	3,334	2.89	9,634	1.20	0.17	0.10	0.07	0.05	372
Measured + Indicated	10,290	2.89	29,742	1.22	0.17	0.10	0.07	0.05	1,167

14.4.9 *Validation*

Not applicable – refer to Pincock, Allen and Holt (2003).

14.4.10 *Previous Estimates*

Since 2003, all of the published resource estimates for the Roby open pit zone relate to the Roby Zone block model produced in 2003 (Pincock, Allen and Holt 2003).

14.5 Powerline Zone

14.5.1 *Introduction*

Chris Roney, P. Geo. and a consultant to NAP, competed a resource estimation of the Powerline Zone at LDIM. The new resource model described below has an effective date of February 2, 2015.

14.5.2 *Database*

NAP maintains all borehole data in a Century Systems Fusion Server®. Header, survey, lithology, and assays tables are saved in the database. A copy of the header, survey, lithology and assays was provided to Chris Roney on January 22, 2015.

The files provided to Chris Roney contained the all the data for surface boreholes for LDIM property. The dataset includes all the surface holes for the Powerline Zone as of January 22, 2015 (cut-off date). The final dataset used for interpretation of the Powerline Zone contained 29 boreholes. In addition to the borehole assay data, NAP had amassed a large surface channel sample database. However, the channel sample assays were completed at NAP's mine laboratory and were considered to be non-complaint, therefore have not been included in the current resource estimate. The resource estimation below was completed using Gemcom GEMS™ 6.6 version software.

14.5.3 Specific Gravity

A specific gravity of 2.89 was used for the resource estimate. A specific gravity of 2.89 is within the accepted range of a gabbro with low-sulphide content.

NAP's exploration group routinely collect specific gravity measurements for all core assay samples using the procedure recommended by Tetra Tech in 2012 (McCracken et al., -2013). However, at the time that the Powerline Zone resource was estimated, the specific gravity data for the relevant boreholes was not all available.

14.5.4 Geological Interpretation

3D wireframe model of palladium mineralization (mineralized envelopes) for the Powerline Zone was developed by Chris Roney in Gemcom GEMS™ software for the Powerline Zone in the Sheriff North Zone area. The basis for the wireframe included minimum grade of 2.0 g/t palladium and minimized waste inclusions.

Sectional interpretations were created in Gemcom GEMS™ software and these interpretations were linked with tie strings and triangulated to build a 3D solid. The solid created has a volume of 176,448.77 m^3. The solid was validated in Gemcom GEMS™ software and no errors were found.

The zone of mineralization interpreted for the Powerline Zone was generally contiguous, however, due to the nature of the mineralization there were portions of the wireframe that had grades less than 2.0 g/t palladium, yet were still within the mineralizing trend. Figure 14-34 shows the spatial relationship of the Powerline Zone in relation to Roby Pit including the previously mined Twilight Zone.

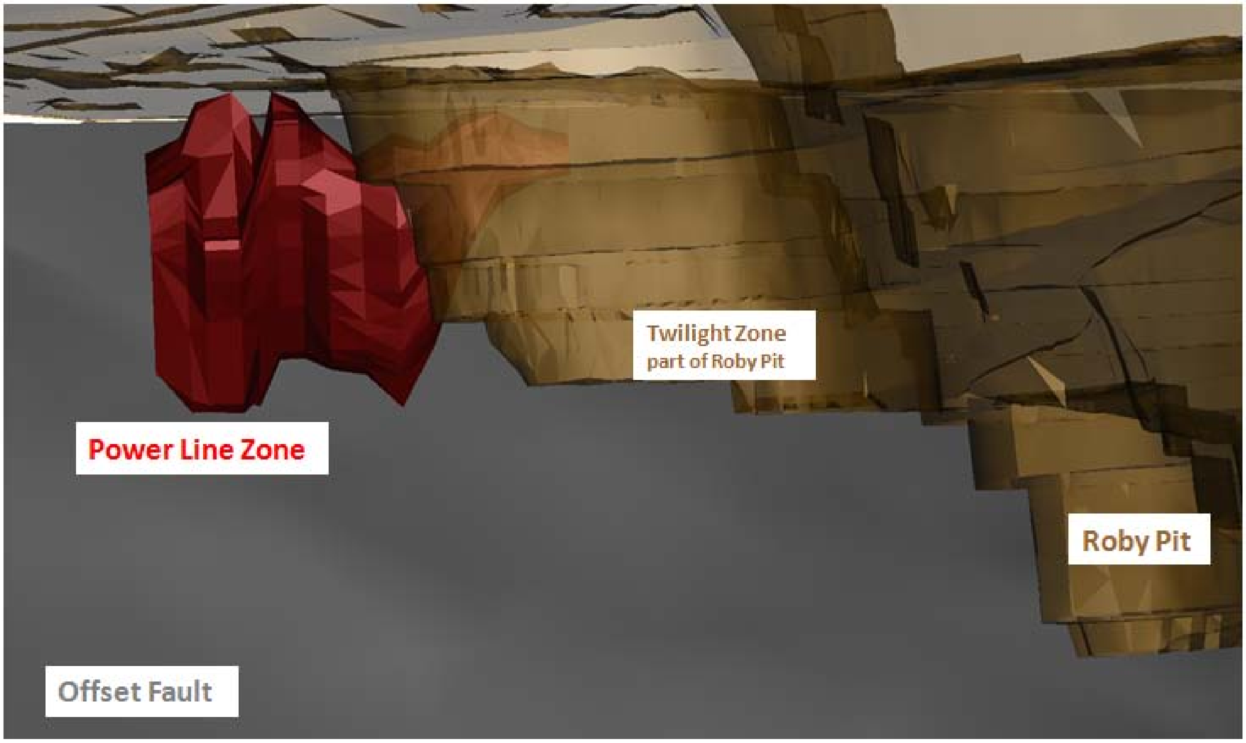


Figure 14-34: Oblique Longitudinal Section Looking South, Showing 3D Shell for the Powerline Zone in Relation to the Roby Pit

14.5.5 Exploratory Data Analysis

14.5.5.1 Assays

The portion of the deposit included in the mineral resource was based on total of 1,138 palladium assays. The assay intervals within the zone were captured using Gemcom GEMS™ cross-table manipulation into individual borehole files. These boreholes files were reviewed to ensure that all relevant assay intervals were captured. Table 14-18 summarizes the basic statistics for the assays in the Powerline Zone for the 2 g/t wireframe. The non-assayed intervals were assigned void (-) values. NAP believes that non-assayed material should not be assigned a zero value, as this does not reflect the true value of the material.

A review of the correlation ratio between the various elements was completed to determine if any strong correlation existed in order to allow for common variables to be utilized during the estimation process. Table 14-19 summarizes the results of this review. As the correlation between palladium and platinum and between nickel and copper are high it was decided to use a single interpolation profile for each of these pairs of elements.

Table 14-18: Summary of Powerline Zone Assay Basic Statistics and Capped Metals

VARIABLE	PD	PT	AU	NI	CU	CO	LENGTH	PD_CUT	PT_CUT	AU_CUT	NI_CUT
Mean	**2.27**	**0.18**	**0.09**	**0.05**	**0.04**	**0.01**	**1.06**	**2.21**	**0.18**	**0.09**	**0.05**
Standard Error	**0.117**	0.006	0.005	0.001	0.001	0.000	0.013	0.108	0.006	0.004	0.001
Median	**0.912**	0.111	0.039	0.047	0.026	0.006	1.000	0.910	0.110	0.040	0.050
Mode	**0.015**	0.003	0.003	0.012	0.007	0.004	1.000	0.010	0.010	0.000	0.010
Standard Deviation	**3.96**	0.216	0.154	0.045	0.046	0.003	0.448	3.650	0.214	0.142	0.042
Sample Variance	**15.7**	0.047	0.024	0.002	0.002	0.000	0.200	13.322	0.046	0.020	0.002
Kurtosis	**14.0**	7.19	32.5	8.25	3.47	71.2	11.6	8.11	6.49	21.2	3.22
Skewness	**3.38**	2.34	4.87	2.17	1.73	4.52	3.23	2.83	2.26	4.04	1.54
Range	**36.6**	1.43	1.53	0.363	0.277	0.052	3.25	18.3	1.25	1.09	0.220
Minimum	**0.001**	0.003	0.001	0.001	0.000	0.001	0.250	0.000	0.000	0.000	0.000
Maximum	**36.6**	1.43	1.53	0.364	0.277	0.053	3.50	18.3	1.25	1.09	0.220
Count	**1,138**	1,138	1,138	1,138	1,138	1,138	1,138	1,138	1,138	1,138	1,138
Confidence Level (95.0%)	**0.230**	0.013	0.009	0.003	0.003	0.000	0.026	0.212	0.012	0.008	0.002
Coefficient of Variation	**1.75**	1.22	1.74	0.84	1.06	0.48	0.42	1.65	1.21	1.63	0.80

Table 14-19: Coefficients of Correlation between Elements

	Pt	Pd	Au	Cu	Ni	Co	Pd_Cap	Au_Cap	Pt_Cap	Ni_Cap
Pt	1	0.8858	0.7465	0.4777	0.5274	0.1642	0.9016	0.754	0.9976	0.6743
Pd	0.8858	1	0.5825	0.2903	0.3659	0.1045	0.9903	0.5857	0.8853	0.4742
Au	0.7465	0.5825	1	0.5137	0.5447	0.1445	0.5897	0.993	0.7445	0.6916
Cu	0.4777	0.2903	0.5137	1	0.8818	0.6771	0.3071	0.5257	0.4831	0.6966
Ni	0.5274	0.3659	0.5447	0.8818	1	0.7808	0.3825	0.5531	0.5327	0.8661
Co	0.1642	0.1045	0.1445	0.6771	0.7808	1	0.1093	0.149	0.1664	0.5641
Pd_Cap	0.9016	0.9903	0.5897	0.3071	0.3825	0.1093	1	0.5976	0.9023	0.4954
Au_Cap	0.754	0.5857	0.993	0.5257	0.5531	0.149	0.5976	1	0.7567	0.7053
Pt_Cap	0.9976	0.8853	0.7445	0.4831	0.5327	0.1664	0.9023	0.7567	1	0.6818
Ni_Cap	0.6743	0.4742	0.6916	0.6966	0.8661	0.5641	0.4954	0.7053	0.6818	1

14.5.5.2 Grade Capping

The raw assay data for the 2 g/t palladium wireframe was examined to assess the amount of metal that is at risk from high-grade assays. Histograms and Cumulative Probability (%) Plots were used to determine if grade capping was required for each element.

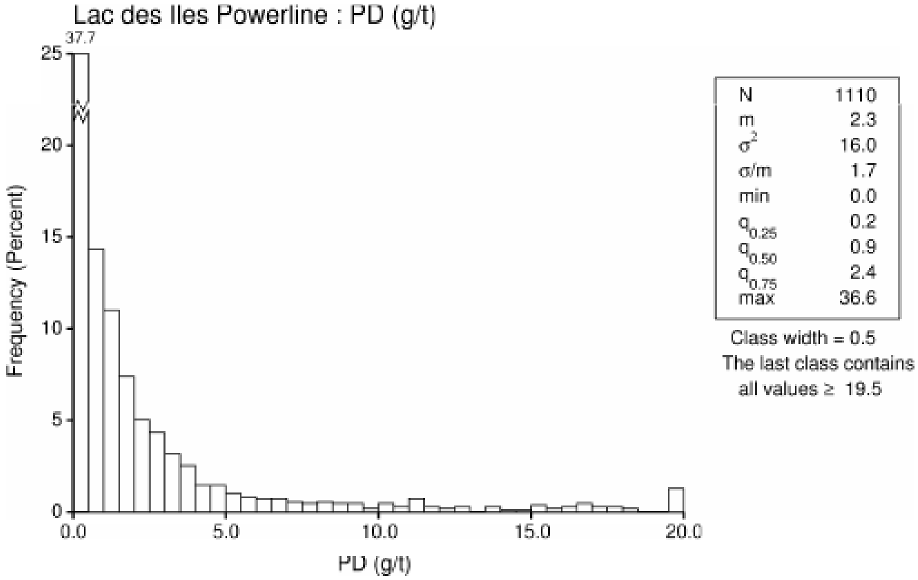
Figure 14-34 to Figure 14-38 show the results of the Histograms and Cumulative Probability (%) Plots. The results of the plots and analysis indicate that grade capping is required for palladium, platinum, gold and nickel. Maximum values of 18.25 g/t for palladium, 1.25 g/t for platinum, 1.09 g/t for gold and 0.22 % for nickel. A summary of the capped metals basic statistics is given in Table 14-18.

14.5.5.3 Compositing

Compositing of all assay data was completed into the 2 g/t Pd wireframe. Compositing was done to 2 m and 3 m intervals. See Table 14-20 for basic statistic of results. The same composite interval of 2m was used as it was in previous estimate completed by Tetra Tech for Sheriff Zone (McCracken et al, 2012). Composite points were extracted from the wireframe.

14.5.6 Spatial Analysis

The 2m composite data was used to create all variograms. Gemcom GEMS™ used to create downhole variograms to determine nugget effect for elements. Subsequently, 3-D semi-variograms were created using Gemcom GEMS™ for palladium-platinum, gold, nickel - copper and cobalt. Figure 14-38 shows the variogram for palladium-platinum. Table 14-21 summarizes the results of the variography done on palladium, gold, nickel and cobalt.



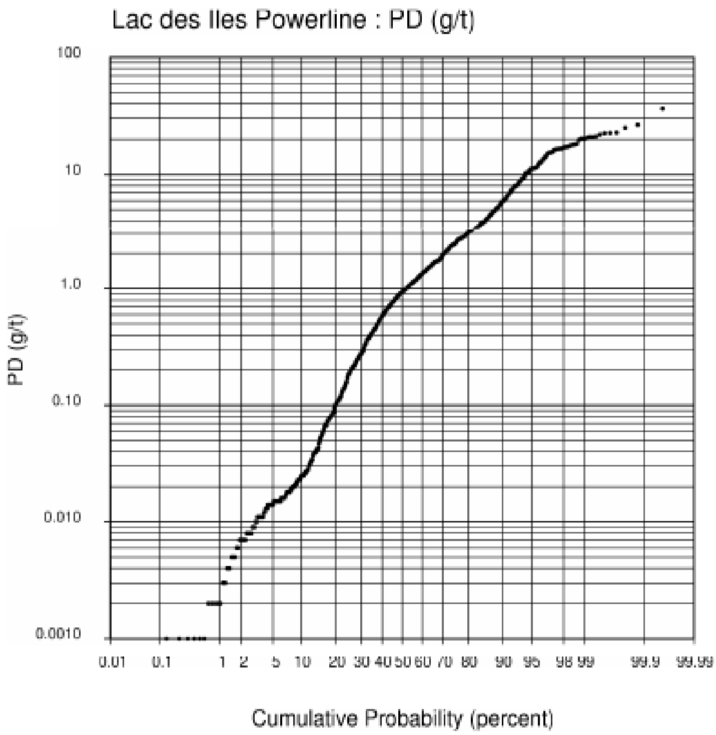


Figure 14-35: Histogram and Cumulative Probability Plot of Raw Powerline Zone Pd Assay Data

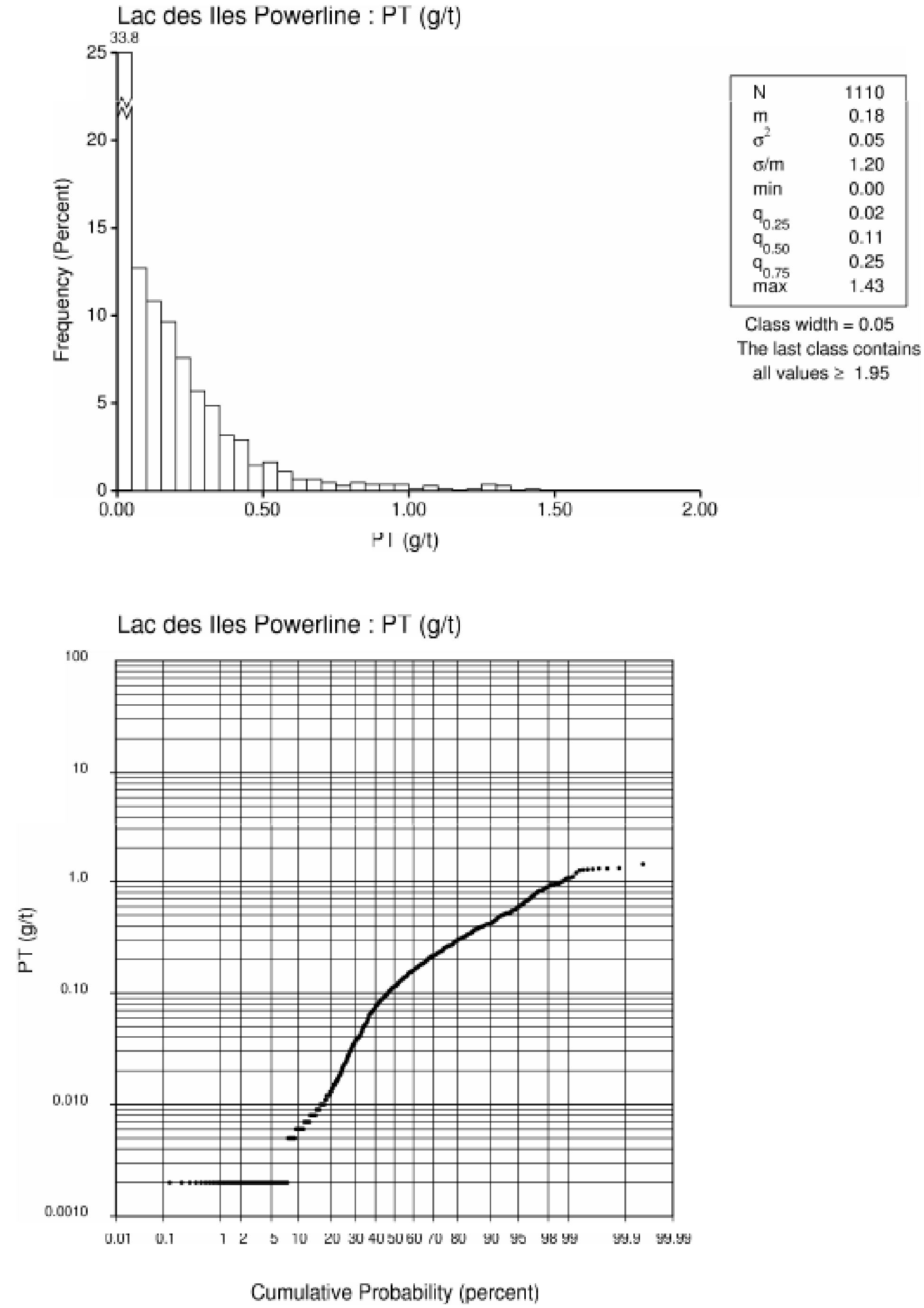


Figure 14-36: Histogram and Cumulative Probability Plot of Raw Powerline Zone Pt Assay Data

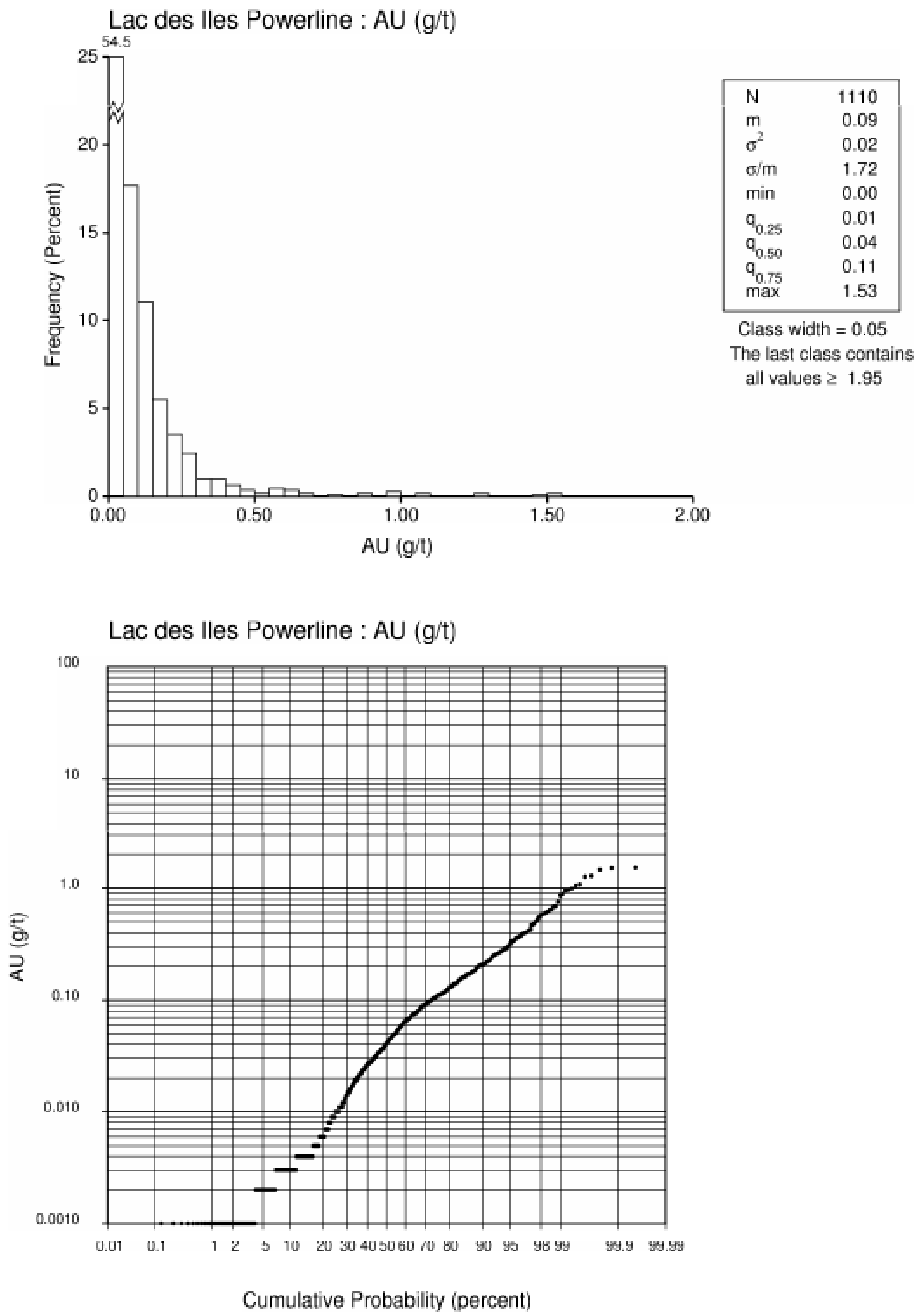
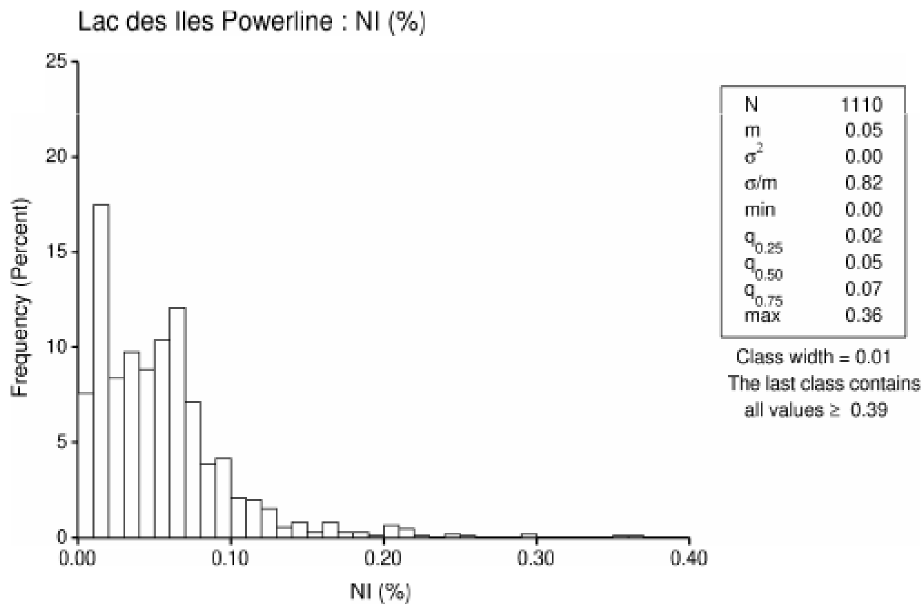


Figure 14-37: Histogram and Cumulative Probability Plot of Raw Powerline Zone Au Assay Data



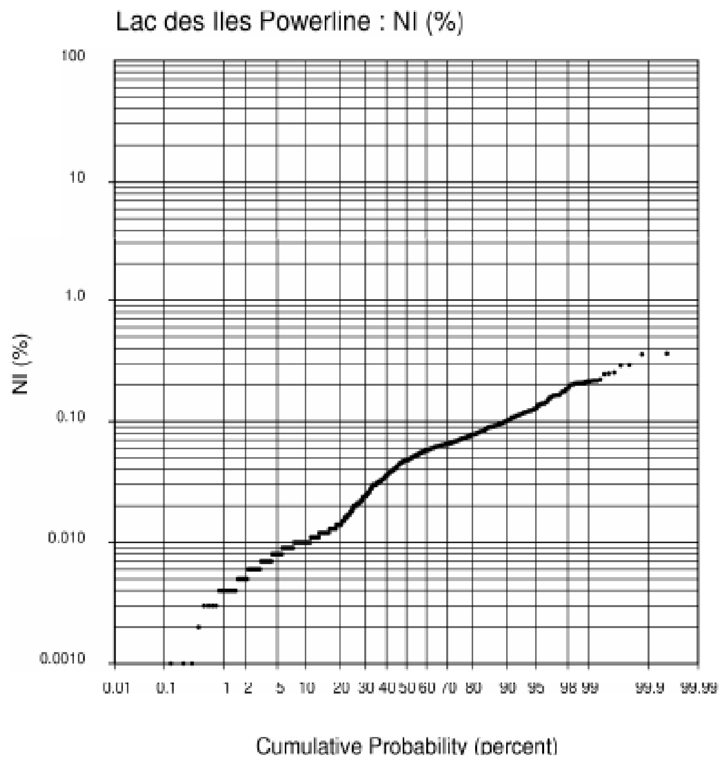


Figure 14-38: Histogram and Cumulative Probability Plot of Raw Powerline Zone Ni Assay Data

Table 14-20: Summary of Powerline Zone Composited Assay Basic Statistics and Capped Metals

2M COMPOSITE -Variable	PD	PT	AU	NI	CU	CO	CLENGTH	PD_CUT	PT_CUT	AU_CUT	NI_CUT
Mean	2.30	0.181	0.092	0.053	0.044	0.006	2.00	2.25	0.181	0.090	0.052
Standard Error	0.156	0.008	0.006	0.002	0.002	0.000	0.002	0.144	0.008	0.005	0.002
Median	1.06	0.130	0.050	0.050	0.030	0.010	2.000	1.06	0.130	0.050	0.050
Mode	0.020	0.010	0.010	0.010	0.010	0.010	2.00	0.020	0.010	0.010	0.010
Standard Deviation	3.71	0.196	0.138	0.039	0.039	0.005	0.038	3.420	0.195	0.128	0.037
Sample Variance	13.7	0.039	0.019	0.002	0.002	0.000	0.002	11.694	0.038	0.016	0.001
Kurtosis	12.6	6.03	26.5	6.23	1.82	-1.87	282	8.07	5.64	18.6	2.70
Skewness	3.26	2.16	4.27	1.77	1.33	-0.375	-16.6	2.79	2.11	3.60	1.33
Range	29.5	1.18	1.29	0.330	0.230	0.010	0.700	18.3	1.14	1.06	0.220
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	1.30	0.00	0.00	0.00	0.00
Maximum	29.5	1.18	1.29	0.330	0.230	0.010	2.00	18.3	1.14	1.06	0.220
Count	566.0	566.0	566.0	566.0	566.0	566.0	566.0	566.0	566.0	566.0	566.0
Confidence Level (95.0%)	0.306	0.016	0.011	0.003	0.003	0.000	0.003	0.282	0.016	0.011	0.003
Coefficient of Variation	1.61	1.08	1.50	0.739	0.900	0.831	0.019	1.52	1.08	1.42	0.710

3M COMPOSITE -Variable	PD	PT	AU	NI	CU	CO	CLENGTH	PD_CUT	PT_CUT	AU_CUT	NI_CUT
Mean	2.32	0.182	0.092	0.053	0.043	0.006	2.99	2.26	0.181	0.091	0.052
Standard Error	0.189	0.010	0.007	0.002	0.002	0.000	0.005	0.174	0.010	0.006	0.002
Median	1.10	0.130	0.060	0.050	0.030	0.010	3.00	1.100	0.130	0.060	0.050
Mode	0.030	0.010	0.010	0.010	0.010	0.010	3.00	0.030	0.010	0.010	0.010
Standard Deviation	3.64	0.192	0.133	0.038	0.037	0.005	0.087	3.37	0.190	0.124	0.036
Sample Variance	13.3	0.037	0.018	0.002	0.001	0.000	0.008	11.3	0.036	0.015	0.001
Kurtosis	12.1	5.81	23.7	6.277	1.72	-1.82	202	8.02	5.37	17.2	2.65
Skewness	3.20	2.10	4.07	1.78	1.28	-0.440	-13.6	2.76	2.04	3.48	1.30
Range	26.9	1.20	1.15	0.300	0.210	0.010	1.42	18.2	1.14	1.01	0.210
Minimum	0.010	0.000	0.000	0.000	0.000	0.000	1.58	0.010	0.000	0.000	0.000
Maximum	26.9	1.20	1.15	0.300	0.210	0.010	3.00	18.3	1.14	1.01	0.210
Count	374	374	374	374	374	374	374	374	374	374	374
Confidence Level (95.0%)	0.371	0.019	0.014	0.004	0.004	0.000	0.009	0.342	0.019	0.013	0.004
Coefficient of Variation	1.57	1.06	1.45	0.72	0.86	0.81	0.03	1.49	1.05	1.37	0.69

*Pd, Pt and Au are reported in g/t and Mo. Cu and Cp are reported in weight percent.

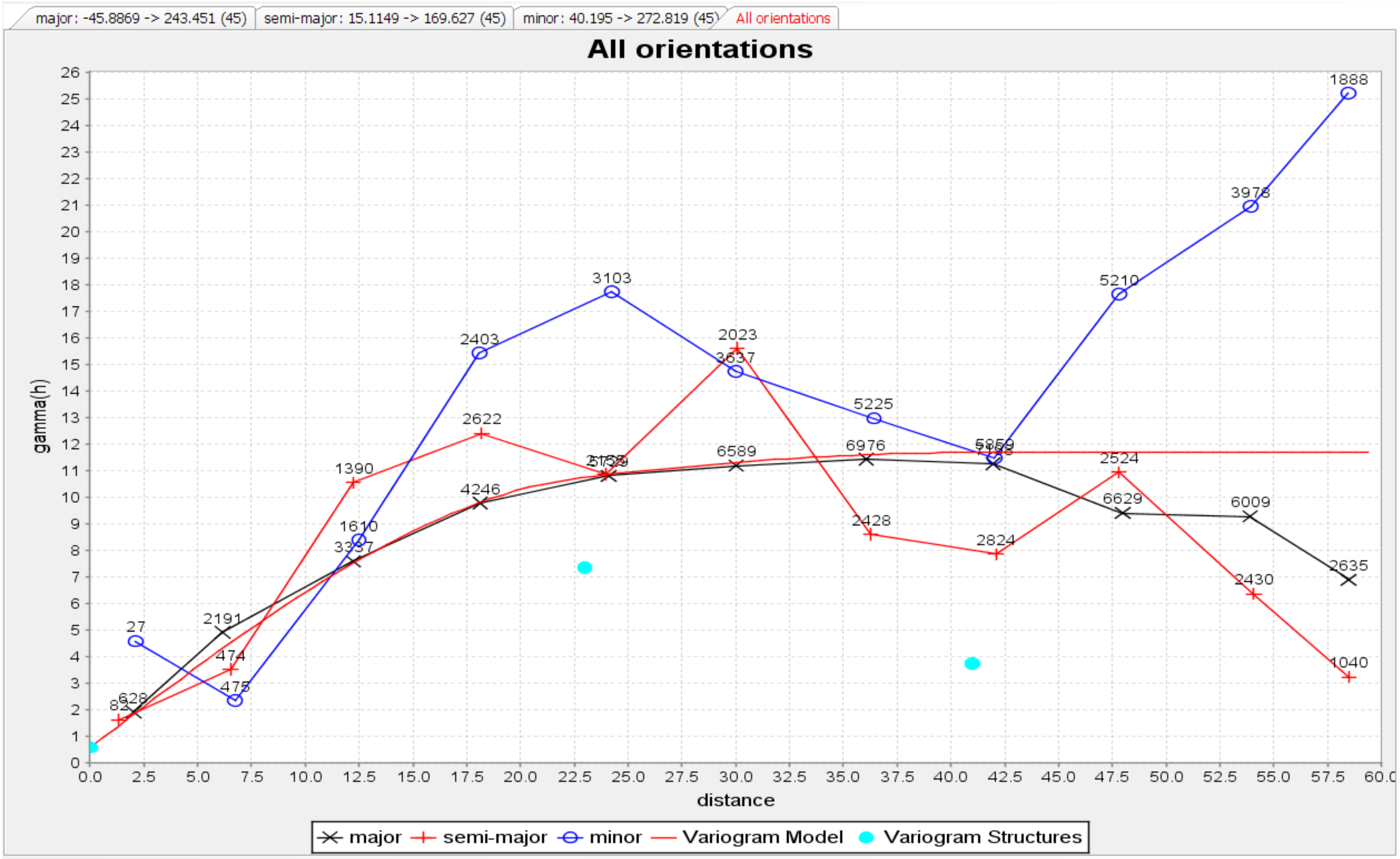


Figure 14-39: Palladium Variogram (Platinum)

Table 14-21: Summary of Variogram Parameters

Element	Principal Azimuth	Principal Dip	Intermediate Azimuth	Nugget	Structure 1 Type	Structure 1 range (X)	Structure 1 range (Y)	Structure 1 range (Z)	Structure 2 Type	Structure 2 range (X)	Structure 2 range (Y)	Structure 2 range (Z)	Total Sill
PD	242.4	-46.58	168.8	0.585	Spherical	22.99	12.09	7.683	Spherical	41.00	21.57	13.70	11.70
AU	190.0	-60.00	110.4	0.0033	Spherical	15.06	10.856	8.924	-	-	-	-	0.013
NI	228.9	-53.41	130.1	0.00007	Spherical	6.541	4.537	3.360	Spherical	15.15	10.51	7.783	0.0014
CO	230.3	-52.86	101.8	0.0000002	Spherical	6.645	4.337	3.592	Spherical	24.86	16.221	13.44	0.000004

14.5.7 Resource Block Model

A 10m by 10m by 10m block model was created for resource evaluation. It was copied from the Sheriff block model (McCracken et al, 2012). Each block has a rock code assigned from the wireframe. The block model is a percent model. Table 14-22 shows the Gemcom GEMS™ coordinates for the block model origins. The coordinate system used is a local (mine) metric system.

The block model was copied from Sheriff block model to make it easier to subtract the Power Line resource from the Sheriff resource.

Table 14-22: Block Model Properties

Origin			Cell Size			Number of Cells		
X Origin	Y Origin	Z Origin	XINC	YINC	ZINC	NX	NY	NZ
31500	31000	-200	10	10	10	175	150	100

14.5.7.1 Estimation Search Parameters

The interpolation profiles were defined for each element. Ordinary Kriging was used with variograms found during the variography study (see Table 14-21). Interpolation parameters used are described in Table 14-23, and search ellipses parameters are described in Table 14-24. Three different passes were used for palladium interpolation. Anistropic Inverse Distance (ID2) and Nearest Neighbour (NN) interpolation profiles were created as well, for validation purposes. Palladium variogram and search ellipse were used for platinum. Nickel variogram and search ellipse used to copper, with gold and cobalt having their own.

The third search ellipse maximum distance was double for gold, nickel and copper due to the short ranges. This allows grade information to be interpolated into the same number of blocks as the palladium and platinum. Over all grades for gold, nickel and copper were not increased.

Table 14-23: Interpolation Parameters for 2 g/t Palladium Shell

Element	Minimum Composite Number	Maximum Composite Number	Maximum Composite per Hole	Maximum Search Radius (m)
Pd (first pass)	10	25	3	41
Pd (second pass)	8	25	3	41
Pd (third pass)	4	25	3	41
Pt (first pass)	10	25	3	41
Pt (second pass)	8	25	3	41
Pt (third pass)	4	25	3	41
Au (first pass)	10	25	3	15
Au (second pass)	8	25	3	15
Au (third pass)	4	25	3	30
Cu (first pass)	10	25	3	15
Cu (second pass)	8	25	3	15
Cu (third pass)	4	25	3	30
Ni (first pass)	10	25	3	15
Ni (second pass)	8	25	3	15
Ni (third pass)	4	25	3	30
Co (first pass)	10	25	3	25
Co (second pass)	8	25	3	25
Co (third pass)	4	25	3	25

Table 14-24: Search Ellipse Parameters

Element	Principal Azimuth (º)	Principal Dip (º)	Intermediate Azimuth (º)	Anisotropy (X) Metres	Anisotropy (y) Metres	Anisotropy (z) Metres
PD	242.41	-46.58	168.76	41.00	21.57	13.70
PT	242.41	-46.58	168.76	41.00	21.57	13.70
AU	190.00	-60.00	110.43	15.06	10.86	8.92
NI	228.94	-53.41	130.07	15.15	10.51	7.83
CU	228.94	-53.41	130.07	15.15	10.51	7.83
CO	230.33	-52.86	101.79	24.85	16.22	13.44

14.5.8 Resource Classification

Several factors are considered in the definition of a resource classification:

- NI-43-101 requirements.
- Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines.
- The author's experience with doing resources and previous experience with doing ultramafic-mafic hosted PGE-Cu-Ni deposits resource on LDI Mine Property (see North VT Resource).
- Spatial continuity based on variography of the assays within the bore holes.

- Bore hole spacing and estimation runs required to estimate the grades in a grade block.
- The number of samples and bore holes used in each of the block estimations.

No environmental, permitting, legal, title, taxation, socio-economic, marking or other relevant issues are known to Chris Roney, P. Geo. that may affect the estimate of mineral resources. Mineral reserves can only be estimated on the basis of economic evaluation that is used in preliminary feasibility study or a feasibility study of mineral project; thus, no reserves have been estimated. As per NI-43-101 guidelines, mineral resources, which are not mineral reserves, do not have to demonstrate economic viability.

14.5.9 Mineral Resource Tabulation

The resource reported as of February 2, 2015 has been tabulated in terms of a palladium cut-off grade. Table 14-25 summarizes the resource estimate for the Powerline Zone (exclusive of reserves; no reserves have been declared for the Powerline Zone). Table 14-26 to Table 14-28 show the grade-tonnage tables for the Powerline Zone for each resource category. The resources are tabulated using various cut-off grades to demonstrate the robust nature of the resource. Based on the current and future operation plans at the mine for near-surface material a 1.0 g/t palladium cut-off grade was used to tabulate the resource. The distribution of the resources categories is displayed in Figure 14-40.

Table 14-25: Powerline Zone Mineral Resource Summary for 1.0 g/t Palladium Cut-off Grade

Classification	Cut-off	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ni (%)	Co (%)
Measured	1.0	251,000	3.04	0.22	0.12	0.04	0.06	0.006
Indicated	1.0	64,860	2.48	0.20	0.08	0.04	0.05	0.004
Total Measured + Indicated		315,900	2.93	0.21	0.11	0.04	0.06	0.006
Inferred	1.0	50,40	3.10	0.28	0.09	0.05	0.05	0.004
Total Inferred		50,740	3.10	0.28	0.09	0.05	0.05	0.004

Table 14-26: Powerline Zone Measured Resource Grade-Tonnage

PD (g/t) Cut-off Grade	TONNES	PD (g/t)	PT (g/t)	AU (g/t)	CU (%)	NI (%)	CO (%)
3.00	99,540	5.22	0.33	0.15	0.04	0.07	0.007
2.50	111,900	4.96	0.32	0.15	0.04	0.07	0.006
2.00	130,700	4.56	0.30	0.14	0.04	0.07	0.006
1.90	139,000	4.40	0.29	0.14	0.04	0.07	0.006
1.80	152,200	4.18	0.28	0.14	0.04	0.06	0.006
1.70	161,700	4.04	0.27	0.14	0.04	0.06	0.006
1.60	164,200	4.00	0.27	0.14	0.04	0.06	0.006
1.50	172,600	3.88	0.26	0.14	0.04	0.06	0.006
1.40	179,100	3.79	0.26	0.13	0.04	0.06	0.006
1.30	186,600	3.70	0.25	0.13	0.04	0.06	0.006
1.20	203,100	3.50	0.24	0.13	0.04	0.06	0.006
1.10	232,200	3.20	0.23	0.12	0.04	0.06	0.006
1.00	251,000	3.04	0.22	0.12	0.04	0.06	0.006
0.90	264,200	2.94	0.21	0.11	0.04	0.06	0.006
0.80	282,000	2.80	0.20	0.11	0.04	0.06	0.006
0.70	295,500	2.71	0.20	0.10	0.04	0.06	0.006
0.60	310,300	2.61	0.19	0.10	0.04	0.05	0.006
0.50	326,000	2.51	0.19	0.10	0.04	0.05	0.006
0.40	335,300	2.46	0.18	0.10	0.04	0.05	0.006
0.00	338,300	2.44	0.18	0.10	0.04	0.05	0.006

Table 14-27: Powerline Zone Indicated Resource Grade-Tonnage

PD (g/t) Cut-off Grade	TONNES	PD (g/t)	PT (g/t)	AU (g/t)	CU (%)	NI (%)	CO (%)
3.00	10,150	7.51	0.43	0.11	0.03	0.06	0.006
2.50	12,300	6.71	0.40	0.12	0.03	0.06	0.006
2.00	19,040	5.10	0.33	0.12	0.03	0.05	0.004
1.90	19,040	5.10	0.33	0.12	0.03	0.05	0.004
1.80	23,420	4.49	0.30	0.11	0.03	0.05	0.003
1.70	24,350	4.38	0.29	0.11	0.03	0.05	0.003
1.60	35,440	3.53	0.25	0.10	0.04	0.06	0.003
1.50	35,440	3.53	0.25	0.10	0.04	0.06	0.003
1.40	41,210	3.23	0.24	0.10	0.04	0.05	0.004
1.30	45,920	3.04	0.23	0.09	0.04	0.05	0.004
1.20	47,970	2.96	0.23	0.09	0.04	0.05	0.004
1.10	58,670	2.63	0.21	0.08	0.04	0.05	0.004
1.00	64,860	2.48	0.20	0.08	0.04	0.05	0.004
0.90	70,950	2.35	0.19	0.08	0.04	0.05	0.004
0.80	73,100	2.31	0.19	0.08	0.04	0.05	0.004
0.70	74,800	2.27	0.19	0.08	0.04	0.05	0.004
0.60	74,800	2.27	0.19	0.08	0.04	0.05	0.004
0.50	79,180	2.18	0.18	0.08	0.04	0.05	0.004
0.40	80,310	2.15	0.18	0.08	0.04	0.05	0.004
0.00	80,360	2.15	0.18	0.08	0.04	0.05	0.004

Table 14-28: Powerline Zone Inferred Resource Grade-Tonnage

PD (g/t) Cut-off Grade	TONNES	PD (g/t)	PT (g/t)	AU (g/t)	CU (%)	NI (%)	CO (%)
3.00	10,730	8.59	0.59	0.12	0.04	0.06	0.004
2.50	11,840	8.03	0.56	0.11	0.03	0.06	0.004
2.00	16,970	6.30	0.46	0.10	0.03	0.05	0.003
1.90	22,490	5.23	0.40	0.09	0.04	0.05	0.004
1.80	22,490	5.23	0.40	0.09	0.04	0.05	0.004
1.70	28,010	4.54	0.36	0.09	0.04	0.05	0.004
1.60	30,550	4.30	0.34	0.09	0.04	0.05	0.004
1.50	34,800	3.96	0.33	0.09	0.04	0.05	0.004
1.40	36,960	3.82	0.32	0.10	0.04	0.05	0.004
1.30	36,960	3.82	0.32	0.10	0.04	0.05	0.004
1.20	45,470	3.33	0.29	0.09	0.05	0.05	0.004
1.10	47,300	3.25	0.28	0.09	0.04	0.05	0.004
1.00	50,740	3.10	0.28	0.09	0.05	0.05	0.004
0.90	57,700	2.84	0.25	0.09	0.05	0.05	0.004
0.80	58,080	2.83	0.25	0.09	0.05	0.05	0.004
0.70	64,940	2.61	0.24	0.08	0.04	0.05	0.004
0.60	65,740	2.58	0.23	0.08	0.04	0.05	0.004
0.50	66,430	2.56	0.23	0.08	0.04	0.05	0.004
0.40	66,620	2.56	0.23	0.08	0.04	0.05	0.004
0.00	67,400	2.53	0.23	0.08	0.04	0.05	0.004

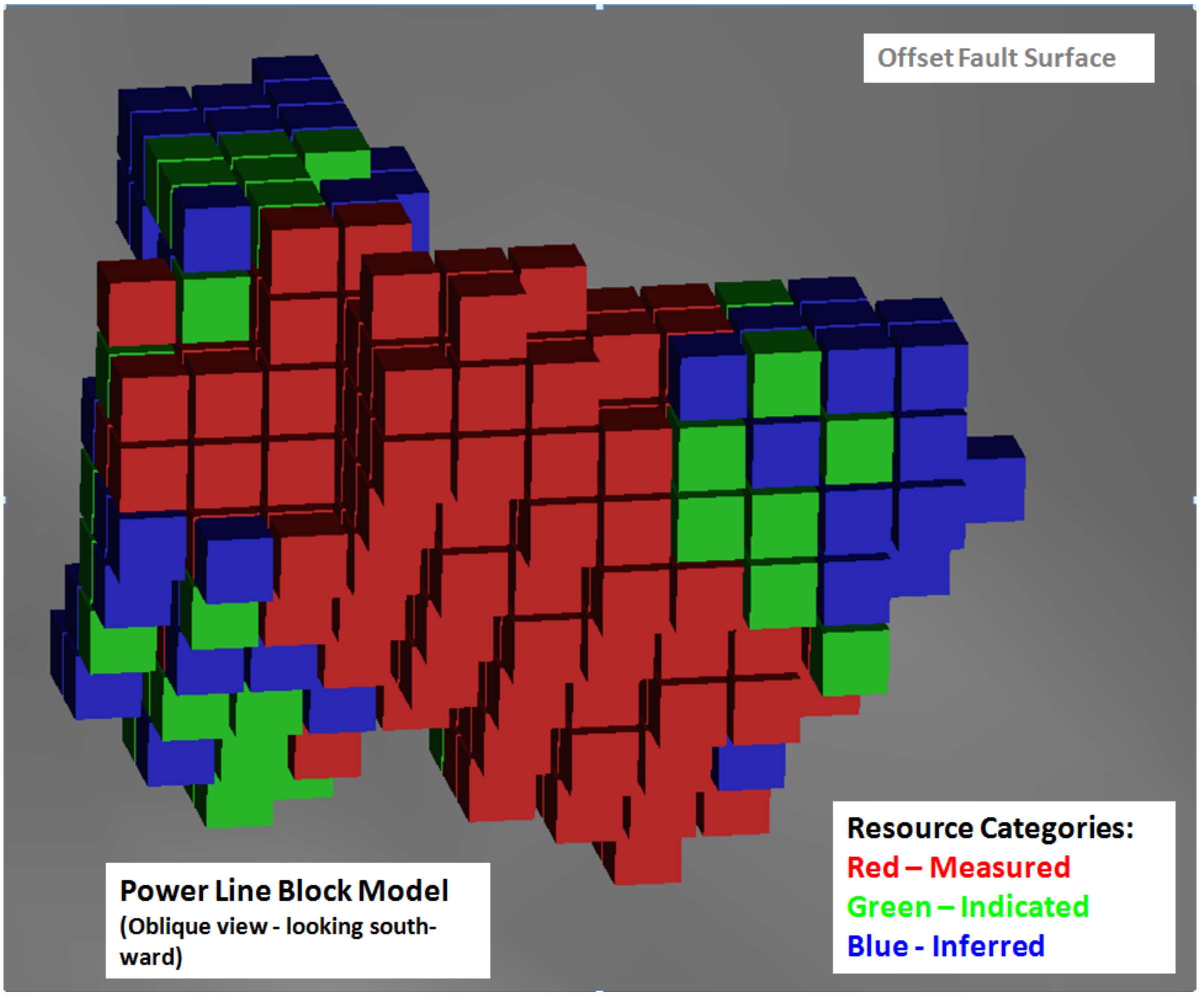


Figure 14-40: Powerline Zone Resource Category Distribution

14.5.10 Validation

The Powerline Zone block model was validated by two methods:

- Visual comparison of colour-coded block model grades with composite grades on section and plan.
- Swath plots of the block model.

14.5.10.1 Visual Validation

The visual comparisons of block model grades with composite grades show a reasonable correlation between the values. No significant discrepancies were apparent from the sections reviewed. Figure 14-41 to Figure 14-44 show representative cross-sections and plan views of the block model with same colour code for composite points from the captured bore hole traces and the block model.

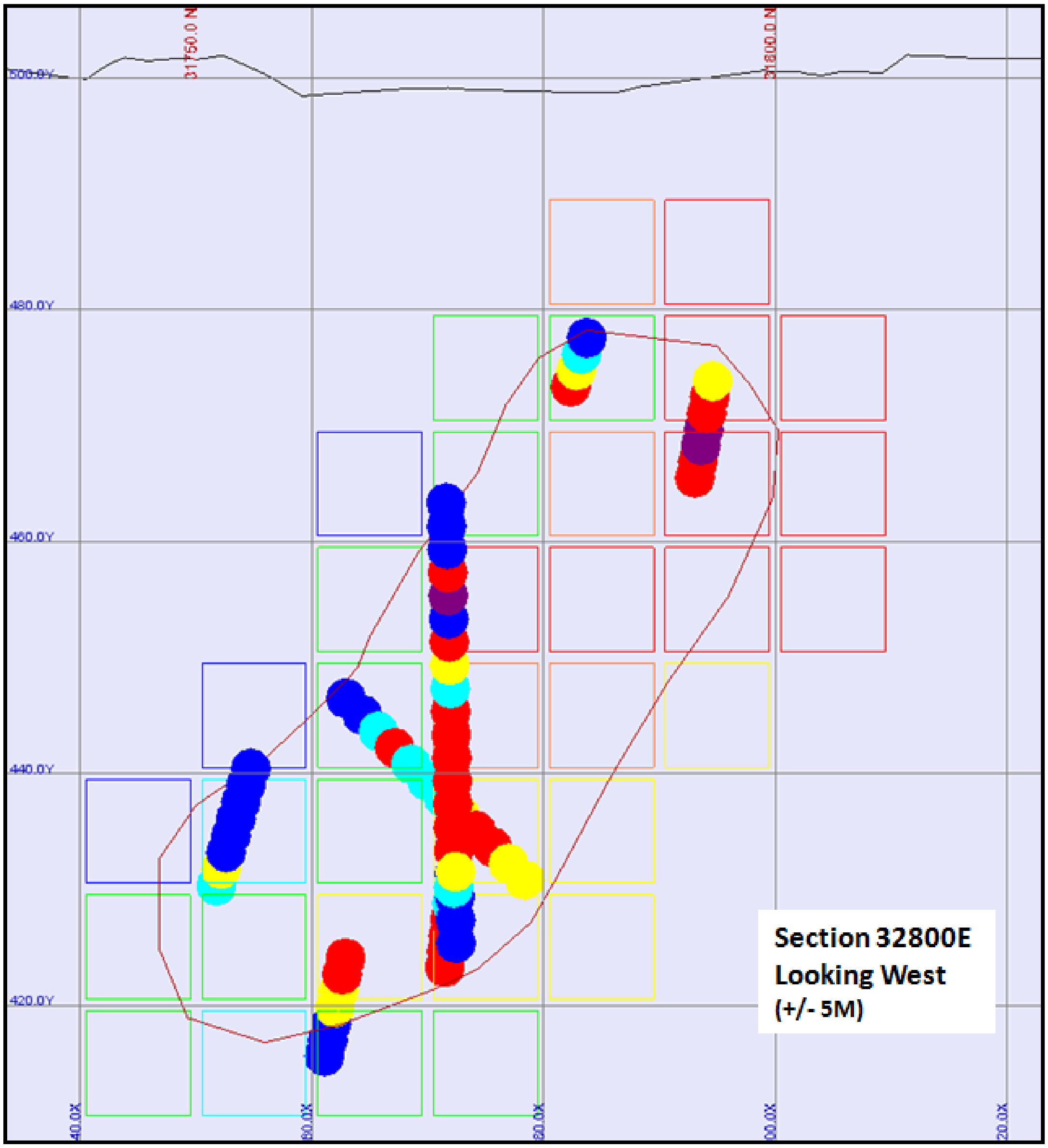


Figure 14-41: Composite Data and Block Model, Section 32800E, Looking West

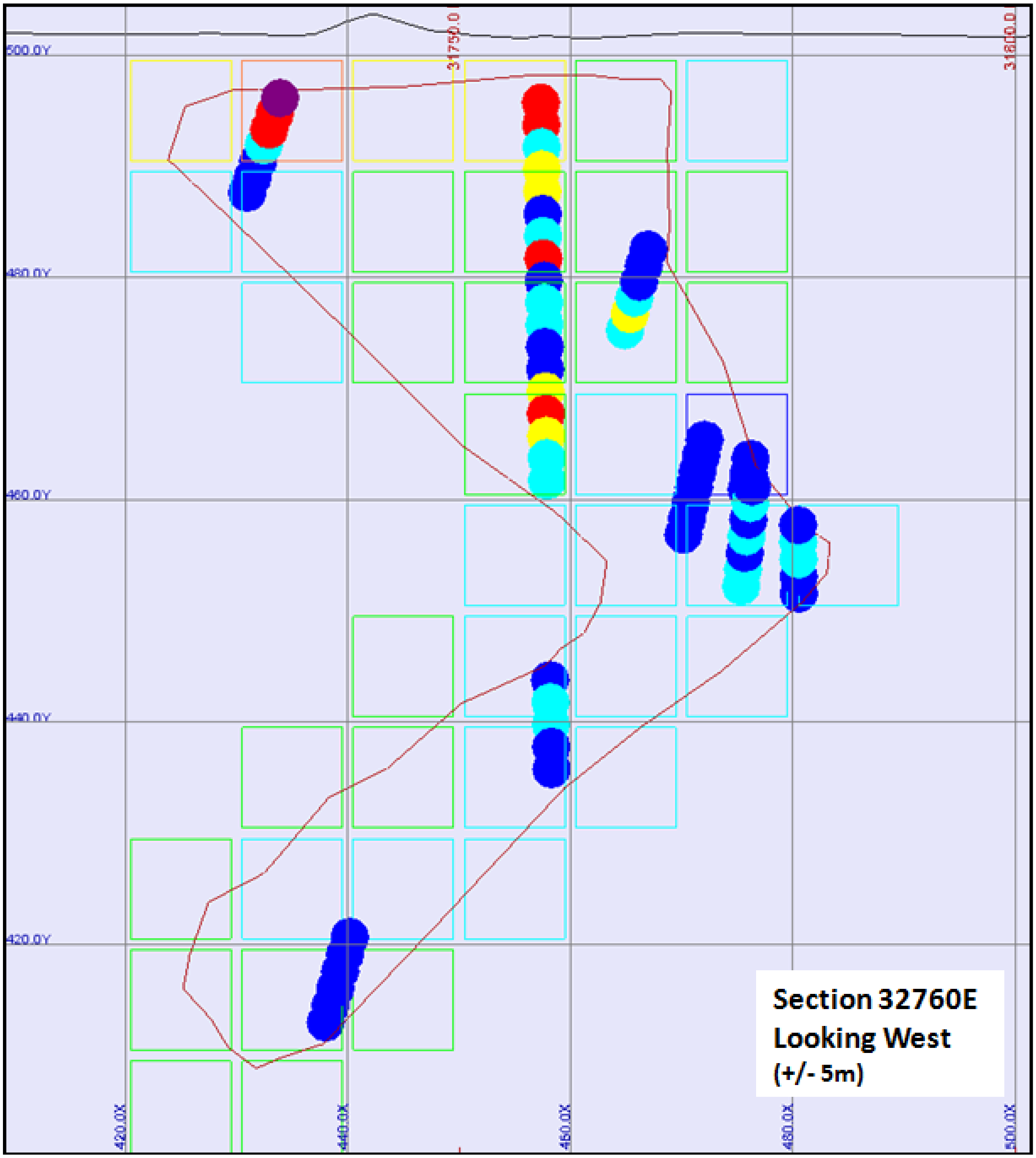


Figure 14-42: Composite Data and Block Model, Section 32760E, Looking West

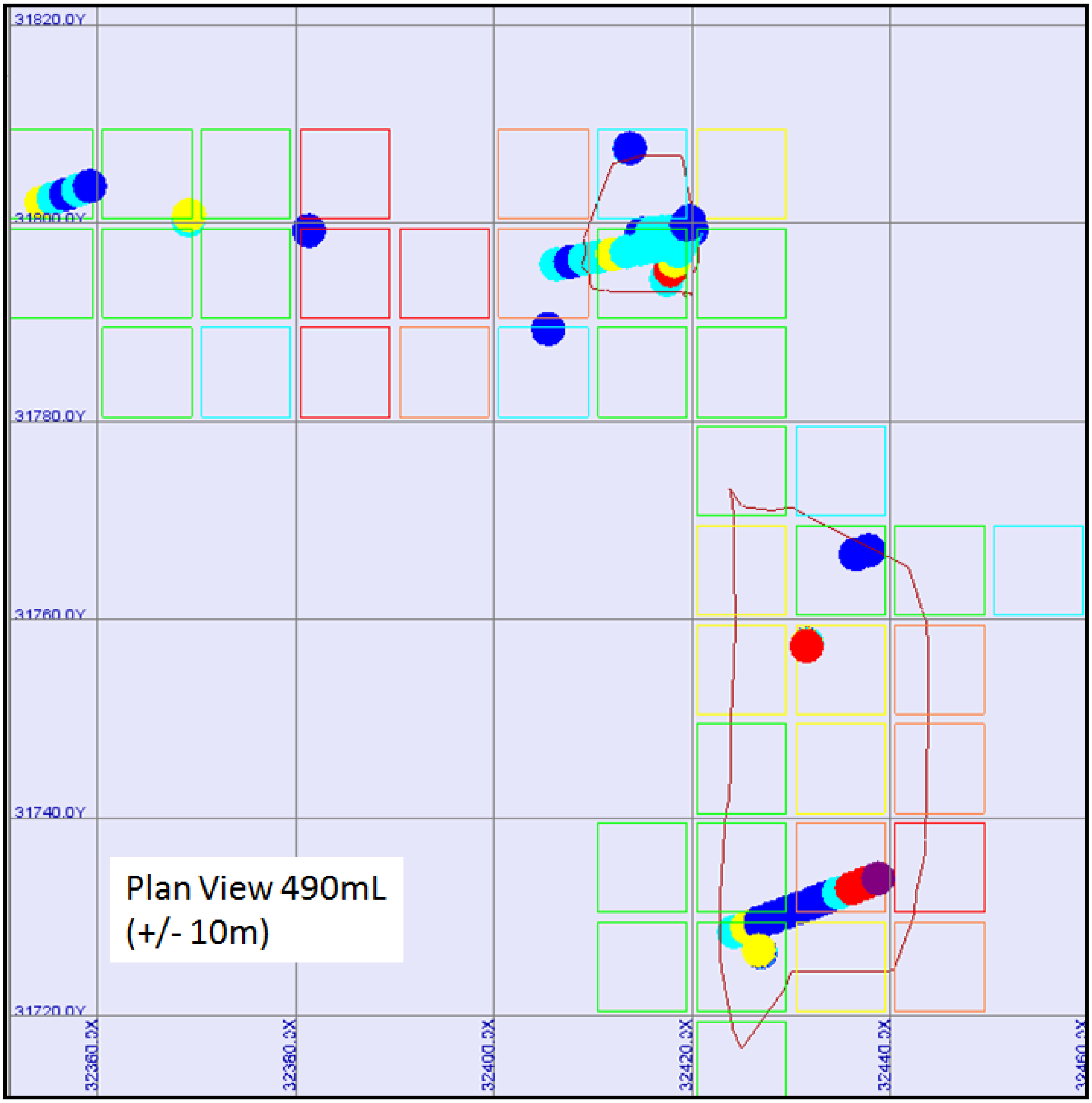


Figure 14-43: Composite Data and Block Model, Plan 490mL

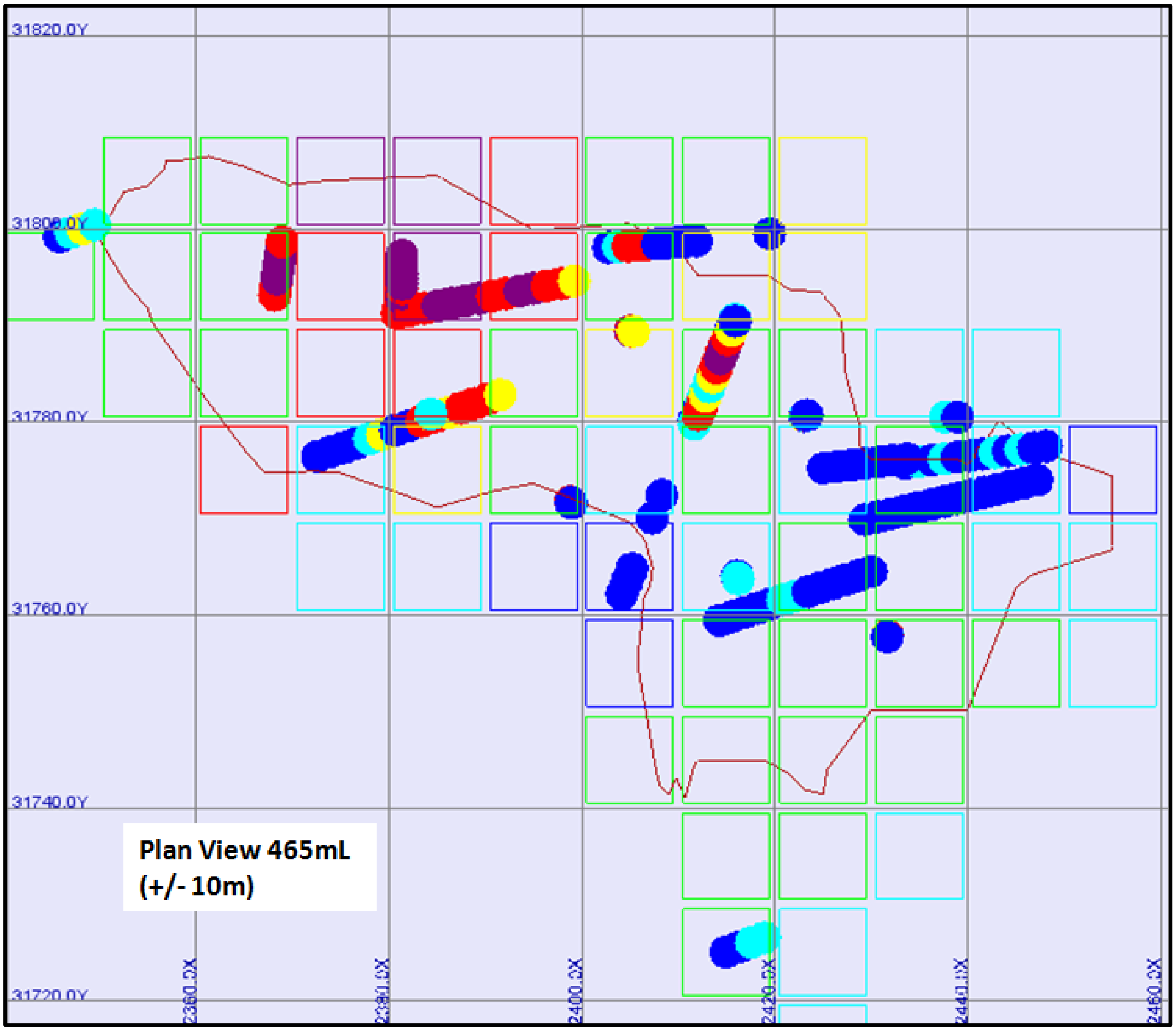


Figure 14-44: Composite Data and Block Model, Plan 465mL

14.5.10.2 Swath Plots

Swath plots of levels, columns and rows have been generated for palladium, platinum, gold, copper, nickel and cobalt. These plots compare the OK estimates (ordinary kriging) with the NN (nearest neighbour) and ID estimates (inverse distance squared). No major issues seen. Level plots can be seen in Figure 14-45 to Figure 14-50.

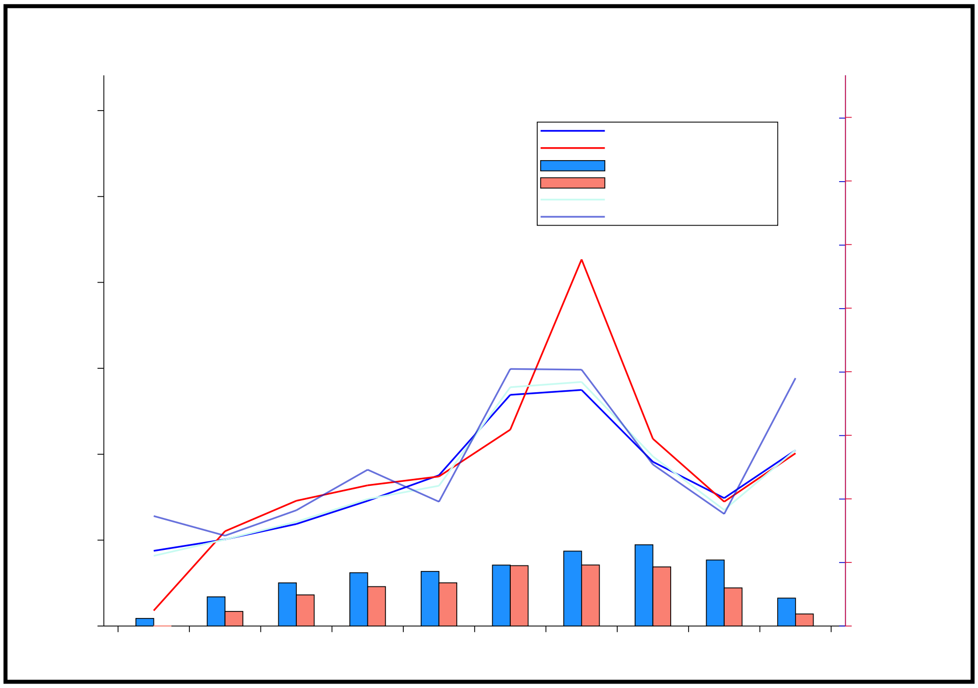


Figure 14-45: Swath Plot of Levels for Palladium

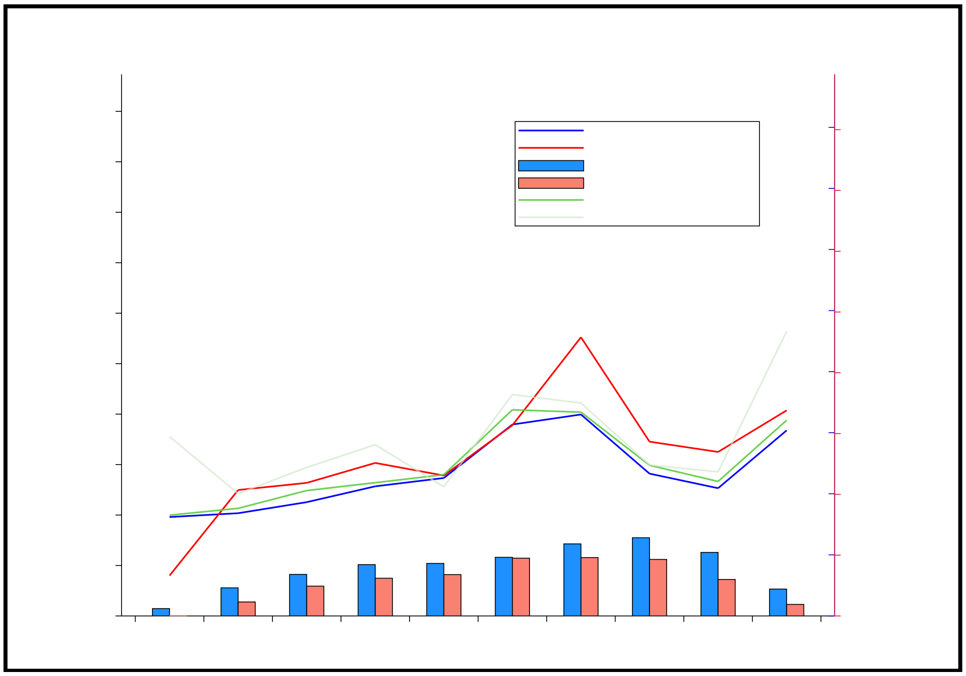


Figure 14-46: Swath Plot of Levels for Platinum

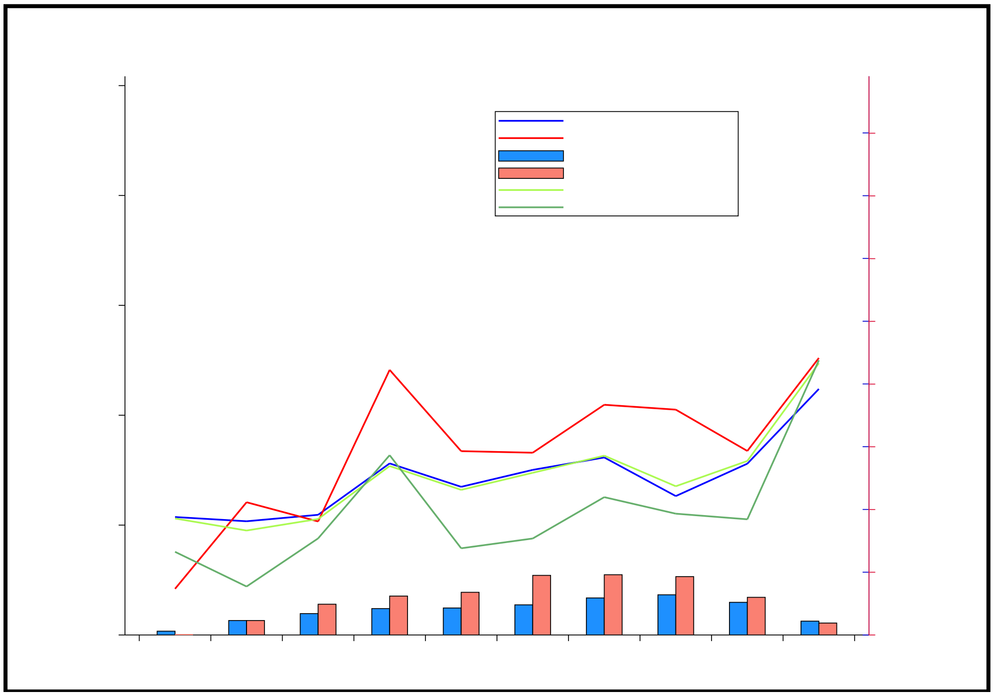


Figure 14-47: Swath Plot of Levels for Gold

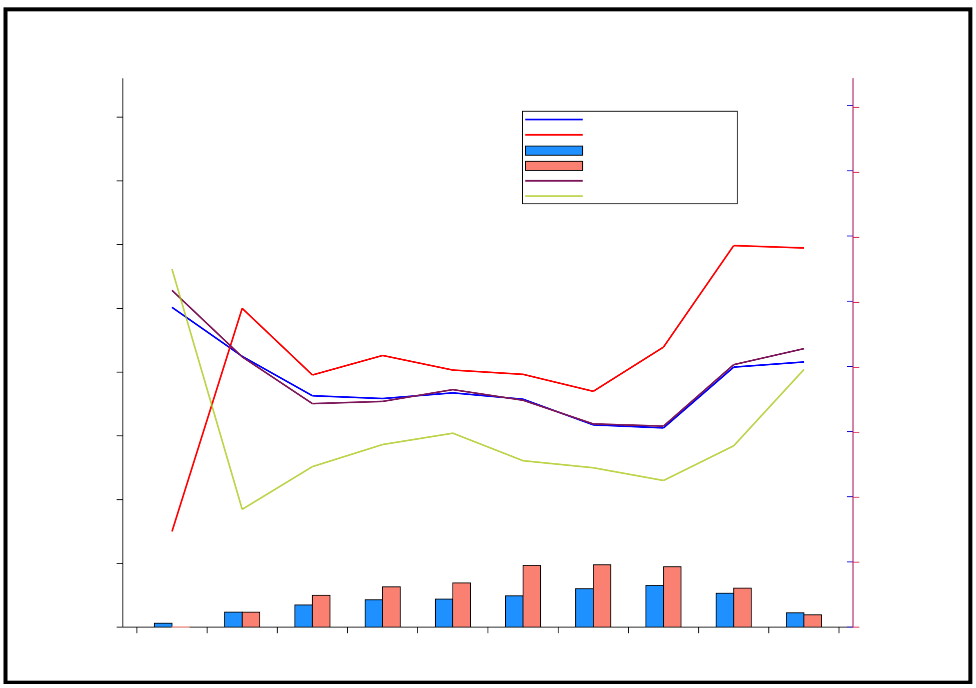


Figure 14-48: Swath Plot of Levels for Copper

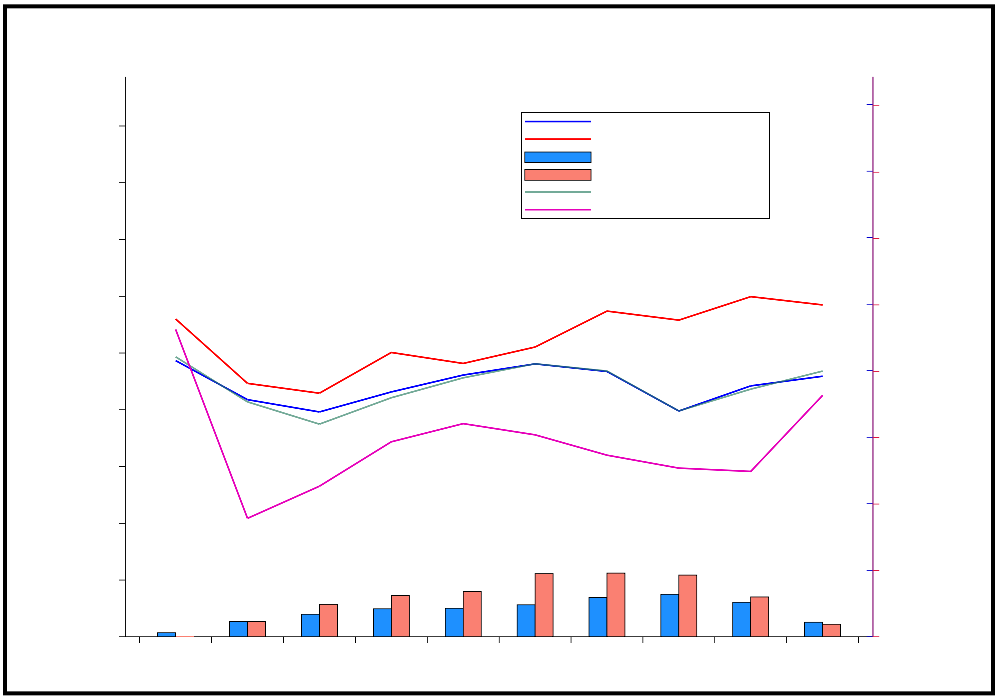


Figure 14-49: Swath Plot of Levels for Nickel

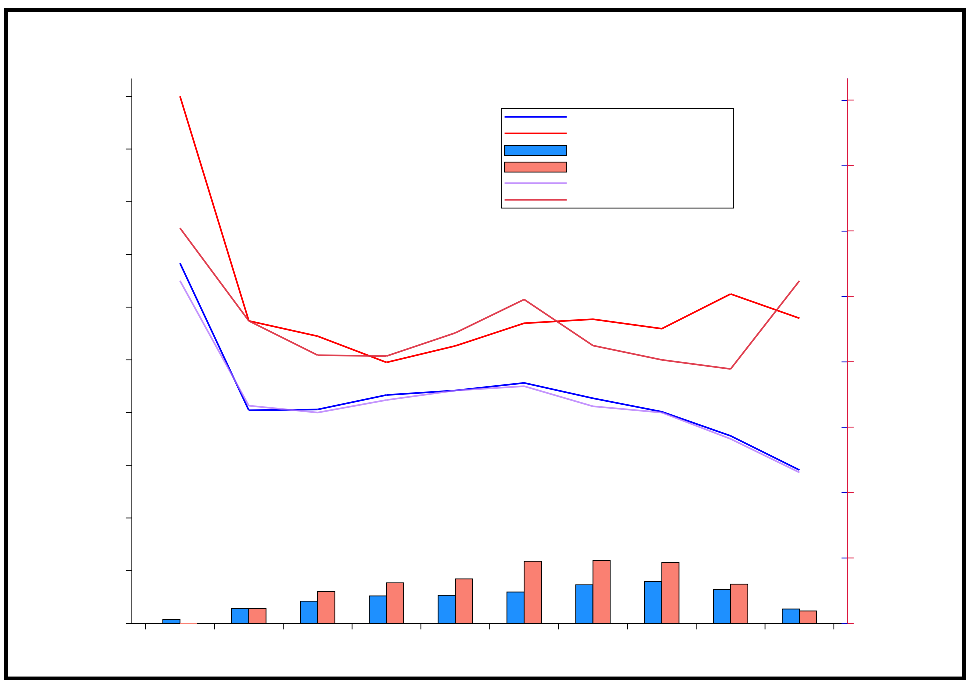


Figure 14-50: Swath Plot of Levels for Cobalt

14.5.11 Previous Estimates

This is the first mineral resource estimate for the Powerline Zone. But it is in the area of the previous Sheriff North Zone resource. Due to a re-interpretation of geology and surface channel sampling a higher grade zone was tested with bore holes in 2013 and 2014. From this information the Powerline Zone was created. Figure 14-51 shows the relationship between Sheriff North and Powerline Zones.

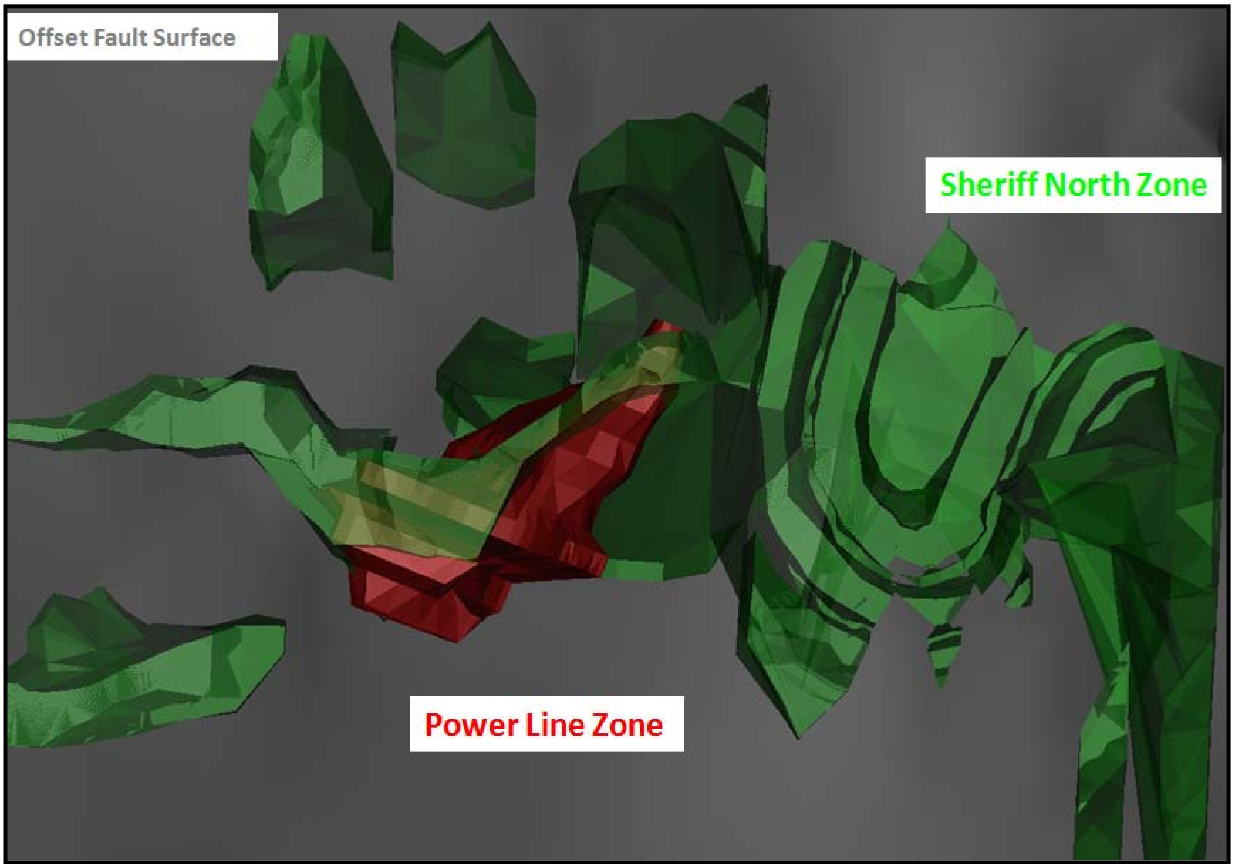


Figure 14-51: Oblique Plan View of Sheriff North Zone and Powerline Zone

14.6 Sheriff Zone

The Sheriff Zone Mineral Resource presented in McKinnon et al. (2013) has been modified because of the necessity of deducting the overlap with the new Powerline Mineral Resource (Section 14.5, above). The Sheriff Mineral Resource was used from the report of McKinnon et al. (2014) at a 1 g/t palladium cut-off grade (see Table 14-29). Then the area which the new Powerline Zone overlaps with the Sheriff North Zone was created into a shell and the tonnage and grade were extracted from the Powerline Zone Block Model (see Table 14-30). Subsequently, this number was subtracted from the original Sheriff Resource to come up with revised Sheriff Zone resource estimate that is effective as of Feb. 2, 2015 (see Table 14-31).

Table 14-29: Summary of Sheriff Zone Mineral Resource as of Feb. 2, 2015 with Powerline Zone

Classification	Zone	Cut-off	Tonnes	Pd	Pt	Au	Cu	Ni
				(g/t)	(g/t)	(g/t)	(%)	(%)
Measured	Sheriff North & South	1.0	5,060,000	1.49	0.16	0.09	0.06	0.06
Indicated	Sheriff North & South	1.0	219,600	1.29	1.16	0.09	0.06	0.06
Total Measured + Indicated			**5,280,000**	**1.48**	**0.20**	**0.09**	**0.06**	**0.06**
Inferred	Sheriff North & South	1.0	479,000	1.50	0.21	0.10	0.07	0.07
Total Inferred			**479,000**	**1.50**	**0.21**	**0.10**	**0.07**	**0.07**

Table 14-30: Summary of Powerline Mineral Resource which intersects Sheriff North Resource

Classification	Zone	Cut-off	Tonnes	Pd	Pt	Au	Cu	Ni
				(g/t)	(g/t)	(g/t)	(%)	(%)
Measured	Powerline	1.0	202,100	2.40	0.20	0.10	0.05	0.04
Indicated	Powerline	1.0	0	-	-	-	-	-
Total Measured + Indicated			**202,100**	**2.40**	**0.20**	**0.10**	**0.05**	**0.04**
Inferred	Powerline	1.0	0	-	-	-	-	-
Total Inferred			**0**	**-**	**-**	**-**	**-**	**-**

Table 14-31: Summary of Sheriff Zone Mineral Resource as of Dec. 31, 2014, with Powerline Zone Removed

Classification	Zone	Cut-off	Tonnes	Pd	Pt	Au	Cu	Ni
				(g/t)	(g/t)	(g/t)	(%)	(%)
Measured	Sheriff North & South	1.0	4,858,000	1.45	0.15	0.09	0.06	0.06
Indicated	Sheriff North & South	1.0	219,600	1.29	0.16	0.09	0.06	0.06
Total Measured + Indicated			**5,078,000**	**1.44**	**0.15**	**0.09**	**0.06**	**0.06**
Inferred	Sheriff North & South	1.0	479,000	1.50	0.21	0.10	0.07	0.07
Total Inferred			**479,000**	**1.50**	**0.21**	**0.10**	**0.07**	**0.07**

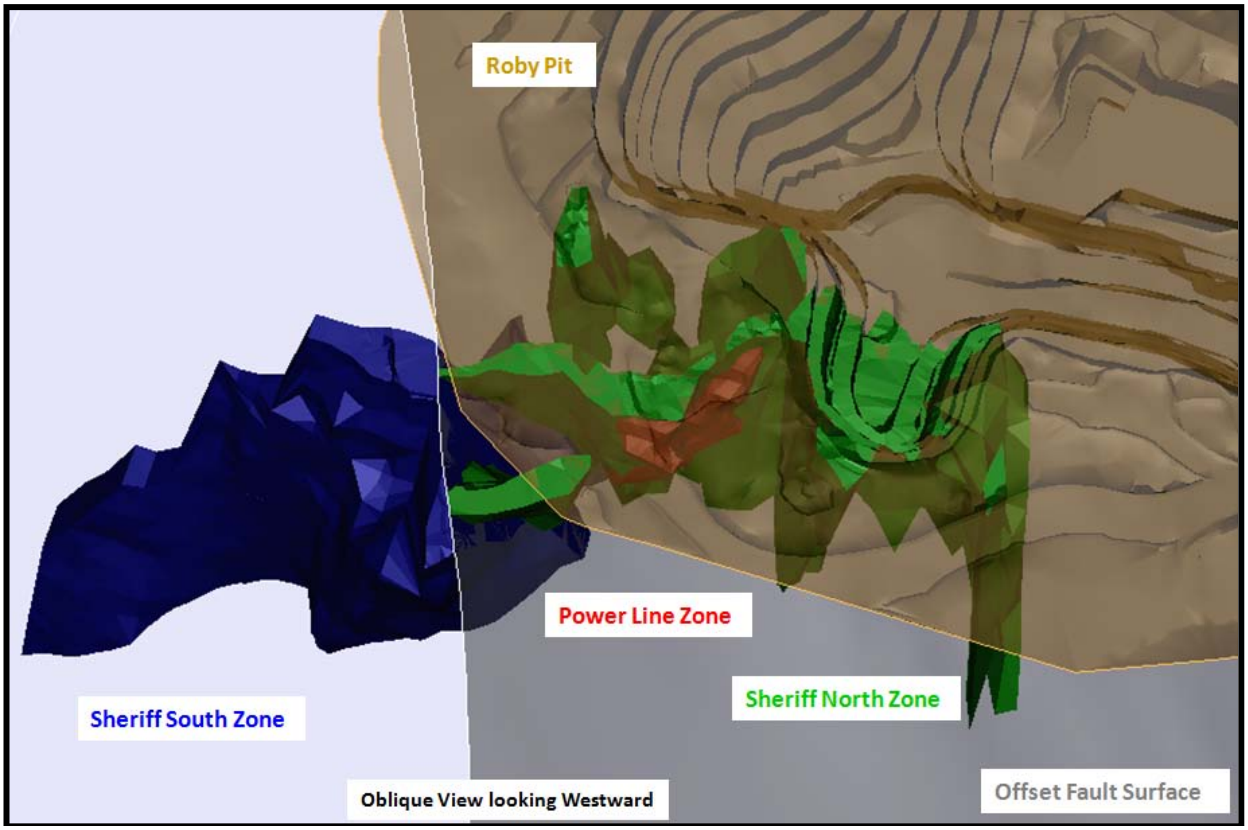


Figure 14-52: Spatial Relationship between Sheriff Zone and Powerline Zone

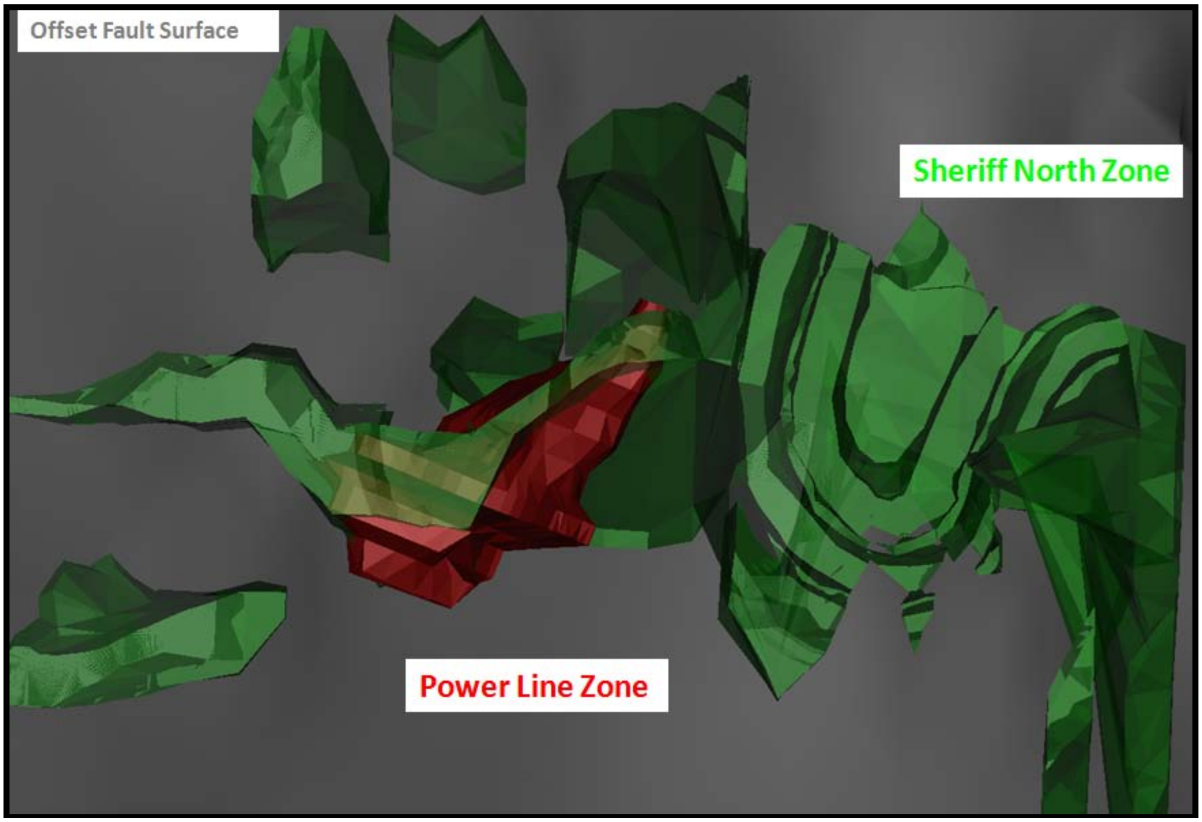


Figure 14-53: Spatial Relationship between Sheriff North Zone and Powerline Zone

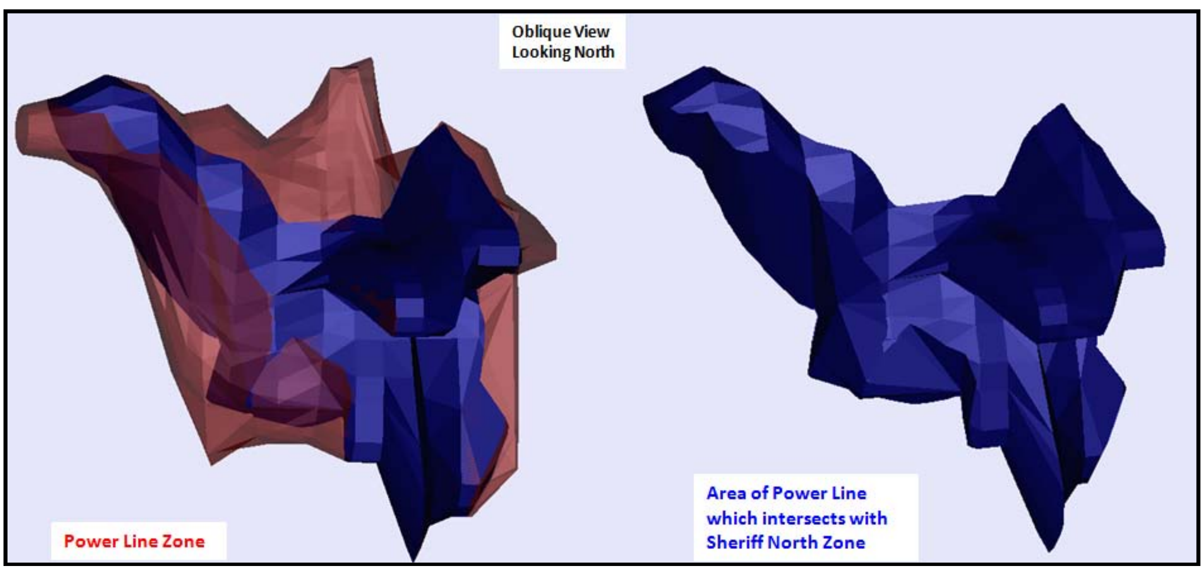


Figure 14-54: Intersection of Sheriff North Zone with Powerline Zone

14.7 North VT RIM Zone

No changes are reported from the resource estimate discusses in McKinnon et al. (2014).

Table 14-32: North VT Rim Zone Mineral Resource Summary

Category	Zone	Cut-off Pd (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Measured	West	1.0	338,900	2.024	0.118	0.034	0.032	0.005
	East_Deep	1.0	97,860	2.025	0.14	0.026	0.031	0.005
	Total	1.0	436,800	2.025	0.123	0.032	0.032	0.005
Indicated	West	1.0	6,034	1.828	0.122	0.032	0.030	0.005
	East_Deep	1.0	7,912	1.694	0.122	0.023	0.031	0.005
	Total	1.0	13,950	1.752	0.122	0.027	0.031	0.005
Measured + Indicated	Total Resource	1.0	450,700	2.016	0.123	0.032	0.032	0.005

14.8 Summary of LDIM Resources

A current summary of mineral resources for the LDIM property are provided in Table 14-33 in which the resources are exclusive of all mineral reserves. The resources are separated into surface resources that are considered to be potentially mineable using surface mining methods and underground resources that are considered to be potentially mineable using underground mining methods. Table 14-34 splits the Offset Zone resources described in Section 14.1 into resources occurring above the 1065 m level in the Offset mine and potentially relating to the current Phase 1 mine plan and resources occurring below this level and potentially relating to the Phase 2 mine expansion discussed in later sections of this report. A cut-off grade of 2.25 g/t Pd was used for the Phase 2 resources based on price assumptions and potential mining approaches discussed in later sections of this report.

Table 14-33: Mineral Resource Estimates for the LDIM Property Effective February 2, 2015
(Mineral Resources are exclusive of Mineral Reserves)

NEAR-SURFACE RESOURCES								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Measured	**(g/t)**	**(000's)**	**(g/t)**	**(g/t)**	**(g/t)**	**(%)**	**(%)**	**(000's oz)**
Roby Zone Pit Expansion	0.60	20,108	1.23	0.17	0.10	0.07	0.05	795
Powerline Zone	1.00	251	3.04	0.22	0.12	0.06	0.04	25
Sheriff Zone	1.00	4,858	1.45	0.15	0.09	0.06	0.06	227
North VT Rim	1.00	437	2.03	0.12	0.03	0.03	0.01	29
Total Measured	-	25,654	1.30	0.17	0.10	0.06	0.05	1,076
Indicated								
Roby Zone Pit Expansion	0.60	9,634	1.20	0.17	0.10	0.07	0.05	372
Powerline Zone	1.00	65	2.48	0.20	0.08	0.05	0.04	5
Sheriff Zone	1.00	220	1.29	0.16	0.09	0.06	0.06	9
North VT Rim	1.00	14	1.75	0.12	0.03	0.03	0.01	1
Total Indicated	-	9,932	1.21	0.17	0.10	0.07	0.05	387
Total Measured + Indicated	**-**	**35,586**	**1.28**	**0.17**	**0.10**	**0.07**	**0.05**	**1,463**
Inferred								
Powerline Zone	1.00	51	3.10	0.28	0.09	0.05	0.05	
Sheriff Zone	1.00	479	1.50	0.21	0.10	0.07	0.07	
Total Inferred	1.00	**530**	1.66	0.21	0.10	0.07	0.07	

UNDERGROUND RESOURCES - HANGINGWALL ZONES								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Measured	**(g/t)**	**(000's)**	**(g/t)**	**(g/t)**	**(g/t)**	**(%)**	**(%)**	**(000's oz)**
Offset Hangingwall Zone	2.50	2,683	4.36	0.31	0.29	0.11	0.08	376
Indicated								
Offset Hangingwall Zone	2.50	4,952	4.36	0.31	0.30	0.12	0.09	694
Total Measured + Indicated	**2.50**	**7,635**	**4.36**	**0.31**	**0.30**	**0.12**	**0.09**	**1,070**
Inferred								
Offset Hangingwall Zone	2.50	4,581	3.90	0.27	0.25	0.10	0.09	
Upper Offset Southeast Extension	TBD	827	3.20	0.28	0.24	0.09	0.08	
Total Inferred	-	5,407	3.80	0.27	0.25	0.10	0.08	
UNDERGROUND RESOURCES - FOOTWALL ZONES								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Measured	**(g/t)**	**(000's)**	**(g/t)**	**(g/t)**	**(g/t)**	**(%)**	**(%)**	**(000's oz)**
Offset Footwall Zone	1.50	10,584	2.22	0.23	0.16	0.08	0.06	755
Roby Footwall Zone	1.50	4,159	2.43	0.21	0.18	0.06	0.05	325
Total Measured	1.50	14,743	2.28	0.22	0.17	0.08	0.06	1,080
Indicated								
Offset Footwall Zone	1.50	11,163	2.10	0.21	0.15	0.08	0.07	754
Roby Footwall Zone	1.50	2,341	2.34	0.20	0.17	0.06	0.05	176
Total Indicated	1.50	13,504	2.14	0.21	0.16	0.08	0.06	930
Total Measured + Indicated	**1.50**	**28,247**	**2.21**	**0.21**	**0.16**	**0.08**	**0.06**	**2,010**

Category/Source	Pd Cut-Off	Tonnes	Pd	Pt	Au	Ni	Cu	Pd Contained
Inferred								
Offset Footwall Zone	1.50	8,853	2.05	0.16	0.12	0.07	0.06	
Roby Footwall Zone	1.50	248	2.43	0.18	0.08	0.03	0.02	
Total Inferred	1.50	9,101	2.06	0.17	0.12	0.07	0.06	
COMBINED RESOURCES - ALL SOURCES								
Category/Source		**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Sub-Category		**(000's)**	**(g/t)**	**(g/t)**	**(g/t)**	**(%)**	**(%)**	**(000's oz)**
Total Measured + Indicated Resources		**71,468**	**1.98**	**0.20**	**0.14**	**0.07**	**0.06**	**4,543**
Total Inferred Resources		15,038	2.70	0.20	0.17	0.08	0.07	

Table 14-34: Summary of Offset Zone Resources Separated into Phase 1 (Above 1065 m Level) and Phase 2 (Below 1065 m Level) Resources

Offset Zone Resources by Phase	Pd COG	Category	Tonnes (000's)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Phase 1 Offset Hangingwall Zone	2.50	Measured	2,247	4.28	0.30	0.27	0.11	0.08
(exclusive of reserves)	2.50	Indicated	1,556	3.99	0.31	0.24	0.09	0.07
	2.50	Inferred	946	4.08	0.30	0.27	0.10	0.10
Phase 1 Offset Footwall Zone	1.50	Measured	10,392	2.22	0.23	0.16	0.08	0.06
	1.50	Indicated	7,971	2.10	0.23	0.16	0.08	0.07
	1.50	Inferred	2,449	2.04	0.18	0.13	0.07	0.06
Phase 2 Offset Hangingwall Zone	2.25	Measured	457	4.63	0.33	0.37	0.14	0.11
	2.25	Indicated	3,633	4.39	0.31	0.32	0.13	0.10
	2.25	Inferred	3,898	3.76	0.26	0.24	0.10	0.08
Phase 2 Offset Footwall Zone	2.25	Measured	58	2.98	0.22	0.18	0.09	0.07
	2.25	Indicated	897	2.82	0.20	0.16	0.09	0.07
	2.25	Inferred	1,761	2.71	0.19	0.13	0.07	0.06
Phase 2 Total	2.25	Measured	515	4.44	0.32	0.35	0.14	0.11
(HWALL & PART OF FWALL)	2.25	Indicated	4,529	4.08	0.29	0.29	0.12	0.10
	2.25	Inferred	5,659	3.43	0.24	0.21	0.09	0.07

15. Mineral Reserve Estimates

15.1 Statement of Reserve

No Mineral Reserves are estimated in this Preliminary Economic Assessment.

To provide a reference to the analysis of the mine expansion opportunities discussed in this report, the Company's current reserve estimate is provided below (Table 15-1) together with a referenced summary of relevant, supporting technical information. The current reserve statement for the Property was disclosed by the Company in a news release dated February 25, 2015. The detailed technical basis for most of the reserves listed in Table 15-1 are given in McKinnon et al. (2014). The only changes to the reserves stated in McKinnon et al. (2014) are the depletion from mining activity that took place in 2014 and the conversion of all remaining RGO resources to reserves as disclosed in the Company's February 25, 2015 news release.

15.2 Near-Surface Reserves

Near-surface reserves at LDIM include a small pit salvage reserve located along the bottom and west wall of the Roby open pit and the remaining RGO stockpile resources estimated in McKinnon et al. (2014) after allowing for mining depletion in 2014.

The Roby open pit reserves were originally estimated by McKinnon et al. (2014). These reserves are based on a validated GEMS™ block model prepared by Denis Decharte, P.Eng., an employee of NAP and a Q.P. under NI 43-101. The validated block model captures resources estimated from a 2003 block model that was subsequently revised by Dave Penna, P.Geo., an employee of NAP and a Q.P. under NI 43-101 (McKinnon et al., 2014). The mineral resources upon which the Roby open pit reserves were estimated used a 1 g/t palladium resource block cut-off grade, and a block size of 15 m by 15 m by 8 m. They are based on a pit salvage mine design described in McKinnon et al. (2014).

Historically, NAP stockpiled materials from open pit mining activities that were assigned grades (from previous resource models) below the prevailing mining cut-off grade. This practice resulted in the development of three different stockpile inventories – each having a narrow range of grade as determined from relevant NI 43-101 compliant block models and validated, in part, by limited post-stockpile reconciliation sampling (blasthole rig sampling). As of December 31, 2013, North American Palladium Ltd. had processed all of its mid- to high-grade stockpiles. In 2014, the remaining surface stockpile was restricted to RGO ("regular-grade ore"). The average grade of the RGO stockpile (0.97 g/t Pd; see Table 15-1) is assumed to be evenly distributed through the stockpile. The resource tonnage of the current RGO stockpile has been estimated by depleting the tonnage of material sent to the Lac des Iles Mine mill for processing in 2013 and 2014 from the resource that existed as of December 31, 2012. An initial reserve estimate for the RGO stockpile was made as part of the mine plan defined in the previous Technical Report for the Property (McKinnon et al., 2014). Positive economics arising from the mining of RGO in 2014 supported the Company's decision to convert the remaining RGO resources to reserves as shown in Table 15-1 and as originally disclosed in the Company's February 25, 2015 news release.

15.3 Underground Reserves

The current estimated underground reserves for the Offset and Roby Zones at the Lac des Iles Mine are based on the Measured and Indicated Resource material included in geological block models described in McKinnon et al. (2014), and originally provided by Denis Decharte P.Eng., an employee of NAP and a Q.P. under NI 43-101. The geological block models used to estimate the current underground reserves reflect the results of diamond drilling completed after March 31, 2012, and prior to December 31, 2013. No new reserves have been estimated from drilling activity occurring after the latter date. The calculations for the underground mineral resources used a 1 g/t palladium resource block cut-off grade, and a block size of 5 m by 5 m by 5 m (McKinnon et al., 2014). Underground reserves at LDIM are restricted to the Hangingwall zones of both the Offset Zone and Roby Zone deposits and reflect depletion from mining activity to the end of 2014 as discussed in the Company's February 25, 2015 news release.

The mine design presented in McKinnon et al. (2014) is still applicable to the current mineral reserve estimate for the underground Roby Zone and Offset Zone deposits.

Table 15-1: Mineral Reserve Estimate for the Lac des Iles Mine Property Effective December 31, 2014

NEAR-SURFACE RESERVES								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Proven	**(g/t)**	**(000's)**	**(g/t)**	**(g/t)**	**(g/t)**	**(%)**	**(%)**	**(000's oz)**
Low-Grade Stockpile (RGO)	0.95	11,199	0.97	0.12	0.08	0.06	0.03	349
Roby Zone Open Pit	1.09	716	1.31	0.16	0.13	0.07	0.06	30
Total Proven	-	11,915	0.99	0.12	0.08	0.06	0.03	**379**
Probable								
Roby Zone Open Pit	1.09	293	1.39	0.18	0.15	0.08	0.07	13
Total Reserve	**-**	**12,208**	**1.00**	**0.12**	**0.08**	**0.06**	**0.03**	**392**
UNDERGROUND RESERVES - HANGINGWALL ZONES								
Category/Source	**Pd Cut-Off**	**Tonnes**	**Pd**	**Pt**	**Au**	**Ni**	**Cu**	**Pd Contained**
Proven	**(g/t)**	**(000's)**	**(g/t)**	**(g/t)**	**(g/t)**	**(%)**	**(%)**	**(000's oz)**
Offset Hangingwall Zone	2.60	3,660	3.95	0.28	0.28	0.11	0.08	465
Roby Hangingwall Zone	2.40	684	3.38	0.23	0.20	0.04	0.05	74
Total Proven	-	4,344	3.86	0.27	0.27	0.10	0.08	539
Probable								
Offset Hangingwall Zone	2.60	3,564	3.82	0.27	0.27	0.10	0.09	438
Roby Hangingwall Zone	2.40	251	3.17	0.22	0.18	0.05	0.04	26
Total Probable	-	3,815	3.78	0.27	0.26	0.10	0.09	464
Total Reserve	**-**	**8,159**	**3.82**	**0.27**	**0.27**	**0.10**	**0.08**	**1,003**

COMBINED RESERVES - ALL SOURCES		Tonnes	Pd	Pt	Au	Ni	Cu	Pd Contained
Category		(000's)	(g/t)	(g/t)	(g/t)	(%)	(%)	(000's oz)
Total Proven Reserve		16,259	1.76	0.16	0.13	0.07	0.04	918
Total Probable Reserve		4,108	3.61	0.26	0.25	0.10	0.09	477
Total Reserve		20,367	2.13	0.18	0.16	0.08	0.05	1,395

Notes (summarized from the Company's February 25, 2015, news release)
1. All reserve estimates were prepared in accordance with National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101) and the Canadian Institute of Mining, Metallurgy and Petroleum classification system.
2. The estimation of mineral reserves may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
3. Palladium ounces are stated as contained ounces. Tonnages are rounded to nearest '000 tonnes. Pd, Pt and Au grades are rounded to nearest .01 g/t. Ni and Cu grades are rounded to nearest .01%. Rounded numbers were used to calculate contained Pd (oz) and average resource and reserve tonnages.
4. All current mineral reserves, excluding the RGO reserve, are based on the estimates, metal price assumptions, and exchange rates provided in McKinnon et al. (2014).
5. The RGO reserves include all RGO resources estimated as of December 31, 2014 by David N. Penna and reflect mining depletion to that date.
6. Underground reserves are estimated to the 1065 Mine Level (4,435 metre elevation), a maximum depth of 1,072.5 m.
7. The effective date of the Lac des Iles Mine resource models that were used in the estimation of the current reserves for the Offset and Roby zones is December 31, 2013.

16. Mining Methods

16.1 Introduction

The 2014 Technical Report confirmed mineral reserves down to the 1065L with additional mineral resources extending to 1600L. The mineral reserves calculated in the 2014 Technical Report totalled 15.0 Mt grading 2.77 g/t Pd and showed a mine life to 2019. These mineral reserves are the basis for the existing LDI mine production plan.

Of significant interest in the 2014 Study was the large mineral resource inventory for the LDI property, which included material at and near surface, an expanded Offset hangingwall (HW) zone and the mineral resource estimate of the Offset and Roby footwall (FW) zone. In addition, a major exploration program was conducted throughout 2014 which continued to demonstrate the potential for more resources.

During 2014, as a result of running the mill on a full time basis, the unit operating costs at LDI decreased. As well, the price for palladium and the US/CAN currency exchange rate both moved in a favourable direction with long-term analyst consensus that it would stay strong.

Given the above, the entire resource was reviewed with the goal of seeking opportunities to extend the mine life. This review included developing preliminary cut-off grades as a result of the updated economic conditions and applying these to the various mineral zones at LDI. Recovery and dilution factors were applied, resulting in potential tonnage and grade for the various zones. These tonnages formed the basis to investigate potential options for the extension of the existing mine production plan.

This section describes the above process on a zone by zone basis. It describes the Current Mine Plan which is based on the 2014 Technical Report, depleted of material mined through December 31, 2014 and augmented with the RGO stockpile and other mineral resources above 1065L. This section further describes two expansion scenarios; a plan for re-activation of the currently idle open pit and a plan for the Phase 2 resources below the 1065L. The existing 2015 plan is common to all production scenarios generated. All mine plans were developed using January 1, 2016 as the start date for production, as this is the valuation date for the economic analysis.

16.2 Preliminary Cut-off Grade

For the purposes of determining the preliminary cut-off grades and mineral resources included in the PEA plan ("mill feed"), and for scheduling purposes, the following assumptions shown in Table 16-1 and Table 16-2, as supplied by NAP, were used:

Table 16-1: Economic Parameters

Assumption	Value
Mill Throughput (nominal)	12,500 tpd
Operating Days	365
Pd Price ($USD/oz)	$830
Pt Price ($USD/oz)	$1,400
Au Price ($USD/oz)	$1,300
Cu Price ($USD/lb)	$3.20
Ni Price ($USD/lb)	$7.50
Smelter Treatment Charge ($USD/dmt)	$350
Concentrate Shipping ($CAD/wmt)	$95
Exchange Rate ($CAD per $USD)	1.12
Royalty on NSR	5%

Table 16-2: Recoveries and Refining Assumptions for Underground Sources

Commodity	Mill Recovery	Smelter Payable	Refining Charge ($USD/lb or oz)
Pd	82.0%	90.5%	$20.22
Pt	80.3%	90.0%	$20.22
Au	80.4%	89.0%	$10.89
Cu	88.7%	90.0%	$0.54
Ni	38.8%	90.0%	$0.79

Mill recoveries represent averages for all the ore zones.

Lower mill recoveries were assumed in estimating the open-pit mineral resources to be included in the LOM plan, primarily because of the lower mill head grade associated with the open pit material at LDI. The assumptions used are shown in Table 16-3.

Table 16-3: Recoveries and Refining Assumptions for Open Pit Sources

Commodity	Mill Recovery	Smelter Payable	Refining Charge ($USD/lb or oz)
Pd	78.0%	90.5%	$20.22
Pt	76.3%	90.0%	$20.22
Au	76.4%	89.0%	$10.89
Cu	84.4%	90.0%	$0.54
Ni	36.9%	90.0%	$0.79

The economic parameters and recovery assumptions shown in this section were only used to determine preliminary cut-off grades for each underground zone.

16.3 Underground Zones above 1065 Level

16.3.1 Roby FW Zone

The Roby FW Zone consists of lower grade material that was previously uneconomic during historical mining activities. This material was re-assessed and it was determined that some salvage mining could be possible by utilizing existing development and uphole drilling, similar to a sub-level cave or sub-level retreat operation. This material would be mucked to the production stream until the samples assays return values below a pre-determined cut-off grade, at which time production from that stope or area would cease. This material would be trucked up the ramp to surface, so as not to displace hoisted material from below this horizon. Figure 16-1 highlights the Roby FW Zone with nearby previously mined out areas. For clarity, the figure does not show any Offset mined out areas.

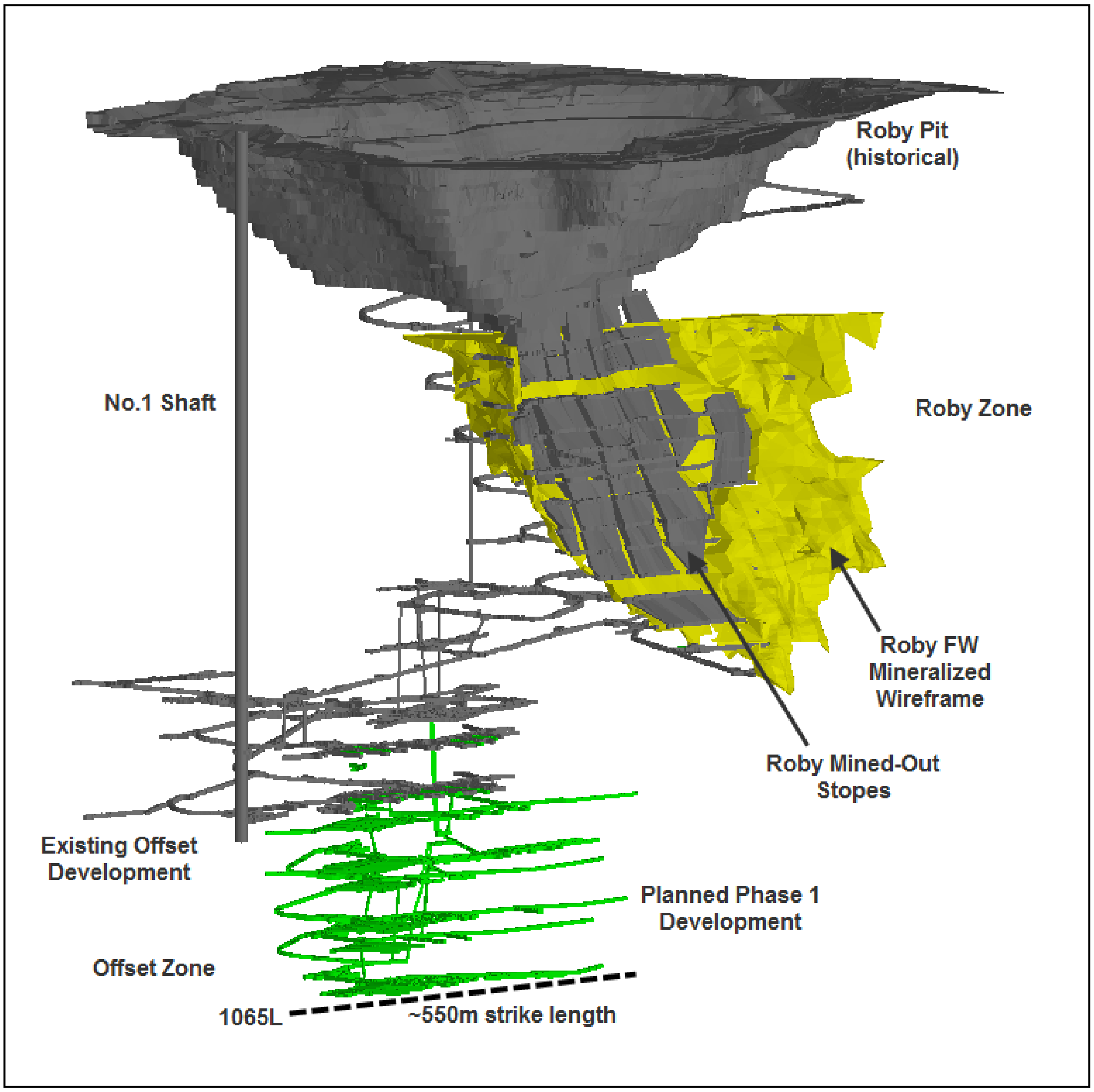


Figure 16-1: Isometric 3D Model View of Roby FW Mineralized Zone – Looking Northwest

Based on the limited number of trucks allowed in the ramp and the number of working faces, the production rate has been estimated at 1,500 tpd.

Based on the operating cost displayed in Table 16-4, it was determined that an operating break-even Pd cut-off grade of 2.00 g/t could be applied to this zone.

Table 16-4: Operating Cost for the Roby FW Zone

Category	Cost
G&A ($/t)	$5.22
Milling ($/t)	$9.87
Surface Handling ($/t)	$2.80
Mining ($/t)	$29.46
Total Operating Cost ($/t)	$47.35
Royalty determined in the financial analysis	

By applying a cut-off grade of 2.00 g/t Pd to the Roby FW Zone, the material available for inclusion in the PEA mine plans is listed in Table 16-5.

Table 16-5: Roby FW Mineral Resources at 2.00 g/t Pd Cut-Off Grade

Category	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Measured & Indicated	4.4	3.06	0.23	0.19	0.06	0.04

Factors were applied to account for mining related dilution and recovery losses (Table 16-6). The planned recovery factor was applied with consideration that detailed design work has not been completed on this zone. The dilution was estimated to be 20% based on sub-level caving retreat mining method.

Table 16-6: Roby FW Mining Factors

Mining Factor	Value
Planned/Resource Recovery	55%
Unplanned/External Dilution*	20%
Mining Recovery	98%

*Zero dilution grade

The mineral resources used in the PEA mine plans after application of the mining factors are displayed in Table 16-7.

Table 16-7: Roby FW Mineral Resources Included in the PEA

Category	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Measured & Indicated	2.8	2.48	0.17	0.07	0.03	0.01

The tonnage and grades shown Table 16-7 were input into the PEA mine schedule.

16.3.2 Offset HW Above 1065L Phase 1

The Offset Zone material above 1065L, referred to as Phase 1 in this report, is comprised of two components in this PEA:

- Production from the existing LDI production plan down to 1065L which comprises entirely mineral reserves that were identified in the previous Technical Report. This mine plan is input directly into the PEA mine plans. This material grades above a 2.50 g/t Pd cut-off grade, which is the current cut-off grade at LDI.

- Additional mineral resources located above 1065L and grading above 2.50 g/t Pd cut-off grade, in the HW zone, as seen in Figure 16-2. This material was identified from further drilling and is incorporated in an updated Mineral Resource Estimate, as described in Section 14. Mining factors were applied to this resource before inclusion into the PEA mine plan.

The mineral resources in the Offset FW Zone material located above 1065L are described in Section 16.5 and currently not considered as part of Phase 1 mining due to their lower grade and subsequent deferral to later in the mine life. They are included as a separate mining zone in the PEA.

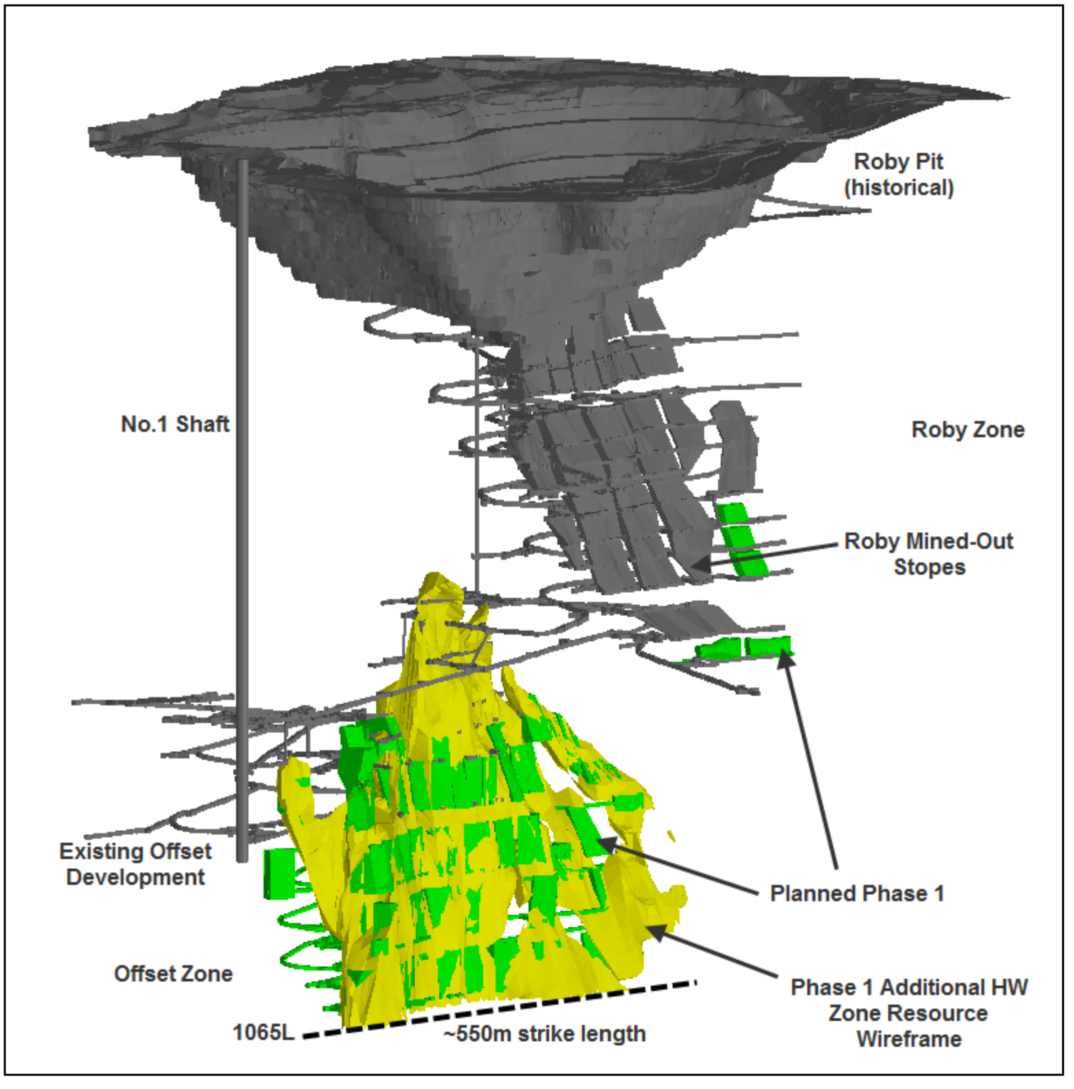


Figure 16-2: The Phase 1 Additional HW Zone Mineral Resource Isometric Wireframe

Table 16-8 displays the additional Offset Zone Phase 1 resources grading above 2.50 g/t cut-off grade, prior to application of mining factors.

Table 16-8: Additional Offset Zone Phase 1 Mineral Resources at 2.50 g/t Pd Cut-Off Grade

Category	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Measured & Indicated	3.8	4.16	0.30	0.26	0.10	0.08

Factors were applied to account for mining related dilution and recovery loss (Table 16-9). The planned recovery factor was applied with consideration that detailed design work had not been done on this zone. The dilution was estimated at 15%, which is a typical dilution factor for a blasthole mining method.

Table 16-9: Offset Phase 1 Mining Factors

Mining Factor	Value
Planned/Resource Recovery	65%
Unplanned/External Dilution*	15%
Mining Recovery	98%

*Zero dilution grade

The additional Phase 1 resources used in the PEA mine plans after application of mining factors are displayed in Table 16-10.

Table 16-10: Additional Offset Zone Phase 1 Mineral Resources Included in the PEA

Category	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Measured & Indicated	2.8	3.62	0.26	0.22	0.09	0.07

These mineral resources were then combined with the mineral resources from the current LDI mine plan for Phase 1, resulting in the total Phase 1 mineral resources input into the PEA mine plans (Table 16-11). The Current LDI Plan details are presented as of January 2016.

Table 16-11: Total Phase 1 Mill Feed Included in the PEA Plan

	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Current LDI Plan	4.9	3.42	0.29	0.27	0.08	0.10
Additional Resources	2.8	3.62	0.26	0.22	0.09	0.07
Total*	7.7	3.49	0.28	0.25	0.08	0.09

*Totals may not add up due to rounding

For the purposes of this PEA, the LDI mine plan schedule was left unchanged, and the additional mineral resources were scheduled at a rate of 4,000 tpd, decreasing towards the end of the zone life.

16.3.3 Offset FW Zone

Similar to the Roby FW Zone, it was determined that under updated economic assumptions there was some potential to mine lower grade material in the footwall of the Offset Zone above 1065 Level. The proposed mining method is blasthole mining using unconsolidated fill for backfill. It is anticipated that a production rate of up to 4,000 tpd would occur during the mining of this zone.

Figure 16-3 shows the Offset FW Zone wireframe. For clarity, the Offset HW Zone and the Roby FW Zone wireframes have been removed.

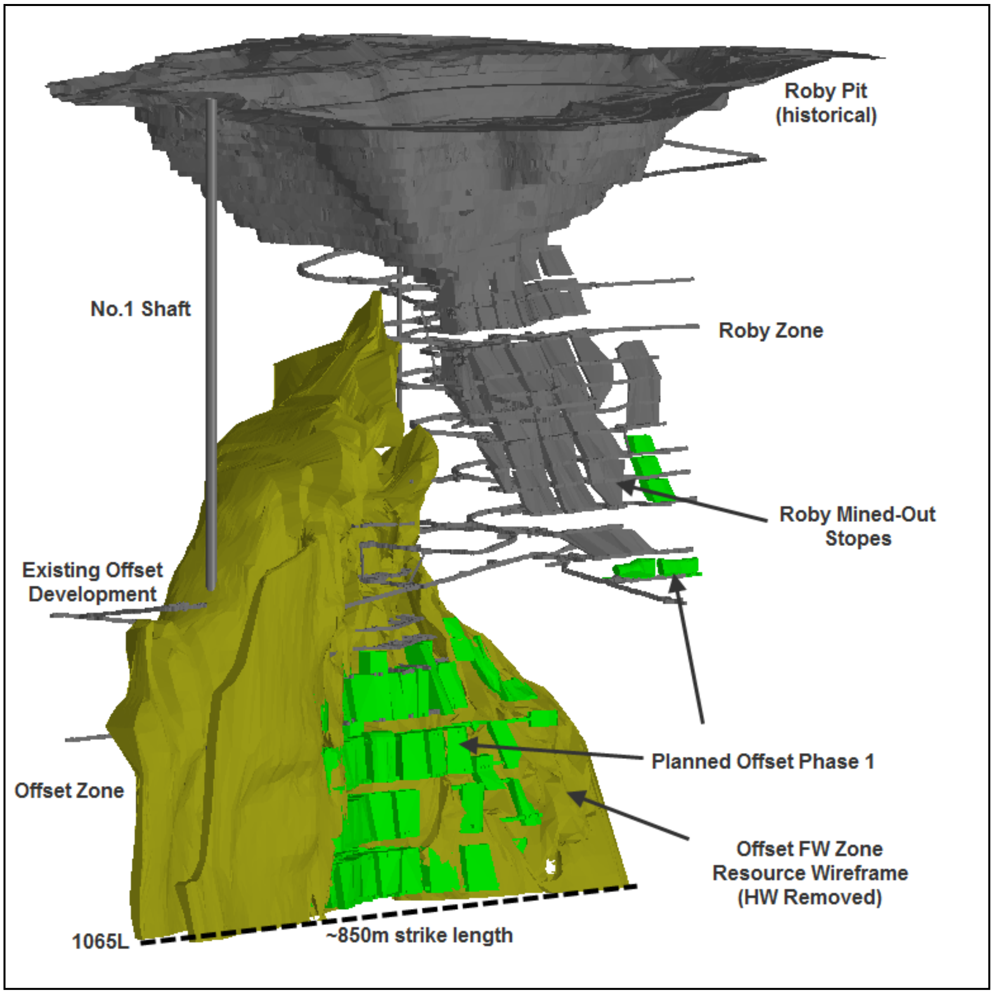


Figure 16-3: Offset FW Zone Mineral Resource Isometric Wireframe

Based on the operating cost displayed in Table 16-12, it was determined that an operating break-even Pd cut-off grade of 2.25 g/t could be applied to this zone.

Table 16-12: Operating Cost for the Offset FW Zone

Category	Cost
G&A ($/t)	$5.22
Milling ($/t)	$9.87
Surface Handling ($/t)	$2.73
Mining ($/t)	$37.61
Total Operating Cost ($/t)	$55.43
Royalty determined in the financial analysis	

By applying the cut-off grade of 2.25 g/t Pd to the Offset FW Zone, the mineral resources to be used in the PEA are listed in Table 16-13.

Table 16-13: Offset FW Mineral Resources at 2.25g/t Pd Cut-off Grade

Category	Tonnes (Mt)	Pd(g/t)	Pt(g/t)	Au (g/t)	Ni (%)	Cu (%)
Measured & Indicated	5.7	2.98	0.28	0.21	0.09	0.08

Factors were applied to account for mining related dilution and recovery losses (Table 16-14). The planned recovery factor was applied with consideration that detailed design work has not been completed on this zone. There exists some risk to recovering a high percentage of the material remaining in the Offset FW Zone. The dilution was determined to be 20%, based on an overhand blasthole method, where these stopes may be more irregular in shapes than in other blasthole zones.

Table 16-14: Offset FW Mining Factors

Mining Factor	Value
Planned/Resource Recovery	55%
Unplanned/External Dilution*	20%
Mining Recovery	98%

*Zero dilution grade

The mineral resources used in the PEA mine plan after application of mining factors are displayed in Table 16-15.

Table 16-15: Offset FW Mineral Resources Included in the PEA

Category	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Measured & Indicated	3.7	2.48	0.24	0.17	0.08	0.07

The tonnage and grades shown in Table 16-15 were subsequently input into the PEA mine plans and the economic analysis.

16.4 Surface Zones

16.4.1 Tailings and RGO Stockpiles

All plans developed in the PEA include inventory contained within the tailings stockpile, as well as the RGO stockpile. The tailings stockpile is included in the current LDI mining plan as a mill feed gain. The RGO stockpile is also included in the current LDI mining plan as supplemental mill feed to other sources.

Table 16-16 summarizes the inventory in the RGO and tailings stockpiles that were included in the Current Mine Plan.

Table 16-16: Tailings and RGO Stockpile Inventory

Stockpile	Tonnes (Mt)	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)
Tailings	1.4	1.37	0.09	0.11	0.02	0.05
RGO	10.5	0.97	0.12	0.08	0.06	0.03

16.4.2 Roby Pit, Sheriff Pit, NVT Pit Mineral Zones

16.4.2.1 Open Pit Input

Mintec's MineSight design software was used to generate the shells for the open pit, based on parameters listed in Table 16-17.

Optimization was based on physical and economic parameters derived from previous operations.

- The overall pit slope angle was obtained from Itasca Canada and is considered to be a preliminary recommendation. Further geotechnical work is necessary.
- Smelter Treatment and Concentrate Shipping charges were obtained from NAP as described in Table 16-1. In the opinion of the QP, those charges are reasonable.

Table 16-17: Open Pit Parameters

Parameter	Value
Overall Pit Slope Angles	55 Degrees
Mining cost	$3.00 / tonne moved
Processing cost	$9.87 / tonne processed
G&A cost	$5.22 / tonne processed
Surface Operations cost	$1.92 / tonne processed
Royalty determined in the financial analysis	

16.4.2.2 Open Pit Results

The resource is contained within four geological block models (NVT and Sheriff models as prepared by McKinnon et al. 2014, Roby surface resources model provided by Pincock, Allen and Holt, 2003, Roby u/g block model as prepared by McKinnon et al. (2014)). The shells generated were compared against these models, and consolidated to present a single pit shell inventory number. All resources contained in the pit shell are in the Measured and Indicated categories.

Table 16-18 outlines the results that were obtained from generating the pit shell and interrogating it against the block models.

Table 16-18: Open Pit Shell Results

Parameter	Value
Mineral Resource Tonnes	36.4Mt
Recovery	90%
Dilution	8%
Mill Feed Tonnes	34.7 Mt
Tonnes Capital Waste	25.6 Mt
Tonnes Operating Waste	18.9 Mt
Pd (g/t) (diluted)	1.14
Pt (g/t) (diluted)	0.16
Au (g/t) (diluted)	0.10
Ni (%) (diluted)	0.05
Cu (%) (diluted)	0.05
Strip ratio	1.3

Figure 16-4 shows the pit shell based on the Roby, Sherriff and North VT Rim mineral resources. Their proximity to the underground workings and shaft pillar is illustrated in Figure 16-5.

The pit shells were generated without regard to existing infrastructure, some of which will be affected by the proposed ultimate pit. These are described in Section 18. Mining of the Open Pit will ultimately have an impact on the existing shaft. The impact on underground and open pit production schedules will need to be further examined in future studies.

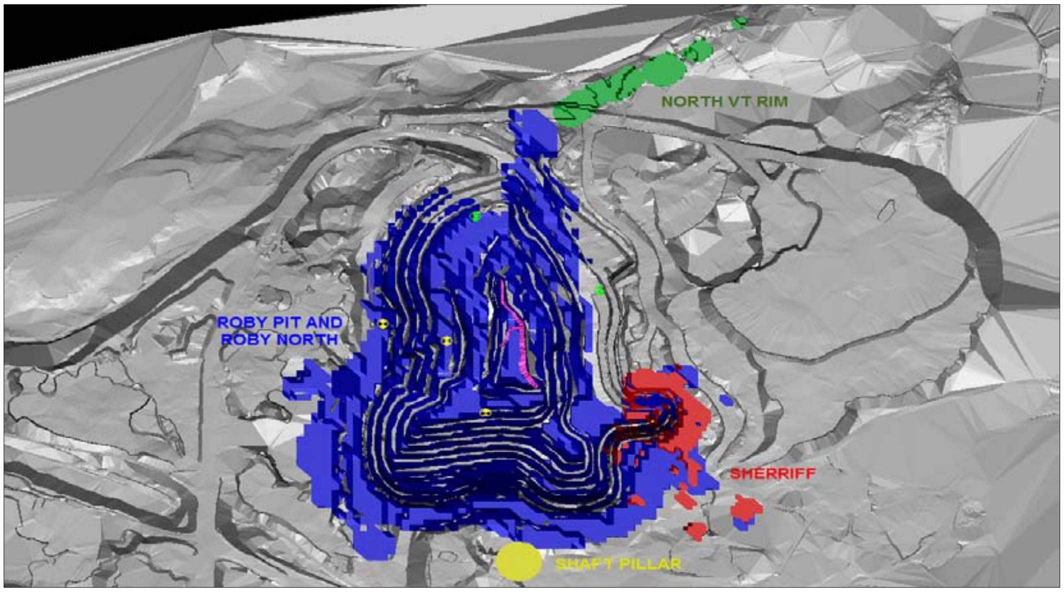


Figure 16-4: Plan View of Open Pit Shell (North is up)

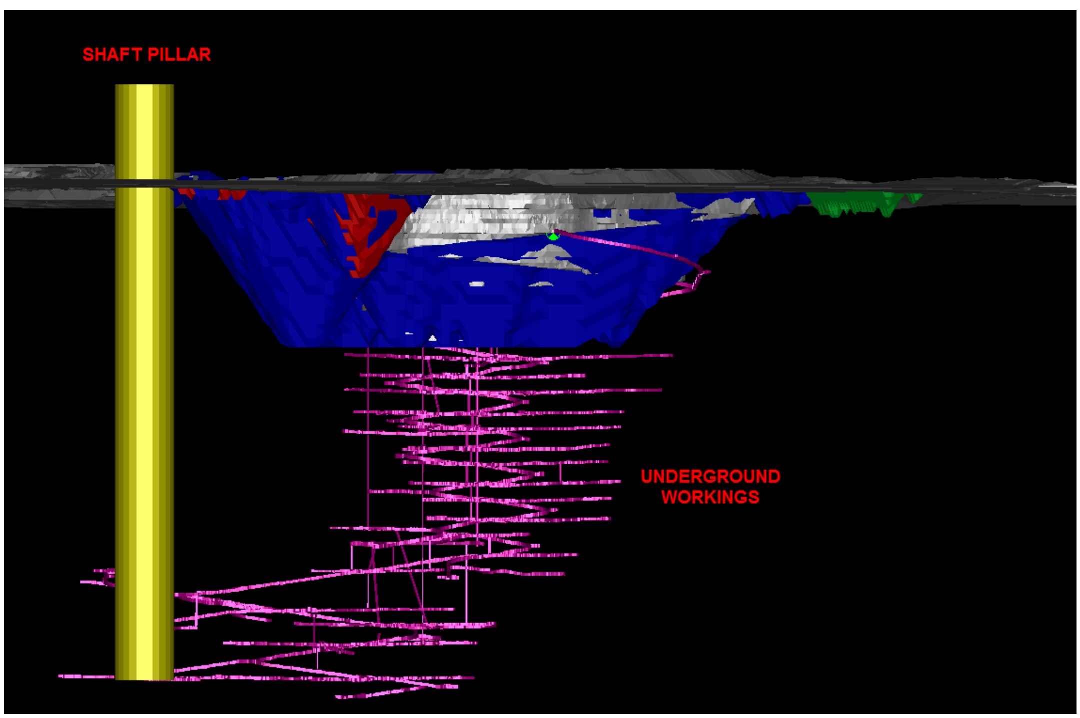


Figure 16-5: Section View of Potential Pits (Looking West)

16.4.2.3 Equipment

Table 16-19 is a list of proposed equipment for the Open Pit operation.

Table 16-19: Open Pit Equipment List

Item	Description	Quantity
Trucks	CAT 775	6
Excavator	CAT 390	2
Loader	CAT 980	1
Drills	12 inch hole capability	2
Dozer	CAT D10	1
Stemming / Blast Truck	Ground force	1

16.5 Offset Zone Phase 2

16.5.1 Overview

The mining method for the Phase 2 expansion scenario has been selected using a qualitative decision matrix analysis. The decision matrix considers a series of key criteria and a relative rank is assigned to each criteria. One of the methods is selected as a reference case and assigned a neutral weighting. The other methods are then scored as better-than (x3), worse-than (x-3) or same-as (x0) the reference case.

The four methods considered are described as follows:

- Overhand Blasthole – Primary/Secondary Sequence – Unconsolidated rock fill (URF) with Rib Pillars – This is considered the base case method as it is similar to the current mining method employed at LDIM.
- Overhand Blasthole – Primary/Secondary Sequence – Paste fill (PF) with no rib pillars – This method employs similar blasthole stoping as the base case, but eliminates the rib pillars and uses pastefill instead of URF.
- Sub-Level Caving – This concept involves a top-down, centre-out progression of drawpoints. Backfill is introduced at the top of the mining zone at the same rate of mucking from drawpoints below.
- Vertical Retreat Mining – “Delayed Pull” – This method progresses bottoms-up, mucking swell from drawpoints developed at the bottom of the zone. Once the entire zone has been blasted, full production is drawn from the bottom draw points, with unconsolidated rockfill being introduced at the top of the zone.

The selection considered the following key criteria:

- Production Rate.
- Capital Cost.
- Operating Cost.
- Reliability/Predictability.
- Resource Recovery.
- Dilution Control.

As a result of the decision matrix analysis shown in Table 16-20, the mining method selected for the Phase 2 expansion scenario was the Overhand Blasthole method with no pillars and the use of paste backfill. The principal factors that resulted in the selection of this method are reliability of production and improved ore recovery. This method is illustrated in Figure 16-6.

Table 16-20: Decision Matrix for Phase 2 Mining Method

Criteria	Rank		Overhand Blasthole URF		Overhand Blasthole Paste Fill		Sub-Level Retreat		Vertical Retreat Delayed Pull	
Health and Safety	High	3	Same	0	Same	0	Same	0	Same	0
Production Rate	High	3	Same	0	Same	0	Same	0	Same	0
CAPEX	Medium	2	Same	0	Worse	-6	Worse	-6	Same	0
OPEX	Medium	2	Same	0	Same	0	Same	0	Same	0
Impact on Schedule	High	3	Same	0	Same	0	Same	0	Same	0
Reliability/Predictability	High	3	Same	0	Better	9	Worse	-9	Worse	-9
Implementation	Low	1	Same	0	Worse	-3	Worse	-3	Same	0
Ore Recovery	Medium	2	Same	0	Better	6	Better	6	Better	6
Stakeholder Acceptance	Low	1	Same	0	Same	0	Worse	-3	Worse	-3
Dilution Control	Medium	2	Same	0	Better	6	Worse	-6	Worse	-6
Total			0		12		-21		-12	

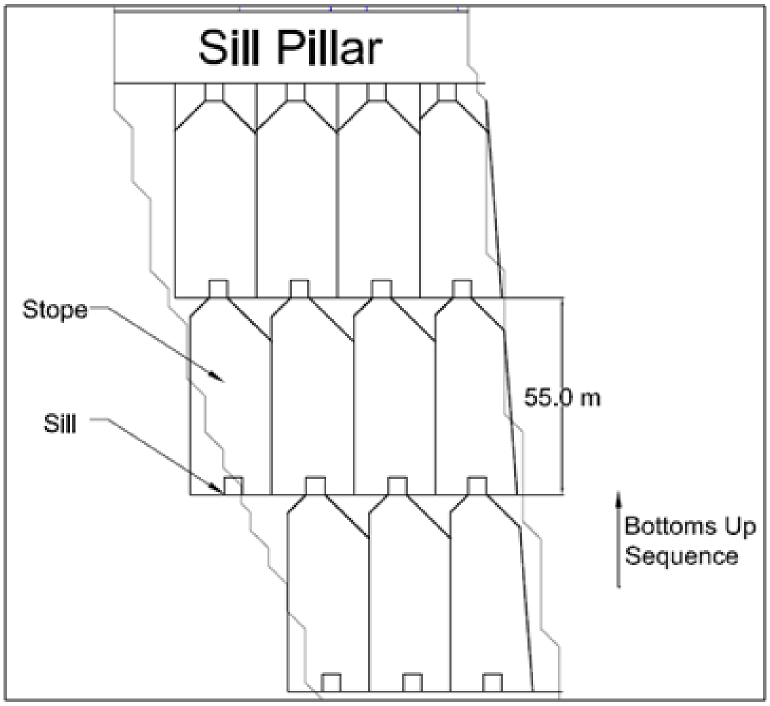


Figure 16-6: Pillarless Overhand Blasthole Mining Section

16.5.2 *Geomechanics*

Hatch and Itasca (ICCI) personnel visited the Lac des Iles site from October 28, 2014 – October 29, 2014. This site visit included an underground tour, as well as a workshop discussion with Lac des Iles operations and management personnel regarding mining methods and stope sizing. In order to determine a stoping dimension for the overhand blasthole with paste fill method in Offset Zone Phase 2, Hatch reviewed the Itasca document "North American Palladium, Lac des Iles Mine – Slope Stability Update" dated November 17, 2014 and in consultation with ICCI, determined the dimensions shown in Table 16-21 to be reasonable for use in the PEA study.

These dimensions were only used to determine level intervals and as a basis for recovery and dilution factors. Detailed stope designs were not done in the PEA. Primary and secondary stopes are assumed to be of the same dimension.

Table 16-21: PEA Stope Dimensions

Height	55 m
Length (Along Strike)	22 m
Width (Perpendicular to Strike)	20 m

Typical mining stope dimensions are displayed in Figure 16-7.

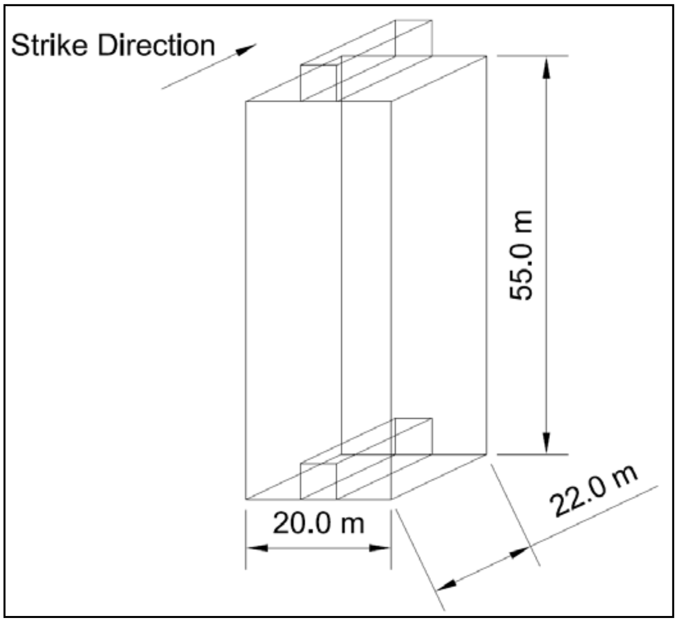


Figure 16-7: Typical Stope Dimensions for Offset Phase 2 Zone

16.5.3 Stoping

The Overhand Blasthole with Pastefill method determined in Section 16.1 is used as the basis for the PEA study for the Offset Zone Phase 2.

The stopes will be drilled using both uppers and downholes assuming 30 m downholes and 20 m uppers in a fan drilling pattern, with 5 m height of sill. Drillholes will have a nominal diameter of 4.5 in. It is anticipated that there will be some losses due to shoulders being left at the top of the stope. This is accounted for by the mining factors previously described. The reason for this recovery loss is due to the requirement of re-entry into the top sill that becomes the bottom sill for the stope above. The walls of this heading must remain intact to provide safe access.

Stoping areas will be developed as follows:

- Crosscut drifts will be developed from the footwall drift into the mineralized zone. The current assumption is that six of these crosscuts will be required for each level (see Section 16.5.4).
- Sill drifts will extend in either direction from the crosscut drifts to serve as drill and mucking horizons for the stopes. The stopes will be developed starting at the midpoint between two crosscuts and retreat back towards the crosscuts. This will provide multiple mining fronts along strike.
- For the purpose of the PEA, single mucking access has been considered.

- Stopes will be filled using pastefill, allowing for tight, consistent filling, and promoting improved mineral resource recovery.
- A 20 m sill pillar between Phase 1 and Phase 2 has been assumed.

Figure 16-8 displays the 3D block model visualization for the Offset Zone Phase 2 material.

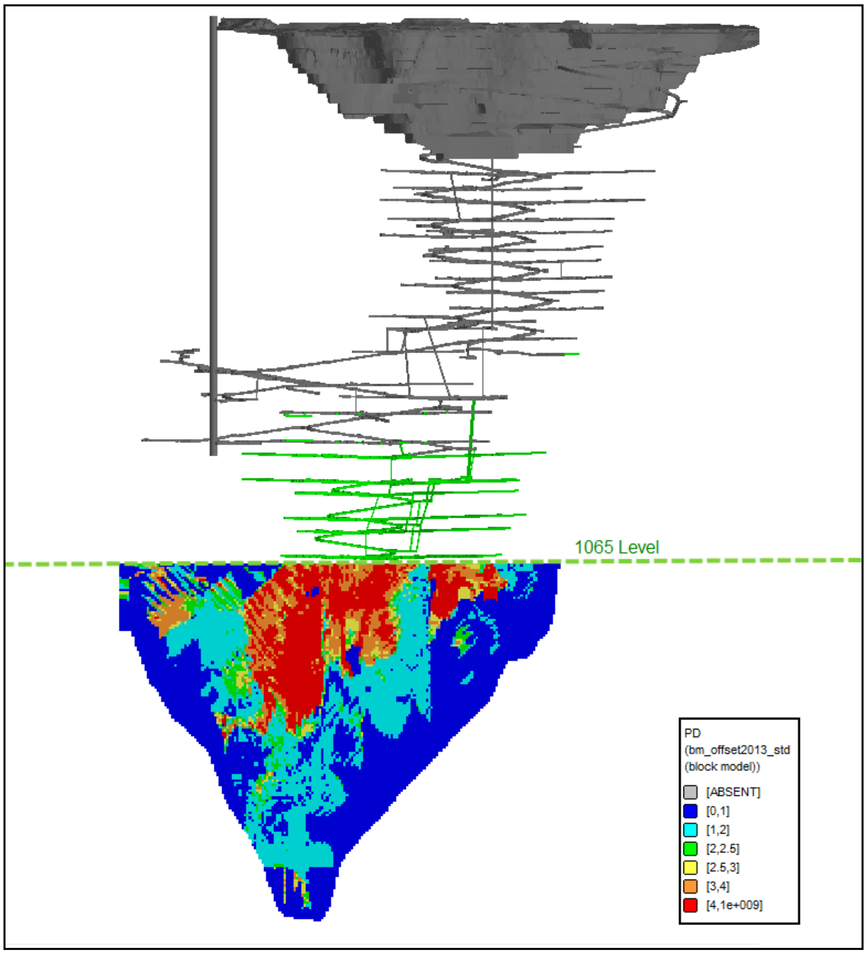


Figure 16-8: 3D Model Looking West

Table 16-22 lists the assumptions used to develop the cut-off grade and determine the resources to be included in the Offset Zone Phase 2 material:

Table 16-22: Operating Costs for Cut-Off Grade Estimation for the Offset Zone – Phase 2

Category	Cost
G&A ($/t)	$5.22
Milling ($/t)	$9.87
Surface Handling ($/t)	$3.03
Mining ($/t)	$33.99
Total Operating Cost ($/t)	$52.11
Royalty determined in the financial analysis	

Using the assumptions described above, the break-even cut-off grade was determined to be 2.25 g/t for the Offset Zone Phase 2. This cut-off grade was applied to both the HW mineral resources and the FW mineral resources, and the results are listed in Table 16-23. The mining factors (Table 16-24) were then applied to this mineral resource.

Table 16-23: Offset Zone Phase 2 Mineral Resources at 2.25 g/t Pd Cut-Off Grade

Offset Zone – HW below 1065L						
Category	**Tonnes (Mt)**	**Pd(g/t)**	**Pt(g/t)**	**Au (g/t)**	**Ni (%)**	**Cu (%)**
Measured & Indicated	4.1	4.42	0.31	0.33	0.13	0.11
Inferred	3.9	3.76	0.26	0.24	0.10	0.08
Offset Zone – FW below 1065L						
Category	**Tonnes (Mt)**	**Pd(g/t)**	**Pt(g/t)**	**Au (g/t)**	**Ni (%)**	**Cu (%)**
Measured & Indicated	1.0	2.83	0.21	0.16	0.09	0.07
Inferred	1.8	2.71	0.19	0.13	0.07	0.06

Factors were applied to account for mining related dilution and recovery losses. These factors were determined using typical factors for an overhand blasthole method using pastefill. The planned recovery factor was estimated by visually observing the grade distribution within the block model, and previous design experience.

Table 16-24: Offset Zone Phase 2 Mining Factors

Mining Factor	Value
Planned/Resource Recovery	90%
Unplanned/External Dilution*	15%
Mining Recovery	98%

*Zero dilution grade

The mineral resources used in the Phase 2 Expansion Plan after application of mining factors are displayed in Table 16-25:

Table 16-25: Offset Zone Phase 2 Mineral Resources Included in the Phase 2 Expansion Plan

Offset Zone – HW below 1065L						
Category	**Tonnes (Mt)**	**Pd(g/t)**	**Pt(g/t)**	**Au (g/t)**	**Ni (%)**	**Cu (%)**
Measured & Indicated	4.1	3.84	0.27	0.29	0.11	0.09
Inferred	4.0	3.27	0.22	0.21	0.09	0.07
Offset Zone – FW below 1065L						
Category	**Tonnes (Mt)**	**Pd(g/t)**	**Pt(g/t)**	**Au (g/t)**	**Ni (%)**	**Cu (%)**
Measured & Indicated	1.0	2.46	0.18	0.14	0.08	0.06
Inferred	1.8	2.35	0.16	0.11	0.06	0.05

The tonnage and grades shown in Table 16-25 were input into the economic analysis for the Phase 2 Expansion Plan.

16.5.4 Development

The development sequence for Phase 2 is as follows:

- Continue the ramp from Phase 1 down to 1405L, excavating level stubs with the advance of the ramp. This ramp advance is scheduled concurrently with shaft deepening activities and represents the critical path of the Offset Zone Phase 2 schedule.
- Establish level connection with the shaft on 1240L.
- Establish material handling infrastructure on 1405L (see Section 18) and continue ramp to shaft bottom at 1500L.
- Develop first stoping level on 1405L.
- Continue level development in a bottom-up fashion, as required to maintain production levels.

The development quantities have been determined using typical level layouts and an assumed effective ramp grade of 13%. Lateral and vertical development quantities have been factored to account for design growth and are listed in Table 16-26 and Table 16-27 respectively.

Table 16-26: Lateral Development Quantities

Ramp Quantities	Meters	Allowance	Total Meters
Ramp 1055L - 1460L @ 13% Grade	3,115	10%	3,427
Level Stubs (6 @ 15m)	90	10%	99
Level Quantities			
1075L	600	25%	750
1130L	600	25%	750
1185L	600	25%	750
1240L	600	25%	750
1295L	550	25%	688
1350L	450	25%	563
1405L	350	25%	438
1405L	400	25%	500
1460L	100	25%	125
Total Lateral Development	7,455	-	8,840

Table 16-27: Vertical Development Quantities

Type	Meters	Allowance	Total Meters
Fresh Air Raise (1055 - 1465)	410	0%	410
Return Air Raise (1055 - 1465)	410	0%	410
Ore Passes	1,000	5%	1,050
Total Vertical Development	1,820		1,870

Figure 16-9 displays a typical level layout for Phase 2. The projected mineral outline is shown in red.

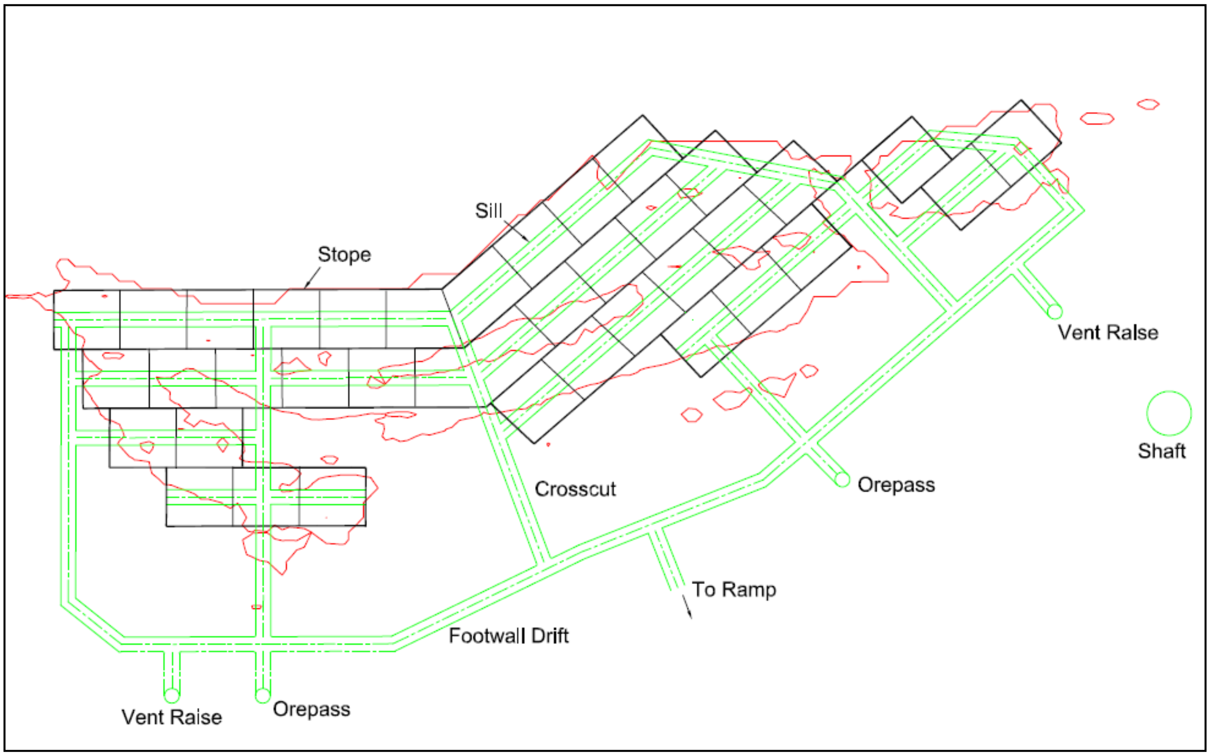


Figure 16-9: Typical Level Layout for Phase 2

16.5.5 *Production Rate*

To determine a possible mining rate, factors were drawn from both benchmarking data and the LDI 2015 budget. Current production rate based on trucking to the 645L rock breaker station is approximately 4,000 to 5,000 tpd.

The production rate for Phase 2 is estimated to be 6,500 tpd, based on the following:

- Elimination of trucking in the ramp by implementing an ore pass system for Phase 2 mining.
- Implementation of a mining method that improves recovery and reduces remote mucking.
- Deepening of the shaft and installation of a loading pocket at 1465L.

16.5.6 *Workforce*

For the purposes of the PEA, labour costs have been included in the operating costs for each method as described in Section 21. A detailed labour table was not created for use in this study.

16.5.7 *Mobile Equipment*

During Offset Zone Phase 2 mining, there will be no truck loading or truck haulage and the current mobile fleet will be able to handle the Phase 2 production.

The proposed material handling system on 1405L will require two large LHDs (Caterpillar R2900 or equivalent) dedicated to re-handling mill feed from passes to the grizzlies.

16.6 Production Forecast

In order to perform an economic analysis of potential expansion plans, it is necessary to include the current production plan (and associated reserves) in the evaluation to properly assess the potential economics of the expansion.

Two alternate expansion plans were developed:

- An Open Pit Expansion which adds an open pit and some additional footwall resources. This ultimately became the Base Case following the economic evaluation contained in this study.
- Phase 2 Expansion, which comprises the Offset Zone resources below 1065L.

16.6.1 *Current Mine Plan*

The Current Mine Plan is based on the previously stated mineral reserves, depleted of material mined through December 31, 2014, and augmented with the RGO stockpile and other Measured and Indicated mineral resources above 1065L. This includes some of the Roby FW mineral resources but does not include any Inferred mineral resources. This scenario has a mine life through 2021, with a total tonnage mined about 22.3 Mt, as illustrated in the yearly production schedule in Figure 16-10.

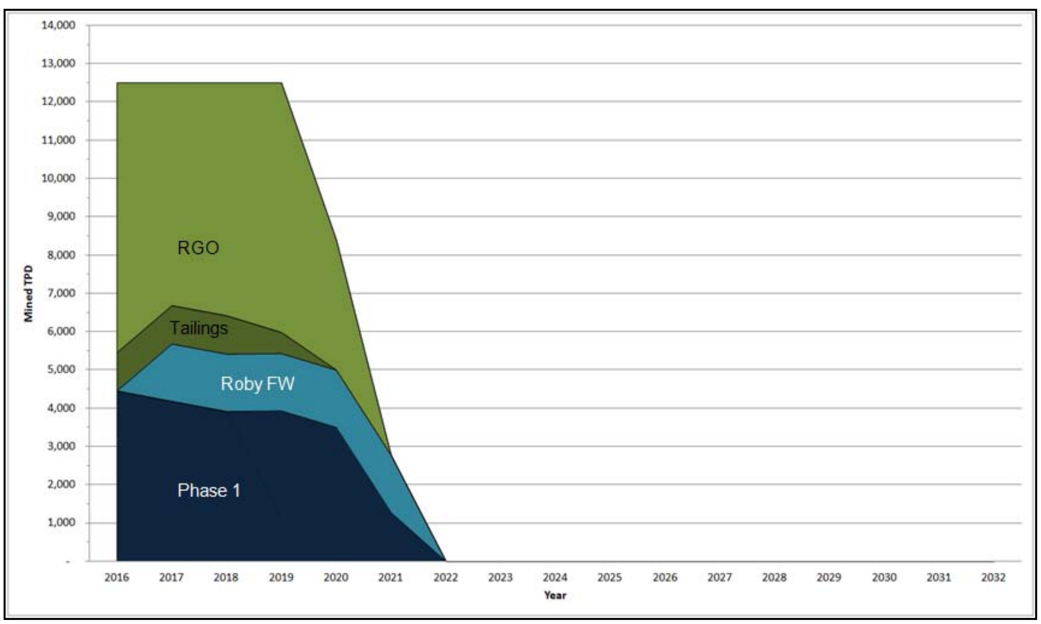


Figure 16-10: Current Mine Plan Yearly Production Schedule

16.6.2 Base Case Plan – Current Mine Plan plus Open Pit Expansion

This scenario adds the Open Pit Expansion as described in Section 16.4.2 and the balance of the Roby FW mineral resource, as well as the Offset FW resources above 1065L as described in Section 16.3.3. There are no Inferred resources included in this scenario. The Base Case Plan yearly production schedule is presented in Figure 16-11: .

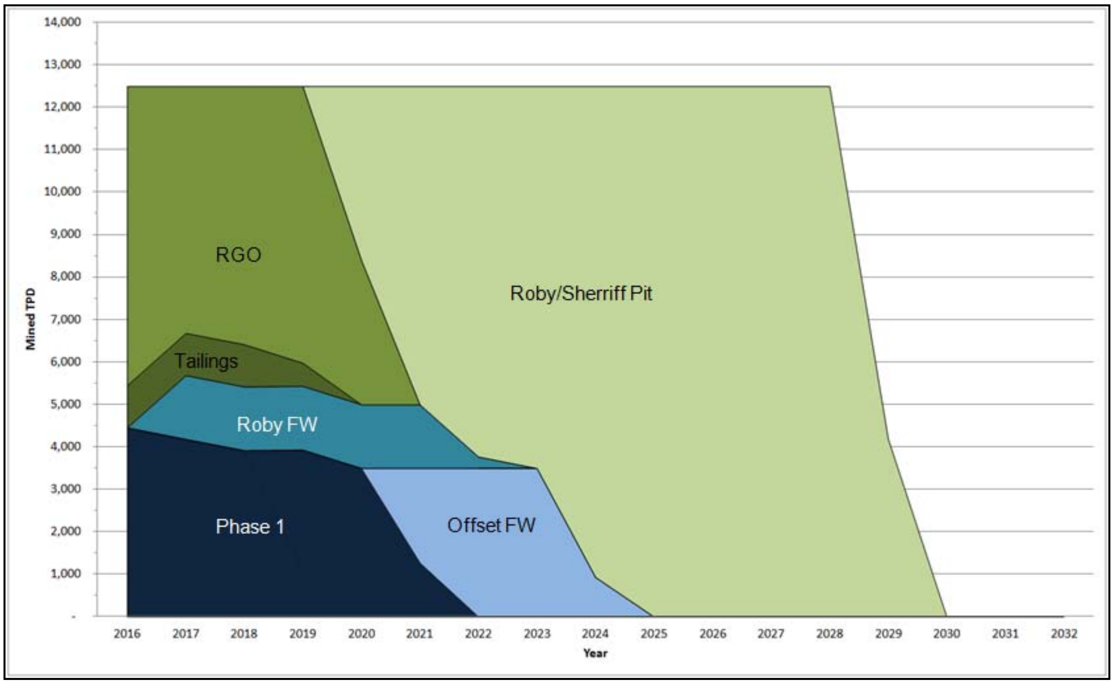


Figure 16-11: Base Case Plan Yearly Production Schedule

Table 16-28 displays the yearly production schedule for the Current Mine Plan and the Base Case.

Table 16-28: Yearly Production Schedules for Current Mine Plan and Base Case

Case	Feed Source (Tonnes per annum)	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	Total**
Current Mine Plan*	Tailings	0.4	0.4	0.4	0.2	-	-	-	-	-	-	-	-	-	-	-	-	1.3
	RGO	2.6	2.1	2.2	2.4	1.2	-	-	-	-	-	-	-	-	-	-	-	10.5
	Roby FW	-	0.5	0.5	0.5	0.5	0.5	-	-	-	-	-	-	-	-	-	-	2.7
	Offset HW - Phase 1	1.6	1.5	1.4	1.4	1.3	0.4	-	-	-	-	-	-	-	-	-	-	7.8
	Total**	**4.6**	**4.5**	**4.5**	**4.5**	**3**	**0.9**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**22.3**
Base Case (Current Mine Plan plus Open Pit Expansion)	Tailings	0.4	0.4	0.4	0.2	-	-	-	-	-	-	-	-	-	-	-	-	1.3
	RGO	2.6	2.1	2.2	2.4	1.2	-	-	-	-	-	-	-	-	-	-	-	10.5
	Open Pit - Roby, Sheriff, NVT	-	-	-	-	1.5	2.7	3.1	3.3	4.2	4.6	4.6	4.6	4.6	1.5	-	-	34.7
	Roby FW	-	0.5	0.5	0.5	0.5	0.5	0.1	-	-	-	-	-	-	-	-	-	2.8
	Offset HW - Phase 1	1.6	1.5	1.4	1.4	1.3	0.5	-	-	-	-	-	-	-	-	-	-	7.8
	Offset FW - Above 1065L	-	-	-	-	-	0.8	1.3	1.3	0.3	-	-	-	-	-	-	-	3.7
	Total**	**4.6**	**4.5**	**4.5**	**4.5**	**4.5**	**4.5**	**4.6**	**4.6**	**4.6**	**4.6**	**4.6**	**4.6**	**4.6**	**2.7**	**-**	**-**	**60.8**

* The Current Mine Plan is based on the previously stated mineral reserves, augmented with the RGO stockpile and other mineral resources above 1065L.
**Totals may not sum due to rounding.

17. Recovery Methods

17.1 Process Description

The present processing plant, which has a nominal capacity of 15,000 tpd, will process all mill feed produced from the Open Pit and Underground mines. The plant runs on a continuous 365 days per year schedule at 92% availability.

The crushing, grinding and flotation flow sheet is shown in Figure 17-1. A primary gyratory crusher is utilized to crush the run-of-mine (ROM) material before stockpiling. Pan feeders feed the main grinding circuit with a split from the coarse stockpile being fed to the secondary crushers to produce finer feed to augment mill throughput. Secondary crushing is accomplished with a cone crusher (standard HP800) with the product fed to the SAG mill.

There is one mill building at the LDI site. It is a 15,000 tpd pre-crush, SABC circuit that was built in 2000 and operated from 2001 until the mine was placed on temporary care and maintenance in October 2008.

Upon re-start of the mine in April 2010, the open pit remained closed, resulting in a drastic reduction in tonnage. The tonnage feed rate was not consistent, so running continuously was not a practical option. The mill process briefly changed to an autogenous/ball mill/crushing (ABC) circuit during the time when the feed grade from underground ore remained high but reverted back to an SABC circuit when underground ore was blended with RGO. The RGO is used to supplement mill feed or to adjust feed grade. The average head grade for the RGO is 0.97 g/t palladium.

The current mill's SABC configuration utilizes two Vertimills® for tertiary grinding to approach a grind of P80 38 microns (current grind varies from P80 47 to 50 microns). A third Vertimill® is currently not in service. The autogenous mill, currently being used as a SAG mill, has dimensions of 9.14 m in diameter by 4.27 m equivalent grinding length (EGL) and is fitted with a 6,400 kW (8,500 hp) motor. The two ball mills each have dimensions of 6.10 m in diameter by 10.36 m EGL and are also each fitted with 6,400 kW (8,500 hp) motors. The two Vertimills® use two 930 kW (1,250 hp) motors. A picture of the SAG and ball mills is shown in Figure 17-2.

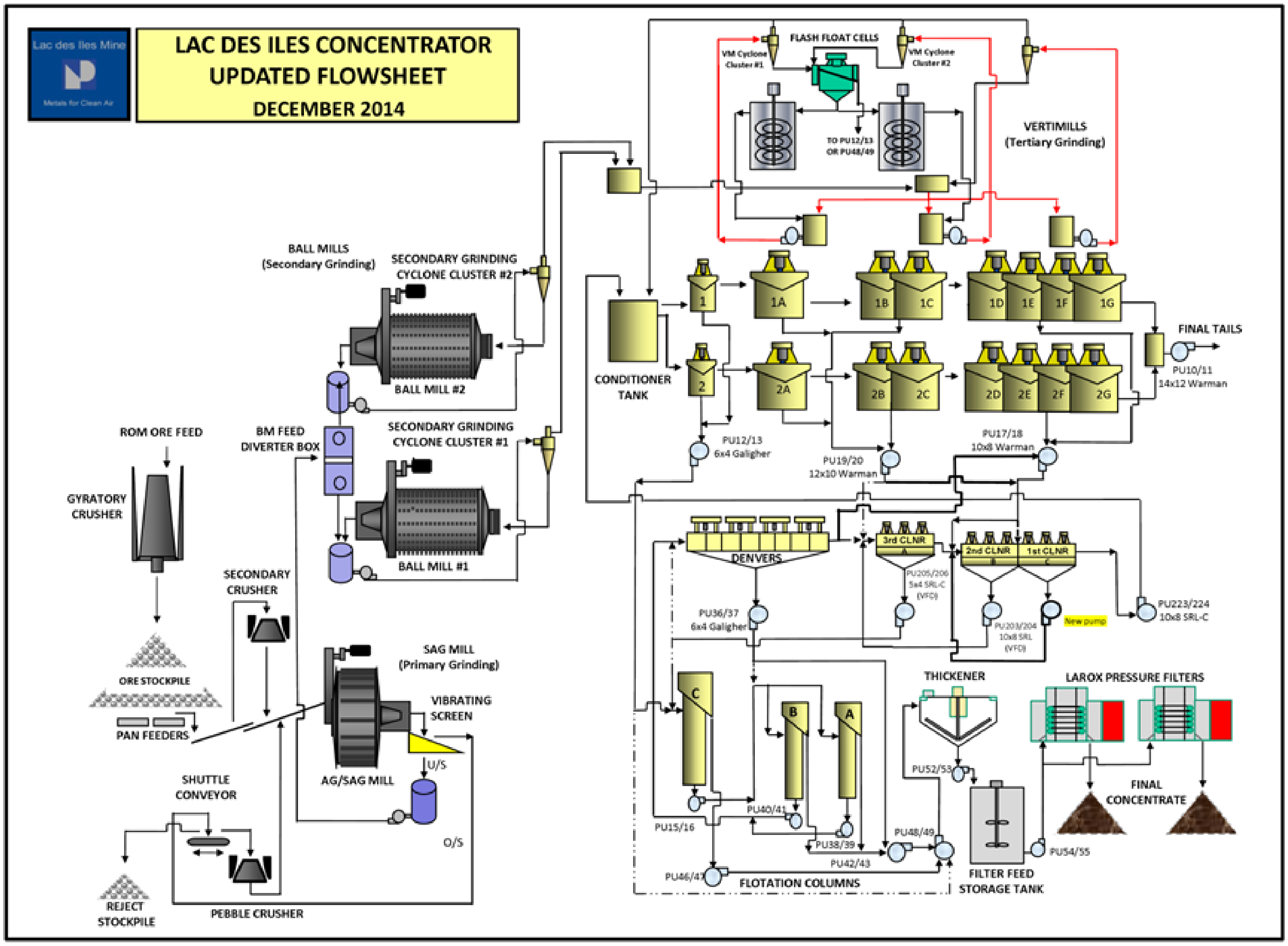


Figure 17-1: Current Lac des Iles Mine Flotation Circuit

Figure 17-2: AG (SAG) Mill & Ball Mills

The SAG product is passed over a vibrating screen with the oversize being belt fed to a pebble crusher (short head cone HP800) with the pebble crusher product recycled back into the feed to the SAG. The SAG discharge vibrating screen undersize feeds the ball mill diverter box which splits the feed between the two ball mills. Each ball mill is in closed circuit with a hydrocyclone cluster. The underflows from the hydrocyclones report back to the ball mills and the overflows report to a collection tank. The overflow is split so that 80% of the flow is transferred to the Vertimills® for tertiary grinding and the remaining 20% is introduced to flotation via the conditioning tank. The flow is split since the Vertimills® and associated sumps do not have the capacity to accept the entire overflow from the hydrocyclones.

Each of the Vertimills® is in closed circuit with hydrocyclone clusters. Hydrocyclone underflows are recycled back into the Vertimills® and the hydrocyclone overflows are transferred to the conditioning tank prior to flotation. Changes around the Vertimill® hydrocyclones were incorporated in December 2013 to increased plant efficiency. The hydrocyclones manufacturer assessed the performance of the hydrocyclones and determined that the feed lines were too small; resulting in high cyclone feed velocity. Proper feed parameters such as feed velocity are essential to cyclone performance as high feed velocity can cause finer particles to report to the cyclone underflow and increase the circulating load of the circuit. Reducing the circulating load of fines on the Vertimills® will allow more efficient grinding of coarser material. The feed line diameter increase has resulted in higher process recoveries since installation.

On December 19, 2014, one flash flotation cell was installed on the tertiary grinding circuit and is being commissioned. NAP plans to install a second flash flotation cell in 2015. Underflow from Vertimill® cyclone cluster 1 and cluster 2 flows to the flash flotation cell to recover fast floating, fine PGMs and base metals. This will alleviate the load to the rougher and scavenger banks, thereby allowing for more residence time in the flotation bank to enhance recoveries. The introduction of the flash flotation cell to the circuit prevents overgrinding of PGMs and base metal minerals which could alter the physical properties/shape of the mineral (flat shapes are difficult to float compared to round shapes) affecting its floatability. Cyclone cluster 3 underflow products will flow back to the distributor box since Vertimill® #3 is not yet operational. Depending on the resulting grade of the flash float cell concentrate products, this can either go to the rougher concentrate pump box (PU12/13) for pumping to Column C for cleaning or directly to the final concentrate pump box (PU48/49) for pumping to the concentrate thickener as a final concentrate.

The flash float cell tailings will gravity flow back to the Vertimills® for further grinding of the coarse cell tailings. Vertimill® discharges will gravity flow to each individual pump box, combining with fresh feed from the secondary grinding cyclone overflow, for pumping back to the Vertimill® cyclone clusters repeating the classification, flash float and tertiary grinding process of the Vertimill® circuit. It is expected that the quantity of flow bypassing tertiary grinding will be reduced after the flash flotation cell is commissioned.

The conditioning tank feeds two lines of roughers and subsequent scavenger flotation. The concentrate from the two lines of rougher flotation (cells 1 and 2), report to the rougher column cleaner cell (column cell C). The tailings from rougher flotation move forward to two lines of scavenger flotation (cells 1B to 1G and 2B to 2G). Tailings from scavenger flotation are final tailings which are pumped to the tailings impoundment.

The concentrate from the first scavenger (cells 1A and 2A) join the rougher concentrate reporting to the rougher cleaner (column cell C). Scavenger concentrate from cells 1B, 1C, 2B, and 2C combined with the tailings from the third cleaner and the concentrate from the first cleaners to feed the secondary cleaner. Scavenger concentrate from cells 1D to 1G and 2D to 2G feed the first cleaner along with tailings from the second cleaner. The tailings from first cleaner are fed back to the flotation conditioning tank. The tailings from the rougher cleaner (column C) are fed to the fourth cleaners (column A and B) along with the concentrate from the third cleaner. The tailings from the fourth cleaners (column A and B) are the feed to the third cleaner along with the concentrate from the second cleaner.

The concentrate streams from the rougher cleaner and the fourth cleaner are the final concentrate which is thickened and filtered to approximately 10% moisture. The concentrate is shipped via trucks to a smelter in Sudbury.

The LDI flotation process utilizes potassium amyl xanthate (PAX) as the flotation collector, methyl isobutyl carbinol (MIBC) as the frother, CMC as a talc depressant, and Aero 3477 as a PGM promoter. A graph of the historical grade – recovery relationship (blue data points) is presented in Figure 17-3 with a best fit regression line shown in black .With the commissioning of the flash cell, an increase in recovery of two percentage points is expected. This is represented by the red curve which has been used to estimate palladium recovery in the financial section of this report.

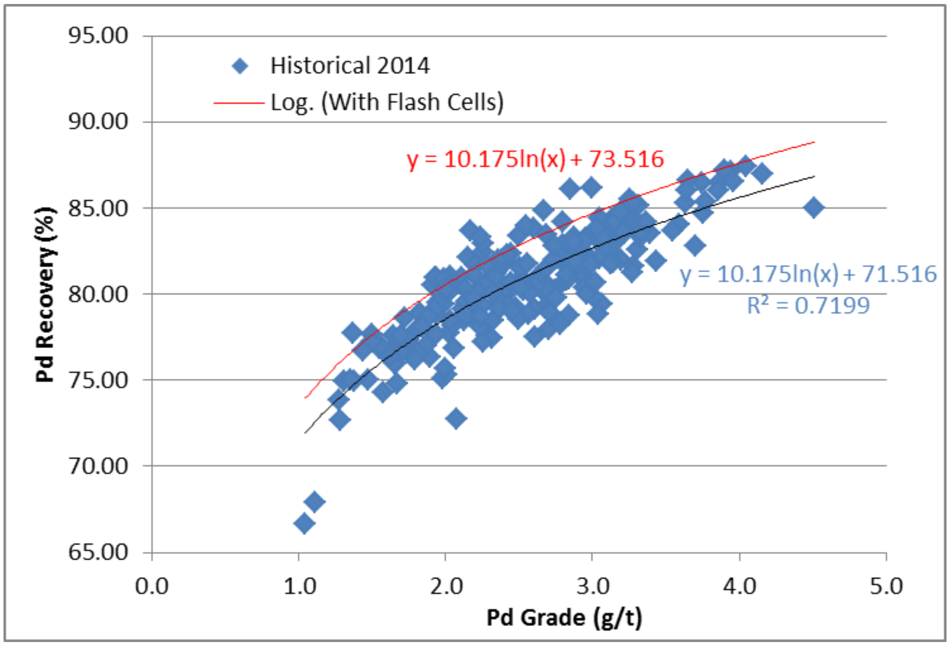


Figure 17-3: Historical Grade – Recovery Relationship

17.2 Historical and Current Mill Production

The historical and current production figures for LDI mine are shown in Table 17-1 and Table 17-2.

Table 17-1: Historical and Current Mill Production

	2014	2013	2012	2011	2010*	2008*	2007	2006**	2005	2004	2003	2002
Tonnes Milled (t)	2,684,783	2,048,083	2,063,260	1,689,781	649,649	3,722,732	5,006,383	4,570,926	4,780,926	5,298,544	5,159,730	4,851,621
Pd Head Grade (g/t)	2.70	2.79	3.44	3.7	6.06	2.33	2.36	2.18	1.66	2.41	2.31	1.91
Pd Recovery (%)	82.43	80.69	78.37	78.35	80.80	75.30	74.80	74.00	69.60	75.20	75.50	73.80
Pt Recovery (%)	77.02	74.79	72.87	72.01	74.20	64.70	64.70	64.90	60.60	66.90	-	-
Au Recovery (%)	79.78	79.00	76.92	74.07	77.20	74.60	72.90	73.00	70.00	74.40	-	-
Cu Recovery (%)	88.36	88.00	85.47	83.45	86.30	84.20	83.80	82.90	80.50	83.50	-	-
Ni Recovery (%)	36.68	39.13	36.80	30.83	34.40	36.90	34.90	35.20	30.00	40.50	-	-
Concentrate Tonnes (t)	21,520	16,966	17,883	11,708	5,517	26,176	33,060	28,979	30,698	45,652	36,714	27,200
Pd (oz) in Concentrate	192,025	148,117	178,835	157,476	102,212	212,046	286,334	237,338	177,167	308,931	288,703	219,300
Pt (oz) in Concentrate	14,556	11,444	12,528	10,265	5,499	16,311	24,442	22,308	18,833	25128	23,742	19,200
Au (oz) in Concentrate	13007	11600	12,487	8,314	4,624	15,921	20,092	17,238	14,308	25679	23,536	16,000
Cu (lb) in Concentrate	3,385,923	3,154,821	2,950,971	1,877,192	774,133	4,623,278	5,536,044	5,155,588	5,514,670	4,320,970	7,142,674	5,295,000
Ni (lb) in Concentrate	1,769,138	1,631,642	1,533,600	916,335	434,750	2,503,902	3,066,973	2,721,042	2,353,227	7,836,183	4,070,674	2,763,000
Mill Availability (%)***	94.6	98.30	98.30	97.90	95.70	88.40	91.10	86.50	86.50	88.40	91.10	90.50

Notes: * Partial year as the mine and mill were placed on temporary care and maintenance.

** Commercial production from the Roby underground mine began May 2006.

*** This factor represents the proportion of total annual time that the mill has performed its design function.

Source: North American Palladium – Lac des Iles Mine

Table 17-2: Historical and Current Concentrate Production Quality

	2014	2013	2012	2011	2010*	2008*	2007	2006	2005
Pd Grade (g/t)	278	272	311	278	576	261	289	272	174
Pt Grade (g/t)	21	21	22	27.3	31	19.9	23	23.5	19.1
Au Grade (g/t)	19	21	22	22.1	26.1	19.4	18.9	18.2	14.5
Cu Grade (%)	7.0	8.0	7.0	7.2	6.4	8.2	7.6	7.9	8.2
Ni Grade (%)	4.0	4.0	4.0	3.6	3.6	4.5	4.2	4.2	3.5
MgO Grade (%)	9.5	7.0	7.9	9.3	9.2	5.7	7.1	7.1	7.6

Note: * Partial year as the mine and mill were placed on temporary care and maintenance.
Source: North American Palladium – Lac des Iles

The assays and metallurgical data in this report are received from LDI/NAP and they are accepted to be correct. Hatch did not verify the accuracy of these data via independent third party laboratories.

17.3 Plant Operating Data

The LDI mill operating data for December 2013 to January 31, 2015 is depicted in Appendix I. One flash flotation cell was installed in December 19, 2014.

The Pd recovery – Pd head grade relationship before and after the addition of the flash cell in December 2014 is shown in Figure 17-4. Over this month the Pd head grade standard deviation is less than January 2015 so the impact of head grade variation interfere less on the Pd recovery after running the flash cell. Although the time period is short and the plant is under commissioning, with operations personnel still working to stabilize the process, there is early indication that the Flash Cell will improve recovery by approximately 2%.

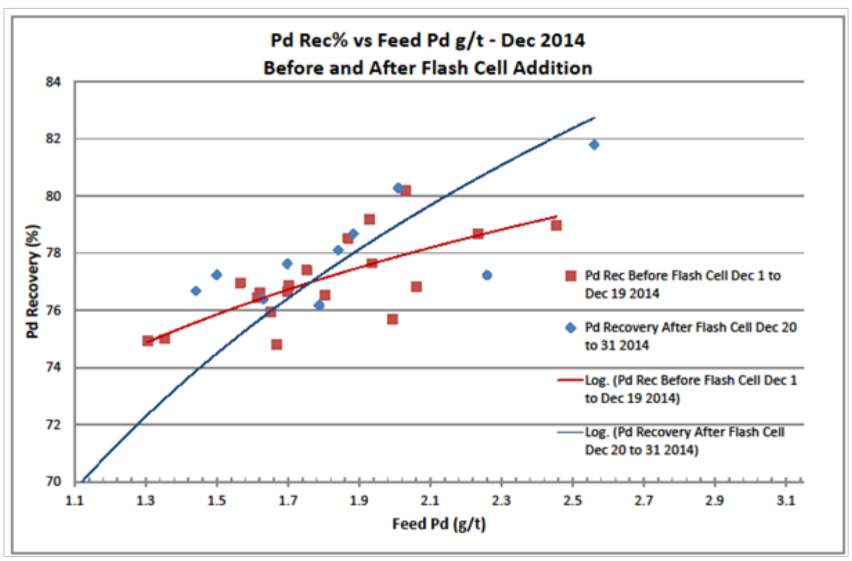


Figure 17-4: Pd Recovery% after and before the Commissioning of the Flash Flotation Cell in December 2014

Achieving higher precious metal recoveries with the introduction of flash cell is expected, based on industry experience. Figure 17-5 shows the gold recovery in the Esperanza flotation circuit with porphyry and andesite ore processed in conventional flotation circuit (squares, almost on top of each other) and in the circuit with the flash flotation cell.

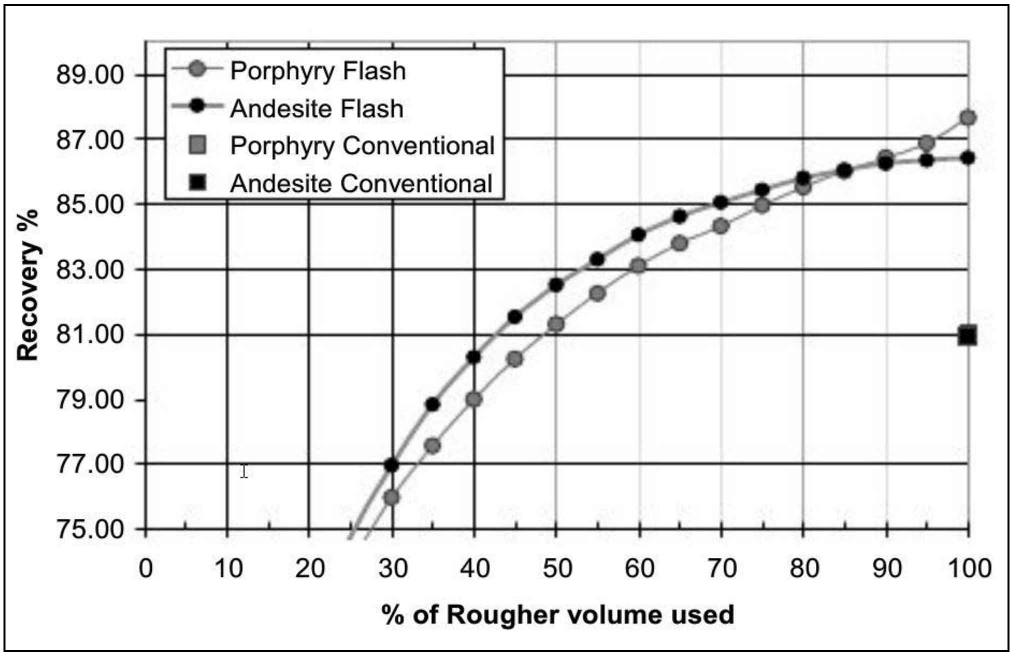


Figure 17-5: Gold Recovery in the Esperanza Flotation Circuit with Conventional and Flash Flotation

Source: Recent Advances in Mineral Processing Plant Design, Deepak Malhotra, Patrick Taylor, Erik Spiller, Marc LeVier, SME 2009.

18. Project Infrastructure

18.1 Site Overview

The site plan, Figure 18-1 shows existing facilities such as the operations camp area, the main administration offices, the tire shop, the Mill area, the open pit shops, the warehouse, the open pit and stockpile area, the underground portal, the shaft, related ventilation accesses, and the Tailings Management Facility.

Overall, these facilities are considered by the QP to be adequate for the needs of the mine, as well as meet the needs of the future open pit and underground mining plan.

The figure also shows the No. 1 Shaft that accesses Phase 1 of the Offset Zone. This shaft will require deepening as part of the Phase 2 project.

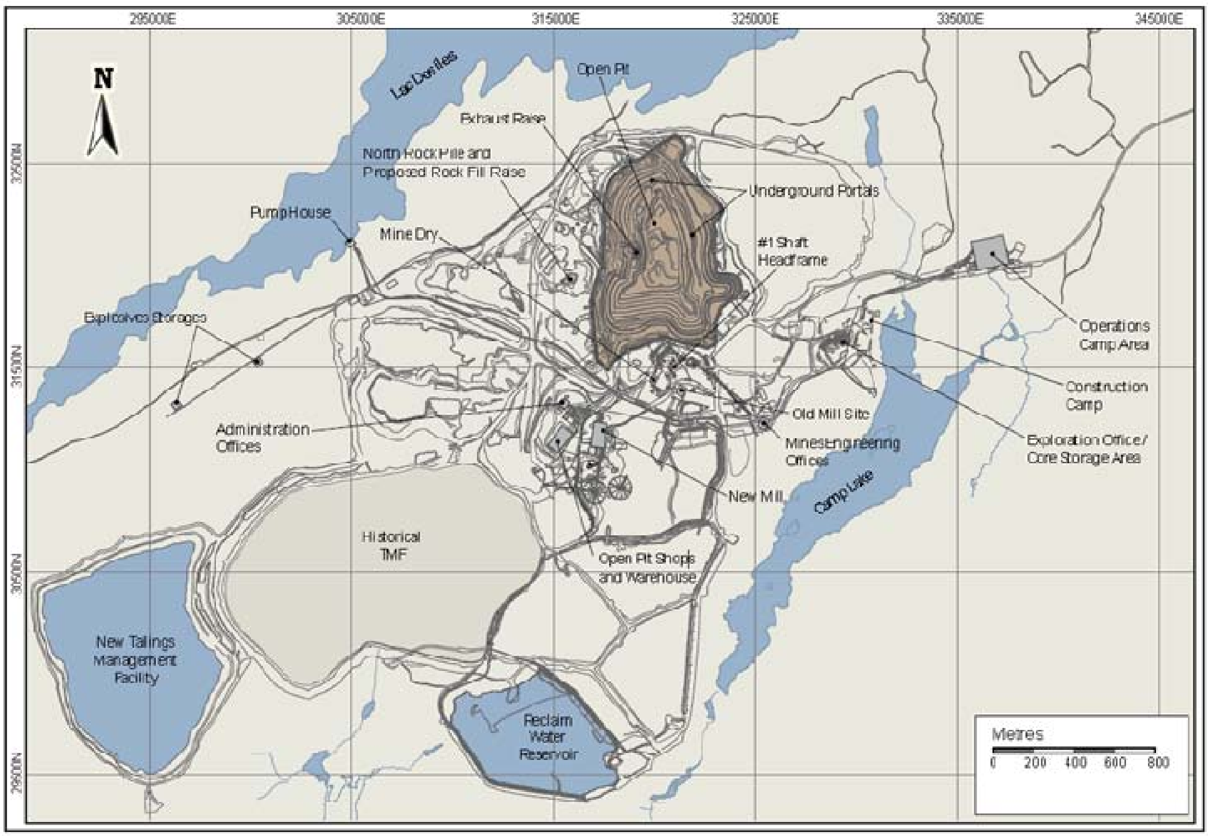


Figure 18-1: Lac des Iles Mine Site

18.2 Camp Facilities

In 2006, a 324-person operations camp and recreational complex was built in conjunction with the construction of the new mill. This has become the only operational camp residence on the Property.

The construction camp established in 2011, to accommodate added workers during Phase 1 construction period has been decommissioned.

Reactivation of at least a portion of the decommissioned construction camp will provide facilities to meet the construction accommodation requirements for the open pit and underground mining expansion.

18.3 Main Administration Offices

A recently constructed office complex centralizes management and supervision teams to improve interaction and communication between the site groups and operational departments.

18.4 Warehouse and Logistics

All purchasing is handled by the on-site staff, with regular freight movement between the site and Thunder Bay. On-site warehouse space accommodates spares for open pit and underground mining, the shaft, as well as milling operations.

The on-site warehouse facilities, originally established to service a 15,000 t/d open pit, have been found to be adequate for the mine’s needs and are considered by LDI to meet the needs of the existing and future mining plan.

18.5 Maintenance Facilities

The primary maintenance facility for the underground mobile fleet is on surface. Minor repairs on mobile equipment, such as drill and boom change outs on drill jumbos, that are difficult to move to surface are performed in underground service bays.

18.6 Fuel and Lube Distribution

Currently, the strategy has fuel delivered underground by mobile fuel/lube trucks that fill up on surface and travel the ramp to re-fuel and service some of the mobile equipment and also replenish the 685 level fuel bay tanks.

The underground haulage trucks are also fuelled on surface utilizing a highway style fuel truck located at the lower portal entrance, whenever they are trucking to surface.

Due to the increase in the underground mobile equipment activity over the past year, LDI is to install a more efficient fuel delivery system. This system will include surface storage tanks and fuel piping from the surface tanks to the 685 level fuel station via a fuel line located in the shaft, and will be capable of transferring fuel automatically on demand.

18.7 Storm Water Management

The current practice of using natural elevation differences, such that water is diverted away from buildings, and drains away in existing ditches to the lower lying areas will be retained. There is no requirement for additional storm water management to be put in place as a result of future underground mining, however, expansion plans for the open pit will address the additional water that will enter the open pit/underground workings as a result of an increased catchment area due to the larger pit surface area. Underground dewatering is discussed in Section 18.14.

18.8 No. 1 Shaft

The No. 1 Shaft hoisting plant, headframe and shaft were commissioned in October 2013 for Phase 1 operations.

The hoist house infrastructure includes the hoist house building, electrical room, and compressor room. It is connected to the shaft sub-collar via a services tunnel. The hoist house contains the following hoisting plants:

- 3.70 m diameter by 1.90 m wide double drum service hoist for the cage and counterweight.
- 2.79 m diameter by 1.80 m wide single drum hoist for the auxiliary cage.
- 4.67 m diameter by 1.91 m wide double drum production hoist for skipping.

The Shaft headframe consists of a conventional back leg arrangement and designed to accommodate the forces exerted by the hoisting plants and associated shaft loads to a depth of 1,350 m below collar elevation. The headframe infrastructure includes:

- A collar and a sub-collar.
- A ventilation plenum entering the shaft at the sub-collar elevation.
- A collar house.
- A bin and load-out facility.
- A man access stairwell and tunnel linking the headframe to the main administration offices and dry.

The No.1 shaft was sunk to a depth of 829 m below collar during Phase 1, and is used for personnel and material movement to/from 740L underground, as well the hoisting of ore from the underground 740L loading pocket. The shaft also serves as a fresh air intake as part of the mine's underground ventilation system.

As shown in Figure 18-2, the No. 1 shaft consists of the following four compartments:

- Two skip compartments: skip size of 1,500 mm by 1,500 mm. Each skip has a payload capacity of 15,420 kg.
- One double deck cage and counterweight (CWT) compartment: cage size of 3,600 mm by 1,800 mm and CWT size of 1,700 mm by 500 mm. The capacity is 84 people or 10,886 kg of payload
- One double deck auxiliary cage compartment: auxiliary cage size of 1,840 mm by 710 mm. The total capacity is 4,600 kg payload.

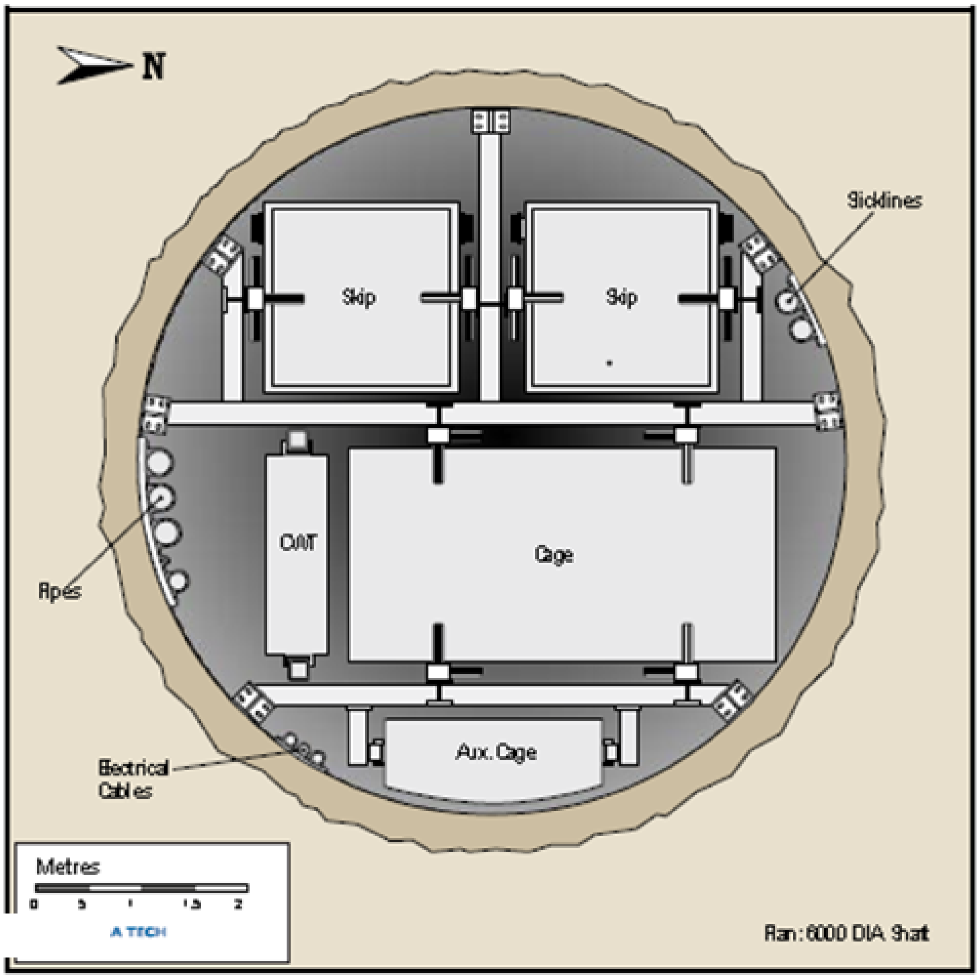


Figure 18-2: Shaft Cross-Section

The shaft has a finished inner diameter of 6.0 m and a concrete lining thickness of 150 mm, with the lining completed to the 822 m elevation. Steelwork for phase 1 has been installed and commissioned to the 790 m elevation. Wooden guides are currently being used.

18.9 Main Ramp

The main ramp is a 15% decline accessed from two portals, the upper portal and the lower portal, both located in the open pit. The decline has a nominal 5.0 m height (flat back) and 5.0 m width profile to approximately 839L, where the profile then becomes 5.5 m high (arched back) and 5 m width profile. The decline profile is sufficient to allow the largest underground mining equipment to quickly and safely access the lowest levels of the mine. Safety stations are installed every 30 m along the ramp.

The ramp provides the means of access for supplies and equipment movement into the mine that cannot be transported down the shaft. This is accomplished using boom trucks and fuel trucks. With the main repair garage located on surface, the ramp is also used to move equipment for servicing and repairs that cannot be completed in the smaller underground service facility.

Portions of the ramp are also used for:

- Haulage of ore from the underground stopes to the truck dump on 645L or to surface.
- Haulage of waste from the lower mine development to the upper levels for backfill purposes.
- Haulage of waste backfill from the 670L waste pass/truck chute to empty stopes as required.

Prior to the commissioning of No. 1 Shaft, the ramp was the only access into the mine and the only route by which ore was removed from underground. The ramp is still utilized for ore/waste movement when the shaft hoisting system is not available.

18.10 Ore Handling System

The current ore handling system planned for mine production down to 1055L is as follows:

- Material is mucked by LHDs and dumped into remucks at the entrance to the level. It is then loaded into trucks for haulage, either to the 645L rock breaker station or to surface for oversize ore.
- Material delivered to the 645L rock breaker station is sized through a grizzly, fed down an ore pass to the 685L crusher station, crushed to ±150 mm and then transferred to storage bins feeding the 740L loading pocket prior to skipping to surface.
- On surface, the skips dump into the load-out system, which incorporates an apron feeder that moves the mill feed into one of two load out bins. Arc gates are used to load surface haulage trucks for overland transport to the main stockpile at the mill area.

18.11 Waste Handling System

Waste generated underground during development and production is placed into mined out stopes as backfill without cement. This may be carried out by LHD's alone or by a LHD and truck combination. Although there is considerable variability, the average is roughly 500 tpd.

18.12 Backfill System

Backfilling of mined out areas is required to support stoping activities. Stopes are filled to minimize wall sloughing and general rock deterioration. Since waste rock from mine development activities is not available in sufficient quantities to meet the mine's total backfill requirement, stockpiled surface waste rock, primarily from the old open pit mining operation, is utilized. This waste rock is crushed to ± 150 mm then sent underground through an existing 2.4 m diameter bored raise to a truck loading chute system on 760L. The waste rock is then trucked where required. The fill is not consolidated with cement.

18.13 Material Movement

At present, mining consumables are distributed from surface to underground storage areas either by the ramp in service vehicles or via the shaft and distributed to the storage areas using service vehicles. This practice will continue as the mining progresses. Additional storage areas will be constructed as required.

18.14 Mine Dewatering

As the mining operations progressed from open pit to underground, two "cascade type" pumping arrangements were implemented for the upper sections of the underground mining operations. Water is pumped out of the mine via pipelines installed in man-ways, and along the ramp. The system is used to dewater the current mining operations and to deal with localized inflows that can occur during spring run-off or heavy rain events. There is an Emergency Water Management Plan in place for such events.

With the commissioning of the shaft, additional dewatering and process water pipelines have been installed in the shaft. A permanent pumping station is planned to be constructed near the shaft on 685L in 2015 which will then allow underground water to be pumped via the vertical shaft system.

18.15 Mine Ventilation

The current ventilation installation consists of a push-pull system using a network of main fans on surface and booster fans underground.

Fresh air is supplied via the production shaft and a ventilation raise that tops out in the open pit. Both shaft and raise systems have propane fired heaters to heat the incoming fresh air in winter months to avoid freezing of water and services in the mine workings.

Return air is drawn from underground via a return air fan located in the open pit, as well as leakage through connections from underground workings to the floor of the pit. Approximately 378 m^3/s of fresh air is presently being supplied to the underground operation and sufficient capacity is available to supply the balance of the Phase 1 mining plan.

18.16 Refuge Stations and Sanitary Facilities

Refuge stations and sanitary facilities are strategically placed throughout the mine. As well as a place for refuge in the event of an emergency, refuge stations are used as a lunchroom and as a meeting and communications area.

18.17 Secondary Egress

An alternate means of egress is required in the event of a disruption in shaft operation. The current means of secondary egress is a combination of the main ramp and a manway located in the fresh air raise.

18.18 Stockpiles

Waste dumps and stockpiles of various grades have been established on surface near the concentrator facilities. One significant aspect is that the waste rock from the pit walls is relatively benign and is classified as non-acid generating.

There is a low grade stockpile of approximately 11.2 Mt grading 0.97 g/t palladium called the RGO Stockpile. It is currently part of the mill feed schedule, and is being reclaimed and blended with ROM underground material to ensure the mill feed is maintained at a nominal 12,500 tpd.

There is approximately 20 Mt of waste material in the surface stockpiles that is used as rock fill for underground. As described in Section 18.11, waste rock is crushed and sent underground for backfill purposes.

18.19 Tailings Waste Management

LDI has been operating a TMF throughout the life of the mine. The design of the operation is expected to facilitate closure and reclamation of the facility at the end of mine life.

The TMF comprises three separate tailings storage areas (i.e., the South, East and West). The TMFs are located adjacent to one another, southwest of the open pit. Tailings from the future mining will be disposed of in the South and East TMFs.

The South TMF, as of January 2014, has enough storage capacity at the 506.2 m Ground Datum for an estimated one year capacity at the 2014 mill budget ore throughput. A downstream rock lift was added to the South tailings dams to increase the overall storage capacity. This project was completed in late 2013. Currently, all wastes created by the milling process are deposited in this facility. Dams and dykes are located around and through the East and South TMFs. The East TMF dams are in the process of being raised by 3 m to increase total tailings capacity to 67 Mt.

The West TMF was closed upon reaching maximum permitted elevation and is undergoing progressive reclamation; however LDI is currently evaluating alternative options for tailings storage capacity, due to the increased construction costs associated with the transition between open pit and underground mining.

All water within both TMFs is reclaimed to the milling process on site and treated before being discharged to the environment, when needed.

The current footprint of the TMF has an ultimate capacity of approximately 60 Mt and can accommodate the planned production from the Base Case mine plan.

18.20 Electrical Distribution

Electrical power is supplied by Hydro One via a 115 kV line to three main substations on the LDI site. Site distribution is the responsibility of LDI and consists of 115 kV, 13.8 kV and 5 kV overhead lines around the site. In December 2013, approval was received for LDI to increase supply power from 38 MW to 47 MW from the Hydro One power grid. This increased allowable power supply is deemed to be sufficient for the current and future open pit and underground mining plan.

There are two separate systems feeding electrical power underground, namely the 5 kV system routed via the underground ramps and the 13.8 kV twin feed down the shaft.

18.21 Compressed Air

Compressed air for underground operations is supplied by two 451 kW Sullair Air twin stage screw compressors located in the compressor room adjacent to the hoist house. (A third compressor is installed as an operational spare).

The compressed air is transferred from surface to underground using 200 mm pipes located in the shaft and then distributed via 152 mm piping along the ramps and mining levels up to stope access cross-cuts. Piping in the stopes is typically 102 mm in diameter.

With an estimated capacity of 3.5 m3/s (7,590 cfm) at 7.6 to 8.3 bar (110 to 120 psi) pressure, the compressors are deemed sufficient to supply the current and future needs of the mining plans.

18.22 Underground Process Water

Process water is sent underground in two 102 mm pipes located in the shaft. This feeds the main 102 mm diameter level distribution lines on the levels and the stope access cross-cuts. Piping in the stope accesses and sills is typically 50 mm. Water pressures and volumes are controlled by pressure reducing valves installed in the shaft at approximately 115 m intervals.

The current system will be extended as the mine gets deeper.

18.23 Surface Services

Water and sewer services are supplied independently for each facility and are considered by LDI to be adequate for current and projected needs. Expansion of potable water and sewer services were completed in 2011 for the underground workforce additions. Grey water discharges into holding tanks that are pumped out on a daily basis with the contents then being discharged into one of the TMF areas.

18.24 Mine Communication and Control Systems

Mine communication networks are in place in the shaft for voice communications, and data networking for the programmable logical controllers (PLC's) for the hoist, loading pocket and underground crusher. Communication is through a leaky feeder and fibre optic system installed in the shaft.

A leaky feeder system is also used through the primary development and level accesses to facilitate radio and data transfer throughout the mine.

18.25 Open Pit Mining - Impact on Mining Related Surface Infrastructure

This section addresses the possible affect that the Open Pit Expansion could have on current infrastructure that supports underground operations.

The expansion plan for the open pit mining will significantly increase in the size of the open pit, and the resumption of surface mining activities at LDI could impact key infrastructure as indicated on Figure 18-3.

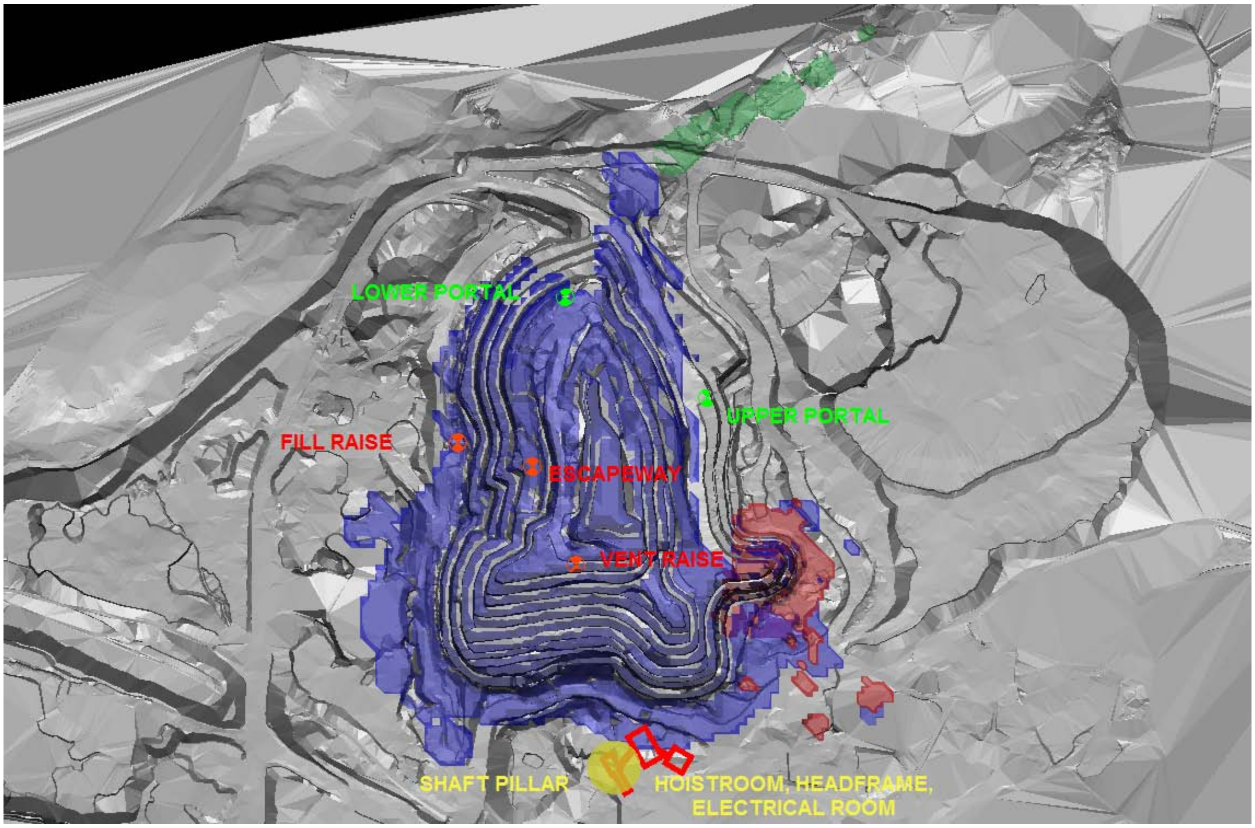


Figure 18-3: Underground Mining Related Surface Infrastructure Affected by the Future Pit Mining

The affected infrastructure, as discussed below, will have to be either removed or relocated (if possible), or production in these areas of the pit will be affected/delayed. Costs have been estimated (summarized in Section 21) to account for the replacement of this key infrastructure, as required.

18.25.1 Shaft Infrastructure (Headframe, Substation, Hoist House and Electrical Substation)

The shaft and associated infrastructure is located at the south end of the pit, within the potential ultimate pit limits. For the purposes of the PEA, it is assumed that the open pit can be scheduled such that the shaft infrastructure will not be compromised until it is no longer needed. This will require further investigation at the next level of study.

18.25.2 Lower Portal

With the mining of the eastern wall of the pit, the ramp leading to the Lower Portal will be mined out completely. A new portal on surface outside of the ultimate pit footprint will be constructed and a new surface ramp would be driven from this portal to intersect with the ramp currently in place. This would require approximately 850 m of underground lateral development.

18.25.3 Ramp Bypass Drift

Approximately 500 m of bypass ramp would be required underground.

18.25.4 Upper Portal

It is not expected that the upper portal would be removed with mining activities in the north eastern wall. However, its future use would be negated with the construction of the new portal described above.

18.25.5 Ventilation Raise

The ventilation raise that currently breaks through at the bottom of the pit will interfere with surface mining activities and will need to be moved outside of the ultimate pit footprint. The new ventilation raise will need to accommodate the volumes of air currently flowing in the escapeway (See Section 18.25.6). This cost is included in the open pit estimate in Section 21. Further design of this will need to be completed in the next study phase.

18.25.6 Escapeway

The escapeway that currently breaks through the west wall of the pit will interfere with surface mining activities. This is no longer required as an escapeway, as secondary egress is provided through the ramp network.

18.25.7 Fill Raise

The rock fill raise will interfere with surface mining activities. An allowance has been made to replace the raise for the Open Pit Expansion Plan.

18.26 Major Infrastructure Changes Required for Phase 2

This section includes the key infrastructure requirements for the Phase 2 underground mining expansion plan. This includes the No. 1 Shaft deepening, the construction of a new ore handling system, and a new backfill system (paste plant and underground distribution network.

18.26.1 No. 1 Shaft Deepening

As part of the Phase 2 mining scenario, it is proposed to deepen the existing shaft to approximately 1,500 m in depth. With a conversion to steel guides and other related changes, the shaft and hoisting system will be capable of 6,500 tonnes per day from the 1465L loading pocket.

A shaft sinking company (Cementation) was engaged to carry out a conceptual study relating to the Phase 2 shaft deepening. This involved a conceptual evaluation of the current sinking hoist and its production capacity at depth. The conclusions arrived at were:

- The existing sinkers bulkhead at 790L will be modified / replaced to allow sinking to be carried out concurrently with skipping operations.
- New temporary shaft sinking equipment will be designed and fabricated.
- The existing temporary shaft sinking winches/winch rope locations will be used for the Galloway.
- The existing cage hoist will be used as the shaft sinking hoist.
- LDI will provide access at 990L and the sinking contractor will use an Alimak pilot and slash method to raise the shaft from 990L up to the 825L.
- This will establish a pre-sink shaft zone to erect and commission the Galloway prior to commencing full shaft sinking.
- The shaft geometry and configuration will be identical to the existing shaft.
- It is anticipated that four stations will be established namely at 1055L, 1240L, 1405L and the 1465L loading Pocket.
- An ore pass system between 1055L and 1405L will be constructed as part of the shaft sinking scope.
- The top of each raise location will incorporate a truck dump/transfer station complete with grizzly and rock breaker. The intent is to use this system to continue production from the upper sections of the mine by using the 1465L loading pocket while the ramp and lower level ore handling system is still being constructed.
- The 1465L loading pocket will consist of a measuring box with a transfer car-style chute at the top and arc gate arrangement to feed the skips.
- The shaft depth is currently planned to be 1500L (shaft bottom).

A ramp will be driven to the shaft bottom to allow for shaft spillage material removal and development of a sump for shaft bottom dewatering.

Cementation has provided a scoping level cost estimate which has been included in the capital estimate.

During shaft deepening/sinking for Phase 2, the main cage will not be operating as the hoist will be used for sinking operations below 825L. Therefore, underground personnel will use the auxiliary cage or mobile man carriers to travel down the ramp. Material and supplies movement will also require use of the ramp or the auxiliary cage when available.

18.26.2 Ore Handling System

All waste generated for development will be placed as fill into mined out stopes. A series of three internal ore passes are expected to be required for the Phase 2 ore handling system, as shown in Figure 18-4. These will begin on alternating levels, with finger raises driven to allow for intermediate loading.

Each production level in Phase 2 is to have at least two accesses into the orepass system. This staggered approach to ore passes will minimize dust at transfer points and at dump horizons.

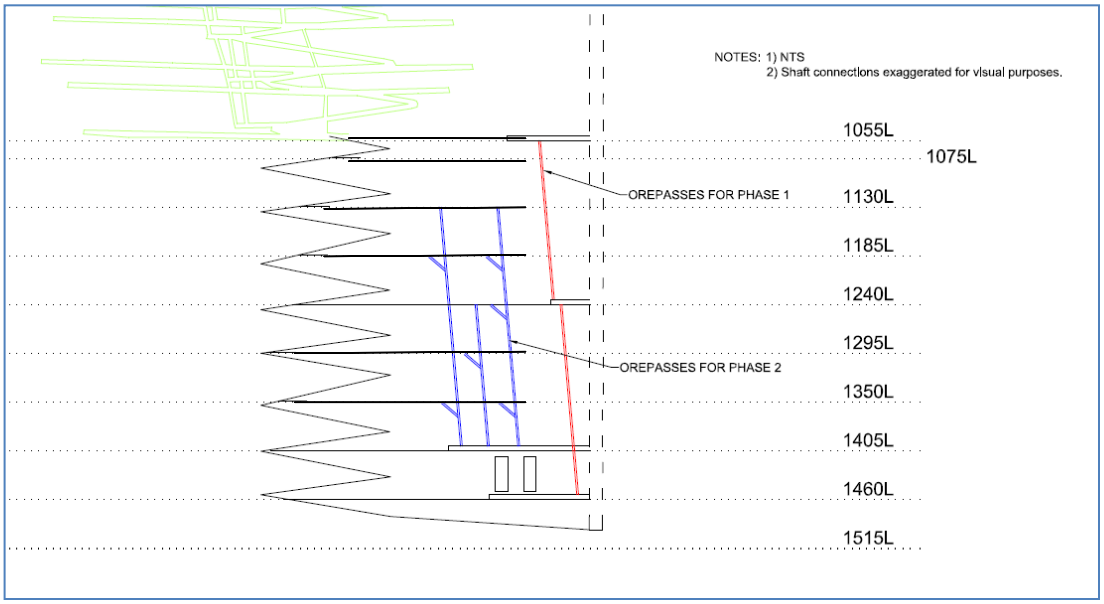


Figure 18-4: Phase 2 Ore Handling Schematic

The ore passes will bottom on 1405L where the ore will be loaded by an LHD and trammed into one of two rockbreaker/grizzly stations. A dedicated tramming drift for the ore above 1405L is allowed for as shown in Figure 18-5.

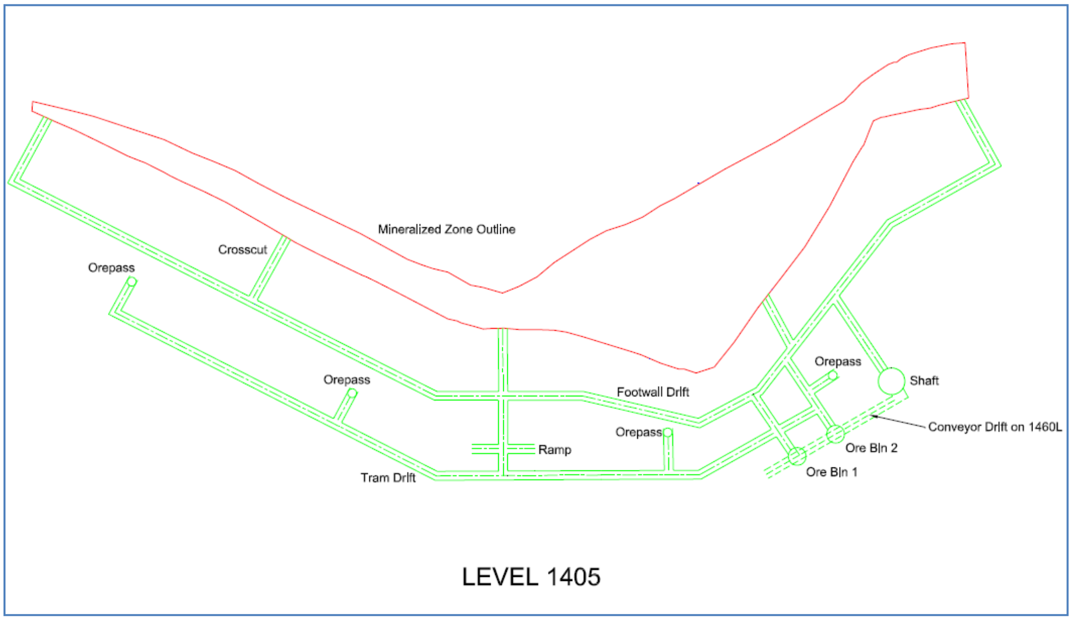


Figure 18-5: Plan of 1405L

A rockbreaker/grizzly arrangement sits over each of the two ore bins. On 1465 L, underneath the bins, a vibrating feeder feeds a conveyer that goes to the shaft loading pocket system. This system is illustrated in Figure 18-6.

Prime ore storage will be in two ore bins located on the 1405L (bin top); feeding to a conveyor system on 1465L to load skips to surface.

Any ore produced below 1405L will be hauled up the ramp to 1405L in haulage trucks and dumped directly onto the grizzlies.

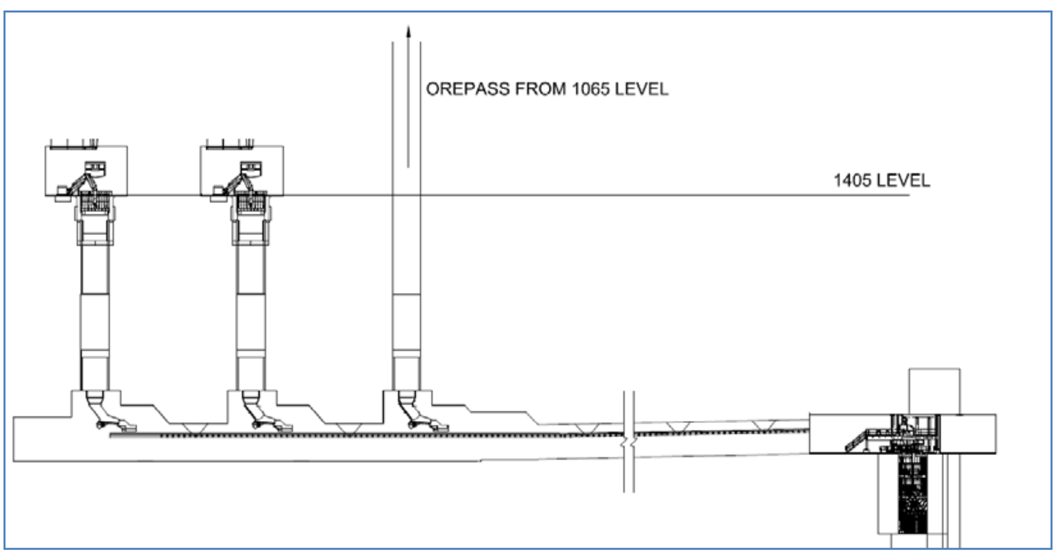


Figure 18-6: 1465L Loading Pocket Level

Concurrently with ramp development and shaft sinking, an additional ore pass between 1065L and 1465L is to be driven to allow for Phase 1 ore to be hoisted from 1465L without trucking to the 645L Truck Dump. The bottom of this raise will have the same feeder/belt arrangement as the bins. Dump points into this additional raise will also require a rockbreaker/grizzly installation.

18.26.3 Backfill

As indicated previously, a Paste Fill Plant and distribution system will be installed for Phase 2.

A preliminary design was completed based on 6,500 tpd mining production with a backfill nominal requirement of 3,050 tpd and a plant design capacity of 4,765 tpd (dry). The design parameters are summarised in Table 18-1.

Table 18-1: Summary of Design Parameters

Description	Units	Value	Source
Mining production	t/d	6,500	NAP
Operating days per year[1]	days	355	NAP
Operating hours per day	Hrs	24	NAP
SG of mined material		2.89	Golder
SG of tailings		2.82	Golder
SG of dry binder (GBFS/T10)		3.00	Golder
Voids created	m^3/day	2249	Calculated
Percent of voids not filled with cemented paste	%	7.3	Calculated
Tonnes of uncemented fill	t/d	475	Mining
Nominal volume of void to be filled with paste (slurry)	m^3/day	2085	Calculated
Nominal fill requirement (solids)	t/d	3040	Calculated
Availability[2]	%	80	Calculated
Operating fill requirement (solids)	t/d	3810	Calculated
Design Factor[3]	%	25	Assigned
Design backfill production (solids)	t/d	4765	Calculated
Design tailings requirement	t/hr (dry)	192	Calculated
Design binder dosage rate[4]	%	3	Golder
Design binder addition	t/hr (dry)	6	Calculated
Design fill rate (solids)	t/hr (dry)	198	Calculated
Design volumetric fill rate (slurry)	m^3/hr	136	Calculated
Solids content[5]	Wt% s	75	Golder
SG of slurry		1.94	Calculated

Mill tailings will be pumped from the mill to the adjacent backfill plant. A thickener will dewater the tailings and the thickener underflow will then be fed into vacuum disc filters to be further dewatered after which the filter cake will then be fed into a paste mixer.

The paste backfill will be delivered by gravity to mined stopes via twin cased boreholes and an underground pipeline distribution system. The boreholes will be drilled from a point in or near the backfill plant and terminate in an underground pipeline connection station. Since Phase 2 mining will commence at the bottom, the distribution system will need to be completed to the bottom at the onset of production.

When backfill is not required, tailings will be pumped from the mill to the TMF.

18.26.4 Dewatering

The dewatering concept for Phase 2 is show in Figure 18-7, and is to consist of:

- A main sump and pump room on 1465L. This will be similar to that currently proposed for 685L for Phase 1.
- Pipe column(s) in the shaft to discharge into the 685L sump.
- A series of collection sumps and drain holes connecting production levels as mining progresses.

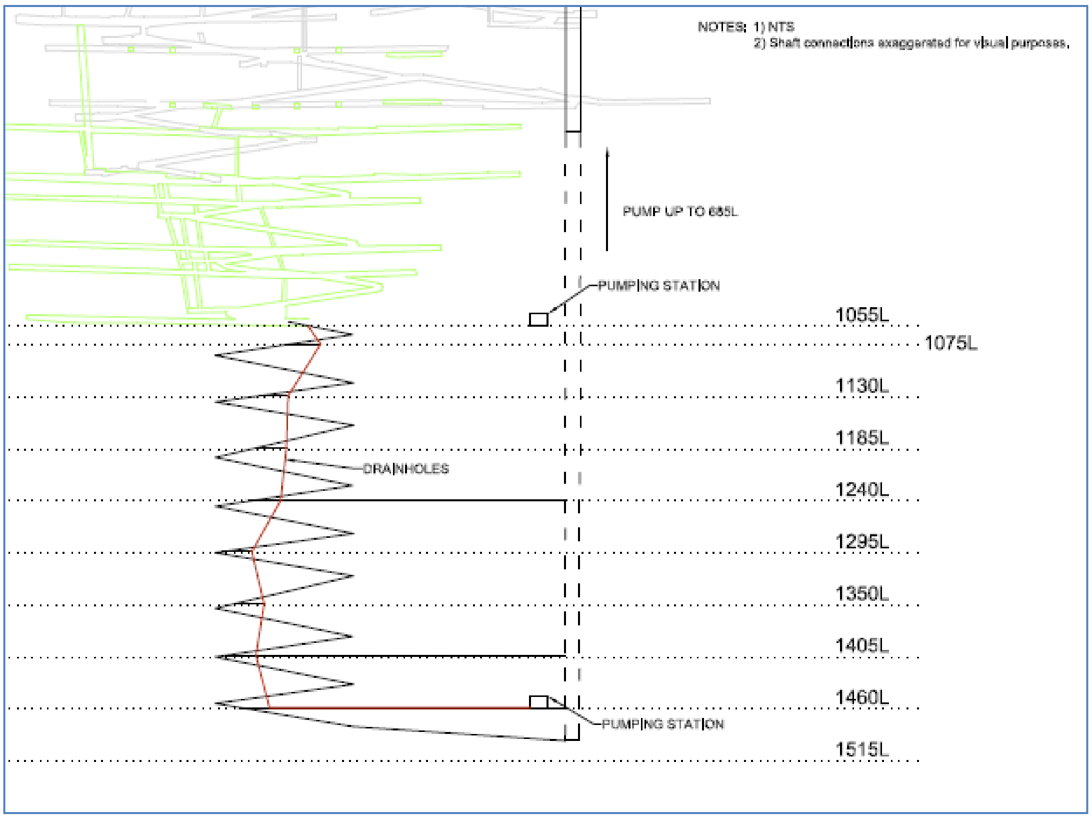


Figure 18-7: Schematic for Phase 2 Dewatering

18.26.5 Ventilation

To supply the deeper headings in Phase 2 with adequate air, a new ventilation circuit will need to be constructed. This will be comprised of new fresh air raises being developed/constructed in multiple stages, dependent on the capacity of the raiseboring equipment (contractor supplied) and rock competencies. Corresponding return air raises will also be required to handle exhaust air, in conjunction with airflows in the new ramp.

Based on the Stantec report from January 2015, the surface fans and heaters are capable of meeting the Phase 2 requirements.

Some form of Ventilation on Demand (VOD) will be assumed for Phase 2. Design of this system is outside of the scope of the PEA, but an allowance will be included, assuming that it will be some extension of VOD installed prior to Phase 2.

No egress ladder ways are necessary in the ventilation raises, as the ramp and shaft form two means of access / egress.

Booster fans will be required to direct air into the new section / areas of the mine. These fans are expected to be of the same or similar types already utilized at the site.

A schematic showing airflows and booster fan locations can be seen in Figure 18-8.

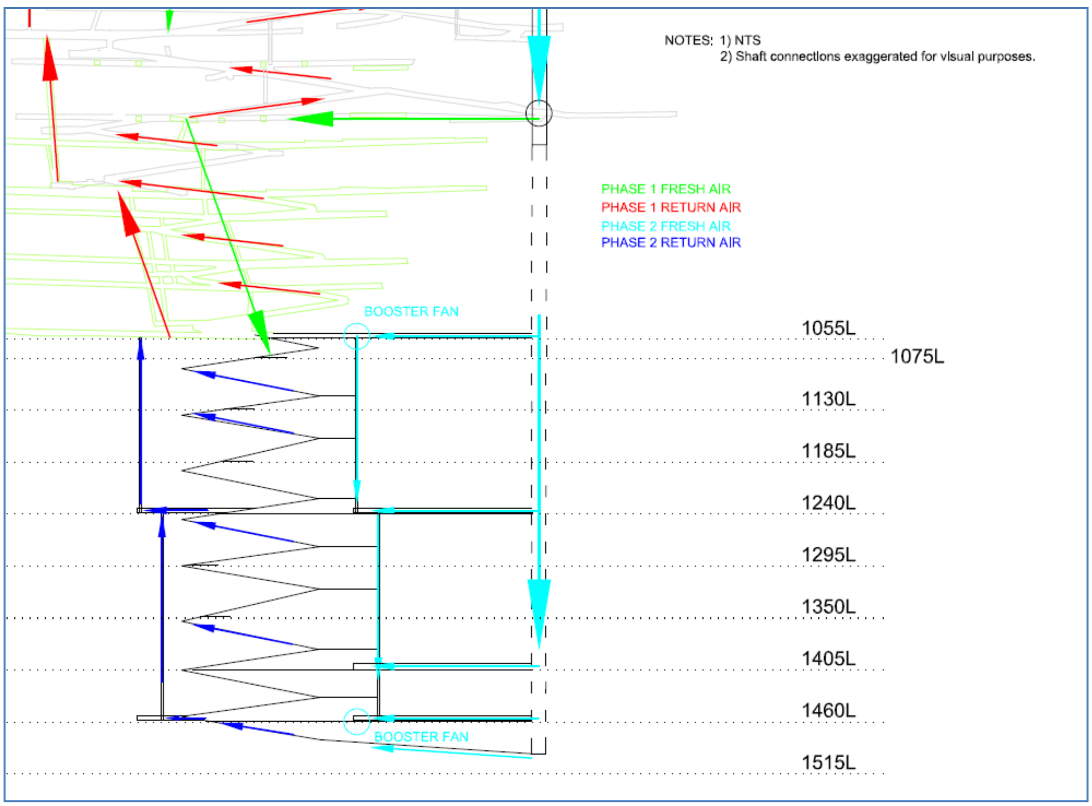


Figure 18-8: Ventilation schematic for Phase 2

18.26.6 Other Infrastructure

For the Phase 2 project, a capital allowance has been made in the estimate for extension of supporting infrastructure deeper into the mine. This includes:

- Electrical power and control system distribution.
- Compressed air distribution.
- An additional underground service bay.
- Shaft fuel line.
- Refuge stations and sanitary facilities.

An independent means of secondary egress will not be required since the ramp will serve that purpose.

18.26.7 Implementation Schedule

An indicative schedule to implement Phase 2 is indicated in Figure 18-9. The proposed schedule is contingent upon a project approval date of mid-2016.

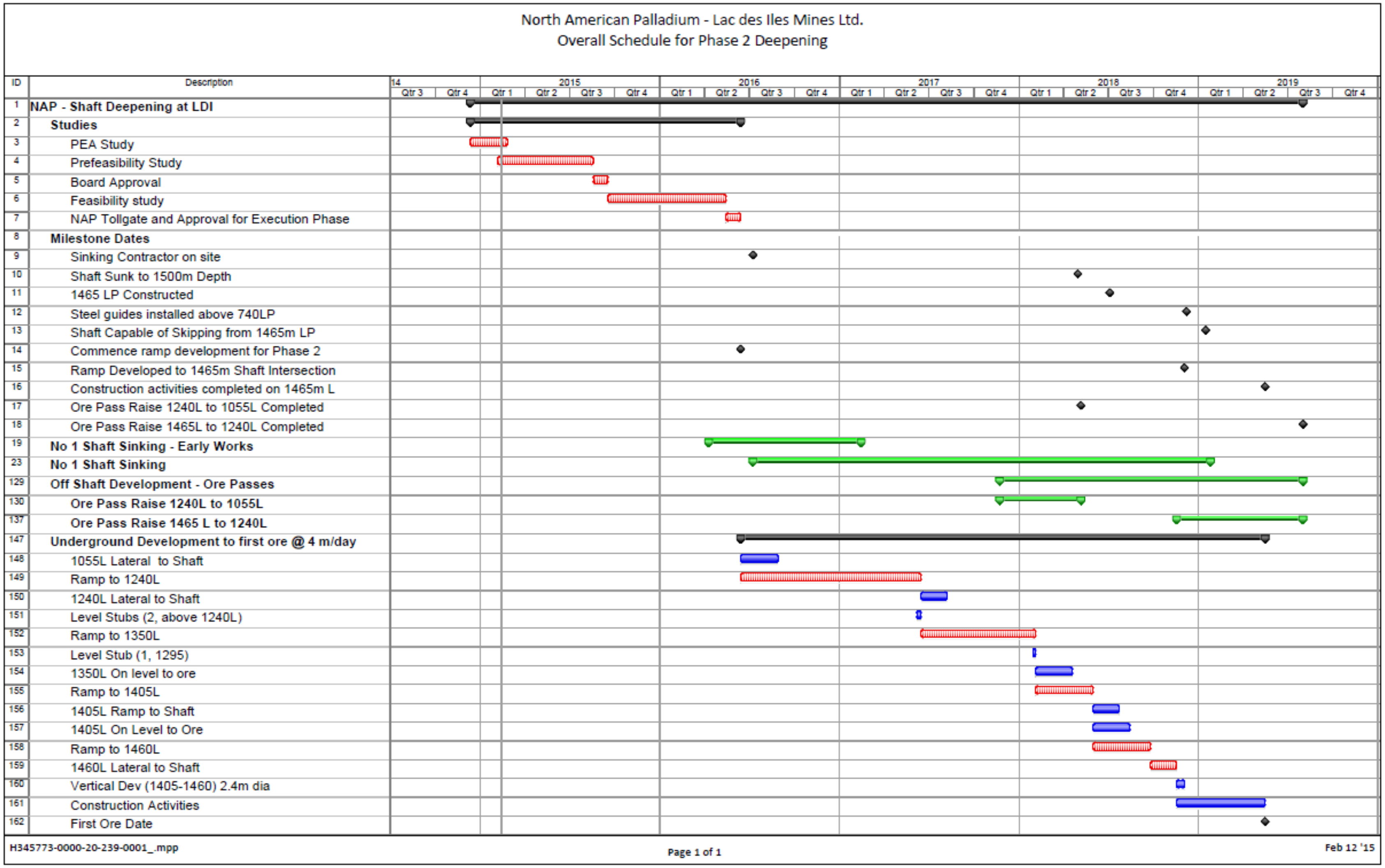


Figure 18-9: Indicative Schedule

19. Market Studies and Contracts

19.1 Concentrate Market

North American Palladium sells a copper nickel sulphide concentrate that can be smelted in copper nickel sulphide smelting-refining complexes that have capability to recover platinum group metals. The terms of the company's existing contract with such a facility were reviewed, found to be reasonable, and used in the economic analysis. It appears that there is an established market for such high PGM grade concentrates with a contract structure that is similar to the terms for comparable nickel copper sulphide concentrates containing precious metals. The net smelter return of the concentrate is linked to the metal commodity prices, smelter terms in the contract and concentrate transportation costs.

NAP currently maintains a contract with Vale for concentrate processing at their Copper Cliff, Ontario Smelter. The Vale contract expires June 30, 2015, and NAP is in discussions with a number of smelters regarding processing of its concentrate. If the Vale contract for concentrate smelting is not renewed before its expiration on June 30, 2015 or alternative options are not arranged, there could be a negative impact on company cash flows.

19.2 Metal Market Price Forecasts

NAP relies on a number of independent organizations for market analysis and metal price forecasts including Bloomberg and a number of major banks with dedicated commodities market research groups. Their research includes thorough discussions, analysis, and statistics concerning market supply (mine production and secondary supply), market demand (fabrication and investment), and other aspects of these markets. The Company also monitors the expectations of competitors in the PGM markets for their views of future prices and supply and demand trends.

The general consensus of the forecasters is that palladium prices will rise, with palladium highlighted as having better prospects over other PGM metals. Their outlook for palladium for at least the next 5 years is one of historically high prices, strong fabrication and investment demand, and constrained supply. The forecasts indicate that demand should continue to rise, driven primarily by the gasoline automotive sector which consumes a majority of global palladium production for the manufacture of catalytic converters. Investment demand as a precious metal is also expected to increase due to the favourable supply demand fundamentals while demand for other industrial uses such as electronics, dental, and jewellery is expected to be flat to slightly negative in the long term.

Palladium supply is concentrated in a few locations worldwide with Russia and South Africa representing approximately 44% and 36% respectively of mine production supply. Palladium produced in Russia and South Africa is a by-product of nickel and platinum production respectively, therefore the level of palladium production is somewhat dependant on the demand and price for those primary metals. Global supply disruptions like the labour issues that recently impacted certain South African platinum producers also impact palladium supply. If the currently known new possible mines come into production as their owners expect, this additional supply is not expected to offset the expected increase in demand.

Existing global mine production has been decreasing for the last three years and is only expected to increase modestly over the next five years.

This supply demand imbalance leads most market analysts to conclude that the palladium market will be in a deficit position for the foreseeable future even after potential new mine production is factored in and therefore, the price of palladium is expected to remain strong in the next decade.

The study base case metal price for palladium, by-products and the exchange rate of the Canadian dollar to the United States dollar is based on a preliminary analyst consensus forecast compiled by NAP with the aid of their financial advisors. The consensus, average, real metal prices and foreign exchange rates used in the economic analysis are shown in Table 19-1:

Table 19-1: Analyst Consensus Metal Pricing

Price	Unit	2016	2017	2018	2019+
Palladium	US$/oz	901	935	948	855
Platinum	US$/oz	1,440	1,543	1,600	1,611
Gold	US$/oz	1,265	1,253	1,246	1,275
Nickel	US$/lb	9.31	9.53	10.11	8.87
Copper	US$/lb	3.11	3.29	3.39	3.01
Exchange Rate	CA$/US$	1.15	1.12	1.11	1.11

Source: Average of recent analyst forecasts as of January 28, 2015

20. Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Studies

Annual monitoring for physical and chemical stability of water surrounding the Property is ongoing. The ongoing operational monitoring, reporting, and regulatory filings will be continued for years after the mine has closed.

Every three years, there is an extensive biological monitoring study that covers an area of 300 km^2 around the mine site. This includes the monitoring of fisheries, plants, soils, water, and benthic invertebrates, as well as algae quality and assessment. No negative trends have been associated with the LDI mine on the surrounding ecosystem.

20.2 Waste and Tailings Disposal, Site Monitoring and Water Management

LDI has been operating a TMF since 1990. The design of the operation is expected to facilitate closure and reclamation of the facility at the end of mine life.

Three TMFs are present on the Property: the West TMF, the East TMF and the South TMF. The TMFs are located adjacent to one another, southwest of the open pit. Tailings from the current mining are disposed of in the South and East TMFs.

Currently, all wastes created by the milling process are deposited in this facility. Dams and dykes are located around and through the East and South TMFs. Dam perimeters are inspected regularly for visual signs of wind and gully erosion, tension cracks at slope crests, seepage stains, alluvial fans, bulging of slopes, piping, sloughing of crests and other indications of instability or failure. A Dam Safety Inspection was conducted in 2014 and indicated the major dam structures were generally in good condition. The review did however identify some minor deficiencies, which are currently being actioned by NAP as part of subsequent design modifications to the TMFs. The tailings area is monitored according to industrial sewage works requirements set out by the Ontario Ministry of Environment. The West TMF has been closed upon reaching capacity.

All water within both TMFs is recycled 100% on-site and is treated before being discharged to the environment when needed. There have been no recorded permit exceedances in treated effluent being discharged to the environment.

Tailings are not leachate toxic waste as defined by Ontario Ministry of Environment Regulation 347. Consequently, there are no long-term risks associated with metal leaching at the site.

Wastes and materials generated during closure activities at the facility are managed and disposed of in accordance with all applicable municipal, provincial, and federal requirements. A landfill site is located on the east side of the West TMF. Solid domestic and industrial waste products such as wood, burn piles, and building demolitions are disposed of at the site.

The waste and materials are inspected and monitored in accordance with the Certificate of Approval (CofA) issued through the Ministry of Environment for a waste disposal site.

Currently, waste rock is stockpiled on surface and is being removed as required for backfill underground. It is not expected that any new material will be added to the existing waste rock dumps as a result of underground mining. Potential re-activation of the open pit operation will result in additional waste rock being generated and will require revisions to the current permits and closure plan.

The mined material from Lac des Iles Mine typically contains less than 3% sulphide by volume. Approximately 80% of sulphide-bearing minerals are typically removed during the milling process. This suggests very little potential for acid rock drainage (ARD). Samples were collected in several locations throughout the TMF and were representative of the age range and texture of the rock. Analyses of these samples indicated that the leachate is non-toxic as defined in Ontario Regulation 347. Monitoring of TMF effluent and downstream water quality also show little evidence of ARD within tailings.

20.3 Permitting

NAP asserts that its operations and facilities comply in all material respects with current legislation, and it holds all necessary approvals and licenses for its operations at the mine and for all planned expansion projects. The site remains current with permitting and licensing requirements Table 20-1. The QP is satisfied that NAP has all permits and approvals to continue its mining operations at Lac des Iles.

Table 20-1: Permits and Licences

Approval	Reason for Approval	Expiration Date
PTTW 0020-97NJES, LDIM Lake	Approved withdraw of 24,000 L/min or 35,000,000 L/d	Expires May 23, 2020
PTTW-U/G #6467-96MN59	Approved withdraw of 18,180 L/min or 26,179,200 L/d	April 18, 2023
PTTW Open Pit #8583-8WPL3N	Approved withdraw of 908 L/min 1,307,520 L/d	expires July 30, 2018
LDIM Closure Plan	Approved the Project and is updated when changes to the mine site occur	Does not expire
Generator Registration Number	Allows the disposal of subject wastes, as per Regulation 501/01	Must be renewed using Ministry of Environment's Hazardous Waste Information Network website on an annual basis
CofA Air 9997-6M4l3B	Air	Does not expire
CofA Sewage 2018-7X4HML	Industrial sewage works	Does not expire. Being amended with MOE
CofA Building 3-1404-98-006	Old administration building	Does not expire
CofA Sewage 8678-4QGGY5	Assay laboratory and mill	Does not expire
CofA Waste Management A900369	Provisional approval waste management system	Does not expire
CofA Waste Disposal Site A770072	Provisional approval waste disposal site	Does not expire

20.4 Social and Community Requirements

The Black Spruce Forest, north of the city of Thunder Bay, is home to the LDI Mine. The site sits in an area of interest to five aboriginal groups which have asserted treaty rights and/or traditional usage, in accordance with federal government criteria.

The mine regularly interacts with these groups in a number of ways ranging from face-to face meetings, information sessions, presentations focused on specific activities (for example amendments to the current closure plan), discussions around potential business opportunities, and the regular dissemination of information relating to employment opportunities at the site.

In 2011, LDI expanded its capability to interact with Communities of Interest (COI) through the addition of a Community Relations Manager. The goal is to ensure all required engagements are addressed and that communities of interest have a single point of contact to ensure open and frequent communication, and that any concerns be addressed in a timely fashion. The communities that have been consulted include:

- Whitesand First Nation.
- Gull Bay First Nation (Kiashke Zaaging Anishinaabek).
- Fort William First Nation.
- Metis Nation of Ontario.
- Red Sky Metis Independent Nation.

LDIM is an active participant in the Local Citizens Committee (LCC) for the Black Spruce Forest (formed in 1995) which meets regularly throughout the year. The LCC is made up of representatives of the many user groups associated with the Black Spruce Forest, which includes aboriginal groups.

No Impact Benefit Assessment (IBA) will be required for the proposed mine expansion.

20.5 Closure Plan

LDIM is responsible for all costs of closure and reclamation at the site. LDIM, in conjunction with the Ontario MNDM, has established a trust fund pursuant to LDI’s mine closure plan for eventual cleanup and restoration of these sites at the end of mine life. LDIM will be submitting a Closure Plan Amendment (CPA) this year for the TMF expansion.

The mine closure plan requires an amount of $14,054,660 to be on deposit. As of the effective date of this report, LDIM has a credit $27,461 on this account.

21. Capital and Operating Costs

21.1 Capital Cost Estimation

This section describes the capital requirements for each of the mining areas. Each mining area, except for Phase 2, includes a capital allowance of $1 per tonne of production added to sustaining capital for the Tailings Management Facility (TMF).

Cost estimates for mobile equipment are based on recent acquisitions at LDI and validated as reasonable by the QP.

Remaining capital cost items were estimated based on current LDI budget information where available or from recent Hatch project experience.

Phase 1 – This includes the currently planned underground mining in the Offset and Roby zones and extraction of the surface RGO and Tailings stockpile. The capital estimate for Phase 1 provides for sustaining underground capital development and equipment replacement and rebuilds. Also included is a major expansion of the TMF and treatment pond.

Open Pit - This portion of the capital estimate includes pre-stripping capital waste and equipment. The relocation of surface infrastructure and portions of the ramp that will be affected by the pit expansion is also required.

Roby FW - No capital development is required for this zone because the existing development that was used for the primary mining will be suitable. The purchase of mobile equipment is allowed for and it is expected to last the duration of mining.

Offset FW - Capital development is not required for this zone because the existing development that was used for the primary mining will be suitable for access. Because this zone is an extension of Phase 1, no major equipment purchase is required, only sustaining replacements and rebuilds are allocated.

In order to perform an economic analysis of potential expansion plans, it is necessary to include a capital expenditure forecast for the current production plan (and associated reserves) in the evaluation to properly assess the potential economics of the expansion. This is summarized in Table 21-1.

The Base Case adds an open pit and some additional footwall resources to the Current Mine Plan. The capital expenditure forecast for the Base Case is summarized in , Table 21-2.

Minor discrepancies in the tables may occur due to rounding.

Table 21-1: Capital Expenditures for the Current Mine Plan

CAD$ x 1 Million	SUM	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Phase 1 to 1055L																	
Sustaining Mobile Equipment	16.5	3.7	3.7	3.1	3.0	3.0											
Sustaining Capital Projects	15.5	4.1	3.7	3.7	2.0	2.0											
UG Capital Development	20.5	9.3	7.2	2.0	2.0												
Tailings Management Facility Upgrade	9.0	9.0															
Sustaining TMF Expansion	19.6	4.6	4.0	4.0	4.0	2.5	0.5										
Phase 1 Mining	**81.1**	**30.7**	**18.6**	**12.8**	**11.0**	**7.5**	**0.5**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Roby Footwall																	
Mobile Equipment	7.5	7.5															
Sustaining TMF Expansion	2.7	0.0	0.5	0.5	0.5	0.5	0.5										
Roby FW	**10.2**	**7.5**	**0.5**	**0.5**	**0.5**	**0.5**	**0.5**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Total Current Plan	**91.3**	**38.1**	**19.1**	**13.4**	**11.6**	**8.1**	**1.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Project Capital	7.5	7.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Capital	83.8	30.6	19.1	13.4	11.6	8.1	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

Table 21-2: Capital Expenditures for the Base Case Scenario

CAD$ x 1 Million	SUM	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Phase 1 to 1055L																	
Sustaining Mobile Equipment	16.5	3.7	3.7	3.1	3.0	3.0											
Sustaining Capital Projects	15.5	4.1	3.7	3.7	2.0	2.0											
UG Capital Development	20.5	9.3	7.2	2.0	2.0												
Tailings Management Facility Upgrade	9.0	9.0															
Sustaining TMF Expansion	20.3	4.6	4.0	4.0	4.0	2.5	1.2										
Phase 1 Mining	**81.8**	**30.7**	**18.6**	**12.8**	**11.0**	**7.5**	**1.2**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Roby Footwall																	
Mobile Equipment	7.5	7.5															
Sustaining Capital	3.0						1.5	1.5									
Sustaining TMF Expansion	2.9		0.5	0.5	0.5	0.5	0.5	0.2									
Roby FW	**13.4**	**7.5**	**0.5**	**0.5**	**0.5**	**0.5**	**2.0**	**1.7**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Studies	**1.0**	**1.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Open Pit																	
Prestripping	84.5				13.5	20.1	20.5	19.3	11.0								
Sustaining Capital	12.0						1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5			
TMF Expansion - Open Pit	34.7					1.5	2.7	3.1	3.3	3.3	4.6	4.6	4.6	4.6	1.5		
Equipment	26.0			8.5	8.5	0.2	0.2					4.3	4.3				
Infrastructure Relocation	19.7					0.0	2.6	0.0	3.6	10.9	2.6						
Total Open Pit	**176.8**	**0.0**	**0.0**	**8.5**	**22.0**	**21.8**	**27.5**	**23.9**	**19.4**	**15.7**	**8.6**	**10.4**	**10.4**	**6.1**	**1.5**	**0.0**	**0.0**
Offset Footwall	**0.0**																
Mobile Equipment	7.5						1.5	1.5	1.5	1.5							
Sustaining Capital	4.5								1.5	1.5							
Sustaining TMF Expansion	4.0	0.0	0.0	0.0	0.0		0.1	1.3	1.3	1.2							
Subtotal Offset FW	**16.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**1.6**	**2.8**	**4.3**	**4.2**			**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Current Plan + Open Pit (Base Case)	**285.0**	**39.2**	**19.1**	**21.8**	**33.6**	**29.9**	**32.4**	**28.4**	**23.7**	**19.9**	**8.7**	**10.4**	**10.4**	**6.1**	**1.5**	**0.0**	**0.0**
Project Capital	58.6	7.5	0.0	8.5	22.0	20.3	20.7	19.3	11.0	0.0	0.0	4.3	4.3	0.0	0.0	0.0	0.0
Sustaining Capital	226.4	31.6	19.2	13.4	11.6	9.5	11.6	9.1	12.7	19.9	8.7	6.1	6.1	6.1	1.5	0.0	0.0

The Offset Zone below 1065L is considered Phase 2. The scope of work to access Phase 2 includes the deepening of No. 1 Shaft, the installation of an ore handling system on 1465L, the extension of infrastructure including power, ventilation and dewatering, and the development of ramps, drifts and raises. The cost of a Paste Backfill Plant and associated distribution system is also included. Since Phase 2 will be mined using mill tailings for backfill, there is a reduction in the capital allowance for the TMF for Phase 2. These capital expenditures are summarized in Table 21-3, assuming a project approval date of mid-2016.

The cost of development was determined from the actual cost at LDI and validated by the QP. The shaft deepening cost has been determined by Cementation Canada and accepted as reasonable by QP. Other infrastructure for the Phase 2 project has been derived from similar, recent Hatch estimates.

Table 21-3: Capital Expenditures for the Phase 2 Expansion Plan

CAD$ x 1 Million	SUM	2016	2017	2018	2019	2020	2021	2022	2023	2024
Phase 2 - Project Capital										
Shaft Deepening Scope incl. LP	61.2	15.1	25.0	21.1	0.0					
Muck Handling	14.5	0.0	3.0	5.2	6.3					
Ramps & Drifts	23.6	4.6	9.2	7.0	1.8					
Ventilation	8.2	0.0	2.5	3.6	2.1					
Infrastructure	7.0	0.0	1.5	4.5	1.0					
Backfill Plant & Distribution System	30.0	0.0	8.9	21.1	0.0					
Mobile Equipment	8.0	5.0	0.0	0.0	3.0					
U/G Mobile Shop	10.0	0.0	0.0	5.0	5.0					
Electrical & Automation	21.0	2.0	8.0	8.0	3.0					
Contingency @ 20%	36.7	5.3	11.6	15.1	4.4	0.2				
EPCM @10%	22.0	3.2	7.0	9.1	2.7	0.1				
Phase 2 - Project Capital	**242.2**	**35.2**	**76.7**	**99.7**	**29.3**	**1.3**	**0.0**	**0.0**	**0.0**	**0.0**
Phase 2 - Sustaining Capital										
Capital Development	17.8				4.6	4.0	4.0	3.5	1.7	0.0
Mobile Equipment	12.0				0.0	1.5	3.0	3.0	3.0	1.5
TMF Expansion - Phase 2	5.9				0.5	0.6	0.6	1.4	1.4	1.4
Electrical & Control	4.0					1.0	1.0	1.0	1.0	
Contingency @ 20%	7.9				1.0	1.4	1.7	1.8	1.4	0.6
Phase 2 - Sustaining Capital	**47.6**				**6.1**	**8.5**	**10.4**	**10.7**	**8.5**	**3.5**
Total Phase 2	**289.8**	**35.2**	**76.7**	**99.7**	**35.4**	**9.8**	**10.4**	**10.7**	**8.5**	**3.5**

21.2 Operating Cost Estimation

The operating costs used for the Preliminary Economic Assessment were derived using the current 2015 LDI Budget as a basis. A cost model was constructed that categorized costs by area and into their fixed and variable components so that adjustments could be made based on forecasted changes to operating scenarios. These costs were used to form the basis for cut-off grade calculations described in Section 16.

The shaft deepening proposed for Phase 2 will alter the underground operating cost. For comparison purposes, two cost scenarios were generated.

Table 21-4: Operating Cost Scenarios

	Base Case Scenario	Phase 2 Expansion Scenario
Surface and ramp production	7,500 tpd	6,000 tpd
U/G production	5,000 tpd	6,500 tpd
Backfill	Unconsolidated waste	Paste
Phase 1 ore	All trucked to 645 level	Some hoisted from 1465 – no trucking required
Development	Required for both topsill and for bottomsill for each mining horizon	One sill required for each mining horizon
Phase 2 ore	N/A	No trucking required
Roby FW	Assumes no additional development – trucking to surface portal	Assumes no additional development – trucking to surface portal
Offset FW	Reduced development, trucked to 645 level, operating towards end of mine life at a reduced rate (3500 tpd due to trucking)	Reduced development, no trucking, operating towards end of mine life at a reduced rate (4000 tpd)

The open pit mining cost in both scenarios was determined from benchmark data for similar size operations.

Table 21-5 displays the operating cost summary for all zones, for each scenario.

Table 21-5: Operating Cost Summary

Scenario	Cost Category	Tailings / RGO	Open Pit	Offset Phase 1	Offset Phase 2	Roby FW	Offset FW
No Shaft Deepening: 7,500 tpd from Surface, 5,000 tpd from U/G	G&A ($/t)	$5.22	$5.22	$5.22	N/A	$5.22	$5.22
	Milling ($/t)	$9.87	$9.87	$9.87		$9.87	$9.87
	Surface Handling ($/t)	$3.25	$1.92	$2.80		$2.80	$2.73
	Mining ($/t)	$0.00	$3.00	$37.50		$29.46	$37.61
	Total (per tonne Milled)	**$18.34**	**$20.01**	**$55.39**		**$47.35**	**$55.43**
Shaft Deepening: 6,000 tpd from Surface, 6,500 tpd from U/G	G&A ($/t)	$5.22	$5.22	N/A	$5.22	$5.22	$5.22
	Milling ($/t)	$9.87	$9.87		$9.87	$9.87	$9.87
	Surface Handling ($/t)	$3.00	$1.92		$3.03	$3.03	$2.65
	Mining ($/t)	$0.00	$3.00		$33.99	$28.21	$33.82
	Total (per tonne Milled)	**$18.09**	**$20.01**		**$52.11**	**$46.33**	**$51.56**

Table 21-6 summarized the operating cost data used for the Base Case

Table 21-6: Operating Cost Summary for the Base Case Scenario

Cost per Tonne Milled					
Base Case	Tailings / RGO	Open Pit	Offset Phase 1	Roby FW	Offset FW
	$18.34	$20.01	$55.39	$47.35	$55.43

22. Economic Analysis

22.1 Introduction

To evaluate the benefits of the required capital investment for the proposed LDI mine expansion plans, the economics were evaluated in an Excel based real basis after-tax discounted cash flow (DCF) model in which the production, revenues, operating costs, capital costs and taxes were considered.

The economics for the Open Pit and Phase 2 expansion plans were evaluated on an incremental cash flow basis whereby positive cash flows are new cash inflows to the company and negative cash flows are new cash outlays relative to the Current Mine Plan and Base Case respectively. The scenarios were structured to incrementally assess the overall economics with or without the projects to calculate the project economics.

The financial model was set up as a single model with multiple sets of input assumptions that can be varied by option. Three different mine plans were considered in the model in order to evaluate the incremental project economics:

- Current Mine Plan which is based on the previously stated mineral reserves, augmented with the RGO stockpile and other mineral resources above the 1065L.
- Base Case scenario which is the Current Mine Plan + Open Pit Expansion.
- Base Case + Phase 2 Expansion to evaluate Phase 2 Expansion on an incremental basis.

The key model assumptions and financial results, project returns and cash flows are presented herein.

This PEA is at a scoping level and is preliminary in nature. One of the expansion scenarios includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. There are no Inferred mineral resources contained within the Current Mine Plan.

For the purposes of this analysis, Inferred resources are included in Phase 2 expansion only. No Inferred resources are included in the Base Case scenario.

All economic assessments are calculated at the LDI mine level, and therefore, do not include certain costs including corporate office, interest, financing and exploration expenses.

22.2 Key Assumptions

The fundamental assumptions used in development of the financial model are shown in Table 22-1.

Table 22-1: Key Financial Model Assumptions

Parameter	Assumption	Description
Units	Metric	The model has been constructed using metric tonnes.
Valuation Date	1-Jan-16	The analysis is based on a valuation date of January 1, 2016. Costs prior to that date are considered sunk and are not considered in the analysis.
Discount Rate	5%	The financial evaluation has considered 5% as the discount rate for precious metals projects in a low risk setting. Start of period discounting has been used in the model. Sensitivity is shown for other discount rates.
Currency	CAD	The model has been constructed using Canadian Dollars. Metal revenue has been converted to Canadian dollars for the analysis.
Inflation	Real basis	All projected revenue and costs are assumed to be in 2015 real terms over the DCF time frame, with no inflation applied.
Capital Structure	Unlevered	The calculated financial results assume a project financed entirely on equity. No Interest Payments have been assumed.
Royalty	5%	5% NSR royalty has been included.
Income Tax	26.5%	Income taxes have been included in the model with preliminary NAP accounting guidance including federal and provincial tax.
Ontario Mining Tax	10%	Ontario mining taxes have been included in the model with preliminary NAP accounting guidance.
Accounts Receivable	120 days	120 days of total revenue has been assumed for accounts receivable for working capital funding requirements.
Accounts Payable	30 Days	30 Days of total OPEX has been assumed for accounts payable for working capital funding requirements.
Inventory & Consumables	30 Days	30 Days of total OPEX has been assumed for inventories and consumables for working capital funding requirements.
Concentrate	7 Days	7 Days of total OPEX has been assumed for concentrate for working capital funding requirements.
Stockpiles	4 Days	4 Days of total OPEX has been assumed for stockpiles for working capital funding requirements.
Closure and Reclamation Costs	$15.4 million	$15.4 million has been included for closure and reclamation costs based on NAP's current forecast.
Long Term Consensus Prices		
Pd	$855/oz USD	Received from NAP on February 13, 2015 - Consensus Price forecasts - source: analysts per details in Appendix G via NAP.
Pt	$1611/oz USD	
Au	$1275/oz USD	
Ni	$8.87/lb USD	
Cu	$3.01/lb USD	
Exchange Rate	1.11 CAD per USD	Metal Prices have been converted to CAD for the evaluation. All costs are assumed to be incurred in CAD.
Treatment and Refining Charges		
Treatment Charge	$350/t USD	Treatment charges per tonne of concentrate as per the current agreement.
Pd	$650/kg USD	Refining charges on payable metal as per the current agreement.
Pt	$650/kg USD	Refining charges on payable metal as per the current agreement.

Parameter	Assumption	Description
Au	$350/kg USD	Refining charges on payable metal as per the current agreement.
Ni	$1.75/kg USD	Refining charges on payable metal as per the current agreement.
Cu	$1.20/kg USD	Refining charges on payable metal as per the current agreement.
Ni Price Participation	8% over $13/kg	The smelter will receive 8% of the Ni revenue above USD$13/kg.
Concentrate Freight	$95/t USD	

22.3 Pricing and Exchange Rate

Given the lower grade resource at the LDI site and the potential to convert this material into a viable mining plan with a life of 15 years or greater, NAP felt it was important to use generally accepted long-term price forecasts. The company compiled, with the aid of its financial partners, a commodity price forecast from an extensive list of financial analysts. These prices are summarized in Table 22-2. The sensitivity to palladium prices from USD$700/oz to USD$1500/oz is shown in the sensitivity analysis Section 0.

Table 22-2: Consensus Commodity Forecast Price and Foreign Exchange Assumptions

Price	Unit	2016	2017	2018	2019+
Palladium	USD$/oz	901	935	948	855
Platinum	USD$/oz	1,440	1,543	1,600	1,611
Gold	USD$/oz	1,265	1,253	1,246	1,275
Nickel	USD$/lb	9.31	9.53	10.11	8.87
Copper	USD$/lb	3.11	3.29	3.39	3.01
Exchange Rate	CAD per USD	1.15	1.12	1.11	1.11

Source: Average of recent analyst forecasts as of January 28, 2015.

The model has been developed in Canadian Dollars based on the consensus forecast exchange rates. Metal prices have been converted to CAD for the evaluation. For the purpose of the sensitivity analysis, costs have been assumed to be 100% incurred in Canadian Dollars.

22.4 Financial Results

As an operating facility considering an expansion, it is necessary to include the Current Mine Plan (and associated reserves) in the evaluation to properly assess the potential economics of the expansion, as this represents the alternative to developing the new resources (i.e., the opportunity cost of the investment decision).

The Current Mine Plan is based on the previously stated mineral reserves, augmented with the RGO stockpile and other mineral resources above the 1065L, but do not include Inferred mineral resources. Using current economic assumptions, the financial summary of the Current Mine Plan is shown in Table 22-3.

Table 22-3: Financial Results for Current Mine Plan

LOM Totals	Units	Current Mine Plan
Production	kt	22,329
Pd (g/t)	g/t	2.06
Pt (g/t)	g/t	0.18
Au (g/t)	g/t	0.16
Ni (%)	%	0.064%
Cu (%)	%	0.057%
Pd contained	oz	1,480,242
Pt Contained	oz	132,325
Au Contained	oz	111,333
Ni Contained	lb	31,740,333
Cu Contained	lb	27,905,328
Recovery		
Pd	%	80.9%
Pt	%	74.1%
Au	%	75.6%
Ni	%	30.2%
Cu	%	86.4%
Payable Metal		
Pd	oz	1,084,414
Pt	oz	88,279
Au	oz	74,927
Ni	lb	8,633,890
Cu	lb	21,695,330
Total Revenue	CAD $M	1,517
TCs	CAD $M	(56)
RCs	CAD $M	(51)
Transportation	CAD $M	(14)
Royalties	CAD $M	(70)
Net Smelter Return	CAD $M	1,326
Total OPEX	CAD $M	(776)
EBITDA	CAD $M	550
Project CAPEX	CAD $M	(8)
Sustaining CAPEX	CAD $M	(84)
Change in Working Capital	CAD $M	70
Closure Costs	CAD $M	(15)
Pre Tax Cash Flow	CAD $M	513
Taxes	CAD $M	(21)
After-tax Cash Flow	CAD $M	493
After-tax NPV @5%	CAD $M	435

To evaluate the potential economic benefits of the required capital investment for each scenario, the Open Pit and Phase 2 expansion scenarios were evaluated independently and the scenarios were structured to incrementally assess the overall economics with or without the expansion projects. Table 22-4 provides a summary of the financial results for the expansion projects.

The economic indicators of the expansion projects are based on scoping level studies that are preliminary in nature. The Phase 2 Expansion scenario includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the economic indicators of the PEA will be realized. The Current Mine Plan and the Base Case do not include any Inferred mineral resources.

Table 22-4: Financial Results for Open Pit and Phase 2 Expansion Scenarios

LOM Totals	Units	Open Pit Expansion	Phase 2 Expansion
Production	kt	38,514	10,857
Pd (g/t)	g/t	1.27	3.27
Pt (g/t)	g/t	0.16	0.23
Au (g/t)	g/t	0.11	0.22
Ni (%)	%	0.065%	0.093%
Cu (%)	%	0.054%	0.075%
Pd contained	oz	1,573,415	1,139,766
Pt Contained	oz	201,556	79,283
Au Contained	oz	137,327	75,739
Ni Contained	lb	55,559,813	22,297,592
Cu Contained	lb	45,692,181	17,953,575
Recovery			
Pd	%	75.9%	84.5%
Pt	%	72.4%	76.6%
Au	%	70.1%	80.0%
Ni	%	30.6%	38.9%
Cu	%	85.7%	89.8%
Payable Metal			
Pd	oz	1,080,105	871,476
Pt	oz	131,424	54,647
Au	oz	85,660	53,919
Ni	lb	15,303,096	7,814,598
Cu	lb	35,223,002	14,517,587
Total Revenue	CAD $M	1,650	1,127
TCs	CAD $M	(55)	(45)
RCs	CAD $M	(67)	(39)
Transportation	CAD $M	(14)	(11)
Royalties	CAD $M	(76)	(52)
Net Smelter Return	CAD $M	1,438	980
Total OPEX	CAD $M	(961)	(538)
EBITDA	CAD $M	477	442
Project CAPEX	CAD $M	(51)	(242)
Sustaining CAPEX	CAD $M	(143)	(50)
Change in Working Capital	CAD $M	-	-
Closure Costs	CAD $M	-	-
Pre Tax Cash Flow	CAD $M	283	151
Taxes	CAD $M	(35)	(33)
After-tax Cash Flow	CAD $M	248	117
After-tax NPV @5%	CAD $M	138	22
After-tax IRR	%	31%	7%

The Open Pit Expansion project with an estimated project capital cost of $51M, relative to the Current Mine Plan, generates an after-tax $NPV_{5\%}$ of $138M and an after-tax IRR of 31%.

Considering the current resource, the Phase 2 Expansion project, with an estimated project capital cost of $242M, generates a potential after-tax $NPV_{5\%}$ of $22M and an after-tax IRR of 7%.

As a result of the economic indicators of the Open Pit Expansion, the Base Case was selected by NAP by adding the Open Pit Expansion to the Current Mine Plan. It was necessary to consider the Current Mine Plan (and associated reserves) in the evaluation of potential economics for the expansion projects to capture the opportunity cost of the investments. The financial results of the Base Case are shown in Table 22-5.

Table 22-5: Financial Results for the Base Case Scenario

LOM Totals	Units	Base Case Scenario
Production	Kt	60,843
Pd (g/t)	g/t	1.56
Pt (g/t)	g/t	0.17
Au (g/t)	g/t	0.13
Ni (%)	%	0.065%
Cu (%)	%	0.055%
Pd contained	oz	3,053,657
Pt Contained	oz	333,881
Au Contained	oz	248,659
Ni Contained	lb	87,300,146
Cu Contained	lb	73,597,509
Recovery		
Pd	%	78.3%
Pt	%	73.1%
Au	%	72.6%
Ni	%	30.5%
Cu	%	85.9%
Payable Metal		
Pd	oz	2,164,520
Pt	oz	219,703
Au	oz	160,587
Ni	lb	23,936,986
Cu	lb	56,918,331
Total Revenue	CAD $M	3,166
TCs	CAD $M	(112)
RCs	CAD $M	(117)
Transportation	CAD $M	(27)
Royalties	CAD $M	(146)
Net Smelter Return	CAD $M	2,765
Total OPEX	CAD $M	(1,738)
EBITDA	CAD $M	1,027
Project CAPEX	CAD $M	(59)
Sustaining CAPEX	CAD $M	(226)
Change in Working Capital	CAD $M	70
Closure Costs	CAD $M	(15)
Pre Tax Cash Flow	CAD $M	797
Taxes	CAD $M	(56)
After-tax Cash Flow	CAD $M	741
After-tax NPV @5%	CAD $M	573

The Base Case (Current Mine Plan + Open Pit Expansion) generates an after-tax $NPV_{5\%}$ of $573M and extends the mine life from 2021 to 2029.

22.5 Project Cash Flows

Based on estimates of revenue, operating costs and capital spending schedule, the incremental after-tax project cumulative cash flows and cumulative discounted cash flows using a 5% discount rate for the Open Pit and Phase 2 expansion projects are illustrated in Figure 22-1.

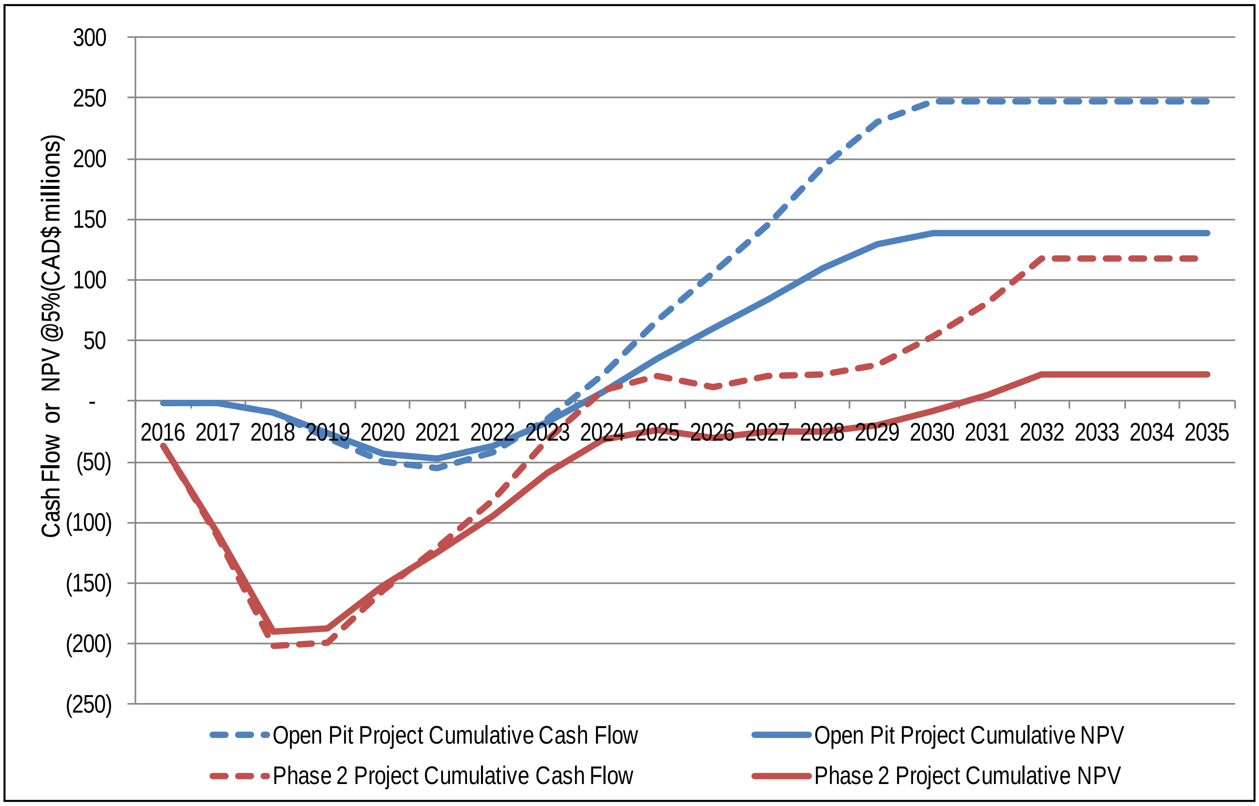


Figure 22-1: Incremental Project Cumulative Cash Flows

The buildup of the project cash flows for the Base Case and the Phase 2 Expansion are detailed in the following sections.

22.5.1 Base Case Scenario (Current Mine Plan + Open Pit Expansion)

Figure 22-2 illustrates the cash flow based on the production schedule, pricing, capital cost, operating cost and tax inputs. The cumulative cash flows and cumulative discounted cash flows using a 5% discount rate are also shown:

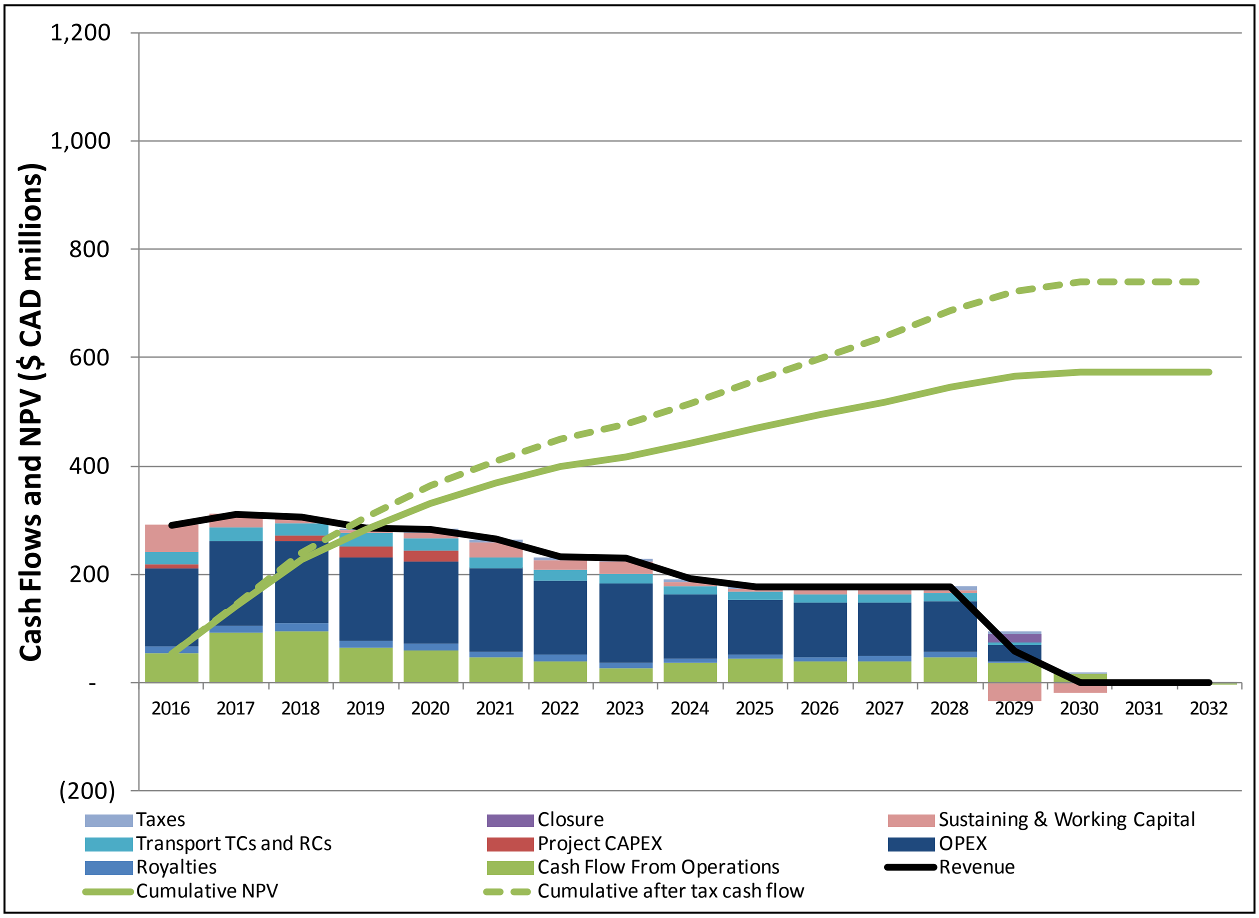


Figure 22-2: Base Case Scenario Cash Flow Chart

The summary of the cash flows for the Base Case is provided in Table 22-6. The Open Pit Expansion incremental cash flow is provided in Table 22-7.

Table 22-6: Base Case Scenario Cash Flow Summary

Area	Unit	Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Pd Price	$US/oz		901	901	935	948	855	855	855	855	855	855	855	855	855	855	855	855	855
Exchange Rate	CAD per USD		1.15	1.15	1.12	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11
Tonnes mined & milled	kt	**60,843**	4,563	4,563	4,563	4,563	4,563	4,563	4,563	4,563	4,563	4,563	4,563	4,563	4,563	1,530	-	-	-
Pd (g/t)	g/t	**1.56**	1.88	2.01	1.96	1.99	1.95	1.80	1.54	1.51	1.24	1.14	1.14	1.14	1.14	1.14	-	-	-
Pt (g/t)	g/t	**0.17**	0.18	0.18	0.18	0.18	0.19	0.19	0.18	0.18	0.16	0.16	0.16	0.16	0.16	0.16	-	-	-
Au (g/t)	g/t	**0.13**	0.15	0.15	0.15	0.15	0.15	0.14	0.12	0.12	0.11	0.10	0.10	0.10	0.10	0.10	-	-	-
Ni (%)	%	**0.07%**	0.07%	0.06%	0.06%	0.06%	0.07%	0.07%	0.07%	0.07%	0.07%	0.06%	0.06%	0.06%	0.06%	0.06%	-	-	-
Cu (%)	%	**0.05%**	0.06%	0.06%	0.05%	0.05%	0.06%	0.06%	0.06%	0.06%	0.05%	0.05%	0.05%	0.05%	0.05%	0.05%	-	-	-
Recovered Pd	koz	**2,165**	199	215	209	213	208	190	160	156	124	113	113	113	113	38	-	-	-
Recovered Pt	koz	**220**	17	18	17	17	18	18	17	17	15	15	15	15	15	5	-	-	-
Recovered Au	koz	**161**	15	15	15	14	15	13	12	12	10	9	9	9	9	3	-	-	-
Recovered Ni	kt	**11**	0.8	0.8	0.8	0.8	0.8	0.8	0.9	0.9	0.8	0.8	0.8	0.8	0.8	0.3	-	-	-
Recovered Cu	kt	**26**	2.0	2.0	1.9	1.9	2.1	2.1	2.0	2.0	1.9	1.8	1.8	1.8	1.8	0.6	-	-	-
Net Smelter Return	**C$M**	**2,765**	**255**	**273**	**268**	**248**	**248**	**231**	**203**	**200**	**167**	**155**	**155**	**155**	**155**	**52**	-	-	-
Mining OPEX	C$M	**(675)**	(61)	(73)	(70)	(70)	(70)	(73)	(60)	(67)	(40)	(25)	(23)	(22)	(18)	(5)	-	-	-
Surface Handling OPEX	C$M	**(145)**	(14)	(14)	(14)	(14)	(12)	(10)	(10)	(10)	(9)	(9)	(9)	(9)	(9)	(3)	-	-	-
Milling OPEX	C$M	**(600)**	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(15)	-	-	-
G&A OPEX	C$M	**(318)**	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(8)	-	-	-
Total OPEX	**C$M**	**(1,738)**	**(144)**	**(156)**	**(153)**	**(153)**	**(151)**	**(152)**	**(138)**	**(145)**	**(118)**	**(102)**	**(101)**	**(99)**	**(95)**	**(31)**	-	-	-
EBITDA	C$M	**1,027**	**111**	**117**	**116**	**96**	**97**	**79**	**64**	**54**	**49**	**53**	**54**	**56**	**60**	**21**	-	-	-
Project CAPEX	C$M	**(59)**	(8)	-	(9)	(22)	(20)	(0)	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	C$M	**(226)**	(32)	(19)	(13)	(12)	(10)	(32)	(28)	(24)	(20)	(9)	(10)	(10)	(6)	(2)	-	-	-
Change in Working Capital	C$M	**70**	(18)	(7)	2	7	0	6	10	1	12	4	0	0	0	36	18	-	-
Closure Costs	C$M	**(15)**	-	-	-	-	-	-	-	-	-	-	-	-	-	(15)	-	-	-
Pre Tax Cash Flow	**C$M**	**797**	**54**	**91**	**96**	**69**	**67**	**52**	**46**	**32**	**41**	**48**	**44**	**45**	**54**	**40**	18	-	-
Taxes	C$M	**(56)**	-	-	-	(4)	(8)	(6)	(5)	(4)	(4)	(5)	(5)	(5)	(6)	(3)	(1)	(0)	(0)
After-tax Cash Flow	**C$M**	**741**	**54**	**91**	**96**	**65**	**59**	**46**	**40**	**28**	**37**	**44**	**39**	**40**	**48**	**37**	17	(0)	(0)
Cumulative after-tax cash flow	C$M		54	145	241	306	364	410	451	478	515	559	598	639	687	723	741	741	741
NPV@5%	C$M	**573**	54	87	87	56	48	36	30	20	25	28	24	24	27	20	9	(0)	(0)

Table 22-7: Open Pit Expansion Incremental Cash Flow

Area	Unit	Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Pd Price	$US/oz		901	935	948	855	855	855	855	855	855	855	855	855	855	855	855	855	855
Exchange Rate	CAD per USD		1.15	1.12	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11
Tonnes mined & milled	kt	38,514	-	-	-	-	1,495	3,551	4,563	4,563	4,563	4,563	4,563	4,563	4,563	1,530	-	-	-
Recovered Pd	koz	1,080	-	-	-	-	36	114	160	156	124	113	113	113	113	38	-	-	-
Recovered Pt	koz	131	-	-	-	-	5	13	17	17	15	15	15	15	15	5	-	-	-
Recovered Au	koz	86	-	-	-	-	3	9	12	12	10	9	9	9	9	3	-	-	-
Recovered Ni	kt	7	-	-	-	-	0.3	0.7	0.9	0.9	0.8	0.8	0.8	0.8	0.8	0.3	-	-	-
Recovered Cu	kt	16	-	-	-	-	0.6	1.5	2.0	2.0	1.9	1.8	1.8	1.8	1.8	0.6	-	-	-
Net Smelter Return	**C$M**	1,438	-	-	-	-	50	148	203	200	167	155	155	155	155	52	-	-	-
Mining OPEX	C$M	(303)	-	-	-	-	(6)	(39)	(60)	(67)	(40)	(25)	(23)	(22)	(18)	(5)	-	-	-
Surface Handling OPEX	C$M	(77)	-	-	-	-	(3)	(8)	(10)	(10)	(9)	(9)	(9)	(9)	(9)	(3)	-	-	-
Milling OPEX	C$M	(380)	-	-	-	-	(15)	(35)	(45)	(45)	(45)	(45)	(45)	(45)	(45)	(15)	-	-	-
G&A OPEX	C$M	(201)	-	-	-	-	(8)	(19)	(24)	(24)	(24)	(24)	(24)	(24)	(24)	(8)	-	-	-
Total OPEX	**C$M**	(961)	-	-	-	-	(32)	(100)	(138)	(145)	(118)	(102)	(101)	(99)	(95)	(31)	-	-	-
EBITDA	C$M	477	-	-	-	-	18	47	64	54	49	53	54	56	60	21	-	-	-
Project CAPEX	C$M	(51)	-	-	(9)	(22)	(20)	(0)	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	C$M	(143)	(1)	(0)	0	-	(1)	(31)	(28)	(24)	(20)	(9)	(10)	(10)	(6)	(2)	-	-	-
Change in Working Capital	C$M	-	-	-	-	-	(17)	(34)	(19)	1	12	4	0	0	0	36	18	-	-
Closure Costs	C$M	-	-	-	-	-	-	15	-	-	-	-	-	-	-	(15)	-	-	-
Pre Tax Cash Flow	**C$M**	283	(1)	(0)	(8)	(22)	(21)	(3)	17	32	41	48	44	45	54	40	18	-	-
Taxes	C$M	(35)	-	-	-	2	0	(2)	(4)	(4)	(4)	(5)	(5)	(5)	(6)	(3)	(1)	(0)	(0)
After-tax Cash Flow	**C$M**	248	(1)	(0)	(8)	(20)	(21)	(5)	13	28	37	44	39	40	48	37	17	(0)	(0)
Cumulative after-tax cash flow	C$M	-	(1)	(1)	(9)	(30)	(50)	(55)	(42)	(15)	22	66	105	146	194	231	248	248	248
NPV@5%	C$M	138	(1)	(0)	(8)	(17)	(17)	(4)	10	20	25	28	24	24	27	20	9	(0)	(0)
IRR	%	31%																	

The waterfall diagram below illustrates the breakdown of the discounted incremental cash flows from the open pit expansion project using a discount rate of 5%:

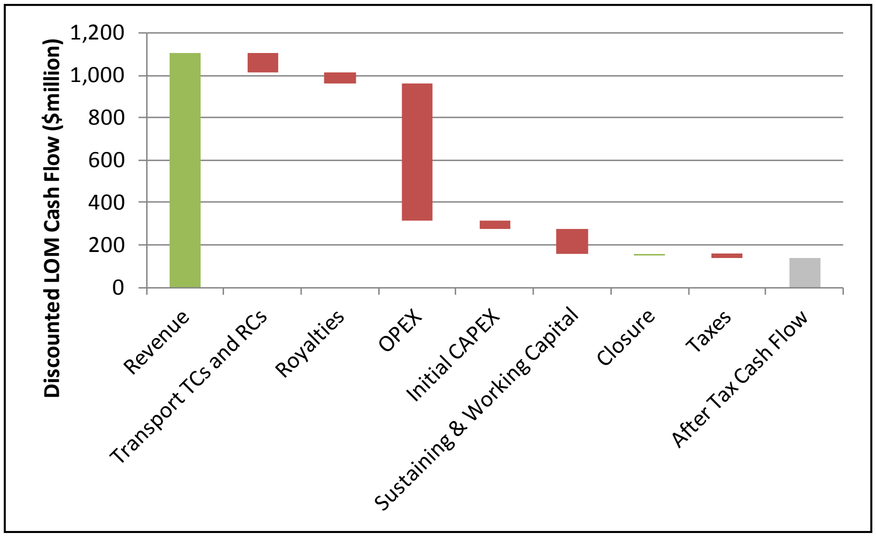


Figure 22-3: Open Pit Expansion project discounted cash flow waterfall diagram

The waterfall diagram below illustrates the breakdown of the undiscounted incremental cash flows from the open pit expansion project:

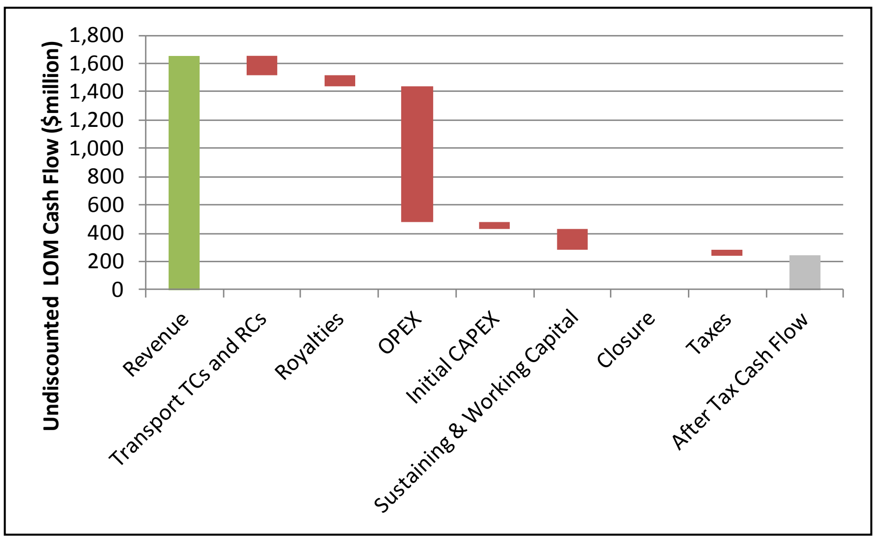


Figure 22-4: Open Pit Expansion project undiscounted cash flow waterfall diagram

22.5.2 Phase 2 Expansion

Table 22-8 shows the incremental cash flow based on the production schedule, revenue, CAPEX and OPEX inputs. The cumulative cash flows and cumulative discounted cash flows are also shown.

Table 22-8: Phase 2 Expansion Incremental Cash Flow

Area	Unit	Total	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Pd Price	$US/oz		901	935	948	855	855	855	855	855	855	855	855	855	855	855	855	855	855
Exchange Rate	CAD per USD		1.15	1.12	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11
Tonnes mined & milled	kt	10,857	-	-	-	-	-	-	-	-	-	-	-	-	-	3,032	4,563	3,262	-
Recovered Pd	koz	871	-	-	-	57	60	76	87	87	115	49	49	20	-	75	113	81	-
Recovered Pt	koz	55	-	-	-	3	2	1	2	2	4	3	3	1	-	10	15	11	-
Recovered Au	koz	54	-	-	-	3	3	4	5	5	6	2	2	1	-	6	9	7	-
Recovered Ni	kt	4	-	-	-	0.2	0.2	0.2	0.3	0.3	0.3	0.1	0.1	0.0	-	0.5	0.8	0.6	-
Recovered Cu	kt	7	-	-	-	0.4	0.1	0.2	0.3	0.3	0.4	0.2	0.2	0.1	-	1.2	1.8	1.3	-
Net Smelter Return	**C$M**	980	-	-	-	61	60	75	88	88	117	51	51	20	-	103	155	111	-
Mining OPEX	C$M	(343)	-	1	1	(28)	(26)	(27)	(32)	(21)	(48)	(38)	(49)	(19)	(6)	(18)	(21)	(12)	-
Surface Handling OPEX	C$M	(31)	1	0	1	0	(2)	(2)	(2)	(2)	(2)	(1)	(1)	(0)	-	(6)	(9)	(6)	-
Milling OPEX	C$M	(107)	-	-	-	(0)	(0)	(0)	(0)	(0)	(0)	-	-	-	-	(30)	(45)	(32)	-
G&A OPEX	C$M	(57)	-	-	-	-	-	-	-	-	-	-	-	-	-	(16)	(24)	(17)	-
Total OPEX	**C$M**	(538)	1	1	1	(27)	(28)	(29)	(34)	(22)	(50)	(39)	(50)	(20)	(6)	(69)	(98)	(68)	-
EBITDA	C$M	442	1	1	1	34	32	46	54	66	67	13	1	1	(6)	34	57	43	-
Project CAPEX	C$M	(242)	(35)	(77)	(91)	(7)	11	(21)	(22)	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	C$M	(50)	(2)	-	-	(6)	(7)	19	15	(11)	(10)	(22)	(11)	(3)	(0)	(3)	(5)	(4)	-
Change in Working Capital	C$M	-	0	0	0	(21)	0	(5)	(4)	0	(10)	22	(0)	11	7	(36)	(18)	15	38
Closure Costs	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15	-	(15)	-
Pre Tax Cash Flow	**C$M**	151	(36)	(75)	(90)	(1)	36	39	44	55	46	13	(11)	9	1	10	34	39	38
Taxes	C$M	(33)	-	-	-	4	6	(3)	(4)	(6)	(5)	(1)	1	0	1	(2)	(10)	(11)	(2)
After-tax Cash Flow	**C$M**	117	(36)	(75)	(90)	3	42	36	39	49	41	12	(10)	9	1	8	24	27	37
Cumulative after-tax cash flow	C$M	-	(36)	(112)	(202)	(199)	(156)	(120)	(81)	(31)	9	21	12	20	21	29	53	81	118
NPV@5%	C$M	22	(36)	(72)	(82)	3	35	28	29	35	28	8	(6)	5	1	4	12	13	17
IRR	%	7%																	

The waterfall diagram below illustrates the breakdown of the discounted incremental cash flows from the Phase 2 expansion project using a discount rate of 5%:

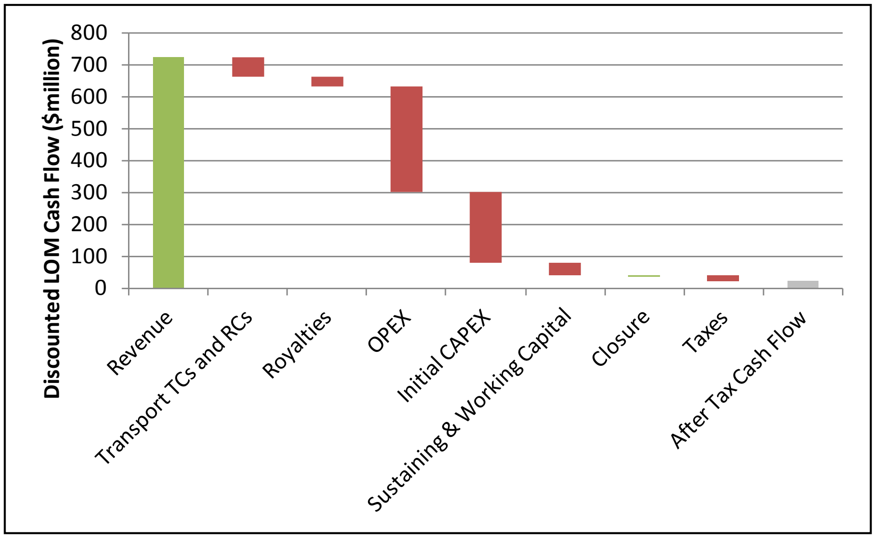


Figure 22-5: Phase 2 Expansion Project Discounted Cash Flow Waterfall Diagram

The waterfall diagram below illustrates the breakdown of the undiscounted incremental cash flows from the open pit expansion project:

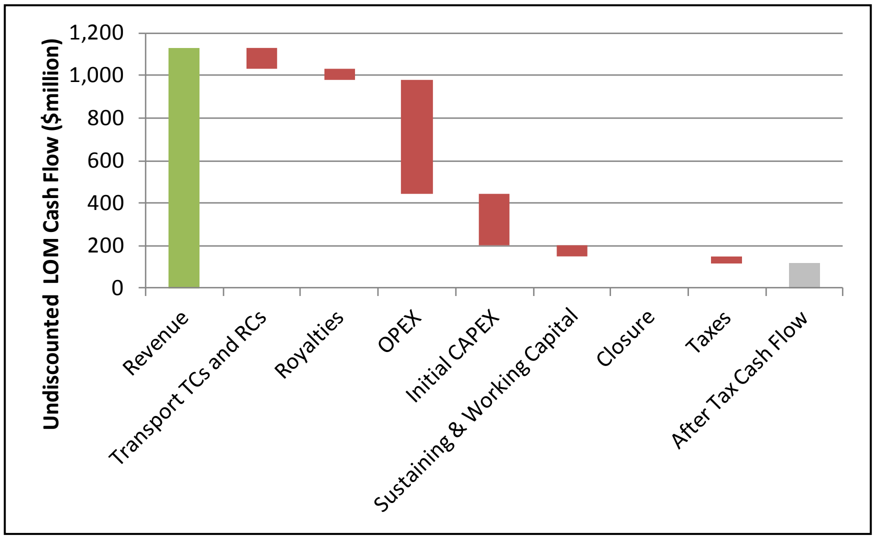


Figure 22-6: Phase 2 Expansion Project Undiscounted Cash Flow Waterfall Diagram

22.6 Sensitivity Analysis

A sensitivity analysis was conducted on the financial model in order to identify key variables with significant impact on forecasted returns. Particularly, the analysis focused on metal prices, operating and capital costs. Forecasted sets of key variables were independently varied and the resulting net present value was recorded.

22.6.1 Base Case (Current Mine Plan + Open Pit Expansion)

The overall operations NPV sensitivity to changes in commodity price, head grade, project CAPEX, OPEX, and TCs/RCs are shown in Figure 22-7. As can be seen, the scenario is most sensitive to changes in grade, followed by exchange rate and palladium price.

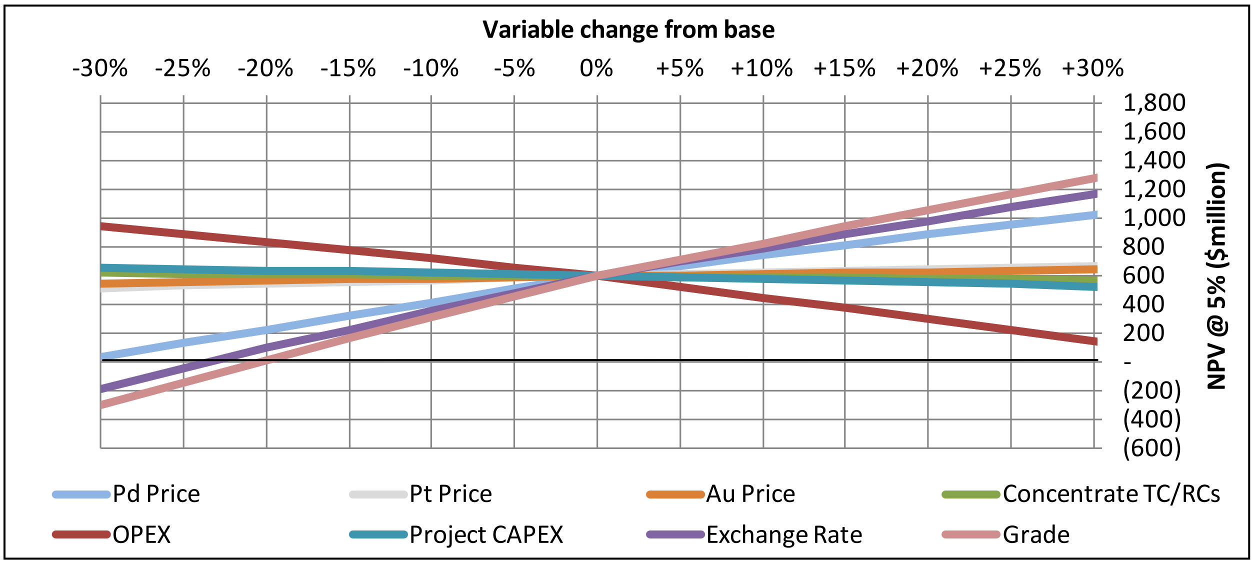


Figure 22-7: NPV Sensitivity – Base Case

The Base Case sensitivity to long term palladium price and exchange rate is shown in Table 22-9. The 3 year average and spot prices (as of February 12, 2015) are highlighted as well as the range of consensus prices used in the evaluation time frame.

Table 22-9: NPV Sensitivity – Base Case

		Palladium Price (USD $/oz)																	
NPV@ 5% CAD $ millions			3 yr Avg		Spot			Consensus											
		700	725	750	772	800	825	855	875	900	925	950	975	1,000	1,100	1,200	1,300	1,400	1,500
Exchange Rate CAD per USD	0.95	104	154	204	246	292	340	393	428	472	516	561	605	645	781	914	1,045	1,175	1,305
	1.00	151	203	254	295	343	394	449	487	533	579	626	663	700	841	980	1,117	1,255	1,392
	1.05	200	254	303	347	397	450	509	548	597	643	681	719	757	904	1,050	1,194	1,339	1,483
	1.10	243	295	347	392	445	501	562	603	649	689	729	768	807	960	1,111	1,263	1,413	1,564
	1.11	252	304	357	402	456	511	573	615	659	699	739	779	818	972	1,125	1,277	1,429	1,581
	1.15	285	340	394	441	496	554	618	655	697	738	779	819	860	1,019	1,177	1,335	1,493	1,650
	1.16	294	349	403	451	507	565	630	664	706	748	789	830	870	1,031	1,190	1,349	1,508	1,668
	1.20	327	384	440	489	547	608	666	701	744	786	828	870	912	1,078	1,243	1,407	1,572	1,737
	1.25	366	425	483	534	595	652	706	742	786	830	874	917	960	1,132	1,303	1,474	1,645	1,816
	1.30	411	472	533	587	646	697	753	790	836	881	926	971	1,016	1,195	1,374	1,552	1,731	1,909
	1.35	453	516	580	634	686	739	796	834	881	928	975	1,021	1,068	1,254	1,439	1,624	1,810	1,995
	1.40	495	560	626	673	726	779	838	877	926	975	1,023	1,071	1,120	1,312	1,504	1,697	1,889	2,081

Note: Sensitivity to other metal prices is also reflected above. Other metal prices are aligned for spot and long term consensus ($855/oz) but interpolated for the other data points using a linear relationship to palladium prices inferred from the spot and long term consensus.

22.6.2 Open Pit Expansion Project

The Open Pit Expansion project NPV sensitivity to changes in commodity price, head grade, project CAPEX, OPEX, and TCs/RCs are shown in Figure 22-8. As can be seen, the scenario is most sensitive to changes in grade, exchange rate and palladium price:

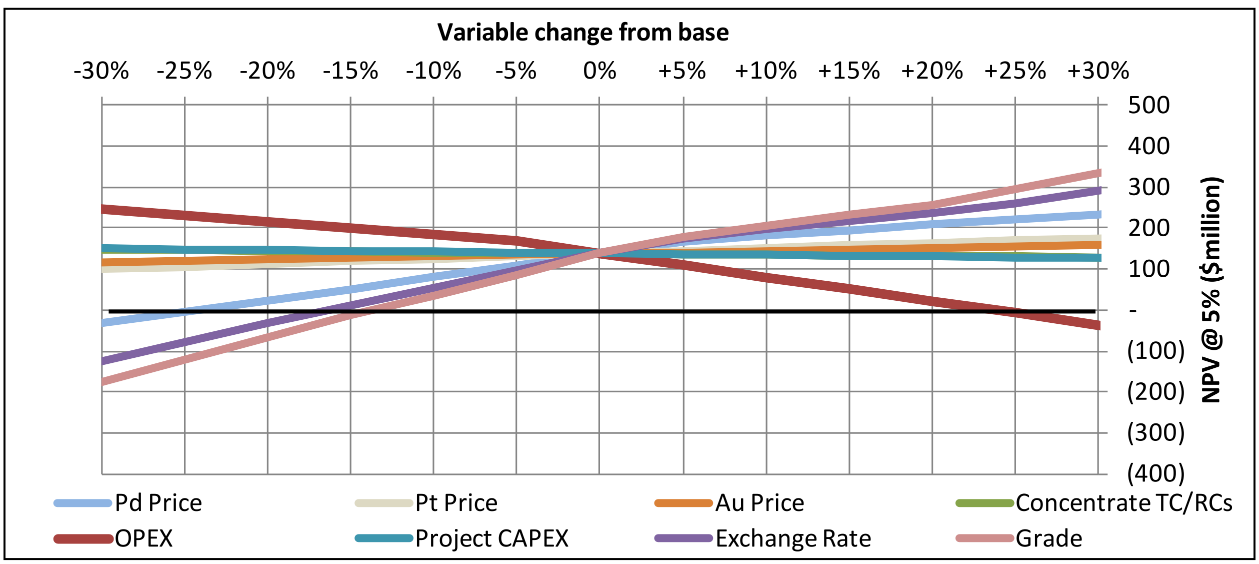


Figure 22-8: NPV Sensitivity – Open Pit Expansion Project

The Open Pit Expansion project sensitivity to long term palladium price and exchange rate is shown in Table 22-10. The 3 year average and spot prices (as of February 12, 2015) are highlighted as well as the range of consensus prices used in the evaluation time frame.

Table 22-10: NPV Sensitivity – Open Pit Expansion Project

		Palladium Price (USD $/oz)																
NPV@ 5% CAD $ millions			3 yr Avg		Spot			Consensus										
		700	725	750	772	800	825	855	875	900	925	950	975	1,000	1,100	1,200	1,300	1,400	1,500
Exchange Rate CAD per USD	0.95	-193	-156	-120	-89	-56	-23	15	39	71	102	133	164	190	275	355	434	512	601
	1.00	-160	-121	-85	-55	-21	14	53	79	112	145	177	201	224	310	394	476	565	660
	1.05	-125	-86	-51	-20	16	53	94	122	156	187	211	235	258	347	435	522	623	723
	1.10	-94	-57	-21	11	48	87	130	159	190	215	240	264	287	380	471	569	674	779
	1.11	-89	-51	-14	18	55	95	138	167	196	221	246	270	294	387	478	579	685	791
	1.15	-66	-27	11	44	83	124	169	193	219	244	269	294	318	414	509	618	728	838
	1.16	-60	-21	17	51	90	131	177	198	224	250	275	300	324	421	518	628	739	850
	1.20	-37	3	42	77	118	160	198	220	247	273	298	324	349	448	553	668	783	898
	1.25	-11	30	72	108	150	189	223	245	272	298	325	351	377	480	595	714	833	953
	1.30	20	63	106	143	184	216	250	273	300	328	355	382	409	519	643	768	892	1017
	1.35	48	93	137	175	208	240	275	298	327	355	383	411	439	559	688	817	947	1076
	1.40	76	123	169	198	231	264	300	324	353	382	411	440	469	599	733	867	1001	1135

Note: Sensitivity to other metal prices is also reflected above. Other metal prices are aligned for spot and long term consensus ($855/oz) but interpolated for the other data points using a linear relationship to palladium prices inferred from the spot and long term consensus.

22.6.3 Phase 2 Expansion Project

The Phase 2 Expansion project NPV sensitivity to changes in commodity price, head grade, project CAPEX, OPEX, and TCs/RCs are shown in Figure 22-9. As can be seen, the scenario is most sensitive to changes in grade, exchange rate and palladium price:

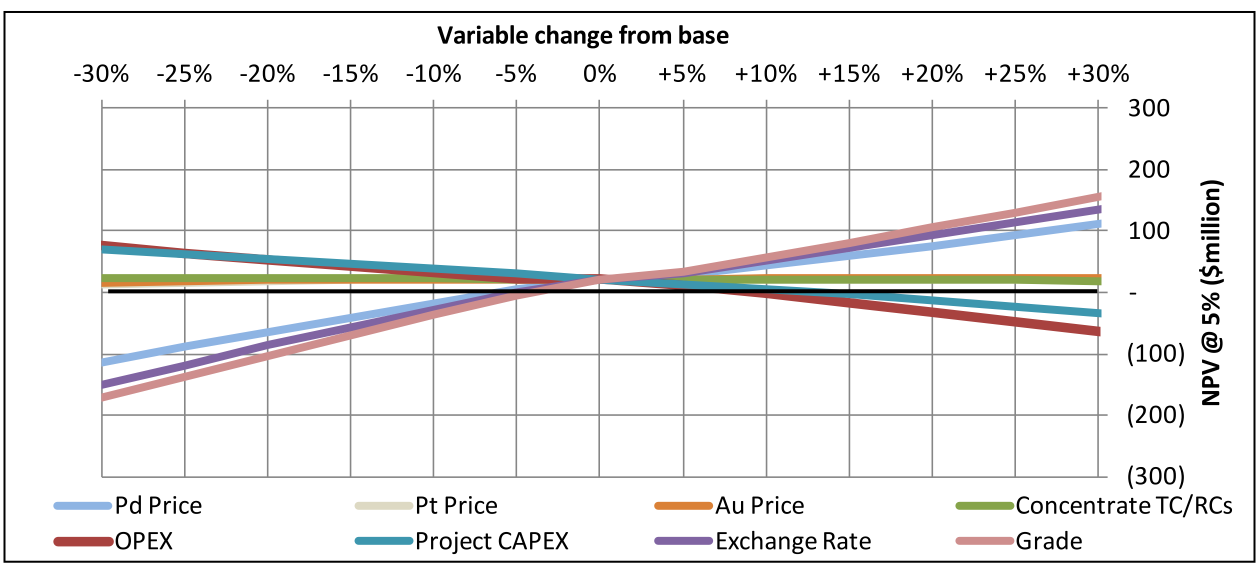


Figure 22-9: NPV Sensitivity – Phase 2 Expansion Project

The Phase 2 Expansion project sensitivity to long term palladium price and exchange rate is shown in Table 22-11. The 3 year average and spot prices (as of February 12, 2015) are highlighted as well as the range of consensus prices used in the evaluation time frame.

Table 22-11: NPV Sensitivity – Phase 2 Expansion Project

		Palladium Price (USD $/oz)																	
NPV@ 5% CAD $ millions			3 yr Avg		Spot			Consensus											
		700	725	750	772	800	825	855	875	900	925	950	975	1,000	1,100	1,200	1,300	1,400	1,500
Exchange Rate CAD per USD	0.95	-178	-157	-137	-118	-96	-76	-54	-39	-21	-2	14	20	29	81	134	188	241	295
	1.00	-156	-135	-111	-92	-72	-51	-28	-13	7	19	24	38	51	106	163	219	275	332
	1.05	-133	-109	-86	-68	-47	-25	-1	15	23	32	46	60	74	133	193	252	312	371
	1.10	-112	-88	-66	-47	-25	-2	20	25	36	50	65	80	96	158	220	282	344	406
	1.11	-107	-83	-62	-43	-21	2	22	27	39	54	69	85	100	163	225	288	351	413
	1.15	-90	-67	-45	-25	-2	20	29	39	55	70	86	102	118	183	248	313	378	443
	1.16	-86	-63	-40	-21	3	22	30	43	58	74	90	107	123	188	254	319	385	450
	1.20	-70	-46	-23	-3	21	29	45	58	74	91	107	124	141	209	277	344	412	480
	1.25	-52	-27	-3	18	29	41	61	75	92	110	127	145	162	233	303	373	444	514
	1.30	-31	-5	20	30	41	60	81	95	113	132	150	168	187	260	334	407	481	554
	1.35	-11	15	31	38	57	77	99	114	133	153	171	191	210	286	362	438	515	591
	1.40	8	31	38	54	73	94	118	134	153	173	193	213	233	312	391	470	549	628

Note: Sensitivity to other metal prices is also reflected above. Other metal prices are aligned for spot and long term consensus ($855/oz) but interpolated for the other data points using a linear relationship to palladium prices inferred from the spot and long term consensus.

22.6.4 Discount Rate Sensitivity

Figure 22-10 illustrates the sensitivity of the Open Pit Expansion and Phase 2 Expansion NPV to varying discount rates:

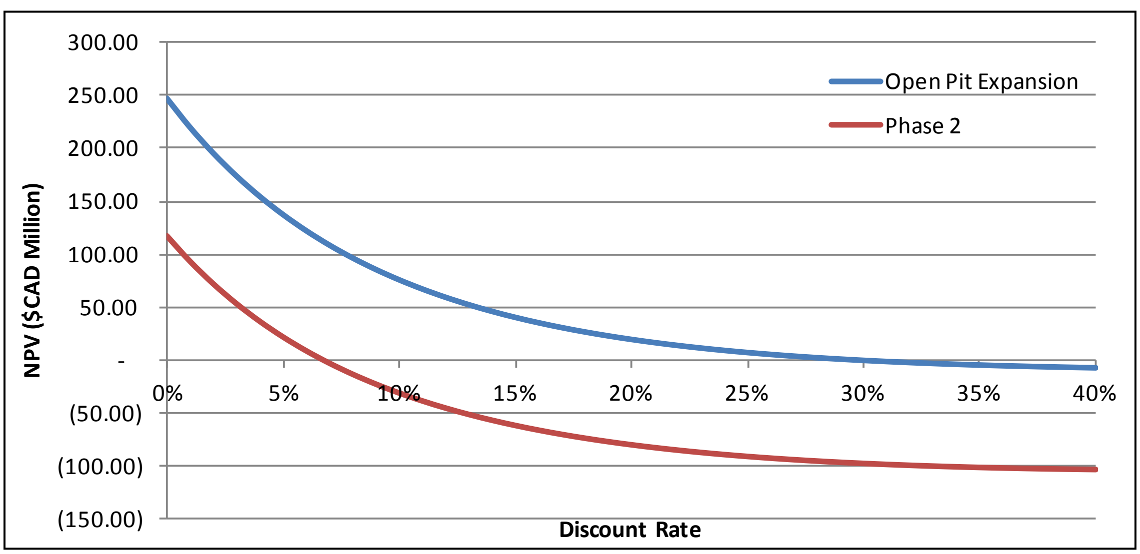


Figure 22-10: Discount Rate Sensitivity for Expansion Scenarios

23. Adjacent Properties

There are several mineral exploration properties adjacent to the Property that cover mafic-ultramafic intrusions thought to belong to the LDIM suite (Figure 23-1). NAP, through the wholly-owned subsidiary LDIM, holds an interest in most of these adjacent properties through staking or option agreements. These include: Legris Lake, Tib Lake, Chisamore, Dog River, Buck Lake, Taman Lake, Wakinoo Lake, Bullseye, Demars Lake, and Varris Lake Figure 23-1). The Shelby property was added to LDIM's portfolio in 2014 through an option agreement with Platinum Group Metals for 20 claims and through the staking of 19 adjacent claims. LDIM returned the Salmi gold property to the underlying vendors in 2014 (Figure 23-1). Small claim blocks occurring in the general vicinity of LDIM are held by prospectors and individuals including: W.J. Richmond and F.A Houghton (both located between LDIM's Buck Lake and Dog River properties), A.K. Siltamaki/J.A. Siltimaki (including the returned Salmi property); W.J. Wheeler (north of Salmi property); R.C. Stenlund (east of highway 527); S.E. Siemieniuk, W.J. Roberts and J.R. Shaver (east of Shelby property). Claims to the south of Legris Lake previously held by M. Magrum have lapsed.

There was very limited exploration work in the LDI surrounding area during 2014. LDIM completed a 575 line km airborne electromagnetic and magnetic survey over portions of the Shelby, Varris Lake, Wakinoo Lake and Demars Lake properties.

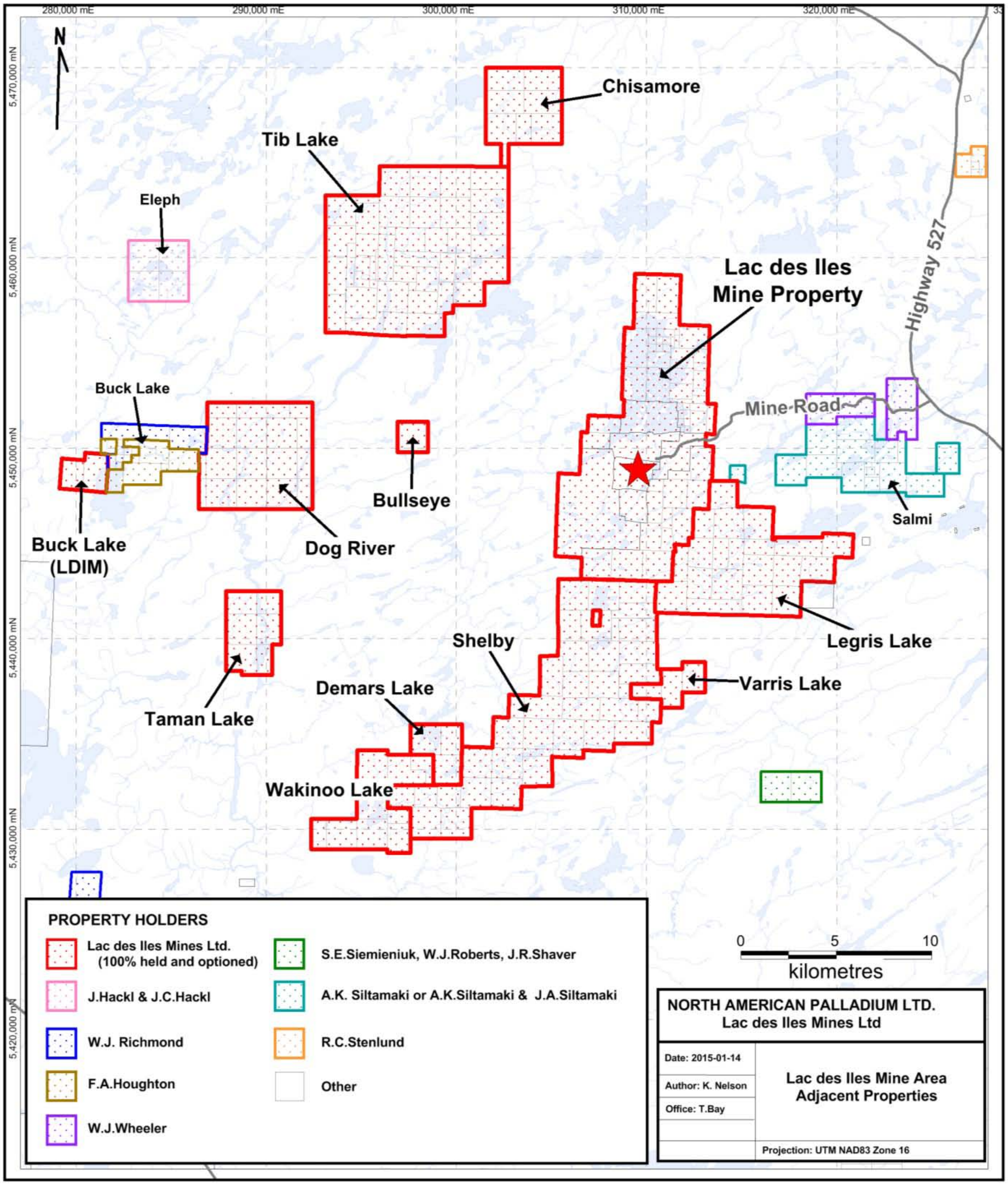


Figure 23-1: Property Location Map

24. Other Relevant Data and Information

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25. Interpretation and Conclusions

25.1 Mineral Resource Estimate

The major changes to the mineral resource estimate for the Property include:

- Updated Offset Zone resource estimate including expanded Lower Offset Zone indicated resources in support of a potential Phase 2 mine expansion.
- Increase to Roby Zone open pit expansion resources based on a revised, preliminary pit expansion shell and an updated Pd cut-off grade of 0.6 g/t.
- Initial resource estimates for the Powerline and Upper Offset southeast extension zones.
- Revision to the Sheriff Zone resource estimate based on a lowering of the Pd cut-off grade and the declaration of an initial resource in the Powerline Zone, part of which overlaps with the Sheriff Zone.

Independent of this report, NAP has determined that all of the RGO mineral resources can be converted to mineral reserves; hence there are no RGO mineral resources.

25.2 Mining

The mining work performed in this PEA included a review of the operating cost model for LDI to estimate operating costs and cut-off grades for the PEA. Based on this review, the following conclusions were determined:

- The full-time mill operation results in decreased operating costs as a result of economies of scale.
- Increasing the hoisting capacity to 6,500 tpd from underground would decrease underground mining unit costs.
- An ore pass handling system rather than trucking up to a truck dump level not only reduces unit operating costs, but also provides potential to increase production rate.
- Improved unit costs decrease cut-off grades, which allows more tonnage to come into the mine plan, increasing the mine life.

Using these improved unit costs and the mine plans generated as described in earlier sections, the following conclusions were determined:

- There is potential for re-opening of the open pit which has not been fully operational since 2008. This would mine life to the year 2029 or beyond.
- There is sufficient underground mining to supplement the RGO feed to its depletion.
- Additional footwall material in Offset Zone Phase 1 and Roby Footwall Zone can potentially be included in the mine plan.
- The use of paste fill for backfilling improves recovery.

25.3 Metallurgy

For the PEA, current operating data was reviewed and a grade/recovery curve generated using that data. From this curve, and from a review of the current milling operating data, the following conclusions were determined:

- The flash flotation cell can provide a recovery increase of 1 to 2%. An additional benefit from flash flotation is the recovery of Pd and other PGM minerals at early stages of flotation, similar to a gravity circuit in gold milling, in order to minimize generation of ultrafine/flat Pd particles that can be problematic to recover.
- The current Pd recoveries range between 75% to 85% and are a strong function of the Pd head grade. The data available indicates that the installation of the flash flotation cell has improved the variation in Pd recovery.

25.4 Economic Analysis

The economics for the LDI expansion project were evaluated using an Excel-based discounted cash flow (DCF) model, in which the production, revenues, operating costs, capital costs and taxes were considered. The Base Case scenario and Phase 2 scenario were evaluated on an incremental basis relative to the Current Mine Plan case as described in earlier sections.

As a result of this analysis, the following conclusions were determined:

- The economics of the Open Pit expansion plan are shown to be robust, with an incremental after-tax $NPV_{5\%}$ of $138M, with an after-tax IRR of 31%. The total $NPV_{5\%}$ of the Base Case is $573M.
- The economics of the Phase 2 expansion plan are shown to be marginally positive, with an incremental after-tax $NPV_{5\%}$ of $22M, with an after-tax IRR of 7%.. This plan will require additional mineral definition to justify the expansion.
- Both expansion plans are most sensitive to foreign exchange, followed by palladium prices.
- The Base Case plan maintains a total positive cash flow from operations in every year.

25.5 Opportunities

- The Roby Footwall Zone indicates a higher margin than the RGO and has the potential to defer the processing of the RGO stockpile. In turn, this may allow an extension of underground mining in the event that additional tonnage is proven up in Phase 1.
- The potential Open Pit indicates a higher margin than the RGO and, if started quickly, could defer production from the RGO stockpile.

25.6 Risks

The following risks are identified in this report:

- Permitting: The project will require periodic permits for expansion of the tailings storage facilities and waste piles. There is a risk that if these permits are delayed, project timelines will lengthen.
- Commodity prices: Due to the low grade of the open pit, the project economics are sensitive to fluctuations in commodity prices.
- Ore grades: Ore grade interpolation is subject to uncertainty. The project economics are sensitive to grade variations from current estimates.
- Cost Inflation/Escalation: The project economics are sensitive to capital and/or operation cost escalation.
- Exchange rates: As the project costs are incurred in $CAD and the project revenues are received in $USD, the project economics are sensitive to exchange rate fluctuations.
- Metallurgical Recovery: The current estimates for recovery are based on historical plant operations performance with a 2% improvement to palladium recovery based on limited recent data since the installation of flash cells. The project economics are sensitive to metallurgical recovery assumptions.
- Processing Contract: If the Vale contract for concentrate smelting is not renewed before its expiration on June 30, 2015 or alternative options are not arranged, there could be a negative impact on company cash flows.

The execution risk for the proposed mining plans is expected to be lower than a greenfield mining project due to:

- It is an operating site within a mining friendly jurisdiction in proximity to existing regional infrastructure.
- It currently operates a mill with known recoveries and a permitted tailings facility with long-term expansion capability.
- It employs experienced personnel.
- It has long established and positive relations with the regional First Nation's communities.

As in any mining project, there are risks associated with geological interpretation and geomechanics. These will require further analysis at the next level of study.

26. Recommendations

Based on the results of this PEA study, it is recommended that NAP:

- Commence a PFS on the Base Case mine plan, which includes the open pit expansion, selective mining of the Roby and Offset FW zones and the additional mineral resources identified in the HW. It is recommended for the PFS that some or all of geological block models be combined into one. The estimated cost of this is approximately $1.5M.
- Investigate opportunities of accelerating the development of some of the higher grade open pit options such as the North VT Rim and Powerline areas that exist within the shell of the major pit expansion. The estimated cost of this is approximately $0.1M.
- Continue exploration and definition drilling at depth to upgrade and expand the Offset Zone resource to support future evaluation of the Phase 2 shaft deepening expansion scenario. The estimated cost of this is approximately $7M.
- Perform engineering studies investigating opportunities to improve the business case for Phase 2. The estimated cost of this is approximately $0.4M.
- To perform an engineering study to investigate the benefits of adding paste backfill capabilities to support the Base Case mine plan through higher mined recoveries and improved stability. The estimated cost of this is approximately $0.1M.

Per NAP, these expenditures are provided in the LDI 2015 budget.

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28. Signature Page and Date

The undersigned prepared this technical report titled “Amended and Restated NI 43-101 Technical Report for Lac des Iles Mine, Ontario Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan”. The effective date of this Technical report is February 18, 2015, and the disclosure date is April 20, 2015.

Signed,

“Signed and Sealed”

Dr. David Peck, P. Geo.	April 20, 2015	North American Palladium Ltd. 200 Bay St., Royal Bank Plaza, South Tower, Suite 2350 Toronto, Ontario, Canada M5J 2J2

“Signed and Sealed”

Mr. Denis Descharte, P.Eng	April 20, 2015	North American Palladium Ltd. 200 Bay St., Royal Bank Plaza, South Tower, Suite 2350 Toronto, Ontario, Canada M5J 2J2

“Signed”

Mr. Dave Penna, P.Geo.	April 20, 2015	Lac des Iles Mines Ltd. P.O. Box 10547 Thunder Bay, Ontario, Canada P7B 6T9

“Signed and Sealed”

Mr. Chris Roney, P.Eng	April 20, 2015	731 22nd Street Brandon, Manitoba, Canada R7B 1S7

"Signed and Sealed"

Mr. Brian Young, P.Eng	April 20, 2015	Hatch Ltd. 40 Elm Street, Unit ND 255 Sudbury, Ontario, Canada P3C 1S5

"Signed and Sealed"

Mr. Babak Houdeh, P.Eng	April 20, 2015	Hatch Ltd. 2800 Speakman Drive Sheridan Science & Technology Park Mississauga, Ontario, Canada, L5K 2R7

"Signed"

Mr. Robert Duinker, P.Eng	April 20, 2015	Hatch Ltd. 2800 Speakman Drive Sheridan Science & Technology Park Mississauga, Ontario, Canada, L5K 2R7

Appendix A:
Certificates of Qualified Persons

CERTIFICATE OF QUALIFIED PERSON

Denis Decharte, P. Eng.

I, Denis Decharte P.Eng. of Montreal, Quebec do hereby certify:

I am a Resource Modeller with Lac des Iles Mines Ltd. with a business address at P.O. Box 10547, Thunder Bay, Ontario, P7B 6T9.

This certificate applies to the technical report entitled “Amended and Restated NI 43-101 Technical Report for the Lac des Iles Mine, Ontario, Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan”, dated April 20, 2015 and effective February 18, 2015 (the “Technical Report”).

I am a graduate of the Ecole Nationale Supérieure de Géologie de Nancy, France (Geological Engineering diploma, 2007). I am a member in good standing of the Association of Professional Engineers of Ontario (licence #100202880) and the Ordre des Ingénieurs du Québec (licence #145332). My relevant experience includes 7 years of experience in mineral exploration and mining operation, including geological, resource and reserve modelling, database management, exploration planning and core logging. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

My most recent personal inspection of the Property was March 18th, 2015 for 3 days.

I am responsible for Sections 14.1 and 14.2 and contributed to Section 14.8 of the Technical Report.

I am not independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.

I have worked as a Resource Modeller for Lac Des Iles Mines Ltd. and North American Palladium Ltd. since April 2012, and prior to that, I had worked as a Junior Geological Engineer for NAP Quebec Mines Ltd., a subsidiary of North American Palladium Ltd. at this time, since May 2010.

I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 20th day of April, 2015 at Montreal, Quebec.

Signed and Sealed

Denis Decharte, P.Eng.
Resource Modeller
Lac Des Iles Mines Ltd.

CERTIFICATE OF QUALIFIED PERSON

David C. Peck, P. Geo.

I, David C. Peck, P. Geo. of Brandon, Manitoba do hereby certify:

I am a Vice President, Exploration with North American Palladium Ltd. with a business address at Suite 2350, 200 Bay Street, Toronto, Ontario, M5J 2J2.

This certificate applies to the technical report entitled “Amended and Restated NI 43-101 Technical Report for the Lac des Iles Mine, Ontario, Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan”, dated April 20, 2015 and effective February 18, 2015 (the “Technical Report”).

I am a graduate of the University of Windsor (B.Sc. Geology, 1984; M.Sc. Geology, 1986) and the University of Melbourne (Ph.D. Geology, 1991). I am a member in good standing of the Association of Professional Engineers and Geoscientists of Manitoba (licence #20431), the Association of Professional Engineers and Geoscientists of British Columbia (licence # 34031) and the Association of Professional Geoscientists of Ontario (licence# 2301). My relevant experience includes 30 years of experience in mineral exploration, strategic business planning and applied mineral deposit research with a continued focus on magmatic platinum-group element and nickel-copper sulphide deposits. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

My most recent personal inspection of the Property was November 4th, 2014 for 1 day.

I am responsible for Sections 4, 6, 7, 8, 9, 10, 11, 12, 14.3, 15 and 23 and contributed to Sections 1, 14, 25 and 26 of the Technical Report.

I am not independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.

I have been responsible for overseeing exploration on The Property that is the subject of the Technical Report since March, 2012.

I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 20th day of April, 2015 at Brandon, Manitoba

Signed and Sealed

David C. Peck, P. Geo.
Head of Exploration
North American Palladium Ltd.

CERTIFICATE OF QUALIFIED PERSON

Chris Roney, P.Geo.

I, Chris Roney, P.Geo., of Brandon, Manitoba, do hereby certify:

I am a Consulting Geologist with Lac des Iles Mines Ltd. with a business address at 731 22nd Street, Brandon, Manitoba, R7B 1S7.

This certificate applies to the technical report entitled “Amended and Restated NI 43-101 Technical Report for the Lac des Iles Mine, Ontario, Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan”, dated April 20, 2015 and effective February 18, 2015 (the “Technical Report”).

I am a graduate of Brandon University (B.Sc. Specialized, 1986). I am a member in good standing of the Association of Professional Geoscientists of Ontario (#2243) and Association of Professional Engineers and Geoscientists of Manitoba (#20332). My relevant experience includes 28 years of experience in exploration and operations, including several years working in narrow vein gold deposit. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property was September 29 to October 8, 2014 for 10 days.

I am responsible for Sections 14.5, 14.6 and 14.7 and contributed to section 14.8 of the Technical Report.

I am not independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.

I have worked as Consulting Geologist for Exploration Group at the mine on the Property that is the subject of this Technical Report since September 2012.

I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 20th day of April, 2015 at Brandon, Manitoba.

Signed and Sealed

Chris Roney, P.Geo.
Consulting Geologist
Lac des Iles Mines Ltd.

CERTIFICATE OF QUALIFIED PERSON

David N. Penna, P. Geo.

I, David N. Penna, of Thunder Bay, Ontario, do hereby certify:

I am a Principal Geologist with Lac Des Iles Mines Ltd with a business address at P.O Box 10547, Thunder Bay, Ontario P7B 6T9.

This certificate applies to the technical report entitled "Amended and Restated NI 43-101 Technical Report for the Lac des Iles Mine, Ontario, Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan", dated April 20, 2015 and effective February 18, 2015 (the "Technical Report").

I am a graduate of Laurentian University, (B.Sc Honors, 1986). I am a member in good standing of the Association of Professional Geoscientists of Ontario, Membership Number 0112. My relevant experience includes 28 years of experience in underground mining including mineral resource/reserve evaluation, grade control, geological mapping, and forecasting/budget preparation. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property is current as I am employed at the mine site.

I am responsible for Section 14.4 and contributed to Section 14.8 of the Technical Report.

I am not independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.

I have been responsible as a Principal Geologist with the Property that is the subject of the Technical Report.

I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.

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

Signed and sealed this 20th day of April, 2015 at Thunder Bay, Ontario.

Signed

David N. Penna
Principal Geologist
Lac Des Iles Mines, Ltd.

CERTIFICATE OF QUALIFIED PERSON

Robert Duinker, P.Eng
Hatch, 2800 Speakman Drive, Mississauga, Ontario, Canada, L5K 2R7

I, Robert Duinker, P.Eng, am employed as a Senior Consultant with Hatch Ltd.

This certificate applies to the Technical Report entitled “Amended and Restated NI 43-101 Technical Report for Lac des Iles Mine, Ontario Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan”, dated April 20, 2015 and effective February 18, 2015 (the “Technical Report”).

I graduated from Queen’s University with a Bachelors of Applied Science (BASc.) in 2002 and from Queen’s University with a Masters of Business Administration (MBA), in 2007. I am a member in good standing of Professional Engineers of Ontario. My relevant experience includes 8 years of experience in management consulting, primarily for the mining industry. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

I have completed a personal inspection of the property on May 2nd 2013

I am responsible for Sections 1.7, 19, 22, 25.4 of the Technical Report.

I had previously completed a review of the Property that is the subject of this Technical Report.

I am independent of North American Palladium Ltd., as defined by Section 1.5 of the Instrument.

I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in accordance with the Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed

Robert Duinker, P.Eng
Senior Consultant
Hatch

April 20, 2015

CERTIFICATE OF QUALIFIED PERSON

Babak Houdeh, P. Eng.
Hatch, 2800 Speakman Drive, Mississauga, Ontario, Canada, L5K 2R7

I, Babak Houdeh, P. Eng, am employed as a Senior Process Engineer with Hatch Ltd.

This certificate applies to the Technical Report entitled "Amended and Restated NI 43-101 Technical Report for Lac des Iles Mine, Ontario Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan", dated April 20, 2015 and effective February 18, 2015 (the "Technical Report").

I graduated from Amirkabir University in Tehran, Iran with a Bachelors of Science (BSc.) Mining Engineering in 1992, and a Masters of Science (MSc.) Mining Engineering in 1995. I am a member in good standing of Professional Engineers of Ontario (PEO# 100168863). My relevant experience includes 10 years of experience in mineral processing including process development, plant design, process simulation, feasibility studies, and forecasting/budget preparation. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

I have not completed a personal inspection of the property.

I am responsible for Sections 1.5, 13, 17 and 25.3 of the Technical Report.

I am independent of North American Palladium Ltd., as defined by Section 1.5 of the Instrument.

I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in accordance with the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed

Babak Houdeh, P. Eng.
Senior Process Engineer
Hatch Ltd.

April 20, 2015

CERTIFICATE OF QUALIFIED PERSON

Brian Young, P.Eng
Hatch, 40 Elm St. Unit 255, Sudbury, Ontario, Canada, P3C 1S8

I, Brian Young, P.Eng, am employed as a Senior Mining Engineer with Hatch Ltd.

This certificate applies to the Technical Report entitled “Amended and Restated NI 43-101 Technical Report for Lac des Iles Mine, Ontario Incorporating A Preliminary Economic Assessment of the Mine Expansion Plan”, dated April 20, 2015 and effective February 18, 2015 (the “Technical Report”).

I graduated from Queen’s University in Kingston, Ontario with a B, Sc. Eng. (Mining), in 1973. I am a member in good standing of the Association of Professional Engineers of Ontario (PEO#51601508). My relevant experience includes 41 years in mine engineering and operations. I have completed scoping studies and life-of-mine studies for underground mines.

I completed a personal inspection of the property on October 28th and October 29th, 2014.

I am responsible for Sections 1.1, 1.4, 1.6, 1.8, 1.9, 1.10, 2, 3, 5, 16, 18, 20, 21, 24, 25.2, 25.5, 25.6, 26 and 27 of the Technical Report.

I am independent of North American Palladium Ltd., as defined by Section 1.5 of the Instrument.

I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and Sealed

____

Brian Young, P.Eng
Senior Mining Engineer
Hatch Ltd.

April 20, 2015