Report to:

NORTH AMERICAN PALLADIUM LTD.

Technical Report Lac des Iles Mine, Ontario, Incorporating Prefeasibility Study Offset Zone Phase I

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NORTH AMERICAN PALLADIUM LTD.

North American Palladium Ltd.

TECHNICAL REPORT LAC DES ILES MINE, ONTARIO, INCORPORATING PREFEASIBILITY STUDY OFFSET ZONE PHASE I

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GLOSSARY

UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic yard	yd^3
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 m^2)	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/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
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	Mbm^3
million bank cubic metres per annum	Mbm^3/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	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
thousand circular mils	MCM
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/hm^3
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt

ABBREVIATIONS AND ACRONYMS

A.R. McPherson Consultants Ltd.	ARMC
Accurassay Laboratories	Accurassay
Activation Laboratories Ltd.	ActLabs
alternating current	AC
ammonium nitrate fuel oil	ANFO
Archean Lac des Iles Intrusive Complex	LDI-IC
atomic emission spectroscopy	AES
Aurubis AG	Aurubis
Autogenous/ball mill/crushing	ABC
Belham Limited	Belham
Bond Work Index	BWI
Canadian development expenses	CDE
Canadian exploration expenses	CEE

Canadian Institute of Mining, Metallurgy and Petroleum	CIM
capital costs allowance	CCA
carboxymethyl cellulose	CMC
CDN Resource Laboratories	CDN
Certificate of Approval	CofA
Communities of Interest	COI
counterweight	CWT
diamond drillhole	DDH
east gabbro	EGAB
Electron Probe Micro Analysis	EPMA
exchange traded funds	ETF
equivalent grinding length	EGL
fully-autogenous grinding	FAG
Fusion Data Management	Fusion
gabbro	GAB
gabbronorite breccia	GNBX
gabbronorite	GN
general and administrative	G&A
Geomechanics Research Centre	GRC
global positioning system	GPS
Harmonized Sales Tax	HST
heterolithic gabbro breccias	HGABBX
hollow structural sections	HSS
induced polarization	IP
inductively coupled plasma	ICP
Internal Rate of Return	IRR
International Electrotechnical Commission	IEC
International Organization for Standardization	ISO
in-the-hole	ITH
inverse distance squared	ID^2
Itasca Consulting Canada Inc.	Itasca
John Patrick Sheridan	Sheridan
Knelson Research and Technology Centre	Knelson
laboratory information management system	LIMS
Lac des Iles Mines Ltd.	LDIM
Lac des Iles Property	the Property
life-of-mine	LOM
load-haul-dump	LHD
Local Citizens Committee	LCC
locked-cycle test	LCT
magnetic gabbro	MTGAB
methyl isobutyl carbinol	MIBC
mill-float-twice	MF2
Mine Block Intrusive	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 return	NSR
North American Palladium Ltd.	NAP
optical emission spectrometry	OES
ordinary kriging	OK
P&E Mining Consultants	P&E
Pacific North West Capital Corp.	PFN
platinum group element	PGE
platinum group metals	PGM
potassium amyl xanthate	PAX
programmable logical controller	PLC
pyroxenite	PYXT
qualified persons	QPs
Qualitative Evaluation of Minerals by Scanning Electron Microsope	QEMSCAN
quality assurance	QA
quality control	QC
regular grade ore	RGO
rock quality designation	RQD
run-of-mine	ROM
SAG/ball mill/crushing	SABC
semi-autogenous	SAG
SGS Lakefield Research Ltd	SGS
SGS Minerals Services	SGS
Sheridan Platinum Group Ltd	SPG
Society for Mining, Metallurgy, and Explortion	SME
Standards Council of Canada	SCC
tailings management facility	TMF
Underwriters Laboratories of Canada	ULC
US Geological Survey	USGS
Vale Canada Limited	Vale
varitextured gabbro	VGAB
Xstrata Process Support	XPS

1.0 SUMMARY

1.1 INTRODUCTION

North American Palladium Ltd. (NAP) retained Tetra Tech to prepare a National Instrument 43-101 (NI 43-101) technical report on the Lac des Iles Mine and a prefeasibility study to support the reserves and resources of the Offset Zone. The Lac des Iles Property (the Property) is located approximately 85 km northwest of Thunder Bay, Ontario.

Table 1.1 summarizes the resources and reserves on the Property. The effective date of the mineral resources and reserves is March 31, 2012. The resources are exclusive of the reserves.

Table 1.1 Lac des Iles Mine Mineral Resources and Reserves

Category	Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pd (oz)
Mineral Reserves								
Proven								
Offset Zone	2.5	5,632,000	4.34	0.30	0.30	0.11	0.08	786,563
Roby Underground	5.8	420,000	6.38	0.42	0.34	0.08	0.07	86,149
Open Pit	1.8	722,000	1.99	0.21	0.22	0.11	0.10	46,192
Probable								
Offset Zone	2.5	2,109,000	4.17	0.30	0.29	0.10	0.08	282,609
Total Reserve	**-**	**8,883,000**	**4.21**	**0.30**	**0.29**	**0.11**	**0.08**	**1,201,513**
Mineral Resources								
Measured								
Offset Zone	2.5	4,942,550	4.56	0.33	0.31	0.12	0.10	724,094
Open Pit	1.8	1,971,000	2.00	0.24	0.15	0.05	0.07	126,735
Stockpile		83,000	1.63	0.17	0.14	0.08	0.06	4,350
Total Measured	**-**	**6,996,550**	**3.80**	**0.31**	**0.27**	**0.12**	**0.10**	**855,179**
Indicated								
Offset Zone	2.5	9,556,600	4.11	0.31	0.29	0.11	0.09	1,262,139
Roby Underground	5.8	1,269,000	7.16	0.41	0.33	0.08	0.06	292,116
Open Pit	1.8	2,565,000	2.20	0.24	0.18	0.07	0.08	181,422
Low-grade Stockpile	0.5	13,188,000	0.97	0.12	0.08	0.06	0.03	411,274
Total Indicated	**-**	**26,578,600**	**2.51**	**0.21**	**0.18**	**0.08**	**0.06**	**2,146,952**

table continues…

Category	Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pd (oz)
Measured + Indicated	**-**	**33,575,150**	**2.78**	**0.23**	**0.20**	**0.08**	**0.05**	**3,002,132**
Inferred								
Offset Zone	2.5	13,669,000	3.59	0.30	0.23	0.09	0.08	1,575,840
Total Inferred	**-**	**13,669,000**	**3.59**	**0.30**	**0.23**	**0.09**	**0.08**	**1,575,840**

Notes: Prepared in accordance with NI 43-101 Standards of Disclosure for Mineral Projects and the CIM classification system.

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

Mineral resources are exclusive of the mineral reserves.

The mineral resource for the Offset Zone was estimated as of June 30, 2012 by Todd McCracken, P.Geo, of Tetra Tech, an independent QP under NI 43-101. The effective date of the resource is March 31, 2012. The mineral resource calculation uses a minimum 2.5 g/t palladium resource block cut-off. The mineral resource estimate is based on the combination of geological modeling, geostatistics, and conventional block modelling (5 m by 5 m by 5 m blocks). Assay grade capping was determined not to be necessary. The Offset Zone resource models used the ordinary kriging (OK) grade interpolation method within a 3Dl block model with mineralized zones defined by wireframed solids. The quality assurance (QA)/quality control (QC) protocols and corresponding sample preparation and shipment procedures for the Offset Zone have been reviewed by Tetra Tech. The following metal price assumptions were used: US$675/oz palladium, US$1,675/oz platinum, US$1,750/oz gold, US$8.00/lb nickel, and US$3.50/lb copper. A US$/CDN$ exchange rate of US$1.00 = CDN$1.00 was also applied.

Mineral reserves for the Offset Zone were estimated by Todd McCracken, P. Geo.; William Richard McBride, P.Eng.; Todd Kanhai, P.Eng.; and Philip Bridson, P.Eng. of Tetra Tech, independent QPs within the meaning of NI 43-101. The mineral reserves were estimated from drilling completed to March 31, 2012. Reserves were estimated to the 990 Mine Level (4,490 m elevation), a maximum depth of 1,017.5 m. The following metal price assumptions were used for reserves estimation: US$675/oz palladium, US$1,675/oz platinum, US$1,750/oz gold, US$8.00/lb nickel, and US$3.50/lb copper. A US$/CDN$ exchange rate of US$1.00 = CDN$1.00 was also applied. An average production rate of 3,500 t/d was used to determine the reserves. The following recoveries were used in the assumptions to determine the reserves: 80.45% palladium, 71.30% platinum, 83.97% gold, 43.13% nickel, and 88.55% copper.

The mineral resource estimate for the Roby Zone open pit and stockpiles were estimated as of June 30, 2010 by Scott Wilson RPA and updated by David N. Penna, P.Geo., an employee of Lac des Iles Mining Ltd. (LDIM) and a qualified person (QP) under NI 43-101 to reflect: (i) additions to mineral reserves in the Roby Zone as a result of a lower cut-off palladium grade; (ii) depletion from production up to March 31, 2012, and (iii) mineral reserves from the crown pillar (supported by an internal engineering report). The following cut-off grades were used: (i) 1.8 g/t palladium equivalent (PdEq) for the Roby open pit, within an optimized pit shell run below the current pit survey; (ii) 1.9 g/t PdEq for the mine stockpiles; and (iii) 5.8 g/t PdEq for the underground Roby Zone. These cut-off grades were determined under the assumption that production would take place at a rate of 14,000 t/d. Metal price assumptions of US$350/oz palladium, US$1,400/oz platinum, US$850/oz gold, US$6.50/lb nickel, and US$2.00/lb copper were used in the estimation of cut-off grade. A US$/CDN$ exchange rate of 1.11 was also applied.

Numbers may not add due to rounding.

Currently, the mine expansion plans consist of the following:

- NAP is currently sinking a shaft 825 m from surface to transition underground operations from mining via ramp to mining via shaft as part of Phase I of the Offset Project. Once completed, operations will benefit from an increased underground mining rate. NAP expects to start hoisting Offset Zone ore from this new shaft by the end of Q3 2013, with full operational benefits of the shaft realized in 2014.
- NAP is setting up underground infrastructure to mine the Offset Zone Phase I using a high-volume bulk mining method (longhole stoping with cemented rock fill). This includes expanding the ramp at depth from the Roby Zone to the Offset Zone, building new mine levels in the Offset Zone, and setting up mining stopes for production in the Offset Zone. This underground development is critical to the mine expansion and is being developed concurrently with the shaft, as it will effectively provide increased ore feed for the new shaft.

The surface infrastructure, built to service the expanded underground operations, is now essentially complete. The headframe, the main substation, the hoist house building, the service hoist, and the auxiliary hoist are all fully operational and complete. Work is being completed on the first Offset Zone stope and shaft sinking is in progress (www.nap.com).

The effective date of this report is January 31, 2013 and the effective date of the resource and reserve estimate is March 31, 2012.

For the purposes of this study, all currencies are expressed in Canadian dollars, unless otherwise specified.

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

The Property is located at latitude 49°10' north, longitude 89°37' west, 85 km northwest of the community of Thunder Bay in northwestern Ontario (Figure 1.1). 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. As of the effective date of this report, all claims and leases on the Property are in good standing.

A 5% net smelter return (NSR) is paid to Sheridan Platinum Group Ltd. (SPG).

Figure 1.1 Property Location Map

1.3 GEOLOGY AND MINERALIZATION

The Lac des Iles Mine area is underlain by mafic to ultramafic rocks of the Archean Lac des Iles Intrusive Complex (LDI-IC). The LDI-IC has intruded granites and greenstones of the Wabigoon Subprovince of the Superior Province. The LDI-IC is the largest of a series of mafic and ultramafic intrusions that occur along the boundary of the Wabigoon-Quetico Subprovince and collectively define a 30 km in diameter circular pattern. The Lac des Iles Mine lies in the southern portion of the LDI-IC, in a roughly elliptical intrusive package measuring 3 km long by 1.5 km wide, termed the Mine Block Intrusive (MBI).

The MBI hosts a number of platinum group element (PGE) deposits; the most important of these are the Roby Zone and the Offset Zone. The MBI comprises rocks with a very wide range of textures, and mafic and ultramafic compositions.

The Offset Zone is a high-grade zone located below the Offset Fault structure where it is displaced down and approximately 300 m to the west. The Offset Zone can be split into three horizons and has been divided into three subzones: the High Grade Subzone, the Mid Subzone, and the Footwall Subzone.

The High Grade Subzone mineralization is stratabound, along the contact between the East Gabbro (EGAB) and the mineralized heterolithic gabbro breccias (HGABBX). The width of the mineralization varies from 4 to 30 m, with an average of 15 m.

The Mid Subzone is proximal to the High Grade Zone, generally sharing a common boundary in the centre sections, and then splitting away near the top and bottom areas. Palladium grades within the Mid Subzone can approximate those high grades found within the High Grade Zone. Apparent widths can vary from 4 to 90 m, with an average of 15 m. Approximately 4% of the zone is occupied by late dikes (dilution) and less than 1% is occupied by shears and faults.

The Footwall Subzone is a stand-alone band of higher-grade mineralization that can be defined based on higher-grade intersections within the footwall mineralization. This subzone, located approximately 2 to 40 m from the Mid Subzone, was interpreted based on vertical continuity seen in the drillhole intersections. It is discontinuous and sinuous in plan and has less of a defined areal extent than the other zones. Apparent widths can vary from 4 to 20 m, with an average of 7 m. Approximately 1% of the zone is occupied by late dikes (dilution).

PGE and base metal mineralization at the Lac des Iles Mine appear to be dominantly stratabound along the contact between the EGAB and the mineralized HGABBX. Within the HGABBX, there is a high-grade core typically constrained to an easily recognized pyroxenite unit. Visible PGE mineralization is rare and its occurrence is difficult to predict. In general, economic PGE grades are anticipated within gabbroic to pyroxenitic rocks. PGE grades show varying degrees of correlation with nickel and copper grades. Some parts of the deposit have very good PGE-copper-nickel (and sulphur) correlations, while others show no significant, positive correlation between PGE and copper or nickel.

1.4 MINERAL RESOURCE ESTIMATE

A mineral resource was generated using 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. A total of 59,995 palladium assay intervals were used in the estimation process. Grade capping was reviewed and applied were necessary and compositing was completed post-capping on a 1 m downhole length interval.

Grade estimations were completed using OK, inverse distance squared (ID^2), and nearest neighbour (NN) processes for six elements on a multi-pass search parameter into 5 m by 5 m by 5 m blocks.

All drilling, logging, sampling, and analytical procedures meet industry standards and are acceptable to use in the resource estimation.

Table 1.2 is a summary of the mineral resource at a 2.5 g/t cut-off. The resources are inclusive of any reserve material.

Table 1.2 Offset Zone Mineral Resource Summary

Category	Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Co (%)	Pd (oz)
Measured	2.5	10,436,550	4.67	0.33	0.32	0.12	0.09	0.01	1,566,989
Indicated	2.5	11,615,600	4.18	0.32	0.30	0.11	0.09	0.01	1,561,026
Measured + Indicated	2.5	22,052,150	4.41	0.32	0.31	0.12	0.09	0.01	3,128,015
Inferred	2.5	13,669,000	3.59	0.30	0.23	0.09	0.08	0.01	1,577,676

Table 1.3 is a summary of the minerals resource above the 4,490 elevation (990 Level), which is the current Phase I focus of NAP.

Table 1.3 Offset Zone Mineral Resources above 4,490 Elevation (990 Level)

Classification	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pd (oz)
Measured	9,363,030	4.68	0.34	0.32	0.12	0.09	1,408,816
Indicated	6,684,300	4.03	0.32	0.27	0.10	0.08	866,070
Measured + Indicated	16,047,330	4.41	0.33	0.30	0.11	0.09	2,274,886
Inferred	9,620,000	2.66	0.29	0.22	0.09	0.07	822,693

1.5 MINERAL RESERVE ESTIMATE

The mineral reserve estimation is solely based upon a NAP provided 3D mining wire frame, from which NAP created individual stoping blocks. This mining wireframe model is a reflection of the mining method selection, the stope sizing parameters, and the geotechnical considerations that assisted in the definition of the mining units. Dilution and recovery were then applied to those mining units.

The hanging wall is a hard boundary between the barren rock and the higher-grade palladium domain. The 2.5 g/t palladium cut-off grade then defines the footwall for the mining wireframe.

The mineral reserves tabulated in Table 1.4 include the Phase I area of the Offset Zone above the 990 Level (elevation 4,490) and only the portion of the Offset Zone

contained within the stope wireframe. The resources below the 990 Level were judged by NAP to require additional definition to ensure they can support an independent later phase capital investment program before they can be incorporated into the reserves. NAP intends to conduct more drilling below the 990 Level before converting those resources into reserves.

The diluted mineable mineral reserves contained within the stope wireframe at a 2.5 g/t palladium cut-off grade are 7,741,000 t grading 4.3 g/t palladium, 0.3 g/t platinum, 0.3 g/t gold, 0.08% copper and 0.11% nickel as summarized in Table 1.4.

Table 1.4 Phase I Offset Zone Mineral Reserves at 2.5 g/t Palladium Cut-off

Classification	Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pd (oz)
Proven	2.50	5,632,000	4.34	0.30	0.30	0.11	0.08	786,535
Probable	2.50	2,109,000	4.17	0.30	0.29	0.10	0.08	282,644
Total Reserve	**-**	**7,741,000**	**4.30**	**0.30**	**0.30**	**0.11**	**0.08**	**1,069,179**

1.6 MINERAL PROCESSING AND METALLURGICAL TESTING

To conserve energy, the Lac des Iles Mine is currently running on an intermittent operating schedule, resulting from an irregular ore supply to the mill. The mine currently runs 14-day campaigns at 13,000 t/d followed by 14 days off. The mill design capacity is 15,000 t/d.

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 however were based on a primary grind size that was too large to achieve an optimum palladium flotation recovery. They were also based on mill throughputs which are lower 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 of 80% passing 38 µm would achieve the optimum palladium flotation recovery. A semi-autogenous (SAG)/ball mill/crushing (SABC) grinding circuit arrangement with a Vertimill™ for tertiary grinding is used to approach this optimum grind size. The investigation by XPS also determined that the Roby and Offset zones behaved similarly with the optimum flotation parameters.

1.7 MINING

The Lac des Iles Mine is currently operating with over 1.2 Mt of ore budgeted to come from the Offset Zone in 2013. Most of the tonnage will come via truck haulage up the main ramp with production from the new No.1 Shaft scheduled to commence by the end of Q3 2013. The No. 1 Shaft is being sunk to the 825 Level and shaft

sinking has reached the 533 m mark. In place are the headframe and the hoist house with associated hoists. Two of the shaft stations were developed from the internal ramp system and the ramp itself has already progressed to the 825 Level by contractor development crews.

The chosen mining method is longhole stoping with cemented rock fill. This method takes into consideration a growing orebody, the future mining of potential expansion targets, and updated information pertaining to geotechnical effects on the Offset Zone's rock mass. Substantial crown pillars are incorporated into the mining plan to provide an earlier production stream and to allow for flexible mining using various mine sequences with enough tonnage to allow for high productivities. Dividing the Phase I area of the Offset Zone into three independent mining areas by means of the 750 Level and 840 Level crown pillars ensures a nominal 3,500 t/d production rate is achievable.

The stope plans and sequencing, along with the development plans, have been applied keeping in mind the future growth potential of the remaining resource. The concept will be to leverage the development of additional reserves through extension of the existing mine development to readily access this anticipated growth area, as well as to provide the exploration platform required to discover new material that would add to the global resource.

1.8 INFRASTRUCTURE

The Lac des Iles Mine has been operating as an open pit since 1993 and became an underground operation in 2006. The Offset Zone Project commenced with preparations for shaft sinking in 2010. To date, the Offset Zone Project has added to the site infrastructure with the completion of the hoist house, installation of all hoists including commissioning of the service and auxiliary hoist, construction of the headframe, and construction of the surface electrical substation. Accordingly, the site infrastructure that has supported these historical efforts is seen by LDIM to be sufficient to continue to meet the ongoing needs of the Offset Zone Phase I Project as well as to meet the needs of the future underground mining plan.

1.9 ENVIRONMENTAL

Annual monitoring for physical and chemical stability of water surrounding the Property is ongoing and will be continued 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 Lac des Iles Mine on the surrounding ecosystem.

LDIM complies in all material respects with current legislation, and it has all necessary approvals and licenses for the operation of the mine and for all planned expansion projects.

The mine 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 to ensure that any concerns, issues, or questions are addressed.

1.10 CAPITAL AND OPERATING COSTS

1.10.1 CAPITAL COSTS

The total capital cost required for the Lac des Iles Mine Phase I is estimated at $110.8 million and is distributed as follows:

- development costs consisting of ramps, level accesses, and raises ($37.4 million)
- underground maintenance and parts ($11.6 million)
- surface and underground construction ($17.1 million)
- procurement ($2.2 million)
- underground equipment ($0.04 million)
- electrical and process control ($2.5 million)
- engineering ($1.0 million)
- Owner's costs and indirects ($3.8 million)
- sustaining capital for the LOM ($35.2 million).

The Offset Zone Phase I capital costs are itemized as initial and sustaining, as summarized in Table 1.5.

Table 1.5 Capital Cost Summary ($ million)

	Initial Capital (2013 to 2014)	Sustaining Capital (2013 to 2018)	Total (2013 to 2018)
Initial Capital	75.6	0.0	75.6
Sustaining Capital	0.0	35.2	35.2
Total	**75.6**	**35.2**	**110.8**

The initial capital by activity is shown in Table 1.6 and sustaining capital by activity is shown in Table 1.7.

Table 1.6 Initial Capital by Activity ($ million)

Activity	2013	2014	Total
Development	24.2	13.2	37.4
Underground Maintenance and Parts	11.6	0.0	11.6
Surface and Underground Construction	17.1	0.0	17.1
Procurement	2.2	0.0	2.2
Underground Equipment	0.04	0.0	0.04
Electrical and Process Control	2.5	0.0	2.5
Engineering	1.0	0.0	1.0
Owner's Costs and Indirects	3.8	0.0	3.8
Total	**62.4**	**13.2**	**75.6**

Table 1.7 Sustaining Capital by Activity ($ million)

Activity	2013	2014	2015	2016	2017	2018	Total
Exploration	3.4	3.2	0.0	0.0	0.0	0.0	6.6
Mine and Maintenance	5.8	2.6	2.0	2.0	2.0	0.0	14.4
Mill	0.6	1.0	0.0	0.0	0.0	0.0	1.6
Tailings Management Facility (TMF) Phase I	6.5	3.0	3.0	0.0	0.0	0.0	12.5
Administration	0.05	0.0	0.0	0.0	0.0	0.0	0.05
Total	**16.4**	**9.8**	**5.0**	**2.0**	**2.0**	**0.0**	**35.2**

1.10.2 *Operating Costs*

The total operating cost for the Lac des Iles Mine Phase I was determined to be $427 million. The total unit operating cost was calculated at $55.16/t mined based on an annual mine run rate 3,500 t/d. A summary of the operating costs, organized by category, is shown in Table 1.8 and are expressed in total Canadian dollars and unit per tonne mined.

Table 1.8 Phase I Operating Cost Summary

Area	Cost ($ million)	Percentage (%)	Unit Cost ($/t mined)
Surface	65.9	16	8.51
Mine	178.6	41	23.07
Mill	118.5	28	15.31
General and Administrative (G&A) Mine	64.0	15	8.27
Total	**427.0**	**100**	**55.16**

1.11 ECONOMIC EVALUATION

An economic evaluation of the Lac des Iles Mine was prepared by LDIM and audited by Tetra Tech. The financial results are summarized in Table 1.9.

Table 1.9 Summary of Phase I Financial Results

Area	Unit	Value	Cost ($/t mined)
Net Present Value (NPV) (5%)	$ million	138	-
Internal Rate of Return (IRR)	%	63.7	-
Payback	years	2.69	-
Gross Revenue	$ million	800	103.40
Smelting, Refining, and Transportation	$ million	(54)	(7.01)
Net NSR	$ million	746	96.39
Royalty	$ million	(37)	(4.82)
Net Revenue	$ million	709	91.57
Operating Cost	$ million	(427)	(55.16)
Operating Margin	$ million	282	36.41
Capital Cost	$ million	(111)	(14.32)
Pre-tax Cash Flow	$ million	171	22.09
Taxes	$ million	0	0.00
Post-tax Cash Flow	$ million	171	22.09

The following financial results were established for the Lac des Iles Mine:

- Gross metal revenue was determined from mill recovered metal grades and metal prices. The gross revenue was determined to be $800 million. The base case metal prices are listed in Table 1.11.
- NSR revenue was determined by metal grades and metal prices and adjusted for smelting, refining, and transportation costs. The NSR revenue was determined to be $746 million.
- Pre-tax cash flow, determined as net revenue less operating and capital costs, is $171 million. The pre-tax NPV of the Lac des Iles Mine is $138 million at a discount rate of 5%.
- The IRR was determined to be 63.7%.
- The payback period is 2.69 years or 32 months.
- The tax calculation performed by LDIM indicates no taxes are payable during Phase I. Therefore the post-tax results are the same as the pre-tax results.

Metal production from the Lac des Iles Mine is presented in Table 1.10 and base case metal prices are shown in Table 1.11. Table 1.12 shows the foreign exchange used in the financial model.

Table 1.10 Metal Production from Lac des Iles

Item	Unit	Value
Total Material to Mill	t	7,741,000
Average Annual Material to Mill	t	1,290,167
Phase I	years	6
NSR		
Average NSR	$/t mined	96.39
Average Grade		
Pd	g/t	4.30
Pt	g/t	0.30
Au	g/t	0.30
Cu	%	0.08
Ni	%	0.11
Total Mill Production		
Pd	oz	860,107
Pt	oz	53,764
Au	oz	61,663
Cu	lb	12,415,097
Ni	lb	8,170,127
Average Annual Mill Production		
Pd	oz	143,351
Pt	oz	8,961
Au	oz	10,277
Cu	lb	2,069,183
Ni	lb	1,361,688

Table 1.11 Base Case Metal Prices used in the Financial Model

Pd (US$/oz)	Pt (US$/oz)	Au (US$/oz)	Cu (US$/lb)	Ni (US$/lb)
675	1,675	1,750	3.50	8.00

Table 1.12 Foreign Exchange used in the Financial Model

Currency Exchange	Rate
CDN$ = US$	1.00

1.12 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- foreign exchange
- palladium price
- palladium grade
- operating cost
- capital cost.

Sensitivity analyses are presented graphically as financial outcomes in terms of NPV and IRR.

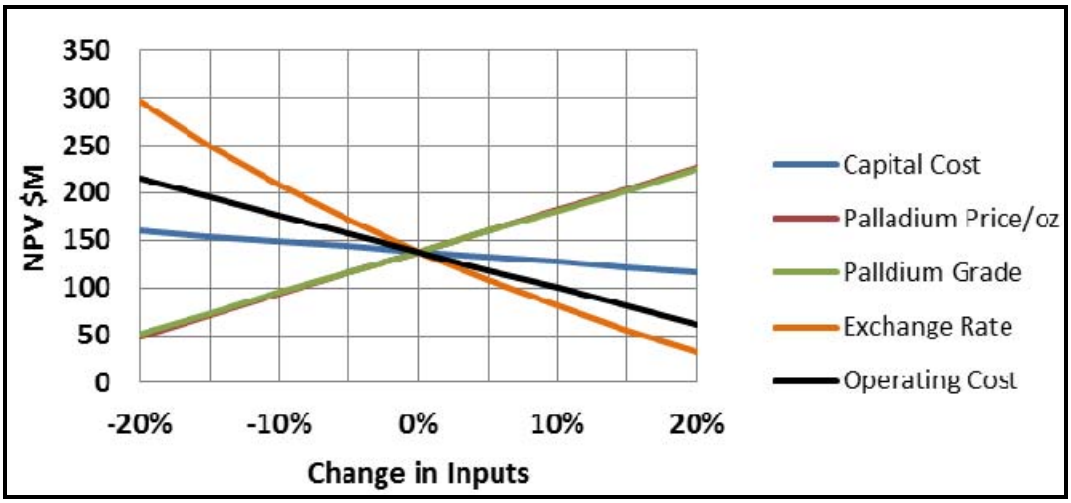
Table 1.13 shows the corresponding sensitivity parameters and the resulting NPVs.

Table 1.13 Sensitivity Parameters and NPV at 5% ($ million)

Parameter	Foreign Exchange	Palladium Price	Palladium Grade	Operating Cost	Capital Cost
-20%	295	49	52	215	160
-10%	208	93	95	176	149
Base	138	138	138	138	138
+10%	81	182	181	99	127
+20%	33	227	224	61	116

Figure 1.2 graphically illustrates that the Lac des Iles Mine NPV is most sensitive to exchange rate, and in deceasing order, palladium price, palladium grade, operating cost, and capital cost.

Figure 1.2 NPV Sensitivity Analysis



Note: Palladium price and palladium grade results are almost identical and show as one line on the graph

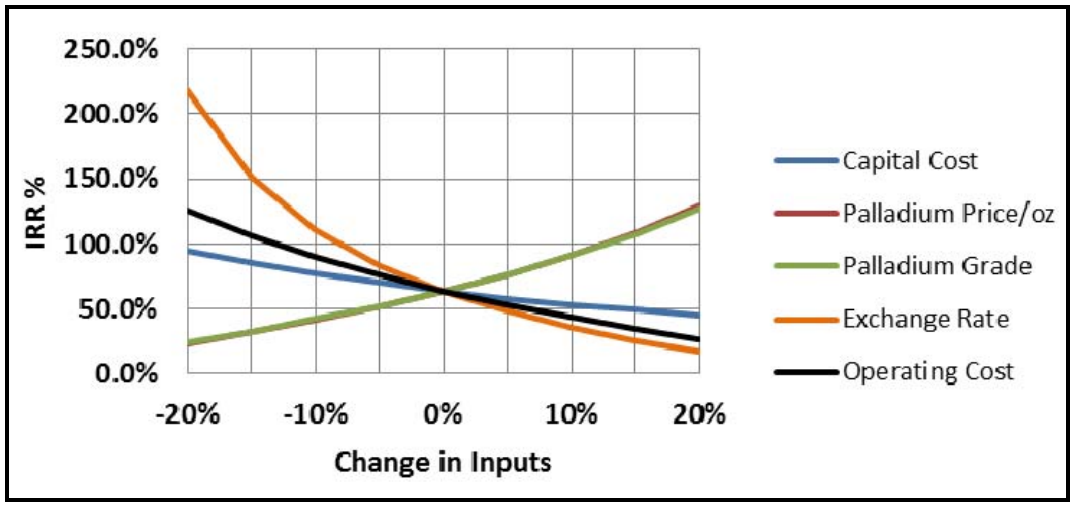
Table 1.14 shows the corresponding sensitivity parameters and the resulting IRRs.

Table 1.14 Sensitivity Parameters and IRR (%)

Parameter	Foreign Exchange	Palladium Price	Palladium Grade	Operating Cost	Capital Cost
-20%	218.8	23.0	24.0	126.4	95.1
-10%	110.7	41.5	42.1	90.0	77.0
Base	63.7	63.7	63.7	63.7	63.7
+10%	36.0	91.6	90.7	43.3	53.4
+20%	16.9	129.3	126.7	26.7	45.2

Figure 1.3 graphically illustrates that the Lac des Iles Mine IRR is most sensitive to exchange rate, and in deceasing order, palladium price, palladium grade, operating cost and capital cost.

Figure 1.3 IRR Sensitivity Analysis



Note: Palladium price and palladium grade results are almost identical and show as one line on the graph

1.13 RECOMMENDATIONS AND CONCLUSIONS

1.13.1 EXPLORATION

- Exploration on the Offset Zone is recommended to focus on infilling the current resource at approximately 20 m centres, in order to improve the understanding of the deposit and upgrade the material in the various resource categories.
- Additional exploration should concentrate on expanding the mineral resource by testing the strike extension to the south of the shaft, as well as both the up dip and down dip extensions.
- A vast majority of the drilling will require underground exploration drifts and stations in order to provide suitable drilling platforms.

1.13.2 METALLURGY

- As recommended in the test work by XPS (XPS 2010), optimization should be completed on the Vertimills™ to achieve the final grind size of P_{80} -38 µm prior to flotation.
- To help optimize flotation recoveries, the sump and Vertimill™ capacity should be created to accept the entire ball mill hydrocyclone overflow into the Vertimills™ rather than divert 20% of the flow to flotation, and thereby optimize the grind size on all of the feed prior to flotation.
- NAP should consider conducting further test work to determine how the Lac des Iles Mine feed would react in a mill-float-twice (MF2) flowsheet, as it is commonly used in South Africa for platinum flotation.

1.13.3 MINING

- With 16.05 Mt of Measured and Indicated Resources above the 990 Level providing 7.74 Mt of mineable reserves for Phase I, a study of the remaining resources above the 990 Level would allow identification of alternative mining options for those resources that fall outside of the NAP provided mineable reserve envelope. As there is an additional 9.62 Mt of Inferred Resources above the 990 Level, it is recommended that the study be kept to a preliminary economic assessment (PEA) level and the net total resources be studied. Exclusive of reserves, the total resources considered for this PEA would then be 17.93 Mt.
- With the resultant 6.00 Mt of Measured and Indicated Resources below the 990 Level, and a further 4.05 Mt of Inferred Resources, a separate PEA on the 10.05 Mt of resources below the 990 Level would allow identification of alternative mining and access options.
- The geotechnical assessments of the crown and stope sequences above the 990 Level should be extended below to define the 990 Level crown pillar requirement, to define the stope sequence at depth, and to assess any potential for fault slip once mining extraction across the total Offset Zone nears maturity (expected to be during removal of the crown pillars).
- The ventilation requirements were determined by a method that is of prefeasibility study nature and it is recommended that further detailed ventilation design is completed to confirm these ventilation quantities.
- With almost all ore trucked up ramp to the dump on the 650 Level in order to feed the shaft ore handling system, it is recommended that a trade-off study be conducted evaluating the added cost of deepening the shaft to just below the 990 Level, versus the cost of retaining truck haulage of the ore and its impact on the ventilation requirements. Consideration should be given to all resources above the 990 Level and not just the mineable reserves.
- With an expectation for an eventual increase in mineable strike length above the 990 Level for the Offset Zone, a trade-off study should be considered to evaluate the cost of trucking the cemented rock fill large horizontal distances versus other distribution methods and fill mediums. Other distribution methods may include distribution of cemented rock fill by conveyor or alternatively through a pipeline by implementing paste fill.
- Once the trade-off studies and the PEA studies have been performed and the results from added exploration drilling are available, completion of a prefeasibility study should be considered that examines the resources above and below the 990 Level in order to tie together all of the studies and added exploration drilling results.
- There is a potential for oscillations in production that may limit operating throughput. It is recommended that the enterprise constraints (i.e. grizzly, ore passes and hoist capacity, ventilation, etc.) be further evaluated so that contingency plans can be identified. It is further recommended that this

study of enterprise constraints be performed on an annual basis during the annual plan preparation process.

- The 7.74 Mt of mineable reserves provide a relatively short production profile for the Offset Zone Phase I. A study should be considered, examining whether the expected increase in annual maintenance costs and decrease in fleet availability jeopardize the annual production plans realizing that the mobile fleet plan retains the same mobile mining equipment throughout Phase I.

1.13.4 ECONOMIC EVALUATION

- The Lac Des Iles Mine Phase I has a robust NPV at 5% of $138 million, an IRR of 63.7%, and a payback on capital of 2.69 years.
- Palladium, the Lac des Iles Mine's major metal, contributes $525 million or 66% of the gross revenue.
- The palladium metal price used in this study is US$675/oz. The January 31, 2013 spot price for palladium US$738.
- The financials used a conservative palladium metal price which, if the current spot price were to be incorporated, would increase the NPV (at 5%) to over $200 million.
- The Lac Des Iles Mine Offset Zone reserves are economically viable.

2.0 INTRODUCTION

Tetra Tech was commissioned by NAP to complete a technical report on the Lac des Iles Mine and a prefeasibility study on the Offset Zone. Tetra Tech has prepared this report in accordance with the guidelines provided in the NI 43-101 Standards of Disclosure for Mineral Projects.

This report was prepared by Tetra Tech at the request of Mr. David Peck, Head of Exploration for NAP.

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

The effective date of this report is January 31, 2013. The effective date of the resource and reserve estimate is March 31, 2012.

A summary of qualified persons (QPs) responsible for each section of this report is provided in Table 2.1. The following QPs completed a site visit of the Property:

- Todd McCracken, P.Geo., visited the site from May 23 to 25, 2012.
- William Richard McBride, P.Eng., visited the site from November 5 to 8, 2012.
- Todd Kanhai, P.Eng., visited the site from November 5 to 7, 2012.

Table 2.1 Summary of QPs

Report Section		Company	QP
1.0	Summary	Tetra Tech	Sign-off by Section
2.0	Introduction	Tetra Tech	Todd McCracken, P.Geo.
3.0	Reliance on Other Experts	Tetra Tech	Todd McCracken, P.Geo.
4.0	Property Description and Location	Tetra Tech	Todd McCracken, P.Geo.
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Tetra Tech	Todd McCracken, P.Geo.
6.0	History	Tetra Tech	Todd McCracken, P.Geo.
7.0	Geological Setting and Mineralization	Tetra Tech	Todd McCracken, P.Geo.
8.0	Deposit Types	Tetra Tech	Todd McCracken, P.Geo.
9.0	Exploration	Tetra Tech	Todd McCracken, P.Geo.
10.0	Drilling	Tetra Tech	Todd McCracken, P.Geo.
11.0	Sample Preparation, Analyses, and Security	Tetra Tech	Todd McCracken, P.Geo.
12.0	Data Verification	Tetra Tech	Todd McCracken, P.Geo.

table continues…

Report Section		Company	QP
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Todd Kanhai, P.Eng.
14.0	Mineral Resource Estimates	-	-
	14.1 Offset Zone	Tetra Tech	Todd McCracken, P.Geo.
	14.2 Roby and Open Pit Zones	LDIM	David N. Penna, P.Geo.
15.0	Mineral Reserve Estimates	-	-
	15.1 Offset Zone	Tetra Tech	William Richard McBride, P.Eng.
	15.2 Roby and Open Pit Zones	NAP	Kevin Small, P.Eng.
16.0	Mining Methods	Tetra Tech	William Richard McBride, P.Eng.
17.0	Recovery Methods	Tetra Tech	Todd Kanhai, P.Eng.
18.0	Infrastructure	Tetra Tech	William Richard McBride, P.Eng.
19.0	Market Studies and Contracts	Tetra Tech	Philip Bridson, P.Eng.
20.0	Environmental Studies, Permitting, and Social or Community Impact	Tetra Tech	Todd McCracken, P.Geo.
21.0	Capital and Operating Costs	Tetra Tech	Philip Bridson, P.Eng.
22.0	Economic Analysis	Tetra Tech	Philip Bridson, P.Eng.
23.0	Adjacent Properties	Tetra Tech	Todd McCracken, P.Geo.
24.0	Other Relevant Data and Information	Tetra Tech	Todd McCracken, P.Geo.
25.0	Interpretation and Conclusions	Tetra Tech	Sign-off by Section
26.0	Recommendations	Tetra Tech	Sign-off by Section
27.0	References	Tetra Tech	Sign-off by Section

2.1 UNITS OF MEASUREMENT

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

3.0 RELIANCE ON OTHER EXPERTS

The QPs who prepared this report relied on information provided by experts who are not QPs. The relevant QPs believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

- Todd McCracken, P.Geo., relied upon Cameron McLean, Exploration Manager of NAP for matters pertaining to mineral claims and mining leases as disclosed in Section 4.0, and Mike Wanecki, Environmental Superintendent of the Lac des Iles Mine, for matters found in Section 20.0.
- Philip Bridson, P.Eng., relied upon Ashvin Lugan, Director, Treasury of NAP concerning tax matters relevant to this report in Section 22.0, as well as information pertaining to market studies and contracts in Section 19.0. Mr. Lugan, CFA, CGA has over 12 years of public company experience in mining and financial services.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 DESCRIPTION

The Property is located at latitude 49°10' north, longitude 89°37' west, 85 km northwest of the community of Thunder Bay in northwestern 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 3,513.2 ha (Table 4.1). Contiguous with these leases are 54 mineral claims held 100% by LDIM (consisting of 331 claim units) covering 5,109.8 ha, for a total area of 8,623.0 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**	**-**

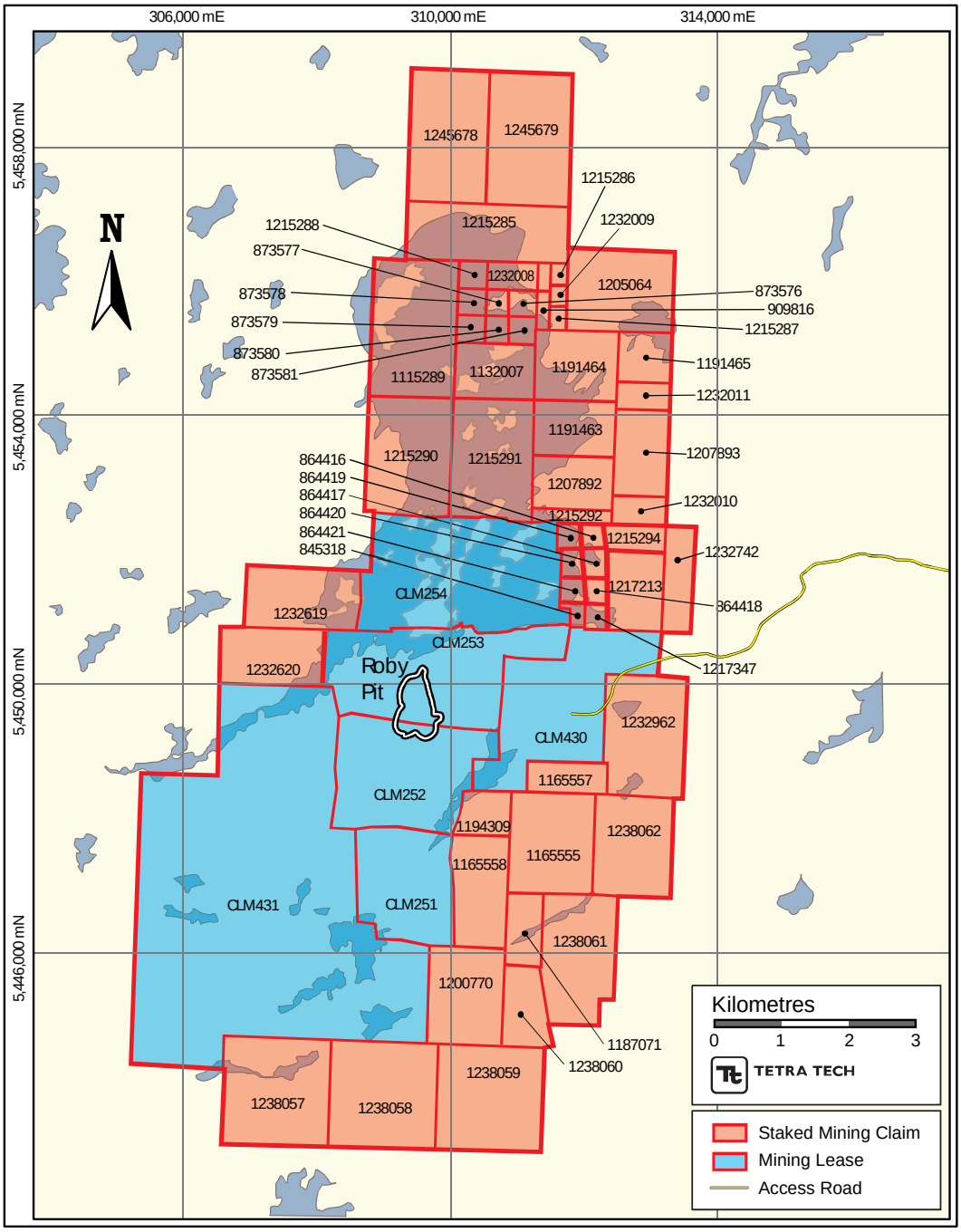
Table 4.2 Claim Summary

Township/Area	Claim Number	Unit	Area (ha)	Recording Date	Claim Due Date	Work Required ($)
Lac des Iles	845318	1	9.0	1985-Dec-04	2015-Dec-04	400
Lac des Iles	864416	1	12.9	1985-Nov-19	2015-Nov-19	400
Lac des Iles	864417	1	16.0	1985-Nov-19	2015-Nov-19	400
Lac des Iles	864418	1	12.6	1985-Nov-19	2015-Nov-19	400
Lac des Iles	864419	1	12.2	1985-Nov-19	2015-Nov-19	400
Lac des Iles	864420	1	13.1	1985-Nov-19	2015-Nov-19	400
Lac des Iles	864421	1	13.0	1985-Nov-19	2015-Nov-19	400
Lac des Iles	873576	1	16.7	1986-May-05	2015-May-05	400
Lac des Iles	873577	1	13.2	1986-May-05	2015-May-05	400
Lac des Iles	873578	1	16.5	1986-May-05	2015-May-05	400
Lac des Iles	873579	1	17.7	1986-May-05	2015-May-05	400
Lac des Iles	873580	1	14.1	1986-May-05	2015-May-05	400
Lac des Iles	873581	1	16.6	1986-May-05	2015-May-05	400
Lac des Iles	909816	1	11.2	1986-May-16	2015-May-16	400
Lac des Iles	1165555	12	189.0	1992-Mar-06	2015-Mar-06	4,800
Lac des Iles	1165557	3	55.5	1992-Mar-06	2015-Mar-06	1,200

table continues…

Township/Area	Claim Number	Unit	Area (ha)	Recording Date	Claim Due Date	Work Required ($)
Lac des Iles	1165558	8	134.9	1992-Mar-06	2015-Mar-06	3,200
Lac des Iles	1187071	4	58.0	1994-Dec-02	2015-Dec-02	1,600
Lac des Iles	1191463	6	99.3	1993-Aug-23	2015-Aug-23	2,400
Lac des Iles	1191464	9	127.3	1993-Aug-23	2015-Aug-23	3,600
Lac des Iles	1191467	4	59.8	1994-Mar-25	2015-Mar-25	1,600
Lac des Iles	1194309	4	51.0	1991-Sep-09	2015-Sep-09	1,600
Lac des Iles	1200770	11	162.1	1994-Dec-02	2015-Dec-02	4,400
Lac des Iles	1205064	12	191	1999-Jul-20	2015-Jul-20	4,800
Lac des Iles	1207892	6	96.3	1995-Feb-03	2015-Feb-03	2,400
Lac des Iles	1207893	6	103.8	1995-Feb-03	2015-Feb-03	2,400
Lac des Iles	1215285	12	206.5	1996-Jun-17	2015-Jun-17	4,800
Lac des Iles	1215286	1	8.4	1996-Jun-17	2015-Jun-17	400
Lac des Iles	1215287	1	9.0	1996-Jun-17	2015-Jun-17	400
Lac des Iles	1215288	1	16.7	1996-Jun-17	2015-Jun-17	400
Lac des Iles	1215289	15	260.6	1996-Jun-17	2015-Jun-17	6,000
Lac des Iles	1215290	15	223.2	1996-Jun-17	2015-Jun-17	6,000
Lac des Iles	1215291	15	218.4	1996-Jun-17	2015-Jun-17	6,000
Lac des Iles	1215292	3	24.5	1996-Jun-17	2015-Jun-17	1,200
Lac des Iles	1215294	3	35.7	1996-Jun-17	2015-Jun-17	1,200
Lac des Iles	1217213	6	99.0	1997-Feb-21	2015-Feb-21	2,400
Lac des Iles	1217347	1	12.5	1998-Apr-14	2015-Apr-14	400
Lac des Iles	1232007	6	98.6	1998-Feb-05	2015-Feb-05	2,400
Lac des Iles	1232008	2	39.0	1998-Feb-06	2015-Feb-06	800
Lac des Iles	1232009	1	8.4	1998-Apr-14	2015-Apr-14	400
Lac des Iles	1232010	2	34.4	1998-Apr-14	2015-Apr-14	800
Lac des Iles	1232011	2	31.6	1998-Apr-14	2015-Apr-14	800
Lac des Iles	1232619	8	151.3	1998-May-07	2015-May-07	3,200
Lac des Iles	1232620	8	128.8	1998-May-07	2015-May-07	3,200
Lac des Iles	1232742	4	69.7	1998-Apr-21	2015-Apr-21	1,600
Lac des Iles	1232962	12	212.5	1999-Jun-29	2015-Jun-29	4,800
Shelby Lake	1238057	16	256.0	1999-Jun-29	2015-Jun-29	6,400
Shelby Lake	1238058	16	257.6	1999-Jun-29	2015-Jun-29	6,400
Shelby Lake	1238059	16	252.9	1999-Jun-29	2015-Jun-29	6,400
Lac des Iles	1238060	6	77.3	1999-Jun-29	2015-Jun-29	2,400
Lac des Iles	1238061	15	199.6	1999-Jun-29	2015-Jun-29	6,000
Lac des Iles	1238062	15	174.0	1999-Jun-29	2015-Jun-29	6,000
Heaven Lake	1245678	15	234.3	2000-Dec-08	2015-Dec-08	6,000
Heaven Lake	1245679	15	246.5	2000-Dec-08	2015-Dec-08	6,000
Total	**-**	**331**	**5109.8**	**-**	**-**	**132,400**

Figure 4.2 Land Tenure of the Property



4.3 ROYALTIES

A 5% NSR is paid to 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 (www.stockhouse.com).

4.4 LIABILITIES AND PERMITS

All environmental liabilities are disclosed in Section 20.0, which covers the mine closure plan. All permits required to conduct work are also disclosed in Section 20.0.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

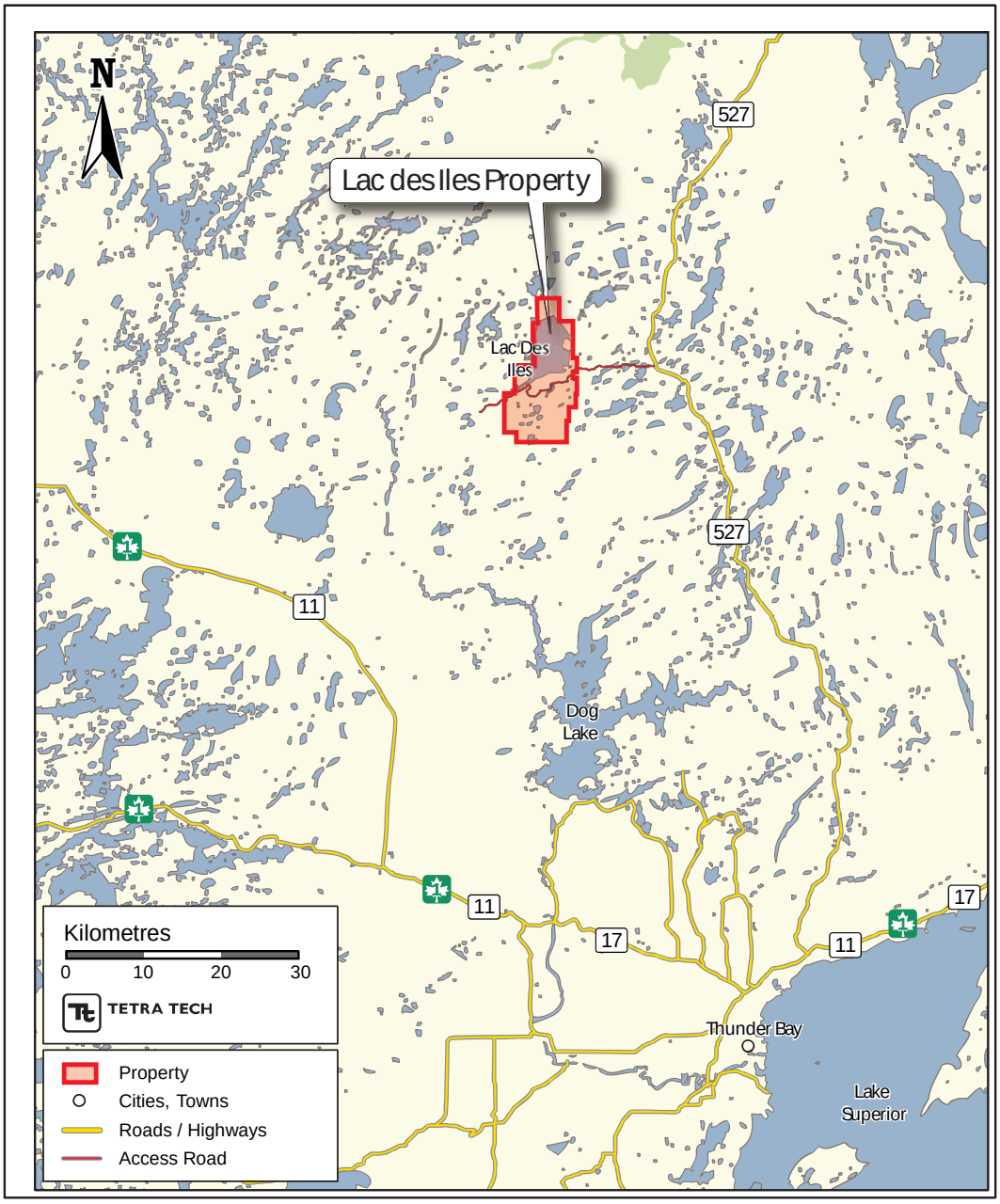
5.1 SITE TOPOGRAPHY, ELEVATION, AND VEGETATION

The Property is located in northwestern Ontario within the Superior Province of the Canadian Precambrian Shield, a boreal forest region typified by uplands, forested mostly by black spruce, birch, poplar, and jack pine, and low areas of numerous lakes and treed swamps. Drainage is poorly integrated and generally south to 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 masl, excluding mining operations. The mine, mill, and tailings impoundment area lie in the Boreal Forest eco-region, characterized by typical northern Ontario forest with numerous lakes and beaver swamps.

5.2 ACCESS

The Property is located approximately 90 km northwest of Thunder Bay. 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 would be at Thunder Bay, to the south or Armstrong, 133 km to the north. Air access is available in Thunder Bay and Armstrong, Ontario. Thunder Bay has an international airport which is serviced by several major airlines (Figure 5.1).

Figure 5.1 Access to Lac des Iles Mine



5.3 CLIMATE

The Property experiences hot summers and cold winters, which is typical of the Boreal 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 (www.yr.no).

Mean annual precipitation for the Property is approximately 469 mm. 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 (www.theweathernetwork.com). 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. The Lac des Iles Mine does not budget for weather-related shutdowns at the mine.

6.0 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 t/d processing plant, producing a bulk nickel-copper-PGE concentrate with gold credits. On October 21, 2008, NAP announced that, due to depressed metal prices, it was temporarily placing Lac des Iles Mine on care and maintenance effective October 29, 2008. Lac des Iles Mine 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 as an underground only mine. 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, Lac des Iles Mine produced 95,057 oz of palladium (www.nap.com). Mine expansion commenced in early 2010 in order to access the Offset Zone. The Offset Zone is believed to be the fault-offset, down-dip 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, Lac des Iles Mine produced 146,624 oz of palladium from the blending of underground ore with lower-grade surface sources (www.nap.com). As of year-end 2012, Lac des Iles Mine produced 163,980 oz of palladium from both underground and surface sources. Mining of the first Offset Zone stope has commenced, and the shaft sinking is in progress (www.nap.com).

Table 6.1 summarizes the activities conducted on the Property by NAP and the predecessor companies.

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 diamond drillholes
1963	Geological Survey of Canada and Ontario Department of Mines	• released aeromagnetic maps covering the Lac Des Iles area
1963-1964	Gunnex	• optioned the Property from prospectors • 11 diamond drillholes totaling 1,516 m
1966-1968	Anaconda American Brass Limited	• optioned the Property • mapping and geophysical surveys • 13 diamond drillholes totaling 1,900 m
1974	Boston Bay	• optioned the Property • 111 diamond drillholes totaling 18,009 m • geophysical surveys
1986	Madeleine Mines	• nine diamond drillholes totaling 3,007 m
1987	Madeleine Mines	• two diamond drillholes totaling 609 m
1988	Madeleine Mines	• six diamond drillholes totaling 1,081 m
1989	Madeleine Mines	• four diamond drillholes totaling 609 m
1992	NAP	• 22 diamond drillholes totaling 1,177 m
1993	NAP	• commercial open pit production of the Roby Zone commences
1995	NAP	• 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	• 19 diamond drillholes totaling 4,243 m
1998	NAP	• 159 diamond drillholes totaling 10,863 m
1998-2001	NAP	• surface exploration over Baker Zone, Moore Zone, North Varitextured Rim Zone, and Creek Zone • extensive prospecting, stripping/trenching, channel sampling, geological mapping, and group induced polarization (IP)/resistivity geophysical surveys
1999	NAP	• 254 diamond drillholes totaling 54,758 m • 16 diamond drillholes tested the main portion of the Baker Zone • 11 diamond drillholes tested Creek Zone • limited diamond drillholes drilled at Moore Zone
2000	NAP	• 253 diamond drillholes totaling 117,324 m including 11 diamond drillholes totaling 11,458.9 m that targeted the Offset Zone

table continues…

Year	Company	Description
2001	NAP	• 36 diamond drillholes totaling 26,792 m including 11 diamond drillholes totaling 11,920 that targeted the Offset Zone
2002	NAP	• 81 diamond drillholes totaling 47,602 m
2003	NAP	• 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 diamond drillholes totaling 2,546 m including 1 diamond drillhole totaling 1,811 m that targeted the Offset Zone
2005	NAP	• 38 diamond drillholes totaling 22,623 m including 22 diamond drillholes totaling 27,697.9 m that targeted the Offset Zone
2006	NAP	• underground commercial producing commenced from ramp access • eight diamond drillholes totaling 9,619 m
2007	NAP	• 45 diamond drillholes totaling 22,195 m all targeting the Offset Zone
2008	NAP	• Lac des Iles Mine is temporarily suspended on October 29, 2008 for care and maintenance due to declining metal prices • 31 diamond drillholes totaling 18,988 m including 4 diamond drillholes totaling 5,582.2 m that targeted the Offset Zone
2009	NAP	• 86 diamond drillholes totaling 41,590 m all targeted the Offset Zone
2010	NAP	• 25 trenches totaling a length of 2,595 m • mining resumed in underground Roby High Grade Zone • commenced a major expansion to mine via shaft from the Offset Zone • 242 diamond drillholes totaling 82,131 m all targeted the Offset Zone
2011	NAP	• Lac des Iles Mine expansion underway • 15 trenches totaling a length of 1,500 m • 234 diamond drillholes totaling 79,991 m

The significant milestones of the operation are summarized below along with Table 6.2, which summarizes the annual production from the mine.

- 1993: Began mining from the Roby Zone using open pit mining methods.
- 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 has progressed substantially, with most of the surface infrastructure completed.

Table 6.2 Annual Production Summary

	2012	2011	2010	2009	2008	2007	2006	2005	2004	2003	2002	2001	2000	1999	1998	1997	1996	1995	1994	1993
Tonnes milled (t)	2,063,260	1,689,781	649,649	-	3,722,732	5,006,383	4,570,926	4,780,599	5,298,544	5,159,730	4,851,621	2,662,240	893,017	894,168	947,924	728,641	686,964	743,580	607,390	50,929
Production (gross)																				
Pd (oz)	178,835	157,476	102,212	-	212,046	286,334	237,338	177,167	308,931	288,703	219,325	123,281	95,116	64,411	84,228	66,086	60,426	76,668	59,026	-
Au (oz)	12,487	8,314	4,624	-	15,921	20,092	17,237	14,308	25,679	23,536	16,030	9,603	6,035	4,888	5,079	4,324	3,811	4,870	3,463	-
Pt (oz)	12,528	10,265	5,499	-	16,311	24,442	22,308	18,833	25,128	23,742	19,180	10,073	6,074	4,744	5,535	4,588	4,285	5,013	3,774	-
Ni (lb)	1,533,600	916,335	434,750	-	2,503,902	3,066,973	2,721,042	2,353,227	4,320,970	4,070,785	2,763,654	1,595,179	1,035,485	973,817	914,030	864,093	740,689	978,857	663,497	-
Cu (lb)	2,950,971	1,877,192	774,133	-	4,623,278	5,536,044	5,155,588	5,514,670	7,836,183	7,142,674	5,295,486	3,123,763	1,362,266	1,377,464	1,229,309	1,133,565	941,105	1,162,677	809,774	-
Pd Head Grade (g/t)	3.44	3.70	6.06	-	2.33	2.38	2.18	1.66	2.41	2.31	1.91	2.14	4.48	-	-	-	-	-	-	-
Pd Recovery (%)	78.37	78.34	80.80	-	75.29	74.74	74.00	69.60	75.16	75.50	73.80	67.40	74.00	-	-	-	-	-	-	-
Tonnes of Ore Mined (t)	**2,063,260**	**1,830,234**	**693,078**	**-**	**3,676,418**	**5,143,066**	**4,648,090**	**3,705,555**	**4,574,134**	**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)	853,600	988,503	615,926	-	615,630	768,841	721,179	-	-	-	-	-	-	-	-	-	-	-	-	-
Open Pit (t)	1,209,660	841,731	77,152	-	3,060,788	4,374,225	3,926,911	3,705,555	4,574,134	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

7.0 GEOLOGICAL SETTING AND MINERALIZATION

This section is sourced and summarized from previous technical reports written on the Property by McCombe et al. (2009) and Blakley (2009). Originally, the geological description was summarized from Clow and Rennie (2004), which sourced the information from Lavigne and Michaud (2001). Additional information can be found in Brugmann et al. (1989) with the most up-to-date work detailed in Hinchey et al. (2005).

7.1 REGIONAL GEOLOGY

The Project area is underlain by mafic to ultramafic rocks of the Archean LDI-IC (Figure 7.1). The LDI-IC has intruded granites and greenstones of the Wabigoon Subprovince of the Superior Province. It lies immediately north of the Wabigoon-Quetico Subprovince boundary, which extends approximately 300 km from Rainy Lake to Lake Nipigon. The LDI-IC is the largest of a series of mafic and ultramafic intrusions that occur along the boundary and which collectively define a 30 km in diameter circular pattern. The Project lies in the southern portion of the LDI-IC, in a roughly elliptical intrusive package measuring 3 km long by 1.5 km wide, termed the MBI (Figure 7.2).

Figure 7.1 Regional Geology

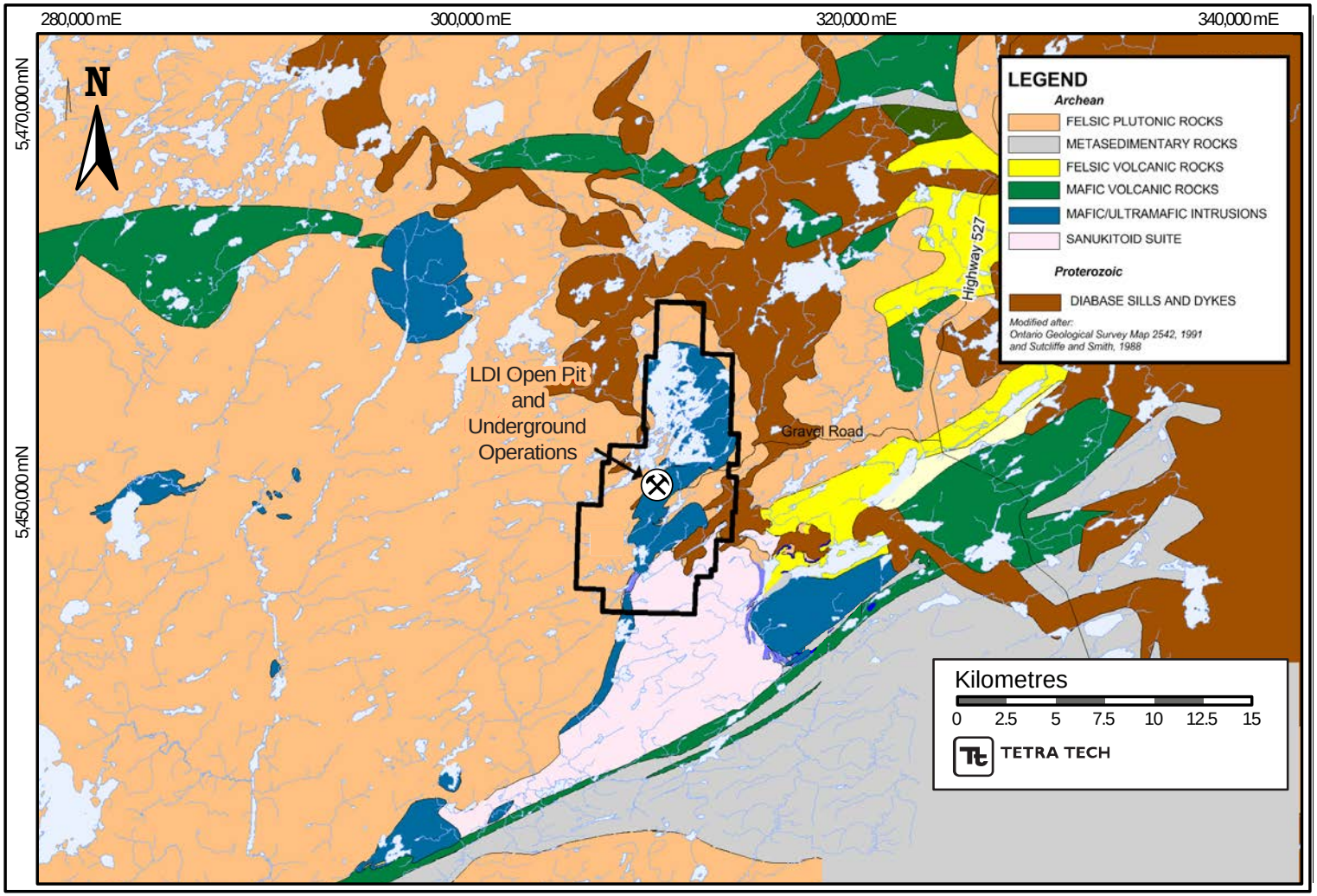
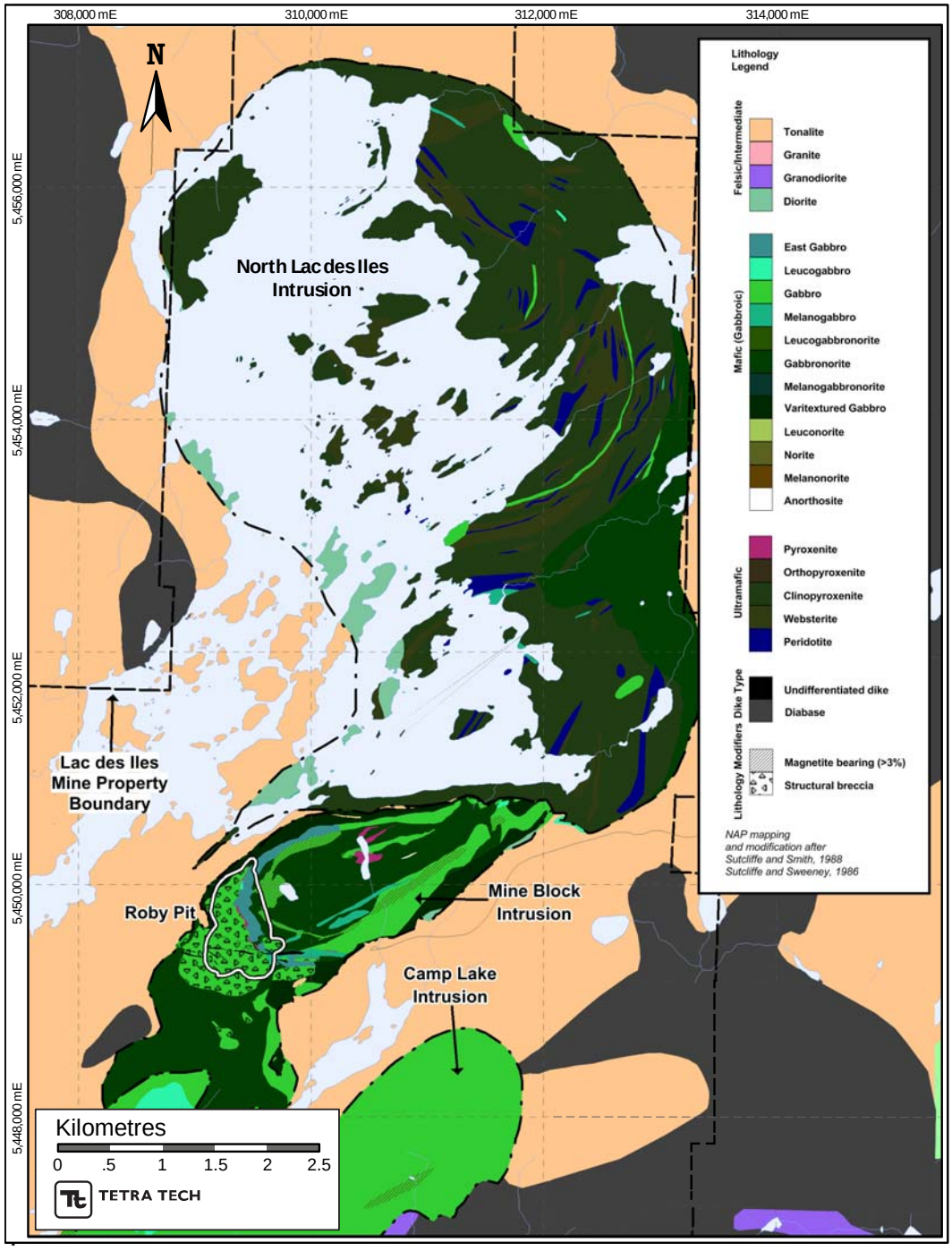


Figure 7.2 Property Geology



7.2 PROPERTY GEOLOGY

The MBI hosts a number of PGE deposits; the most important of these are the Roby Zone (including its three subzones: the North Roby Zone, the High Grade Zone, and the Breccia Zone) and the Offset Zone.

The MBI comprises rocks with a very wide range of textures, and mafic and ultramafic compositions, ranging from anorthosite to orthopyroxenite, leucogabbronorite to melanonorite, and includes magnetite-rich gabbro. Textures include equigranular, fine to coarse grained, porphyritic and pegmatitic, varitextured units, and heterolithic gabbro breccias (HGABBX). The last three textural types are the most common host to PGE mineralization. The MBI consists of two lithologically distinct domains. The oval-shaped domain immediately south of the Property is lithologically complex and contains widespread PGE mineralization, while the domain further to the south is dominated by massive, medium-grained, PGE-barren gabbronorite. The interior of the oval-shaped domain has an abundance of monolithic and heterolithic breccia with an average composition of gabbronorite. Within this area, individual lithological units are not laterally extensive and chaotically distributed. The most laterally continuous unit is a massive, medium-grained gabbro, referred to as EGAB. EGAB is adjacent to a varitextured gabbro "rim" to the west and more equigranular gabbronorite (GN) to the east. The varitextured rim is host to the Roby palladium deposit, where HGABBX commonly occurs as pipes and pods, and large blocks (approximately 60 m) of varying composition. A pyroxenite (PYXT) unit, at the contact between the EGAB and the HGABBX, is host to much of the Offset Zone.

7.2.1 ROCK TYPES

The principal rock types in the Offset Zone area include the following:

East Gabbro: EGAB is a well-known gabbro "marker unit" that is characteristically uniform and compositionally homogeneous. EGAB has very minor alteration, with local trace pyrite and epidote. It has no significant associated mineralization, and bounds the Offset Zone to the east.

Heterolithic Gabbro Breccia: HGABBX is the principal host for the Offset Zone, consisting of a melanogabbro to gabbro matrix with variable clast composition, ranging from leucogabbro to pyroxenite. Clast percentage varies commonly from 15 to 60%.

Varitextured Gabbro (VGAB): VGAB includes the majority of rock types, excluding EGAB, and have a varitextured counterpart. The VGAB varies from leucocratic to pyroxenitic, with grain sizes from fine to very coarse, to pegmatitic. The coarser-grained units form patches and "veinlets" within finer-grained counterparts.

Gabbro (GAB): GAB is the most common gabbro in the MBI and is medium grained and equigranular, but range from fine to coarse grained and may locally be leucocratic to melanocratic.

Magnetic Gabbro (MTGAB): MTGAB is medium-grained, equigranular gabbro, which occurs within the MBI and contains black, fine-grained, interstitial magnetite (typically less than 20%); magnetite content ranges from trace amounts to local, narrow layers of 60 to 95% magnetite.

Pyroxenite: PYXT is a steeply dipping, sheared, thin layer, situated along the contact between the HGABBX and EGAB. It hosts the highest proportion of the Offset Zone. This unit is responsible for much of the high PGE grades. Not all pyroxenites units carry economic PGE grades.

Gabbronorite: GN is a 20 to 50 m thick, steeply dipping slab located along the north western contact of the EGAB. It is also a host unit of the Offset Zone, although to a lesser degree than the PYXT. The gabbronorite appears to be a gradational extension of the pyroxenite to the northeast of the mine site.

Gabbronorite Breccia (GNBX): GNBX is a palladium-mineralized (Twilight Zone) heterolithic breccia, similar to the HGABBX but without pegmatitic phases or VGAB. It occurs as a roughly cylindrical pod, approximately 150 m in diameter, completely enclosed by the EGAB.

Dikes: Late, post-mineralization, mafic dikes vary from small, discrete bodies that occupy space within the modeled mineralized wire frames, to large bodies that control the northern termination of the Offset Zone. A dike swarm approximately 30 m wide and trending approximately easterly was mapped at the southern extent of the Roby Zone.

7.3 STRUCTURE

Two major faults have been interpreted to influence the Offset Zone. The Offset Fault structure displaces the Offset Zone down and approximately 300 m to the west. This fault is often marked by extensive fault gouge, fracturing and alteration of adjacent country rock, and infilling by mafic dikes. The B2 Fault was interpreted from the underground Offset Zone diamond drilling since 2007. It lies approximately 20 to 40 m below and parallel to the westerly dipping Baker Fault, and is marked by narrow intersections of fault gouge, fracturing, and late mafic dikes.

7.4 MINERAL ZONES

7.4.1 ROBY OPEN PIT ZONE

The Roby Zone is a bulk-mineable, PGE-enriched disseminated sulphide deposit with a minimum north to south length of 950 m, and a width of 815 m, including the Twilight Zone in the southwestern portion of the deposit. The Roby Zone consists of three distinct ore types: High Grade (7.6% of volume), North Roby (5.3% of volume), and Breccia (87.1% of volume). The High Grade is the primary material type mined underground (Figure 7.3).

7.4.2 ROBY UNDERGROUND ZONE

The Roby Underground Zone (High Grade Zone) is hosted mainly within a portion of a 15 to 25 m thick unit of locally sheared pyroxenite/melanogabbro. A host to high-grade PGE mineralization, it is located in the east-central portion of the Roby Zone, bounded by the barren EGAB hanging wall and HGABBX-hosted Breccia to the west. The Roby Underground Zone is primarily confined to a 400 m long segment of the pyroxenite, although it does extend northward into the gabbronorite. The High Grade Zone, striking north-northwest to north-northeast, dips almost vertically near surface and flattens to nearly 45° at depth. The zone appears to be terminated down dip the Offset Fault.

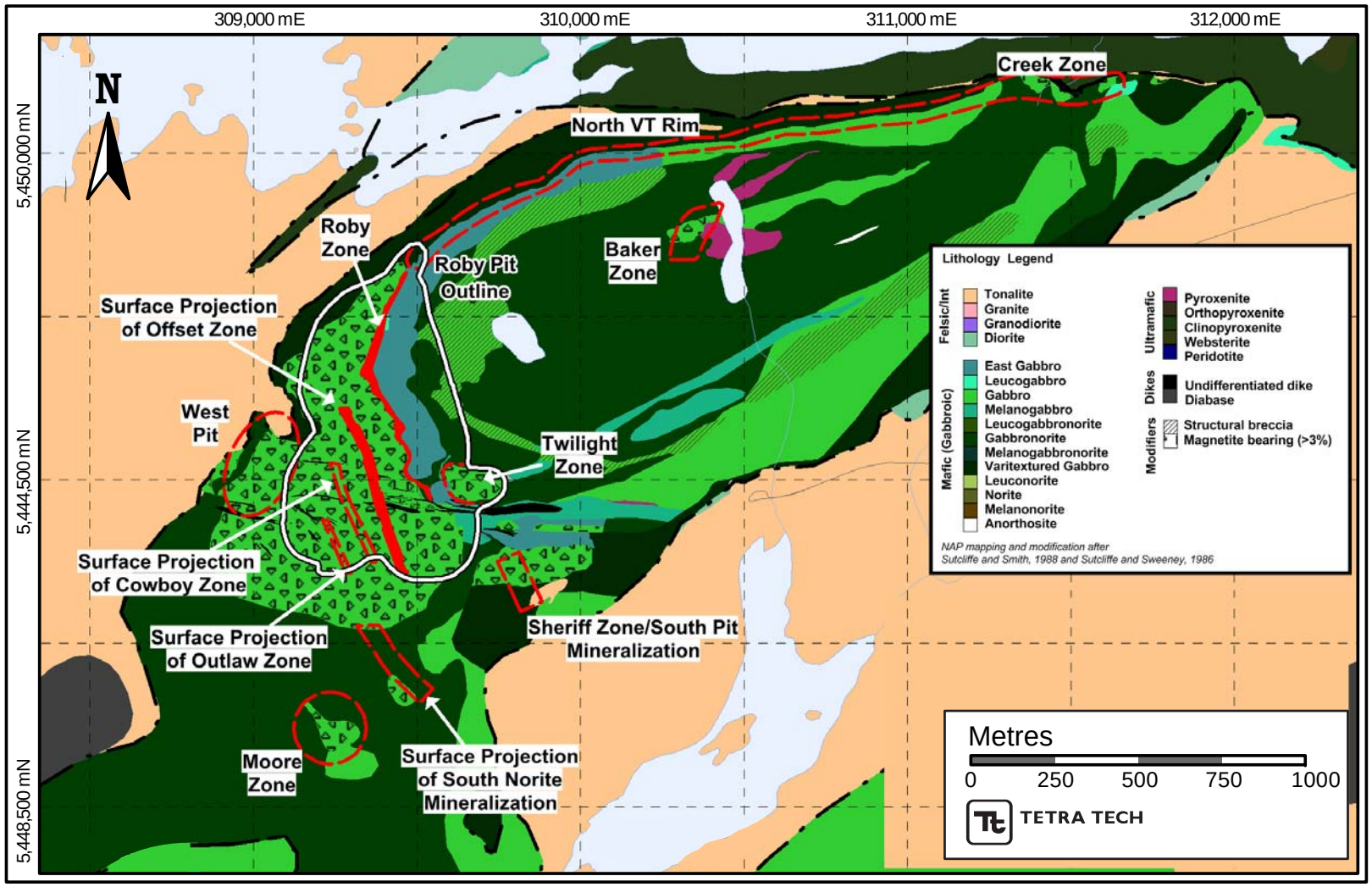
7.4.3 OFFSET ZONE

The Offset Zone, a high-grade zone similar to the High Grade Zone, is located below the Offset Fault structure, where it is displaced down and approximately 300 m to the west. The Offset Zone can be split into three horizons and has been divided into three subzones: the High Grade Subzone, the Mid Subzone, and the Footwall Subzone.

HIGH GRADE SUBZONE

The High Grade Subzone mineralization is stratabound, along the contact between the EGAB and the mineralized HGABBX. Within the HGABBX, there is a high-grade core typically constrained to an easily recognized ultramafic unit, the pyroxenite (see Figure 7.3). Its width varies from 4 to 30 m, with an average of 15 m. Approximately 2% of the zone is occupied by late dikes (dilution) and less than 1% is occupied by shears and faults.

Figure 7.3 Lac des Iles Property Mineralized Zones



MID SUBZONE

The Mid Subzone is proximal to the High Grade Zone, generally sharing a common boundary in the centre sections, and then splitting away near the top and bottom areas. Palladium grades within the Mid Subzone can approximate those high grades found within the High Grade Zone. Apparent widths can vary from 4 to 90 m, with an average of 15 m. Approximately 4% of the zone is occupied by late dikes (dilution) and less than 1% is occupied by shears and faults.

FOOT WALL SUBZONE

The Foot Wall Subzone is a stand-alone band of higher-grade mineralization that can be defined based on higher-grade intersections within the footwall VGAB mineralization. This subzone, located approximately 2 to 40 m from the Mid Subzone, was interpreted based on vertical continuity seen in the drillhole intersections. It is discontinuous and sinuous in plan and has less of a defined areal extent than the other zones. Apparent widths can vary from 4 to 20 m, with an average of 7 m. Approximately 1% of the zone is occupied by late dikes (dilution).

7.4.4 BAKER ZONE

The Baker Zone is located approximately 1 km northeast from the Roby and Twilight zones and contains similar rock types and textures. Gabbronorites/norites have been intruded by east-northeast trending, heterolithic melanogabbro breccia and lesser melanogabbro, leucogabbro breccia, varitextured gabbro and late pyroxenite dikes. Surface exploration has exposed the Baker Zone breccias and associated lithologies over a 150 m by 55 m area. The heterolithic melanogabbro breccia hosts blebby to disseminated to narrow veinlets of sulphide with sporadic mineralization in the adjacent lithologies. The north-trending, shallowly westerly dipping Baker Fault appears to truncate the Baker Zone mineralization at depth. Extensive surface exploration by NAP occurred mainly from 1998 to 2001 and consisted of prospecting, stripping/trenching (including the main stripped area of approximately 200 m by 120 m), channel sampling, geological mapping, and ground IP/resistivity surveys. Sixteen diamond drillholes in 1998/1999 tested the main portion of the Baker Zone over a 250 m strike length and to a maximum depth of 200 m. Subsequent exploration (trenching and diamond drilling) has tested possible strike extensions of the zone and the area below the Baker Fault. A 2012 review of the Baker Zone included 1,155 continuous channel samples along 9 north-south lines spaced approximately 20 m apart.

7.4.5 THE MOORE ZONE

The Moore Zone is a low-grade, presently uneconomic, mineralized zone, located approximately 500 m south of the current Roby 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, varying from approximately 15 to 115 m wide, which occurs within the massive, medium-grained gabbronorite typical of the more southerly domain of the MBI. The main Moore Zone mineralization is located in the eastern portion of the breccias pod and appears to be structurally controlled (trending approximately 030°, dipping 70° east), ranging from 5 to 25 m thick. Prospecting, mapping, trenching, sampling, and limited diamond drilling of the Moore Zone have indicated limited economic potential.

7.4.6 CREEK ZONE

The Creek Zone is located approximately 2 km northeast of the Roby Pit in the northeastern nose of the MBI, near the contact with the north LDI-IC. Surface trenching has exposed the main portion of the Creek Zone 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. Unlike the Roby Zone, mineralization is not dominantly hosted by the breccia matrix but seems to occur within the pegmatitic gabbronorite. Prospecting, mapping, trenching, sampling, and limited diamond drilling of the Creek Zone have indicated limited mineralized potential.

7.4.7 COWBOY ZONE

The Cowboy Zone is a new underground PGE-bearing zone. The Cowboy Zone contains higher sulphides than the Offset Zone and consists of approximately 1% chalcopyrite mineralization, in addition to pyrite, hosted by VGAB, pyroxenite, and brecciated gabbro. The zone is located 30 to 50 m down section to the west of the Offset Zone and extends for up to 250 m along strike and 350 m down dip. The zone is open in all directions and appears to consist of several mineralized subzones or lenses. During Phase 1 and 2 drilling of the Offset Zone in 2009, the toes of drillholes on alternating cross sections (at ±60 m) were pushed ±100 m into the footwall to trace the Cowboy Zone and explore for other PGE mineralization.

7.4.8 OUTLAW ZONE

The Outlaw Zone lies approximately 30 to 60 m west of the Cowboy Zone and further down section into the footwall of the Offset Zone.

7.4.9 SHERIFF ZONE

The Sheriff Zone, a combination of the former Southeast Roby and South Pit zones connected through further drilling. It may represent a large-tonnage, low-grade resource amenable to a bulk mining scenario in close proximity to the current infrastructure.

7.4.10 SOUTHERN NORITE ZONE

A surface drillhole, 11-003, discovered the Southern Norite Zone while attempting to drill test the extent of the Offset Zone towards the south. This zone sits relatively along strike from the Offset Zone and 150 m south of the shaft.

7.5 MINERALIZATION

PGE and base metal mineralization at the Lac des Iles Mine appears to be dominantly stratabound along the contact between the EGAB and the mineralized HGABBX. Within the HGABBX, there is a high-grade core typically constrained to an easily recognized pyroxenite unit. Visible PGE mineralization is rare and its occurrence is difficult to predict. In general, economic PGE grades are anticipated within gabbroic to pyroxenitic rocks (in close proximity to the marker unit EGAB) that exhibit strong sausseritization of plagioclase feldspars, strong talcose alteration and association with either disseminated or blebby secondary sulphides. Higher PGE grades (mean – 7.89 g/t palladium, maximum – 55.95 g/t palladium) occur in those portions of the pyroxenite that are altered to an assemblage of amphibole (anthophyllite-actinolite-hornblende)-talc-chlorite. PGE grades show varying degrees of correlation with nickel and copper grades. Some parts of the deposit have very good PGE-copper-nickel (and sulphur) correlations, while others show no significant, positive correlation between PGE and copper or nickel. Sulphides typically occur as fine-grained disseminations of less than 1 to 2 to 3 percent and of varying proportions of pyrrhotite, pyrite, chalcopyrite, and pentlandite. The high-grade mineralization is located primarily within the western, highly altered portion of the pyroxenite, since much of the pyroxenite between the barren EGAB and the High Grade Zone is low-grade. The higher-grade “High Grade ore” is not restricted to the pyroxenite as it commonly straddles the pyroxenite/gabbro breccia contact to widths exceeding 250 m.

The majority of platinum-group minerals occur either interstitially to sulphides as cumulus grains or are associated with sulphides at sulphide-silicate boundaries, occurring as discrete mineral inclusions within secondary silicates of altered rocks (Sweeny 1989; Lavigne and Michaud 2001). Palladium and platinum mineralization within the High Grade Zone consists primarily of fine-grained palladium-telluride minerals; kotulskite, michenerite, palladium-arsenide, PGE sulphide, and vysotskite (Jon and Barnes 2012; Lavigne and Michaud 2001). The PGEs at the Lac des Iles Mine are shown in Table 7.1.

Table 7.1 Minerals

Mineral	Chemical Formula
Braggite	(Pt, Pd, Ni)S
Kotulskite	Pd(Te, Bi)
Isometrieite	$Pd_{11}(Sb, Te)_2As_2$
Merenskyite	$PdTe_2$
Michenerite	PdBiTe
Moncheite	$PtTe_2$
Palladoarsenide	Pd_2As
Sperrylite	$PtAs_2$
Stibiopalladinite	Pd_5Sb_2
Stillwaterite	Pd_8As_3
Vysotskite	PdS
Unnamed	$Ag_4Pd_3{}_te_4$
Unnamed	Pd_5As_2
Melonite, gold, pentlandite	Pd in solid solution

Source: Lavigne and Michaud (2001)

8.0 DEPOSIT TYPES

Two styles of mineralization have been observed at the Offset Zone; contact nickel-PGE mineralization (US Geological Survey (USGS) #5b) and reef PGE mineralization (USGS #2b) (http://pubs.usgs.gov/bul/b1693/html/bullfrms.htm).

Barnes and Gomwe (2011) put forward a schematic model illustrating a deposit model for the history of mineralization in the MBI (Figure 8.1).

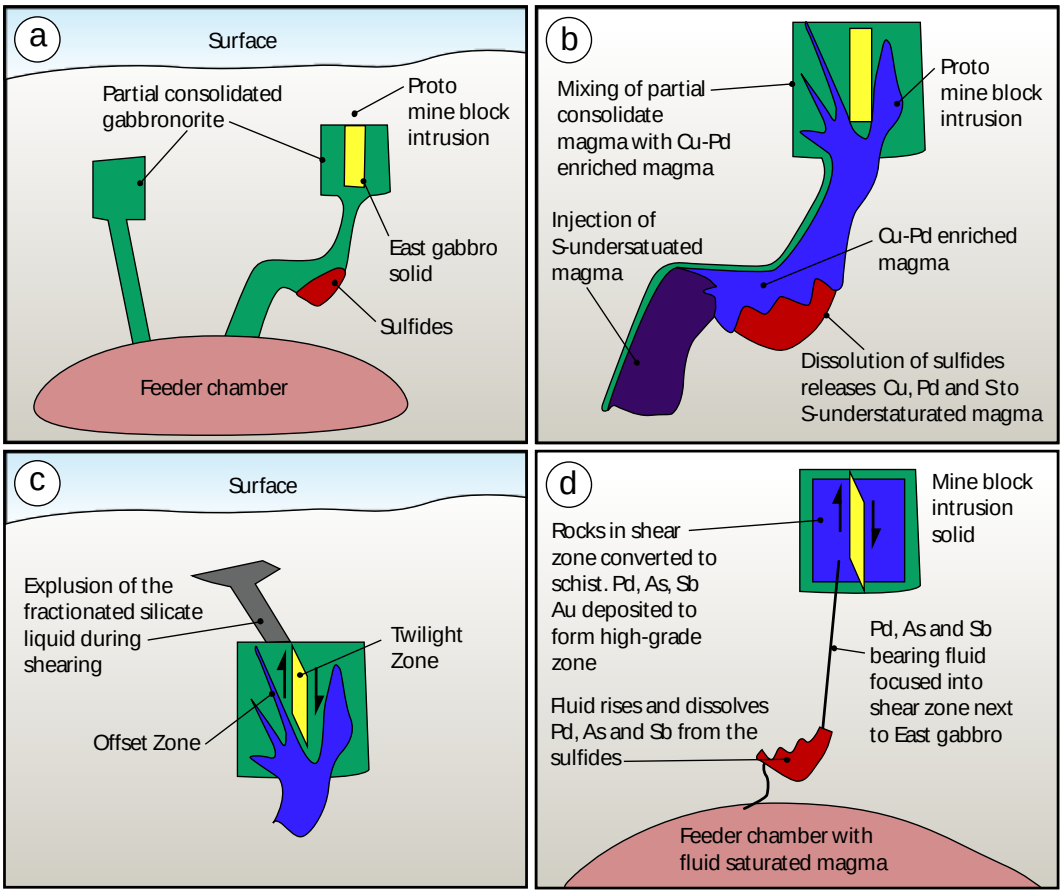
Any model for the formation of the Lac des Iles ores must consider the following observations. The deposits occur in zones of magmatic breccia that underwent deformation at high temperature. There were repeated injections of magma into these zones. Each injection caused the fragmentation of the rocks already in place. The rocks are essentially orthopyroxene-plagioclase cumulates with a very low melt fraction and the matrix material does not represent a silicate liquid composition. This implies that the fractionated liquid has been squeezed out of the breccia. The similarity in composition between the fragments and matrix also implies that they are comagmatic. The presence of pegmatites and varitextured rocks suggests the presence of a fluxing agent during their crystallization, but these rocks are neither enriched nor depleted in PGE, thus this fluxing agent did not play a role in the mineralization. Control of the metals by a sulfide liquid can explain much of the variation in metal content observed in the gabbronorites and metagabbronorites. The partition coefficients of the PGE into sulphide liquid are thought to be similar for all of the PGE, thus collection by a sulfide liquid will not explain the extremely high Pd/Ir and Pd/Pt ratios of the rocks. All of the rocks in the Mine Block intrusion show very strong fractionation of Pd from Pt and Ir, and yet, no rocks depleted in Pd have been found. Therefore, the process of fractionating the PGE must have taken place elsewhere. The High-grade zone samples have undergone intense low temperature alteration to a greenschist facies assemblage and appear to be located close to or within a shear zone.

Considering all of these factors, we suggest the following model. The Lac des Iles Complex is one of a number of small intrusions in the area that together form a circular array about 50 km in diameter. It has been suggested that the intrusions are underlain by a large feeder chamber (Hinchey et al., 2005). In the first stage of the model the East Gabbro was emplace and solidified. After the emplacement of the East Gabbro a gabbronorite magma intruded on either side of it [Figure 8.1a]*. At depth a sulfide liquid segregated from the magma, collected in a structural trap, and solidified there* [Figure 8.1a]*. Subsequently an S-undersaturated magma passed over the sulfides and partially dissolved them. We suggest by analogy with the experimental work of Peregoedova et al. (2004, 2006), that Cu, Pd, and Au dissolved into the magma* [Figure 8.1b]*. This magma was then injected on either side of the East Gabbro. It mixed with the partially consolidated magma already*

present in the chamber to form the magmatic breccia zones [Figure 8.1c]. *As the magma crystallized it became saturated in sulfide liquid and because the magma was enriched in Pd a Pd-rich sulfide liquid segregated from it and precipitated among the cumulate grains to form the Roby and Twilight zones. The chamber was being deformed during crystallization and much of the fractionated magma was squeezed out of the Roby and Twilight zones. As the temperature fell the magma in the upper chambers became saturated in a magmatic fluid. This resulted in partial recrystallization of the magma to form pegmatites and varitextured rocks. This fluid could also have converted many of the rocks to metagabbronorites.*

To model the formation of the High-grade zone, we suggest that as the temperature decreased in the feeder chamber this magma became saturated in fluid and that this fluid as it rose, scavenged Pd, As, Sb, and to a lesser extent Pt and Au, from the sulfide formed at depth [Figure 8.1d]. *At this point the magma in the Mine Block chamber was consolidated, thus the fluid could not penetrate everywhere and was confined to the shear zone next to the East Gabbro* [Figure 8.1d]. *The rising fluid reacted with the host rocks to precipitate Pd, Pt, and Au accompanied by Sb and As rather than S, creating the low-S, Pd-rich High-grade zone. The focusing of the lower temperature fluid and the continuing shearing accentuated the level of alteration in the High-grade zone. The correlation between Pd, Pt, and Sb plus As, the low Cu content and the enrichment in Au observed in the High-grade zone rocks suggests involvement of thiosulfate ligands (Wood, pers. comm.). Thiosulfates may exist in significant concentrations during the oxidation of sulfide bodies and can significantly transport Pd, Pt, and Au (Wood 2002). The presence of thiosulfates would require the process to take place at temperatures well below 240°C (Wood, 2002).*

Figure 8.1 Deposit Model



8.1 CONTACT-STYLE PGE MINERALIZATION (USGS #5B)

Contact-style PGE mineralization develops as the result of sulphur-saturation brought on by the interaction of the fertile parental magma with the surrounding country rock lithologies. The contamination of the initial fertile parental magma by the addition of either silicon dioxide and/or sulphur can directly result in sulphur-saturation and the separation of a PGE-rich immiscible sulphide. The addition of silicon dioxide and/or sulphur is typically achieved by the assimilation of either local country rock lithologies and/or the assimilation of breccia fragments previously developed along the contact margin. Analogies for this model include River Valley (northeastern Ontario), the Platreef (South Africa), and the Portimo Complex (Finland).

8.2 REEF-STYLE PGE MINERALIZATION (USGS #2B)

Reef-style PGE mineralization is a strata-bound or strata-form style of mineralization that typically occurs higher up in the stratigraphy of the intrusion at the contact

between two separate and distinct lithological units. Sulphur-saturation and therefore sulphide segregation can be the result of the interaction between distinctly different types of magma, with sulphur-saturation occurring at their interface. Geochemical evolution of the overlying magma can also cause sulphur-saturation and the separation of immiscible sulphides can accumulate between the two units.

The Lac des Iles ore zone has a different shape to the PGE-bearing reefs elsewhere. It is approximately 900 m by 700 m and open at depth. All the reefs, except the Platreef of South Africa, are narrow layers of 1 to 3 m thick. The Platreef is wider, but it occurs at the margin of the Bushveld Complex, while the deposits of Lac des Iles occur within the body (Barnes and Gomwe 2010).

Due to the stratigraphic control and typical narrow widths of reef-style PGE mineralization, exploration programs must be focused entirely on the productive horizon. In order to identify the proper horizon, geochemical traverses are essential with the goal being to look for systematic changes in PGE and/or nickel-copper tenors across lithological boundaries. Once the specific horizon is identified then grid sampling and ground-based geophysics should be used over the target area.

9.0 EXPLORATION

9.1 INTRODUCTION

Other than diamond drilling, there has been no current exploration on the Offset Zone. Below is a summary of the surface exploration conducted on the mining lease.

9.2 TRENCHING

Overburden stripping and trenching is a technique commonly used for exploration across the Property. Large number of areas and metres are stripped, mapped, and continuously sampled every year. The results from the trenching programs are used to generate future drilling targets and correlate well with previous surface operations.

In 2010, 18 separate trenches were dug for a total length of 908 m. In 2011, 14 separate trenches were opened for a total of approximately 1,000 m. In 2012, eight trenches were completed for a total of 1,245 m (Table 9.1).

Table 9.1 Trench Location Summary

Trench	Zone	Length (m)
TR10-01	VT North	5
TR10-02	VT North	96
TR10-03	VT North	132
TR10-04	VT North	53
TR10-05	VT North	37
TR10-06	VT North	62
TR10-07	VT North	42
TR10-08	Creek	17
TR10-09	Creek	75
TR10-10	Creek	45
TR10-11	Creek	52
TR10-12	Creek	49
TR10-20	SLDI	59
TR10-21	SLDI	26
TR10-22	SLDI	71
TR10-23	SLDI	65

table continues...

Trench	Zone	Length (m)
TR10-24	SLDI	9
TR10-25	SLDI	13
TR11-01	VT North	31
TR11-02	VT North	67
TR11-03	VT North	60
TR11-04	VT North	172
TR11-05	VT North	38
TR11-06	VT North	204
TR11-07	VT North	67
TR11-09	VT North	175
TR11-10	VT North	89
TR11-11	VT North	60
TR11-12	SLDI	3
TR11-13	SLDI	4
TR11-14	SLDI	17
TR11-15	SLDI	13
BZ12-2870	Baker	57
BZ12-2890	Baker	130
BZ12-2910	Baker	134
BZ12-2930	Baker	163
BZ12-2950	Baker	196
BZ12-2970	Baker	191
BZ12-2990	Baker	164
BZ12-3010	Baker	98

Overburden stripping was performed by Belham Limited (Belham) of Kaministiquia, Ontario. They were contracted to provide a Link Belt 240LX excavator and operator; the project was planned and supervised by Mike Grieve from NAPs Metals Exploration Division. Excavated areas were surveyed using a Leica SR 530 real-time kinetic global positioning system (GPS), accurate to within 2 cm. Surveys were then imported into Gemcom™ and MapInfo/Discover for map generation (Nelson et al. 2010). Belham was also contracted to pressure wash the areas stripped and cut channel samples over the course of the 2010, 2011, and 2012 campaign.

Cutting was performed using a Stihl TS400 water-cooled cut-off saw equipped with a diamond blade; two parallel saw cuts were made 4 to 5 cm apart and 8 to 10 cm deep; sample intervals were indicated by a saw cut placed perpendicular to the direction of the sampling lines (Nelson et al. 2010).

Samples were chipped free using a masonry chisel and hammer, then packaged in individually labelled plastic sample bags (as a backup, the bags also included a labelled manila shipping tag). The plastic bags were then taped shut, packed four to

five to a rice bag, and transported to the Activation Laboratories Ltd. (ActLabs) assaying facility in Thunder Bay by Courtesy Courier. Aluminum tags inscribed with the appropriate sample number were secured to the starting saw cut for each sample interval using a masonry nail (Nelson et al. 2010) (Figure 9.1).

Figure 9.1 Example of a Completed Trench

Figure 9.2 shows the locations relative to the Lac des Iles Mine area and Figure 9.3 shows an example of the completed trench mapping. Table 9.2 summarizes the results from the trenching programs.

Figure 9.2 Trench Location Map

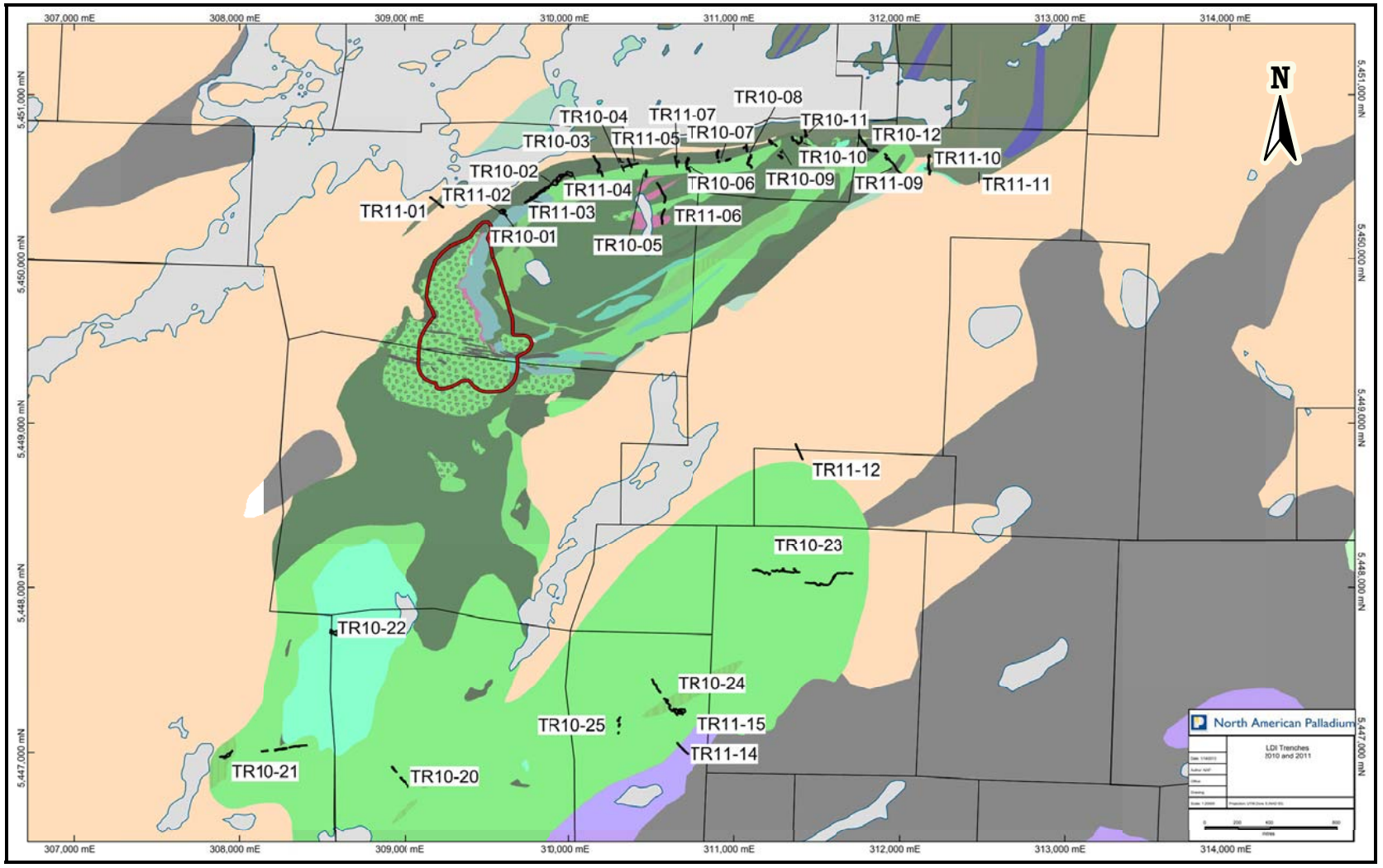


Figure 9.3 Example of Trench Mapping

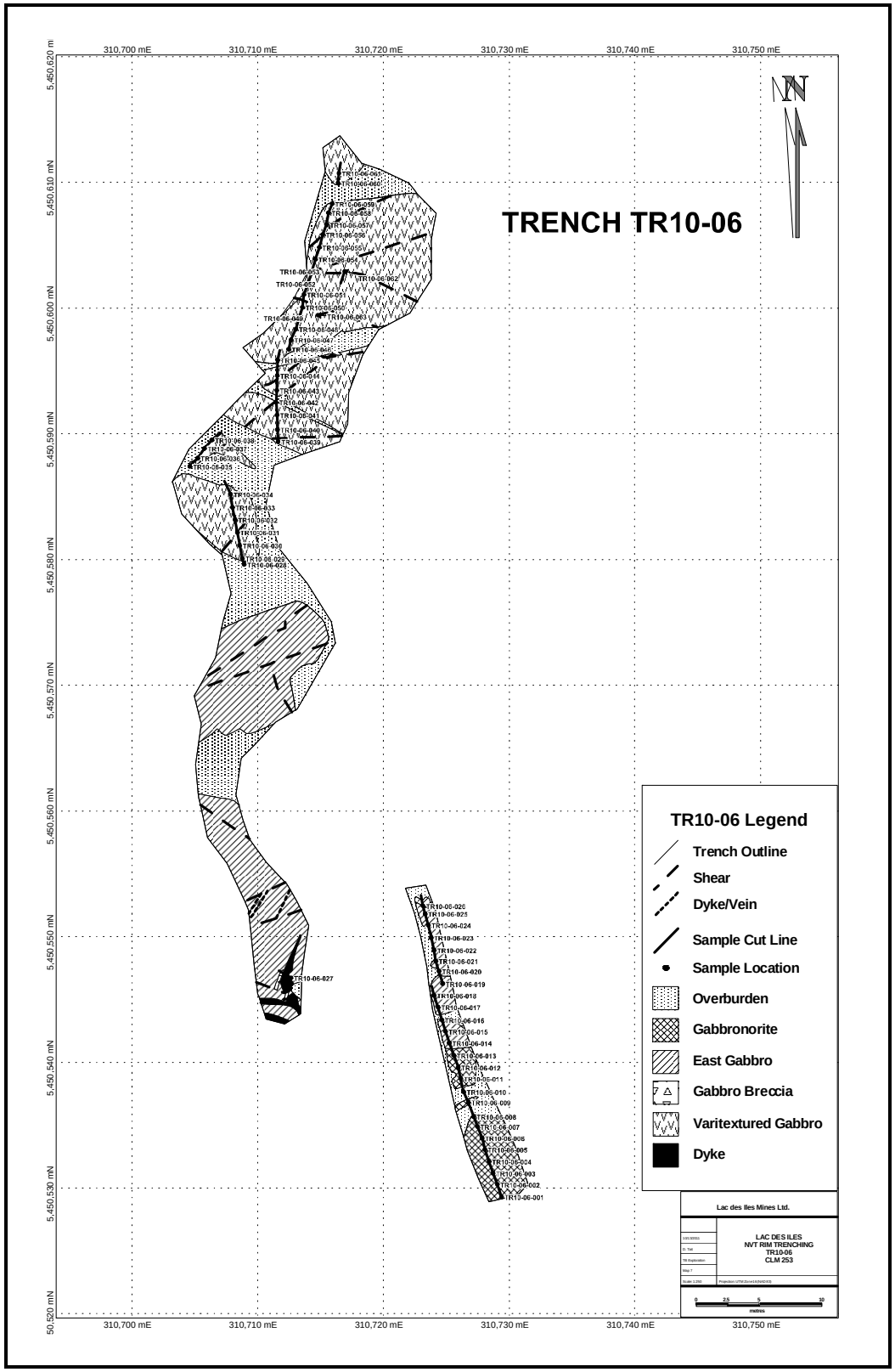


Table 9.2 Summary of Trench Results

Trench	Sample ID	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ni (%)	Co (%)
TR10-02	TR10-02-001	3.38	0.145	0.049	0.0067	0.0298	0.0045
TR10-02	TR10-02-032	4.18	0.409	0.029	0.0010	0.0280	0.0039
TR10-02	TR10-02-033	5.81	0.442	0.015	0.0005	0.0293	0.0037
TR10-02	TR10-02-056	2.77	0.200	0.017	0.0013	0.0400	0.0049
TR10-02	TR10-02-063	2.71	0.416	0.058	0.0040	0.0371	0.0042
TR10-02	TR10-02-076	2.48	0.157	0.015	0.0007	0.0327	0.0041
TR10-02	TR10-02-086	24.60	1.870	0.102	0.0015	0.0315	0.0038
TR10-03	TR10-03-042	2.29	0.467	0.056	0.0144	0.0324	0.0056
TR10-03	TR10-03-049	3.15	0.404	0.077	0.0258	0.0373	0.0058
TR10-04	TR10-04-008	5.70	0.616	0.100	0.0077	0.0313	0.0054
TR10-07	TR10-07-018	3.09	0.360	0.184	0.0091	0.0349	0.0050
TR10-10	TR10-10-010	2.11	0.317	0.126	0.0389	0.0788	0.0087
TR11-02	TR11-02-026	22.30	0.994	0.325	0.0076	0.0358	0.0041
TR11-02	TR11-02-027	23.00	0.976	0.577	0.0228	0.0499	0.0046
TR11-02	TR11-02-028	6.79	0.301	0.238	0.0092	0.0354	0.0040
TR11-02	TR11-02-031	15.80	0.845	0.344	0.0063	0.0377	0.0049
TR11-02	TR11-02-041	3.06	0.192	0.053	0.0058	0.0357	0.0048
TR11-02	TR11-02-069	2.45	0.111	0.015	0.0022	0.0302	0.0036
TR11-02	TR11-02-081	8.32	0.450	0.103	0.0091	0.0321	0.0040
TR11-03	TR11-03-028	6.20	1.030	0.037	0.0005	0.0205	0.0025
TR11-03	TR11-03-050	3.08	0.136	0.023	0.0018	0.0317	0.0036
TR11-04	TR11-04-007	3.80	0.284	0.021	0.0023	0.0444	0.0062
TR11-04	TR11-04-008	6.13	0.510	0.034	0.0044	0.0279	0.0048
TR11-04	TR11-04-010	4.25	0.289	0.031	0.0011	0.0193	0.0029
TR11-04	TR11-04-011	64.40	5.170	1.190	0.0210	0.0679	0.0046
TR11-04	TR11-04-098	2.08	0.208	0.013	0.0026	0.0375	0.0057
TR11-04	TR11-04-100	4.53	0.236	0.043	0.0008	0.0281	0.0042
TR11-04	TR11-04-101	2.97	0.151	0.016	0.0014	0.0410	0.0061
TR11-04	TR11-04-102	37.30	2.440	0.326	0.0047	0.0465	0.0065
TR11-04	TR11-04-103	45.80	2.480	1.590	0.0179	0.0525	0.0052
TR11-04	TR11-04-104	15.20	0.698	0.277	0.0035	0.0380	0.0047
TR11-04	TR11-04-105	2.23	0.134	0.012	0.0006	0.0199	0.0033
TR11-04	TR11-04-106	6.21	0.420	0.025	0.0010	0.0228	0.0032
TR11-04	TR11-04-172	4.23	0.219	0.035	0.0016	0.0231	0.0036
TR11-04	TR11-04-176	5.60	0.688	0.087	0.0284	0.0489	0.0050
TR11-07	TR11-07-042	2.09	0.220	0.024	0.0005	0.0348	0.0046
TR11-09	TR11-09-019	2.70	0.218	0.012	0.0002	0.0291	0.0037
TR11-09	TR11-09-066	3.54	0.433	0.042	0.0199	0.0223	0.0051
12BZ2890	12BZ2890-126	2.15	0.205	0.167	0.1840	0.1470	0.0065
12BZ2890	12BZ2890-127	3.18	0.309	0.251	0.2440	0.1760	0.0078

table continues…

Trench	Sample ID	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ni (%)	Co (%)
12BZ2890	12BZ2890-128	4.70	0.445	0.300	0.3140	0.2530	0.0097
12BZ2890	12BZ2890-129	2.45	0.225	0.171	0.1960	0.1550	0.0072
12BZ2890	12BZ2890-136	2.45	0.239	0.189	0.1630	0.1460	0.0069
12BZ2890	12BZ2890-138	2.23	0.209	0.179	0.2020	0.1470	0.0076
12BZ2890	12BZ2890-148	2.39	0.274	0.148	0.2070	0.1650	0.0077
12BZ2890	12BZ2890-149	2.07	0.226	0.166	0.2130	0.1580	0.0081
12BZ2890	12BZ2890-185	2.01	0.184	0.102	0.2040	0.7520	0.0279
12BZ2910	12BZ2910-167	2.18	0.229	0.135	0.1460	0.1320	0.0075
12BZ2910	12BZ2910-174	2.80	0.375	0.231	0.8300	0.4400	0.0195
12BZ2910	12BZ2910-175	6.06	0.301	0.224	0.6320	0.8250	0.0484
12BZ2930	12BZ2930-144	2.07	0.232	0.119	0.1470	0.1330	0.0082
12BZ2930	12BZ2930-159	2.25	0.230	0.094	0.1390	0.1320	0.0102
12BZ2950	12BZ2950-169	2.26	0.214	0.185	0.1750	0.1460	0.0091
12BZ2970	12BZ2970-098	2.24	0.288	0.108	0.1960	0.1670	0.0088
12BZ2970	12BZ2970-100	2.03	0.239	0.166	0.2240	0.1880	0.0080
12BZ2970	12BZ2970-102	2.71	0.388	0.263	0.2590	0.2360	0.0096
12BZ2970	12BZ2970-127	2.16	0.222	0.116	0.1670	0.1400	0.0080
12BZ2970	12BZ2970-128	2.61	0.263	0.150	0.2200	0.1600	0.0079
12BZ2970	12BZ2970-129	2.21	0.233	0.099	0.1470	0.1410	0.0079
12BZ2970	12BZ2970-148	2.72	0.249	0.254	0.1860	0.1470	0.0074
12BZ2990	12BZ2990-082	2.10	0.296	0.249	0.3250	0.2270	0.0122
12BZ2990	12BZ2990-088	2.40	0.341	0.258	0.3410	0.1800	0.0079
12BZ2990	12BZ2990-091	2.78	0.351	0.196	0.3720	0.2180	0.0092
12BZ2990	12BZ2990-092	2.20	0.275	0.092	0.2480	0.1680	0.0075
12BZ3010	12BZ3010-150	2.08	0.213	0.148	0.1640	0.1220	0.0063

10.0 DRILLING

10.1 PRIOR OWNERS

Prior owners drilled 135 holes totalling 21,425 m on the Property, while LDIM and its predecessor company drilled 1,747 holes totalling 594,858 m on the Property (Table 10.1).

Table 10.1 Drilling Summary

Year	No. of Holes	Length (m)	Operator	Series		
1964	11	1,516	-	G64-1	-	G64-9
1966	13	1,900	-	A66-12	-	A66-24
1970s	111	18,009	Boston Bay	P001	-	P114
1986	9	3,007	Madeleine Mines	86-01	-	86-26
1987	2	609	Madeleine Mines	87-37	-	87-40
1988	6	1,081	Madeleine Mines	88-1	-	88-8
1989	4	609	Madeleine Mines	89-1	-	89-4
1992	22	1,177	LDIM	92-01	-	92-22
1995	56	7,802	LDIM	95-01	-	95-57
1997	19	4,243	LDIM	97-01	-	97-19
1998	51	7,591	LDIM	98-01	-	98-53
1998	108	3,272	LDIM	EX-P-01	-	EX-P-141
1999	254	54,758	LDIM	99-001	-	EXP99093
2000	233	117,324	LDIM	00-001	-	00-339
2001	36	26,792	LDIM	01-001	-	01-086
2002	81	47,602	LDIM	02-001	-	02-094
2003	25	10,211	LDIM	03-001	-	03-029
2004	4	2,546	LDIM	04-001	-	04-004
2005	38	22,623	LDIM	05-001	-	05-048
2006	8	9,619	LDIM	05-006W2	-	05-016W4
2007	45	22,195	LDIM	07-001	-	07-024
				07-028	-	07-037
				07-041	-	07-051
2008	31	18,988	LDIM	Jul-52	-	07-053
				08-001	-	08-108
2009	86	41,590	LDIM	09-001	-	09-955
2010	242	82,131	LDIM	10-001	-	10-999

table continues…

Year	No. of Holes	Length (m)	Operator	Series		
2011	234	79,991	LDIM	11-001	-	11-904
2012	153	50,148	LDIM	12-001	-	12-842
Total	**1,882**	**637,334**	**-**	**-**	**-**	**-**

10.2 NORTH AMERICAN PALLADIUM LTD.

In 2012 drilling was stopped in late May due to influx of water into the mine workings. Numerous areas flooded and thus reduced access to the drill platforms. Underground drilling started back up in October, and tested both the Offset and Roby zones. There were a total of 44 surface holes drilled and a total of 109 drillholes drilled from underground.

All drilling on the Property in 2012 was contracted to Bradley Diamond Drilling of Thunder Bay, a division of Major Drilling of Moncton, New Brunswick. All drilling was drilled with NQ sized core (47.6 mm diameter).

10.2.1 OFFSET ZONE

From 2000 to 2012, 543 surface and underground diamond drillholes totalling 210,242 m have been drilled on the Offset Zone (Table 10.2).

Table 10.2 Offset Zone Drilling Summary

Year	No.	Length	No. Surface	No. Underground	Surface (m)	Underground (m)
2000	11	11,458.90	11	0	11,458.90	0.00
2001	11	11,920.50	11	0	11,920.50	0.00
2003	2	1,773.00	2	0	1,773.00	0.00
2004	1	1,811.00	1	0	1,811.00	0.00
2005	22	27,697.90	22	0	27,697.90	0.00
2007	45	22,195.42	1	44	1,491.50	20,703.92
2008	4	5,582.20	4	0	5,582.20	0.00
2009	86	41,589.58	0	86	0	41,589.58
2010	179	39,994.00	17	162	14,036.00	25,958.00
2011	132	35,010.00	11	121	10,640.00	24,370.00
2012	72	17,724.00	-	72	-	17,724.00
Total	**543**	**210,242.50**	**80**	**463**	**86,411.0**	**123,831.50**

10.2.2 SURVEYING

COLLAR SURVEY

All hole collars have been surveyed using known mine stations. Collar locations on surface and underground are pre-located by geologists and/or technicians and a final survey is done by conventional methods by mine surveyors or differential GPS.

DOWNHOLE SURVEY

Since 1995, all holes have been surveyed downhole either with a Tropari instrument or the Reflex Maxibor®, a non-magnetic light log method. In the spring of 2008, Sperry Sun was contracted by NAP's Exploration department to resurvey surface holes 07-052 and 07-053 with a North Seeking Gyro. The holes intersected the extreme south portion of the Offset Zone in the area of Inferred Resources. The results of these gyro surveys were updated within the Offset Zone digital database. A comparison of the original Reflex Maxibor® survey results with the Sperry Sun North Seeking Gyro results as shown in Table 10.3 (Routledge et. al. 2010). Variance at the toe of the hole between the methods is 42 m and 34 m, respectively, or 2.3% of the hole length. The precision difference of up to 42 m between the methods suggests that controlled directional drilling from surface is required for the level of accuracy to classify resources as Indicated where a pattern of 30 m by 60 m is to be maintained.

Table 10.3 Survey Comparison

Survey Type	Hole No. 07-052 (1,809 m)			Hole No. 07-053 (1,489 m)		
	Northing (m)	Easting (m)	Elevation (m)	Northing (m)	Easting (m)	Elevation (m)
Reflex Maxibor®	32151.65	31638.58	-1222.31	32127.67	31431.02	-821.09
North Seeking Gyro	32114.50	31654.41	-1210.91	32109.30	31459.57	-815.45
Absolute Variance	37.15	15.83	11.40	18.37	28.55	5.64

10.2.3 CORE DELIVERY

Core boxes are delivered from the drill site by the drill contractor to the core logging facility on the Property. If required, core is shipped to the NAP Exploration office in Thunder Bay for logging.

Drill core that is processed at the NAP Thunder Bay Exploration office is shipped to Thunder Bay from the mine via Courtesy Courier. The drill core is in drill core boxes which are secured to pallets by metal strapping during transport. Drill core at the Thunder Bay facility is stored in a gated area.

Once the boxes are in the core facility, the drillhole numbers and box numbers are checked and reported to the logging geologists in order to confirm the correct core bench where the core will be transferred.

The boxes are laid out on the core bench maintaining numeric continuity. The core is cleaned to remove any drill grease or additive.

10.2.4 GEOTECHNICAL LOGGING

These are the steps to record geotechnical data for the core:

1. Load core boxes on logging benches in numerical order with box numbers increasing down rows and left to right. Check with the logging geologist beforehand to ensure correct placement (Figure 10.1).
2. Carefully inspect the new core, reconstructing blocky intervals wherever possible. Visually check that the meters written on the drill blocks increase down row, and left to right, and are in intervals of no more than 3.05 m. These intervals will be used as From-To intervals in the subsequent measurements below. Alert logging geologists of any discrepancies.
3. Take recovery length and rock quality designation (RQD) measurements.
4. Recovery Length: Measure length of core in core boxes between From-To intervals in meters. With broken/rubbly core, record the best estimate of recovery by reconstructing, and measuring, competent pieces, while visualizing rubbles zones as whole core.
5. RQD: Measure the total length of solid core pieces (less than 10 cm) measured along the centerline of the core between From-To intervals.
6. Mark meter intervals on core. Use drill blocks for reference. Use core recovery for intervals in which there is a discrepancy between blocks and meter intervals. Look for obvious breaks/fault zones, and signs of grinding/rounding from drilling, to account for missing core. Alert logging geologists of any discrepancies.
7. Label core boxes with individual From-To intervals in the top left, and bottom right corners, using a permanent black felt marker. “From” equals the depth at start of box (top left corner) and “To” equals the depth at end of box (bottom right corner).

Figure 10.1 Core Logging Facility

10.2.5 CORE LOGGING

Core is logged in detail and all the data about lithology, alteration, mineralization, veining and structure is recorded manually on paper forms.

Sample lengths are variable, varying from a 20 cm minimum sample length to a 2 m maximum sample length. Samples are terminated at lithological boundaries when possible.

Since late 2008, NAP's Exploration office in Thunder Bay has used the Fusion Data Management (Fusion) suite of software by Century Systems Technology Inc. (acquired by CAE Mining in January 2011) to manage borehole data including analytical results. Boreholes are created within the software and geologists and technicians enter the required information including collar, survey, geological logs, RQD, specific gravity and lists for sample analysis. ActLabs reports analytical results digitally in a format that is imported directly into the Fusion software. Copies of the result certificates are maintained on the exploration file server as .pdf files, catalogued by certificate number.

10.2.6 CORE PHOTOS

The core boxes are photographed in numerical order and five boxes are photographed at a time (Figure 10.2). Core photos are backed-up daily on the main office server by the logging geologist.

Figure 10.2 Core Photo Setup

10.2.7 CORE SAMPLING

From the core station, core is transferred to pallets located outside the core cutting facility (Figure 10.3).

Sample intervals are transferred to a sample booklet with pre-printed sample numbers.

One box of core is loaded to the core-splitting bench at the time, and the interval of core to be sampled is split in half using a pneumatic wedge splitter (Figure 10.4). One-half of the core is placed in a clean transparent bag, which has been previously labeled, and a pre-printed sample tag is placed inside of the bag. The bag is then sealed with a plastic tie strap and placed on the floor in an orderly manner for easy tracking. The core-splitting bench and splitter are thoroughly cleaned after each sample to avoid contamination.

The remaining half of the core is put back in the core box, and it is transferred to covered storage racks where it will be kept for future reference. In general, one in five Offset Zone boreholes are split to maintain the Lac des Iles Mine core reference library with the remaining boreholes whole-core sampled. Individual samples are placed in plastic sample bags.

QA/QC samples are inserted into the sample stream (see Section 11.0 for details).

While on site, all the samples are stored in a secure warehouse, only accessible to key NAP personnel.

Figure 10.3 Core Storage Prior to Sampling

Figure 10.4 Core Splitting Setup

10.2.8 SAMPLE SHIPPING

The 2011 and 2012 Offset Zone drill core was processed both at the Lac des Iles Mine site and the NAP Thunder Bay Exploration office. Drill core is generally sent for analysis in batches of 70 samples containing six certified standards and blanks.

Multiple samples are packaged in rice bags and groups of rice bags totalling up to 140 samples are placed in auto parts transport bins (Figure 10.5) (or shrink-wrapped on pallets if no bins are available) before being transported to ActLabs' Thunder Bay facility by Courtesy Courier.

Figure 10.5 Sampling Shipping Container

10.2.9 AUTHOR'S OPINION

Tetra Tech is of the opinion that the diamond drilling, core logging and core sampling procedures meet industry standards and are acceptable to use for resource estimation.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 SAMPLE PREPARATION, ANALYSES, AND SECURITY 2003 TO 2008

From 2003 to mid-2008, the Lac des Iles Mine drillhole core was 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, 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 in the building. Prior to 2009, all samples were processed at the mine. Core samples are trucked by exploration staff or by Courtesy Courier from the mine to ActLabs (or Accurassay in the past) in Thunder Bay and from the Thunder Bay office to the laboratory.

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 ANALYTICAL METHODOLOGY

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 labelled small 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 10.0 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 and 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 Lac des Iles Mine by email and hardcopy in the mail.

11.2 SAMPLE PREPARATION, ANALYSES, AND SECURITY 2009 TO 2010

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. The mine itself has a gatehouse and barriers to restrict public access. Core samples are 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 ANALYSIS

PALLADIUM-PLATINUM-GOLD

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).

NICKEL-COPPER-COBALT-SILVER

Four acid, near total, digestion and ICP-OES finish.

11.3 SAMPLE PREPARATION, ANALYSES, AND SECURITY 2011 TO 2012

For the 2011 through 2012 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:

- drying at 60°C
- size reduction in a TM Engineering Ltd. Terminator jaw crusher to 80% passing -10 mesh (1.7 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).
- cleaner sand is used between each sample
- pulps and rejects are retained at ActLabs for a variable period of time
- once QA/QC checks have been completed the pulps and rejects are then shipped back to the mine where they are stored in sea-can shipping containers.

11.3.2 SAMPLE ANALYSIS

PALLADIUM-PLATINUM-GOLD (ACTLABS SCHEDULE OF SERVICES 1C-OES FIRE ASSAY ANALYSIS)

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.

NICKEL-COPPER-COBALT-SILVER (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.

11.4 QA/QC

Due to the atypical character of the mineralization in the Offset Zone, NAP Exploration staff decided in 2009 to custom-make their QA/QC duplicate program. In addition to the laboratories internal duplicates (known as lab dup and prep dup), NAP randomly and specifically selected samples for re-assays either through ActLabs or SGS. The pulps of those samples re-assayed by ActLabs are considered as the duplicate samples, whereas the samples sent to SGS as the check samples.

11.4.1 DUPLICATE ANALYSES

Around 10% of pulp splits, from 2009 to 2011 Offset Zone drilling were randomly selected and re-run by ActLabs, in order to control the reproducibility of the fire assay method results. The analytical technique used by ActLabs to re-run those pulps is identical to the one used for measuring the original assays.

NAP compiled ActLabs duplicates analyses on pulps for 2009, 2010, and 2011 drilling, and compared these with the original ActLabs assays (Table 11.1).

Table 11.1 NAP 2009 to 2011 Duplicate QA/QC Results

	Type	Pd (ppb)	Pt (ppb)	Au (ppb)
2009 to 2011				
Mean	Original Assay	1,366.95	139.20	105.50
Mean	Duplicate Assay	1,370.69	138.85	103.42
% Difference	Original vs. Duplicate	-0.27	0.25	1.97
Minimum	Original Assay	2.5	2.5	1
Minimum	Duplicate Assay	2.5	2.5	1
Maximum	Original Assay	39,400	2,530	8,580
Maximum	Duplicate Assay	40,400	2,530	9,090
N	Original Assay	6,243	6,244	6,244
N	Duplicate Assay	6,243	6,244	6,244
2009				
Mean	Original Assay	1,854.18	216.61	200.99
Mean	Duplicate Assay	1,884.45	218.88	201.53
% Difference	Original vs. Duplicate	-1.63	-1.05	-0.27
Minimum	Original Assay	2.5	2.5	4
Minimum	Duplicate Assay	2.5	2.5	6
Maximum	Original Assay	39,400	1,990	2,310
Maximum	Duplicate Assay	40,400	2,130	1,910
N	Original Assay	373	373	373
N	Duplicate Assay	373	373	373
2010				
Mean	Original Assay	1,017.34	96.06	80.78
Mean	Duplicate Assay	1,003.15	94.72	81.07
% Difference	Original vs. Duplicate	1.39	1.4	-0.35
Minimum	Original Assay	2.5	2.5	1
Minimum	Duplicate Assay	2.5	2.5	1
Maximum	Original Assay	25,900	2,530	8,580
Maximum	Duplicate Assay	30,400	1,790	9,090
N	Original Assay	1,717	1,718	1,717
N	Duplicate Assay	1,717	1,718	1,717
2011				
Mean	Original Assay	1,467.73	150.08	107.14
Mean	Duplicate Assay	1,476.50	149.92	103.85
% Difference	Original vs. Duplicate	-0.6	0.11	3.08
Minimum	Original Assay	2.5	2.5	1
Minimum	Duplicate Assay	2.5	2.5	1
Maximum	Original Assay	27,200	2,010	2,580
Maximum	Duplicate Assay	27,600	2,530	2,110
N	Original Assay	4,153	4,155	4,154
N	Duplicate Assay	4,153	4,155	4,154

For the drilling programs running from 2009 to 2011, 6,244 paired original and dup assays were compiled. Overall, the assays reproducibility was within 2% for the three precious metals measured by fire assay method.

The pairs were graphed in scatter plot for palladium (Figure 11.1). For the assaying programs from 2009 to 2011, the overall linear correlation is good at 97% (r2 = 0.9698) with an almost 1:1 correlation between original and duplicate values. The same observation is valid if the assays are broken down per year of drilling (Figure 11.2 to Figure 11.4).

Most of samples with significant disparity were looked at. Most of the lack of reproducibility can be associated to the extreme low concentration of the metals in the samples (detection limit of the analytical method), inversion of samples name during re-assays, or wrong calibration of the analytical machines. In the case of the latest, triplicates are then run to confirm the suspicion of analytical error. Whatever the reason, samples with significant discrepancy are reported in the database data entry control sheet.

Figure 11.1 NAP 2009 to 2011 Palladium Duplicate Plot

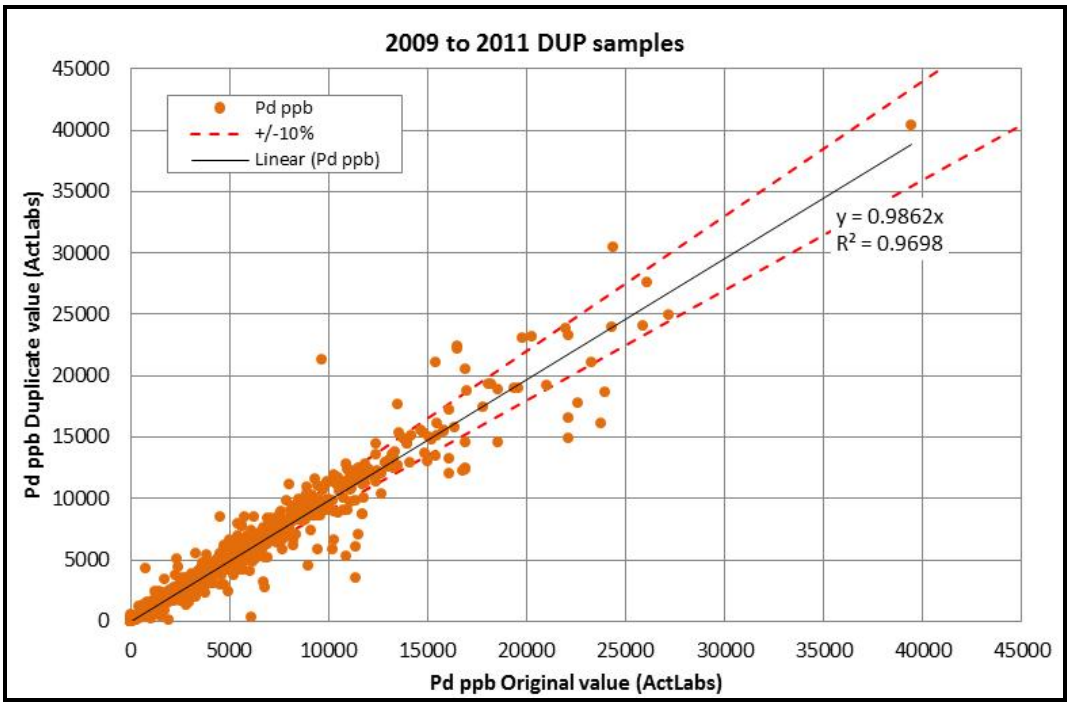


Figure 11.2 NAP 2009 Palladium Duplicate Plot

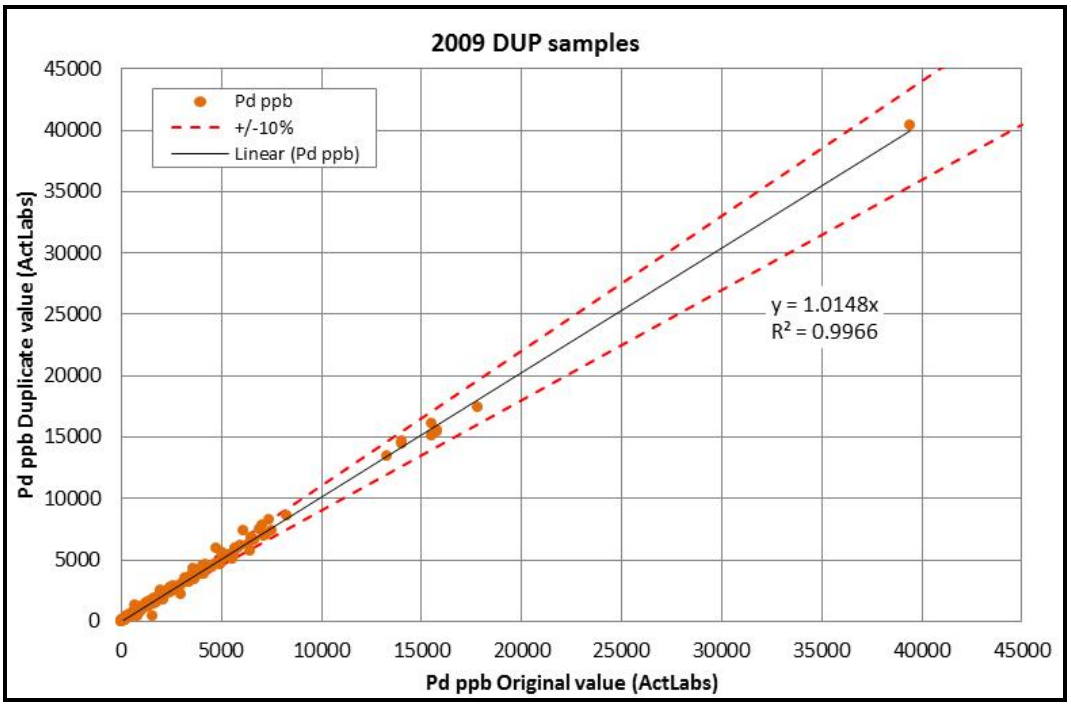


Figure 11.3 NAP 2010 Palladium Duplicate Plot

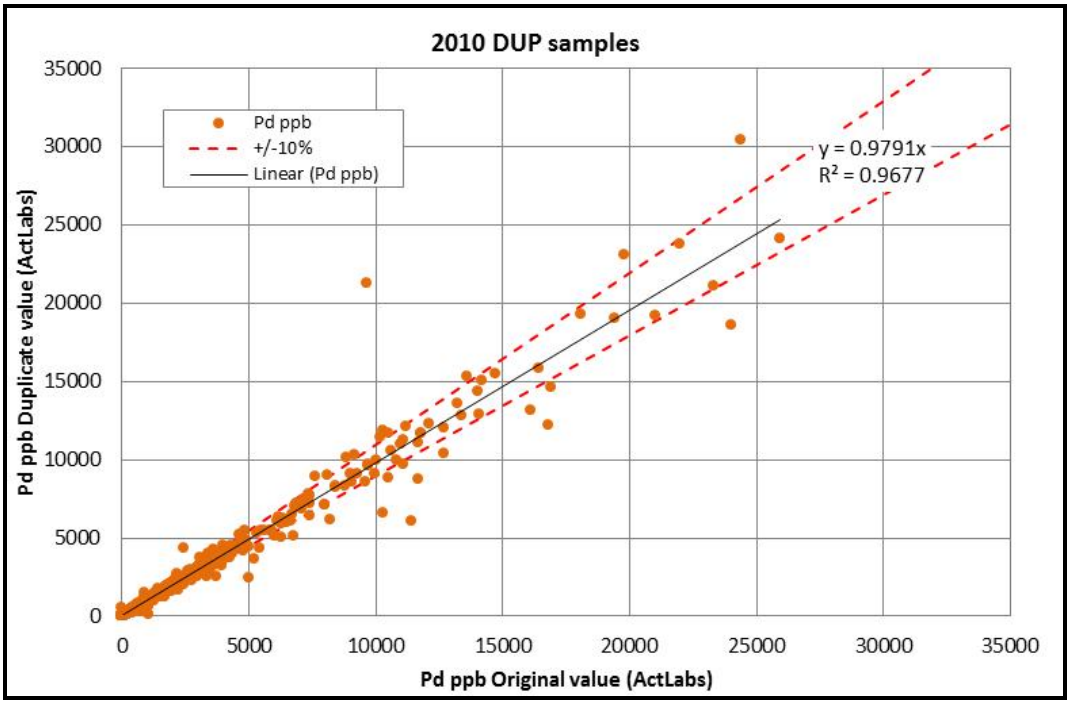
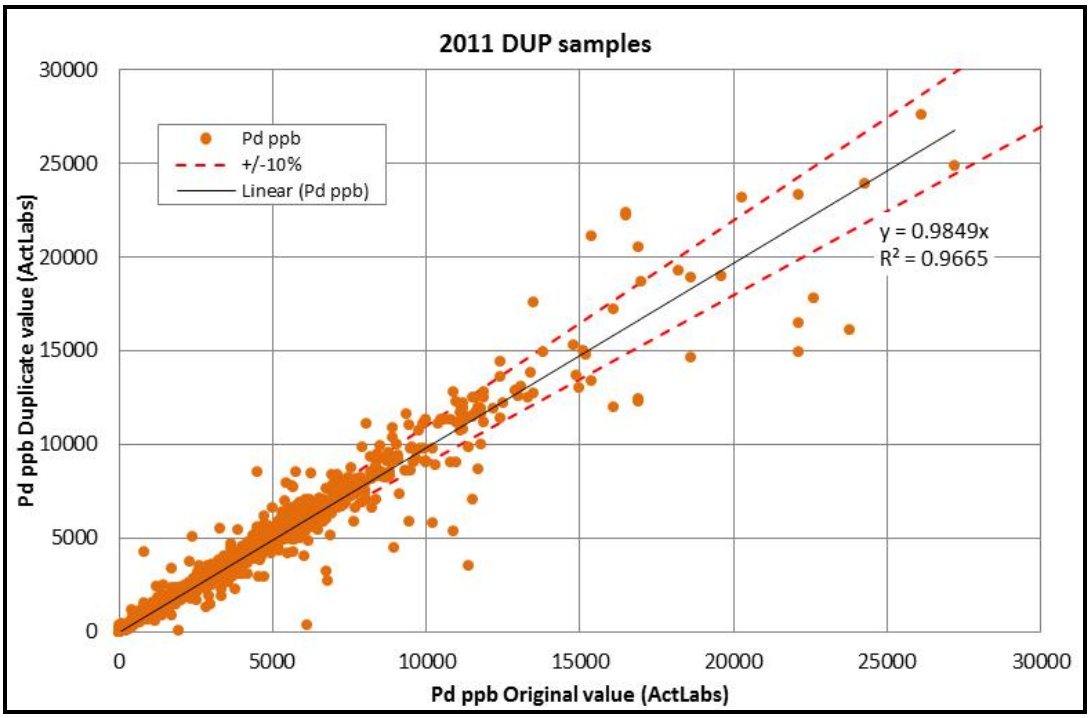


Figure 11.4 NAP 2011 Palladium Duplicate Plot



11.4.2 CHECK ANALYSES

Approximately 10% of the pulp splits, from the expected high-grade zone, were forwarded to SGS, in order to check the reproducibility and consistency of the high-grade zone. For base metals (copper, cobalt and nickel), SGS uses four acid digestion and ICP-atomic emission spectroscopy (AES) analysis (code ICP40B). The palladium, platinum, and gold metals are determined by fluxing and fire assay fusion with lead oxide collection, two acid digestion of the doré bead and ICP-AES analysis (FAI323).

NAP compiled SGS laboratory check analyses on pulps for 2009, 2010, and 2011 drilling, and compared these with the original ActLabs assays. The results are presented in Table 11.2.

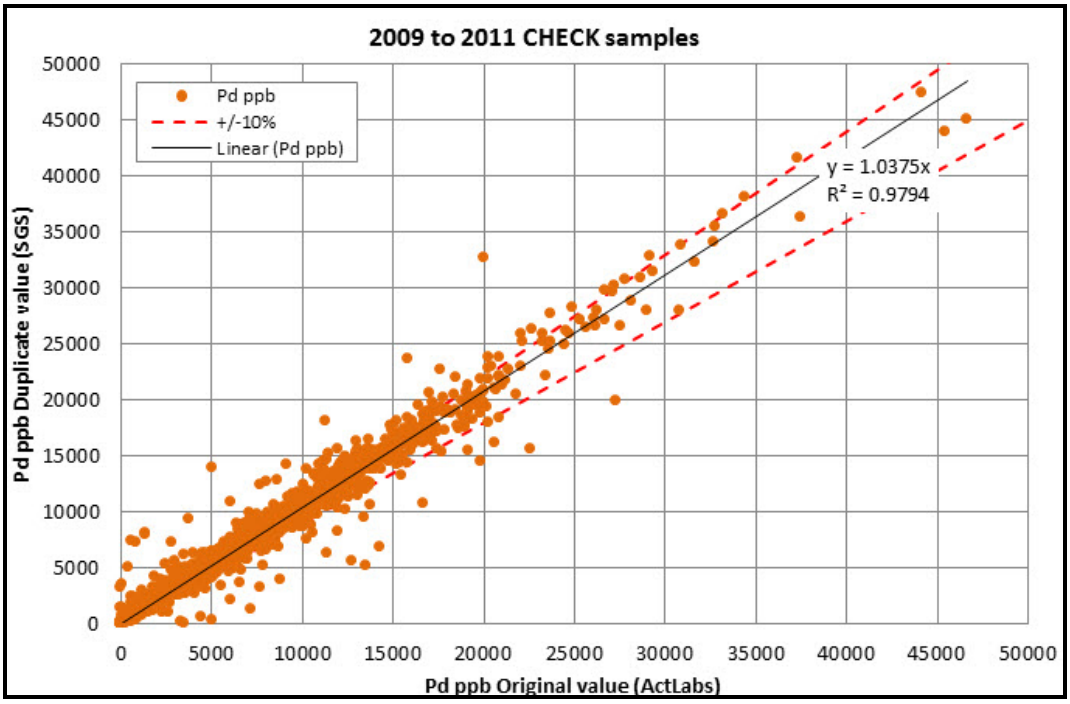
Table 11.2 NAP 2009 to 2011 Check Sample QA/QC

	Type	Pd (ppb)	Pt (ppb)	Au (ppb)	Cu (ppm)	Ni (ppm)	Co (ppm)
2009 to 2011							
Mean	Original Assay	3,669.15	292.60	241.22	818.88	1048.46	69.51
Mean	Duplicate Assay	3,843.84	306.85	248.88	804.76	932.71	64.48
% Difference	Original vs. Duplicate	-4.76	-4.87	-3.18	1.72	11.04	7.24
Minimum	Original Assay	2.5	2.5	1	0.5	5	0.5
Minimum	Duplicate Assay	2.5	5	2.5	0.25	5	0.5
Maximum	Original Assay	46,700	3,180	5,150	5,000	5,000	642
Maximum	Duplicate Assay	47,300	3,240	6,750	10,000	10,000	596
N	Original Assay	6,295	6,294	6,294	6,768	6,768	6,768
N	Duplicate Assay	6,295	6,294	6,294	6,768	6,768	6,768
2009							
Mean	Original Assay	3,464.93	273.89	203.78	711.08	959.74	61.89
Mean	Duplicate Assay	3,546.98	285.24	207.02	688.64	881.08	57.59
% Difference	Original vs. Duplicate	-2.37	-4.14	-1.59	3.16	8.2	6.95
Minimum	Original Assay	2.5	2.5	1	5	18	3
Minimum	Duplicate Assay	2.5	5	2.5	7.9	11	4
Maximum	Original Assay	34,400	2,220	2,080	4,790	5,000	260
Maximum	Duplicate Assay	38,100	2,010	2,150	4,490	7,470	249
N	Original Assay	904	904	904	904	904	904
N	Duplicate Assay	904	904	904	904	904	904
2010							
Mean	Original Assay	3,988.70	288.63	246.03	746.22	1,018.89	63.60
Mean	Duplicate Assay	4,162.18	303.77	263.76	739.60	907.66	58.90
% Difference	Original vs. Duplicate	-4.35	-5.25	-7.2	0.89	10.92	7.38
Minimum	Original Assay	2.5	2.5	1	0.5	6	2
Minimum	Duplicate Assay	2.5	5	2.5	1.6	5	0.5
Maximum	Original Assay	37,300	2,680	4,170	4,920	5,000	193
Maximum	Duplicate Assay	41,500	2,640	6,750	5,410	5,160	210
N	Original Assay	1,431	1,431	1,431	1,599	1,599	1,599
N	Duplicate Assay	1,431	1,431	1,431	1,599	1,599	1,599
2011							
Mean	Original Assay	3,600.30	298.31	248.02	868.97	1,078.35	73.35
Mean	Duplicate Assay	3,796.57	312.90	253.05	853.80	953.05	68.04
% Difference	Original vs. Duplicate	-5.45	-4.89	-2.03	1.75	11.62	7.24
Minimum	Original Assay	2.5	2.5	1	1	5	0.5
Minimum	Duplicate Assay	2.5	5	2.5	0.25	6	0.5
Maximum	Original Assay	46,700	3,180	5,150	5,000	5,000	642
Maximum	Duplicate Assay	47,300	3,240	3,760	10,000	10,000	596
N	Original Assay	3,960	3,959	3,959	4,265	4,265	4,265
N	Duplicate Assay	3,960	3,959	3,959	4,265	4,265	4,265

For the drilling programs running from 2009 to 2011, more than 6,700 paired original and check assays were compiled. Overall, the assays reproducibility is within 5% for the three precious metals measured by fire assays method, and less than 12% for the base metals measured by ICP technique.

The pairs were graphed in scatter plot for palladium (Figure 11.5). For the assaying programs from 2009 to 2011, the overall linear correlation is good at 98% (r2 = 0.9794) with an almost 1:1 correlation between original and duplicate values. The same observation is valid if the assays are broken down per year of drilling.

Figure 11.5 Palladium Scatter Plot



Samples showing significant disparity between their original assay and their duplicate one were looked at. Most of the lack of reproducibility can be associated to the extreme low concentration of the metals in the samples (detection limit of the analytical method), inversion of samples name during re-assays, or wrong calibration of the analytical machines. In the case of the latest, triplicates are then run to confirm the suspicion of analytical error. Whatever the reason, samples with significant discrepancy are reported in the database data entry control sheet.

11.4.3 BLANKS

Blanks were inserted into the drill core continuous sampling series at regular intervals during the 2010 to 2012 drill programs.

Throughout 2010 and in the start of 2011, the first three samples submitted for each drillhole were a blank sample (001) followed by a standard sample (002) and then by another blank sample (003). Blank samples were then inserted into the sample stream with the following pattern, blank (020), blank (040), blank (080), blank (100). All of the blank material was sourced from CDN Resource Laboratories (CDN) of Langley, British Columbia. A number of blank materials have been used as the available supply changes and are summarized by year in Table 11.3.

Table 11.3 Summary of Blank and Standard Reference Material

Year	Blank Sample Reference Material	Standard Sample Reference Material	Source
2010	CDN-BL-4, CDN-BL-6 and CDN-BL-7	CDN-PGMS-15, CDN-PGMS-16 and CDN-PGMS-17	CDN Resource Laboratories
2011	CDN-BL-7, CDN-BL-8, CDN-BL-9 and CDN-BL-10	CDN-PGMS-15, CDN-PGMS-17, CDN-PGMS-19 and CDN-PGMS-20	CDN Resource Laboratories
-	Blank Crush (marble)	-	Lockstone Yard and Patio Centre, Thunder Bay
2012	CDN-BL-9 and CDN-BL-10	CDN-PGMS-10, CDN-PGMS-15, CDN-PGMS-17, CDN-PGMS-18, CDN-PGMS-19. CDN-PGMS-20	CDN Resource Laboratories
-	Blank Crush (marble)	-	Lockstone Yard and Patio Centre, Thunder Bay

In April/May of 2011, a problem was noted by exploration staff regarding the frequency of CDN-BL-7 failures. The main concern was gold failures but slightly elevated palladium was also noted (Appendix B). CDN was contacted who were already aware of the issue and all unused CDN-BL-7 material was returned to CDN. Bulk, crushed marble purchased from a local landscape yard was used instead of the CDN-BL-7 blanks in approximately 16 drillholes (noted in database as “blank crush”). Once the new CDN-BL-8 blank reference material was received, a new sampling series was established.

From mid-2011 through 2012, most samples were submitted to ActLabs in batches of 70 samples except for those at the end of the holes. The reference materials were inserted following this pattern: batch one contained reference material at 001 (blank crush), 002 (CDN standard), 003 (CDN blank), 020 (CDN blank), 040 (blank crush), 060 (CDN standard); batch 2 contained reference material at 071 (blank crush), 072 (CDN standard), 073 (CDN blank), 090 (CDN blank), 110 (blank crush), 130 (CDN standard).

The CDN blank samples and the blank crush samples are considered failed if one or more of the palladium, platinum, or gold results exceed the maximum allowable 10 ppb. Once a failure is detected, the lab is instructed to re-assay from pulps all

samples from the preceding blank that passed to the following blank that passed. These re-assays are then used in the drillhole database. The graphs displaying the original results of the blanks samples from 2010 through May 2012 for all holes drilled on the Property are available in Appendix B.

11.4.4 STANDARD REFERENCE MATERIAL

Standard reference material samples were inserted into the drill core continuous sampling series at regular intervals during the 2010 to 2012 drill programs.

Throughout 2010 and 2011, the first three samples submitted for each drillhole were a blank sample (001) followed by a standard sample (002) and then by another blank sample (003). Blank and standard reference materials were then inserted at every 20th sample position downhole in the pattern "blank (020), blank (040), standard (060), blank (080), blank (100), standard (120)". All of the standard reference material was sourced from CDN. A number of standard reference materials have been used as the available supply changes. These are summarized by year in Table 11.3.

The CDN standard samples results for palladium, platinum and gold are assessed based on the recommended values and standard deviations reported on the CDN certificates. Palladium fails if it exceeds two standard deviations. Platinum and gold fail if they exceed three standard deviations. Once a failure is detected, the lab is instructed to re-assay from pulps all samples from the preceding standard that passed to the following standard that passed. These re-assays are then used in the drillhole database. The graphs displaying the original results of the standards from 2010 through May 2012 for all holes drilled on the Property are available in Appendix B. Copies of the certificates are available in Appendix C.

11.4.5 AUTHOR'S OPINION

Tetra Tech is of the opinion that the sample preparation and analytical procedures meet industry standards, and that the analytical results are acceptable to use for resource estimation.

The review of the QA/QC program indicates that the program is operating to industry standards, and the data is acceptable for the use in resource estimation.

12.0 DATA VERIFICATION

Tetra Tech carried out an internal validation of the diamond drillhole file against the original drillhole logs and assay certificates. The validation of the data files was completed on 18 of the 172 drillholes in the database or approximately 10% of the dataset. Data verification was completed on collar coordinates, end-of-hole depth, down-the-hole survey measurements, and "from" and "to" intervals. No errors were encountered. A total of 10% of the assays data were validated against the original assay certificate. The error rate from this validation was 0%.

All assays entered in the database as being below detection limit with a "<" sign were converted to half the detection limit and were not considered to be errors in the data. All the data was converted to a consistent unit as over the year different units were used (Table 12.1).

Table 12.1 Database Modifications

Element	Detection Limit Edits Made to Original Assay Data	Original Assay Unit	Database Unit	Conversion Used
Au	<2 = 1	ppb	ppm	0.001
Pd	<5 = 2.5	ppb	ppm	0.001
Pt	<5 = 2.5	ppb	ppm	0.001
Ag	<0.3 = 0.15	ppm	ppm	N/A
Co	N/A	ppm	per	0.0001
Cu	<1 = 1, >500 = 500	ppm	per	0.0001
Fe	N/A	per	per	N/A
Ni	<0.1 = 0.005 >5000 = 5000	ppm	per	0.0001
S	< 0.01 = 0.005	per	per	N/A

Tetra Tech imported the drillhole data into the Datamine™ program, which has a routine that checks for duplicate intervals, overlapping intervals, and intervals beyond the end of hole. The errors identified in the routine were checked against the original logs and corrected.

Tetra Tech visually observed the diamond drill setups underground at the Lac des Iles Mine. Tetra Tech also observed the mineralized zone underground.

Eighteen independent samples of mineralized course rejects were collected for check assaying representing different mineralization grade ranges. The samples were sent to ALS Minerals in Thunder Bay, Ontario. The samples were prepared in Thunder Bay and the pulps were shipped by ALS Minerals to Vancouver, British

Columbia for analysis. Each course reject was processed twice to generate two duplicate assay results for each original sample (Table 12.2).

ALS Minerals is accredited to international quality standards through ISO/IEC 17025 (ISO/IEC 17025 includes ISO 9001 and ISO 9002 specifications) with CAN-P-1579 (Mineral Analysis).

The results of the validation check samples for palladium and platinum indicate that the results of the check samples are typically high compared to the originals samples (Figure 12.1 and Figure 12.2).

The following QPs completed a site visit of the Property:

- Todd McCracken, P.Geo. visited the site from May 23 to 25, 2012
- Rick McBride, P.Eng. visited the site from November 5 to 8, 2012
- Todd Kanhai, P.Eng. visited the site from November 5 to 7, 2012.

12.1 QP OPINION

The Offset data set is deemed to be valid and is acceptable for the use in resource estimation.

Table 12.2 Validation Check Sample Results

Hole	Sample	Lithology	From	To	North American Palladium Samples			Tetra Tech Check Samples (Course Rejects)			Tetra Tech Check Samples Duplicates (Course Rejects)		
					Pd (ppm)	Pt (ppm)	Au (ppm)	Pd (ppm)	Pt (ppm)	Au (ppm)	Pd (ppm)	Pt (ppm)	Au (ppm)
11-829	11-815-080	GAB	70.00	71.00	0.954	0.084	0.010	4.01	0.333	0.054	4.12	0.347	0.053
11-829	11-815-112	GAB/VEIN	100.00	101.00	4.190	0.256	0.183	4.43	0.258	0.211	4.18	0.248	0.219
11-829	11-815-118	PYXT	106.00	107.00	3.910	0.262	0.231	4.89	0.276	0.236	4.65	0.258	0.198
11-822	11-820-128	MGAB	116.00	117.00	0.840	0.164	0.511	0.798	0.163	0.464	0.855	0.174	0.475
11-822	11-820-140	PYXT	127.00	128.00	1.910	0.160	0.294	2.43	0.181	0.312	2.42	0.185	0.362
11-822	11-820-150	PYXT	134.00	135.00	8.460	0.468	0.235	9.59	0.49	0.34	9.55	0.5	0.26
11-820	11-822-046	GAB	40.00	41.00	0.869	0.126	0.044	0.973	0.168	0.062	0.956	0.159	0.043
11-820	11-822-087	MGAB	77.00	78.00	2.290	0.164	0.030	2.43	0.169	0.035	2.42	0.162	0.034
11-820	11-822-105	PYXT	94.00	95.00	6.700	0.413	0.180	9.04	0.568	0.313	9.03	0.6	0.296
11-815	11-829-089	GAB	79.00	80.00	0.841	0.065	0.050	1.83	0.142	0.072	1.98	0.172	0.133
11-815	11-829-132	GAB	119.00	120.00	4.020	0.263	0.549	5.06	0.33	0.733	5.12	0.362	0.782
11-815	11-829-170	PYXT	153.00	154.00	3.920	0.193	0.129	3.94	0.207	0.15	4.08	0.209	0.164
12-806	12-806-150	MGAB	134.00	135.00	0.814	0.132	0.355	1.37	0.158	0.425	1.41	0.153	0.485
12-806	12-806-163	PYXT	146.00	147.00	5.800	0.330	0.539	7.9	0.419	0.529	7.99	0.42	0.517
12-806	12-806-173	PYXT	156.00	157.00	3.220	0.386	0.251	4.01	0.515	0.321	4.02	0.485	0.451
12-809	12-809-033	GAB	28.00	29.00	0.900	0.083	0.007	1.015	0.092	0.012	0.978	0.085	0.01
12-809	12-809-111	GAB	99.00	100.00	2.040	0.129	0.028	2.22	0.182	0.042	2.18	0.169	0.04
12-809	12-809-120	PYXT	108.00	109.00	4.920	0.358	0.122	5.26	0.373	0.159	5.54	0.357	0.147

Figure 12.1 Palladium Validation Plot

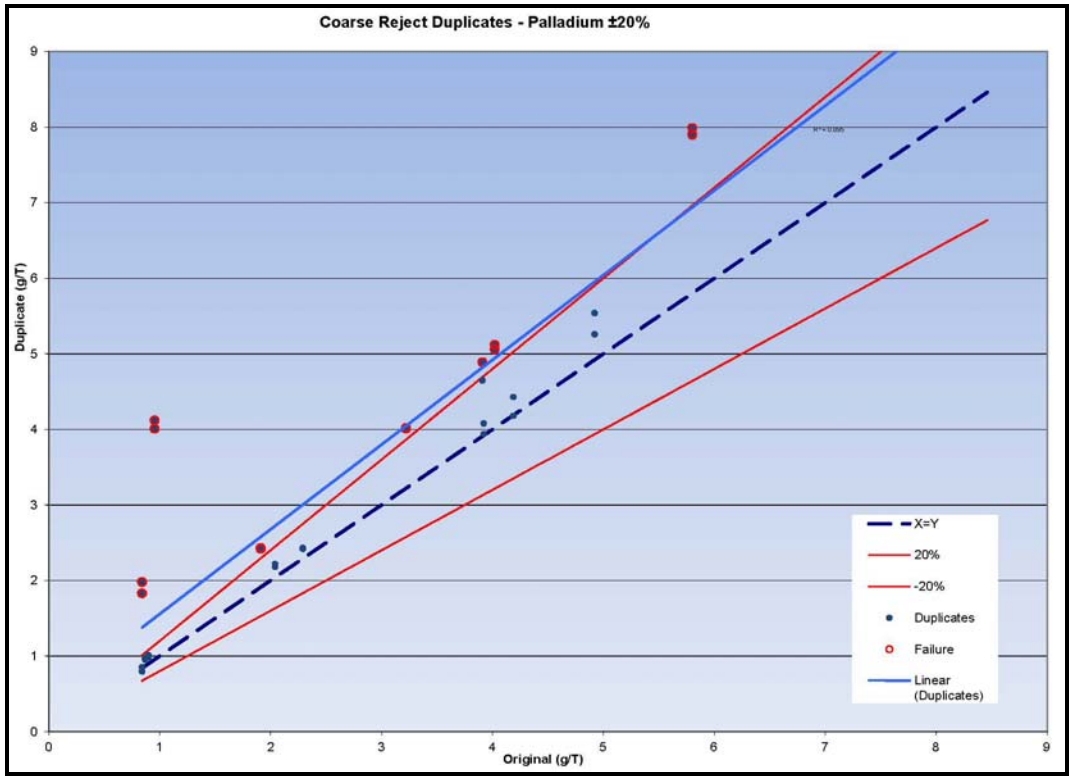
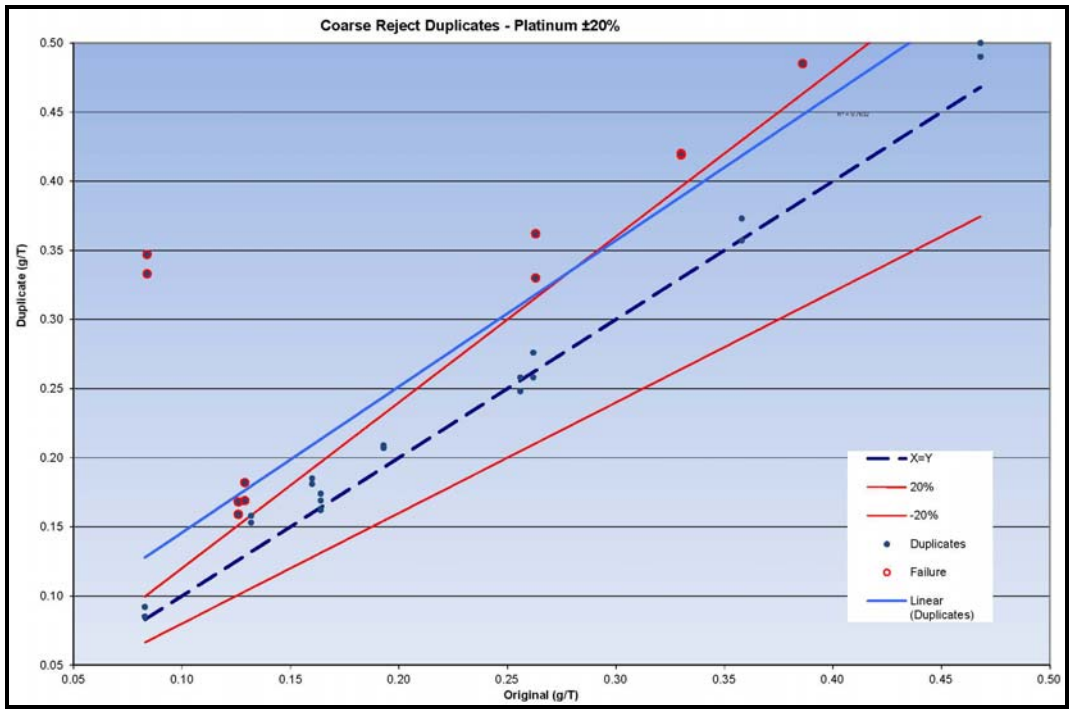


Figure 12.2 Platinum Validation Plot



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Two historical metallurgical test programs were carried out; one by SGS and the other by XPS. The titles of the two test reports are:

- SGS - Grinding Circuit Evaluation for the Lac Des Iles Mine, June 2008 (SGS 2008)
- XPS - Lac des Iles Roby and Offset Zone Ore Characterization Report, June 2010 (XPS 2010).

It should be noted that these two test programs and reports were undertaken by the Lac des Iles Mine prior to review by Tetra Tech. The sections below summarize the data presented in each of these test reports. The SGS (2008) and XPS (2010) reports are included in Appendix D of this report.

13.1 SGS GRINDING CIRCUIT EVALUATION

13.1.1 GRINDING TEST WORK

The current Lac des Iles Mine operation is run on an intermittent operating schedule as the mill is 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 in 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 t/d 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 t/d

- rod milling, ball milling and vertimilling at 3,500 t/d.

Table 13.1 summarizes the breccia grindability test results 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 Grindability Test Results

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

Source: SGS (2008)

The ultimate goal of these simulations was to determine a feasible flowsheet that would limit capital and operating costs. Pebble milling is an excellent option, but the complexity of adding the circuit into the current infrastructure made it a non-ideal option. Refurbishing the equipment in the 2,400 t/d mill also could not meet the long-term tonnage requirement.

Given these constraints, the preferred option of Lac des Iles Mine is 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 the defined size. This particular circuit was not one of the configurations simulated in this study. It is the configuration currently used at Lac des Ile Mine with a 13,000 t/d throughput on the intermittent process schedule.

13.2 XPS Roby and Offset Zone Ore Characterization

XPS was retained for characterization of Roby and Offset Zone ore. Drill core composite samples from each zone were submitted to XPS by Lac des Iles Mine in February 2010 for this purpose, with XPS' assistance in 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 (LCT) 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 depressant tests
- LCT for metallurgical performance of the Roby and Offset zones

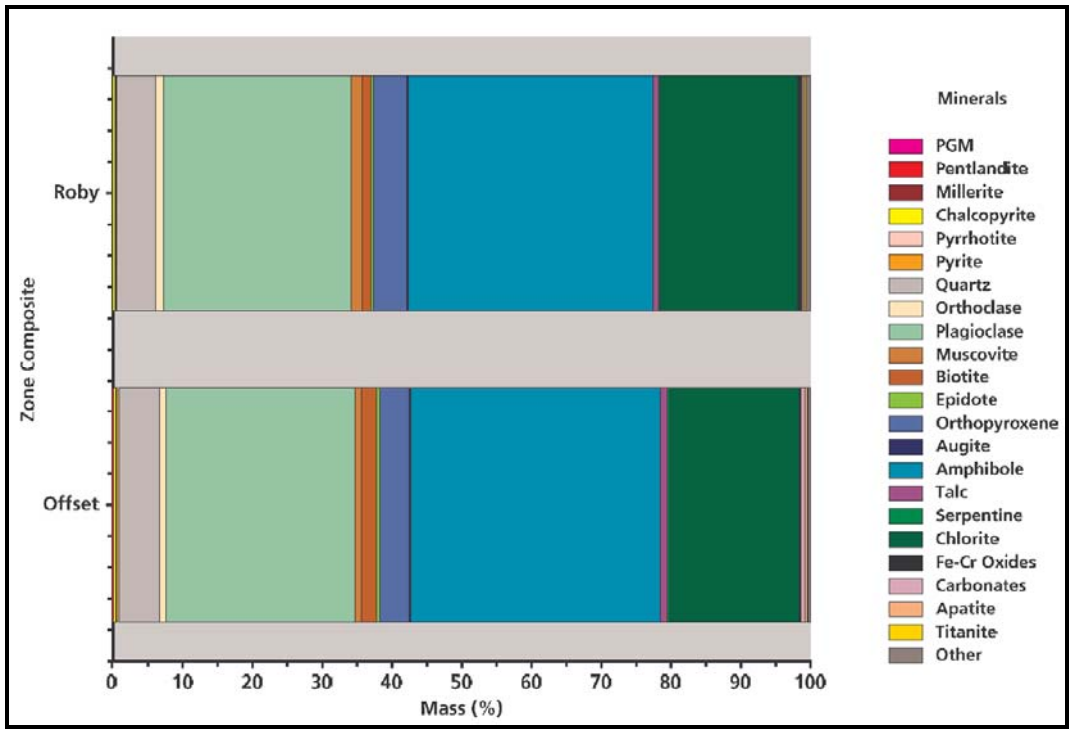
- Bond's ball mill work index for each zone composite sample
- gravity recovery tests.

13.2.1 MINERALOGY

MODAL MINERALOGY

The modal mineralogy of the Roby and Offset zones is 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 Modal Mineralogy of Roby and Offset Zones



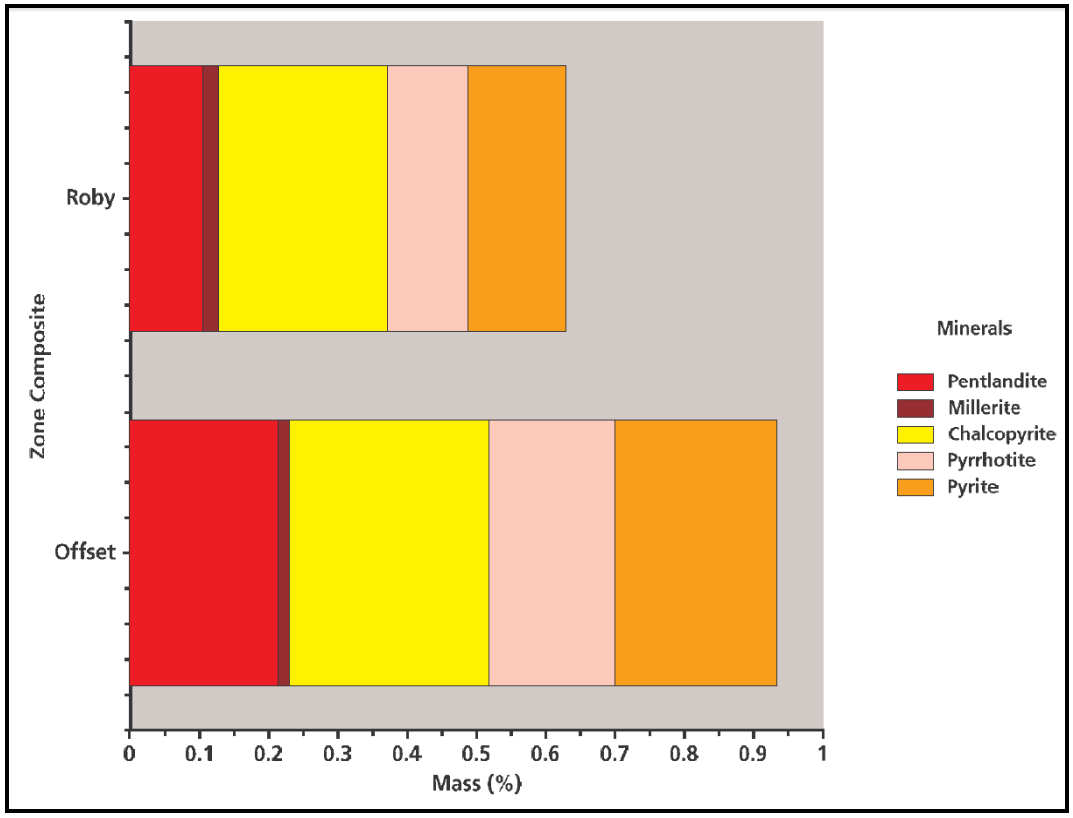
Source: XPS (2010)

The predominant difficulty in processing these ore types is in dealing with floatable gangue. The ore 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 is 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

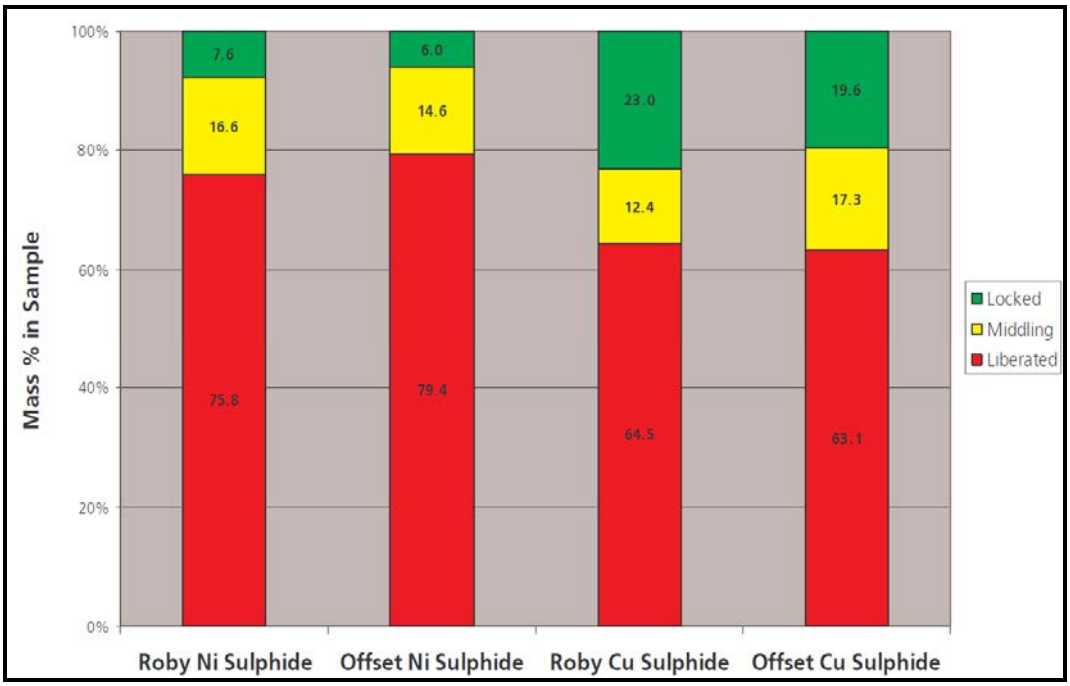


Source: XPS (2010)

NICKEL AND COPPER SULPHIDE LIBERATION

The liberation of both zones composites has been evaluated at P_{80} -75 µm grind size. This was the operation's grind size at the time of the report. The liberation of the nickel and copper sulphides is represented by the bars in Figure 13.3.

Figure 13.3 Total Liberation of Nickel and Copper Sulphide Roby and Offset Zones



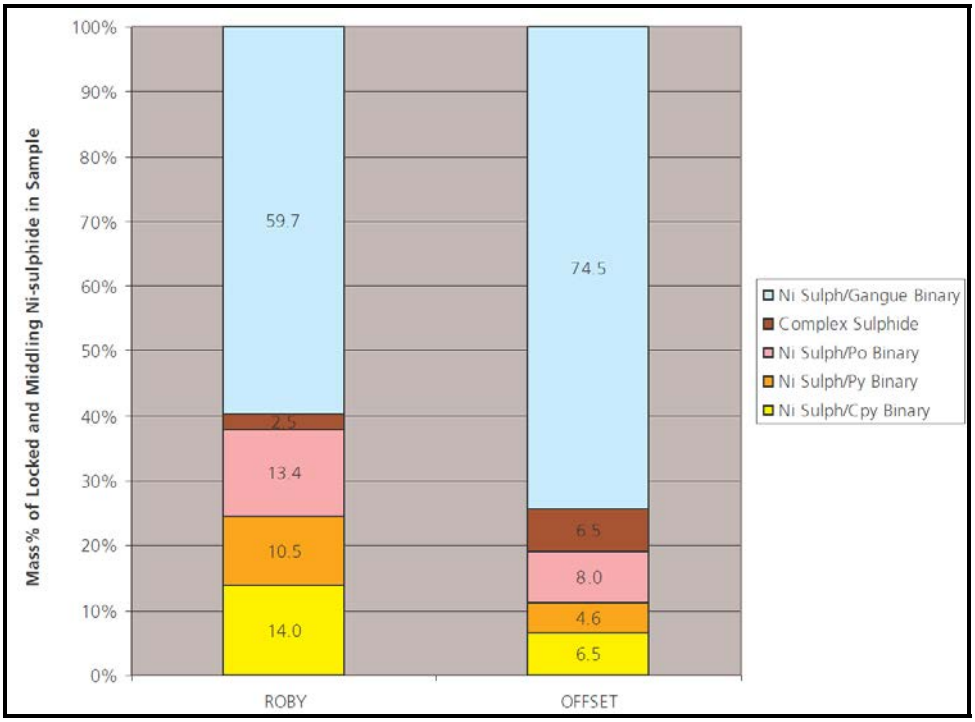
Source: XPS (2010)

The nickel sulphides are a compilation of the total liberation of the pentlandite and millerite and the copper sulphide is representative of the liberation of the chalcopyrite. The nickel sulphides are moderately well liberated while the copper sulphides are contained in more locked particles. This result was surprising, given Lac des Iles Mine's history of poor nickel recoveries but excellent copper recoveries.

NICKEL DEPORTMENT AND RECOVERY ISSUES

In an attempt to explain the historically poor nickel recoveries at Lac des Iles Mine, 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 completed and is 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 lower nickel recovery than for the Roby Zone.

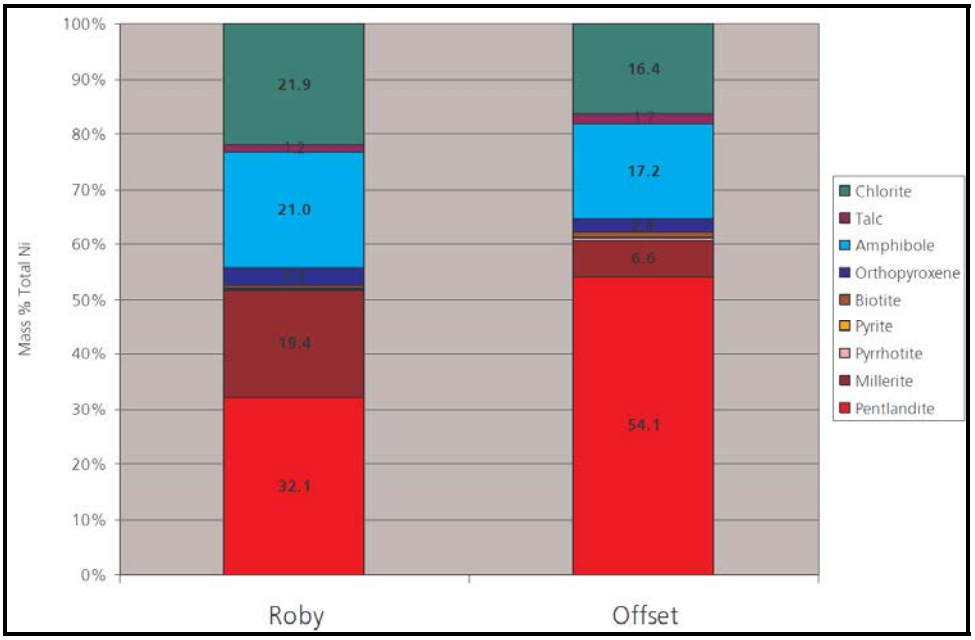
Figure 13.4 Nickel Sulphide Locking Characteristics



Source: XPS (2010)

Figure 13.5 shows that 52% and 61% 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)

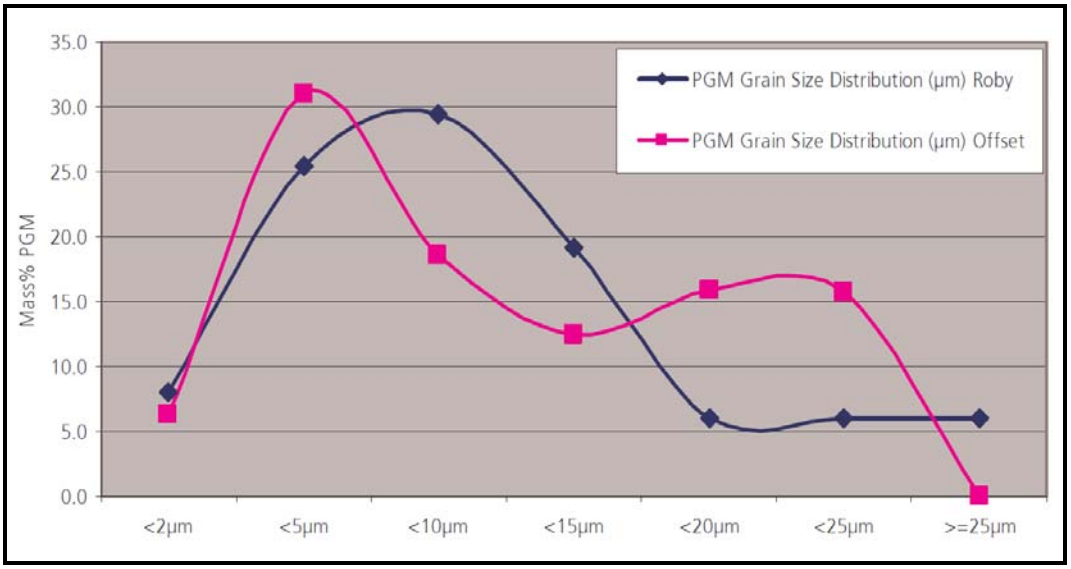
There is a considerable amount of nickel associated with gangue minerals which may be lost to tails. Given that the historical nickel recovery has been in the order of 30%, there is a great deal of potential to improve nickel recovery and will require further investigation into the flotation process itself.

PLATINUM GROUP METALS MINERALOGY

The predominant palladium minerals are sulphides (vysotskite), tellurides (kotulskite, merenskyite), and antimonides (mertieite). There are minor amounts of palladium in gold/silver species (electrum), arsenides, and other tellurium/bismuth minerals. By mass, the majority of platinum group metals (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)

13.2.2 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} values of 38 to 75 µm
- two levels of carboxymethyl cellulose (CMC) addition and the use of Aero 3477 (a PGM promoter)

- staged grinding.

The head grades of the feed to flotation for each sample are presented below in Table 13.2.

Table 13.2 Head Grade of Roby and Offset Composites

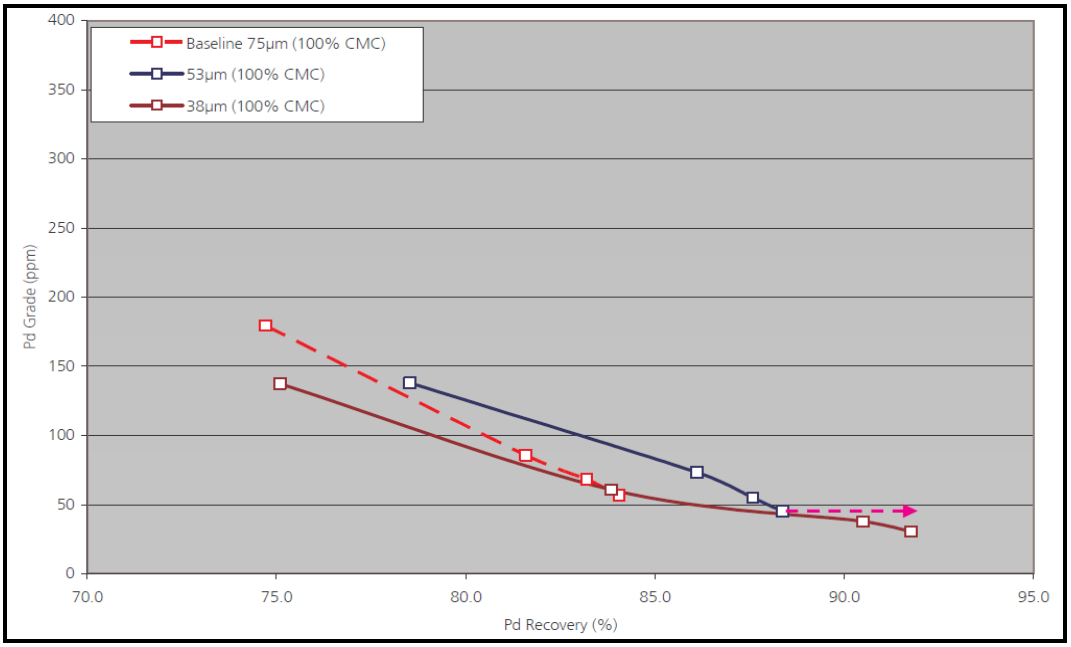
Head Grade	Ni (%)	Cu (%)	S (%)	MgO (%)	Au (g/t)	Pt (g/t)	Pd (g/t)
Roby Zone	0.12	0.07	0.18	12.20	0.37	0.31	6.12
Offset Zone	0.14	0.10	0.28	11.60	0.29	0.35	5.52

Source: XPS (2010)

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

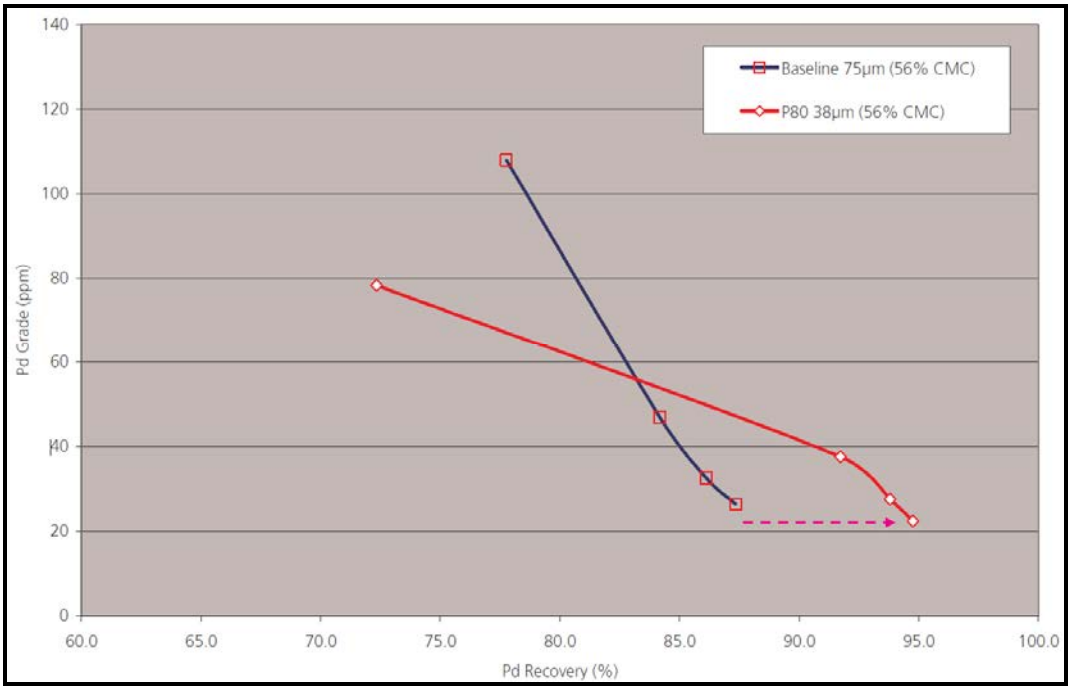


Source: XPS (2010)

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

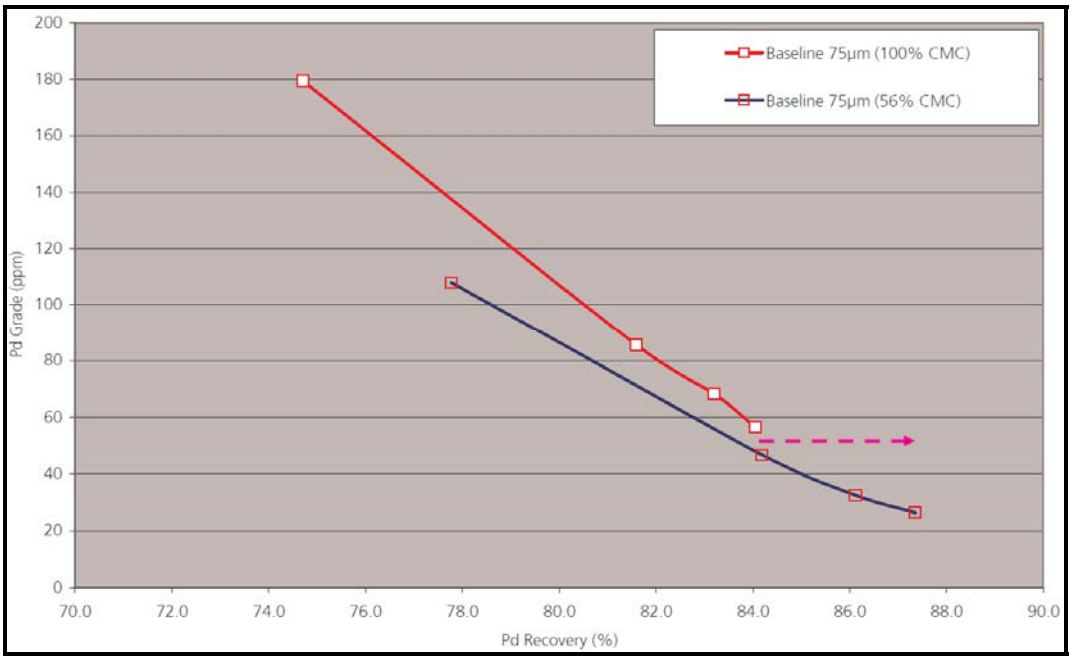


Source: XPS (2010)

ROUGHER-SCAVENGER – CMC DOSAGE

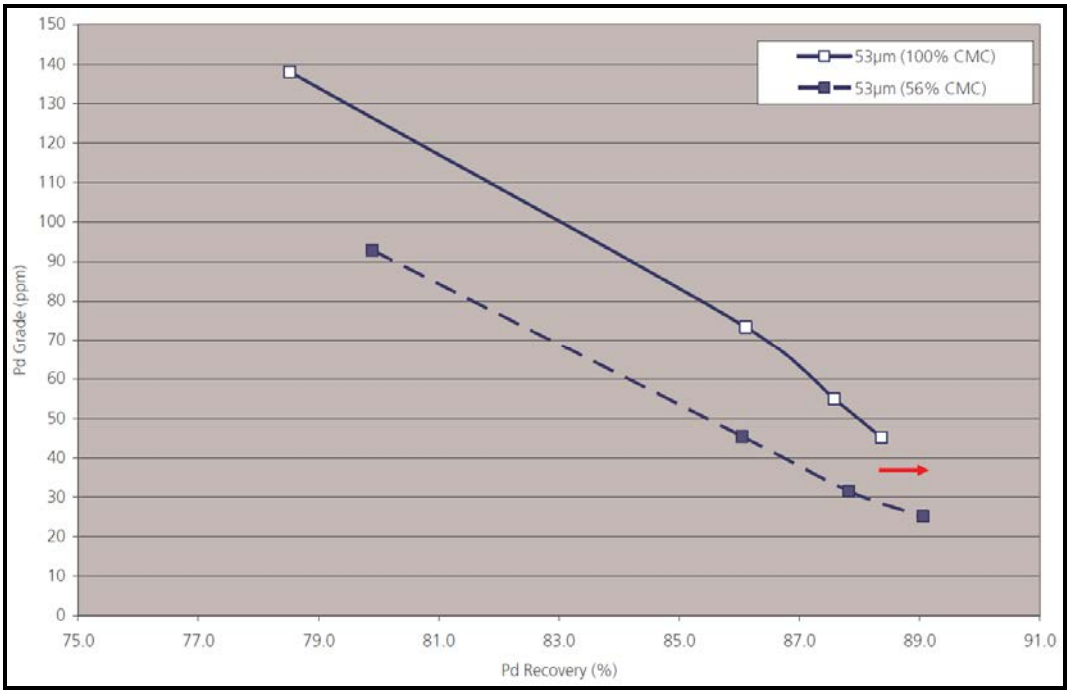
A series of rougher-scavenger tests using a primary grind of 75 µm were completed on the Roby Zone sample to determine the effect of CMC dosage on the palladium recovery. The results of these tests are presented in Figure 13.9. The recommended CMC dosage of 172 g/t was tested as well as 56% of the recommended dosage (i.e. 92 g/t). The test illustrates that the higher CMC dosage had 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)

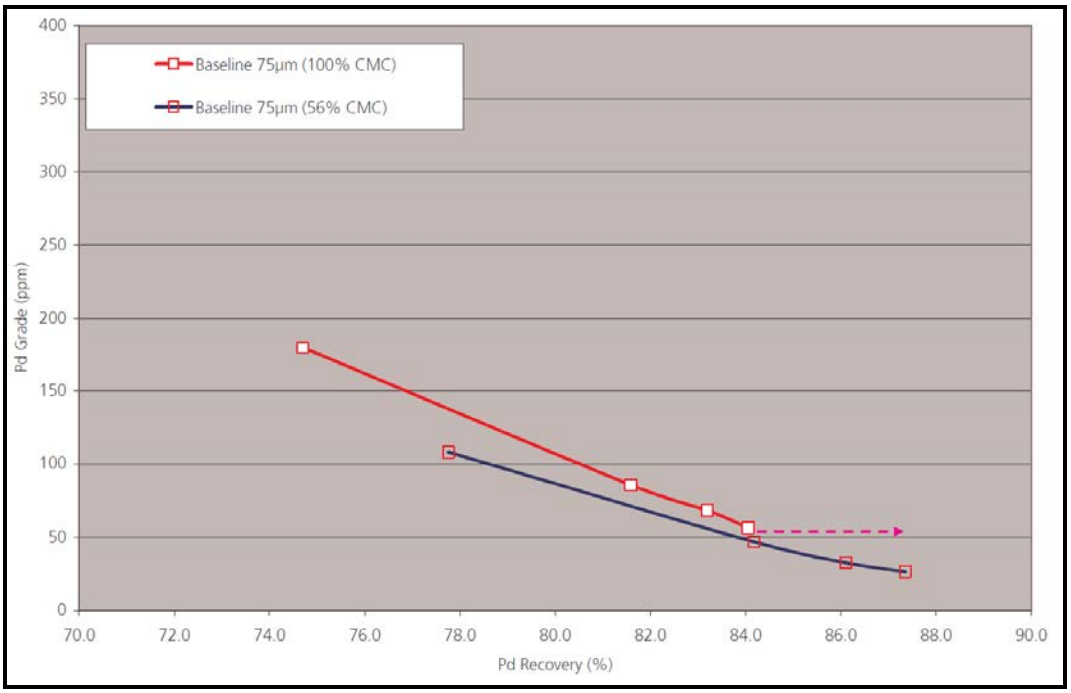
Figure 13.10 Roby Zone – Rougher-Scavenger Palladium Grade/Recovery versus CMC Dosage Primary at Primary Grind of 53 µm



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

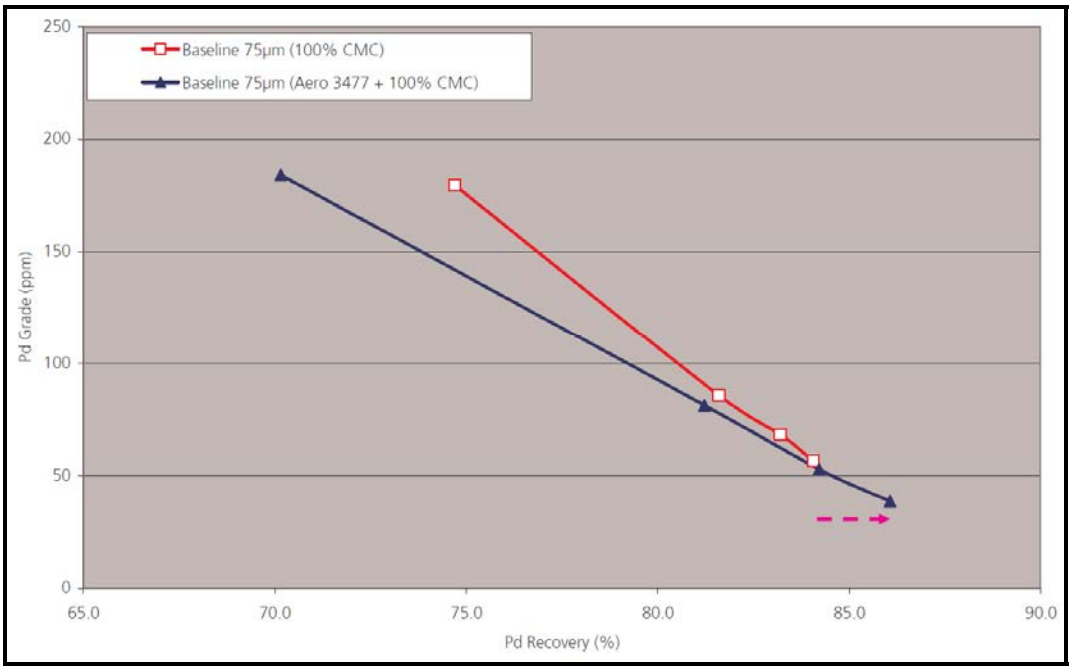


Source: XPS (2010)

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 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



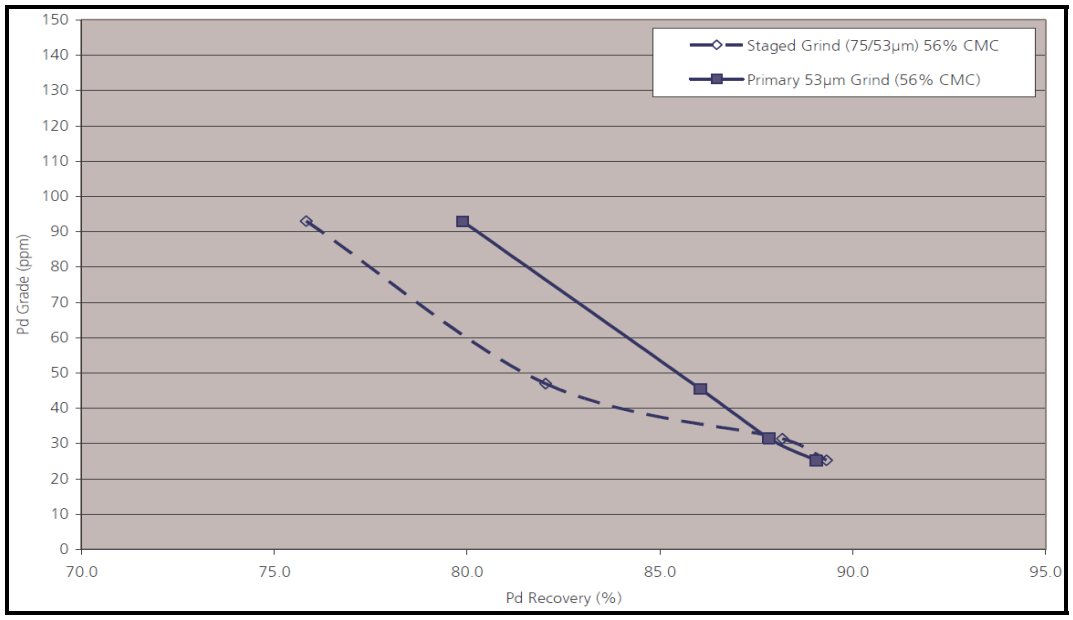
Source: XPS (June 2010)

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.

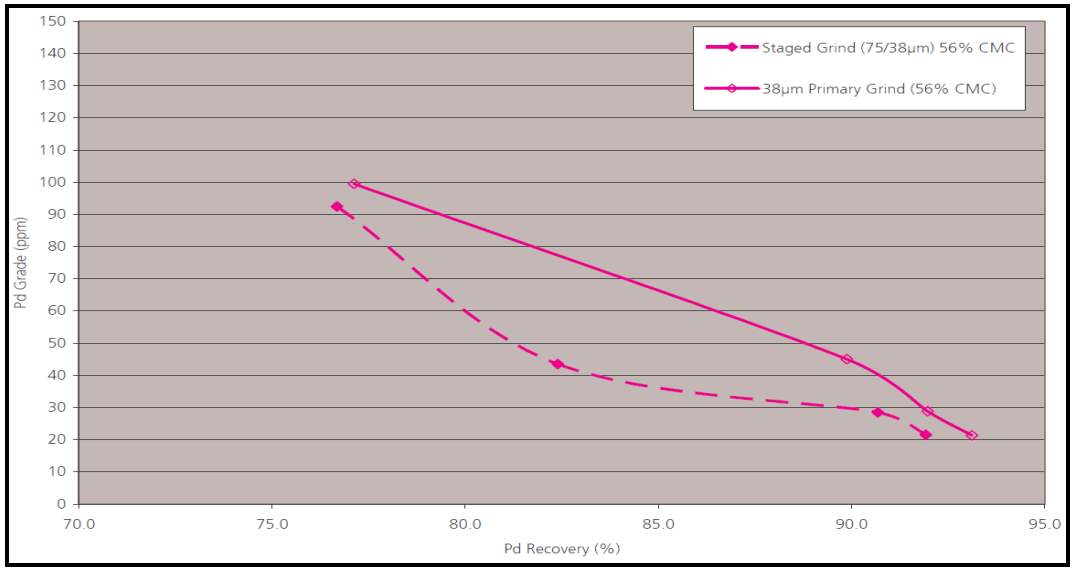
From Figure 13.13 and Figure 13.14, it can be seen that the single stage fine grind resulted in higher palladium recoveries than the two-stage grinding.

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



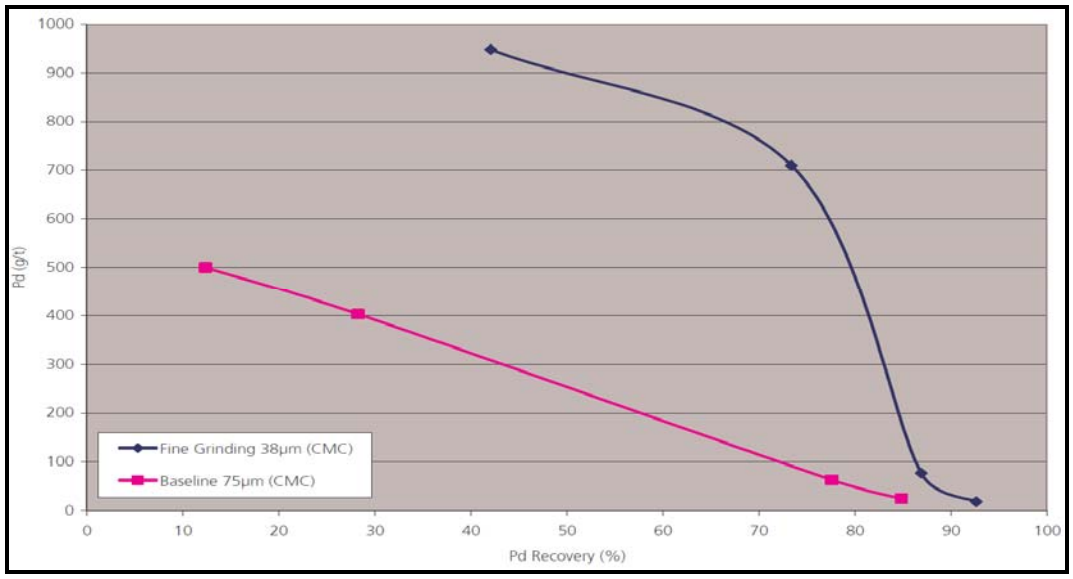
Source: XPS (2010)

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

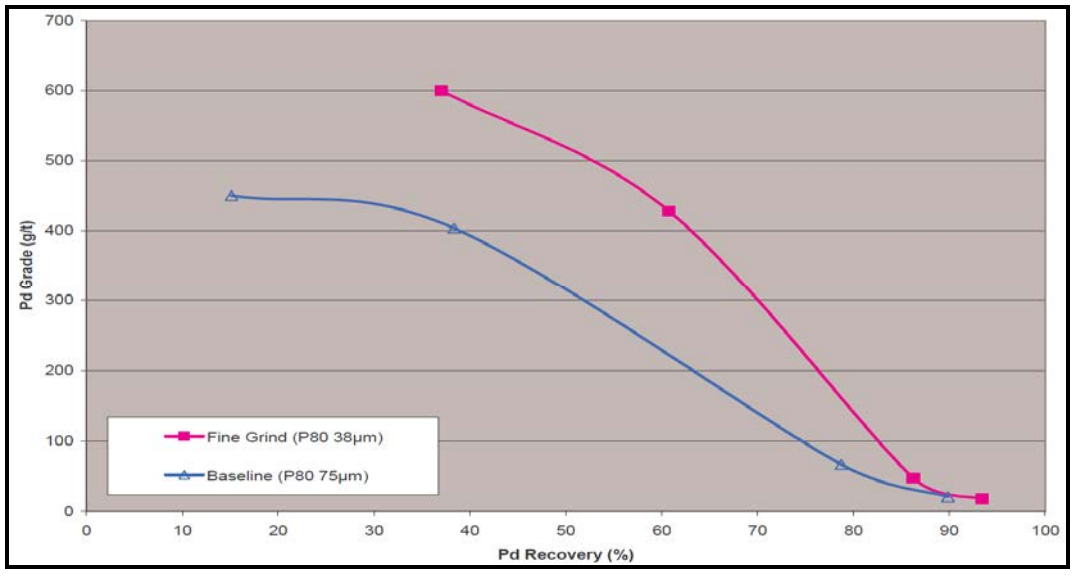
tests at the specified grind sizes for the Roby and Offset samples respectively. In both the Roby and Offset tests, the benefits of the finer grind to the palladium recovery can be seen.

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

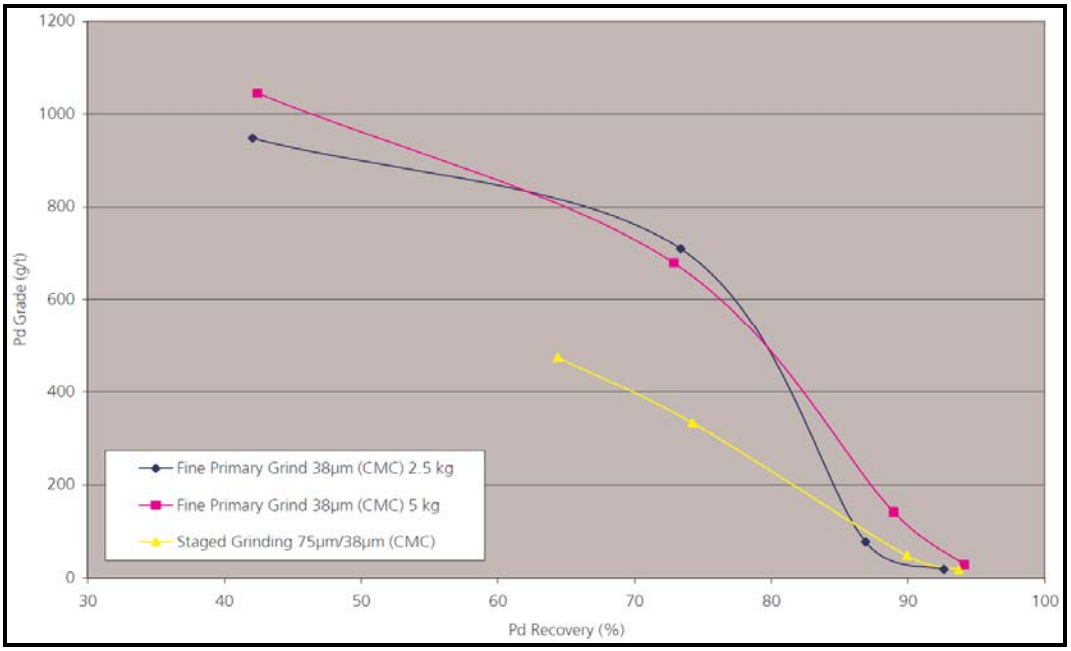


Source: XPS (2010)

OPEN CIRCUIT CLEANER TESTS – FINE GRINDING VERSUS STAGED GRINDING

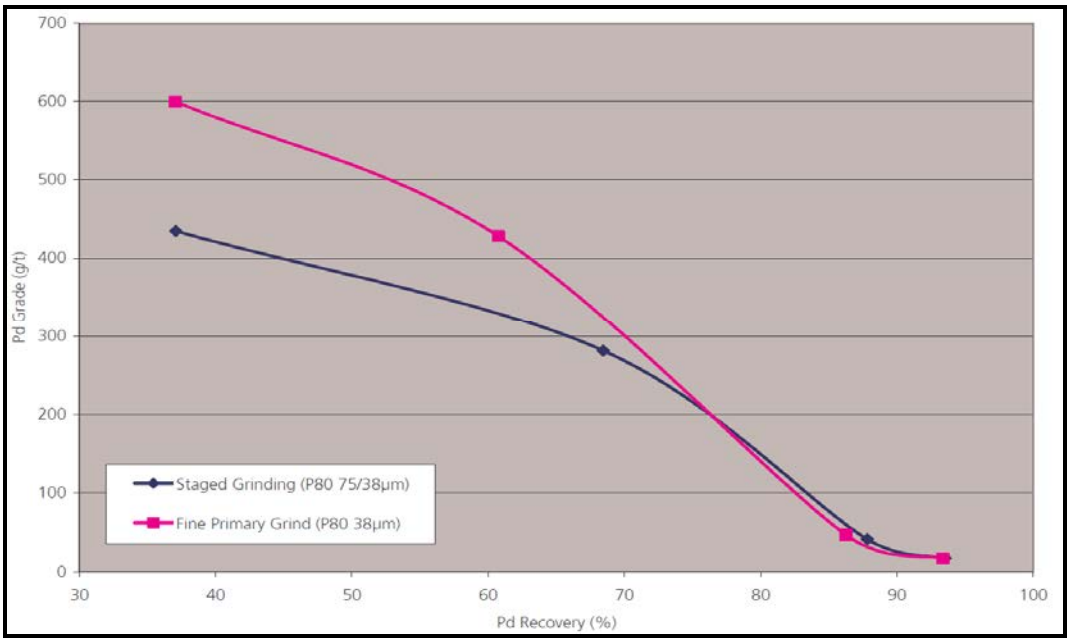
A series of cleaner flotation tests were performed to determine the effect of fine grinding versus staged grinding as performed previously in the rougher-scavenger test work. Figure 13.17 and Figure 13.18 show the cleaner flotation tests for the Roby and Offset samples respectively. The fine grind is 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



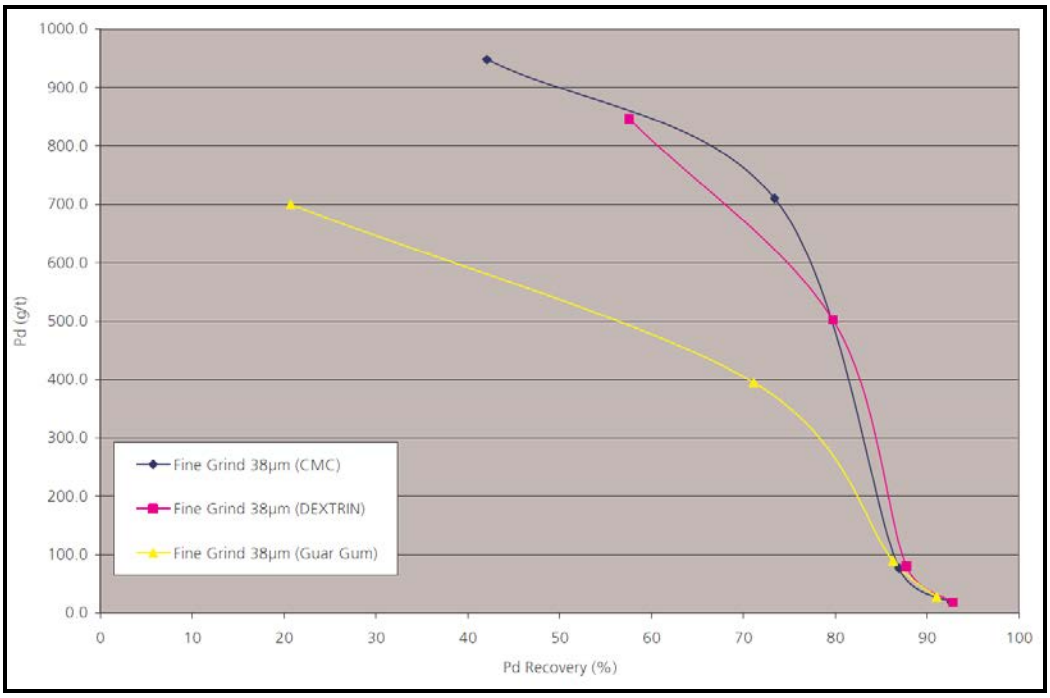
Source: XPS (2010)

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.

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)

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 Ore 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	0.80

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

Source: XPS (2010)

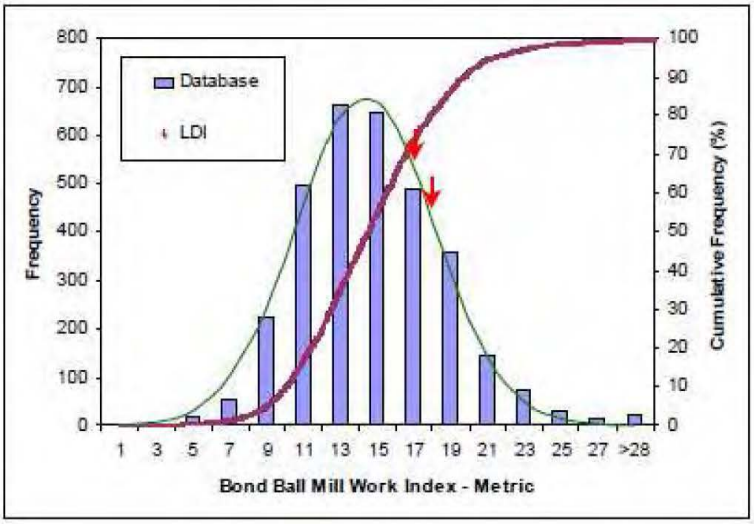
13.2.3 ORE HARDNESS

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.

Figure 13.20 Roby and Offset Bond Ball Mill Work Indices

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



Source: XPS (2010)

13.2.4 GRAVITY RECOVERY TEST WORK

Reject material from the ore 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
P_{80} 840 µm	Stage 1 Concentrate	0.4	183.00	12.90	47.90	11.5	13.1	33.8
P_{80} 206 µm	Stage 2 Concentrate	0.4	185.20	11.70	13.50	11.3	11.6	9.3
P_{80} 54 µm	Stage 3 Concentrate	0.2	152.50	11.20	18.90	6.0	7.2	8.4
Knelson Concentrate	Stage 1-3 Concentrate	1.0	176.50	12.00	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, these poor gravity recoveries are as expected.

13.3 REPRESENTATIVITY

The test samples were taken from the Roby and Offset zones and were deemed representative of the mineralization in these two different deposits by LDIM personnel. The samples were taken from different areas and at different elevations so that they would be representative of the deposit.

13.4 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 has proven its effectiveness and the test work reinforces the use of this strategy.

14.0 MINERAL RESOURCE ESTIMATES

14.1 OFFSET ZONE

14.1.1 INTRODUCTION

Tetra Tech completed a resource estimation of the Offset Zone of the Lac des Iles Mine. The effective date of the resource is March 31, 2012.

14.1.2 DATABASE

NAP maintains all borehole data in a Century Systems Fusion Server®. Header, survey, assays, and lithology tables are saved in the database. A copy of the header, survey, lithology and assays were provided to Tetra Tech on June 4, 2012.

The files provided to Tetra Tech contained the data for 2004 boreholes. The dataset were for all surface and underground holes for the Roby underground and Offset zones. All boreholes that ended above the Offset Fault were removed from the dataset in order to concentrate on the Offset Zone. Four boreholes drilled in the Offset Zone were also removed from the data set as the samples were analyzed at the mine lab and were deemed to be non-compliant with the rest of the data set. The final dataset used for interpretation of the Offset Zone contained 539 boreholes. Table 14.1 summarizes the borehole database.

The resource estimation was conducted using Datamine™ Studio 3 version 3.21.7164.0.

Table 14.1 Borehole Database

Field	No. of Records	No. of Samples	Percent of Data Set (%)	Minimum	Maximum	Mean	Variance	Standard Deviation
Length	208,795	208,795	100	0.0100	57.600	1.328	1.077	1.038
Pdppm	208,795	182,147	87	0.0000	49.800	0.860	3.878	1.969
Ptppm	208,795	182,147	87	0.0000	3.203	0.090	0.022	0.147
Auppm	208,795	182,147	87	0.0000	11.618	0.068	0.025	0.159
Cuper	208,795	179,072	86	0.0001	3.719	0.037	0.004	0.060
Niper	208,795	179,068	86	0.0001	4.175	0.050	0.003	0.053
Coper	208,795	179,072	86	0.0001	0.150	0.006	0.000	0.003
Agppm	208,795	153,118	73	0.1000	232.000	0.388	1.657	1.287
Feper	208,795	146,820	70	0.0050	50.500	5.811	5.370	2.317

14.1.3 SPECIFIC GRAVITY

The Lac des Iles Mine 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 Lac des Iles Mine.

Tetra Tech used a specific gravity of 2.89 for the resource estimate, which is the same number used in previous estimate. A specific gravity of 2.89 is within the accepted range of a gabbro with low-sulphide content.

Tetra Tech recommends that the Lac des Iles Mine continue to collect specific gravity measurements from the various rocks types and grade distributions in order to build up the data set. At a minimum, 2% of the data set should have specific gravity measurements.

14.1.4 GEOLOGICAL INTERPRETATION

Two, 3D wireframe models of mineralization were developed by Tetra Tech in 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 with the second wireframe having a minimum grade of 2.0 g/t palladium. The 2 g/t palladium wireframe is located within the 1 g/t palladium wireframe as a discrete, higher-grade domain.

Sectional interpretations were in Datamine™ software, and these interpretations were linked with tag strings and triangulated to build 3D solids. Table 14.2 tabulates the solids and associated volumes. The solids were validated in the Datamine™ software and no errors were found.

Table 14.2 Wireframe Summary

Zone	Wireframe Dimensions						Volume (m^3)
	Minimum X	Maximum X	Minimum Y	Maximum Y	Minimum Z	Maximum Z	
Offset 1 g/t	31,877.52	32,256.40	31,325.17	32,225.17	(1,271.23)	259.11	53,624,843.8
Offset 2 g/t	31,959.16	32,253.89	31,325.17	32,205.17	(985.85)	(15.93)	8,260,307.9

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 and estimation of material above the fault plane. In addition, the mineral wireframe contacts with the east gabbro and a dyke were treated as a hard boundary to avoid the estimation into these barren units.

The wireframes extend at depth and along strike beyond the last borehole. This is to provide the exploration group with target areas for future exploration. The resource model will not estimate grades into the full volume of the wireframes due to sheer size of the wireframes.

Figure 14.1 is an oblique view of the 1 g/t palladium mineral wireframe while Figure 14.2 is an oblique view of the 2 g/t palladium mineral wireframe.

Figure 14.1 Oblique View (Looking West) of Offset Zone 1 g/t Palladium Mineral Wireframe

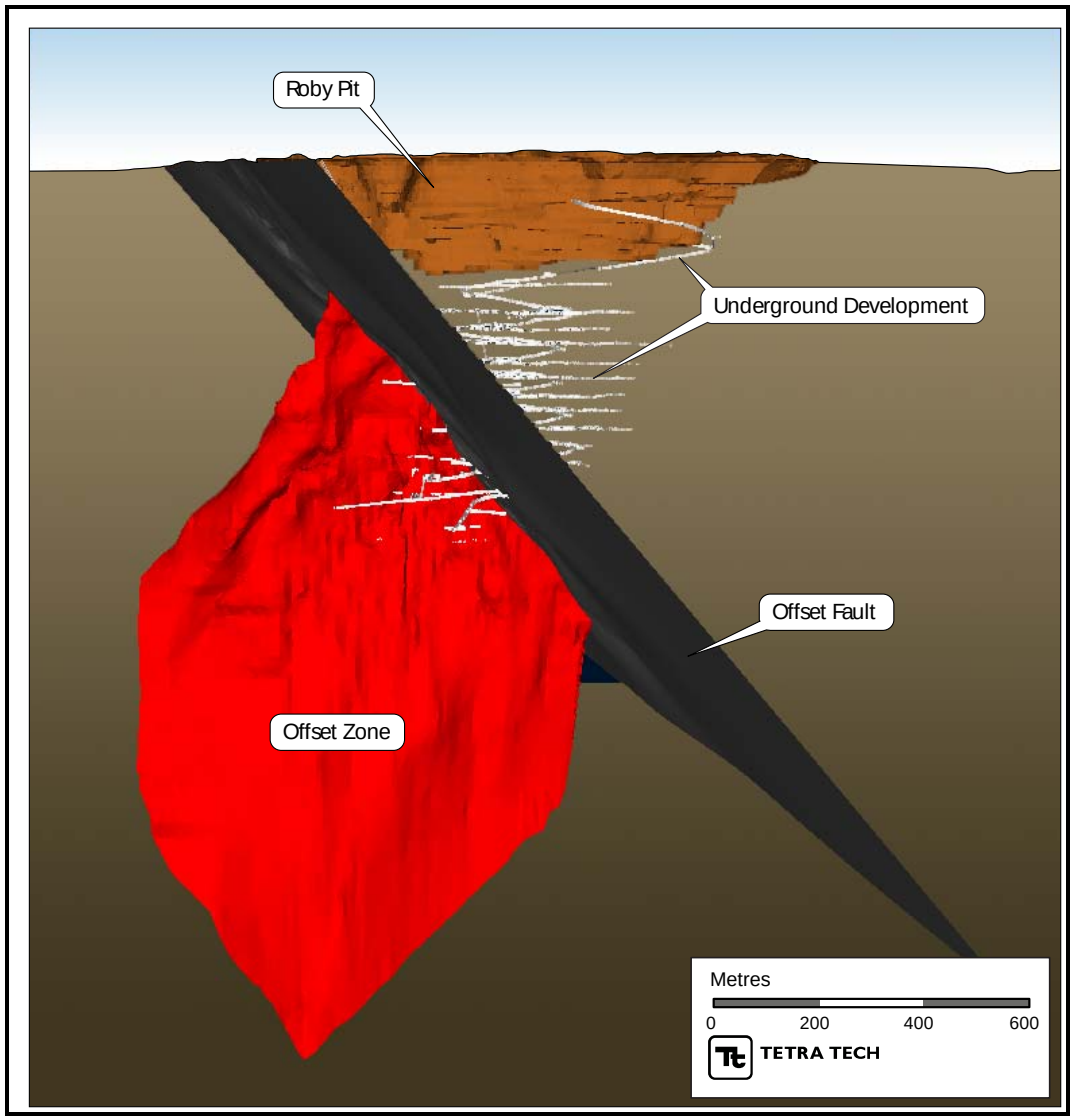
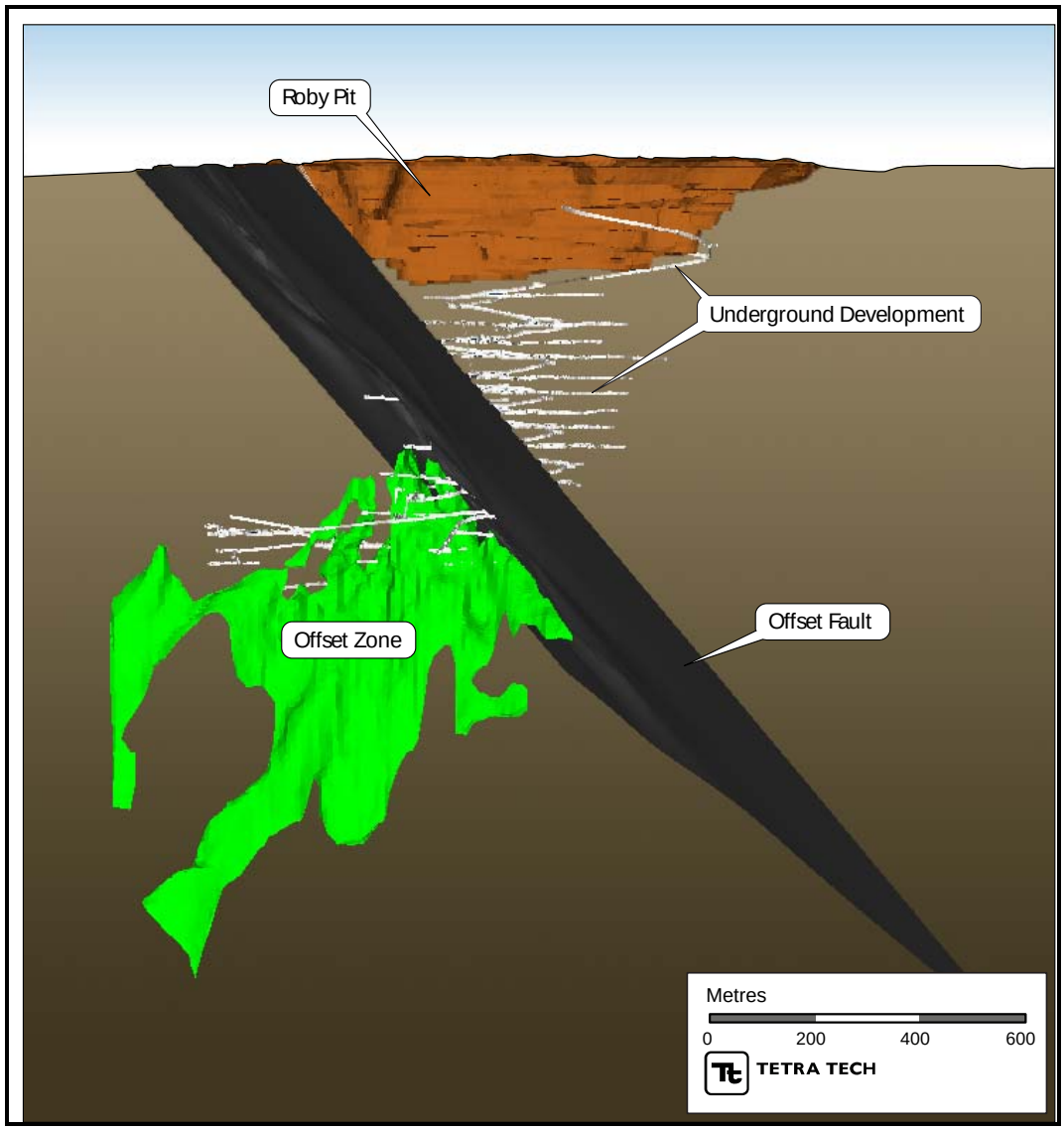


Figure 14.2 Oblique View (Looking West) of Offset Zone 2 g/t Palladium Mineral Wireframe



The final wireframes shapes were validated within Datamine™ to ensure the shapes were closed solids and contained no cross overs. Table 14.2 summarized the dimensions of the two wireframes.

14.1.5 *Exploratory Data Analysis*

Assays

The portion of the deposit included in the mineral resource was sampled by a total of 59,995 palladium assays. The assay intervals within each zone were captured using

a Datamine™ macro into individual borehole files. These borehole files were reviewed to ensure all the proper assay intervals were captured. Table 14.3 summarizes the basic statistics for the assays in the Offset Zone for the 1 and 2 g/t wireframes.

The non-assayed intervals were assigned void (-) value. Tetra Tech believes that non-assayed material should not be assigned a zero value, as this does not reflect the true value of the material.

Table 14.3 Summary of Offset Zone Borehole Statistics

Field	No. of Records	No. of Samples	Percent of Samples (%)	Minimum	Maximum	Mean	Standard Deviation
Offset Zone 1 g/t							
Length	60,385	60,385	100.00	0.0100	7.50	0.96	0.382
Pd (ppm)	60,385	59,995	99.35	0.0010	49.80	1.98	2.912
Pt (ppm)	60,385	59,995	99.35	0.0000	3.18	0.18	0.199
Au (ppm)	60,385	59,995	99.35	0.0000	5.44	0.15	0.230
Cu (%)	60,385	60,281	99.83	0.0001	1.59	0.06	0.067
Ni (%)	60,385	60,278	99.82	0.0001	1.25	0.08	0.062
Co (%)	60,385	60,281	99.83	0.0001	0.06	0.01	0.002
Ag (ppm)	60,385	55,863	92.51	0.1000	100.00	0.50	1.021
Fe (%)	60,385	51,814	85.81	0.0050	31.00	5.34	1.836
S (%)	60,385	51,814	85.81	0.0050	3.74	0.19	0.228
Offset Zone 2 g/t							
Length	16,589	16,589	100.00	0.0100	3.30	0.92	0.295
Pd (ppm)	16,589	16,586	99.98	0.0025	49.80	4.15	4.151
Pt (ppm)	16,589	16,586	99.98	0.0000	2.68	0.31	0.247
Au (ppm)	16,589	16,586	99.98	0.0000	5.15	0.28	0.323
Cu (%)	16,589	16,587	99.99	0.0001	1.59	0.09	0.083
Ni (%)	16,589	16,586	99.98	0.0001	1.25	0.11	0.081
Co (%)	16,589	16,587	99.99	0.0001	0.03	0.01	0.002
Ag (ppm)	16,589	15,109	91.08	0.1000	46.58	0.68	1.033
Fe (%)	16,589	13,338	80.40	0.2500	23.40	5.61	1.751
S (%)	16,589	13,338	80.40	0.0050	3.74	0.26	0.254

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 14.4 summarizes the results of this review. The results indicate that although a couple of elements have a moderate association, the correlations are not strong enough to warrant the use of the same estimation criteria.

Table 14.4 Offset Zone Correlation Ratios

	Au	Pt	Pd	Ni	Cu	Fe	Co	Ag	S
Au	1.0000								
Pt	0.6926	1.0000							
Pd	0.6472	0.8867	1.0000						
Ni	0.7397	0.6981	0.6239	1.0000					
Cu	0.7131	0.6123	0.4958	0.8452	1.0000				
Fe	0.2391	0.2321	0.1639	0.2794	0.3107	1.0000			
Co	0.4714	0.4884	0.3966	0.6606	0.5972	0.8073	1.0000		
Ag	0.2147	0.1965	0.1567	0.2216	0.2973	0.1106	0.1532	1.0000	
S	0.5668	0.5504	0.4253	0.6622	0.8029	0.3861	0.5603	0.3157	1.0000

GRADE CAPPING

Raw assay data for the1 g/t wireframe was examined individually to assess the amount of metal that is at risk from high-grade assays. The Datamine™ Decile function was used to assist in the determination if grade capping was required on each of the elements in the dataset by using the Parrish analysis (Parrish 1997).

When using the Parrish analysis, the following criteria may warrant grade capping:

- the top decile of 90 to 100% contains more than 40% of the metal content, or
- the top decile of 90 to 100% has more than twice the metal content of the next decile at 80 to 90%, or
- the top percentile of 99 to 100% has more than 10% of the metal content, or
- the top percentile of 99 to 100% has more than twice the metal content of the next percentile of 98 to 99%.

Table 14.5 summarizes the results of the Parrish analysis. The results of the analysis indicate that capping of palladium and silver maybe required with the dataset from the 1 g/t mineral wireframe. Further review indicated that the sample requiring capping from the initial 1 g/t dataset are located within the 2 g/t wireframe and the Parrish analysis of the 2 g/t dataset indicates no capping is required for any of the elements.

In addition to the Parrish analysis, the spatial distribution of the samples was reviewed to determine if the population of anomalous samples are in close proximity and may represent a subset within the data. The review of the data resulted in no capping of any elements with the Offset Zone data set.

Table 14.5 Parrish Analysis Summary

Offset Zone 1 g/t Palladium Mineral Wireframe										Offset Zone 2 g/t Palladium Mineral Wireframe									
Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap	Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap
Palladium	Decile	0	10	5,428	0.152	0.100	0.216	0.689	-	Palladium	Decile	0	10	1,665	0.387	0.101	0.687	0.892	-
		10	20	5,373	0.294	0.217	0.379	1.335	-			10	0	1,630	0.978	0.688	1.260	2.258	-
		20	30	5,304	0.482	0.379	0.592	2.185	-			20	30	1,632	1.571	1.260	1.880	3.629	-
		30	40	5,270	0.723	0.592	0.860	3.280	-			30	40	1,644	2.171	1.880	2.461	5.015	-
		40	50	5,282	1.018	0.860	1.184	4.620	-			40	50	1,626	2.808	2.464	3.147	6.487	-
		50	60	5,220	1.389	1.184	1.622	6.302	-			50	60	1,616	3.521	3.148	3.950	8.134	-
		60	70	5,315	1.922	1.622	2.260	8.719	-			60	70	1,595	4.443	3.950	4.990	10.260	-
		70	80	5,407	2.720	2.260	3.271	12.339	-			70	80	1,587	5.712	4.990	6.583	13.192	-
		80	90	5,454	4.131	3.274	5.304	18.737	-			80	90	1,582	7.754	6.586	9.267	17.913	-
		90	100	5,424	9.209	5.310	49.800	41.795	Yes			90	100	1,557	13.947	9.270	49.800	32.219	No
	Percentile	90	91	543	5.488	5.310	5.670	2.488	-		Percentile	90	91	154	9.456	9.270	9.640	2.183	-
		91	92	542	5.851	5.670	6.060	2.653	-			91	92	159	9.926	9.640	10.219	2.289	-
		92	93	549	6.298	6.060	6.560	2.861	-			92	93	159	10.543	10.234	10.900	2.428	-
		93	94	540	6.825	6.560	7.120	3.093	-			93	94	148	11.139	10.900	11.400	2.579	-
		94	95	558	7.434	7.120	7.740	3.375	-			94	95	154	11.807	11.400	12.200	2.735	-
		95	96	536	8.118	7.740	8.530	3.686	-			95	96	150	12.708	12.200	13.249	2.926	-
		96	97	543	9.047	8.530	9.610	4.109	-			96	97	160	13.997	13.300	14.800	3.230	-
		97	98	536	10.469	9.620	11.400	4.746	-			97	98	159	15.784	14.800	16.900	3.663	-
		98	99	531	12.720	11.400	14.600	5.770	-			98	99	158	18.268	16.900	20.000	4.203	-
		99	100	546	19.824	14.600	49.800	9.013	No			99	100	156	25.780	20.000	49.800	5.984	No

table continues…

Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap	Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap
Platinum	Decile	0	10	3,342	0.109	0.100	0.119	3.777	-	Platinum	Decile	0	10	1,390	0.120	0.100	0.140	3.282	-
		10	20	3,353	0.129	0.119	0.139	4.478	-			10	20	1,406	0.160	0.140	0.179	4.370	-
		20	30	3,325	0.151	0.139	0.163	5.256	-			20	30	1,367	0.198	0.179	0.217	5.415	-
		30	40	3,341	0.177	0.163	0.190	6.140	-			30	40	1,363	0.237	0.217	0.257	6.473	-
		40	50	3,360	0.206	0.190	0.223	7.154	-			40	50	1,356	0.279	0.257	0.301	7.607	-
		50	60	3,319	0.242	0.223	0.263	8.404	-			50	60	1,375	0.328	0.302	0.357	8.963	-
		60	70	3,388	0.288	0.263	0.317	10.011	-			60	70	1,332	0.390	0.357	0.424	10.650	-
		70	80	3,384	0.354	0.317	0.398	12.312	-			70	80	1,333	0.465	0.424	0.514	12.704	-
		80	90	3,406	0.459	0.398	0.537	15.948	-			80	90	1,325	0.581	0.514	0.664	15.849	-
		90	100	3,435	0.763	0.538	3.180	26.520	No			90	100	1,339	0.903	0.664	2.680	24.688	No
	Percentile	90	91	349	0.549	0.538	0.560	1.908	-		Percentile	90	91	131	0.675	0.664	0.687	1.832	-
		91	92	346	0.571	0.560	0.583	1.983	-			91	92	134	0.701	0.687	0.717	1.927	-
		92	93	341	0.597	0.583	0.610	2.078	-			92	93	136	0.729	0.717	0.744	1.997	-
		93	94	340	0.626	0.611	0.643	2.175	-			93	94	130	0.761	0.744	0.778	2.079	-
		94	95	338	0.661	0.643	0.680	2.296	-			94	95	139	0.801	0.779	0.823	2.186	-
		95	96	346	0.704	0.680	0.730	2.443	-			95	96	132	0.850	0.823	0.879	2.313	-
		96	97	334	0.759	0.730	0.791	2.645	-			96	97	139	0.913	0.879	0.947	2.499	-
		97	98	352	0.838	0.791	0.889	2.903	-			97	98	134	0.999	0.947	1.050	2.739	-
		98	99	342	0.968	0.890	1.060	3.379	-			98	99	130	1.112	1.050	1.180	3.035	-
		99	100	347	1.354	1.060	3.180	4.710	No			99	100	134	1.491	1.180	2.680	4.080	No

table continues...

Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap	Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap
Nickel	Decile	0	10	5,531	0.020	0.010	0.028	2.584	-	Nickel	Decile	0	10	1,679	0.032	0.010	0.044	2.725	-
		10	20	5,795	0.032	0.028	0.037	4.105	-			10	20	1,649	0.050	0.044	0.057	4.314	-
		20	30	5,779	0.040	0.037	0.044	5.127	-			20	30	1,633	0.063	0.057	0.070	5.388	-
		30	40	5,802	0.048	0.044	0.052	6.083	-			30	40	1,615	0.076	0.070	0.082	6.471	-
		40	50	5,786	0.056	0.052	0.061	7.089	-			40	50	1,633	0.089	0.082	0.097	7.656	-
		50	60	5,797	0.066	0.061	0.072	8.377	-			50	60	1,599	0.105	0.097	0.113	8.967	-
		60	70	5,824	0.079	0.072	0.088	10.065	-			60	70	1,611	0.123	0.113	0.134	10.503	-
		70	80	5,834	0.098	0.088	0.111	12.465	-			70	80	1,617	0.147	0.134	0.162	12.544	-
		80	90	5,905	0.128	0.111	0.151	16.208	-			80	90	1,601	0.185	0.162	0.215	15.829	-
		90	100	5,919	0.220	0.151	1.253	27.897	No			90	100	1,587	0.299	0.215	1.253	25.603	No
	Percentile	90	91	599	0.154	0.151	0.158	1.948	-		Percentile	90	91	156	0.219	0.215	0.224	1.875	-
		91	92	610	0.161	0.158	0.165	2.041	-			91	92	161	0.228	0.224	0.232	1.947	-
		92	93	608	0.169	0.165	0.173	2.141	-			92	93	164	0.238	0.232	0.244	2.038	-
		93	94	588	0.178	0.173	0.183	2.259	-			93	94	163	0.251	0.244	0.258	2.142	-
		94	95	576	0.189	0.183	0.195	2.395	-			94	95	158	0.265	0.258	0.273	2.270	-
		95	96	596	0.203	0.195	0.211	2.565	-			95	96	161	0.281	0.274	0.289	2.416	-
		96	97	570	0.221	0.211	0.231	2.797	-			96	97	158	0.300	0.289	0.314	2.565	-
		97	98	603	0.245	0.231	0.262	3.100	-			97	98	155	0.329	0.314	0.346	2.815	-
		98	99	595	0.284	0.262	0.313	3.606	-			98	99	155	0.373	0.346	0.408	3.187	-
		99	100	574	0.398	0.314	1.253	5.045	No			99	100	156	0.507	0.409	1.253	4.347	No

table continues…

Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap	Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap
Copper	Decile	0	10	4,738	0.012	0.010	0.014	1.711	-	Copper	Decile	0	10	1,489	0.015	0.010	0.020	1.501	-
		10	20	4,698	0.017	0.014	0.020	2.413	-			10	20	1,522	0.025	0.020	0.031	2.603	-
		20	30	4,737	0.024	0.020	0.027	3.339	-			20	30	1,485	0.038	0.031	0.045	3.899	-
		30	40	4,659	0.032	0.027	0.037	4.502	-			30	40	1,492	0.052	0.045	0.059	5.331	-
		40	50	4,703	0.042	0.037	0.048	5.991	-			40	50	1,488	0.067	0.059	0.075	6.865	-
		50	60	4,633	0.055	0.048	0.062	7.792	-			50	60	1,483	0.084	0.075	0.093	8.650	-
		60	70	4,741	0.071	0.062	0.081	10.072	-			60	70	1,475	0.104	0.093	0.115	10.694	-
		70	80	4,685	0.093	0.081	0.108	13.235	-			70	80	1,482	0.130	0.115	0.147	13.401	-
		80	90	4,789	0.128	0.108	0.154	18.141	-			80	90	1,455	0.171	0.147	0.201	17.593	-
		90	100	4,780	0.231	0.154	1.590	32.804	No			90	100	1,446	0.286	0.201	1.590	29.463	No
	Percentile	90	91	472	0.158	0.154	0.162	2.233	-		Percentile	90	91	153	0.205	0.201	0.210	2.111	-
		91	92	483	0.166	0.162	0.171	2.353	-			91	92	143	0.214	0.210	0.220	2.196	-
		92	93	476	0.175	0.171	0.180	2.484	-			92	93	153	0.226	0.220	0.231	2.333	-
		93	94	474	0.186	0.181	0.192	2.631	-			93	94	146	0.237	0.231	0.243	2.439	-
		94	95	486	0.199	0.192	0.206	2.818	-			94	95	142	0.251	0.243	0.260	2.594	-
		95	96	482	0.214	0.206	0.224	3.042	-			95	96	144	0.268	0.260	0.277	2.764	-
		96	97	481	0.234	0.224	0.245	3.318	-			96	97	144	0.289	0.277	0.302	2.997	-
		97	98	483	0.260	0.245	0.277	3.687	-			97	98	141	0.319	0.302	0.338	3.289	-
		98	99	475	0.302	0.277	0.336	4.279	-			98	99	139	0.359	0.338	0.393	3.711	-
		99	100	468	0.420	0.336	1.590	5.958	No			99	100	141	0.487	0.393	1.590	5.029	No

table continues…

Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap	Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap
Gold	Decile	0	10	2,342	0.108	0.100	0.117	3.372	-	Gold	Decile	0	10	1,145	0.114	0.100	0.129	2.852	-
		10	20	2,313	0.126	0.117	0.137	3.956	-			10	20	1,140	0.145	0.129	0.161	3.631	-
		20	30	2,332	0.148	0.137	0.160	4.632	-			20	30	1,124	0.182	0.161	0.201	4.552	-
		30	40	2,358	0.175	0.160	0.190	5.466	-			30	40	1,142	0.223	0.202	0.245	5.586	-
		40	50	2,381	0.208	0.191	0.229	6.514	-			40	50	1,113	0.269	0.245	0.295	6.748	-
		50	60	2,408	0.250	0.229	0.274	7.830	-			50	60	1,082	0.325	0.295	0.357	8.147	-
		60	70	2,364	0.304	0.274	0.338	9.518	-			60	70	1,100	0.397	0.358	0.440	9.961	-
		70	80	2,352	0.383	0.338	0.434	11.969	-			70	80	1,116	0.495	0.440	0.559	12.414	-
		80	90	2,415	0.516	0.434	0.622	16.136	-			80	90	1,096	0.655	0.560	0.777	16.439	-
		90	100	2,373	0.979	0.622	5.440	30.607	No			90	100	1,069	1.183	0.777	5.150	29.671	No
	Percentile	90	91	241	0.637	0.622	0.651	1.991	-		Percentile	90	91	104	0.795	0.777	0.814	1.991	-
		91	92	239	0.670	0.651	0.689	2.092	-			91	92	106	0.835	0.814	0.855	2.076	-
		92	93	244	0.709	0.689	0.730	2.218	-			92	93	109	0.881	0.856	0.903	2.225	-
		93	94	245	0.753	0.730	0.776	2.362	-			93	94	105	0.924	0.903	0.952	2.327	-
		94	95	231	0.806	0.776	0.838	2.513	-			94	95	104	0.989	0.952	1.027	2.417	-
		95	96	235	0.874	0.839	0.910	2.739	-			95	96	108	1.067	1.030	1.110	2.738	-
		96	97	230	0.959	0.910	1.020	2.992	-			96	97	108	1.154	1.110	1.220	2.884	-
		97	98	230	1.082	1.020	1.160	3.398	-			97	98	109	1.303	1.220	1.390	3.266	-
		98	99	235	1.284	1.160	1.440	4.000	-			98	99	107	1.537	1.390	1.728	3.857	-
		99	100	243	2.007	1.440	5.440	6.302	No			99	100	109	2.329	1.730	5.150	5.889	No

table continues…

Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap	Element	Field	Q% From	Q% To	No. of Samples	Mean	Minimum	Maximum	Metal (%)	Parrish Cap
Silver	Decile	0	10	5,897	0.149	0.100	0.150	2.952	-	Silver	Decile	0	10	1,676	0.149	0.100	0.150	2.136	-
		10	20	5,691	0.150	0.150	0.150	2.969	-			10	20	1,477	0.150	0.150	0.150	2.148	-
		20	30	5,586	0.150	0.150	0.150	2.969	-			20	30	1,469	0.150	0.150	0.150	2.149	-
		30	40	5,552	0.150	0.150	0.150	2.969	-			30	40	1,309	0.183	0.150	0.300	2.616	-
		40	50	5,551	0.150	0.150	0.150	2.969	-			40	50	1,612	0.345	0.300	0.400	4.939	-
		50	60	4,724	0.244	0.150	0.300	4.829	-			50	60	1,541	0.441	0.400	0.500	6.314	-
		60	70	5,848	0.379	0.300	0.400	7.510	-			60	70	1,567	0.549	0.500	0.600	7.858	-
		70	80	5,834	0.508	0.400	0.600	10.047	-			70	80	1,556	0.740	0.600	0.900	10.602	-
		80	90	5,781	0.777	0.600	1.100	15.379	-			80	90	1,491	1.278	0.900	1.877	18.314	-
		90	100	5,399	2.395	1.100	100.000	47.408	Yes			90	100	1,411	2.996	1.881	46.579	42.924	Yes
	Percentile	90	91	571	1.162	1.100	1.200	2.299	-		Percentile	90	91	140	1.943	1.881	2.016	2.781	-
		91	92	560	1.323	1.200	1.400	2.617	-			91	92	141	2.096	2.016	2.197	2.999	-
		92	93	545	1.506	1.400	1.600	2.983	-			92	93	141	2.277	2.198	2.371	3.250	-
		93	94	538	1.722	1.600	1.838	3.407	-			93	94	141	2.448	2.371	2.525	3.506	-
		94	95	537	1.974	1.838	2.100	3.912	-			94	95	141	2.624	2.525	2.731	3.777	-
		95	96	527	2.242	2.100	2.375	4.435	-			95	96	140	2.822	2.731	2.900	4.042	-
		96	97	523	2.491	2.376	2.626	4.930	-			96	97	142	3.011	2.904	3.115	4.301	-
		97	98	529	2.790	2.626	2.952	5.523	-			97	98	141	3.229	3.115	3.360	4.634	-
		98	99	536	3.177	2.957	3.432	6.284	-			98	99	140	3.554	3.370	3.835	5.110	-
		99	100	533	5.557	3.432	100.000	11.017	No			99	100	144	5.947	3.845	46.579	8.523	No

COMPOSITING

Compositing of all assay data within the 1 and 2 g/t wireframes was completed at 1, 2, and 3 m intervals. The downhole intervals honoured the interpretation of the geological solids. The backstitching process was used in the compositing routine to ensure all captured sample material was included. The backstitching routine adjusts the composite lengths for each individual borehole in order to compensate for the last sample interval.

The 1 m composites were selected as the optimal composite length to use in the estimation. Table 14.6 summarizes the statistics for the boreholes after compositing.

Table 14.6 Drillhole Compositing Statistics

Offset 1 g/t									Offset 2 g/t								
Field	No. of Records	No. of Samples	Percent of Samples (%)	Minimum	Maximum	Mean	Variance	Standard Deviation	Field	No. of Records	No. of Samples	Percent of Samples (%)	Minimum	Maximum	Mean	Variance	Standard Deviation
1 m Composite									**1 m Composite**								
Length	57,800	57,800	100	0.5000	1.07	1.00	0.000	0.003	Length	15,239	15,239	100	0.8000	1.02	1.00	0.000	0.004
Pd (ppm)	57,800	57,311	99	0.0010	49.80	1.97	8.173	2.859	Pd (ppm)	15,239	15,237	100	0.0025	49.80	4.21	17.074	4.132
Pt (ppm)	57,800	57,311	99	0.0000	3.18	0.18	0.038	0.195	Pt (ppm)	15,239	15,237	100	0.0000	2.68	0.31	0.060	0.245
Au (ppm)	57,800	57,311	99	0.0000	5.15	0.15	0.051	0.226	Au (ppm)	15,239	15,237	100	0.0000	5.15	0.28	0.105	0.324
Cu (%)	57,800	57,583	100	0.0001	1.59	0.06	0.004	0.066	Cu (%)	15,239	15,237	100	0.0001	1.59	0.09	0.007	0.083
Ni (%)	57,800	57,583	100	0.0001	1.25	0.08	0.004	0.061	Ni (%)	15,239	15,237	100	0.0001	1.25	0.11	0.006	0.080
Co (%)	57,800	57,583	100	0.0001	0.06	0.01	0.000	0.002	Co (%)	15,239	15,237	100	0.0001	0.03	0.01	0.000	0.002
Ag (ppm)	57,800	51,455	89	0.1000	100.00	0.51	1.094	1.046	Ag (ppm)	15,239	11,965	79	0.2500	23.40	5.59	3.109	1.763
Fe (%)	57,800	47,496	82	0.0050	31.00	5.30	3.403	1.845	Fe (%)	15,239	13,695	90	0.1000	46.58	0.70	1.127	1.062
S (%)	57,800	47,496	82	0.0050	3.74	0.18	0.050	0.224	S (%)	15,239	11,965	79	0.0050	3.74	0.25	0.064	0.253
2 m Composite									**2 m Composite**								
Length	29,036	29,036	100	1.0000	2.75	1.99	0.001	0.024	Length	7,708	7,708	100	1.0000	2.12	1.98	0.002	0.046
Pd (ppm)	29,036	28,797	99	0.0010	35.50	1.97	6.629	2.575	Pd (ppm)	7,708	7,708	100	0.0025	38.70	4.21	13.184	3.631
Pt (ppm)	29,036	28,797	99	0.0003	3.11	0.18	0.030	0.173	Pt (ppm)	7,708	7,708	100	0.0005	2.58	0.31	0.045	0.213
Au (ppm)	29,036	28,797	99	0.0005	2.86	0.15	0.040	0.200	Au (ppm)	7,708	7,708	100	0.0000	3.01	0.28	0.078	0.279
Cu (%)	29,036	28,933	100	0.0001	1.23	0.06	0.003	0.059	Cu (%)	7,708	7,708	100	0.0002	1.34	0.09	0.005	0.074
Ni (%)	29,036	28,933	100	0.0003	0.69	0.08	0.003	0.056	Ni (%)	7,708	7,708	100	0.0007	0.70	0.11	0.005	0.072
Co (%)	29,036	28,933	100	0.0001	0.03	0.01	0.000	0.002	Co (%)	7,708	7,708	100	0.0002	0.02	0.01	0.000	0.002
Ag (ppm)	29,036	25,872	89	0.1000	50.62	0.51	0.749	0.865	Ag (ppm)	7,708	6,068	79	0.2700	14.85	5.59	2.547	1.596
Fe (%)	29,036	23,892	82	0.2700	19.91	5.30	2.940	1.715	Fe (%)	7,708	6,939	90	0.1000	24.51	0.70	0.877	0.936
S (%)	29,036	23,892	82	0.0050	3.22	0.18	0.037	0.193	S (%)	7,708	6,068	79	0.0050	1.99	0.25	0.046	0.215
3 m Composite									**3 m Composite**								
Length	19,269	19,269	100	2.0000	4.00	3.00	0.002	0.042	Length	5,074	5,074	100	2.0000	4.00	3.00	0.009	0.095
Pd (ppm)	19,269	19,114	99	0.0010	27.81	1.97	5.936	2.436	Pd (ppm)	5,074	5,074	100	0.0025	35.00	4.21	11.431	3.381
Pt (ppm)	19,269	19,114	99	0.0003	2.66	0.18	0.027	0.163	Pt (ppm)	5,074	5,074	100	0.0005	1.55	0.31	0.038	0.196
Au (ppm)	19,269	19,114	99	0.0005	2.59	0.15	0.035	0.188	Au (ppm)	5,074	5,074	100	0.0002	2.58	0.28	0.068	0.260
Cu (%)	19,269	19,203	100	0.0001	0.98	0.06	0.003	0.056	Cu (%)	5,074	5,074	100	0.0014	1.13	0.09	0.005	0.069
Ni (%)	19,269	19,203	100	0.0008	0.56	0.08	0.003	0.053	Ni (%)	5,074	5,074	100	0.0009	0.58	0.11	0.005	0.068
Co (%)	19,269	19,203	100	0.0001	0.02	0.01	0.000	0.002	Co (%)	5,074	5,074	100	0.0003	0.02	0.01	0.000	0.002
Ag (ppm)	19,269	17,171	89	0.1000	33.37	0.51	0.618	0.786	Ag (ppm)	5,074	3,988	79	0.3725	12.52	5.59	2.311	1.520
Fe (%)	19,269	15,849	82	0.3357	15.09	5.30	2.732	1.653	Fe (%)	5,074	4,566	90	0.1000	17.42	0.70	0.797	0.893
S (%)	19,269	15,849	82	0.0050	3.26	0.18	0.032	0.180	S (%)	5,074	3,988	79	0.0050	1.76	0.25	0.040	0.200

14.1.6 SPATIAL ANALYSIS

Variography, using Datamine™ software, was completed for each element globally 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.

Table 14.7 summarizes results of the variography, while Figure 14.3 to Figure 14.11 depicts the correlograms for each of the elements being estimated.

Table 14.7 Variogram Parameters

VDESC	VREFNUM	VANGLE1	VANGLE2	VANGLE3	VAXIS1	VAXIS2	VAXIS3	NUGGET	ST1	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4
pt	1	30	-60	90	3	2	1	0.04500	1	20	20	5	0.02100	1	60	50	40	0.04500
au	2	30	-60	90	3	2	1	0.00115	1	7	21	11	0.00069	1	90	100	70	0.00231
pd	3	30	-60	90	3	2	1	0.35000	1	23	22	17	0.37000	1	70	110	60	0.45500
ni	4	30	-60	90	3	2	1	0.00005	1	27	12	5	0.00006	1	100	110	57	0.00010
cu	5	210	-30	90	3	2	1	0.00140	1	10	9	10	0.00100	1	80	100	60	0.00300
co	6	-60	0	120	3	2	1	150.00000	1	14	8	20	82.83900	1	60	70	60	90.54700
rh	7	210	-30	90	3	2	1	325.00000	1	13	13	5	77.99300	1	30	45	20	642.48000
mg	8	210	-30	90	3	2	1	0.18000	1	12	11	4	0.16400	1	30	40	12	0.30700
s	9	210	-30	90	3	2	1	0.01600	1	10	7	2	0.00800	1	20	40	31	0.01900
fe	10	0	0	0	0	0	0	50.00000	1	3	3	3	317.66600	1	20	20	20	674.58800
ag	11	0	0	0	0	0	0	0.42000	1	10	10	10	0.28400	0	0	0	0	0.00000

Figure 14.3 Palladium Variogram

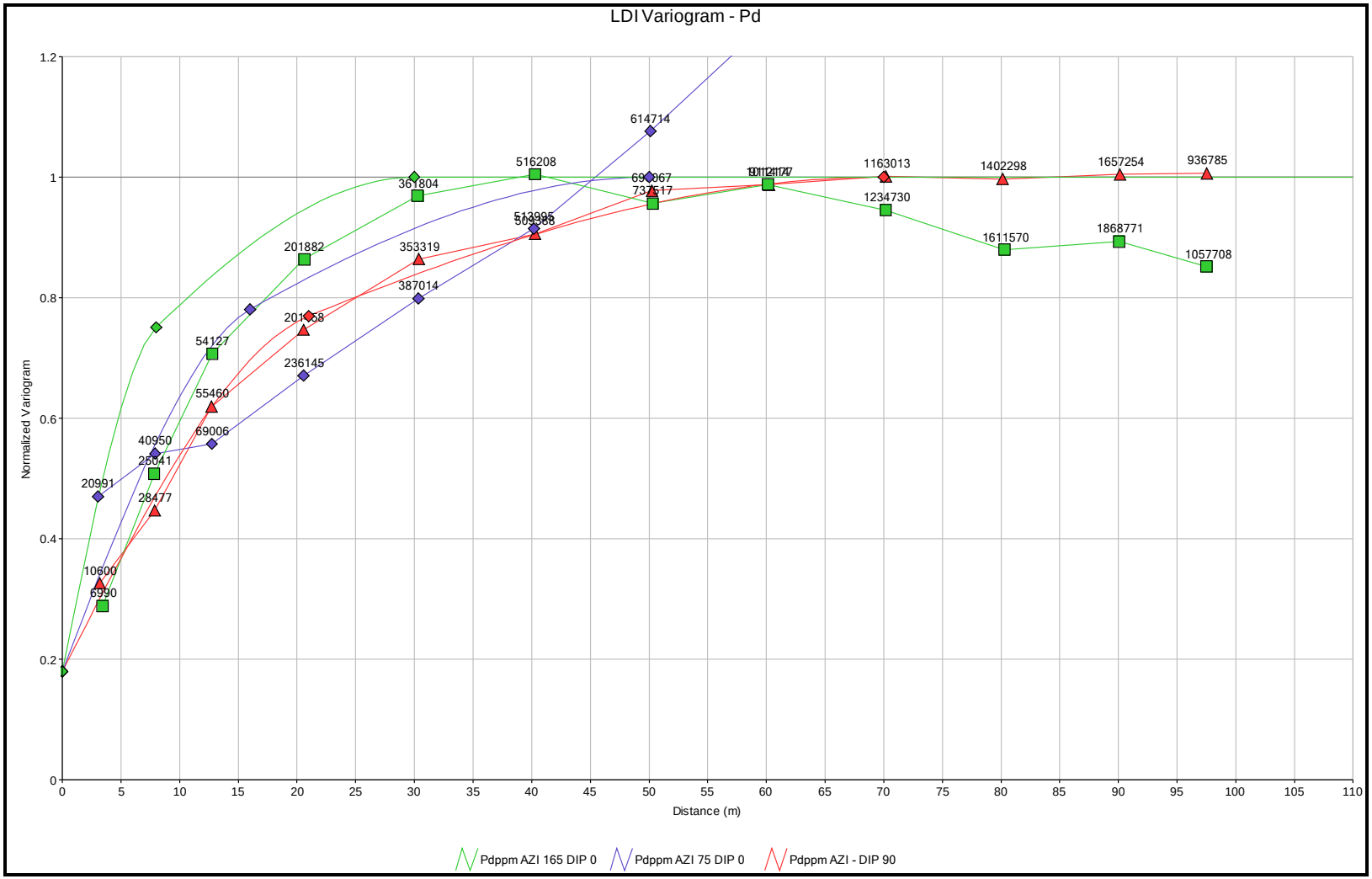


Figure 14.4 Platinum Variogram

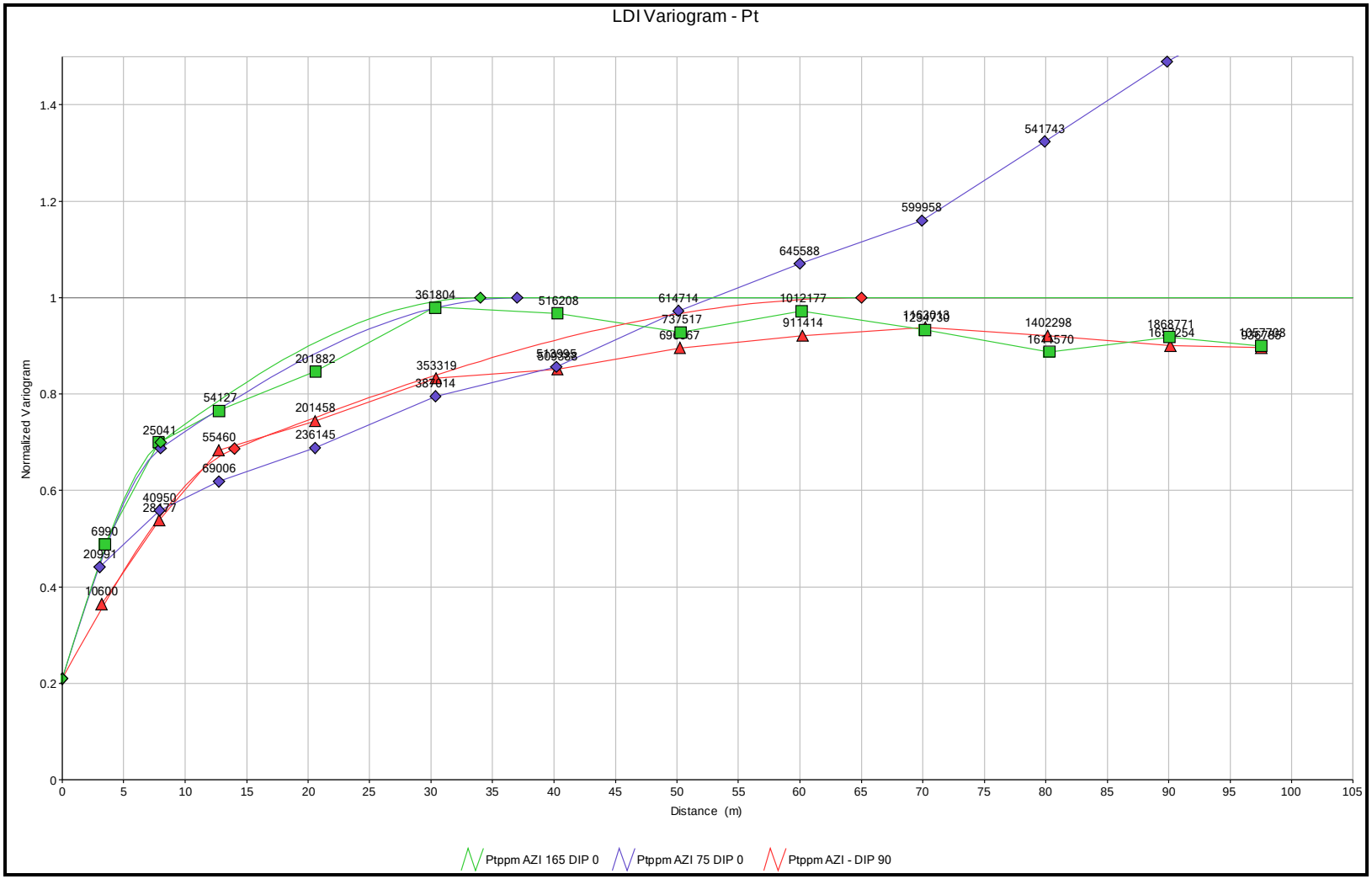


Figure 14.5 Nickel Variogram

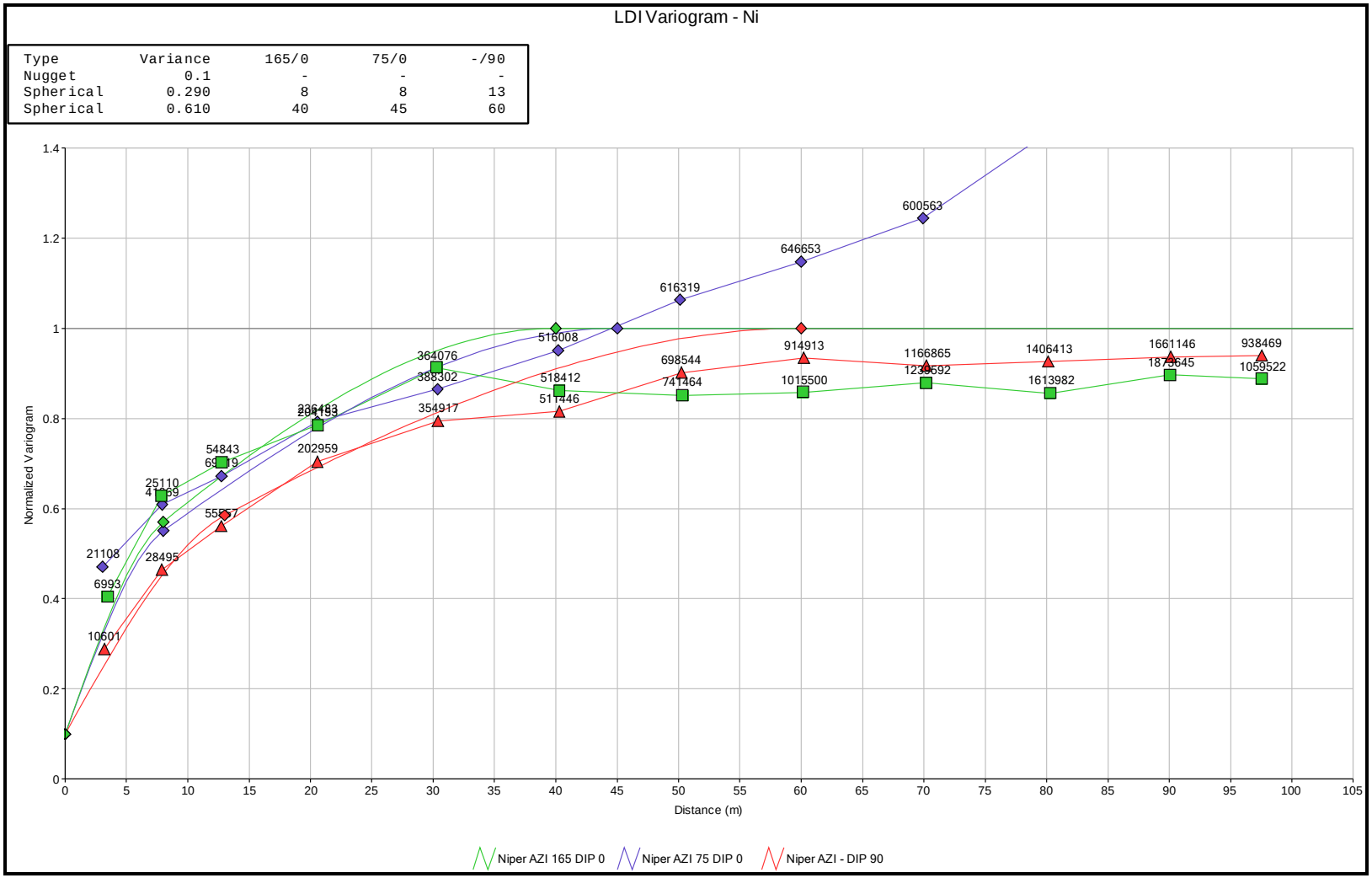


Figure 14.6 Copper Variogram

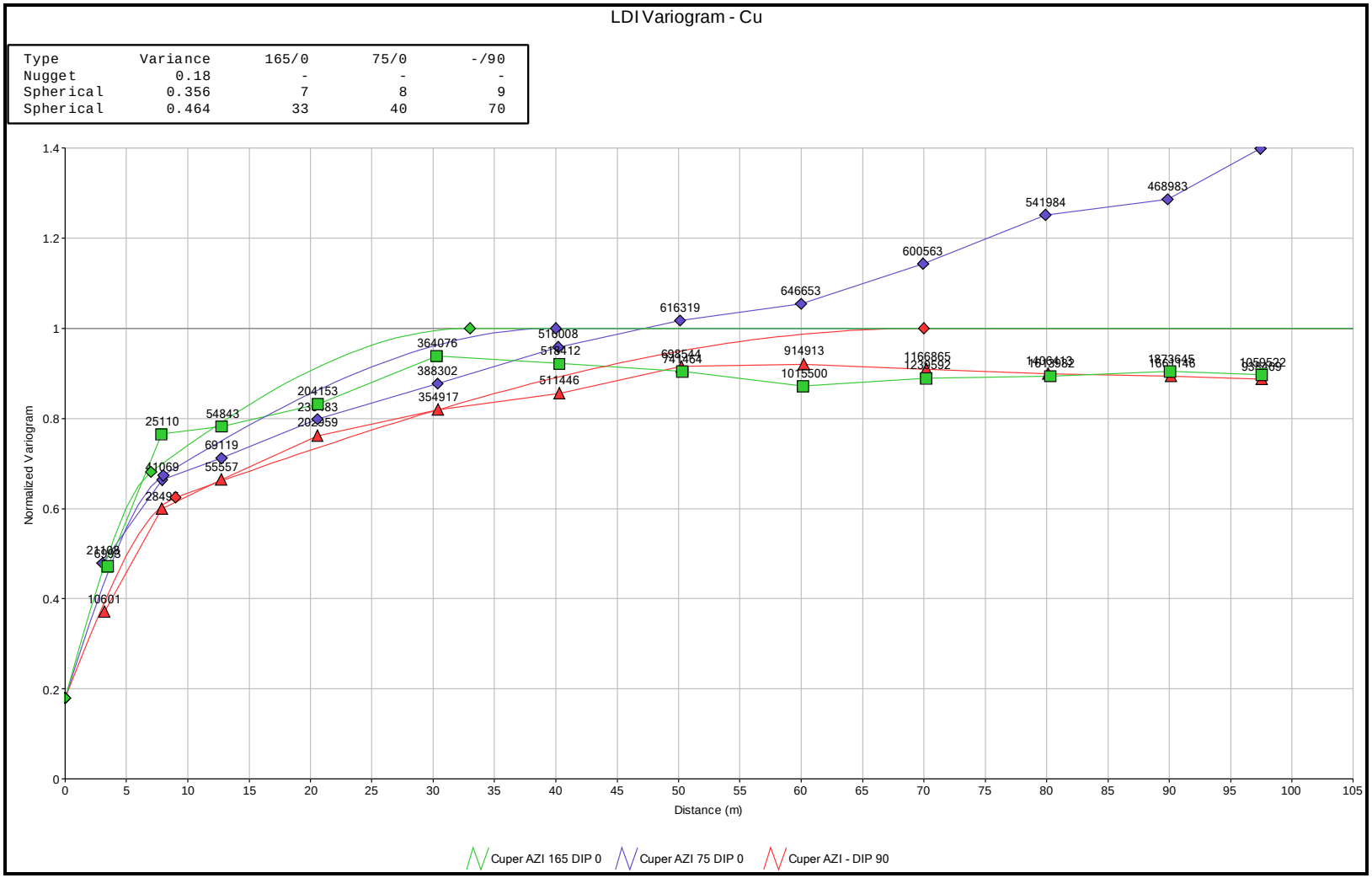


Figure 14.7 Gold Variogram

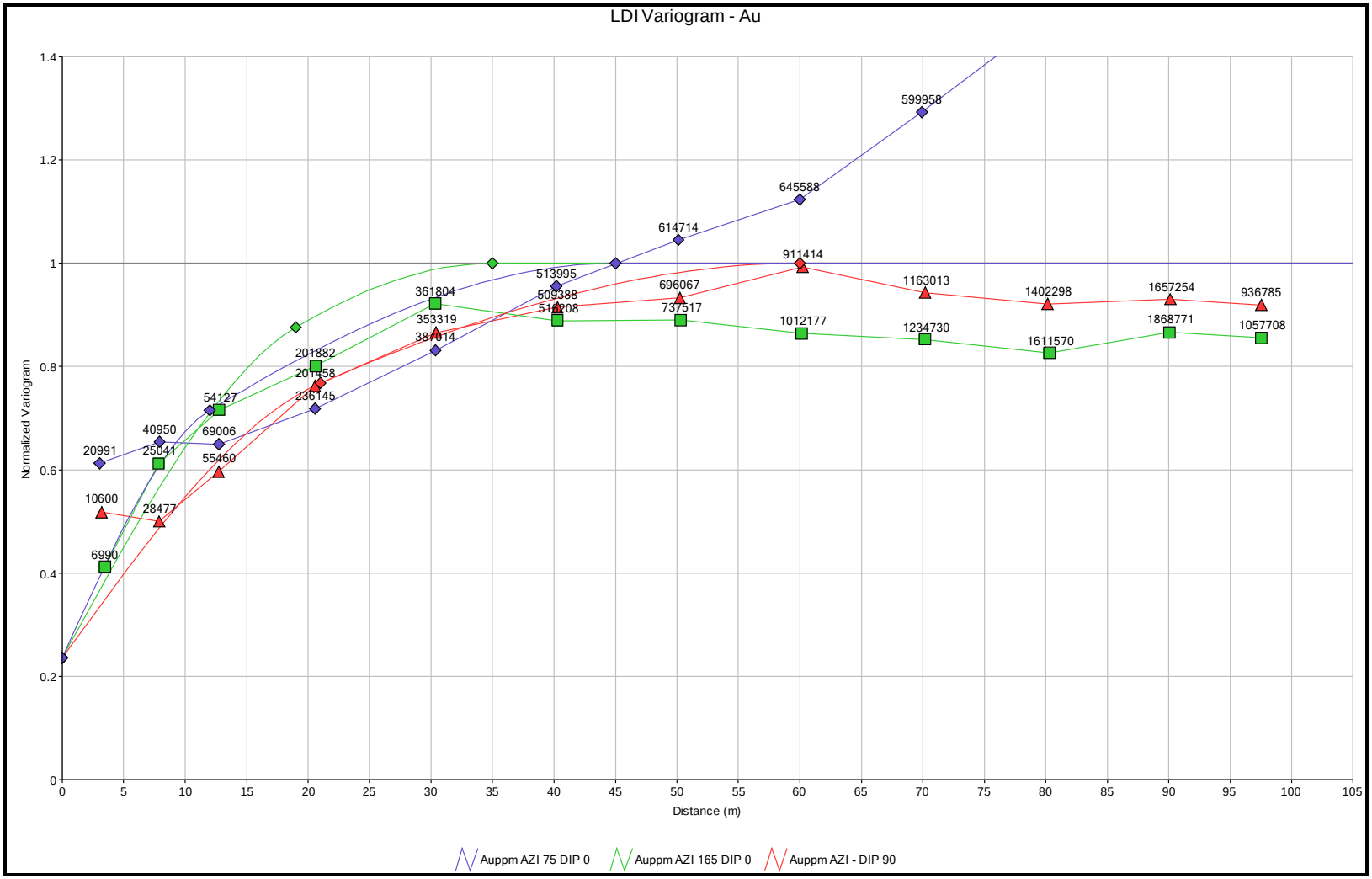


Figure 14.8 Silver Variogram

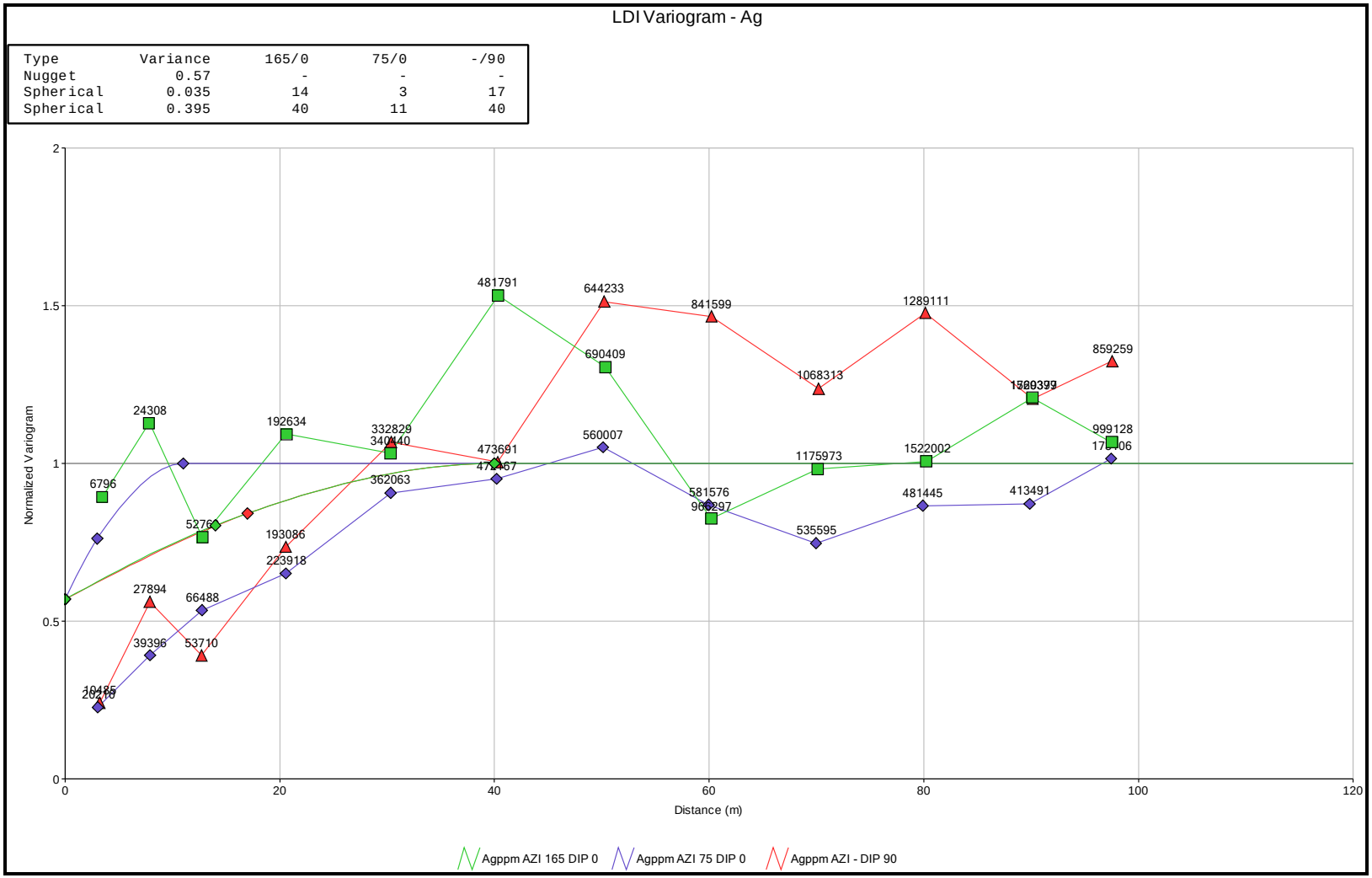


Figure 14.9 Cobalt Variogram

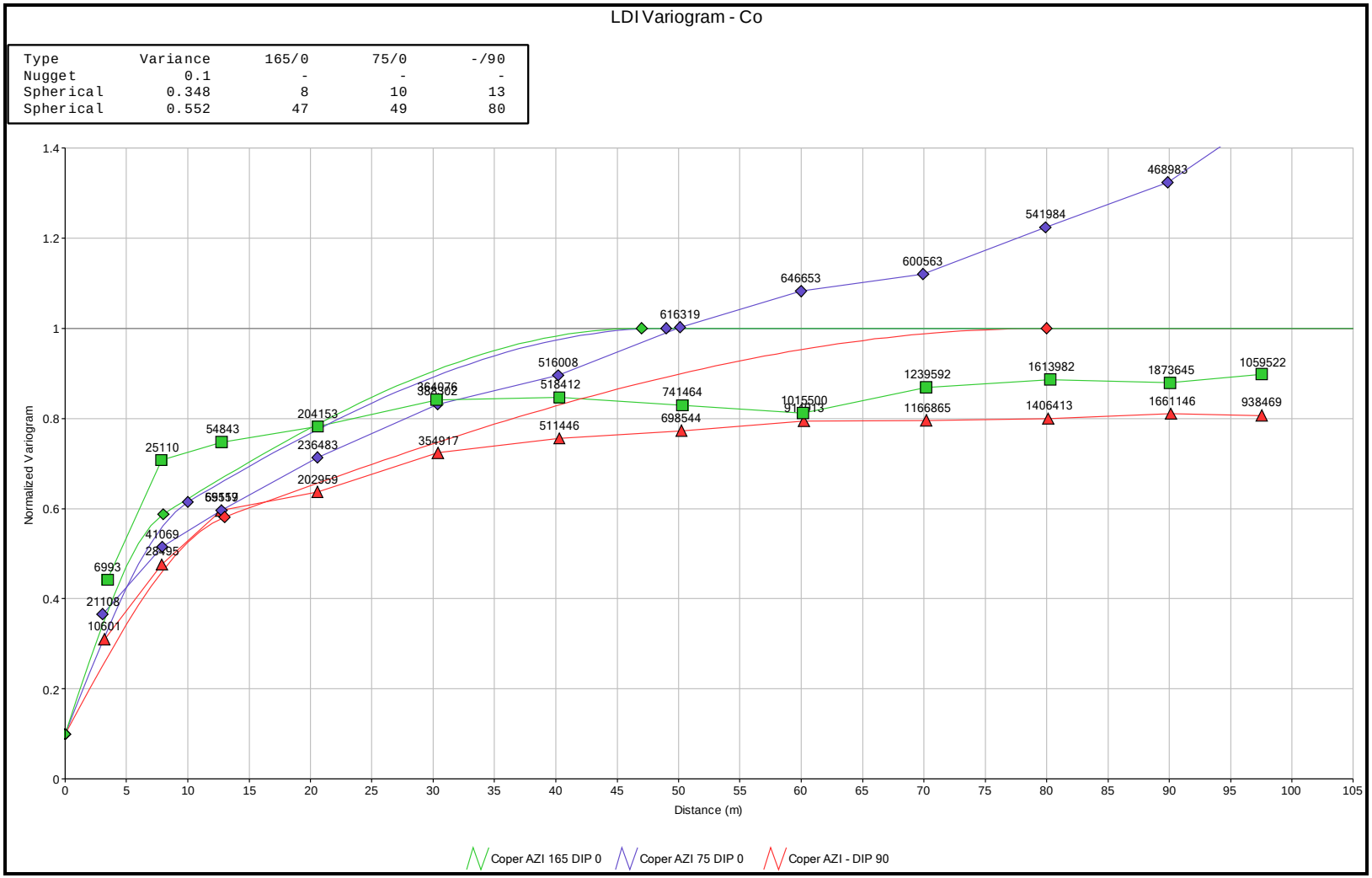


Figure 14.10 Sulphur Variogram

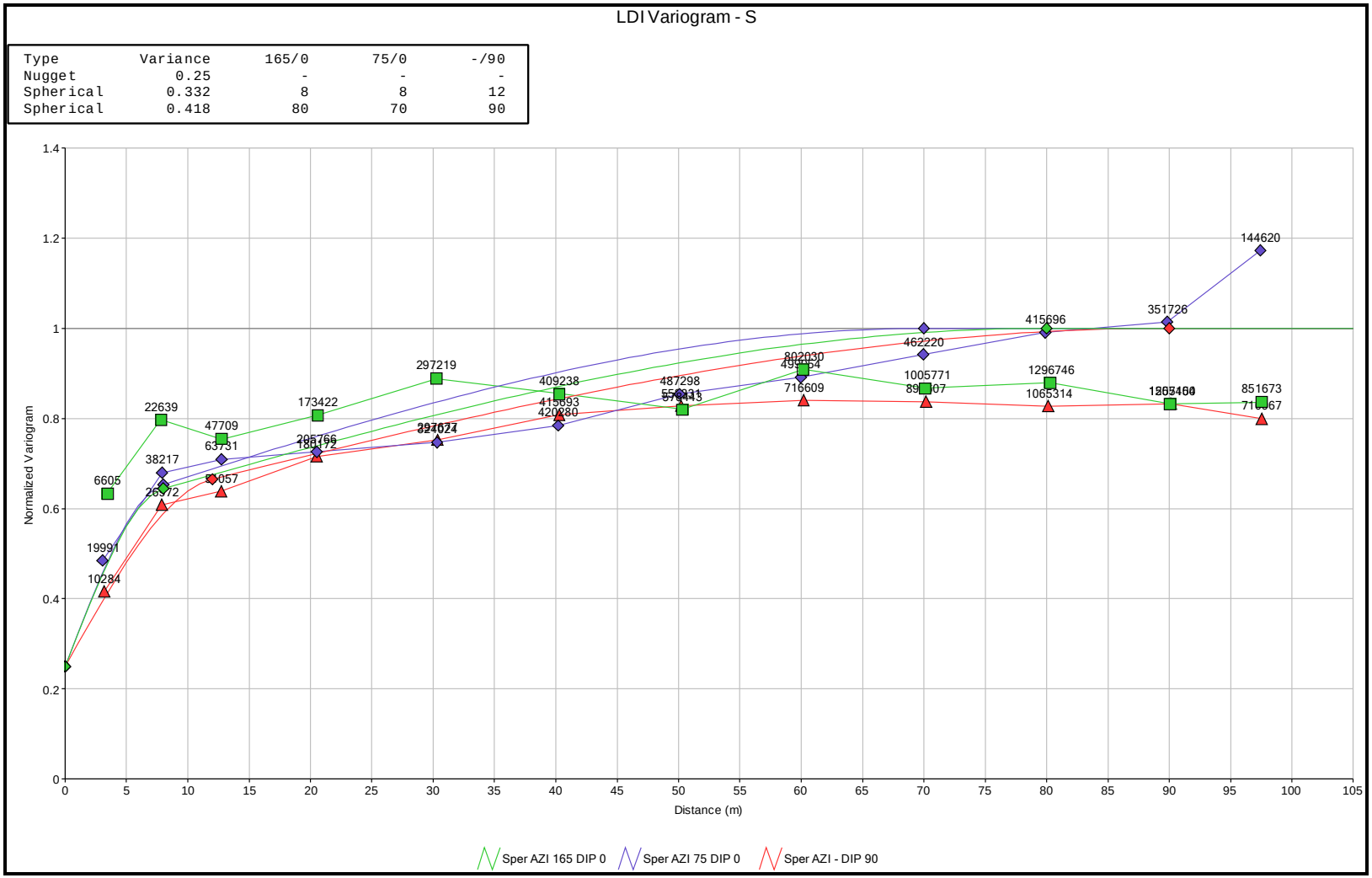
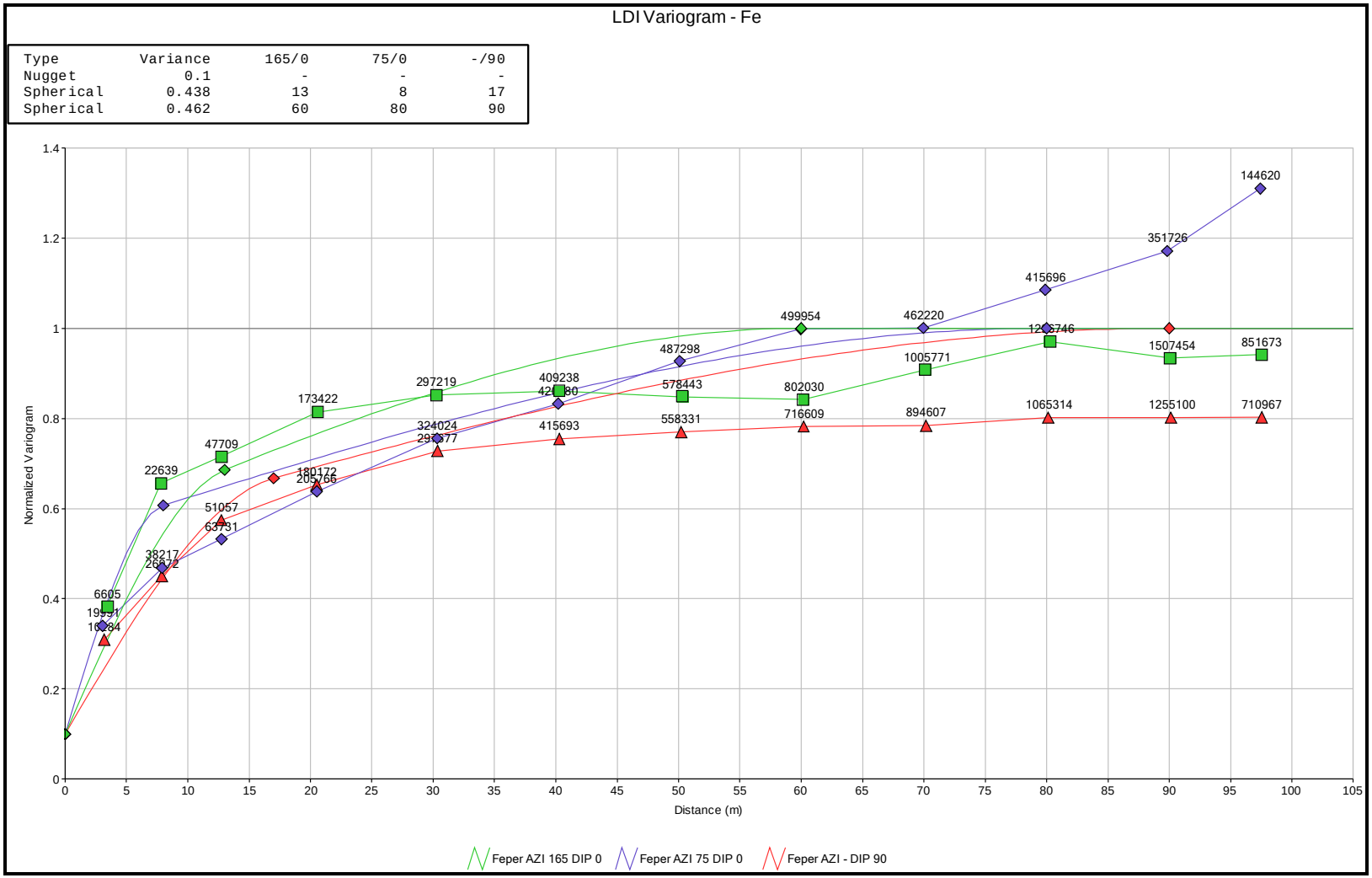


Figure 14.11 Iron Variogram



14.1.7 RESOURCE BLOCK MODEL

Individual block models were established in Datamine™ for the 1 and 2 g/t mineral wireframes using one parent model as the origin. The model was not rotated.

Drillhole spacing is variable with the majority of the surface drilling spaced at 25 m section and 25 to 100 m on sections. A block size of 5 m by 5 m by 5 m was selected in order to accommodate the nature of the mineralization and be amenable for underground mining potential.

Sub-celling of the block model on a 2.5 by 2.5 by 2.5 pattern in the ZY plane allows the parent block to be split in each direction to more accurately fill the volume of the wireframes, thus more accurately estimate the tonnes in the resource.

At the end of the modelling process, the 2 g/t model was overlain on the 1 g/t model.

Table 14.8 summarizes details of the parent block model.

Table 14.8 Parent Model Summary

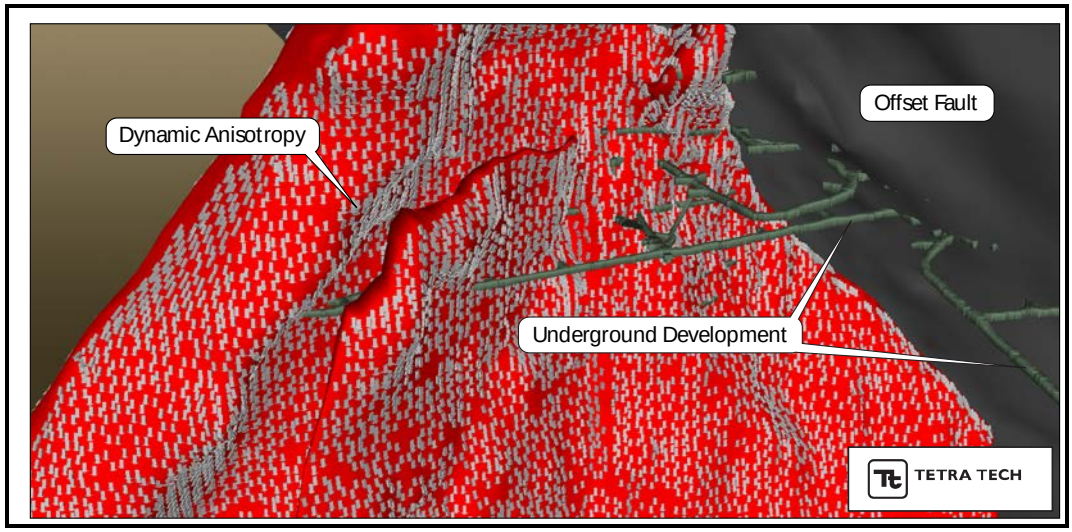
Origin			Cell Size			Number of Cells		
X Origin	Y Origin	Z Origin	XINC	YINC	ZINC	NX	NY	NZ
31000	31000	-1300	5	5	5	400	400	320

DYNAMIC ANISOTROPY

Due to the erratic nature of the wireframes compared to the likely geology geometry and the distribution of the mineralization within the zones, a single search ellipse would not be practical and would result in the smearing of grades.

Dynamic anisotropy is an option in Datamine™ Studio 3 that allows the anisotropy rotation angles that define search volumes and variogram models to be defined individually for each cell in the model, thus allowing the search volume to be precisely oriented to follow the trend of the mineralization. Figure 14.12 is an example on how the orientation of the search ellipse will vary across the mineralized zone.

Figure 14.12 Dynamic Anisotropy Directions



ESTIMATION AND SEARCH PARAMETERS

The interpolations of the zones were completed using the estimation methods: NN, ID^2 and OK. 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 borehole in order to satisfy the estimation criteria.

Estimation runs were completed in two steps. Step one involved the estimation was completed on the 1 g/t grade shell. The second step was the estimation on the more restricted 2 g/t grade shell. The results of the 2 g/t grade shell model were overlain on the results of the 1 g/t grade shell. This allowed the higher-grade hanging wall domain to be preserved and eliminate the potential to grade smearing across strike.

Table 14.9 and Table 14.10 summarize the interpolation criteria for the two mineral wireframes.

Table 14.9 Estimation Parameters

Description	Estimate Reference No.	VALUE_IN	VALUE_OUT	Search Reference No.	NUMSAM_F	SVOL_F	IMETHOD	VREFNUM
1 g/t Grade Shell								
Pdnn	1	Pdppm	Pdnn	1.000	-	-	1	1
Pdid	2	Pdppm	Pdid	1.000	-	-	2	1
Pdok	3	Pdppm	Pdok	1.000	NUMSAMP	SVOL	3	1
Ptnn	4	Ptppm	Ptnn	2.000	-	-	1	2
Ptid	5	Ptppm	Ptid	2.000	-	-	2	2
Ptok	6	Ptppm	Ptok	2.000	-	-	3	2
Aunn	7	Auppm	Aunn	3.000	-	-	1	3
Auid	8	Auppm	Auid	3.000	-	-	2	3
Auok	9	Auppm	Auok	3.000	-	-	3	3
Cunn	10	Cuper	Cunn	4.000	-	-	1	4
Cuid	11	Cuper	Cuid	4.000	-	-	2	4
Cuok	12	Cuper	Cuok	4.000	-	-	3	4
Ninn	13	Niper	Ninn	5.000	-	-	1	5
Niid	14	Niper	Niid	5.000	-	-	2	5
Niok	15	Niper	Niok	5.000	-	-	3	5
Conn	13	Coper	Conn	6.000	-	-	1	6
Coid	14	Coper	Coid	6.000	-	-	2	6
Cook	15	Coper	Cook	6.000	-	-	3	6

table continues…

Description	Estimate Reference No.	VALUE_IN	VALUE_OUT	Search Reference No.	NUMSAM_F	SVOL_F	IMETHOD	VREFNUM
2 g/t Grade Shell								
Pdnn	1	Pdppm	Pdnn	1.000	-	-	1	1
Pdid	2	Pdppm	Pdid	1.000	-	-	2	1
Pdok	3	Pdppm	Pdok	1.000	NUMSAM	SVOL	3	1
Ptnn	4	Ptppm	Ptnn	2.000	-	-	1	2
Ptid	5	Ptppm	Ptid	2.000	-	-	2	2
Ptok	6	Ptppm	Ptok	2.000	-	-	3	2
Aunn	7	Auppm	Aunn	3.000	-	-	1	3
Auid	8	Auppm	Auid	3.000	-	-	2	3
Auok	9	Auppm	Auok	3.000	-	-	3	3
Cunn	10	Cuper	Cunn	4.000	-	-	1	4
Cuid	11	Cuper	Cuid	4.000	-	-	2	4
Cuok	12	Cuper	Cuok	4.000	-	-	3	4
Ninn	13	Niper	Ninn	5.000	-	-	1	5
Niid	14	Niper	Niid	5.000	-	-	2	5
Niok	15	Niper	Niok	5.000	-	-	3	5
Conn	13	Coper	Conn	6.000	-	-	1	6
Coid	14	Coper	Coid	6.000	-	-	2	6
Cook	15	Coper	Cook	6.000	-	-	3	6

Table 14.10 Search Parameters

1 g/t Grade Shell											
Element	SREFNUM	Search Method	Search Distance Along Strike (X)	Search Distance Down Dip (Z)	Search Distance Across Strike (Y)	Z Axis Rotation	Y Axis Rotation	X Axis Rotation	DA Angle Z	DA Angle Y	DA Angle X
Pd	1	ellipse	9	25	30	-15	15	0	TRDIPDIR	-	TRDIP
Pt	2	ellipse	9	20	30	-15	15	0	TRDIPDIR	-	TRDIP
Au	3	ellipse	9	20	30	30	15	0	TRDIPDIR	-	TRDIP
Cu	4	ellipse	9	20	30	-15	15	0	TRDIPDIR	-	TRDIP
Ni	5	ellipse	9	20	30	-15	15	0	TRDIPDIR	-	TRDIP
Co	6	ellipse	9	24.5	30	-15	15	0	TRDIPDIR	-	TRDIP

SVOLFAC1	Minimum No. of Samples	Maximum No. of Samples	SVOLFAC2	Minimum No. Samples	Maximum No. of Samples	SVOLFAC3	Minimum No. of Samples	Maximum No. of Samples
1	15	20	2	8	15	8	3	10

Octant Method	Minimum No. of Octant	Minimum Octant	Maximum Octant	Maximum Samples/Borehole
0	0	0	0	5

2 g/t Grade Shell											
Element	SREFNUM	Search Method	Search Distance Along Strike (X)	Search Distance Down Dip (Z)	Search Distance Across Strike (Y)	Z Axis Rotation	Y Axis Rotation	X Axis Rotation	DA Angle Z	DA Angle Y	DA Angle X
Pd	1	ellipse	10	30	30	-15	15	0	TRDIPDIR	-	TRDIP
Pt	2	ellipse	10	30	30	-15	15	0	TRDIPDIR	-	TRDIP
Au	3	ellipse	10	30	30	30	15	0	TRDIPDIR	-	TRDIP
Cu	4	ellipse	10	30	30	-15	15	0	TRDIPDIR	-	TRDIP
Ni	5	ellipse	10	30	30	-15	15	0	TRDIPDIR	-	TRDIP
Co	6	ellipse	10	30	30	-15	15	0	TRDIPDIR	-	TRDIP

SVOLFAC1	Minimum No. of Samples	Maximum No. of Samples	SVOLFAC2	Minimum No. Samples	Maximum No. of Samples	SVOLFAC3	Minimum No. of Samples	Maximum No. of Samples
1	10	25	2	6	25	8	3	25

Octant Method	Minimum No. of Octant	Minimum Octant	Maximum Octant	Maximum Samples/Borehole
0	0	0	0	5

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
- authors experience with ultramafic-mafic hosted PGE deposits
- 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 Tetra Tech 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, mineral resources, which are not mineral reserves, do not have to demonstrate economic viability.

14.1.9 MINERAL RESOURCE TABULATION

The resource reported as of March 31, 2012 has been tabulated in terms of a palladium cut-off grade. Figure 14.13 to Figure 14.15 and Table 14.11 to Table 14.13 are the grade tonnage curve and tables for the Offset Zone for each of the resource categories. The resources are tabulated using various cut-off grades to demonstrate the robust nature of the resource.

Table 14.11 Offset Zone Measured Grade-Tonnage Table

Palladium Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Co (%)
1.0	18,771,980	3.32	0.26	0.24	0.10	0.08	0.01
1.1	17,941,240	3.43	0.27	0.24	0.10	0.08	0.01
1.2	17,157,330	3.53	0.27	0.25	0.11	0.08	0.01
1.3	16,412,690	3.64	0.28	0.26	0.11	0.08	0.01
1.4	15,697,430	3.74	0.29	0.26	0.11	0.08	0.01
1.5	15,015,780	3.84	0.29	0.27	0.11	0.08	0.01
1.6	14,451,930	3.93	0.30	0.28	0.11	0.09	0.01
1.7	13,884,040	4.03	0.30	0.28	0.11	0.09	0.01

table continues…

Palladium Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Co (%)
1.8	13,361,620	4.12	0.31	0.29	0.11	0.09	0.01
1.9	12,885,690	4.20	0.31	0.29	0.12	0.09	0.01
2.0	12,447,940	4.28	0.32	0.30	0.12	0.09	0.01
2.1	12,019,850	4.36	0.32	0.30	0.12	0.09	0.01
2.2	11,609,470	4.44	0.32	0.31	0.12	0.09	0.01
2.3	11,212,640	4.51	0.33	0.31	0.12	0.09	0.01
2.4	10,826,850	4.59	0.33	0.32	0.12	0.09	0.01
2.5	**10,436,550**	**4.67**	**0.33**	**0.32**	**0.12**	**0.09**	**0.01**
2.6	10,062,080	4.75	0.34	0.33	0.12	0.09	0.01
2.7	9,701,740	4.83	0.34	0.33	0.13	0.10	0.01
2.8	9,344,170	4.91	0.34	0.34	0.13	0.10	0.01
2.9	9,006,310	4.98	0.35	0.34	0.13	0.10	0.01
3.0	8,662,920	5.06	0.35	0.35	0.13	0.10	0.01
3.1	8,355,940	5.14	0.36	0.35	0.13	0.10	0.01
3.2	8,038,610	5.22	0.36	0.36	0.13	0.10	0.01
3.3	7,732,940	5.29	0.36	0.36	0.13	0.10	0.01
3.4	7,436,620	5.37	0.37	0.37	0.13	0.10	0.01
3.5	7,135,820	5.45	0.37	0.37	0.13	0.10	0.01
3.6	6,858,460	5.53	0.37	0.37	0.13	0.10	0.01
3.7	6,574,580	5.61	0.38	0.38	0.14	0.10	0.01
3.8	6,294,440	5.69	0.38	0.38	0.14	0.10	0.01
3.9	6,011,230	5.78	0.38	0.39	0.14	0.10	0.01
4.0	5,738,760	5.87	0.39	0.39	0.14	0.10	0.01
4.1	5,471,420	5.96	0.39	0.39	0.14	0.10	0.01
4.2	5,191,360	6.05	0.40	0.40	0.14	0.11	0.01
4.3	4,953,630	6.14	0.40	0.40	0.14	0.11	0.01
4.4	4,702,020	6.24	0.40	0.41	0.14	0.11	0.01
4.5	4,484,680	6.32	0.41	0.41	0.14	0.11	0.01

Figure 14.13 Offset Zone Measured Grade-Tonnage Curve

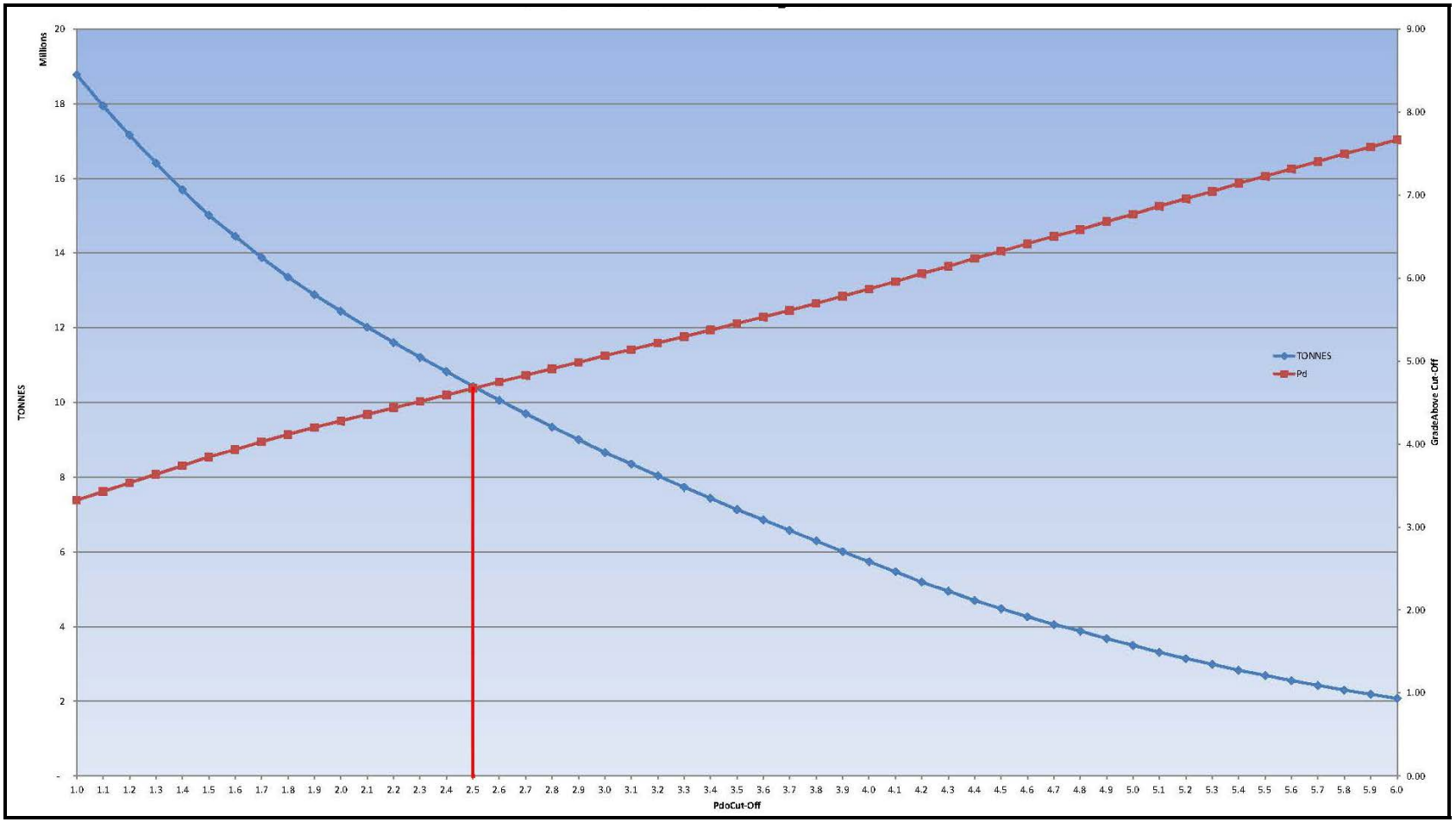


Table 14.12 Offset Zone Indicated Grade-Tonnage Table

Palladium Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Co (%)
1.0	36,256,100	2.41	0.22	0.19	0.09	0.07	0.01
1.1	33,440,000	2.52	0.23	0.20	0.09	0.07	0.01
1.2	30,811,800	2.64	0.24	0.20	0.09	0.07	0.01
1.3	28,374,300	2.76	0.25	0.21	0.09	0.07	0.01
1.4	26,051,400	2.88	0.25	0.22	0.09	0.08	0.01
1.5	23,920,300	3.01	0.26	0.23	0.10	0.08	0.01
1.6	21,976,300	3.14	0.27	0.24	0.10	0.08	0.01
1.7	20,184,200	3.27	0.27	0.24	0.10	0.08	0.01
1.8	18,564,500	3.40	0.28	0.25	0.10	0.08	0.01
1.9	17,105,400	3.53	0.29	0.26	0.10	0.09	0.01
2.0	15,852,600	3.66	0.29	0.27	0.11	0.09	0.01
2.1	14,779,370	3.78	0.30	0.27	0.11	0.09	0.01
2.2	13,832,900	3.89	0.30	0.28	0.11	0.09	0.01
2.3	12,958,600	4.00	0.31	0.29	0.11	0.09	0.01
2.4	12,219,400	4.10	0.31	0.29	0.11	0.09	0.01
2.5	**11,615,600**	**4.18**	**0.32**	**0.30**	**0.11**	**0.09**	**0.01**
2.6	10,999,400	4.28	0.32	0.30	0.11	0.09	0.01
2.7	10,431,200	4.36	0.32	0.31	0.12	0.10	0.01
2.8	9,882,800	4.45	0.33	0.31	0.12	0.10	0.01
2.9	9,412,000	4.53	0.33	0.32	0.12	0.10	0.01
3.0	8,970,500	4.61	0.33	0.32	0.12	0.10	0.01
3.1	8,541,700	4.69	0.34	0.33	0.12	0.10	0.01
3.2	8,153,600	4.76	0.34	0.33	0.12	0.10	0.01
3.3	7,775,800	4.84	0.34	0.33	0.12	0.10	0.01
3.4	7,409,700	4.91	0.35	0.34	0.12	0.10	0.01
3.5	7,062,300	4.98	0.35	0.34	0.12	0.10	0.01
3.6	6,704,800	5.06	0.36	0.34	0.12	0.10	0.01
3.7	6,353,600	5.14	0.36	0.35	0.12	0.10	0.01
3.8	6,017,400	5.22	0.36	0.35	0.13	0.10	0.01
3.9	5,661,300	5.30	0.37	0.36	0.13	0.10	0.01
4.0	5,320,100	5.39	0.37	0.36	0.13	0.10	0.01
4.1	4,988,800	5.48	0.37	0.37	0.13	0.11	0.01
4.2	4,673,800	5.57	0.38	0.37	0.13	0.11	0.01
4.3	4,359,900	5.66	0.38	0.38	0.13	0.11	0.01
4.4	4,070,100	5.75	0.39	0.38	0.13	0.11	0.01
4.5	3,792,000	5.85	0.39	0.39	0.13	0.11	0.01

Figure 14.14 Offset Zone Indicated Grade-Tonnage Curve

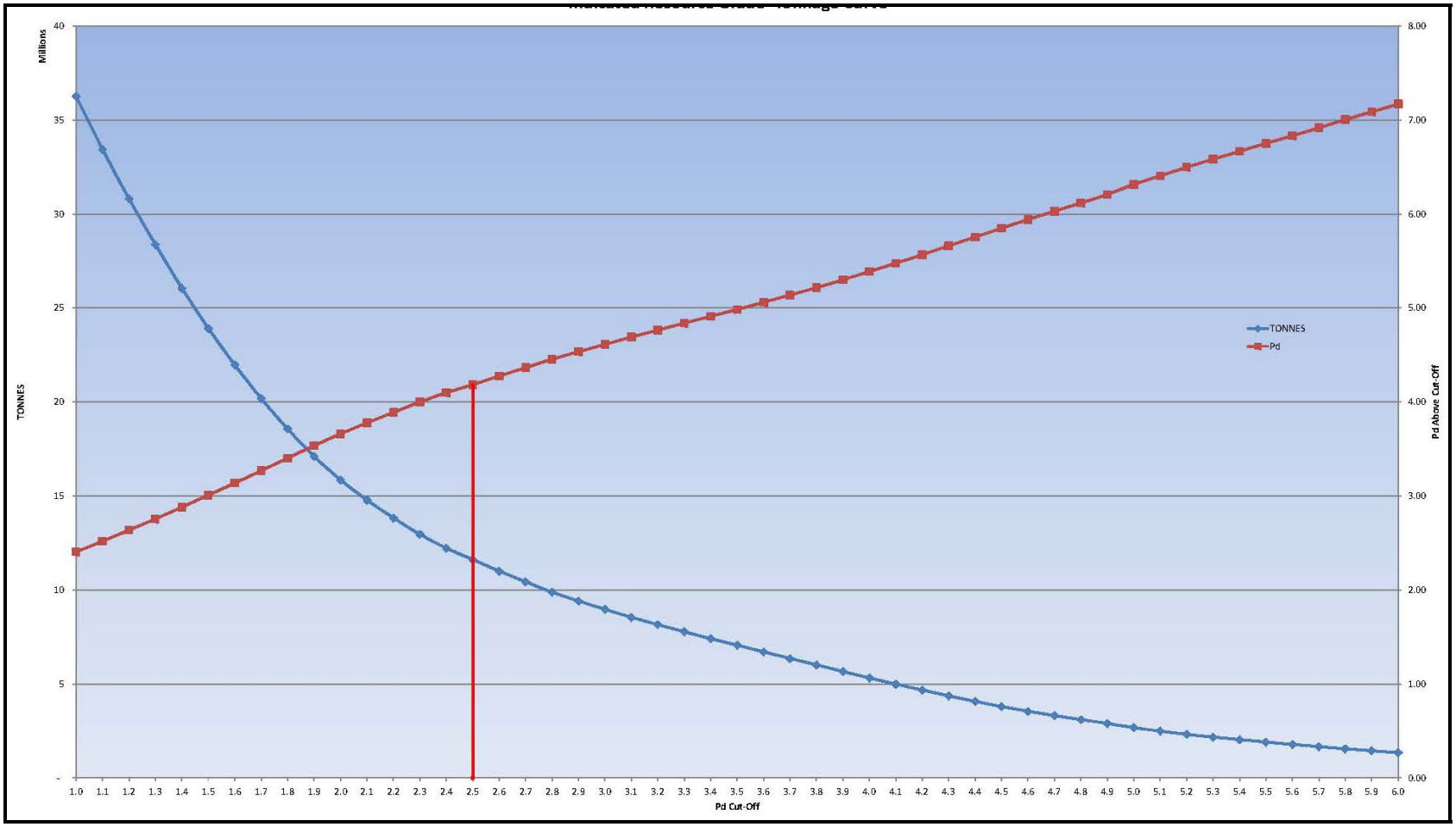
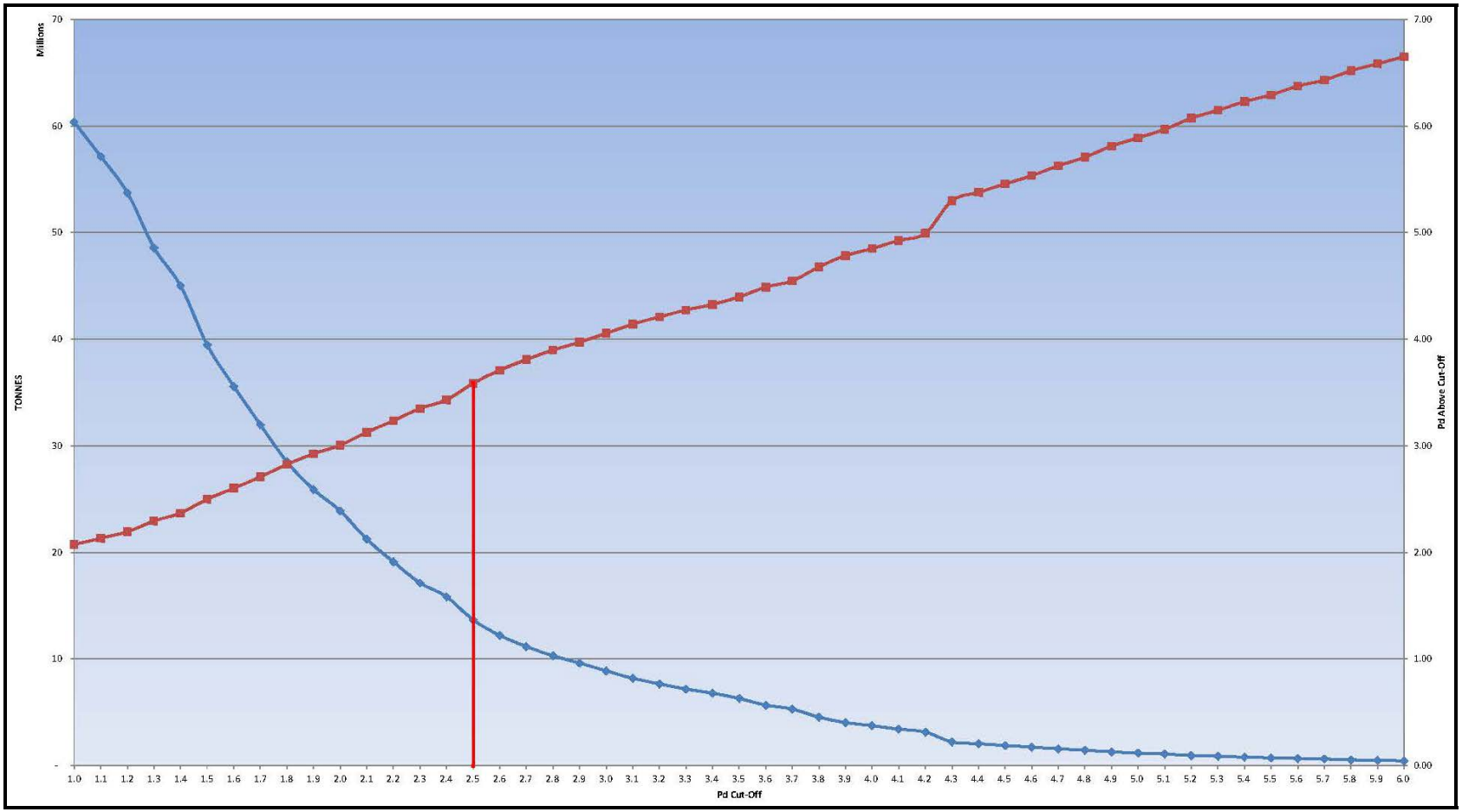


Table 14.13 Offset Zone Inferred Grade-Tonnage Table

Palladium Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Co (%)
1.0	60,370,000	2.07	0.20	0.15	0.07	0.06	0.01
1.1	57,137,000	2.13	0.20	0.15	0.07	0.06	0.01
1.2	53,732,000	2.19	0.21	0.15	0.07	0.06	0.01
1.3	48,581,000	2.29	0.21	0.16	0.07	0.06	0.01
1.4	45,002,000	2.37	0.22	0.16	0.07	0.06	0.01
1.5	39,460,000	2.50	0.23	0.17	0.08	0.06	0.01
1.6	35,577,000	2.60	0.23	0.17	0.08	0.06	0.01
1.7	31,968,000	2.71	0.24	0.18	0.08	0.06	0.01
1.8	28,505,000	2.83	0.25	0.18	0.08	0.07	0.01
1.9	25,894,000	2.92	0.25	0.19	0.08	0.07	0.01
2.0	23,895,000	3.01	0.26	0.19	0.08	0.07	0.01
2.1	21,242,000	3.13	0.27	0.20	0.09	0.07	0.01
2.2	19,117,000	3.23	0.27	0.21	0.09	0.07	0.01
2.3	17,134,000	3.35	0.28	0.22	0.09	0.07	0.01
2.4	15,838,000	3.43	0.29	0.23	0.09	0.07	0.01
2.5	**13,669,000**	**3.59**	**0.30**	**0.23**	**0.09**	**0.08**	**0.01**
2.6	12,215,000	3.71	0.31	0.25	0.10	0.08	0.01
2.7	11,166,000	3.81	0.32	0.25	0.10	0.08	0.01
2.8	10,299,000	3.90	0.32	0.26	0.10	0.08	0.01
2.9	9,615,000	3.97	0.33	0.27	0.10	0.08	0.01
3.0	8,890,000	4.06	0.34	0.27	0.10	0.08	0.01
3.1	8,190,000	4.14	0.34	0.28	0.10	0.08	0.01
3.2	7,669,000	4.21	0.35	0.28	0.10	0.08	0.01
3.3	7,184,000	4.27	0.36	0.29	0.10	0.08	0.01
3.4	6,797,000	4.33	0.36	0.29	0.10	0.09	0.01
3.5	6,292,000	4.40	0.37	0.30	0.10	0.09	0.01
3.6	5,675,000	4.49	0.37	0.30	0.11	0.09	0.01
3.7	5,302,000	4.55	0.38	0.31	0.11	0.09	0.01
3.8	4,553,000	4.68	0.38	0.31	0.11	0.09	0.01
3.9	4,037,000	4.78	0.38	0.32	0.11	0.09	0.01
4.0	3,745,000	4.85	0.39	0.33	0.11	0.09	0.01
4.1	3,418,000	4.93	0.39	0.33	0.11	0.09	0.01
4.2	3,142,000	4.99	0.40	0.34	0.11	0.09	0.01
4.3	2,221,000	5.30	0.37	0.33	0.11	0.09	0.01
4.4	2,050,000	5.38	0.38	0.34	0.11	0.09	0.01
4.5	1,888,000	5.46	0.38	0.34	0.11	0.09	0.01

Figure 14.15 Offset Zone Inferred Grade-Tonnage Curve



Based on the current and future operating plans at the mine, a 2.5 g/t palladium cut-off was used to tabulate the resource. Table 14.14 summarizes the resource estimate for each of the resource categories in the Offset Zone.

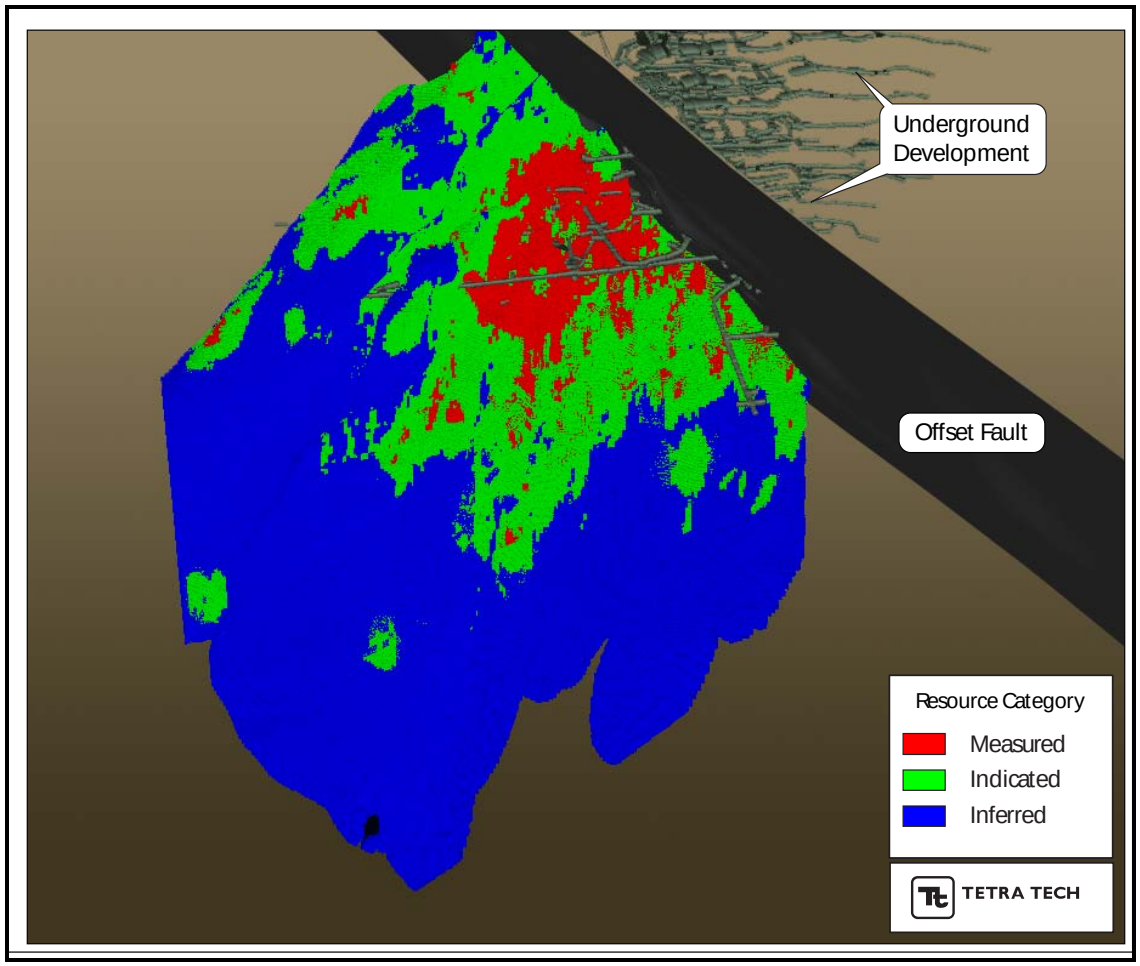
Table 14.14 Offset Mineral Resource Summary

Category	Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Co (%)	Pd (oz)
Measured	2.5	10,436,550	4.67	0.33	0.32	0.12	0.09	0.01	1,566,989
Indicated	2.5	11,615,600	4.18	0.32	0.30	0.11	0.09	0.01	1,561,026
Measured + Indicated	2.5	22,052,150	4.41	0.32	0.31	0.12	0.09	0.01	3,128,015
Inferred	2.5	13,669,000	3.59	0.30	0.23	0.09	0.08	0.01	1,575,840

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. As per NI 43-101, mineral resources, which are not mineral reserves, do not have to demonstrated economic viability

The distribution of the resource categories is displayed in Figure 14.16.

Figure 14.16 Offset Zone Resource Category Distribution



14.1.10 VALIDATION

The Offset models were validated by three methods:

- visual comparison of colour-coded block model grades with composite grades on section and plan
- comparison of the global mean block grades for OK, ID^2, NN and composites
- swath plots of the various zones in both plan and section views.

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, yet grade smoothing is

apparent in some locations due to the distance between drill samples being broader in some regions.

Figure 14.17 and Figure 14.18 display the comparison between the block model and the original drillholes.

Figure 14.17 Offset Zone Section

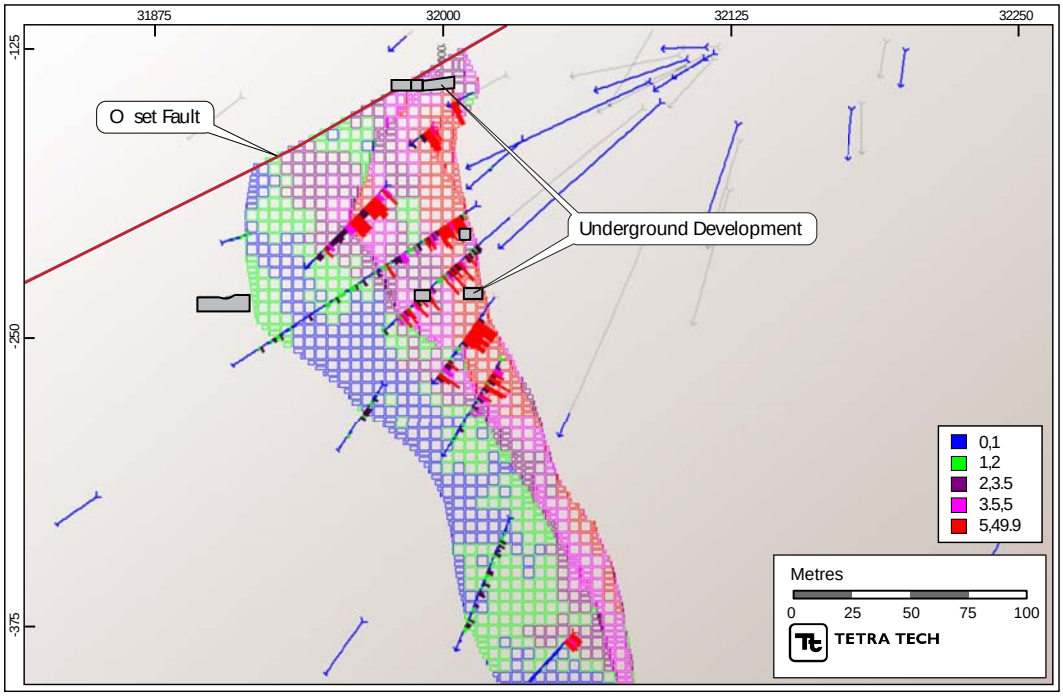
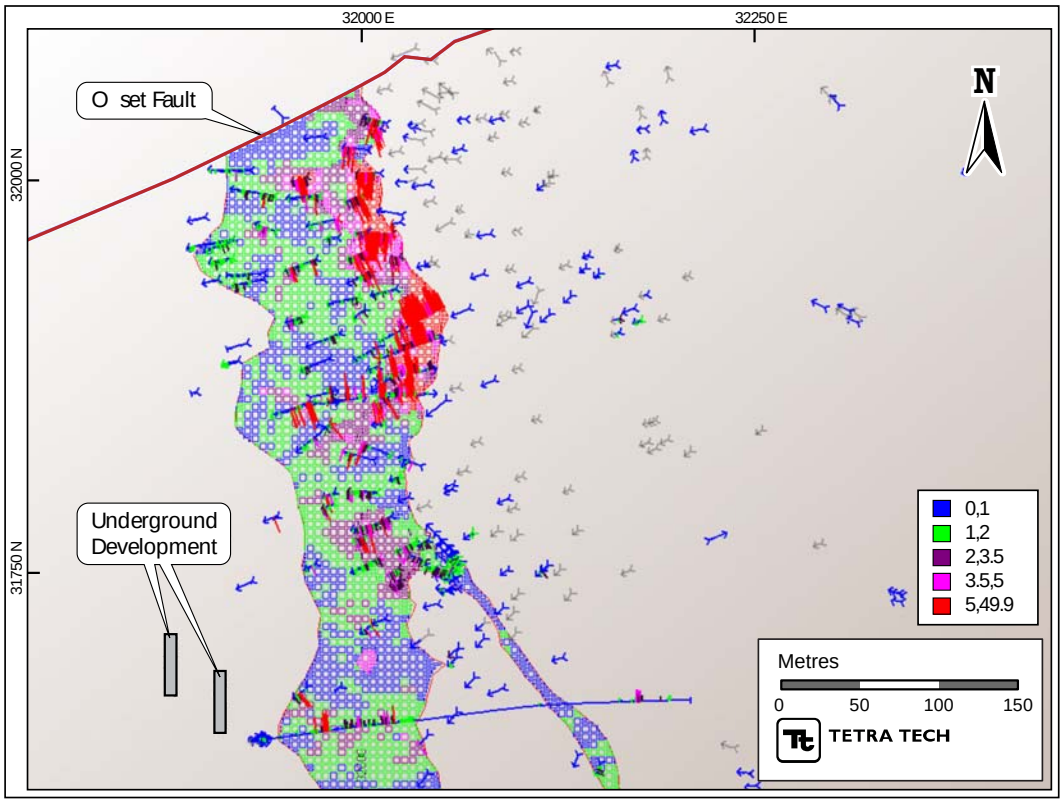


Figure 14.18 Offset Zone Plan



GLOBAL COMPARISON

The global block model statistics for the OK model were compared to the global ID^2 and NN model values as well as the composite capped drillhole data. Table 14.15 shows this comparison of the global estimates for the three estimation method calculations. In general, there is agreement between the OK model, the ID^2 model, and the NN 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 diamond drill statistics. Comparisons were made using all blocks at a 0% cut-off.

Table 14.15 Offset Global Statistics Comparison

	Palladium				Platinum				Gold			
	DDH	OK	ID^2	NN	DDH	OK	ID^2	NN	DDH	OK	ID^2	NN
Minimum (g/t)	0.00	0.01	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum (g/t)	49.80	19.29	21.10	38.70	3.07	1.52	1.42	3.11	5.38	1.70	1.88	3.01
Mean (g/t)	2.45	2.26	2.27	2.21	0.21	0.21	0.21	0.21	0.18	0.17	0.16	0.16
Standard Deviation	3.18	1.63	1.72	2.41	0.21	0.11	0.12	0.17	0.25	0.12	0.13	0.20
	Copper				Nickel				Cobalt			
	DDH	OK	ID^2	NN	DDH	OK	ID^2	NN	DDH	OK	ID^2	NN
Minimum (%)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum (%)	1.59	0.53	0.54	1.34	1.25	0.40	0.44	0.70	0.06	0.02	0.02	0.03
Mean (%)	0.06	0.06	0.06	0.06	0.08	0.08	0.08	0.08	0.01	0.01	0.01	0.01
Standard Deviation	0.07	0.04	0.04	0.06	0.07	0.04	0.04	0.05	0.00	0.00	0.00	0.00

Note: DDH = diamond drillhole

SWATH PLOTS

Swath plots of eastings, northings, and elevations were generated for the Offset Zone. These plots are comparing the OK estimates with the NN and ID^2 estimates and are illustrated Figure 14.19, Figure 14.20, and Figure 14.21.

Figure 14.19 Offset Zone Easting Plot

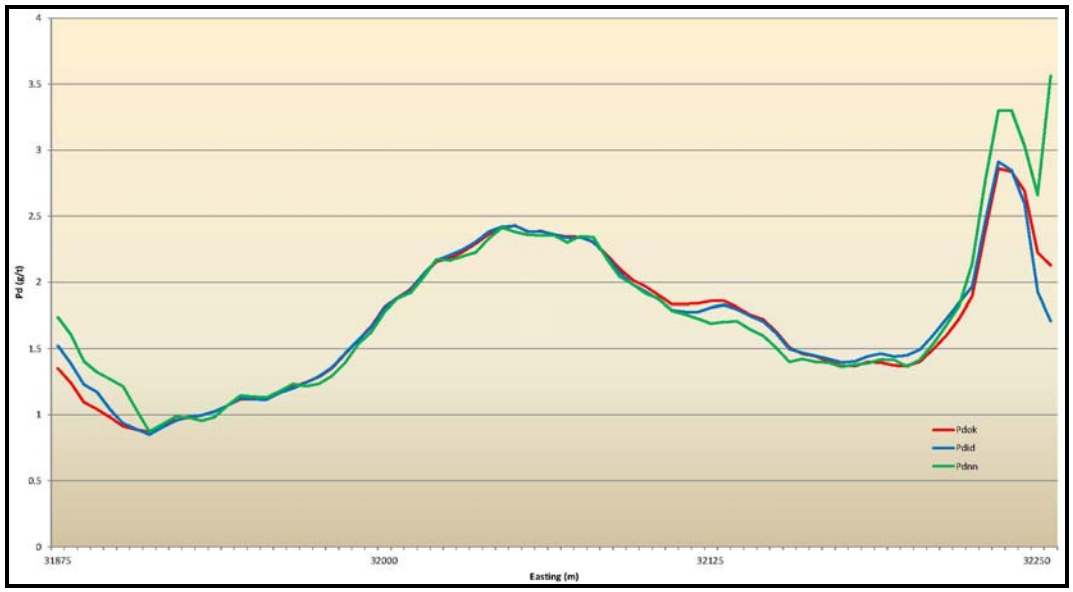


Figure 14.20 Offset Zone Northing Plot

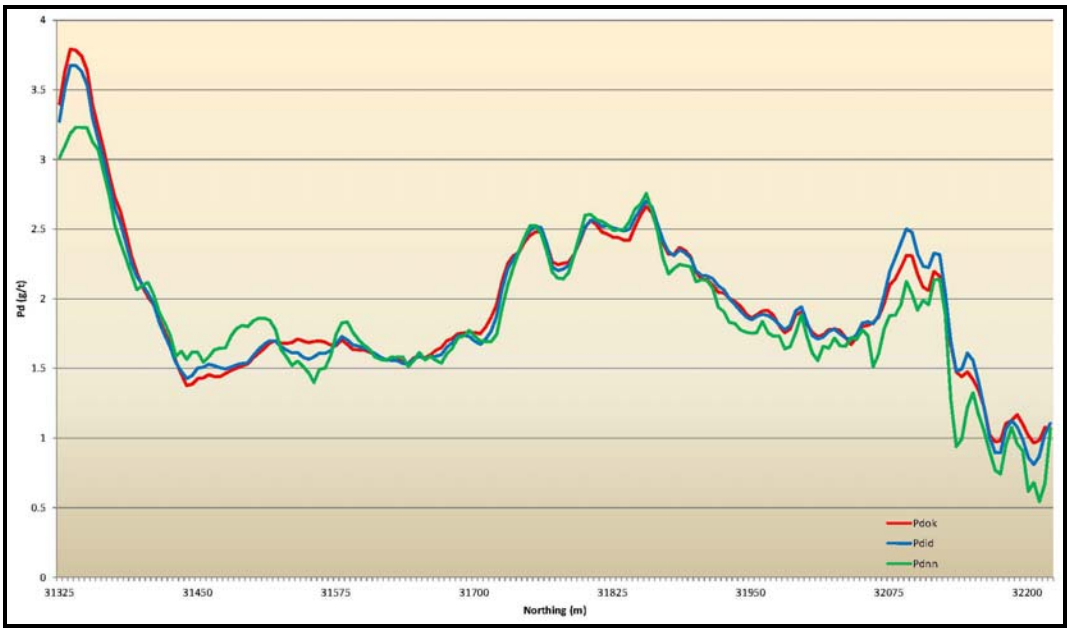
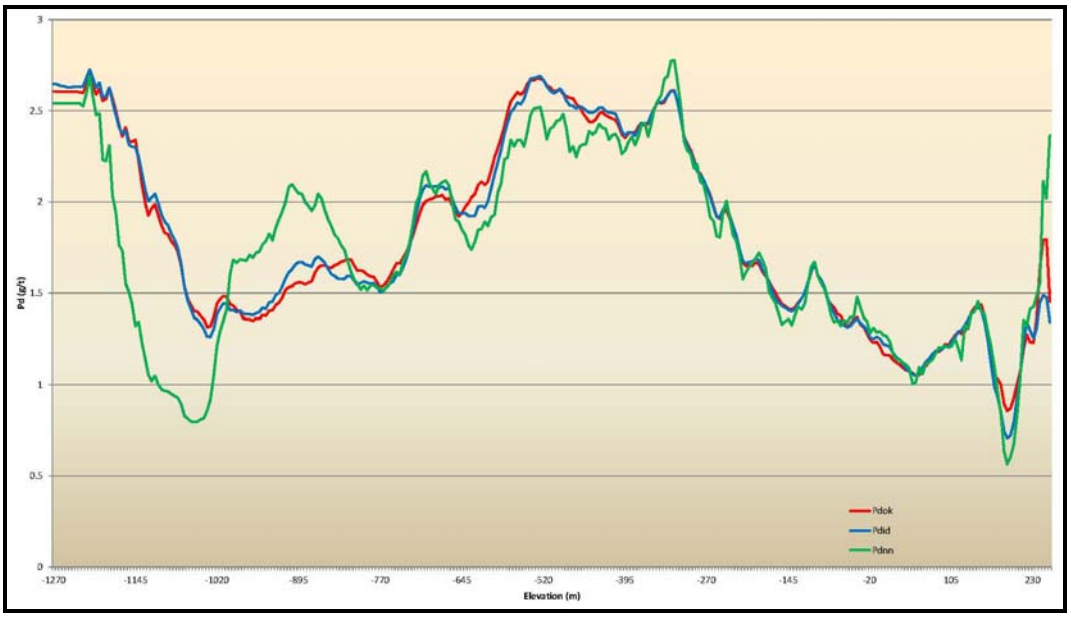


Figure 14.21 Offset Zone Plan Plot



14.1.11 PREVIOUS ESTIMATES

NAP had commissioned Scott Wilson RPA to generate a resource estimate in 2010 (www.nap.com). This estimated was based on the premise that the resource could potentially be extracted using traditional underground mining methods.

Table 14.16 compares the basic parameters of the previous 2010 estimate with the current 2012 mineral resource.

Table 14.16 Modelling Parameter Comparison

	2010 Scott Wilson RPA	2012 Tetra Tech
Number of Drillholes	188	539
Grade Capping	Pd capped at 20 g/t globally Pd capped at 30 g/t locally	Parrish Analysis no grade capping applied
Composite Length	1.5 m downhole	1.0 m average, back stitching allows for "tail" material to be spread evenly over the entire hole composite
Cut-off Grade	4.0 g/t Pd	2.5 g/t Pd for potential underground resource
Specific Gravity	2.89	2.89
Number of Mineral Zones	One zone generated around a 4.0 g/t minimum grade	One 1 g/t grade shell with a 2 g/t internal grade shell
Block Size	5 m by 5 m by 5 m	5 m by 5 m by 5 m (125 m^3) - single sub-cell
Estimation Method	OK with ID^2 and NN validation	OK with ID^2 and NN validation

The primary difference between the 2010 resource model and the 2012 resource model is due to a 187% increase in the number of boreholes used in the estimate and the use of a lower cut-off grade to highlight the potential for larger bulk mining particularly in the footwall environment. The reason for using a lower cut-off grade is explained in Section 15.0. The result is a 79% increase in the contained ounces of palladium being reported in the combined Measured and Indicated Resource and a 159% increase in the contained ounces of palladium being reported in the Inferred Resource. Table 14.17 illustrates the differences in the 2010 resource estimate with the current NI 43-101 compliant resource from 2012.

Table 14.17 Comparisons of 2010 and 2012 Resource Model

	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Co (%)	Pd (oz)
2010 Scott Wilson RPA Consulting OK Model								
Measured Resource @ 4.0 g/t Au cut-off	0	0	0	0	0	0	0	-
Indicated Resource @ 4.0 g/t Au cut-off	8,628,000	6.29	0.42	0.40	0.14	0.11	0.01	1,744,828
2012 Tetra Tech Model								
Measured Resource @ 2.5 g/t Pd cut-off	10,436,550	4.67	0.33	0.32	0.12	0.09	0.01	1,566,920
Indicated Resource @ 2.5 g/t Pd cut-off	11,615,600	4.18	0.32	0.30	0.11	0.09	0.01	1,562,692
Inferred Resource @ 2.5 g/t Pd cut-off	13,669,000	3.59	0.30	0.23	0.09	0.08	0.01	1,575,879

14.2 ROBY AND OPEN PIT ZONES

14.2.1 INTRODUCTION

Lac des Iles Mine has updated the resource balance for the remaining zones, exclusive of the Offset Zone, on the Property. The resources were generated in June 2010 by Scott Wilson RPA (Buck et al. 2010). The update is based in the depletion of resource material or conversion to reserves by LDIM. The effective date of the resource is March 31, 2012.

Table 14.18 summarizes the resource on the Property, excluding the Offset Zone.

Table 14.18 Lac des Iles Mine Resource Summary

Category	Pd Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pd ('000 oz)
Measured								
Open Pit	1.8	1,971,000	2.00	0.24	0.15	0.05	0.07	127
Stockpile	1.9	83,000	1.63	0.17	0.14	0.08	0.06	4
Total Measured	-	2,054,000	1.99	0.24	0.15	0.05	0.07	131
Indicated								
Roby Underground	5.8	1,269,000	7.16	0.41	0.33	0.08	0.06	292
Open Pit	1.8	2,565,000	2.20	0.24	0.18	0.07	0.08	181
Low-grade Stockpile	0.5	13,188,000	0.97	0.12	0.08	0.06	0.03	411
Total Indicated	-	17,022,000	1.62	0.16	0.11	0.06	0.04	885
Measured + Indicated	-	19,076,000	1.66	0.17	0.12	0.06	0.04	1,016

Notes: Prepared in accordance with NI 43-101 Standards of Disclosure for Mineral Projects and the CIM classification system.

The mineral resource estimate for the Roby Zone open pit and stockpiles were estimated as of June 30, 2010 by Scott Wilson RPA and updated by David N. Penna, P.Geo., an employee of LDIM and a QP under NI 43-101 to reflect: (i) additions to mineral reserves in the Roby Zone as a result of a lower cut-off palladium grade; (ii) depletion from production up to March 31, 2012, and (iii) mineral reserves from the crown pillar (supported by an internal engineering report). The following cut-off grades were used: (i) 1.8 g/t palladium equivalent (PdEq) for the Roby open pit, within an optimized pit shell run below the current pit survey; (ii) 1.9 g/t PdEq for the mine stockpiles; and (iii) 5.8 g/t PdEq for the underground Roby Zone. These cut-off grades were determined under the assumption that production would take place at a rate of 14,000 t/d. Metal price assumptions of US$350/oz palladium, US$1,400/oz platinum, US$850/oz gold, US$6.50/lb nickel, and US$2.00/lb copper were used in the estimation of cut-off grade. A US$/CDN$ exchange rate of 1.11 was also applied.

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

Numbers may not add due to rounding.

The open pit Measured Resource represents the resource inside the designed open pit shell provided by P&E Mining Consultants (P&E). A portion of these resources have been reclassified as Probable Reserves.

The open pit Indicated Resource represents the resource outside the designed P&E pit shell and is unchanged since June 30, 2010.

The stockpile Measured Resource consisted of approximately 600,000 t of open pit oversize material in 2010. The bulk of this material was depleted in 2010 and the remainder is lower-grade stockpiles.

The stockpile includes an Indicated Resource of approximately 13 Mt of supplemental material grading 0.97 g/t palladium. Depletions have been applied since June 30, 2010.

The Roby Zone underground Indicated Resource consists of pillars and material inside the resource solid, but currently uneconomic. This has been unchanged since June 30, 2010.

15.0 MINERAL RESERVE ESTIMATES

15.1 OFFSET ZONE

The reserves estimated for the Lac des Iles Mine are based on the Measured and Indicated Resource material included in the resource block model for the Lac des Iles Offset Zone. The block model was prepared by an independent QP within the meaning of NI 43-101. The mineral resource calculations used a 1 g/t palladium resource block cut-off grade, and a block size of 5 m by 5 m by 5 m. The estimated in situ Measured and Indicated Resource reported from the block model at the 2.5 g/t palladium cut-off grade is summarized in Table 14.14. This is reflective of the current and future operating plans of the Lac des Iles Mine, which is outlined in Section 16.0.

The mine design presented in Section 16.0 reflects the most likely access, mining, and production scenarios for the Offset Zone's Measured and Indicated Resources. The mine design is based on current practice at the mine site. The mine design includes all development and construction required to access the Measured and Indicated Resources for mining by a bulk mining method and to transport the mined ore to the existing milling facilities on site.

15.1.1 OFFSET ZONE MINING STOPE SIZE

The stoping block size was derived by reviewing the Roby Zone mining stopes and reducing the size to account for increased field stresses with depth. The generalized stoping block size was 40 to 50 m high, 22 m wide and 35 m deep, which allowed for flexible mining using various mine sequences and enough tonnage to allow for high productivities.

15.1.2 MINING COSTS

The costs of the base case mine design used in this report were based on the current costs and future forecasted mining costs, which are summarized in Table 15.1.

Table 15.1 Offset Zone Operating Costs ($/t)

Area	2013	2014	2015	2016	2017	2018	Total
Surface	7.78	7.78	8.26	9.14	9.14	9.14	8.51
Mine	34.37	28.80	23.70	17.34	16.61	16.48	23.07
Mill	14.44	14.44	15.14	15.91	16.72	15.35	15.31
G&A	9.15	8.02	7.89	8.96	8.36	7.29	8.27
Total	**65.74**	**59.04**	**54.99**	**51.35**	**50.83**	**48.26**	**55.16**

15.1.3 METAL PRICES

NAP relies on a number of independent industry organizations for market analysis and metal price forecasts for precious metals. These organizations include CPM Group, Johnson Matthey, Mitsui Global Precious Metals, GFMS, as well as a number of major banks with dedicated commodities market research groups. These organizations and market research groups are recognized worldwide as authorities in PGM markets. 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. Table 15.2 summarizes the long term forecasts of metals and exchange rate.

Table 15.2 Long-term Metal Prices and Exchange Rate (Five Years)

Metal Price/ Exchange Rate	Amount ($)
Pd (US$/oz)	675
Pt (US$/oz)	1,675
Au (US$/oz)	1,750
Cu (US$/lb)	3.50
Ni (US$/lb)	8.00
Foreign Exchange (US$ = CDN$)	1.00

15.1.4 MILL RECOVERY

The predicted mill recovery to concentrate is shown in Table 15.4. Mill improvements referenced in Section 17.0 have led to a prediction by NAP for better recoveries. The actual Lac des Iles Mine operating mill recoveries for previous years of operation are summarized in Table 15.3.

Table 15.3 Milling Recovery

Area	2012	2011	2010	2008	2007	2006	2005	2004	2003	2002
Tonnes Milled	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	3.44	3.70	6.06	2.33	2.36	2.18	1.66	2.41	2.31	1.91
Pd Recovery	78.37	78.35	80.80	75.30	74.80	74.00	69.60	75.20	75.50	73.80
Pt Recovery	72.87	72.01	74.21	64.70	64.70	64.90	60.60	66.90	-	-
Au Recovery	76.92	74.07	77.21	74.60	72.90	73.00	70.00	74.40	-	-
Cu Recovery	85.47	83.45	86.34	84.20	83.80	82.90	80.50	83.50	-	-
Ni Recovery	36.8	30.83	34.38	36.90	34.90	35.20	30.00	40.50	-	-
Concentrate Tonnes	17,883	11,708	5,517	26,176	33,060	28,979	30,698	45,652	36,714	27,200
Pd (oz)	178,835	157,476	102,212	212,046	286,334	237,338	177,167	308,931	288,703	219,300
Pt (oz)	12,528	10,265	5,499	16,311	24,442	22,308	18,833	25,128	23,472	19,200
Au (oz)	12,487	8,314	4,624	15,921	20,092	17,238	14,308	25,679	23,536	16,000
Cu (lb)	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)	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	98.3	97.9	95.7	88.4	91.1	86.5	86.5	88.4	91.1	90.5

15.1.5 MINE CUT-OFF GRADE

The mine cut-off grade was calculated using the cost data presented above as well as long term metal prices and milling recovery. Smelter returns, treatment costs, and freight costs were included to calculate the NSR for in situ material. The net smelter return value equates to a cut-off grade of 2.5 g/t of palladium. The NSR calculation includes the smelter treatment and shipping costs per tonne of concentrate and the metal prices and predicted concentrate recoveries summarized in Table 15.4.

Table 15.4 Long-term Metal Prices and NSR Calculation

Element	Metal Price (US$)	Metal Price (US$)	Concentrate Recovery (%)
Ni	8.00/lb	176.37/t	43.13
Cu	3.50/lb	77.16/t	88.55
Co	-	-	-
Au	1,750.00/oz	56.26/g	83.97
Pt	1,675.00/oz	53.85/g	71.30
Pd	675.00/oz	21.70/g	80.45

Note: NSR at 2.5 g/t palladium cut-off grade to mill is $66.20/t milled

15.1.6 MINING DILUTION AND ORE LOSS

The mining dilution and ore loss factors are based on actual results from past mining in the upper Roby Mine. These results were reviewed and updated to reflect changes in mining method and stope geometry. Table 15.5 summarizes the expected mining recovery and dilution factors applied to determine the Offset Zone ore reserves.

Table 15.5 Mining Dilution and Ore Loss Factors

Mining Method	Recovery (%)	External Dilution (%)	
		525 L-735 L	825 L-990 L
Bulk Open Stoping	90	15	14
Upper Crown Mining	60	25	25

15.1.7 3D MINING SHAPES

The Offset Zone block model was reviewed in plan and in section to identify zones of measured and indicated resource material above 2.5 g/t of palladium cut-off grade and to determine the most appropriate bulk mining shape.

Sub levels were established at 40 to 50 m vertical intervals, and vertical sections were cut through the model at 10 m intervals along strike. Mining shapes were created on each section based on interpolation of the hanging wall contact and the interface

between the high grade material and low grade blocks (below 2.5 g/t). When the high grade material thickness was greater than 35 m, the mining shape was cut to keep the shape thickness less than or equal to 35 m. No foot wall shape was created for the reserve calculation. The shape was created to respect the mining method and could therefore contain some low grade material (which is the internal dilution component of total dilution). A 3D wire frame was created utilizing all the sectional shapes resulting in a mining wireframe from which individual stoping blocks were created.

15.1.8 EXTERNAL DILUTION

The mining shape created boundaries from which individual stope blocks were created. The bulk mining stope size was limited to 40 to 50 m high, 22 m wide, and 35 m thick (hanging wall to foot wall). This size was used to estimate an external dilution factor (unplanned dilution) to apply to the in situ stope resource. The weighted average equivalent external dilution has been estimated to be between 14 and 25% at an average 2.5 g/t palladium grade. To calculate the dilution grade, a 5 m thick 3D wireframe has been created on the hanging wall of the mining wire frame. The average grade of this wire frame was estimated from the block model using the percent method, which assigns a 0 g/t palladium grade for the percentage of each block outside the general 1 g/t palladium wireframe, and the grade from the resource block model for the percentage of each block inside the general wireframe. The same process was repeated for the other three sides. All four sides were then aggregated to depict the total external dilution to the stope block.

15.1.9 OFFSET ZONE RESERVES

The tonnes and grades contained within the stope wireframes were extracted from the block model of measured and indicated resource data. Stope plans were made on those resources that qualified for conversion to reserves. External dilution and mining recovery factors have been applied to complete the reserve estimate. The results are summarized in Table 15.6.

Table 15.6 Offset Zone Estimated Reserves at 2.5 g/t Palladium Cut-off*

Classification	Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pd (oz)
Proven	2.50	5,632,000	4.34	0.30	0.30	0.11	0.08	786,535
Probable	2.50	2,109,000	4.17	0.30	0.29	0.10	0.08	282,644
Total Reserve	**-**	**7,741,000**	**4.30**	**0.30**	**0.30**	**0.11**	**0.08**	**1,069,179**

Note: *This zone of 7.74 Mt of reserves grading above 2.5 g/t palladium represents only a portion of the 22.05 Mt of the overall Measured and Indicated Resources also grading above 2.5 g/t, with most of the variance being Indicated Resources that require better definition.

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. U.S. investors should refer to the Annual Information Form for an overview on how Canadian standards differ significantly from U.S. requirements.

Palladium ounces are stated as contained ounces. Disclosure of contained ounces is permitted under Canadian regulations; however, the SEC generally permits resources to be reported only as in place tonnage and grade.

Mineral reserves for the Offset Zone were estimated by Todd McCracken, P. Geo.; William Richard McBride, P.Eng.; Todd Kanhai, P.Eng.; and Philip Bridson, P.Eng. of Tetra Tech, independent QPs within the meaning of NI 43-101. The mineral reserves were estimated from drilling completed to March 31, 2012. Reserves were estimated to the 990 Mine Level (4,490 m elevation), a maximum depth of 1,017.5 m. The following metal price assumptions were used for reserves estimation: US$675/oz palladium, US$1,675/oz platinum, US$1,750/oz gold, US$8.00/lb nickel, and US$3.50/lb copper. A US$/CDN$ exchange rate of US$1.00 = CDN$1.00 was also applied. An average production rate of 3,500 t/d was used to determine the reserves. The following recoveries were used in the assumptions to determine the reserves: 80.45% palladium, 71.30% platinum, 83.97% gold, 43.13% nickel, and 88.55% copper.

Numbers may not add due to rounding.

As stated in the note in Table 15.6, the remaining resources from Table 14.14 require better definition and were not included in the current reserve estimate. However, the stope plans and sequencing along with the development plans have been applied to these mindful of the future growth potential of the remaining resource. The concept will be to leverage the development for the reserves listed above to accommodate extensions of the existing mine development to readily access this anticipated growth as well as provide the exploration platform required to discover new material that could come into the global resource.

15.1.10 RISKS

Some of the key risks that have currently been identified which may materially affect the mineral reserve estimate are outlined below.

CAPITAL COST ESTIMATE

Cost pressures on capital expansion, such as labour and materials cost increases along with lower labour productivity could materially affect the mineral reserves due to increased carrying costs for project expansions.

FUTURE METAL PRICES

Price increase or decrease in metal values would have an effect on the total amount of mineral resources converting to reserves by changing the overall cut-off grade and NSRs used to calculate mineral profitability.

MILLING RECOVERY

Metallurgical changes due to mineral composition within the Offset Zone may result in decreased or increased recovery. This may affect the amount of material mined to be either increased or decreased for each unit of metal recovered to concentrate.

SITE OPERATING COSTS

Future cost increases or process improvements may cause increases or decreases in direct mining or recovery costs per tonne of reserve extracted.

15.2 ROBY AND OPEN PIT ZONES

15.2.1 INTRODUCTION

Lac des Iles Mine has updated the reserve balance for the remaining zones on the Property, exclusive of the Offset Zone. The reserve balance was generated in June 2010 by Scott Wilson RPA (Buck et al. 2010). The update is based on the depletion of reserve material by Lac des Iles Mine. The effective date of the reserve is March 31, 2012.

Table 15.7 summarizes the reserves on the Property excluding the Offset Zone.

Table 15.7 Roby Zone Reserve Summary

Classification	Cut-off (g/t)	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pd (oz)
Proven								
Roby	5.8	420,000	6.38	0.42	0.34	0.08	0.07	86,149
Open Pit	1.8	722,000	1.99	0.21	0.22	0.11	0.10	46,192
Probable	-	-	-	-	-	-	-	-
Total Reserve	**-**	**1,142,000**	**3.60**	**0.29**	**0.26**	**0.10**	**0.09**	**132,342**

Notes: Prepared in accordance with NI 43-101 Standards of Disclosure for Mineral Projects and the CIM classification system.

The mineral resource estimate for the Roby Zone open pit and stockpiles were estimated as of June 30, 2010 by Scott Wilson RPA and updated by David N. Penna, P.Geo., an employee of LDIM and a QP under NI 43-101 to reflect: (i) additions to mineral reserves in the Roby Zone as a result of a lower cut-off palladium grade; (ii) depletion from production up to March 31, 2012, and (iii) mineral reserves from the crown pillar (supported by an internal engineering report). The following cut-off grades were used: (i) 1.8 g/t PdEq for the Roby open pit, within an optimized pit shell run below the current pit survey; (ii) 1.9 g/t PdEq for the mine stockpiles; and (iii) 5.8 g/t PdEq for the underground Roby Zone. These cut-off grades were determined under the assumption that production would take place at a rate of 14,000 t/d. Metal price assumptions of US$350/oz palladium, US$1,400/oz platinum, US$850/oz gold, US$6.50/lb nickel, and US$2.00/lb copper were used in the estimation of cut-off grade. A US$/CDN$ exchange rate of 1.11 was also applied.

Numbers may not add due to rounding.

Roby underground reserves are based on mining depletions applied to the June 30, 2010 reserves for December 31, 2010 through March 31, 2012.

The open pit Measured Resource represents the resource inside the designed open pit shell provided by P&E. A portion of these resources have been reclassified as Probable Reserves.

16.0 MINING METHODS

16.1 MINING METHOD

16.1.1 ALTERNATIVES CONSIDERED

The PEA for the Lac des Iles Mine Offset Zone published on September 30, 2010 (Buck et al. 2010), proposed a bulk mining method, called super shrinkage, for use at the Lac Des Iles Mine. Since then, further information has led the Lac des Iles Mine personnel to consult with geotechnical experts including W. F. Bawden of Mine Design Engineering (MDE), Rimas Pakalnis of the University of British Columbia, and Itasca Consulting Canada Inc. (Itasca) for the purpose of reconsidering the use of the super shrinkage mining method in the Offset Zone.

In general, the geotechnical effects on the Offset Zone's rock mass have been updated with new information and further assessed.

Under the initial geotechnical assumptions, the super shrinkage bulk mining method, which typically offers higher productivity and lower operating costs than longhole stoping with backfill, was viewed as well suited for wide and continuous ore bodies such as the Lac des Iles Mine Offset Zone. This method was initially successfully deployed at Agnico-Eagle's Goldex Mine until uncontrolled block caving was induced, causing closure of this mine.

Shortly after publication of the PEA (Buck et al. 2010), geotechnical issues began to manifest themselves in some of the rib pillars in the middle portions of the adjoining Roby Zone. While these developments neither threatened life nor the Property, they clearly became the focus for new data collection by Lac des Iles Mine, as it was realized further improvements to the overall geotechnical model were necessary for the model to be applied to the deeper Offset Zone of the mine. Consequently, further infill diamond drilling for structures and regional baseline stress assessments were performed and the new data factored into the geotechnical model to test the potential effects on using the super shrinkage mining method in the Offset Zone.

This new 3D modelling assessment indicated a very highly-stressed sill pillar which could be prone to rock bursting with the large unfilled stope sizes contemplated under a super shrink scenario.

A report by MDE on March 10, 2011 (MDE 2010) also indicated significant borehole squeezing and heavy dynamic support would be required in sill drifts.

At that time a diamond drilling campaign was underway to better define the Offset Zone. That campaign showed the Offset Zone was growing and that sub-parallel resource potential existed.

This potential growth heightened geotechnical concerns for using the super shrinkage mining method in the Offset Zone since geotechnical issues could develop in the Offset Zone that could potentially sterilize the new sub-parallel resource potential.

These findings resulted in an engineering review and alternative considerations for the super shrinkage mining method to be used in the Offset Zone.

The principal underground mining methods considered were longhole stoping with cemented rock fill and the super shrinkage mining method (which was considered in the original MDE (2010) report). The two scenarios were developed to compare the overall mine stability.

This review concluded that significant areas of the deposit were not suitable for super shrinkage mining and therefore longhole mining became the chosen method for planning purposes.

16.1.2 CHOSEN METHOD

Based on this review, longhole stoping with cemented rock fill, using a primary and secondary stoping block sequence, was selected as the preferred mining method. This method provides the optimal level of flexibility within an overall extraction concept and allows modifications to the panel sizes, as needed to best suit the deposit.

The north and south sides of the core Offset Zone tend to be generally narrower in width and accordingly will be predominantly mined on longitudinal retreat instead of following the primary/secondary extraction sequence. It was further observed that this can be easily switched back to a primary/secondary format if additional infill drilling successes or improved economics lead to a wider ore body at the extremities of the Offset Zone.

This longhole stoping with cemented rock fill method best addresses the deposit's rock characteristics and allows for support of large openings as required. This mining method is better suited than super shrinkage for the size and geometry of the deposit, and has the added advantage of flexibility to coincidently access the sub-parallel mineralized zones without ore sterilization, which could have occurred under the super shrinkage scenario.

16.2 MINE ACCESS AND ORE TRANSPORTATION

As the Offset Zone is open at depth, the deepest mining level (based on current Phase I plans) will be located 990 m below surface, which represents the current economic limit of the resource, even though indications are that mineralization continues well below the 990 Level.

Mining will utilize mechanized equipment as was established previously in the 2010 PEA (Buck et al. 2010) mine design. Primary access for the workers and equipment will be provided via a shaft from surface. Underground movement of ore and waste will be through internal ore and waste passes then transferred to the shaft loading pockets for skipping to the surface.

The shaft option was selected over trucking up ramp based on the following considerations:

- excessive trucking distances from underground to surface
- a steeply dipping ore zone that is narrow in thickness and limited along strike
- shaft level connections are minimized and only required on a few key levels, which allows for central underground warehouses and maintenance complexes
- the existing primary ramp can readily be advanced into the Offset Zone to provide access to various production levels and provide a secondary egress from the mine in event of problems with the shaft.

The economics for the two access alternatives, as judged through NPV, can be considered to be equal.

16.3 NO. 1 SHAFT

The new production shaft referenced as the No. 1 Shaft going forward will be 6.0 m in diameter and concrete-lined. It will have a conventional back leg headframe equipped with:

- a production hoist for skipping
- a service hoist for a main cage
- the space for a Mary-Ann utility hoist should the Lac des Iles Mine elect to install this in the future.

As illustrated in Figure 16.1, the planned depth for the No. 1 Shaft is 825 m below surface, which includes a collar, sub-collar, and a ventilation plenum entering the

shaft below the sub-collar. The headframe infrastructure includes the headframe structure proper, collar house, and bin house.

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 all hoists which include:

- production hoist: 4.72 m in diameter, double drum hoist, 4.70 m diameter sheaves
- service hoist: 3.70 m in diameter, double drum hoist, 3.67 m diameter sheaves
- auxiliary hoist: 2.79 m in diameter, single drum hoist, 3.67 m diameter sheave.

All hoists will be converted to alternating current (AC) drives with a standardized motor size for all units.

16.3.1 SHAFT SIZE

The completed No. 1 Shaft will have a finished inner diameter of 6.0 m and a concrete lining of 150 mm. The diameter was selected to allow compartmentalized transport of men, materials, ventilation, and production at a rate of 5,234 t/d from a possible depth of 1,360 m. Shaft geotechnical support will include 2.1 m long friction bolts and 102 mm (4 in) square chain link followed by over-forming of the concrete liner.

16.3.2 SHAFT CONFIGURATION

As shown in Figure 16.2, the No. 1 Shaft will consist of the following four compartments:

- two skip compartments: skip size of 1,500 mm by 1,500 mm
- one 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
- one auxiliary cage compartment: auxiliary cage size of 1,840 mm by 710 mm.

Figure 16.1 No. 1 Shaft Sectional View

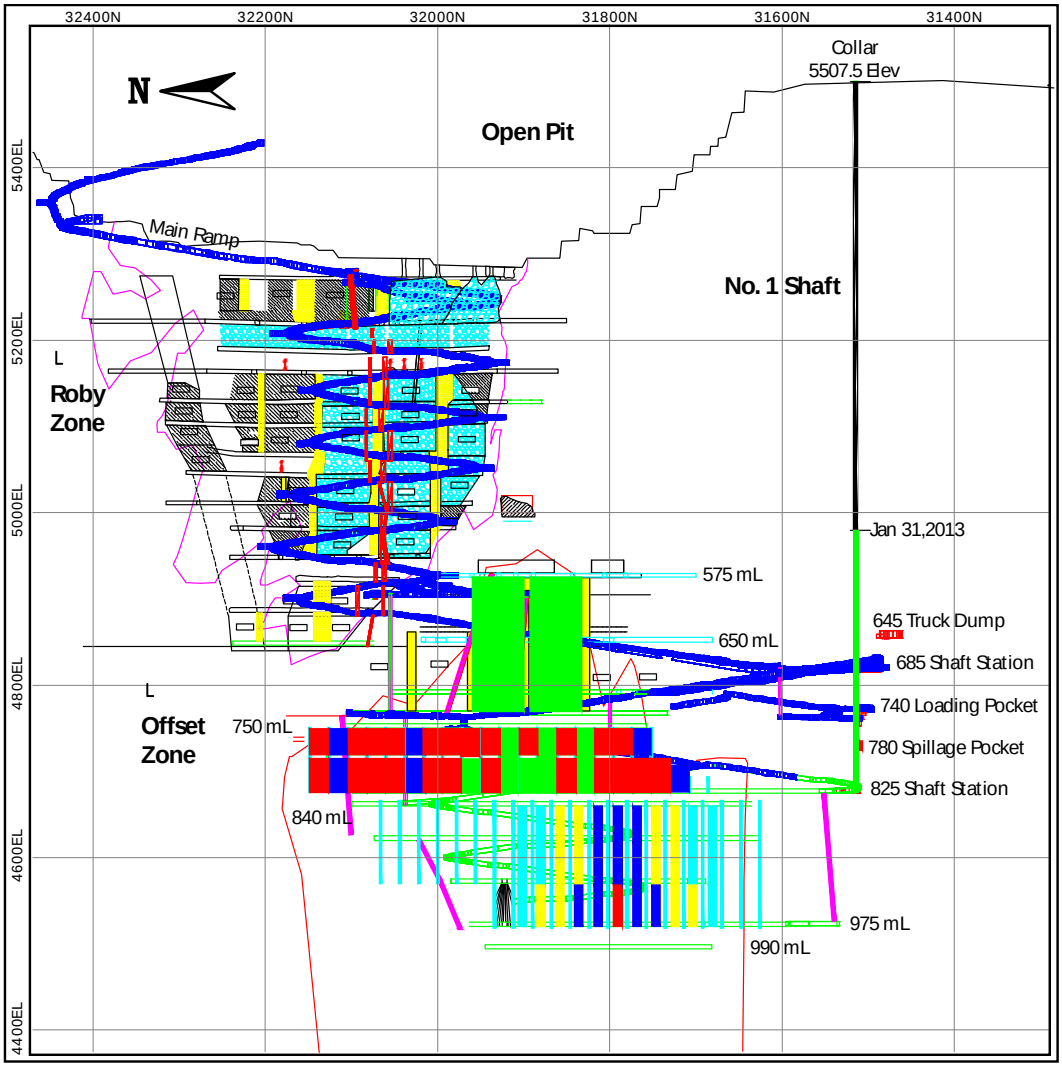
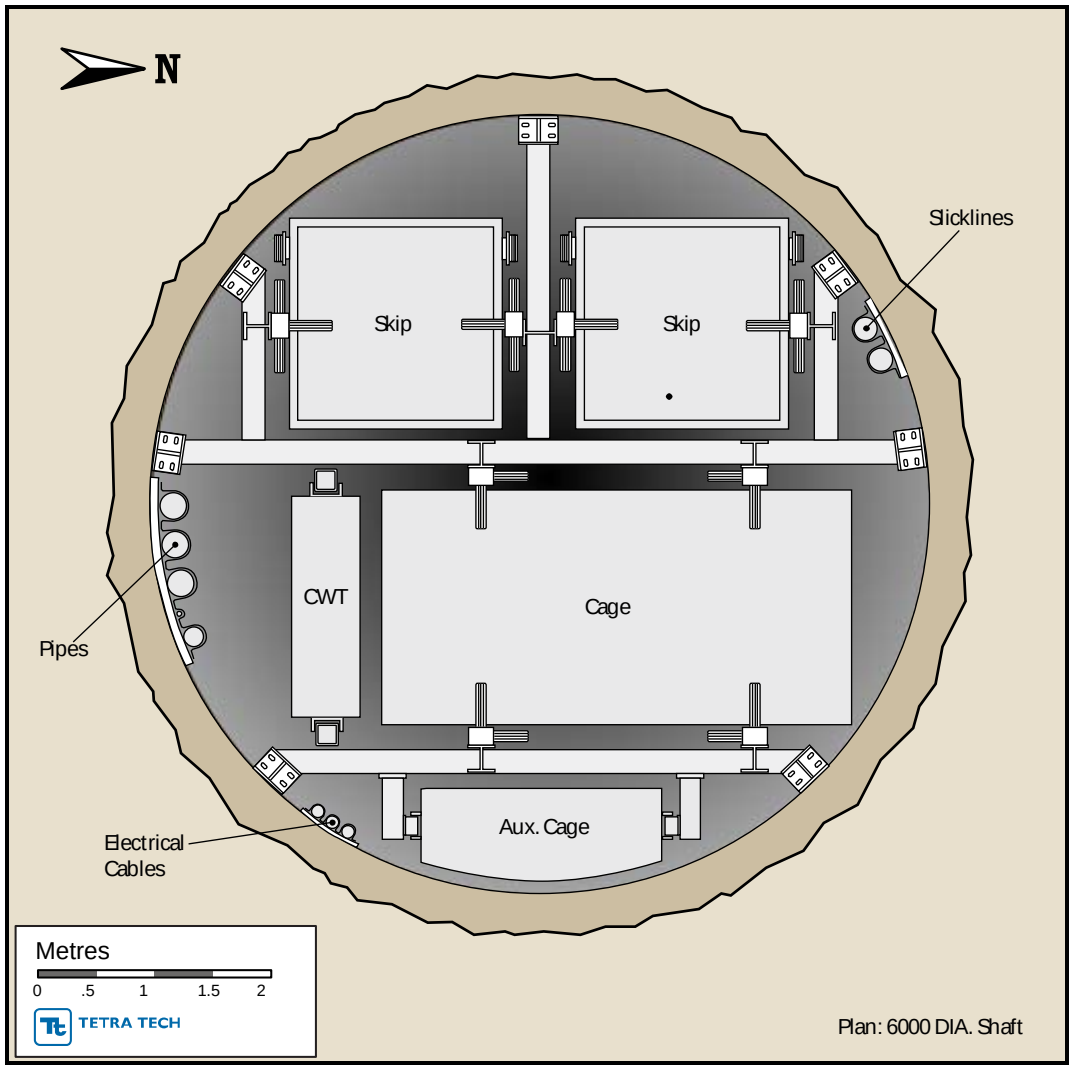


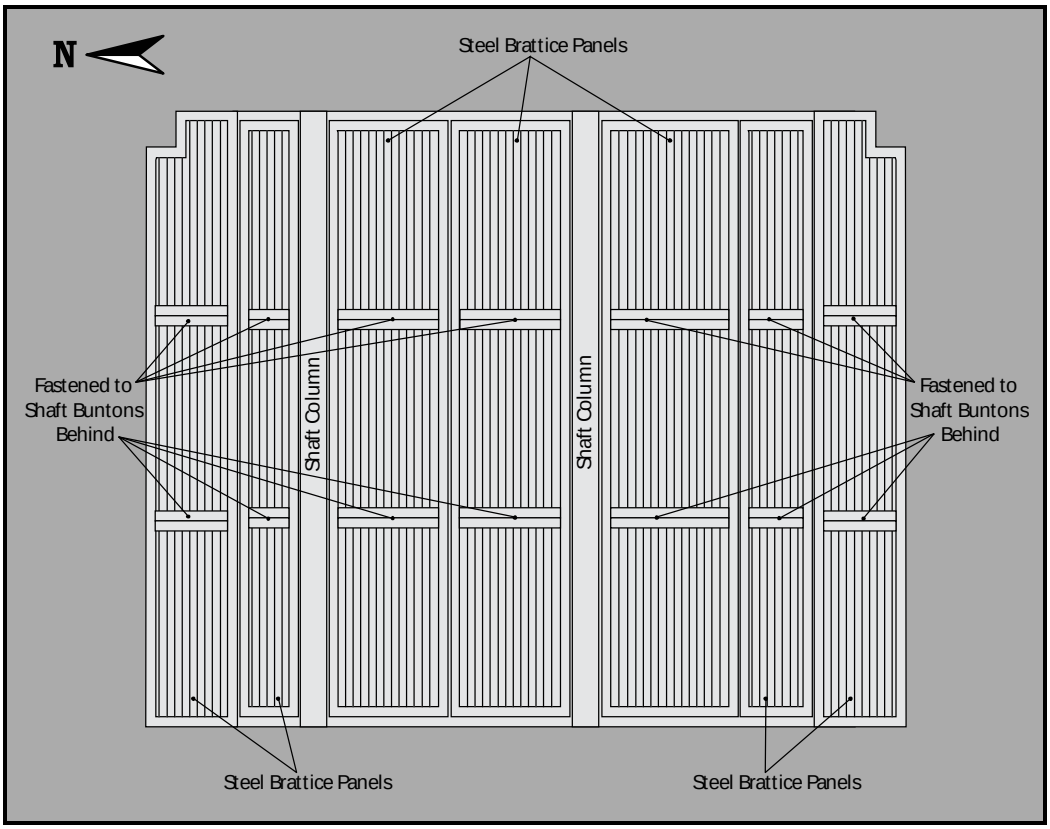
Figure 16.2 No. 1 Shaft Configuration



16.3.3 SHAFT STEEL SETS AND GUIDES

Main dividers will be constructed of hollow structural sections (HSS) 203 mm by 203 mm by 9.5 mm and buttons HSS 152 mm by 152 mm by 9.5 mm and bolted together using structural steel bolts. The skipping compartment will have steel brattice from collar to shaft bottom as shown in Figure 16.3. The remaining compartments will be open to permit visual inspection and regular maintenance.

Figure 16.3 Typical Sectional View, No. 1 Shaft Steel Brattice



Both the cage and skips will run on wooden guides 190 mm by 140 mm in size. The counterweight will run on steel guides, HSS 178 by 178 by 12.7 in size. The auxiliary cage will run on 102 mm by 140 mm wooden guides.

16.3.4 SHAFT SERVICES

The following shaft services will be installed in a services shaft compartment to support ongoing production activities in the Offset Zone Phase I ore body.

- piping
  - one 200 mm diameter compressed air line
  - two 200 mm diameter dewatering lines
  - two 100 mm diameter process water lines
  - one 50 mm dry fuel delivery line
  - two 150 mm concrete delivery slick lines
- electrical, communications and process control:
  - two 500 MCM, 13.8 kVA power cables
  - one shaft leaky feeder cable

- one communication bracket – shaft signals, fibre optics line, and telephone.

16.3.5 SHAFT STATIONS

Shaft stations provide access from No. 1 Shaft to various levels underground as shown in Table 16.1.

Table 16.1 Summary of Shaft Stations No 1 Shaft

Shaft Station	Elevation of Station*	Description
Collar Top of Concrete	5,507.5 Elevation	Collar Top of Concrete
685 Level	4,815 Elevation	Level Access
740 Level	4,760 Elevation	Loading Pocket
780 Level	4,720 Elevation	Spillage Pocket
825 Level	4,675 Elevation	Main Level – Maintenance Shop, Supply Handling Level

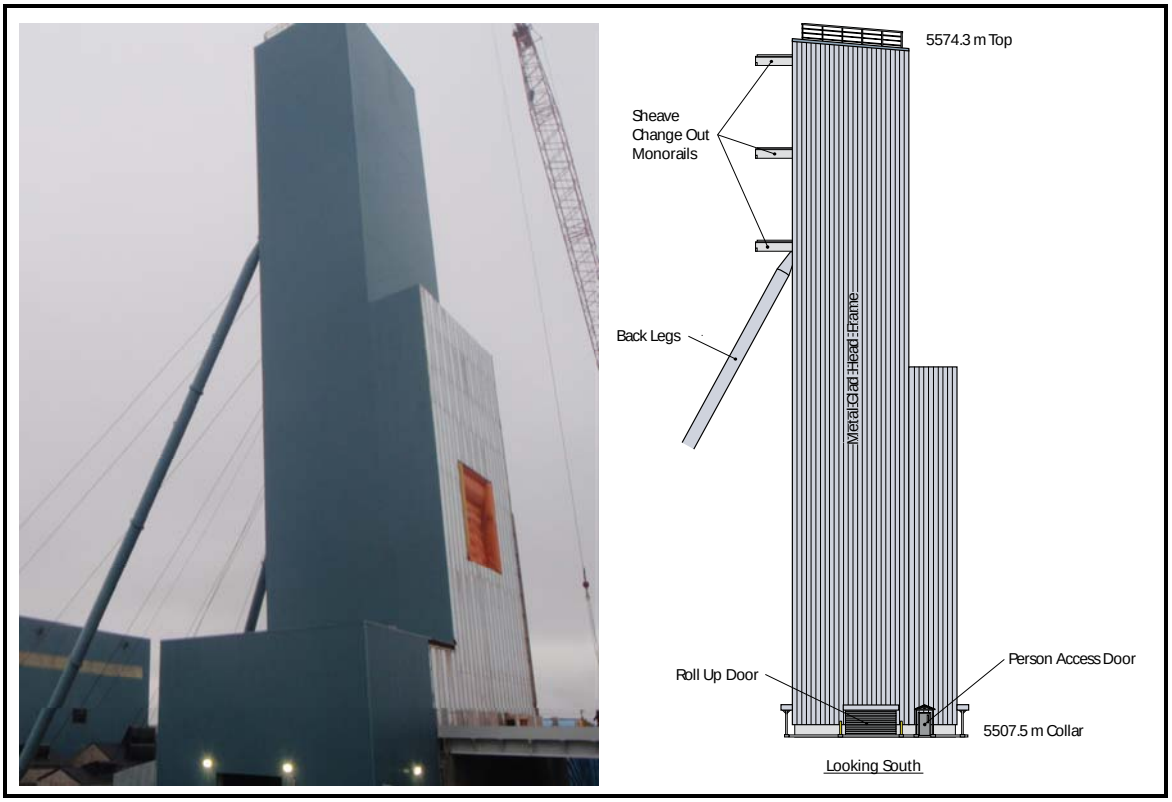
Note: *Elevations decrease when the deeper levels are established, eventually reaching a zero elevation 5,507.5 m below the shaft collar.

16.3.6 HEADFRAME

The headframe, shown in Figure 16.4, will be a structural steel construction with a clad/insulated exterior. Stairs inside the headframe will provide access to the various levels of the headframe. The headframe will include three sheave decks in order to facilitate the three sets of hoist ropes, as well as support for the shaft sinking phase. The top sheave deck will house the production hoist sheaves, the second will house the service/sinking hoist sheaves, and the lowest deck will house the auxiliary hoist sheave.

The sub-collar will be constructed to support skip changing and ventilation air routing, and will transfer loads from the headframe to the shaft collar. The collar house will be constructed at the base of the headframe and top of the sub-collar to facilitate personnel, equipment, and materials entering the shaft. The collar house will be equipped with a set of large overhead doors to facilitate transport truck or concrete truck access. A 20 t overhead crane will be included in the collar house to lift loads. The shaft collar will be equipped with full-height access doors and gates to allow for the rapid removal of large items or the removal of any other conveyance. The section of rigid guide extending from the collar to the third set in the headframe will be constructed as a “swing guide” installation, again, to facilitate conveyance removal. In addition, jack-knife monorails will be provided for each shaft compartment to support conveyance removal operations.

Figure 16.4 No. 1 Shaft Headframe



16.3.7 SHAFT SINKING

SLASHING SURFACE TO 825 LEVEL

The 825 m of production shaft will be developed using a pilot raise and slash method to permit high-efficiency sinking to the full shaft diameter. Slashing rates are faster than full face sinking which shortens the construction schedule. To further optimize the shaft sinking schedule, a pilot raise using an Alimak raise climber will be excavated from the 825 Level to within 3 m of the 685 Level's station. The 3 m pillar will be left in place so that the loading and spill pockets can be developed in parallel with further shaft development above the 685 Level's station.

When the shaft has reached the mid-shaft spill pocket, a deflection-style sinking bulkhead and spill pocket will be installed to permit concurrent shaft sinking and skipping operations to occur.

To date, the shaft work completed comprises:

- construction of the hoist house
- installation of all hoists and commissioned the service and auxiliary hoist

- construction of the headframe
- No. 1 Shaft sunk to 533 m below collar
- excavation of the 685 Level and 740 Level shaft stations.

16.4 GEOTECHNICAL

16.4.1 BASIS OF INVESTIGATION

The rock mechanics assessment for the Offset Zone has been based on site investigations including core logging, outcrop mapping, and logging of oriented geotechnical core holes. Rock mass quality values have been determined and used to establish design criteria and recommendations for the following aspects of the mine design that include:

- stability assessment of crown pillars
- ground support requirements for ramp and development headings
- stope dimensions
- stability of stope hanging walls and dilution potential
- appropriate mining methods and sequences.

Please refer to Appendix E for detailed geotechnical reports prepared by MDE.

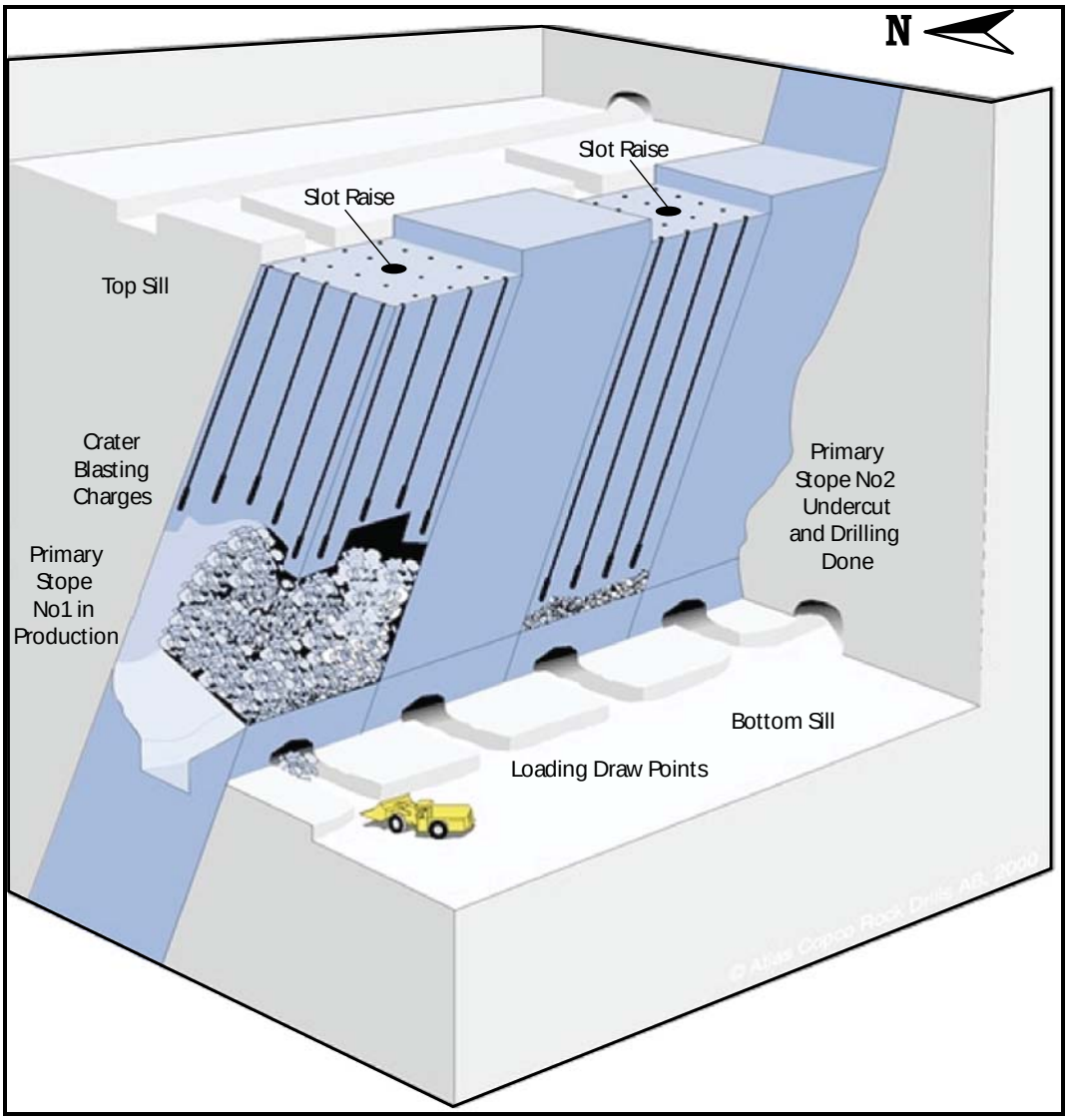
16.4.2 ROCK MASS QUALITY

Based on the assessment of rock mass quality and the engineering geological model, it was determined that the rock mass properties do not vary by rock type, but tend to be the same within the rock type regardless of the mineralized zone.

16.4.3 STOPE DESIGN

Underground mining will be carried out using longhole mining techniques. Longhole mining with primary and secondary transverse stopes, as shown in Figure 16.5, will be used for the longitudinal longhole retreat for both north and south extents.

Figure 16.5 Isometric View of the Longhole Mining Method



Assessments of the potential longhole stope dimensions were carried out using the Mathew/Potvin stability method with both deterministic and probabilistic approaches. Based on these assessments, Table 16.2 outlines the proposed stope dimensions.

Table 16.2 Stope Dimensions

	Height (m)	Width (m)	Length (m)
Primary Stope Block	45	22	35
Secondary Stope Block	45	22	35
Uppers Stope Block	25	22	35

Using these dimensions the following stope design criteria were established:

- A sublevel interval of 40 to 50 m was selected for larger stoping block tonnages. This means that the effective stope heights will be approximately 45 m including 5 m for the overcut.
- The width (back span) of primary and secondary stopes will be 35 m. Additional support will be applied in the back of the secondary stopes.

16.4.4 IN SITU STRESS

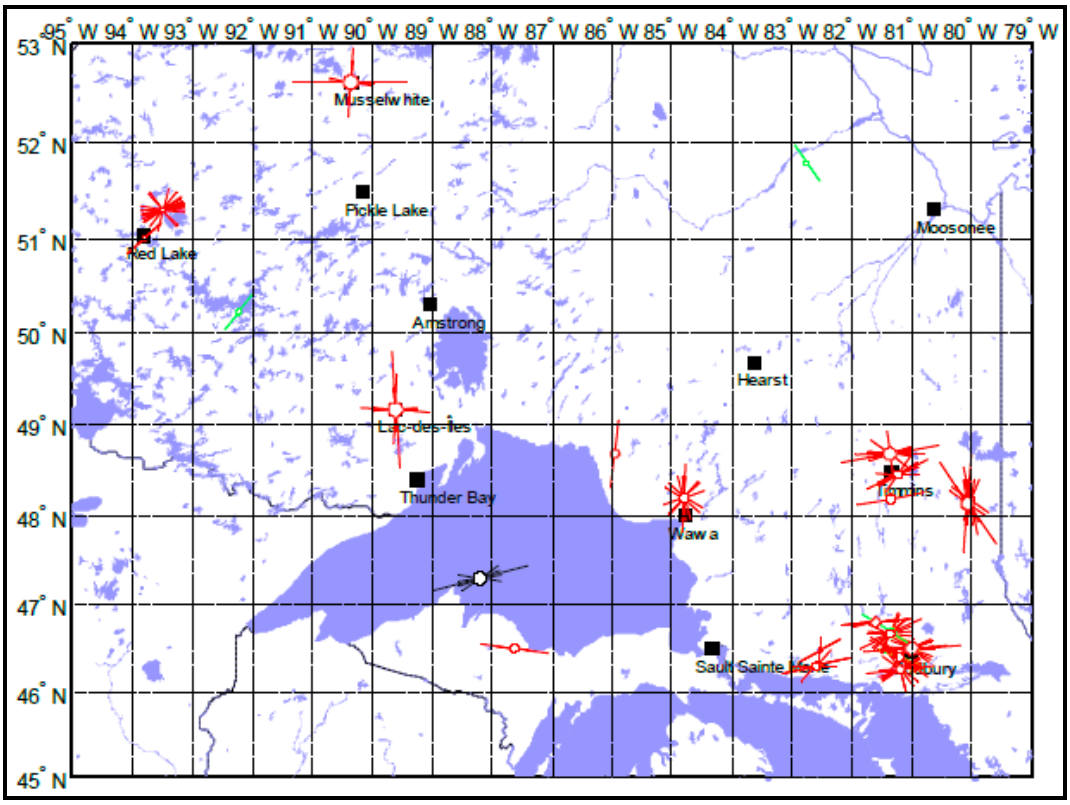
The Geomechanics Research Centre (GRC) at Laurentian University was contracted to undertake in situ stress measurements. Table 16.3 summarizes the stress fields from testing performed on the 625 Level.

Table 16.3 In Situ Stresses

Principle Stress	Magnitude (Mpa)	Orientation
Major S1	30 ± 7	356/16
Intermediate,S2	22 ±	102/43
Minor, S3	16 ± 3	251/42

As well, Figure 16.6 shows orientation of the principle stress (S1) from Lac des Iles Mine with other locations shown for northern Ontario. The results obtained at Lac des Iles Mine are in agreement with trends generally measured in the Canadian Shield.

Figure 16.6 Orientation of S1 Northern Ontario



Source: Adams (1995) database

16.4.5 NUMERICAL MODELLING

Three-dimensional numerical modelling was carried out to assess the mining sequence and stress conditions around the stopes in the mineralized zones. In general, the results did not identify any areas that will be adversely affected by the proposed mining sequence in the upper Phase I portion of the Offset Zone.

As the ore zone increases to depth, the magnitude of the principal stresses can be expected to increase beyond those determined for the 625 Level. As part of the detailed planning process underway at the mine site, mining sequence and crown pillar thickness determinations continue to be refined through additional 3D analyses by Itasca. However, refinements to the mining sequence to address increased stresses is not seen to significantly impact the current production plans for the Offset Zone Phase I, as depicted in this report.

16.4.6 BACKFILL

An estimation of the backfill strength requirements was completed for primary, secondary and drift and fill openings. For both the open stoping and drift and fill mining techniques, the main recommendations are as follows:

- Between 3 and 6% content of cement material (Portland and fly ash approximately 50/50) should be used.
- Fill shall be placed as tightly as practical in order to realize the maximum support effect on the stope/drift walls.
- The minimum delay between the completion of backfilling and mining of the adjacent drift panel shall be 30 days (to be confirmed by further laboratory and field testing).

16.4.7 DILUTION

Historic average dilution for the Roby Zone is 14%; however there is variation by stope geometry and location. As with the PEA (Buck et al. 2010), dilution for the Offset Zone Phase I was estimated to be 14% grading 2.1 g/t palladium. Internal dilution was calculated using the volumetric function in Gemcom when the individual stope tonnages were evaluated. Once the internal dilution cut-off and the volumetric functions were established, this allowed automatic categorization of the stopes into ore and waste tonnes.

16.4.8 CROWN PILLAR DESIGN

Assessments of the stability of the crown pillars located on the 750 Level, 840 Level, and the 990 Level were carried out using numerical methods. Planned final crown pillar geometries are summarized in Table 16.4.

Table 16.4 Summary of Potential Ore Zone Crown Pillar Thicknesses

Level	Maximum Elevation of Stope Back (m)	Minimum Crown Pillar Thickness (m)
750	4,750*	25 (4,775 top of crown elevation)
840	4,660*	25 (4,685 top of crown elevation)

Note: * Elevations decrease when the deeper levels are established, eventually reaching a zero elevation 5,507.5 m below the shaft collar.

Based on the results of the stability analyses, practical considerations regarding mining of the crown pillars have been identified as follows:

MAIN ORE ZONE CROWN

- Crown pillar rock quality and thickness shall be confirmed by drilling prior to recovery of the crown.
- Crown pillar stability shall be monitored, as stopes are mined above the 750 Level and 840 Level.

- The recovery of crown pillar stopes would be by up-hole retreat mining methods.

Tetra Tech recommends that basic engineering of the crown pillars be completed as part of the detailed engineering during project construction. Any future pillar recovery plans for the 750 Level and 840 Level crown pillars may require further detailed 3D numerical modelling to see if early crown extraction can be scheduled concurrently during primary and secondary stope extraction, and would need to be carefully integrated into the overall project plan.

16.5 STOPING

16.5.1 MINING METHOD DESIGN PARAMETERS

The mining method selected is a longhole stoping with cemented rock fill using a primary and secondary stoping sequence. During the 2010 updated mine expansion study, two major modifications were introduced to improve the Offset Zone Project economics: the addition of longhole stoping on a primary and secondary sequence, and mining of the north and south limbs using longhole stoping on a retreating sequence.

16.5.2 LONGHOLE MINING DESIGN BASIS – PRIMARY AND SECONDARY SEQUENCE

Based on practical mining considerations, the following design basis was established for the longhole stoping mining method:

- The sub-level interval will be between 40 and 50 m in height.
- A transverse stoping layout will be used with primary and secondary stopes.
- The width along strike for both primary and secondary stopes will be 22 m.
- Each stope will be accessed by a single 5 m by 5 m draw point driven on the centreline of the stope.
- Remote load-haul-dump (LHD) vehicles will be used to muck all ore within the stope that is beyond a safe mucking location at the draw point brow.
- Stopes will be mined on a “centre out” primary-secondary sequence.
- Primary stopes will be mined and filled with a consolidated backfill (cemented rock fill) before secondary stopes are mined. Secondary stopes can be filled with unconsolidated backfill (waste rock) when there is no crown pillar beneath the stope being filled.
- Cable bolts may be installed in the hanging wall sills as needed.
- Longhole drilling will be by in-the-hole drills, drilling 101 mm (nominal) diameter holes.

- The maximum stope length will be 35 m, at which point the stope will be filled and the remainder mined as a second stage.

16.5.3 NORTH AND SOUTH LIMBS

The north and south limbs reduce to between 5 and 20 m in width. The resulting change in width and geometry requires the use of longhole stoping on a retreating sequence based on the following design basis:

- The sub-level interval will be between 40 to 50 m in height.
- A longitudinal stoping layout will be used.
- The width along strike will be between 40 to 50 m, depending on stope geometry and hanging wall exposure.
- Each stope will be accessed by a single 5 m by 5 m draw point driven on the centreline of the stope.
- Remote LHD vehicles will be used to muck all ore within the stope and beyond safe mucking at the draw point brow.
- Stopes will be mined on a retreating sequence.
- Stopes will be mined and filled with a combination of consolidated rock fill and loose rock fill (dependent on whether crown pillar is beneath the stope).
- Cable bolts will be installed in the drill horizon as required.
- Longhole drilling will be by in-the-hole drills drilling 101 mm (nominal) diameter holes.
- Slot raises will be excavated by drop-raising or blasting against the adjoining filled stope negating the need for the 2.5 m diameter pilot raise.

16.5.4 CROWN RECOVERY – 750 LEVEL AND 840 LEVEL

At the end of mine life, uphole mining will be used for the final recovery of the 750 Level and 840 Level crown pillars based on the following design basis:

- The crown height will be 25 m and drilled with upholes.
- A transverse stoping layout will be used.
- The width along strike will be between 40 to 50 m, depending on stope geometry and hanging wall exposure.
- Each stope will be accessed by a single 5 m by 5 m draw point driven on the centreline of the stope.
- Remote LHD vehicles will be used to muck all ore within the stope and beyond safe mucking at the draw point brow.
- Stopes will be mined on a stope and pillar sequence.

- Stopes will be mined and filled with loose rock fill.
- Longhole drilling will be by top hammer machines drilling 101 mm (nominal) diameter holes.
- Uppers slots will be blind slot raises.

Further assessment will be required before a definitive crown pillar-mining plan can be adopted.

16.6 DEVELOPMENT

16.6.1 ACCESS RAMP

The mine will extend to a depth of 990 m below surface in the Offset Zone Phase I deposit, and will still be open along strike and at depth. Primary access to the mine from surface will be the No. 1 Shaft and the secondary means of egress will be via a 15% ramp from the 990 Level to surface. A 3D view of the shaft and ramp is shown in Figure 16.7, while a typical level plan for the 825 Level is shown in Figure 16.8.

Figure 16.7 3D View of the Mine Ramps and Levels

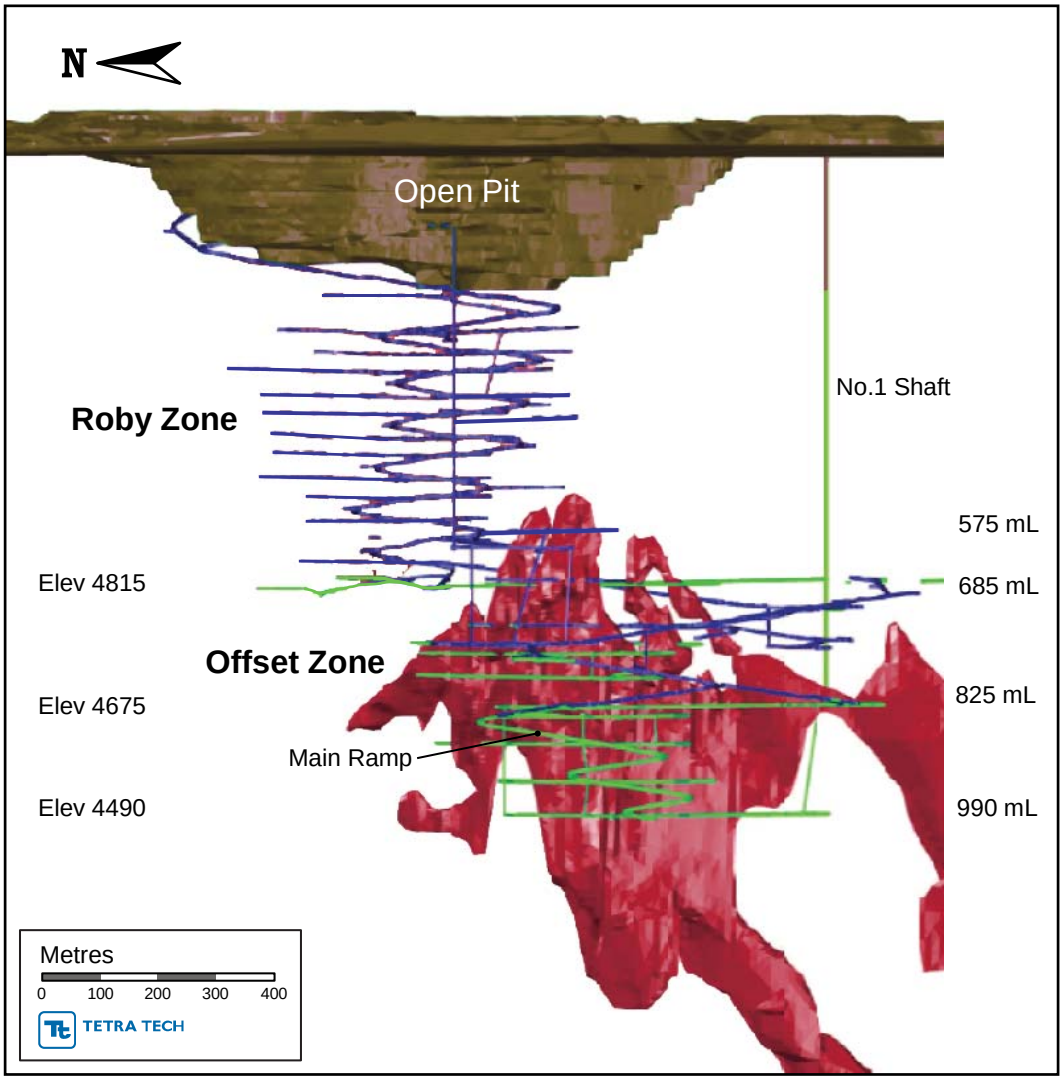
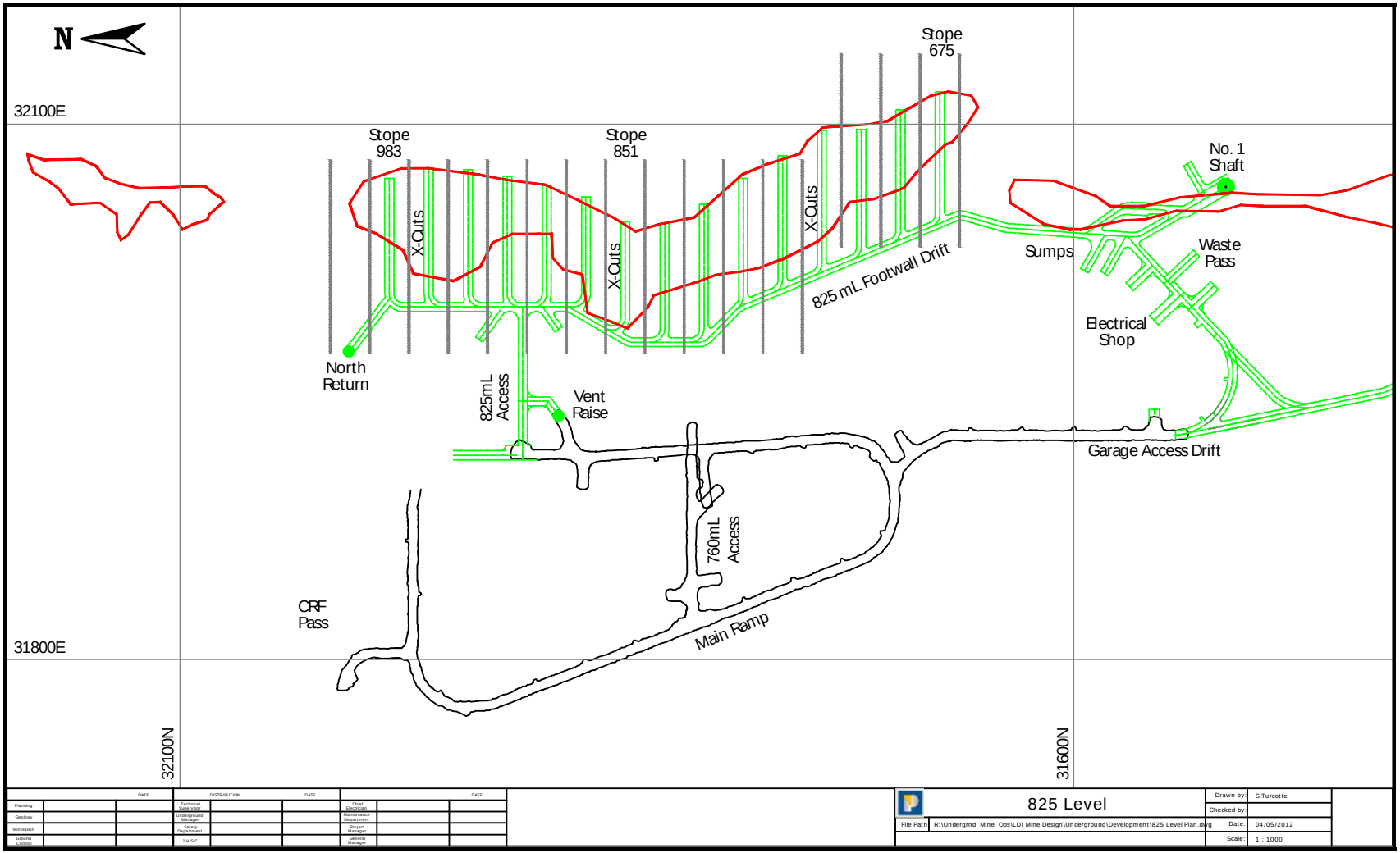


Figure 16.8 825 Level – Typical Level Plan



The main ramp will be 5 m wide by 5 m high, sized to accommodate 50 t trucks and 6.1 m^3 LHD units. The back profile will be arched and standard ground support will consist of a 2.1 m long resin grouted rebar in the back, and a 1.8 m long resin grouted rebar in the walls, all on a 1.2 m by 1.5 m pattern. In addition to the rebar, a #6 galvanized wire mesh screen will be installed throughout the ramp. Major intersections of the ramp and levels, with effective spans in excess of 11 m, will be further supported by a 2.4 m resin grouted rebar on a 1.2 m by 1.5 m pattern. Seven-metre long grouted cable bolts will also be installed in the back on a 1.2 m by 1.5 m pattern over the complete intersection area.

The design criteria for the main ramp are summarized in Table 16.5.

Table 16.5 Main Ramp Design Criteria

Parameter	Criterion
Dimensions	5.0 m wide by 5.0 m high with arched back
Curve Radius	27 m
Grade (General)	15% except at level take-off locations
Grade (At Level Take-offs)	4% for minimum 15 m, plus 5 m long 9% transition zones at each end
Remuck Bays*	5.0 m by 5.0 m, 15 m long and located a minimum of every 150 m
Safety Bays	0.6 m deep by 2 m high by 1.5 m long and located every 30 m

Note: * Remuck bays will be used as passing bays for vehicles in the ramp.

16.6.2 ACCESS DEVELOPMENT

Access to the deposits and mine infrastructure will be provided by levels and in some cases secondary ramps from the main ramp. The mine has been planned with 40 to 50 m vertical level intervals, which corresponds to the sub-level spacing selected for blasthole mining. Mining levels and their nominal elevations are listed in Table 16.6.

Table 16.6 Mining Levels and Elevations

Level Designation	Nominal Elevation* (m)
750	4,750
825	4,675
840	4,660
875	4,620
925	4,570
975	4,520
990	4,490

Note: * Elevations decrease when the deeper levels are established, eventually reaching a zero elevation 5507.5 metres below the shaft collar.

The level drifts will be 5 m wide by 5 m high, with an arched back profile. Standard ground support will be the same as that used for the main ramp. The design criteria for the levels and secondary ramps are summarized in Table 16.7.

Table 16.7 Design Criteria for Levels and Secondary Ramps

Parameter	Criterion
Dimensions	5.0 m wide by 5.0 m high with arched back
Curve Radius	27 m
Grade (Levels)	3% minimum
Grade (Secondary Ramps)	14% (nominal), 15% (maximum)
Grade (Level Transitions)	10% for 5 m, plus 5% for 5 m
Remuck Bays	5.0 m by 5.0 m, 15 m long and located a minimum of every 150 m
Safety Bays	0.6 m by 2.0 m, 1.5 m long and located every 30 m

16.6.3 *INFRASTRUCTURE DEVELOPMENT*

Mine infrastructure will include detonator and explosive magazines, refuge stations, electrical substations, maintenance facilities, a fuel bay, a material storage, main sump and pumping station, secondary sumps and drain-hole cut-outs, and ore remuck bays. Infrastructure development included in the mine plan is shown in Table 16.8.

Sizing for infrastructure development has been based on the design of similar facilities at the Roby Zone. Infrastructure development will receive standard ground support the same as specified for the levels. The ground support for certain facilities will be supplemented with the application of shotcrete.

Table 16.8 Infrastructure Development Summary

Description	Quantity	Location	Comments
Explosive and Detonator Magazines	2 each	825 Level	Main magazines
Refuge Stations	3	740 Level, 825 Level	-
Electrical Substations	2	825 Level	-
Maintenance Facility	1	825 Level	Main underground shop
Fuel Bay	2	825 Level	-
Material Storage Bay	2	825 Level	-
Dirty Water Sump	2	825 Level	-
Main Pump Station	2	685 Level	-
Secondary Sumps	6	All Levels	-
Ore Remuck Bays	6	All Levels	-

16.6.4 RAISES

Mine development requirements include raises for ore passes and ventilation. Separate raises will be provided for fresh air, stoping, return air, and ore passes. Some of the fresh air raises may be equipped with a man-way suitable for a third means of egress for personnel underground.

All ore passes will be driven using Alimak raise climbing equipment, supported by 1.8 m resin grouted bolts on a 1.2 m by 1.2 m pattern. Ore pass fingers will be supported by resin grouted rebar and 15 m long cable bolts, to provide additional ground support at the ore pass dumping points.

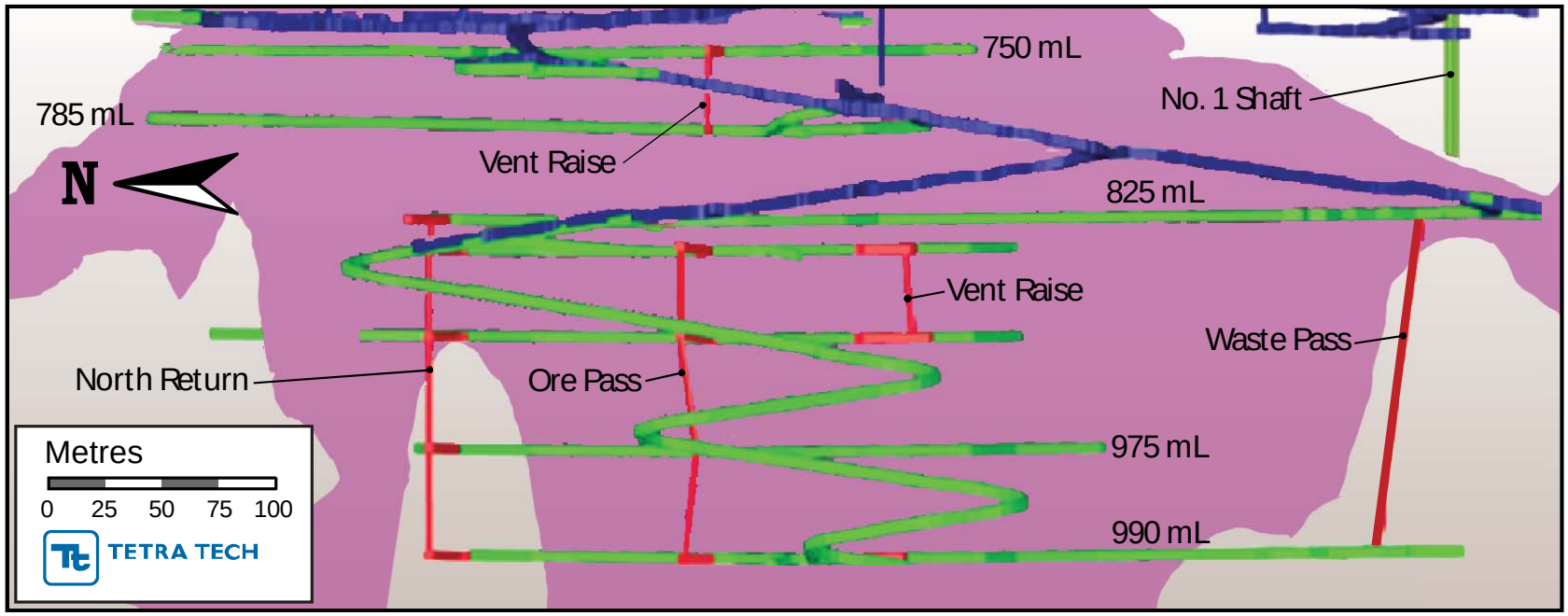
All ventilation raises will be driven using raise boring equipment. The upper section of the main exhaust raise to surface has a 5.0 m diameter circular cross-section. Lower sections of the return system will consist of a north and south return air raise at a diameter of 3.7 m. Main fresh air will be delivered via the No. 1 Shaft and transferred to main fresh air raises on the 825 Level. The fresh air raises are centrally located on the main levels and will have a diameter of 4.0 m.

The raises are depicted in red on the mine schematic in Figure 16.9. A total of 2,556 m of raises will be developed. Table 16.9 summarizes raise requirements and specifications.

Table 16.9 Mine Raise Requirements

Description	Dimensions	Dip (°)	Length (m)
No. 1 Return Air Raise (600 Level to Surface)	5.0 m diameter	90	456
990 South Return (990 Level to 750 Level)	3.7 m diameter	85	260
990 Centre Ore Pass (990 Level to 750 Level)	3.1 m by 2.4 m	85	260
990 Main Fresh Air (990 Level to 750 Level)	4.0 m diameter	85	260
990 North Return (990 Level to 750 Level)	3.7 m diameter	85	260
990 Main Waste Pass	2.4 m by 2.1 m	85	260
735 Backfill Raise (735 Level to Surface)	2.5 m diameter	90	800
		Total	**2,556**

Figure 16.9 Raise Development Looking East



16.6.5 DEVELOPMENT FORECAST BY YEAR

A summary development schedule is provided in Table 16.10. Pre-production development is the portion completed prior to the commencement of commercial production. Detailed development forecasts showing pre-production and Phase I development requirements are provided in Appendix F

A mining contractor(s) will complete all level and waste raise development during the pre-production period, while Owner's crews will be used for draw point ore development. During the production phase, Owner's crews will be responsible for all development with the exception of waste raises.

Advance rates based on actual performance for different headings for both contractor and Owner's crews are shown in Table 16.11. The advance rates represent a 5 d/wk, two 12 h/d, shift schedule and therefore, allows time to make up for any missed rounds in a given week.

Table 16.10 Summary Development Forecast by Year (m)

Description	2013	2014	2015	2016	Total
Ramp Development	2,272	-	-	-	2,272
Level Access and Foot Wall Development	1,235	1,653	-	-	2,888
Ore Sills	1,040	2,261	2,055	-	5,356
Total Lateral	4,547	3,914	2,055	-	10,516
Vent and Escape Raises	1,176	560		-	1,736
Total Development	5,723	4,474	2,055	-	12,252

Table 16.11 Advance Rate Assumptions for the Development Schedule

Type	Advance Rate		Delay Allowance	Schedule Advance Rate
	Daily	Monthly		
Ramps and Levels				
Contractor				
Single Heading (all sizes)	2.8 m/d	87 m/mo	3 d/mo	80 m/mo
Multiple Heading (all sizes) - Four Crews	6.0 m/d	470 m/mo	3 d/mo	470 m/mo
Owner's Crews				
Multiple Heading (all sizes)	5.0 m/d	-	-	150 m/mo
Alimak Raises				
Alimak Set-up (includes first 10 m)	-	-	-	2 wk
Ore Passes (3.1 m by 2.3 m)	2.5 m/d	-	-	77 m/mo
Bored Raises				
Surface Raises (5.0 m diameter)	2.5 m/d	Pilot and Ream		70 m/mo

The mining method and sequence have been designed to eliminate pillars within the Offset Zone Phase I. Consequently, mining will commence from the lowest level of each mining block in Phase I, and it will be necessary to develop the ramp in its entirety before 990 Level development and ore production can begin. As indicated in Table 16.10, 100% of the Phase I lateral waste development and 100% of the waste raise development will be completed during the next three years.

The top development priority will be the completion of the main ramp to the existing bottom of the Offset Zone Phase I on the 990 Level. One development crew will be dedicated to driving the main ramp and once there, excavating the required levels. The ramp development plan will include critical portions of the levels needed to access the ventilation raise system located adjacent to the ramp. This will limit the length of ventilation ducting required to support development activities until the permanent ventilation system is available. Critical level development will also include driving the 990 Level to the base of the ventilation and ore pass system.

The ventilation raises will be excavated upon completion of related access development to create a ventilation path extending progressively downward with the ramp development.

16.7 PRODUCTION FORECAST

16.7.1 BASIS OF PRODUCTION FORECAST

The production forecast was created based on the criteria depicted in Table 16.12 and the following operating philosophy:

- Blasthole stopes will be mined in a primary and secondary sequence with production commencing from two horizons. Primary stopes at the bottom of each horizon and centre of each zone will be scheduled first, while subsequent primary stopes will be sequenced in an upward and outward manner or inverted “V” shape. Secondary stopes will also be scheduled in an inverted “V” configuration after the primary stopes have been mined and filled on either side, plus one level above.
- Production from the north and south limbs will be scheduled in a longitudinal retreat, starting from furthest stoping block and retreating back towards the main footwall access.
- Mine production will be ramped up to the full design rate as quickly as practical, following completion of mine development and construction.
- Production from the two main mining zones will be scheduled relatively uniformly over the Phase I life-cycle to maximize the stope extraction at the forecast mining rate.

Table 16.12 Production Forecast Criteria

Parameter	Value	Comments
General		
Target production rate	3,500 t/d	-
Operating days	365 d/a	-
Annual target production rate	1,260,000 t/a	-
Shift schedule	Two 12 hour shifts/day	-
Longhole Mining		
Longhole drilling	20 t/m (6.2 t/ft)	101 mm (4 in) diameter
Blasthole stope mucking production	70 t/h (2,400 t/d)	-
Build backfill barrier	7 d	-
Fill stope (cemented rockfill)	20 d	-
Fill set time before mining beside stope	28 d	-

16.7.2 PRODUCTION FORECAST

A summary of the mine production forecast by mining zone is shown in Table 16.13. Full annual production (1,260,000 t/a) will be achieved at the start of 2014. A detailed production forecast by zone has been created by month for the years 2013 and 2014 and by year for 2015 to 2017.

The annual mining schedule by stope is illustrated in longitudinal projection in Figure 16.10, which illustrates the mining sequence by depicting the extent of stope mining at the end of each project year.

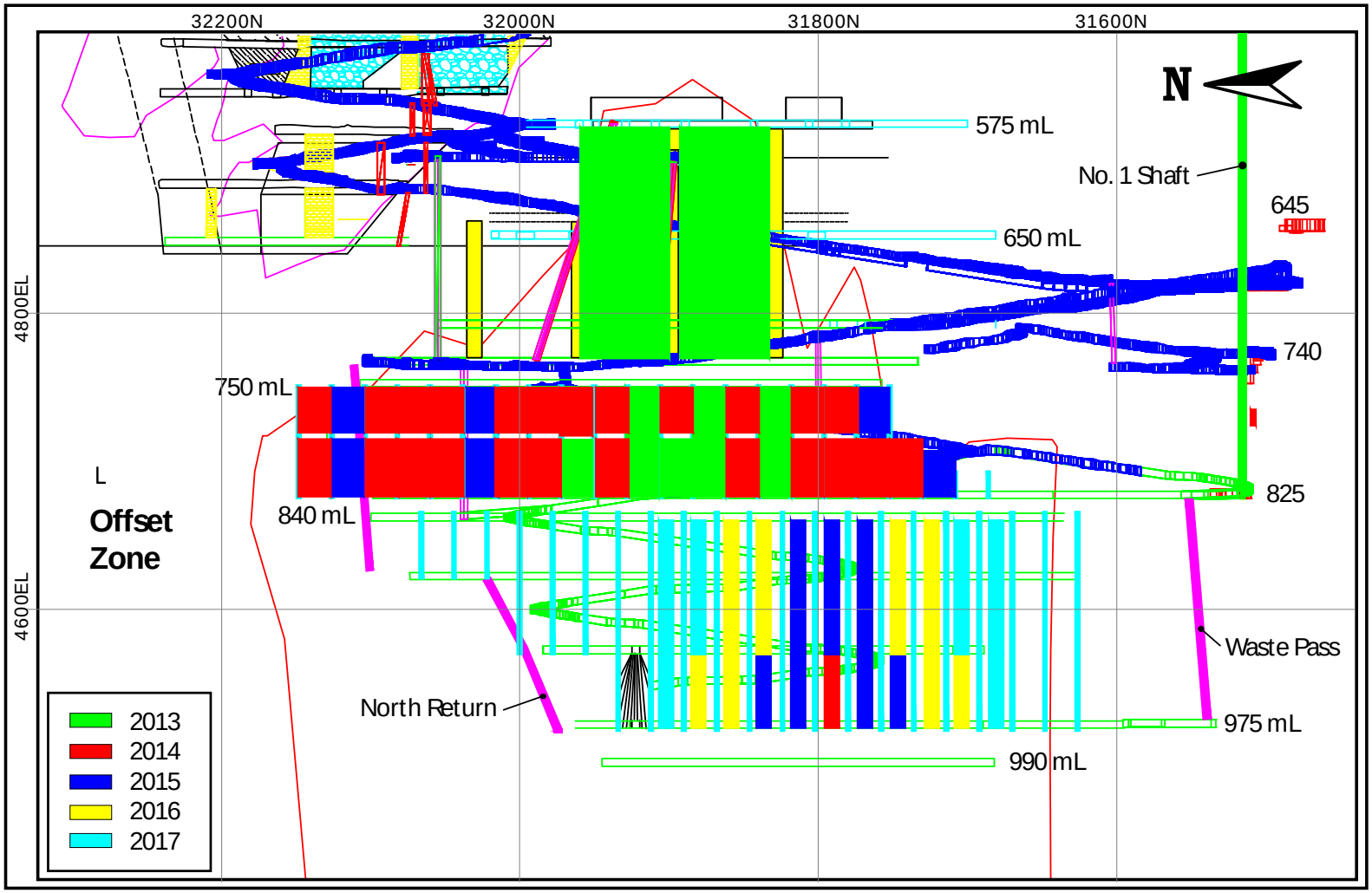
The schedule provides for the already developed stopes above the 740 Level to immediately commence production in 2013 (Year 0). Full production is achieved in 2014 (Year 1) once multiple mining and filling faces are available between the 825 Level and the 760 Level. A schedule illustrating the month-by-month build-up to full production is provided in Table 16.13. Prior to reaching full production in 2014, a total of 1.6 Mt will have been mined from the Offset Zone. The annual production rate for the Offset Zone Phase I ramps up from 1.2 Mt in 2013, 1.4 Mt in 2014, 1.6 Mt in 2015 and with completion of the ore sill development, the annual production rate from the Offset Zone above the 990 Level will level off at 1.2 Mt in 2016 to 2018 with sustained production at 3,500 t/d (105,000 t/mo) being achieved.

Over the life of Phase I, 95% of production will come from blasthole stoping, and 5% from ore development. All development ore will be sourced from the mining of draw points for the blasthole stopes.

Table 16.13 Yearly Build-up to Full Production

	Year						
	2013	**2014**	**2015**	**2016**	**2017**	**2018**	**Total**
Offset Zone, Phase I (t/d)	3,300	4,000	3900	3,452	3,452	3,112	-
Total Phase I ('000 t/a)	**1,208**	**1,447**	**1,430**	**1,260**	**1,260**	**1,136**	**7,741**
Full Production (%)	96	100	100	100	100	90	-
Palladium (g/t)	4.62	4.24	4.33	4.29	4.29	4.00	4.30
Platinum (g/t)	0.29	0.29	0.31	0.31	0.31	0.35	0.30
Gold (g/t)	0.28	0.28	0.31	0.33	0.33	0.24	0.30
Copper (%)	0.08	0.08	0.09	0.10	0.10	0.03	0.08
Nickel (%)	0.11	0.11	0.11	0.12	0.12	0.09	0.11

Figure 16.10 Offset Phase I Production Long Section



16.8 BACKFILL

16.8.1 BACKFILL SYSTEM SELECTION

Backfilling of mined out areas will be required to support blasthole stoping.

Blasthole stopes will be filled immediately following ore extraction to minimize wall sloughing and general rock deterioration. Blasthole stopes will be mined on an alternating primary and secondary basis. Primary stopes will be mined first then filled with cemented rock fill to minimize backfill dilution during the mining of adjacent secondary stopes. Secondary stopes can be filled with unconsolidated backfill or development waste as available, as long as there is no crown pillar immediately below the empty blasthole stope.

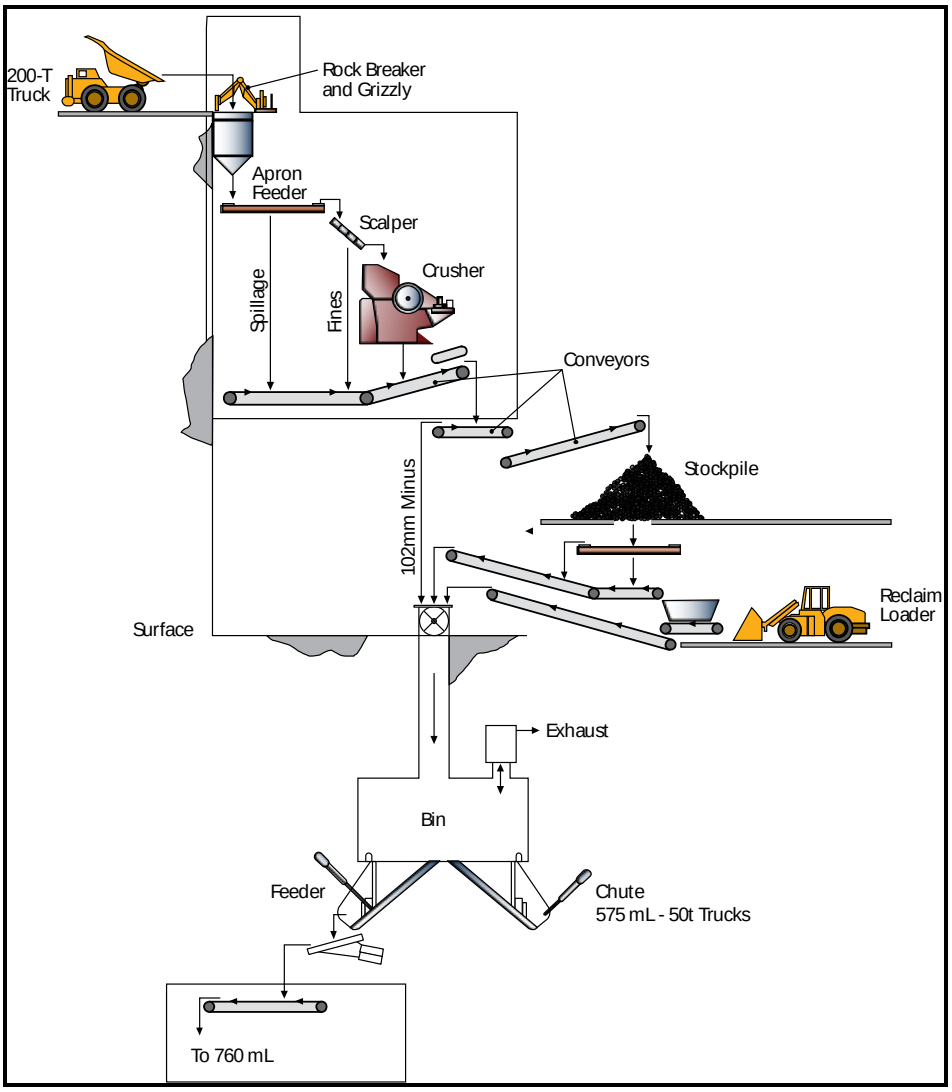
CEMENTED ROCK FILL

Cemented rock fill can achieve excellent strength characteristics, well exceeding the requirements of the mining methods proposed for the Offset Zone Phase I deposit.

Mine development will generate an inexpensive supply of waste rock. However, available quantities will not be sufficient to meet the mine's total backfill requirement. The use of surface waste rock from the previous open pit can be utilized to meet the development waste deficit and would dramatically reduce the cost for this type of fill.

The use of cemented rock fill was selected because of the low capital costs for a plant and distribution system as depicted in Figure 16.11 and the abundance of underground and surface waste rock availability.

Figure 16.11 Backfill Process Flow Diagram



16.8.2 BACKFILL SCHEDULE

Primary blasthole stopes will be filled with cemented rock fill while secondary blasthole stopes will be filled with development waste rock (if no underground waste rock is available, surface waste stockpiles will be used). Waste rock will be reclaimed and sent back underground using the backfill system.

A detailed schedule of backfill requirements over the mine life is summarized in Table 16.14.

Table 16.14 Backfill Forecast by Year

Type	2013	2014	2015	2016	2017	2018	Total
Backfill ('000 m^3)							
CRF	0	440	440	440	440	398	2,158
Waste Rock	0	440	440	440	440	398	2,158
Total	**0**	**880**	**880**	**880**	**880**		**4,316**
Binder m^3 – 4% by Weight of Rock Fill							
Binder	0	18	18	18	18	16	88

The average required binder is based on a blend of 20% T10 CSL (cement) and 80% slag. Test work indicates that it may be possible to reduce the binder content below 3% and this opportunity should be explored during mine operations.

16.8.3 BACKFILL SOURCING

Sourcing for backfill will come from underground generated development waste rock and from the existing surface stockpile of open pit waste rock, in excess of 10 Mt. The quantity of backfill required will be less than the mined tonnage due to the swell factor of the replacement waste, cement content of the cemented rock fill and the internal voids associated with cemented rock fill. Table 16.15 outlines the backfill sources required by year. In 2013, approximately 383,000 t of development waste from the Offset Zone Phase I will be being routed to backfill empty Roby Zone stopes. Backfilling for the Offset Zone Phase I stopes will not be required until the 2014.

Table 16.15 Backfill Forecast by Year ('000 t)

	2013	2014	2015	2016	2017	2018	Total
Backfill Required	0	880	880	880	880	880	**4,316**
Development Waste	383	159	0	0	0		**537**
Surface Waste	0	721	880	880	880	880	**4,241**

16.9 ORE AND WASTE MANAGEMENT

16.9.1 ORE HAULAGE

Ore will be mucked by LHDs and dumped down one of three ore passes. The ore production fleet will consist of two 11 yd^3 LHD units for level tram to the ore pass. With the majority of production from below the 685 Level crusher location, ramp haulage of the ore will be required. As shown in Table 16.16, the LHD fleet will be complemented by as many as five 45 t and 50 t sized haul trucks to move material to

the 645 Level truck dump location that feeds the 685 Level crusher and thereby the mid-shaft loading pocket for hoisting to the surface load out. Alternatively the trucks can haul the ore up ramp to the surface re-handle pad at the bottom of the pit where it is then loaded into the open pit haul trucks.

Table 16.16 Summary of Ore Trucking and LHD Requirements by Year

Description	Production Year						
	2013	2014	2015	2016	2017	2018	Total
Ore Haul ('000 t)	1,208	1,447	1,430	1,260	1,260	1,136	**7,741**
Trucking (50 t trucks)							
Trucks Required	4	4	5	5	5	5	**-**
LHDs (11 yd^3) – Production Only							
LHDs Required	2	2	2	2	2	2	**-**

16.9.2 *WASTE HAULAGE*

Additional to the ore haulage fleet, the development fleet will comprise one 6.1 m^3 LHD, one 3.5 yd^3 LHD and two 35 t trucks for remucking development ore and waste to the remuck locations. All longer haul development waste requirements will be met by 45 t and 50 t trucks, either directly to a secondary blasthole for use as backfill, or to mined out stoping blocks in the Roby Zone. The waste production and associated equipment schedules are shown in Table 16.17 and Table 16.18, respectively.

Table 16.17 Waste Production and Haulage Schedule by Year

Description	2013	2014	2015	2016	2017	2018	Total
Total Development Waste ('000 t)	383	159			-	-	**542**
Haulage Backfill (to Roby Zone)	383	-	-	-	-	-	**383**
Waste Hoisted to Surface Stock Pile	-	-	-	-	-	-	**0**
Waste Haulage Backfill (Offset)	0	281	440	440	440	440	**2,041**

Table 16.18 Summary of Waste Trucking and Development LHD Requirements by Year

Description	Production Year					
	2013	2014	2015	2016	2017	2018
Trucking (35 t trucks)						
Trucks Required	3.0	3.0	3.0	3.0	3.0	3.0
LHDs (6.1 m^3 and 3.5 yd^3) – Development Only						
LHDs Required	2.0	2.0	2.0	2.0	2.0	2.0

16.10 PRODUCTION SUPPORT ACTIVITIES

In addition to the development, backfilling, and trucking requirements discussed above, there are a number of other activities required to support production. Those activities are summarized in Table 16.19.

The diamond drilling indicated in Table 16.19 is stope definition drilling and was estimated on the basis of preliminary layouts of the diamond drillholes and a detailed examination of the plans and sections. An allowance has also been included for exploration drilling.

Cable bolting was estimated on the basis of 12 m of cable bolting per metre of development. This factor is based on data from similar bulk mines and generally better ground conditions within the Offset Zone Phase I ore body. This development factor was derived from an estimate of the number of cable bolts required to support a typical intersection and the expected number of intersections that will require cable bolting.

Slot raise length was estimated by allowing for one slot raise per blasthole stope. Longhole drilling is based on 18 t of ore per metre of drilling for blasthole production. This factor includes all drilling required for slot as well as 10% re-drilling. No allowance was made in the drill factor for dilution.

Table 16.19 Activity Schedule by Year

Description	2013	2014	2015	2016	2017	2018	Total
Diamond Drilling (m)	84,400	47,800	0	0	0	-	**132,200**
Cable Bolting (m)*	54,635	53,700	24,660	0	0	-	**132,995**
Shotcrete (t)*	6,000	3,000	1,000	1,000	500	-	**11,500**
Longhole Drilling (m)	6,889	100,800	63,000	63,000	63,000	63,000	**359,689**

Note: * For production support only – does not include development activity which is included with development.

16.10.1 ORE HANDLING SYSTEM

The ore handling system consists of a small jaw crusher, grizzly, rock breakers and ore/waste remuck dump locations. The crusher is being installed on the 685 Level and allows pre-production and production ore and waste to be hoisted to surface utilizing the shaft. The ore handling system consists of a portable 864 mm by 1,118 mm crusher, fed from a 2.4 m by 2.4 m raise, which extends 20 m above the crusher station to the 645 Level. On the 645 Level, ore will be sized using a typical grizzly and rock breaker arrangement as depicted in Figure 16.12 and Figure 16.13. The waste handling system will also feed the No. 1 Shaft loading pocket directly via a 2.4 m by 2.4 m waste pass parallel to the ore raise. Waste material will be sized prior to entering the waste pass using a grizzly and rock breaker at the 645 Level. Unless the No. 1 Shaft is deepened and a lower pocket installed, then the ore

handling system described above would remain in place for any future resource growth in the upper Offset Zone as well as neighbouring resource targets such as the Sherriff Zone and southern extensions to the Roby Zone.

Figure 16.12 Plan View 645 Level Truck Dump and Rock Breaker

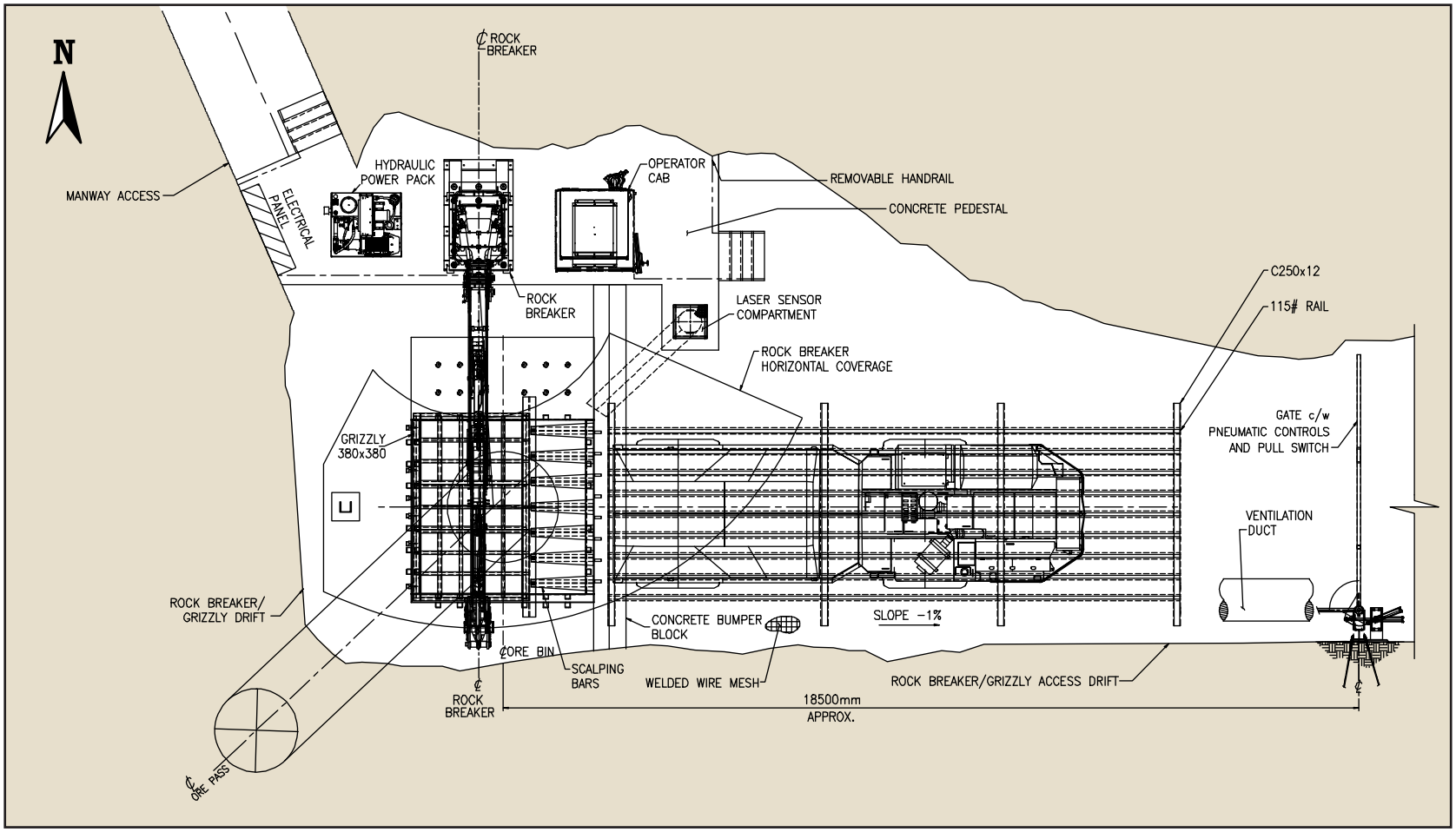
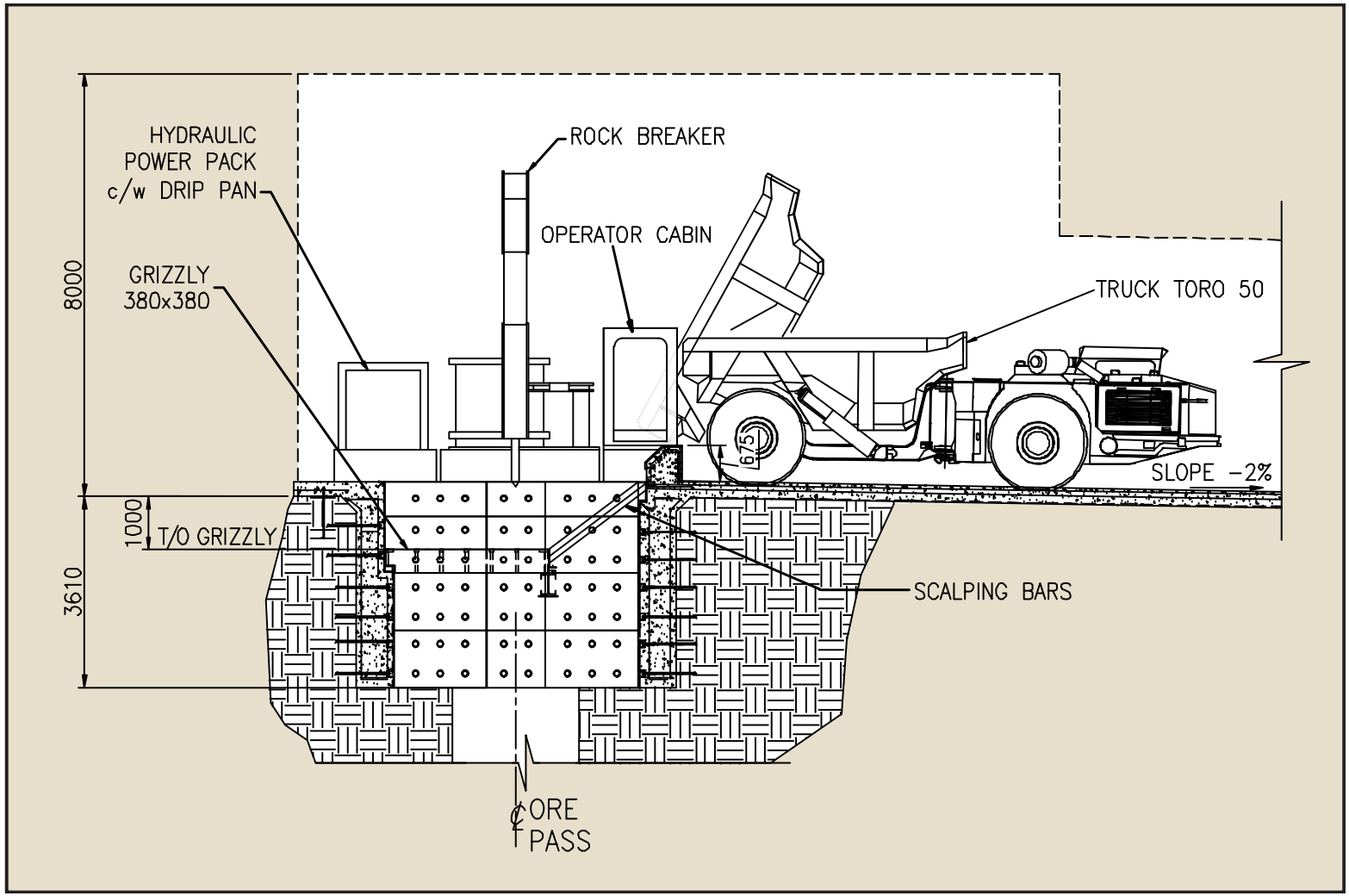


Figure 16.13 Sectional View 645 Level Truck Dump and Rock Breaker



16.10.2 MINE DEWATERING

Water entering the mine will consist of industrial water, mine water inflows, and water entrained in cemented rock fill. The supply of industrial water will average 962 m^3/d. Water in the cemented rock fill will be hydrated by cement, and it has therefore been assumed that water release will be minimal. Based on the existing pumping, a mine water inflow of 400 m^3/d has been estimated. The dewatering requirement for the mine has been estimated as follows:

- industrial water: 962 m^3/d
- mine water inflow: 400 m^3/d
- emergency water storage: 200 m^3/d
- pumping requirement: 1,562 m^3/d.

SHAFT BOTTOM SUMP

This sump will be located at the No. 1 Shaft bottom. The water generated from the No. 1 Shaft and loading pocket will be directed down to this sump. A dirty water submersible pump will pump the water up the shaft to the 685 Level's pump station. The proposed design includes a two-ton overhead monorail for pump maintenance, a submersible sump, access walkway, and discharge piping. The sump is large enough to allow access for a 6.1 m^3 LHD for mucking out slimes. The submersible sump discharge line is comprised of a flexible rubber hose connected to a steel discharge line containing a check valve, a main shut-off valve, and a drain valve. Power at 600 V is supplied down the shaft from the 740 Level and feeds an electrical panel located above the walkway.

LEVEL SUMPS

Water collection sumps will be located on each level. The sumps will be located near the point where the ramp and level access cross-cut intersect and will be designed to prevent water entering the ramp from the levels. The proposed design includes a benched floor sump, a separation wall containing an overflow weir, two drilled boreholes with collars down to the next horizon, and a steel walkway out to the holes for unplugging and maintenance access. The sump is large enough to allow periodical cleaning using an LHD to remove settled slimes. There will normally be two drilled boreholes collared near the entrance of the sump to send overflow water down to next level sump or main collection sump.

MAIN SUMPS

The main dirty and clear water dewatering sumps will be installed below the 685 Level. There will be two dirty water sumps, each measuring approximately 4.5 m wide and 21 m long, with water overflowing from the dirty water sump into a

clear water sump. A concrete dam will be built at the opening of each dirty water sump and removable timber baffle walls constructed, dividing the dirty water sumps into three sections. The dirty water sumps are alternated and when built-up sediment needs cleaning out, the flow is switched to the alternative sump. Slimes are removed from the sumps with LHDs and brought to a remuck bay for drying, and subsequently hauled to the loading pocket for hoisting to surface. The clear water sumps, similar in size to the dirty water sumps but at a -20% gradient, will be utilized to treat and store clear water prior to recirculation within the mine or discharge. It will be pumped to a recharge tank for underground process water, as required, or discharged to the water treatment facility on surface.

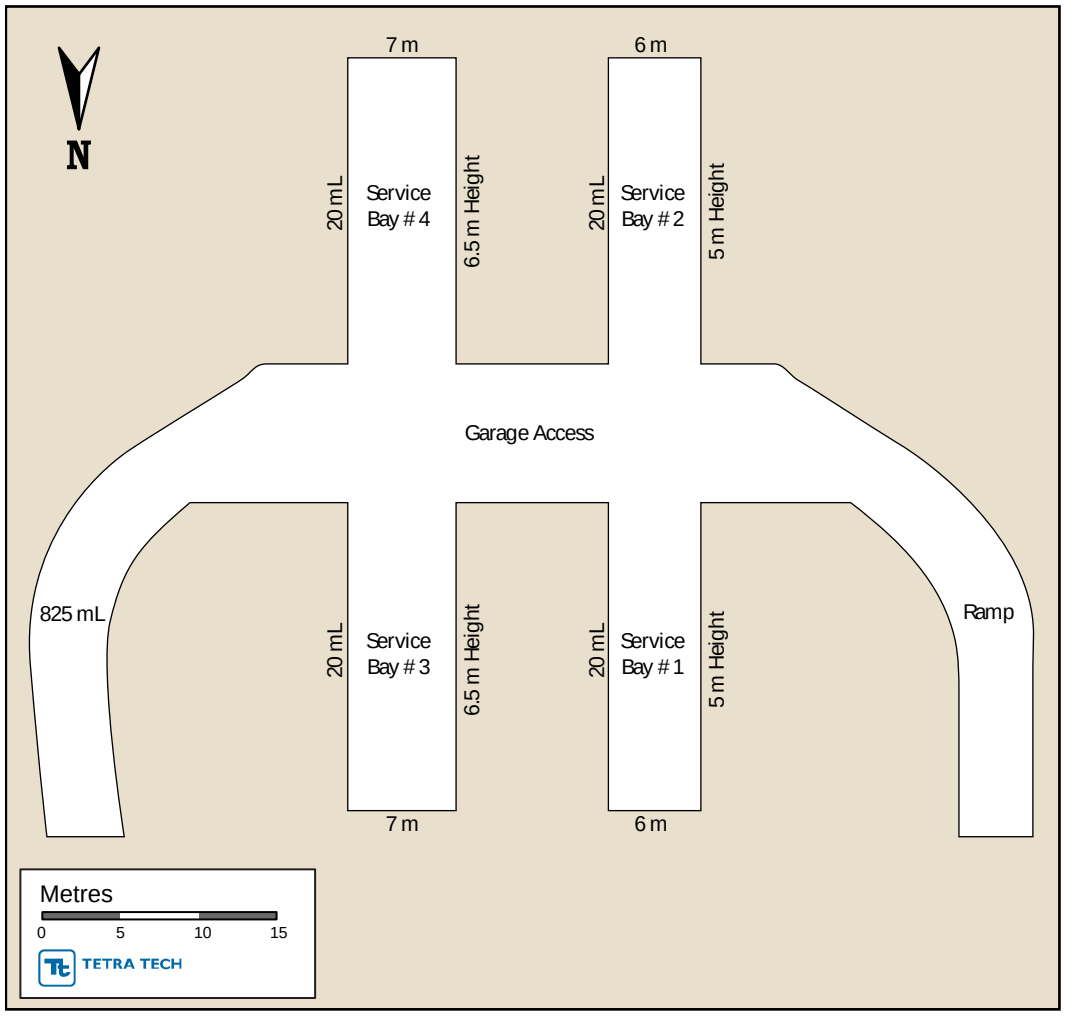
EMERGENCY WATER MANAGEMENT

Due to the high quantity of groundwater infiltrating the mine, during spring runoff and exceptional thunderstorm activity, it was decided that an emergency water storage area would be required. For this study, it was assumed an emergency storage will be built on 345 Level and provide 12 hours of groundwater storage capacity. This would likely provide enough time to cover a major power outage, to change out a pump, or repair a failed pump line. During runoff events, the groundwater from the surface pit enters the Roby Zone workings and flows freely down the main decline. An emergency runoff storage area will be located on 650 Level which is at the bottom of the cross over incline from Roby Zone to the Offset Zone; here the water will be intercepted from the ramp and stored in the down ramp to the lower levels in the Roby Zone.

16.10.3 MAINTENANCE SHOP

The maintenance facility for the Offset Zone Phase I will be comprised of three single bays, which will be used primarily for maintenance and minor servicing of trackless equipment. The main shop bay will be equipped with an overhead 10 t bridge crane, with a warehouse located at the back end of the bay. Tool boxes and work benches will be stationed along one side of the bay. The second equipment service bay will not have a crane. This bay will be mostly used for services on equipment that does not require the use of the crane. Similar to the main bay, tool boxes and work benches will be stationed along one wall of the bay. Each bay will also be equipped with a fire suppression system and a fume exhaust system. The garage's fire suppression system supervisory flow and pressure switches will be wired to a small fire alarm panel mounted in the garage office. In the event of a fire, a hardwired signal from the fire alarm panel will be used to close the garage bay door. The garage door can also be closed manually with a hand-held remote control and local push button.

Figure 16.14 Plan View 825 Level Maintenance Shop



An office will be located across from the secondary bay in the shop. In the office, a phone and a business computer work station will be installed. Outside the office, away from the walkway, there will be sufficient room for a parts staging area. Parts will be delivered and stored here temporarily.

Outside the garage door, a triple Sat-Stat station (satellite fuel/oil station) will be installed on a concrete pad, situated away from equipment travel. The Sat-Stat station will be equipped with its own fire suppression system and a thermal activated, Underwriters Laboratories of Canada (ULC)-approved four-hour rated roll-up fire door. The concrete floor beneath the Sat-Stat station will facilitate the cleanup of potential oil and fuel spills.

An oil/water separator will be installed to serve both the maintenance facility and the wash bay. The maintenance facility will have a concrete floor to facilitate the control and clean-up of spills. Only one underground garage is planned for basic servicing and repairs of the mining trackless equipment. This garage will be located on the

825 Level near the No. 1 Shaft station. All major repairs of mobile equipment are planned to be done in the surface heavy equipment workshop.

Bollards will be placed in front of the fire doors to prevent inadvertent entry from mobile equipment.

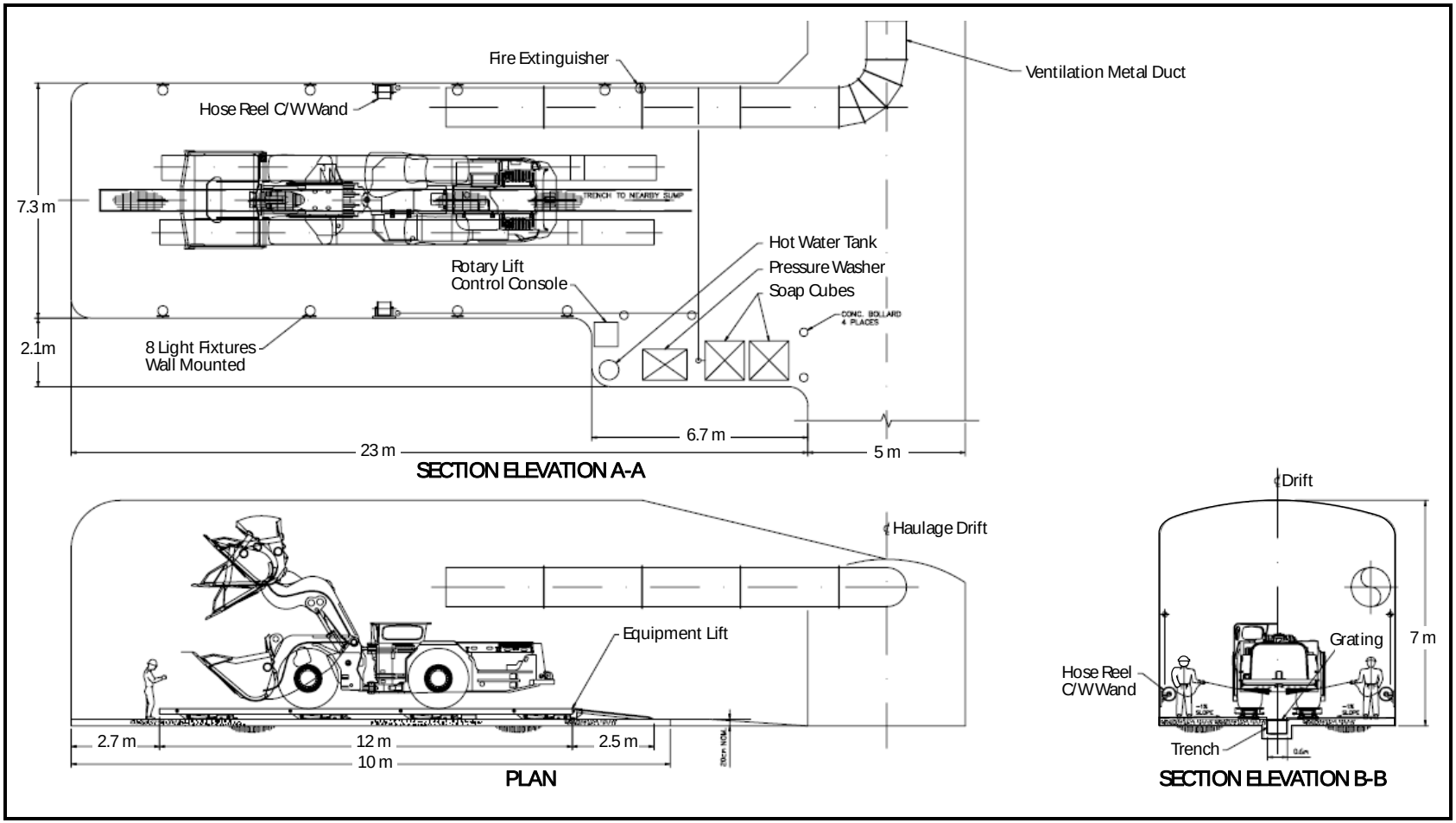
16.10.4 WASH BAY

A wash bay will be located near the main maintenance facility to ensure the equipment can be properly cleaned before servicing. The wash bay as depicted in Figure 16.15 will be comprised of one bay, large enough to accommodate the largest piece of equipment and will have a concrete floor for ease of clean-up. Water from the area will be drained to a nearby sump. One wash bay will be constructed in conjunction with one multi-vehicle garage to clean the trackless equipment before servicing. The wash bay will be located on the 825 Level.

16.10.5 FUEL STATION

Diesel fuel will be transferred underground via a dry fuel line located in the shaft to refill underground portable diesel fuel stations. Satellite diesel fuel stations (Sat-Stat) will be utilized underground in order to store the fuel for underground slower mobile equipment like drills and LHDs during the Offset Zone Project. Each station has a capacity exceeding 9,060 L. Stations will be equipped with either an electric or pneumatic dispensing system surrounding a 1.5 m high concrete wall, acting as a catchment basin for any fuel that might leak from the fuel tanks.

Figure 16.15 Wash Bay 825 Level



16.10.6 REFUGE STATION

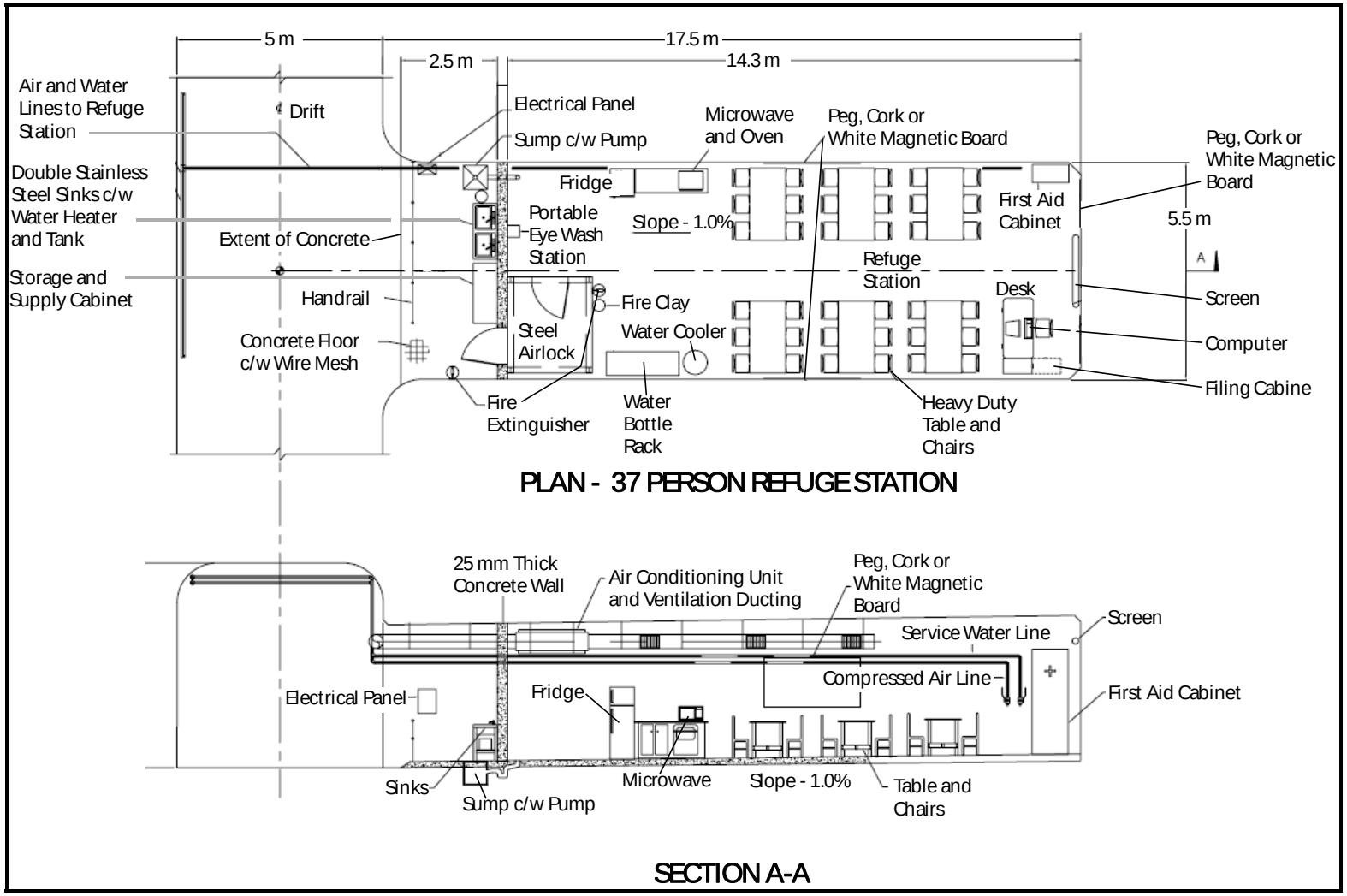
Typically, a person in a confined space requires 1 m^3 of air per hour to survive. Refuge stations as depicted by Figure 16.16 will be designed to hold 37 people (290 m^3). The stations will serve a dual purpose:

- Personnel working in the area can use the refuge station for their lunch room, and as a meeting and communications area. The area supervisor will also have an office with direct communications to surface. Washing facilities will be provided outside of the refuge station to assist in housekeeping and keeping the interior clean. Appliances will be provided, inside and out, as required to serve these purposes.
- The second purpose is in the event of an underground emergency. Workers will be able to gather in the refuge station for refuge and communications. The bulkhead enclosing the room will be fireproof and recessed away from the main access drift and protected from any explosive concussion. Each refuge station will be equipped with an air-lock system. The access man door will be fire-rated and tightly fitted to the frame. Fire clay is provided immediately inside the doorway to seal any gaps in the door against smoke, fumes, or fire. The room will be equipped with first aid equipment, air and water lines, and an air conditioning unit. The floor and bulkhead will be fitted with a submerged drain to prevent smoke and fumes from entering the refuge station.

Using these underground facilities for a dual purpose (as lunch rooms as well as refuge stations) will encourage miners to maintain their familiarity with the location and layout. The refuge stations will have the potential to be readily converted for use as an underground fresh air base, in case of emergency.

The two permanent refuge stations will be located on the 740 and 825 Levels. When considering the number of refuge stations and their locations in the Offset Zone, the number of people working underground per shift, walking distances, access to ventilation raises, and production areas were taken into account. The area specific supervisor will have an office equipped with a phone and capability for a computer work station.

Figure 16.16 Refuge Station Typical



16.10.7 EXPLOSIVES STORAGE

The size of the powder magazine will be 6.7 m wide by 6.7 m high by 14 m long and will be at least 60 m from the following:

- the main access into or from the mine
- key mechanical and electrical installations that remain in service during a mine emergency
- areas of refuge or other areas where workers may congregate
- storage areas for fuels or other potential sources of fire.

The powder magazine as depicted by Figure 16.17 will include a one ton electric-powered jib crane for travel and lift, a concrete floor, an elevated concrete storage pad for pallet storage, ventilation, lighting, and will be capable of holding eighteen, 450 kg totes of ammonium nitrate fuel oil (ANFO) or emulsion. The location of the powder magazine will on the 825 Level.

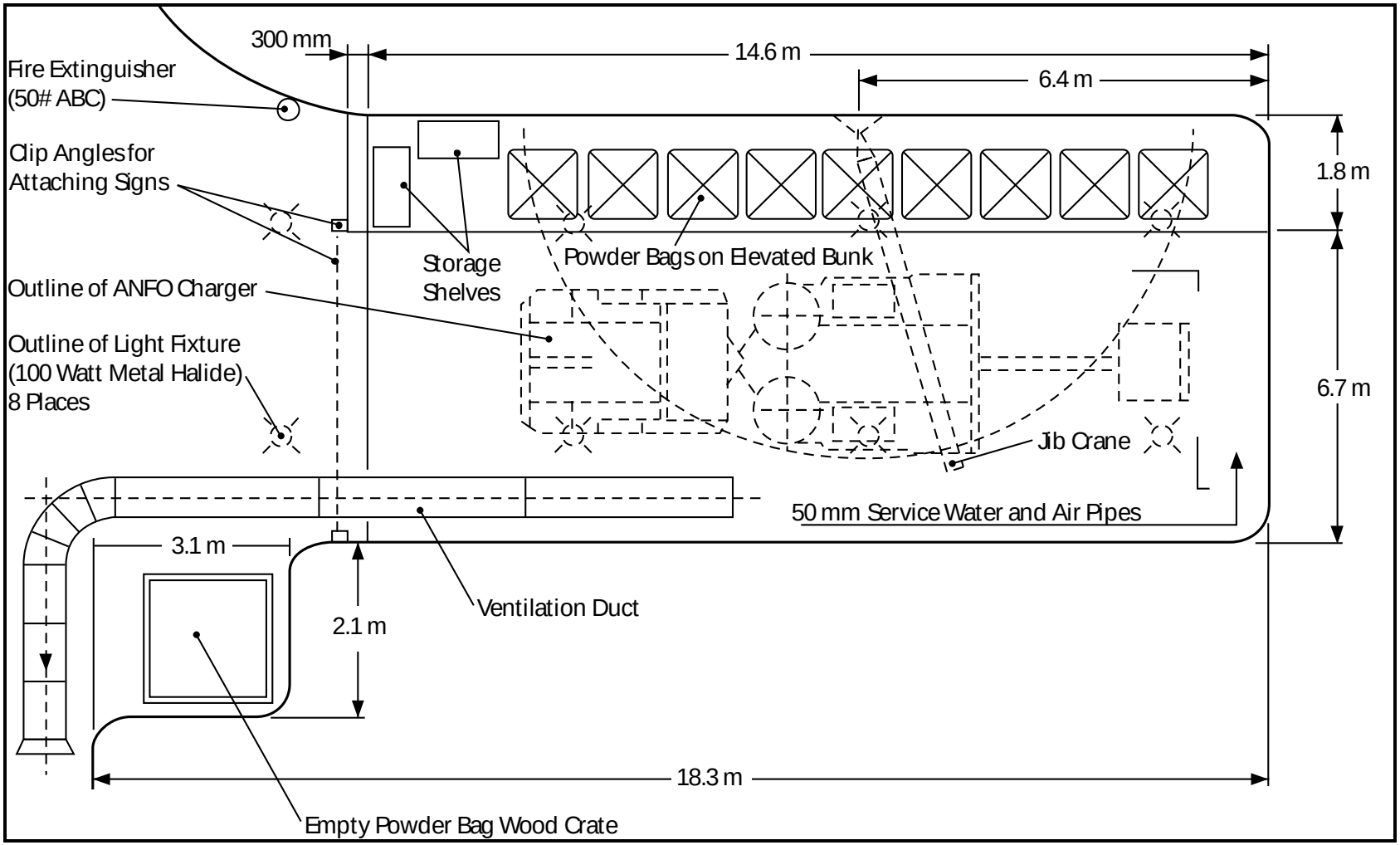
16.10.8 DETONATOR MAGAZINE

A detonator magazine will be located near the explosives magazine. The detonator magazine will be 6 m deep by 3 metres wide by 3 m high. The magazine will include a timber and screen wall with an access door, providing for the inside length of the magazine to be 4 m. The magazine will be equipped with suitable wooden shelving to allow stacking of detonator boxes on each side. Access controls and sign out documentation will be placed inside the detonator chamber. An explosion-proof red light will be installed outside the magazine to indicate its location. The magazine will have bollards in front to prevent inadvertent entry by mobile equipment.

16.10.9 MATERIALS HANDLING AND STORAGE AREA

A large storage area for mining consumables including pipe and fittings, ground support materials, ventilation supplies, etc. will be developed on the 825 Level. The storage area will be 20 m long by 7 m wide by 5 m high. The storage area will include shelving and low wooden racking to safely store articles. Materials and parts will be palletized or placed in specially designed containers (for bulk materials and parts) for sending underground via the ramp and shaft (when completed). Service vehicles will transport the bulk materials to the storage areas. Materials will be distributed from the storage areas to work place storage areas by service vehicles.

Figure 16.17 Explosives Storage Magazine



16.10.10 SANITARY FACILITIES

Underground toilets will be provided on the 740 and 825 Levels. These will be located in openings of approximately 5.1 m deep by 5.5 m wide by 3.3 m high with a gravel floor. The facility will include a recessed concrete wall 2.5 m high, with a double hinged door in the wall for access. Two walled toilet stalls will be provided, each having their own door. The actual toilet is a portable mining toilet. The latrine will be equipped with coat pegs, cold water tank, and a stainless steel sink. A cleaning hose and water connection will also be provided, as will a hot water tank for the use of the personnel.

16.11 MINE SERVICES

16.11.1 ELECTRICAL DISTRIBUTION

The primary electrical distribution system for the Offset Zone Phase I will comprise a new 10 MVA sub-station at the collar of the shaft. The existing Lac des Iles Mine sub-station will feed a new underground mine surface sub-station via a 44 kV overhead line. The sub-station will transform power down to 13.8 kV for transmission underground and 4,160 V for surface feeds. The 13.8 kV feed will be sent underground by two 500 MCM cables, both powered and interlocked so that both are kept “warm”. Either cable could carry the entire mine load, providing 100% redundancy in case of failure. The power cables will tie in on the levels, and will be Verlock-type in construction. Loads on the level will be handled by mobile power centres that are 2 MVA in capacity and transform power down from 13.8 kV to 4,160 V and 600 V, as required.

On surface, the 4,160 V feed will drive the hoists, fans, and other power requirements for the underground mine surface facilities. The system will utilize a switch room/motor control centre (MCC) room at the hoist house to direct and control the power distribution around the surface plant for the shaft.

The anticipated loads are depicted in Table 16.20 for the fixed equipment and Table 16.21 for the mobile electrical equipment.

Table 16.20 Electrical Underground Electrical Fixed Equipment

Load Description	Quantity	Connected (kW)	Energy per Month (kWh)
Underground Booster Fans			
600 hp	1	450	307
Sump and Mud Pumps	1	14	4
Lighting	1	7	2

table continues…

Load Description	Quantity	Connected (kW)	Energy per Month (kWh)
750 Level			
125 hp Vent Fan	2	186	93
Sump and Mud Pumps	2	28	8
Lighting	1	7	2
740/760 Level			
125 hp Vent Fan	2	186	93
Sump and Mud Pumps	2	28	8
Loading Pocket	1	20	8
Lighting	1	7	2
Refuge Station	1	12	1
Spill Pocket	1	12	3
825 Level			
125 hp Vent Fan	2	186	93
Sump and Mud Pumps	1	14	4
Lighting	1	7	2
Main Dewatering Pumps	1	521	240
Shaft Station	1	7	2
Refuge Station	1	12	1
Underground Wash Bay	1	15	2
Underground Maintenance Shop	1	25	3
Underground Storage	1	7	2
Subtotal	**-**	**-**	**977**
Contingency 20%	-	-	195
Total	**-**	**-**	**1,172**

Table 16.21 Electrical Underground Electrical Mobile Equipment

Load Description	Quantity	Connected (kW)	Energy per Month (kWh)
Jumbo	2	158	172
Mclean Bolter	2	45	49
Cable Bolter	1	116	20
Shotcrete Sprayer	1	110	19
ITH Drill c/w Compressor	2	148	137
Diamond Drill	2	37	57
Contingency 20%	-	-	90
Total	**-**	**-**	**544**

16.11.2 COMPRESSED AIR

Compressed air will be supplied by two new compressors located in the compressor room of the new hoist house as depicted by Figure 16.18. The system will consist of three 451 kW (650 hp) Sullair Air twin stage screw compressors with an estimated capacity of 3.5 m^3/s (7,590 cfm) at 7.6 to 8.3 bar (110 to 120 psi) pressure. The existing compressors will continue to supply the Roby Zone ore body and will be connected to the Offset Zone Phase I compressed air system to supplement air supply if required.

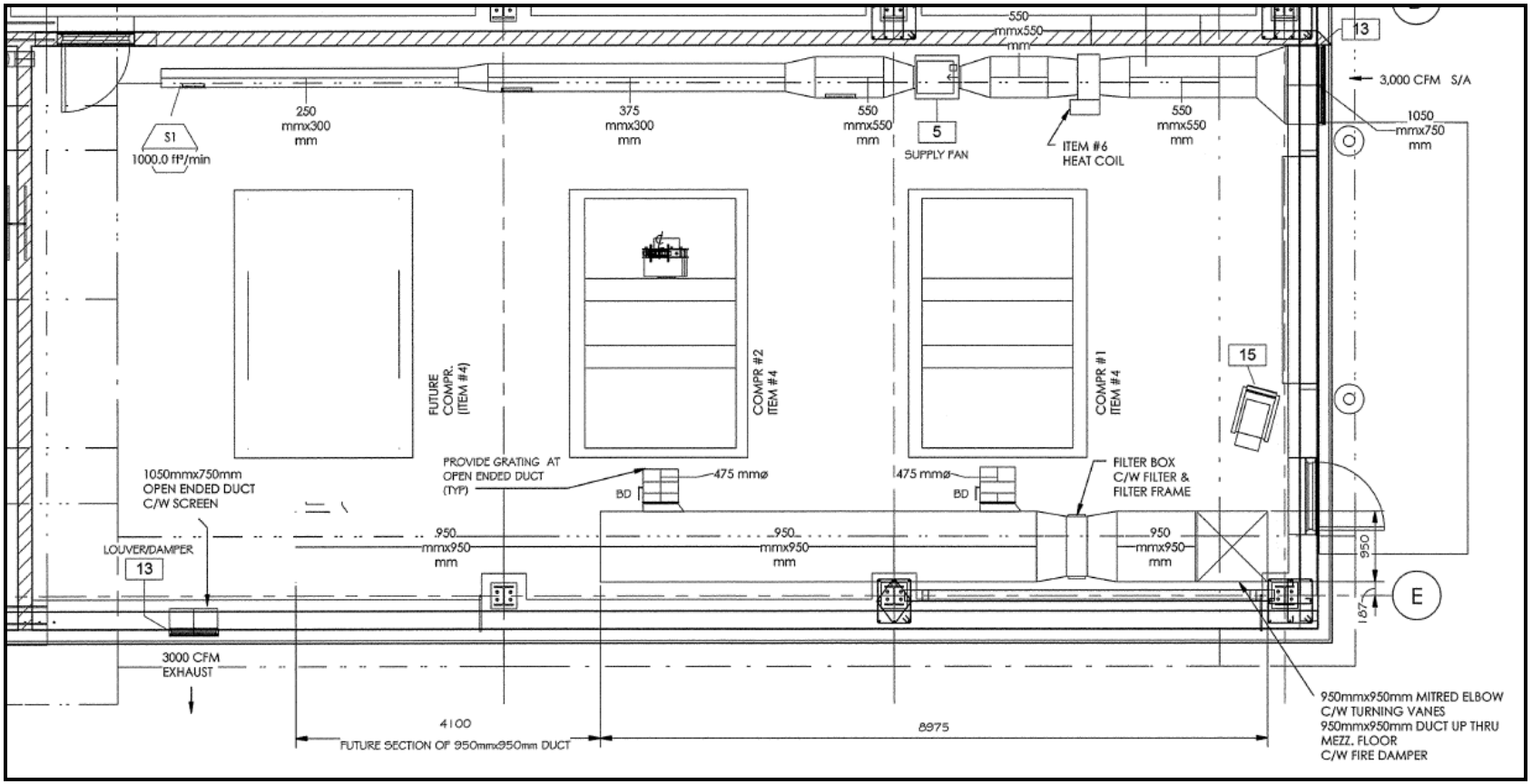
The compressed air is transferred from surface to underground using 200 mm pipes located in the shaft and then distributed via 152 mm piping on the levels up to stope access cross-cuts. Piping in the stopes will be 102 mm in diameter.

The compressed air requirements are shown in Table 16.22.

Table 16.22 Compressed Air Equipment Consumption

Equipment	Units Required		Usage (%)	Unit Flow (cfm)	Requirements (cfm)	
	Average	Peak			Average	Peak
Underground						
In-the-hole (ITH) Drills	2	2	90	450	810	810
Top Hammer	0	1	75	250	0	188
ANFO Loader	1	2	25	50	13	25
Hole Cleaning	1	1	25	200	50	50
Air Powered Doors	10	14	75	20	25	40
Jackleg/Stoper	2	4	90	180	1,350	1,890
Alimak	0	1	100	95	0	95
Shotcrete Machines	1	2	100	750	750	1,500
Underground Garage	1	1	50	60	30	30
Service Bay	1	1	25	20	5	5
Blow Pipe(2 in)	0	1	15	1,000	0	150
Miscellaneous/Air Leaks (15%)	-	-	-	-	503	810
Total	**-**	**-**	**-**	**-**	**3,855**	**6,207**

Figure 16.18 Compressor Room



16.11.3 PROCESS WATER

Water will be sent underground in two 102 mm pipes located in the shaft. This will feed the main 102 mm diameter level distribution lines on the levels, which will send water to the stope access cross-cuts. Piping in the stope accesses and sills will also be 50 mm. Water pressures and volumes will be controlled by pressure reducing valves installed in the shaft at approximately 115 m intervals.

The service water requirements are shown in the following Table 16.23.

Table 16.23 Service Water Consumption

Equipment	Units	Flow Rate per Unit (m^3/h)	Cycle Factor (%)	Utilization (%)	Total Usage	
					(m^3/h)	(m^3/d)
Drill Jumbo	2	4.1	60	85	4.18	76
MacLean Bolter	2	4.1	60	85	4.18	75
Jackleg/Stoper	10	0.2	33	60	0.40	7
Water Spray	5	1.2	65	95	3.71	67
Diamond Drill	2	2.3	55	80	2.02	36
ITH Drill	2	15.0	65	95	18.53	333
Emulsion Loader	1	4.5	100	100	4.50	81
Service Area	1	3.0	30	50	0.45	8
Shotcrete Machine	1	4.5	20	100	0.90	16
Dust Suppression	3	4.5	80	90	9.72	175
Miscellaneous (10%)	-	-	-	10	4.86	87
Total	**-**	**-**	**-**	**-**	**53.44**	**962**

16.11.4 MINE COMMUNICATIONS AND CONTROL SYSTEMS

The Offset Zone Phase I area will be provided with a communications and data network to provide voice communications, and a programmable logical controller (PLC). A leaky feeder system will be used through the primary development and level accesses to facilitate radio and data transfer throughout the mine. A separate Ethernet line will be run to the main shop on the 825 Level, as well as to the crushing and pump stations.

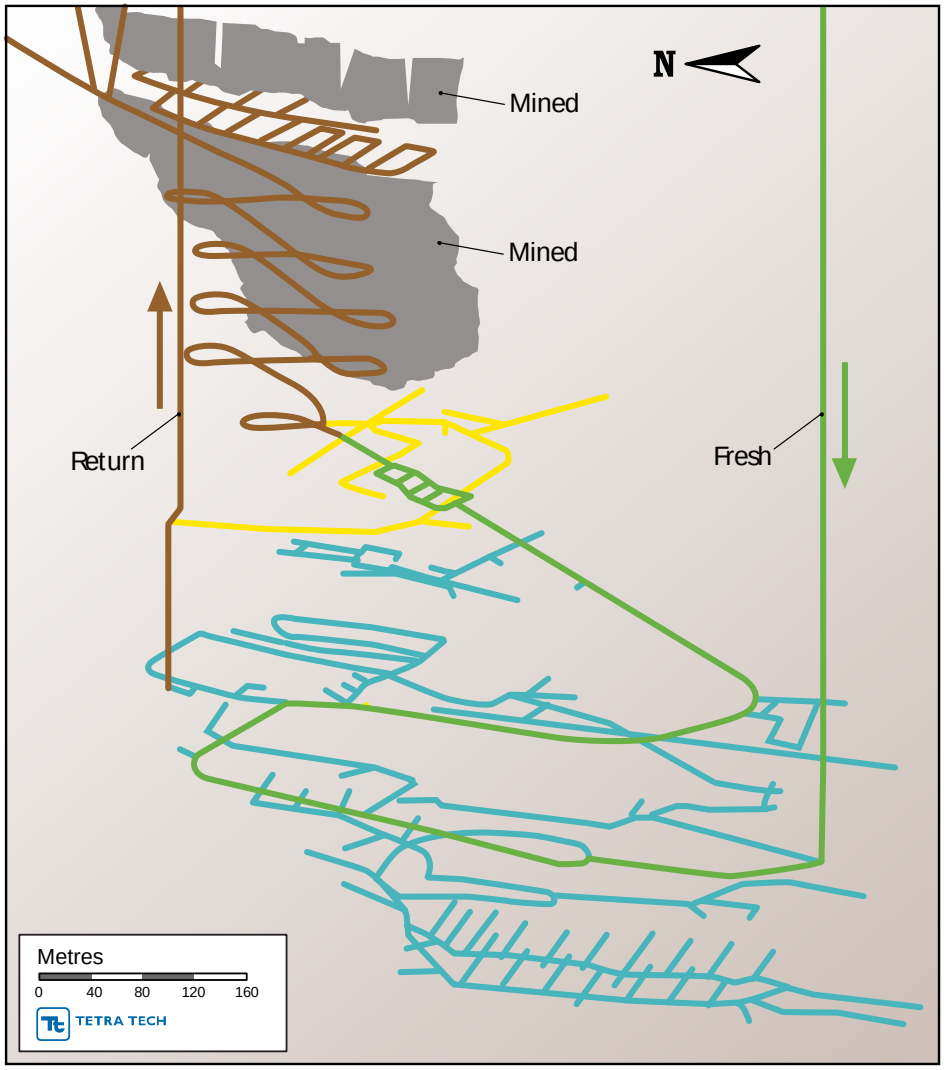
16.12 VENTILATION

16.12.1 SYSTEM OVERVIEW

The mine ventilation system, as depicted by Figure 16.19, will provide 750 m^3/s of fresh air to the lower mine through the No. 1 Shaft and existing fresh air system,

which will extend from surface to No. 1 Shaft bottom. Fresh air will be heated during winter to +3°C by direct-fired propane mine air heaters to prevent freezing. The mine will exhaust a total of 750 m³/s through a combination of the main ramp and the 5 m diameter exhaust raise. The preliminary design will be revisited as part of the detailed design of the ventilation system.

Figure 16.19 Mine Ventilation Long Section



16.12.2 VENTILATION REQUIREMENTS

The mine ventilation system has been designed to support the planned fleet of diesel haulage equipment, as well as to provide adequate ventilation for drilling and other activities.

Provision of sufficient dilution air for the haulage equipment is the main requirement. The mine ventilation requirement has been estimated based on the proposed diesel

fleet and the dilution air required for each vehicle. A design factor of 0.06 m^3/s/kW of running diesel power was used to determine the ventilation requirement. The ventilation estimate used for this study includes a utilization factor that was applied to each type of equipment, thereby more accurately representing the actual mining scenario by determining the running diesel power.

Ventilation requirements are summarized in Table 16.24.

Table 16.24 Ventilation System Requirements – (0.06 m³/s/kW Design Basis)

Area	Unit	kW/ Unit	Units	Total kW	Unfactored Requirement (m³/s)	Utilization Factor (%)	Factored Requirement (m³/s)
Production	LHD 11 yd^3	354	2	708	84.96	80	68
	LHD 8 yd^3	242	2	484	58.08	35	20
	LHD 3 yd^3	140	2	280	33.6	5	2
	Truck 45/50 t	390	5	1950	585	80	468
	Truck 30 t	298	2	596	71.52	60	43
	Jumbo	74	2	148	17.76	15	3
	Rock Bolter	113	2	226	27.12	10	3
	Blockholer	110	1	110	6.6	10	1
	Scissor Lift	58	2	116	13.92	10	1
	ANFO Loader	130	2	260	31.2	10	3
	Shotcrete Sprayer	110	1	110	6.6	10	1
	Transmixer	155	2	310	37.2	25	9
Service	Scissor Lift	130	2	260	31.2	10	3
	Tractor	68.6	2	137.2	16.464	25	4
	Grader	110	1	110	6.6	25	2
	Toyota	95	7	665	279.3	5	14
	Boom Truck	60	2	120	14.4	25	4
	Fuel Truck	130	2	260	31.2	25	8
	Main Garage	-	-	-	-	-	17
	685 Crusher	-	-	-	-	-	8
				Subtotal	**1,353**	**-**	**681**
				Leakage @ 10%	-	-	68
				Total	**-**	**-**	**749**

As illustrated in Table 16.24, the design ventilation airflow totals 750 m³/s and would allow some flexibility during operations.

16.12.3 SYSTEM DESIGN

The mine ventilation system will utilize a two-part design. The upper levels of the mine will be a push system through the Roby main fan. The portals will also exhaust to atmosphere based on activity in the upper mine. The lower mine will use an

exhaust fan on the 5360 bench in the Roby Pit, which will downcast the shaft and heated at the shaft collar.

During detailed design, the distribution of airflows throughout the mine, and in particular along the ramp, will be reviewed using a program such as Ventsim or VnetPC PRO+. As the ramp will continue to be used for diesel truck haulage as well as being a travel way for support equipment, it is critical that sufficient ventilating air is provided in the ramp throughout its length. It is recognized that the ventilation schematic will need to be refined to meet any future extension of the Offset Zone as well as other ancillary discoveries as the mine grows in the future.

An analysis of the mine's current ventilation system, based on anticipated flows and distribution within the mine, was undertaken to establish fan requirements. During detailed design, calculations for the mine ventilation circuit analysis and fan requirements must be updated to reflect final volumes, as fleet requirements have been refined to accommodate the increased production rate, while the ventilation calculations were not updated.

Ventilation control within the mine will be achieved through the use of doors, regulators, and booster fans. Auxiliary ventilation using axial vane fans and ducting will be used to supply air to development and operating headings.

16.13 EQUIPMENT

16.13.1 MOBILE EQUIPMENT

The mobile equipment requirements for the Offset Zone Phase I Project are shown in Table 16.25.

Table 16.25 Mobile Equipment Requirements

Description	Typical Model	Units	Service/Comments
Truck (35 t)	MTI	3	Waste haul per Table 16.9.3
Truck (50 t)	Toro	5	Ore haul per Table 16.9.1
Truck (45 t)	CAT	4	On hand placed "On Standby"
3.5 yd Scoop	Wagner/EJC	2	One on waste haul per Table 16.9.3; Second "on standby"
8 yd	Toro/MTI	2	One on waste haul per Table 16.9.3; Second "on standby"
11 yd	Toro/CAT	3	Two on ore haul per Table 16.9.1; Third "on standby"
Two Boom Jumbo	Axera/MTI	2	-
ITH Drill Complete With Booster Compressors	Cubex	2	Drilling both downholes and upholes

table continues…

Description	Typical Model	Units	Service/Comments
Mechanised Bolter	Maclean	2	Ground support
Blockholer	Maclean	1	Drilling oversize in draw point
Fuel Truck	Getman	1	-
ANFO Loader	Getman	1	-
Scissor Lift	Various	9	-
Grader	CAT	1	-
Tractor	Miller	1	-
Boom Trucks	Tamrock/Getman	2	-
Transmixer Truck	Marcott	2	-
Shotcrete Sprayer		1	-
Carrier	Getman	2	-
Man Carrier		10	-

Equipment for a typical production unit will consist of one 11 yd^3 LHD mucking from the stope, loading three 50 t trucks hauling ore to the No. 1 Shaft's ore handling system. Each LHD will be sized with sufficient bucket reach to load the trucks without the need for a loading ramp. Each LHD will be equipped with radio remote control for mucking ore beyond the draw point brow.

Production drilling will be done with ITH drills complete with booster compressors. A cost/benefit analysis of automated drills will be completed during detailed engineering.

Development will be carried out with one 3.5 yd^3 LHD and one 6.1 m^3 LHD and three 35 t trucks.

Rock bolts will be placed immediately to the face with the rock bolters. Scissor lifts will be used for installing screen for ground support and for services. Cable bolt holes will be drilled with the Mclean Bolter and the cables will be placed and grouted from a scissor lift.

16.13.2 *FIXED EQUIPMENT*

Fixed equipment requirements for the mine are summarized in Table 16.26.

Table 16.26 Fixed Equipment Requirements

Surface	Units	Comments
Production Hoist	1	-
Service Hoist	1	-
Auxiliary Hoist	1	-
Compressors	3	New Compressors
Surface Intake Fan and Heater	1	New Fan
Surface Exhaust Fan	1	New Fan
Underground		
Underground Booster Fans		
600 hp	1	-
Auxiliary Fans – 125 hp	21	-
Auxiliary Fans – Various Sizes	10	Various Sizes
Electrical Mine Power Centres	14	Portable, 13 kVa-4160,600 V
645 Level		
Rock Breaker with Grizzly	2	1-Ore Pass, 1-Rock Pass
685 Level		
Crusher	1	-
740 Level		
Loading Pocket	1	-
825 Level		
Main Dewatering Pumps	3	New Pumps
Underground Wash Bay	1	-
Underground Maintenance Shop	1	-
Refuge Station	1	-
Underground Storage	1	-
Ore Pass Controls	3	-
Shaft Bottom Pump	1	-

16.14 WORKFORCE

Mine manpower requirements for Phase I are summarized in Table 16.27. Manpower will drop off from a high of 378 persons in 2013 to 289 persons by 2017, principally due to the completion of major infrastructure development by the end of 2014, and a winding down of definition diamond drilling/exploration activities.

Following the pre-production period, during which a mining contractor will be used for development and construction, all underground activities will be performed by owner's crews with the exception of Alimak raising and diamond drilling, which will continue to be done by a contractor.

Table 16.27 Manpower Schedule – Mine Operations

Description	Shift	Year (No. of Persons)						
		2013	2014	2015	2016	2017	2018	2019
Mine Site Administration								
General Manager	4 by 3	1	1	1	1	1	1	-
Community Relations Manager	4 by 3	1	1	1	1	1	1	-
Executive Assistant	4 by 3	1	1	1	1	1	1	-
Materials Management								
Materials Manager	4 by 3	1	1	1	1	1	1	-
Buyer	7 by 7	1	1	1	1	1	1	-
Buyer	7 by 7	1	1	1	1	1	1	-
Warehouse Supervisor	4 by 3	1	1	1	1	1	1	-
Buyer	7 by 7	1	1	-	-	-	-	-
Junior Buyer/Warehouse Assistant	4 by 3	1	1	1	-	-	-	-
Materials Controller 1	7 by 7	8	8	6	6	6	6	-
Human Resources								
Human Resources Coordinator	4 by 3	2	2	2	1	1	1	-
Senior Human Resources Coordinator	4 by 3	1	1	1	1	1	1	-
Nurse/Occupational Health and Safety Shared	7 by 7	1	1	1	1	1	1	-
Nurse/Occupational Health and Safety	7 by 7	1	1	1	1	1	1	-
Safety								
Safety Coordinator	7 by 7	2	2	2	2	1	1	-
Safety Superintendent	4 by 3	1	1	1	1	1	1	-
Trainer	7 by 7	4	4	3	2	2	2	-
Union Safety Representative	7 by 7	1	1	1	1	1	1	-
Technical Services								
Technical Services Manager	4 by 3	1	1	1	1	1	1	-
Senior Mine Engineer	7 by 7	1	1	1	1	1	1	-
Mine Production Planner	7 by 7	2	2	2	2	1	1	-
Chief Geologist	4 by 3	1	1	1	1	1	1	-
Geology Technician	4 by 3	2	2	2	2	1	1	-
Junior Mine Geologist	4 by 3	2	2	2	2	1	1	-
Mechanical Engineer in Training	4 by 3	1	1	1	-	-	-	-
Mine Development Planner	7 by 7	2	2	1	1	1	1	-
Open Pit Planner	7 by 7	2	-	-	-	-	-	-
Principal Geologist	7 by 7	1	1	1	1	-	-	-
Projects Engineer	4 by 3	1	1	1	-	-	-	-
Rock Mechanics Technician	7 by 7	1	1	1	1	1	1	-

table continues…

Description	Shift	Year (No. of Persons)						
		2013	2014	2015	2016	2017	2018	2019
Senior Geologist	7 by 7	1	1	1	1	1	1	-
Survey Technician	7 by 7	2	2	2	2	2	2	-
Mine Technologist	4 by 3	1	1	1	1	1	1	-
Environmental								
Environmental Superintendent	7 by 7	1	1	1	1	1	1	-
Environmental Technician	7 by 7	1	1	1	1	1	1	-
Environmental Technician	5 by 2	1	1	1	1	-	-	-
Finance and IT (Staff)								
Senior Financial Analyst	5 by 2	1	1	1	1	1	1	-
Local Area Network Technician	4 by 3	1	1	1	1	1	1	-
Local Area Network Technician	7 by 7	1	1	1	1			-
Accounts Payable Administrator/ Payroll Clerk	5 by 2	1	1	1	1	1	1	-
Cost Accountant	7 by 7	2	2	1	1	1	1	-
Accounts Payable/Accounting Clerk	5 by 2	1	1	1	1	1	1	-
Maintenance								
Business Improvement Manager	4 by 3	1	1	1	1	1	1	-
Project Engineer	7 by 7	2	2	1	1	1	1	-
Fixed Plant Maintenance								
Senior Fixed Plant Supervisor	4 by 3	1	1	1	1	1	1	-
Fixed Plant Supervisor	7 by 7	1	1	1	1	1	1	-
Surface Electrical Supervisor	7 by 7	2	2	2	2	2	2	-
Fixed Plant Supervisor	7 by 7	1	1	1	1	1	1	-
Chief Electrician	4 by 3	1	1	1	1	1	1	-
Fixed Plant Planner	4 by 3	1	1	1	1	1	1	-
Surface Electrical Planner	4 by 3	1	1	1	1	1	1	-
Electrical Engineer	4 by 3	1	1	1	1	-	-	-
Electrician Trades 5	7 by 7	8	8	8	8	8	8	-
Mill Trades 5	7 by 7	18	18	18	18	18	18	-
Surface Maintenance								
Maintenance Superintendent	4 by 3	1	1	1	1	1	1	-
Surface Mobile Supervisor	7 by 7	1	1	1	1	1	1	-
Surface Mobile Supervisor	7 by 7	1	1	1	1	1	1	-
Mine Maintenance Planner	7 by 7	1	1	1	1	1	1	-
Data Control Coordinator	7 by 7	2	2	2	2	2	2	-
Mechanic Trades 3	7 by 7	6	6	6	6	6	6	-
Mechanic Trades 5	7 by 7	6	6	6	6	6	6	-
Electrician Trades 5	7 by 7	2	2	2	2	2	2	-
Maintenance Welder Trades 5	7 by 7	2	2	2	2	2	2	-

table continues…

Description	Shift	Year (No. of Persons)						
		2013	2014	2015	2016	2017	2018	2019
Underground Maintenance								
Underground Electrical Planner	7 by 7	1	1	1	1	1	1	-
Underground Electrical Supervisor	7 by 7	1	1	1	1	1	1	-
Underground Electrical Supervisor	7 by 7	1	1	1	1	1	1	-
Underground Mobile Planner	7 by 7	1	1	1	1	1	1	-
Underground Mobile Planner	7 by 7	1	1	1	1	1	1	-
Underground Electrician Trades 5	7 by 7	14	14	14	14	14	14	-
Underground Mechanic Trades 3	7 by 7	5	5	5	5	5	5	-
Underground Mechanic Trades 5	7 by 7	16	16	16	16	16	16	-
Underground Maintenance Welder Trades 5	7 by 7	2	2	2	2	2	2	-
Underground Electrician Trades 5 (Roby Zone)	7 by 7	4	4	-	-	-	-	-
Surface Operations								
Senior Site Operations Supervisor	7 by 7	1	1	1	1	1	1	-
Site Operations Supervisor	7 by 7	4	4	4	2	2	2	-
Prod Shovel/Loader Operator	7 by 7	0	0	0	0	0	0	-
Site Services Heavy Equipment Operator	7 by 7	18	18	18	10	8	8	-
UG Operations								
Mine Manager	4 by 3	1	1	1	1	1	1	-
Senior Underground Coordinator	4 by 3	1	1	1	1	1	1	-
Underground Mine Supervisor	7 by 7	4	4	4	4	4	4	-
Underground Production Miner 2	7 by 7	21	21	21	21	21	21	-
Underground Production Miner 3	7 by 7	17	17	17	17	17	17	-
Underground Production Miner 4	7 by 7	34	34	34	34	34	34	-
Underground Capital Development								
Project Shaft Superintendent	7 by 7	1	1	1	1	1	1	-
Project Shaft Superintendent	7 by 7	1	1	1	1	1	1	-
Senior Underground Project Coordinator	7 by 7	1	1	-	-	-	-	-
Construction Project Manager	7 by 7	1	1	-	-	-	-	-
Underground Construction Coordinator	7 by 7	1	1	-	-	-	-	-

table continues…

Description	Shift	Year (No. of Persons)						
		2013	2014	2015	2016	2017	2018	2019
Construction Superintendent	7 by 7	1	1	-	-	-	-	-
Development Supervisor	7 by 7	1	1	-	-	-	-	-
Underground Construction Supervisor	7 by 7	1	1	-	-	-	-	-
Underground Construction Supervisor	7 by 7	1	1	-	-	-	-	-
Document Controller	4 by 3	1	1	1	1	-	-	-
Underground Production Miner 1	7 by 7	6	6	3	2	2	2	-
Underground Production Miner 2	7 by 7	4	4	3	2	2	2	-
Underground Production Miner 3	7 by 7	13	13	6	3	3	3	-
Underground Production Miner 4	7 by 7	10	10	5	3	3	3	-
Mill Operations								
Mill Superintendent	4 by 3	1	1	1	1	1	1	-
Senior Mill Operations Supervisor	4 by 3	1	1	1	1	1	1	-
Mill Operations Supervisor	7 by 7	3	3	3	3	3	3	-
Process Control Supervisor	7 by 7	2	2	2	2	2	2	-
Mill Operator 1	7 by 7	1	1	1	1	1	1	-
Mill Operator 2	7 by 7	1	1	1	1	1	1	-
Mill Operator 4	7 by 7	12	12	12	12	12	12	-
Assay Lab								
Chief Assayer	4 by 3	1	1	1	1	1	1	-
Assayer	7 by 7	4	4	4	4	4	4	-
Lab Technician	7 by 7	8	8	8	8	8	8	-
Metallurgical Lab								
Metallurgical Engineer in Training	7 by 7	1	1	1	1	1	1	-
Metallurgical Process Assistant	7 by 7	1	1	1	1	1	1	-
Exploration								
Manager	5 by 2	1	1	1	1	1	1	-
Senior Geologist	7 by 7	2	2	2	2	2	2	-
Exploration Manager	5 by 2	1	1	1	1	1	1	-
Exploration Geologist	7 by 7	4	4	4	4	4	4	-
Exploration Consultant	7 by 7	3	3	-	-	-	-	-
Exploration Surveyor	7 by 7	1	1	-	-	-	-	-
Geological Technician	7 by 7	1	1	1	1	1	1	-
Database Manager/Land Manager	5 by 2	1	1	1	1	1	1	-

table continues…

Description	Shift	Year (No. of Persons)						
		2013	2014	2015	2016	2017	2018	2019
Database Geologist	7 by 7	1	1	1	1	1	1	-
Exploration Technician	7 by 7	1	1	1	-	-	-	-
Assistant Exploration Manager	4 by 3	1	1	-	-	-	-	-
Exploration Office Manager	5 by 2	1	1	-	-	-	-	-
Exploration Supervisor	7 by 7	1	1	-	-	-	-	-
Exploration Geologist	7 by 7	1	1	-	-	-	-	-
Exploration Tech Services Supervisor	7 by 7	1	1	-	-	-	-	-
Geological Technician	7 by 7	1	1	1	-	-	-	-
Exploration Technician Supervisor	7 by 7	1	1	1	-	-	-	-
Geological Technician	5 by 2	1	1	1	-	-	-	-
Exploration Geologist	4 by 3	1	1	1	-	-	-	-
Exploration Technician	7 by 7	1	1	1	-	-	-	-
Geological Technician	7 by 7	2	1	1	-	-	-	-
Exploration Surveyor	7 by 7	1	1	1	-	-	-	-
Junior Geologist	5 by 2	1	1	1	-	-	-	-
Exploration Cost Accountant	5 by 2	1	1	1	-	-	-	-
Total Mine	**-**	**378**	**375**	**332**	**300**	**289**	**289**	**-**

17.0 RECOVERY METHODS

The present processing plant, which has a nominal capacity of 15,000 t/d, will continue to process all ore produced from the Roby and Offset zones underground mines. As the Offset Zone will target a mining rate of 5,500 t/d, the processing plant will operate on a two-week operating and two-week shutdown rotation. The current mill throughput is 13,000 t/d based on this operating schedule.

17.1 PROCESS DESCRIPTION

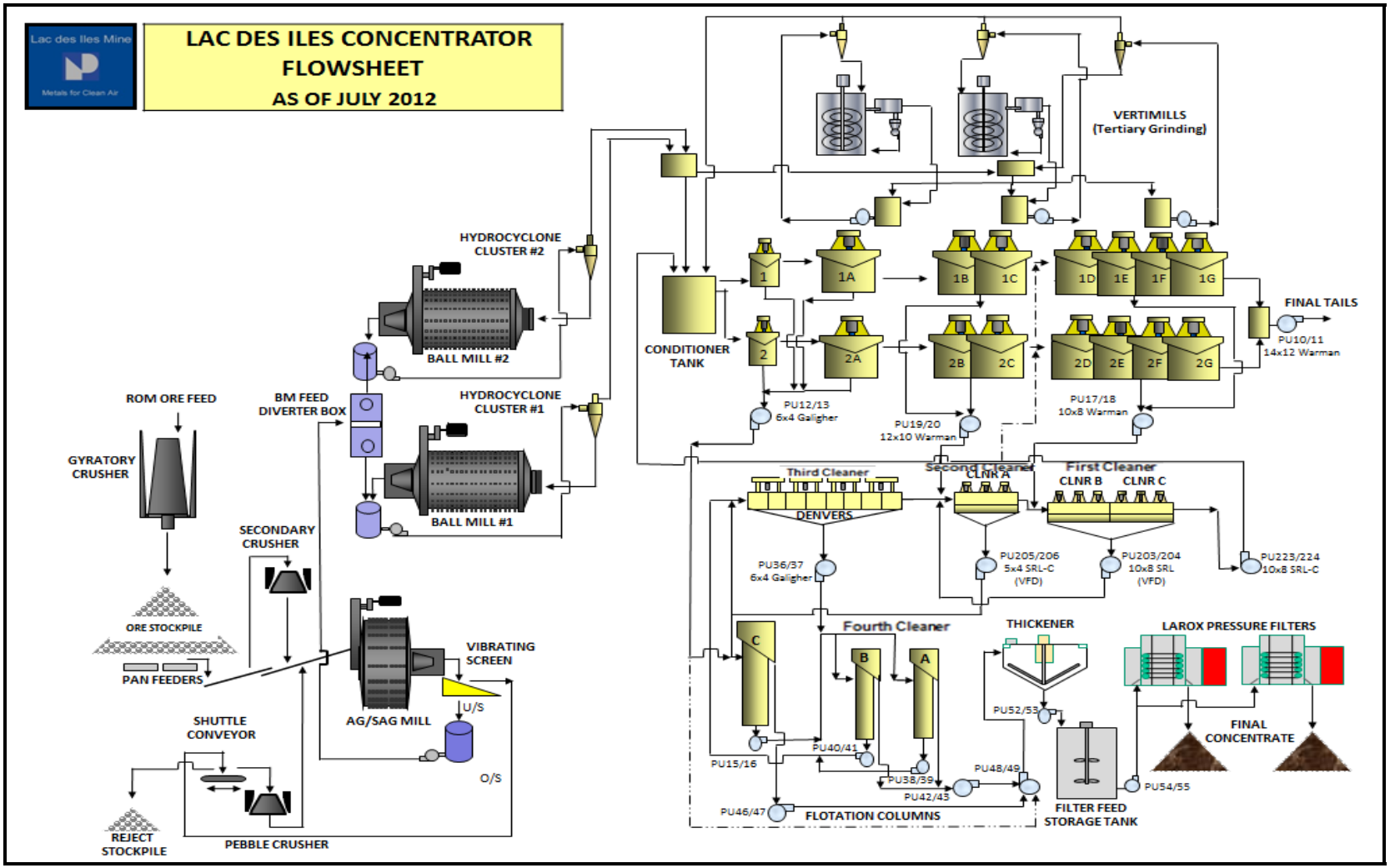
The current crushing, grinding and flotation flowsheet is shown in Figure 17.1. A primary crusher (gyratory) is utilized to crush the run-of-mine (ROM) ore which is then stockpiled. Pan feeders feed the secondary crushers from the stockpile. Secondary crushing is accomplished with a cone crusher (standard HP800). The secondary crusher product is the feed to the SAG mill.

There are two mill buildings at the Lac des Iles Mine site. One is a 2,400 t/d conventional rod/ball mill circuit that operated from 1993 to 2001. The other is a 15,000 t/d 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 to 2,525 t/d. The tonnage is 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 feed grade from underground ore remained high but reverted back to an SABC circuit when underground ore was blended with pit ore (broken oversize). This pit ore used for the blend is from the early days of the open pit and is referred to as regular grade ore (RGO). The RGO is used to supplement mill feed or adjust feed grade. The average head grade for the RGO is 0.97 g/t palladium.

The current mill's SABC configuration runs at 13,000 t/d throughput and utilizes two Vertimills™ (and one standby) for tertiary grinding to approach a grind of P_{80} -38 µm. The mill is run on 14-day campaigns with 14 days off. The autogenous mill (currently being used as a SAG mill) has dimensions of 9.144 m in diameter (30 ft) by 4.270 m (14 ft) 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 (20 ft) by 10.360 m EGL (34 ft) 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 and Figure 17.3 is a picture of one of the Vertimills™.

Figure 17.1 Current Lac des Iles Mine Flotation Circuit



Source: North American Palladium – Lac des Iles

Figure 17.2 SAG Mill and Ball Mills

Source: North American Palladium – Lac des Iles

Figure 17.3 Vertimill™

Source: North American Palladium – Lac des Iles

The SAG product is screened over a vibrating screen with the oversize being belt fed to a pebble crusher (short head cone HP800) and the pebble crusher product is recycled back into the feed to the SAG. The SAG discharge vibrating screen undersize feeds the ball mill diverter box which splits the feed in 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. Plans are in place to revise the Vertimill™ circuit to accept the entire ball mill hydrocyclone overflow.

Each of the Vertimills™ is in closed circuit with hydrocyclone clusters. Hydrocyclone underflows are recycled back in to the Vertimills™ and the hydrocyclone overflows are transferred to the conditioning tank prior to flotation. The Vertimills™ are currently being optimized to achieve a P_{80} -38 µm grind product. The current P_{80} to flotation is approximately 58 µm.

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 tails from rougher flotation move forward to two lines of scavenger flotation (cells 1B to 1G and 2B to 2G). Tails from scavenger flotation are final tails 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 tails 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 tails from the second cleaner. The tails from first cleaner are fed back to the flotation conditioning tank. The tails 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 tails 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. Most of the concentrate is shipped via trucks to a smelter in Sudbury but some are bagged prior to shipping to foreign smelters.

The Lac des Iles Mine 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.

17.2 HISTORICAL AND CURRENT MILL PRODUCTION

The historical and current production figures for the Lac des Iles Mine are shown in Table 17.1 and Table 17.2.

Table 17.1 Historical Production

	2012	2011	2010	2008*	2007	2006**	2005	2004	2003	2002
Tonnes Milled (t)	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)	3.44	3.7	6.06	2.33	2.36	2.18	1.66	2.41	2.31	1.91
Pd Recovery (%)	78.37	78.35	80.8	75.3	74.8	74.0	69.6	75.2	75.5	73.8
Pt Recovery (%)	72.87	72.01	74.2	64.7	64.7	64.9	60.6	66.9	-	-
Au Recovery (%)	76.92	74.07	77.2	74.6	72.9	73.0	70.0	74.4	-	-
Cu Recovery (%)	85.47	83.45	86.3	84.2	83.8	82.9	80.5	83.5	-	-
Ni Recovery (%)	36.8	30.83	34.4	36.9	34.9	35.2	30.0	40.5	-	-
Concentrate Tonnes (t)	17,883	11,708	5,517	26,176	33,060	28,979	30,698	45,652	36,714	27,200
Pd (oz)	178,835	157,476	102,212	212,046	286,334	237,338	177,167	308,931	288,703	219,300
Pt (oz)	12,528	10,265	5,499	16,311	24,442	22,308	18,833	25,128	23,742	19,200
Au (oz)	12,487	8,314	4,624	15,921	20,092	17,238	14,308	25,679	23,536	16,000
Cu (lb)	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)	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 (%)	98.3	97.9	95.7	88.4	91.1	86.5	86.5	88.4	91.1	90.5

Notes: *The mine and mill were placed on temporary care and maintenance on October 22, 2008.
**Commercial production from the Roby underground mine began May 2006.

Source: North American Palladium – Lac des Iles

Table 17.2 Historical and Current Concentrate Production Quality

	2012	2011	2010	2008*	2007	2006	2005
Pd Grade (g/t)	311.0	418.4	576.6	258.8	269.5	250.4	179.5
Pt Grade (g/t)	22.0	27.3	31.0	19.9	23.0	23.5	19.1
Au Grade (g/t)	22.0	22.1	26.1	19.4	18.9	18.2	14.5
Cu Grade (%)	7.0	7.2	6.4	8.2	7.6	7.9	8.2
Ni Grade (%)	4.0	3.6	3.6	4.5	4.2	4.2	3.5
MgO Grade (%)	7.9	9.3	9.2	5.7	7.1	7.1	7.6

Notes: *The mine and mill were placed on temporary care and maintenance on October 22, 2008.
Source: North American Palladium – Lac des Iles

17.3 OPERATIONS

The processing plant has a nominal capacity of 15,000 t/d. The plant will operate on a batch basis due to the low mining feed rate.

Modifications to the grinding circuit are being made to improve recoveries to approximately 80%. The XPS metallurgical study on both Roby and Offset material indicates that both ores are compatible and may be processed together. It has also shown that a palladium recovery of 85% might be achieved. Metallurgical test work directed toward optimizing processing conditions continues at the mill.

17.4 CAPITAL EXPENDITURES

The existing processing plant has been restarted with ongoing maintenance and upgrading of equipment and facilities included in daily operations. No capital expenditures are currently envisaged to process Offset Zone material. Table 17.3 shows a list of the major equipment already installed and running in the mill that is being used for the process circuit.

Table 17.3 Major Equipment List

Major Equipment	Quantity	Description
Primary Crusher	1	1,372 mm by 2,235 mm Gyratory, 447 kW
Secondary Crusher	1	Standard Head HP800
Pebble Crusher	1	Short Head HP800, 298 kW
SAG Mill	1	9,144 mm by 4,270 mm, 6,400 kW
SAG Discharge Vibrating Screen	1	2,440 mm by 6,100 mm, 9.5 mm opening, 44.8 kW
Ball Mill	2	6,100 mm by 10,360 mm, 6,400 kW
Primary Hydrocyclone Cluster	2	10 hydrocyclones/cluster, 381 mm diameter
Tertiary Grinding	3	932 kW Vertimills™
Conditioning Tank	1	6,300 mm diameter by 5,200 mm height

table continues…

Major Equipment	Quantity	Description
Rougher Cells	2	50 m^3 tank cells, 75 kW
Scavenger Cells	14	130 m^3 tank cells, 150 kW
First Cleaner Flotation Cells	6	30 m^3 tank cells, 55.9 kW
Second Cleaner Flotation Cells	3	30 m^3 tank cells, 55.9 kW
Third Cleaner Flotation Cells	8	5 m^3 cells, 11.2 kw
Fourth Cleaner Flotation	2	1,219 mm diameter by 10,973 mm tall column cells
Rougher Column Cleaning	1	1,676 mm diameter by 10,973 mm tall column cell
Dewatering Thickener	1	High rate, 17,000 mm diameter
Pressure Filters	2	LAROX pressure filters, 10 plates/filter
Concentrate Load Out	1	1,500 t capacity

Source: North American Palladium – Lac des Iles

17.5 MANPOWER

The mill manpower requirements for the Offset Zone processing period will be essentially the same as for present processing operations. Manpower will operate the mill on the batch or continuous basis with only minor changes to overall staffing requirements. The total processing plant manpower complement, including staff and management, will be 83 employees as shown in Table 17.4. Processing plant operating personnel will work two 12-hour shifts.

Table 17.4 Process Plant Manpower

Position	Quantity
Process Plant Operations	
Mill Superintendent	1
Metallurgical Supervisor	1
Metallurgical Technician	1
Senior Mill Operations Supervisor	1
Shift Supervisors	3
Process Control Supervisors	2
A – Crew Operators	6
B – Crew Operators	6
C – Crew Operators	2
Total Process Plant Operations	**23**
Assay Laboratory	
Chief Assayer	1
Assayers	4
Laboratory Technicians	8
Total Assay Laboratory	**13**
Mill Maintenance	
Senior Mill Maintenance Supervisor	1

table continues…

Position	Quantity
Mill Maintenance Supervisor	2
Mill Electrical Supervisor	2
Maintenance Planner	2
Millwrights	17
Electricians	20
Instrumentation	3
Total Mill Maintenance	**47**
Total Processing Plant Manpower	**83**

Source: North American Palladium – Lac des Iles

17.6 OPERATING COSTS

The actual Lac des Iles Mine operating costs were submitted by NAP and include maintenance costs (by contractor), manpower, supplies, reagents, as well as similar operating costs for the tailings area. The annual operating costs per tonne milled are presented in Table 17.5 from 2009 (when the mill went into care and maintenance) to October 2012.

Table 17.5 Annual Operating Costs

	2012	2011	2010	2009*
Annual Operating Cost ($)	27,778,186	20,447,237	12,196,195	2,567,628
Annual Production (t)	2,063,260	1,689,781	649,649	n/a
Cost per Tonne Milled ($)	13.46	11.33	18.77	n/a

Notes: *The mine and mill were placed on temporary care and maintenance on October 22, 2008.

Source: North American Palladium – Lac des Iles

18.0 PROJECT INFRASTRUCTURE

18.1 LOCAL RESOURCE

Thunder Bay, with a population of 121,596 (2011 Census – www.thunderbay.ca), is the major service centre for northwestern Ontario and provides most of the services required by NAP's Lac des Iles Mine and its Offset Zone Phase I Project. This includes an airport with regular daily service to major Canadian cities, rail connections, and ocean access via the Great Lakes and St. Lawrence Seaway.

Most mine and mill consumables including fuel, cement, and propane are readily available in Thunder Bay. With the proximity of Lac des Iles Mine to Thunder Bay, there is access to experienced staff and personnel with good mining and processing expertise.

Most employees work on either a four-day-on/three-day-off, seven-day-on/seven-day-off or a 14-day-on/14-day-off shift schedule. Contract miners operate on a 21-day-in/14-day-out schedule.

18.2 SITE TOPOGRAPHY

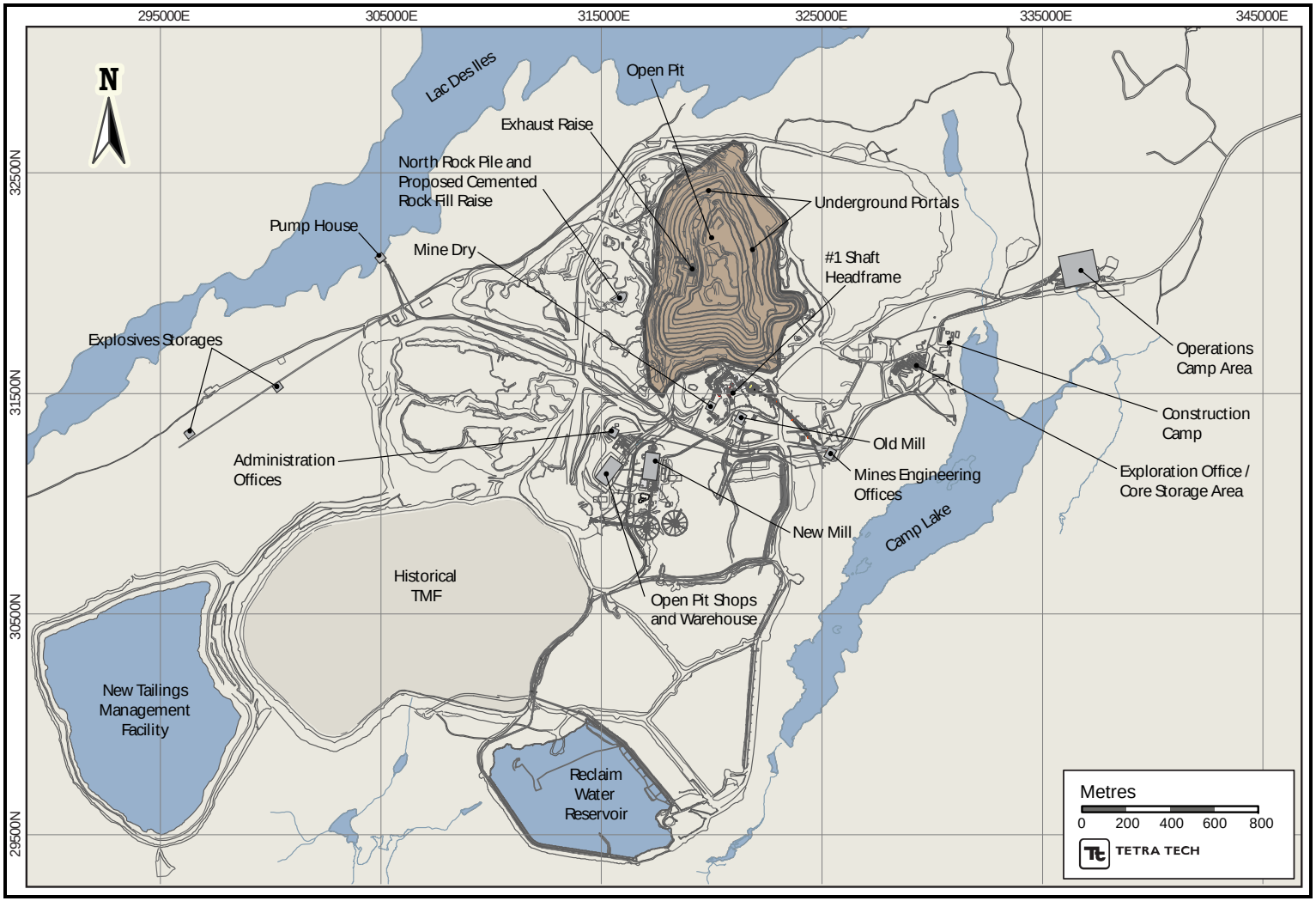
Lac des Iles Mine is located in northwestern Ontario which lies within the Superior Province of the Canadian Precambrian Shield. The topography of the site is favourable for the placement of facilities, being generally of low relief. Elevations on the Property range from 418 to 550 masl, exclusive of the open pit.

18.3 SITE OVERVIEW

A site plan is shown in Figure 18.1. It shows facilities such as the operations camp area, the old camp area, main office and tire shop, the old mill area, the new mill area, the open pit shops, the warehouse and operational offices, the old concentrator building, the open pit and stockpile area, the underground portal and related ventilation accesses, and the TMF.

New infrastructure related specifically to the Offset Zone and depicted on the site plan includes the No. 1 Shaft headframe, the hoist house, the compressor building, the mine dry, the substation and the surface load out facility, all constructed in the past two years to accommodate the future mine expansion. The reader is referred to Section 16.0 for further details on these new Offset Zone facilities.

Figure 18.1 Lac des Iles Mine Surface Site Plan Inclusive of Project Infrastructure 2012



Overall, the facilities are considered by LDIM to be adequate for the needs of the Lac des Iles Mine, as well as meet the needs of the future underground mining plan.

18.4 CAMP FACILITIES

In 2006, a 324-person operations camp and recreational complex was built in conjunction with the construction of the new mill. Once the Offset Zone Phase I construction activities wind down, this facility will be sufficient to accommodate the underground workforce.

In the interim, a construction camp was established in 2011, to accommodate added surface and underground construction workers during the peak construction period associated with readying the Offset Zone Phase I for production through the No. 1 Shaft facilities. The construction camp added approximately 200 spaces and will be decommissioned once the Lac des Iles Mine related workforce requirements are reduced.

To date these camp facilities have been found to be adequate for mine's needs and are considered by LDIM to meet the needs of the future underground mining plan.

18.5 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, as well as milling operations. The trucking contractor maintains a trans-shipping warehouse in Thunder Bay for Lac des Iles Mine material.

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 LDIM to meet the needs of the future underground mining plan.

18.6 ROADS

The Lac des Iles Mine site is approximately 135 km from the Thunder Bay International Airport by travelling approximately 116 km along paved highways with the majority of the distance on Provincial Highway 527 and a further distance of approximately 19 km along a gravel access road. The gravel access road is open to the public. It is maintained year round by Lac des Iles Mine using their personnel and equipment. The paved highway is maintained year round by the Ontario Ministry of Transportation personnel and equipment.

The route to the site has been in existence since the commencement of open pit mining in 1993 and to date has been found to be adequate for the mine's needs. All Offset Zone Phase I materials, including oversize mining equipment, have been

brought to site along this access route. This access is considered by LDIM to meet the needs of the future underground mining plan.

Once at the site, access is restricted to site personnel. On the mine site, all existing roads are of a crushed rock construction. Much of the roads in existence at the site were constructed to accommodate open pit haul trucks. There is no requirement related to the Offset Zone Phase I Project for the construction of any further roads on the site.

18.7 STOCKPILES

Waste dumps and ore 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 classified as non-acid generating.

The intent is to crush the waste rock stockpile and then use this as the source of the underground cemented rockfill. There is approximately 20 Mt of waste material in the surface stockpiles that can be used as cemented rock fill. Similarly, the plan is to place waste rock from the underground workings as backfill into the empty underground stopes, once a stope is mined out.

18.8 TAILINGS MANAGEMENT FACILITY

The Lac des Iles Mine 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.

Two TMFs are present on the Property, the West TMF, and the new South TMF. 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 TMF.

The new South TMF is expected to have sufficient capacity to hold all of the tailings generated until the end of the multiple Offset Zone phases. There are presently additional rock lifts being added to the tailings dams to increase the storage capacity. Currently, all wastes created by the milling process are deposited in this facility. Dams and dykes are located around and through the West and South TMFs. The South TMF dams are in the process of being raised by 5 m to increase capacity of the South TMF. The TMF was expanded in 2012 and the capital plan includes for three ensuing dam expansions over the next three years (stage 2 lifts) to meet all of the mine's long-term requirements.

The West TMF was closed upon reaching capacity and is undergoing progressive remediation.

All water within both TMFs is recycled 100% on site and treated before being discharged to the environment when needed.

18.9 SERVICES

Water and sewer services are supplied independently for each facility and are considered by LDIM to be adequate for current 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 the TMF areas. The reader is referred to Section 20.0 for more details around water usage and sewage disposal practices.

18.10 POWER

Electrical power is supplied by Hydro One via a 118 kV line to a two-main substation on-site. Site distribution is the responsibility of Lac des Iles Mine and consists of 13.8 V and 4,160 V overhead lines around the site. There is a services agreement with Hydro One in place. In June 2011, conditional approval was received for Lac des Iles Mine to add up to 6 MW, associated with the Offset Zone Project, to the Hydro One power grid.

18.11 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 the Offset Zone Phase I Project. As shown in Section 16.0, there will be emergency water storages established underground to capture any spring runoff and surface drainage generated by exceptional thunderstorm activity that makes its way into the open pit.

19.0 MARKET STUDIES AND CONTRACTS

NAP relies on a number of independent industry forecast organizations for market analysis and metal price forecasts for precious metals. These organizations include CPM Group, Johnson Matthey, Mitsui Global Precious Metals, GFMS, as well as a number of major banks with dedicated commodities market research groups. These organizations and market research groups are recognized worldwide as authorities in PGM markets. 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 general consensus of the leading forecasters is that palladium prices will rise, with palladium highlighted as having better prospects over other PGM metals. Their outlook for palladium over the next 10 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 automotive sector which consumes over 60% of global palladium production (for the manufacture of catalytic converters). Investment demand and fabricated demand for other industrial uses such as electronics, dental, and jewellery is also expected to increase in the long term.

The fundamentals that support the forecasters views that demand will increase are largely related to rising vehicle production in emerging economies (which are currently leading the growth in the automobile market), stricter emission controls that require the use of catalytic converters, technological advancements for palladium's use in diesel catalytic converters, and the recent growth in palladium exchange traded funds (ETF).

Supply, by contrast, has been essentially stagnant in the past several years and is forecast to remain tight in the next decade. Palladium supply is concentrated in a few locations worldwide (predominately Russia, South Africa and North America), with minimal new supply originating from those areas. Moreover, considering the recent supply disruptions and reduced output in South Africa, and a contraction of underlying output in Russia, supply is expected to be significantly challenged. Accordingly, the price of palladium is expected to remain strong in the next decade.

This study's base case metal prices for palladium and platinum (the two primary revenue metals) are based on current market consensus for these metals. The study assumes the following metal prices:

- $675 palladium
- $1,675 platinum

- $1,750 gold
- $3.50 copper
- $8.00 nickel
- a foreign exchange rate of 1.0.

These assumptions were also used in NAPs 2013 corporate budget, financial model (Section 22.0), and reserve estimates (Section 15.0). These prices are below the 2013 and longer-term (e.g. 5-year) consensus pricing for all pay metals, as shown in the accompanying tables. NAP has taken a prudent and slightly conservative view on metal price assumption for the current LOM of the Lac des Iles mining operation.

Table 19.1 to Table 19.6 show the price forecasts for palladium, platinum, gold, copper, and nickel from various independent forecasters. It should be noted that the base case palladium and platinum prices used in this report, relative to the forecasts identified here, are considered to be conservative as they are below these forecasts.

Table 19.1 Two Year Term Metal Price Forecasts

Commodity	Unit	2013 Forecasts				Annual Averages		
		Q1	Q2	Q3	Q4	2012	2013	2014
Palladium	$/oz	700	875	950	975	644	874	931
Platinum	$/oz	1,725	1,825	1,900	1,925	1,554	1,844	1,838
Gold	$/oz	1,825	1,875	1,925	1,950	1,670	1,895	1,806
Copper	$/lb	3.80	3.84	3.60	3.50	3.61	3.69	3.55
Nickel	$/lb	8.25	8.75	8.25	8.00	7.95	8.32	8.48

Source: TD Securities, January 17, 2013, “Commodities Trade Strategy” Research Report

Table 19.2 Long-term Metal Price Consensus for Palladium

Forecaster	2012 ($)	2013 ($)	2014 ($)	2015 ($)	2016 ($)	Long-term ($)
BMO	643	725	750	700	650	650
Canaccord Genuity	656	775	850	-	-	900
CIBC	610	700	900	1,000	800	700
Cormark	650	750	-	-	-	750
Credit Suisse	640	700	800	850	-	900
Deutsche Bank	679	750	800	900	1,000	1,000
Dundee	678	700	775	750	700	600
J.P.Morgan	635	700	881	-	-	1,000
Macquarie	746	1,019	950	-	-	800
Morgan Stanley	638	708	780	850	1,092	1,106
National Bank Financial	672	750	750	750	750	550
Paradigm	700	560	560	560	560	560
Raymond James	648	856	1,000	950	850	800

table continues…

Forecaster	2012 ($)	2013 ($)	2014 ($)	2015 ($)	2016 ($)	Long-term ($)
RBC	650	750	850	850	750	750
Scotiabank	644	725	750	750	700	650
Societe Generale	710	800	-	-	-	-
TD Securities	653	600	-	-	-	550
Average	660	740	815	810	785	765

Source: Street Analyst consensus, as compiled by Scotiabank, December 2012

Table 19.3 Long-term Metal Price Consensus for Platinum

Forecaster	2012	2013	2014	2015	2016	Long-term
BMO	1,565	1,650	1,650	1,650	1,650	1,650
Canaccord Genuity	1,565	1,700	1,725	-	-	1,800
CIBC	1,500	1,500	2,200	2,400	1,900	1,600
Cormark	1,713	1,750	-	-	-	1,700
Credit Suisse	1,560	1,710	1,800	1,850	-	1,900
Deutsche Bank	1,581	1,700	1,800	1,850	1,900	2,000
Dundee	1,678	1,900	2,100	2,000	1,900	1,600
J.P.Morgan	1,516	1,481	1,750	-	-	1,800
Macquarie	1,548	1,750	1,850	1,900	1,950	1,800
Morgan Stanley	1,538	1,715	1,785	1,865	1,941	2,212
National Bank Financial	1,548	1,600	1,600	1,600	1,600	1,300
Paradigm	1,650	1,750	1,750	1,750	1,750	1,750
Raymond James	1,577	1,788	2,000	1,900	1,800	1,600
RBC	1,550	1,700	1,800	1,750	1,700	1,700
Scotiabank	1,565	1,650	1,650	-	-	1,400
Societe Generale	1,600	1,650	-	-	-	-
TD Securities	1,578	1,750	1,750	1,750	1,750	1,750
Wilson HTM	1,525	1,750	1,800	1,900	2,000	2,000
Average	1,575	1,695	1,815	1,860	1,820	1,740

Source: Street Analyst consensus, as compiled by Scotiabank, December 2012

Table 19.4 Long-term Metal Price Consensus for Gold

Forecaster	2012	2013	2014	2015	2016	Long-term
BAML	1,690	1,850	2,090	-	-	1,620
BMO	1,689	1,950	2,000	1,800	1,600	1,400
Canaccord Genuity	1,683	1,850	1,950	-	-	1,600
CIBC	1,700	2,000	1,500	1,500	1,500	1,500
Clarus	1,500	1,500	1,500	1,500	1,500	1,500
Cormark	1,690	1,800	1,600	1,400	1,400	1,400
Credit Suisse	1,680	1,840	1,750	1,500	-	1,300
Deutsche Bank	1,726	2,050	1,800	1,606	1,413	1,025

table continues…

Forecaster	2012	2013	2014	2015	2016	Long-term
Dundee	1,636	1,707	-	-	-	1,375
GMP	1,613	1,575	1,575	1,575	1,575	1,575
HSBC	1,761	1,775	1,750	-	-	-
Jennings	1,695	1,900	2,000	1,900	1,300	1,300
J.P.Morgan	1,670	1,775	1,819	-	-	1,300
Macquarie	1,681	1,876	1,713	1,394	1,300	950
Morgan Stanley	1,683	1,853	1,800	1,750	1,550	1,350
National Bank Financial	1,689	1,800	1,800	1,800	1,800	1,400
Paradigm	1,600	1,400	1,400	1,400	1,400	1,400
RBC	1,650	1,700	1,500	1,300	1,200	1,200
Raymond James	1,680	1,780	1,800	1,650	1,450	1,200
Salman	1,657	1,650	1,450	1,300	-	1,250
Scotiabank	1,677	1,800	1,800	1,800	1,600	1,300
Societe Generale	1,650	1,600	-	-	-	-
Stonecap	1,700	1,700	1,600	1,500	1,350	1,350
TD Securities	1,651	1,750	-	-	-	1,500
Wilson HTM	1,751	1,798	1,591	1,440	1,335	1,025
Average	1,670	1,770	1,720	1,560	1,455	1,340

Source: Street Analyst consensus, as compiled by Scotiabank, December 2012

Table 19.5 Long-term Metal Price Consensus for Copper

Forecaster	2012	2013	2014	2015	2016	Long-term
BAML	3.68	3.54	3.40	-	-	-3.08
BMO	3.62	3.60	3.50	3.50	2.75	2.75
Canaccord Genuity	3.61	3.50	3.50	3.25	3.00	2.75
CIBC	3.55	3.75	3.50	2.50	2.50	2.50
Clarus		3.50	3.25	3.00	2.40	2.40
Cormark	3.71	4.00	4.00	3.50	3.25	2.50
Credit Suisse	3.60	3.63	3.40	3.18	-	2.49
Deutsche Bank	3.58	3.63	3.40	3.23	3.06	2.77
Dundee	3.65	4.00	4.05	3.50	3.50	2.75
HSBC	3.86	3.40	3.18	-	-	-
J.P.Morgan	3.93	3.91	3.86	3.86	-	2.49
Macquarie	3.64	3.66	3.48	3.43	3.40	2.95
Morgan Stanley	3.63	3.90	3.70	3.40	3.20	2.77
National Bank Financial	3.56	3.50	3.25	2.60	2.60	2.60
Paradigm	3.65	4.25	4.00	3.50	-	2.75
RBC	3.55	3.75	3.75	3.75	4.25	2.75
Raymond James	3.64	3.75	4.00	4.00	3.67	3.00
Salman	3.52	3.36	3.34	3.04	-	2.75
Scotiabank	3.70	4.29	4.80	4.95	2.75	2.75

table continues…

Forecaster	2012	2013	2014	2015	2016	Long-term
TD Securities	3.65	3.90	3.50	3.50	-	2.75
Wilson HTM	3.86	3.94	3.70	3.46	3.30	2.77
Average	3.65	3.75	3.65	3.45	3.10	2.70

Source: Street Analyst consensus, as compiled by Scotiabank, December 2012

Table 19.6 Long-term Metal Price Consensus for Nickel

Forecaster	2012	2013	2014	2015	2016	Long-term
BAML	8.14	8.28	8.17	-	-	9.50
BMO	8.02	8.50	8.50	8.50	8.50	8.50
Canaccord Genuity	8.09	8.25	8.50	-	-	9.00
CIBC	8.00	8.50	8.50	7.50	7.50	7.50
Clarus	-	9.00	8.50	8.50	8.50	8.50
Cormark	8.11	8.00	7.50	7.50	7.50	7.50
Credit Suisse	8.07	8.56	9.07	9.53		9.07
Deutsche Bank	8.29	8.85	9.07	8.85	8.62	8.17
Dundee	8.25	9.25	9.30	8.50	8.50	8.00
HSBC	9.07	8.39	9.07	-	-	7.00
J.P.Morgan	8.12	8.73	10.04	-	-	8.16
Macquarie	7.91	8.28	9.58	11.00	11.00	11.00
Morgan Stanley	8.07	8.30	9.00	9.50	10.50	11.89
National Bank Financial	7.96	8.00	8.00	8.00	8.00	8.00
Paradigm	8.25	8.50	9.00	9.00	9.00	9.00
RBC	8.00	8.75	9.50	10.00	11.00	9.50
Raymond James	9.25	10.00	10.00	10.00	9.50	9.00
Salman	7.72	7.13	7.17	8.39	-	8.24
Scotiabank	8.27	10.50	12.50	13.00	10.50	10.50
Societe Generale	8.79	9.53	10.43	11.20	11.11	7.98
TD Securities	8.30	9.00	8.50	8.50	8.50	8.50
Wilson HTM	8.38	9.58	9.80	10.01	10.04	7.71
Average	8.25	8.70	9.10	9.30	9.25	8.75

Source: Street Analyst consensus, as compiled by Scotiabank, December 2012

19.1 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 Lac des Iles Mine claims and leases to LDIM in exchange for a NSR royalty.

According to management, NAP is required to pay SPG and Sheridan 5% of the NSR at Lac des Iles Mine until the expiration of the Lac des Iles Mine leases. The term NSR is defined in the royalty agreement as the net proceeds receivable by LDIM from the production and sale of concentrates from Lac des Iles Mine after deducting: the costs of sampling, assaying, transportation and insuring of

concentrate; smelting, processing and refining charges and penalties (excluding Lac des Iles Mine milling costs). All mining operations at Lac des Iles Mine are on the mining leases covered by the royalty agreement.

19.2 SMELTER CONTRACT

19.2.1 VALE

During the second quarter of 2012, NAP entered into an agreement with Vale Canada Limited (Vale) for the smelting of Lac des Iles Mine concentrate. Vale's smelter is located in Sudbury, Ontario. Shipments are made by truck, with the material delivered in the form of friable filter cake. The agreement with Vale includes an advance payment option to NAP (less processing charges). Accountabilities for payable metals are within industry norms. The agreement expires June 30, 2015.

19.2.2 AURUBIS

During the second quarter of 2012, NAP entered into an agreement with Aurubis AG (Aurubis) for the smelting of Lac des Iles Mine concentrate. Aurubis' smelter is located in Hamburg, Germany. The agreement includes an advanced payment option to NAP (less treatment and refining charges). Accountabilities for payable metals are within industry norms. The agreement expires December 31, 2013.

20.0 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 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 Lac des Iles Mine on the surrounding ecosystem.

20.2 WASTE AND TAILINGS DISPOSAL, SITE MONITORING, AND WATER MANAGEMENT

The Lac des Iles Mine 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.

Two TMFs are present on the Property, the West TMF, and the new South TMF. The TMFs are located adjacent to one another, southwest of the open pit. Tailings from the current and future mining are being disposed of in the South TMF.

Currently, all wastes created by the milling process are deposited in this facility. Dams and dykes are located around and through the West 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. The tailings area is monitored according to industrial sewage works requirements set out by the Ministry of Environment.

The West TMF has been closed upon reaching capacity and is undergoing progressive remediation.

All water within both TMFs is recycled 100% on-site and treated before being discharged to the environment when needed. There have been no recorded 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. It is 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. No new material is planned to be added to the existing waste rock dumps on the east and west sides of the Roby pit. Waste rock is non-acid generating.

20.3 PERMITTING

LDIM 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. Lac des Iles Mine remains current with permitting and licensing requirements (Table 20.1).

Table 20.1 Summary of Permits

Approval	Reason for Approval	Expiration Date
Air CoA (Amended) 0112-889QZX	Approval for site wide air emissions	Does not expire Amended 2010 Comprehensive CofA
Industrial Sewage Works CofA 4-0048-92-936	Approves the TMF; the discharge of sanitary sewage to the facility, and landfill.	Amendment submitted to MOE
Municipal and Private Sewage Works CoA 6806-4KTPB9	Approves sewage treatment plant at operation camp	Does not expire
Land Use Permit LUP SPR1056	Approves the hydro line to Silver Falls	Does not expire
Land Use Permit LUP SPR1055	Approves the operations camp and parking lot	Does not expire

table continues…

Approval	Reason for Approval	Expiration Date
Land Use Permit LUP SPR1054	Approves the expansion TMF	*Public Lands Act*, Does not expire
Permit to take water 6WHP2Y	Approved withdraw of 908 L/min or 1,307,520 L/d	Amended, expires 2018
Permit to take water 6835-725PXK	Approved withdraw of 24,310 L/min or 35,006,400 L/d	Expires April 11, 2013 Process of Amending
Permit to take water 1607-7S8LCB	Approved withdraw 2,700 L/d	Expires May 20, 2019
Lac Des Iles Mine 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
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 Lac des Iles 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 LDIM 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 were 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.

In addition to the above, LDIM has been working with the local on-reserve school in the development and delivery of a literacy initiative designed to support the students and staff. LDIM has also supplied requested materials to assist in the delivery of existing core programming.

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.

20.5 CLOSURE PLAN

LDIM is responsible for all costs of closure and reclamation at the Lac des Iles Mine. LDIM, in conjunction with the Ontario Ministry of Northern Development and Mines (MNDM), has established a trust fund pursuant to LDIM's mine closure plan for eventual cleanup and restoration of these sites at closure. The mine closure plan calls for a total amount of $14,054,660 million to be accumulated in the fund. Commencing in February 2001, the trust fund, controlled by MNDM, started to accumulate through monthly deposits of $100,000. As of August 2012, LDIM reported that it had $12,973,600 million on deposit with MNDM.

Recently, a closure plan amendment had been submitted to MNDM to include new developments within the footprint, including the shaft. As well as the inclusion of LDIMs plan to expand mining operations by developing the north expansion of the existing pit and the Baker Zone.

21.0 CAPITAL AND OPERATING COSTS

21.1 SUMMARY

The total initial capital cost is $76 million. The total sustaining capital cost is $35 million for a total capital estimate of $111 million.

The capital cost estimates are based on budget pricing from suppliers, consultants, contractors, and a review of current capital spending on site. Smaller equipment and facilities component costs were factored based on industry norms for the type of facility being constructed and, where possible, adjusted to reflect local conditions.

Construction and installation labour rates are based on contractor actual costs in the region, and country, for similar types of work.

All cost estimates are in 2013 constant Canadian dollars.

The Phase I Offset Zone capital costs are itemized as initial capital cost and sustaining capital cost and are summarized in Table 21.1.

Table 21.1 Capital Cost Summary ($ million)

Area	Initial Capital (2013 to 2014)	Sustaining Capital (2013 to 2018)	Total (2013 to 2018)
Initial Capital	75.6	0.0	75.6
Sustaining Capital	0.0	35.2	35.2
Total	**75.6**	**35.2**	**110.8**

The total operating costs for the Offset Zone are $427 million. The total cost and unit costs ($/t) are summarized in Table 21.2.

Table 21.2 Phase I Operating Cost Summary

Area	Cost ($ million)	Cost (%)	Unit Cost ($/t mined)
Surface	65.9	15	8.51
Mine	178.6	42	23.07
Mill	118.5	28	15.31
G&A Mine	64.0	15	8.27
Total	**427.0**	**100**	**55.16**

21.2 CAPITAL COSTS

21.2.1 INITIAL CAPITAL COSTS

The initial capital period of 2013 and 2014 has an estimated total capital cost of $75.6 million. The initial capital costs, contingency not included, are:

- development ($37.4 million)
- underground maintenance and parts ($11.6 million)
- surface and underground construction ($17.1 million)
- procurement ($2.2 million)
- underground equipment ($0.04 million)
- electrical and process control ($2.5 million)
- engineering ($1.0 million)
- Owner's costs and indirects ($3.8 million).

A detailed breakdown of the initial capital cost is shown in Table 21.3.

Table 21.3 Initial Capital Cost ($ million)

Area	2013	2014	Total
Development			
Ramp	7.7		7.7
Level Access	11.8	8.7	20.5
Raising	4.7	4.5	9.2
Subtotal Development	**24.2**	**13.2**	**37.4**
Underground Maintenance and Parts	11.6	0.0	11.6
Surface and Underground Construction			
Surface Construction	2.0	0.0	2.0
Shaft Sinking	9.8	0.0	9.8
Shaft Infrastructure	2.2	0.0	2.2
Underground Construction	3.1	0.0	3.1
Subtotal Surface and Underground Construction	**17.1**	**0.0**	**17.1**
Procurement	2.2	0.0	2.2
Underground Equipment	0.04	0.0	0.04
Electrical and Process Control	2.5	0.0	2.5
Engineering	1.0	0.0	1.0
Owner's Costs and Indirects	3.8	0.0	3.8
Total	**62.4**	**13.2**	**75.6**

21.2.2 SUSTAINING CAPITAL COST

The sustaining capital period from 2013 through 2018 has a cost of $35.2 million. It is comprised of:

- exploration ($6.6 million)
- mine and maintenance ($14.4 million)
- mill ($1.6 million)
- TMF (Phase I) ($12.5 million)
- administration ($0.05 million).

Sustaining capital costs are estimated to cover major repairs and replacement of mine equipment, in order to maintain continuous mine production levels. A summary of the sustaining capital cost is outlined in Table 21.4.

Table 21.4 Sustaining Capital Summary ($ million)

Area	2013	2014	2015	2016	2017	2018	Total
Exploration	3.4	3.2	0.0	0.0	0.0	0.0	6.6
Mine and Maintenance	5.8	2.6	2.0	2.0	2.0	0.0	14.4
Mill	0.6	1.0	0.0	0.0	0.0	0.0	1.6
TMF (Phase I)	6.5	3.0	3.0	0.0	0.0	0.0	12.5
Administration	0.05	0.0	0.0	0.0	0.0	0.0	0.05
Total	**16.4**	**9.8**	**5.0**	**2.0**	**2.0**	**0.0**	**35.2**

21.3 OPERATING COSTS

The Phase I operating costs have been categorized and summarized in Table 21.5.

Table 21.5 Phase I Operating Cost Summary

Area	Cost ($ million)	Cost (%)	Unit Cost (/t mined)
Surface	65.9	15	8.51
Mine	178.6	42	23.07
Mill	118.5	28	15.31
G&A Mine	64.0	15	8.27
Total	**427.0**	**100**	**55.16**

The costs for consumables are based on prices provided by suppliers. The cost of power is based on rates currently charged to the Lac des Iles Mine, which is supplied from the Ontario provincial power grid by a 65 km transmission line owned by LDIM.

Labour costs for the operating period are based on the manpower schedules presented for each department and the associated labour costs. Labour costs are based on Lac des Iles Mine rates for each category of employee and the current collective agreement, and also include the benefits component.

The total Phase I operating cost is $55.16/t mined. The average surface cost is $8.51/t mined, the average mine cost is $23.07/t mined, the average mill cost is $15.31/t mined, and the average G&A cost is $8.27/t mined.

A summary of the mine operating costs by activity is shown in Table 21.6 and the unit cost per tonne is shown in Table 21.7.

Table 21.6 Mine Operating Costs by Activity ($ million)

Area	2013	2014	2015	2016	2017	2018	Total
Surface Operations							
Load	0.08	0.10	0.11	0.11	0.11	0.10	0.63
Haul	0.16	0.19	0.21	0.21	0.21	0.19	1.18
Auxiliary Equipment	0.55	0.66	0.74	0.74	0.74	0.67	4.10
Services	8.11	9.72	10.07	9.77	9.77	8.81	56.25
General	0.50	0.60	0.67	0.67	0.67	0.61	3.73
Subtotal Surface Operations	9.40	11.26	11.82	11.51	11.51	10.38	65.88
Underground Mine							
General	6.92	5.76	5.76	4.61	3.69	2.95	29.68
Sill/Level Development	5.15	11.19	10.17	0.00	0.00	0.00	26.52
Crusher/Conveyor	0.05	0.33	0.34	0.41	0.41	0.41	1.95
Production	9.34	7.89	7.36	6.58	6.58	6.58	44.32
Services	2.71	2.86	2.86	2.86	2.86	2.86	17.03
Diamond Drilling	6.24	6.24	0	0.00	0.00	0.00	12.48
Maintenance	11.11	7.40	7.40	7.40	7.40	5.92	46.62
Subtotal Underground Mine	41.52	41.67	33.90	21.86	20.93	18.72	178.60
Mill							
Mill Process	17.44	20.89	21.64	20.04	21.06	17.44	118.51
G&A							
Mine	2.88	3.33	3.77	3.77	3.77	3.40	20.93
Non-mine	8.18	8.28	7.51	7.51	6.76	4.87	43.11
Subtotal G&A	11.06	11.61	11.28	11.28	10.53	8.28	64.04
Total	**79.42**	**85.43**	**78.64**	**64.69**	**64.03**	**54.82**	**427.03**

Table 21.7 Mine Operating Unit Costs by Activity ($/t)

	2013	2014	2015	2016	2017	2018	Total
Surface Operations							
Load	0.07	0.07	0.08	0.09	0.09	0.09	0.08
Haul	0.13	0.13	0.15	0.17	0.17	0.17	0.15
Auxiliary Equipment	0.45	0.45	0.52	0.59	0.59	0.59	0.53
Services	6.72	6.72	7.04	7.75	7.75	7.75	7.27
General	0.41	0.41	0.47	0.54	0.54	0.54	0.48
Subtotal Surface	7.78	7.78	8.26	9.14	9.14	9.14	8.51
Underground Mine							
General	5.73	3.98	4.03	3.66	2.93	2.60	3.83
Sill/Level Development	4.26	7.73	7.11	0.00	0.00	0.00	3.43
Crusher/Conveyor	0.04	0.23	0.24	0.32	0.32	0.36	0.25
Production	7.73	5.46	5.15	5.22	5.22	5.79	0.73
Services	2.24	1.98	2.00	2.27	2.27	2.52	2.20
Diamond Drilling	5.17	4.31	0.00	0.00	0.00	0.00	1.61
Maintenance	9.20	5.11	5.17	5.87	5.87	5.21	6.02
Subtotal Underground	34.37	28.80	23.70	17.34	16.61	16.48	23.07
Mill							
Mill Process	14.44	14.44	15.14	15.91	16.72	15.35	15.31
G&A							
Mine	2.38	2.30	2.64	3.00	3.00	3.00	2.70
Non-mine	6.77	5.72	5.25	5.96	5.36	4.29	5.57
Subtotal G&A	9.15	8.02	7.89	8.96	8.36	7.29	8.27
Total	**65.74**	**59.04**	**54.99**	**51.35**	**50.83**	**48.26**	**55.16**

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

An economic evaluation of the Lac Des Iles Mine was prepared by LDIM based on a pre-tax financial model and audited by Tetra Tech. For Phase I Offset Zone's 7.7 Mt reserve the financial model was based on the following:

- The base case metal prices for palladium and platinum are based on current market prices:
  - The study assumes the following metal prices: US$675/oz palladium, US$1,675/oz platinum, US$1,750/oz gold, US$3.50/lb copper and US$8.00/lb nickel (Table 22.1).
  - These assumptions were also used in the Lac Des Iles Mine 2013 corporate budget, financial model, and reserve estimates.
  - These prices are below 2013 and longer-term consensus pricing for all pay metals.
  - LDIM has taken a prudent and slightly conservative view on metal price assumption for Phase I of the mining operation.
- Lac Des Iles has used an exchange rate of CDN$1.00 = US$1.00 for this study (Table 22.2).

Table 22.1 Base Case Metal Prices used in the Financial Model

Pd (US$/oz)	Pt (US$/oz)	Au (US$/oz)	Cu (US$/lb)	Ni (US$/lb)
675	1,675	1,750	3.50	8.00

Table 22.2 Foreign Exchange used in the Financial Model

Currency Exchange	Rate
CDN$ = US$	1.00

22.2 PRINCIPAL ASSUMPTIONS

For the purpose of this study's financial analysis, LDIM applied the following assumptions to this phase of the LDI mine expansion:

- All costs, revenues, and financial evaluations are calculated and presented in Canadian dollars.
- The Lac Des Iles Phase I is six years.
- A detailed mine production schedule, tonnes, and grades, was included in the financial model.
- An average of 130,000 to 140,000 oz of palladium will be produced annually.
- The mill recovery rates and grades were provided by LDIM and reviewed by the technical expert. The projected recovery rates were based on overall metal recovery rates and test results released in independent test labs' reports available at the time of writing the report.
- The initial capital cost for Phase I of the Lac Des Iles Mine is $76 million.
- The sustaining capital cost for the Lac Des Iles Mine is $35 million.
- The TMF sustaining capital estimates are included in the financial model.
- A mine closure cost was not included in the evaluation.
- A salvage value was not included in the evaluation.
- Smelting and refining terms, and transportation costs are based on contracts with Vale.
- Payable metals for palladium, platinum, gold, copper, and nickel are based on confidential agreements between NAP and smelters.
- A NSR royalty of 5% is payable to SPG and John Sheridan.
- The discount rate of 5% was established for use in determining the NPVs of the Lac Des Iles Mine. The discount rate of 5% is consistent with that used at other comparable mines producing base metals, also at similar stages of production.
- No remedial measures were assumed required to meet proposed production rates and recoveries used in the NPV calculations and the economic analysis.

The financial model can be used to evaluate the sensitivity of the Lac Des Iles Mine to various inputs. The ability to vary these inputs allows the impact of updated field results to be examined quickly and the effects on the Lac Des Iles Mine determined.

22.3 FINANCIAL MODEL

The mine production schedule has been incorporated into the 100% equity financial model to develop annual recovered metal production from the relationships of tonnage milled, head grades, and recoveries.

The mineralized material tonnages, grades, and metal production are tabulated in Table 22.3.

Table 22.3 Metal Production from Lac Des Iles

Item	Unit	Value
Total Material to Mill	t	7,741,000
Average Annual Material to Mill	t	1,290,167
Phase I	years	6
NSR		
Average NSR	$/t mined	96.39
Average Grade		
Pd	g/t	4.30
Pt	g/t	0.30
Au	g/t	0.30
Cu	%	0.08
Ni	%	0.11
Total Mill Production		
Pd	oz	860,107
Pt	oz	53,764
Au	oz	61,663
Cu	lb	12,415,097
Ni	lb	8,170,127
Average Annual Mill Production		
Pd	oz	143,351
Pt	oz	8,961
Au	oz	10,277
Cu	lb	2,069,183
Ni	lb	1,361,688

Gross metal revenue was determined from mill recovered metal grades and metal prices. The gross revenue was determined to be $800 million or $103.40/t mined.

The smelting, refining, and transportation costs are $54 million or $7.01/t mined.

NSR revenue is gross revenue less smelting, refining, and transportation costs. The net NSR revenue was determined to be $746 million or $96.39/t mined.

A 5% royalty, per the SPG agreement, was applied to the NSR and was determined to be $37 million or $4.82/t mined.

Net revenue, determined as NSR revenue less royalty, was $709 million or $91.57/t mined.

Unit operating costs for surface, mining, milling, and G&A were applied to annual mined tonnages to determine the overall mine site operating cost, which has been deducted from the net revenue to derive annual operating margin. The operating

costs were determined to be $427 million or $55.16/t mined. The operating margin was determined to be $282 million or $36.41/t mined.

Initial capital and sustaining capital were considered in the year spent. Initial capital development consisting of ramps, level accesses, and raises was determined to be $76 million over Phase I. Sustaining capital for mining, process and infrastructure was determined to be $35 million.

Pre-tax cash flow, determined as net revenue less operating and capital, was estimated as $171 million. The Lac Des Iles Mine has:

- a pre-tax NPV of $138 million at a discount rate of 5%
- an IRR of 63.7%
- a payback period of 2.69 years or 32 months.

The tax calculation performed by LDIM indicates no taxes are payable during Phase I. Therefore the post-tax results are the same as the pre-tax results.

A summary of Phase I financials for the Lac Des Iles Mine is shown in Table 22.4.

Table 22.4 Summary of Phase I Financial Results

	Unit	Value	$/t mined
NPV (5%)	$ million	138	-
IRR	%	63.7	-
Payback	years	2.69	-
Gross Revenue	$ million	800	103.40
Smelting, Refining and Transportation	$ million	(54)	(7.01)
Net NSR	$ million	746	96.39
Royalty	$ million	(37)	(4.82)
Net Revenue	$ million	709	91.57
Operating Cost	$ million	(427)	(55.16)
Operating Margin	$ million	282	36.41
Capital Cost	$ million	(111)	(14.32)
Pre-tax Cash Flow	$ million	171	22.09
Taxes	$ million	0	0.00
Post-tax Cash Flow	$ million	171	22.09

The average cost per ounce of palladium (C1) is US$313 when by-product credits for platinum, gold, copper, and nickel are applied.

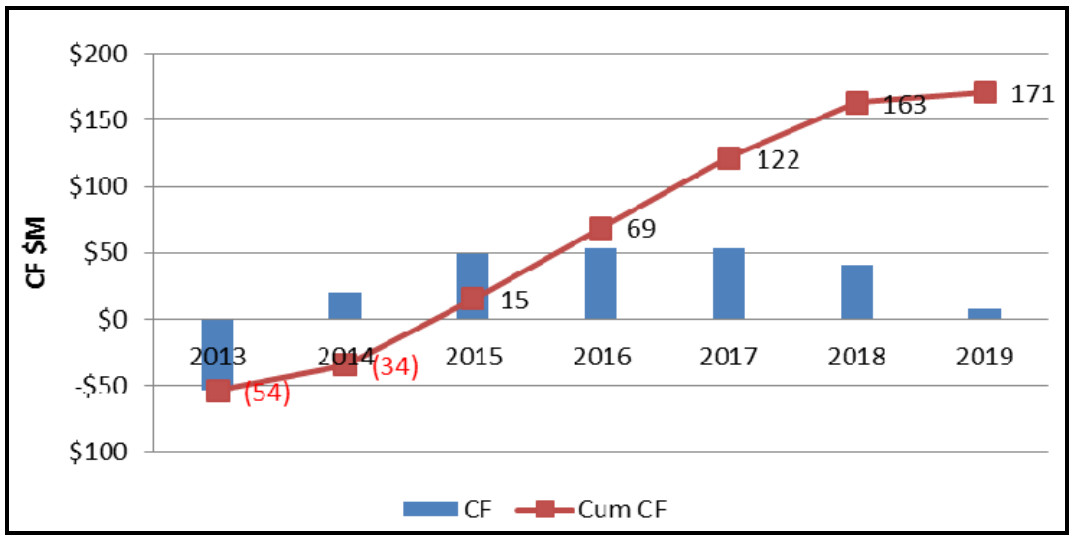
A breakdown of the gross revenue by metal is shown in Table 22.5.

Table 22.5 Gross Revenue Breakdown by Metal

Metal	Gross Revenue ($ million)	Percentage (%)
Pd	525	66
Pt	81	10
Au	96	12
Cu	39	5
Ni	59	7
Total	**800**	**100**

The undiscounted annual cash flows are illustrated in Figure 22.1.

Figure 22.1 Annual Undiscounted Cash Flow



22.3.1 METAL PRICE SCENARIOS

The average metal prices for three cases were identified with their corresponding NPVs at discount rates of 8%, 5%, and 0%. The results are shown in Table 22.6.

Table 22.6 Pre-tax NPV by Metal Price Scenario

Case	Average Metal Prices					NPV ($ million)		
	Pd (US$/oz)	Pt (US$/oz)	Au (US$/oz)	Cu (US$/lb)	Ni (US$/lb)	@ 8%	@ 5%	@ 0%
Base +20%	810	2,010	2,100	4.20	9.60	248	273	323
Base +10%	743	1,843	1,925	3.85	8.80	185	206	247
Base	675	1,675	1,750	3.50	8.00	121	138	171
Base -10%	608	1,508	1,575	3.15	7.20	58	71	95
Base -20%	540	1,340	1,400	2.80	6.40	(5)	3	19

22.3.2 *SENSITIVITY ANALYSIS*

NPV and IRR sensitivity analyses were carried out on the following parameters:

- foreign exchange
- palladium price
- palladium grade
- operating cost
- capital cost.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR.

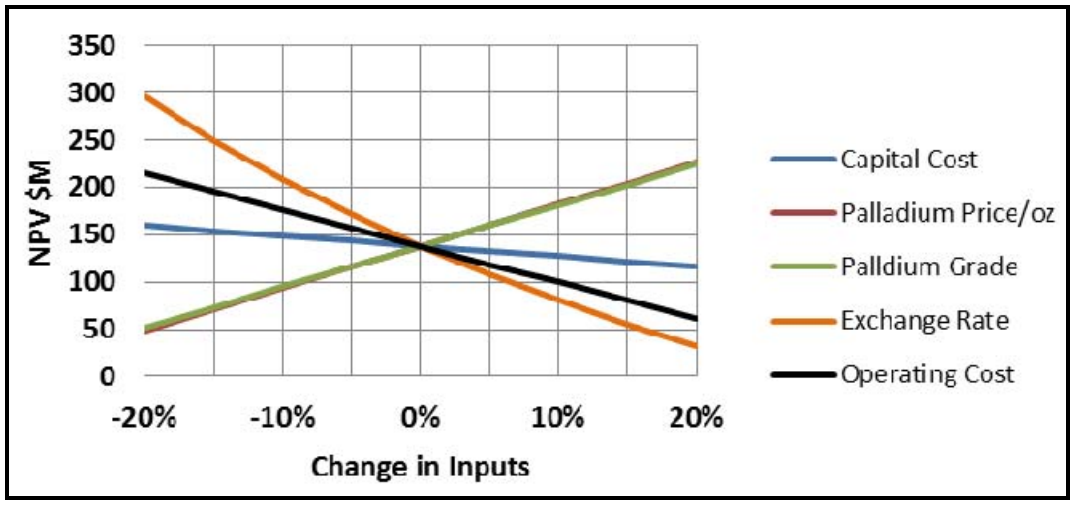
Table 22.7 shows the corresponding sensitivity parameters and the resulting NPVs.

Table 22.7 Sensitivity Parameters and NPV at 5% ($ million)

Parameter	Foreign Exchange	Palladium Price	Palladium Grade	Operating Cost	Capital Cost
-20%	295	49	52	215	160
-10%	208	93	95	176	149
Base	138	138	138	138	138
+10%	81	182	181	99	127
+20%	33	227	224	61	116

Figure 22.2 illustrates that the Lac Des Iles Mine NPV is most sensitive to the exchange rate, and in decreasing order, palladium price, palladium grade, operating cost, and capital cost.

Figure 22.2 NPV Sensitivity Analysis



Note: Palladium price and palladium grade results are almost identical and show as one line on the graph.

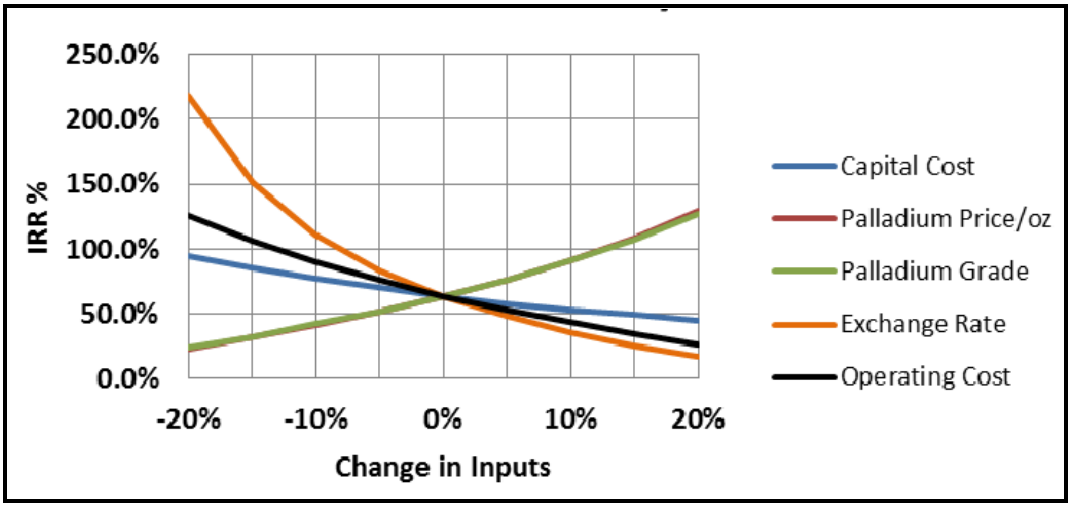
Table 22.8 shows the corresponding sensitivity parameters and the resulting IRRs.

Table 22.8 Sensitivity Parameters and IRR (%)

Parameter	Foreign Exchange	Palladium Price	Palladium Grade	Operating Cost	Capital Cost
-20%	218.8	23.0	24.0	126.4	95.1
-10%	110.7	41.5	42.1	90.0	77.0
Base	63.7	63.7	63.7	63.7	63.7
+10%	36.0	91.6	90.7	43.3	53.4
+20%	16.9	129.3	126.7	26.7	45.2

Figure 22.3 illustrates that the Lac Des Iles Mine IRR is most sensitive to the exchange rate, and in decreasing order, palladium price, palladium grade, operating cost, and capital cost.

Figure 22.3 IRR Sensitivity Analysis



Note: Palladium price and palladium grade results are almost identical and show as one line on the graph.

22.4 TAX CONSIDERATIONS

LDIM provided a tax model that accurately predicts the post-tax cash flow.

The tax calculation indicated no taxes were payable during the LOM, therefore the post-tax cash flow, NPV, and IRR are the same as the pre-tax values.

The following analysis highlights the economics of the cash flow model if taxes are considered. This includes federal and provincial corporate income taxes, the federal and provincial Harmonized Sales Tax (HST), Ontario mining tax and municipal property and educational taxes when labelled “post-tax".

According to currently enacted legislation, the combined anticipated federal and provincial income tax rate and mining tax rate, at the time when production is expected to commence, will be approximately 30%. The federal and provincial income tax components are expected to be 15% and 10%, respectively. The “effective” Ontario mining tax rate is expected to be approximately 5%, after taking into account various deductions including a processing allowance permitted under the taxing statute.

The model has applied the tax rate to taxable income as computed under the relevant statute. Generally deductions are allowed for operating expenses and capital development and exploration. The post-tax cash flow model considers income tax pools currently available to NAP and LDIM. Income tax pools such as non-capital losses of approximately $209 million, capital costs allowance (CCA) and cumulative eligible capital of $119 million, Canadian exploration expenses (CEE) and Canadian development expenses (CDE) of $263 million by NAP and LDIM are available as tax deductions to the Project. CEE are generally exploration expenses

incurred to determine the existence of a mineral resource in Canada while CDE are generally the cost to acquire a mining interest or development costs relating to a mine after it has come into production in reasonable commercial quantities. Where tax pool deductions are limited, the cash flow model assumes LDIM will claim maximum deductions to generate non-capital losses which are assumed to be utilized which maximizes the present value of such pools.

An operator of a mine in Ontario is subject to a 10% tax on profit which is generally computed as gross value of production less permitted mine operating expenses, depreciation, and development allowances. No deduction is allowed for interest expense or royalties. A processing allowance is permitted which given the level of processing expenditures for LDIM lowers its effective tax rate to approximately 5%.

All goods and services are subject to HST at a rate of 13%. However, the HST is anticipated to be refundable to NAP and is therefore not included in the analysis.

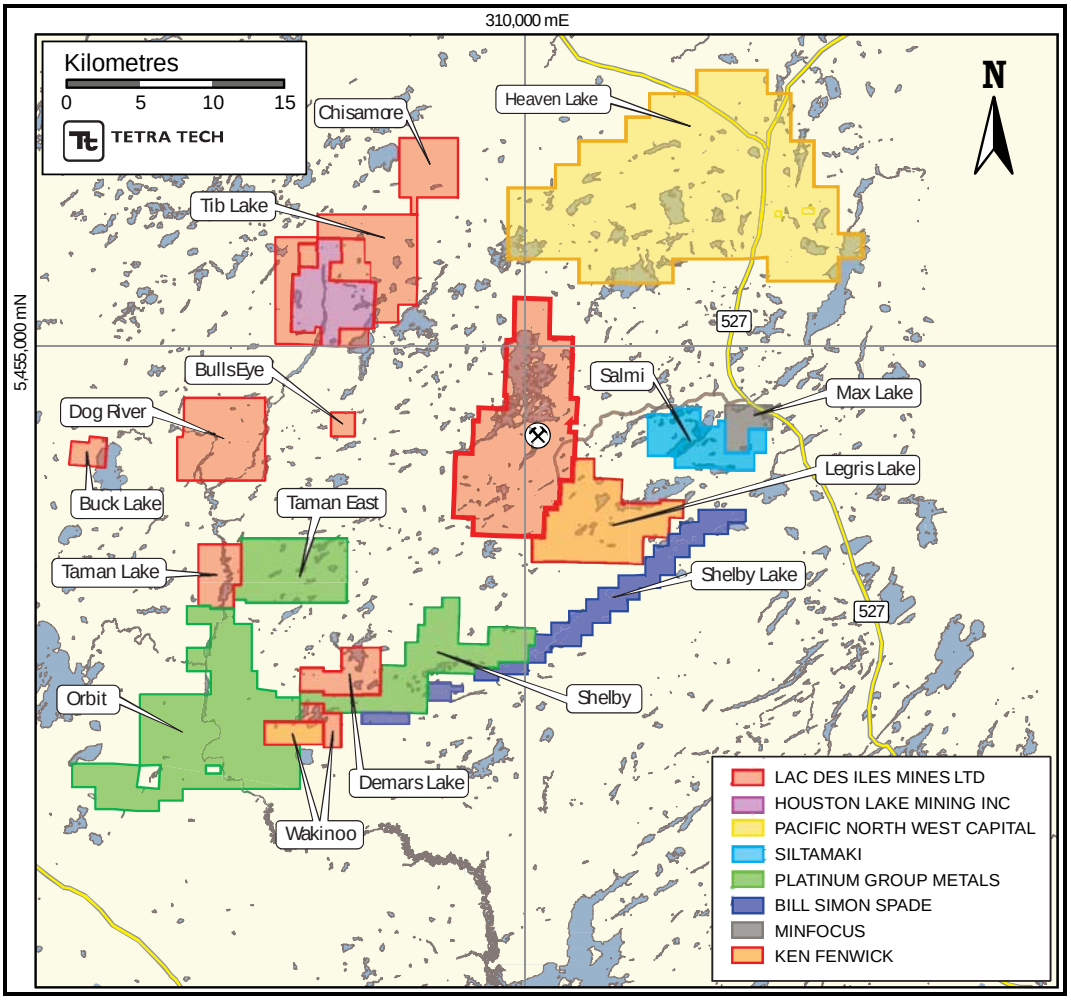
The municipal property tax and education taxes have already been included in the G&A expense line of the cash flow analysis and have been estimated based on a similar scale to other mining operations in the Province of Ontario.

23.0 ADJACENT PROPERTIES

There are several properties adjacent to the Property and are found to be covering mafic-ultramafic intrusions thought to belong to the Lac des Iles suite (Figure 23.1).

NAP, through the wholly-owned subsidiary LDIM, has interest in several properties in the region through staking, option agreements, or joint ventures. These include; Legris, Tib Lake, Chisamore, Eleph, Dog River, Buck Lake, Taman Lake, Wakinoo, Salmi, Bulleye and Demars. As these properties do not meet the definition of an adjacent property by definition in NI 43-101 (1.2), they are displayed in Figure 23.1 for interest sake only.

Figure 23.1 Adjacent Properties



The Heaven Lake property totals 22,110 ha and is held by Pacific North West Capital Corp. (PFN). Historic showings of gabbro containing PGEs exist. Heaven Lake is located approximately 3 km to the north of the Property. No recent activity has been reported on the property and PFN is currently seeking a joint venture partner on the property (http://pfncapital.com).

The Max Lake property located 10 km to the east, totals 766 ha and is held by Minfocus Exploration Corp. No significant mineral occurrences or work programs have been reported.

The Shelby Lake property totals 3,162 ha and is held by Bill Spade. Limited historic work has been conducted on this property.

Platinum Group Metals Ltd. holds three properties 4 km south of the Property. Taman East totals 3,200 ha, Shelby totals 4,067 ha and Orbit totals 10,368 ha. Airborne geophysical surveys, geochemical surveys, and drilling have been conducted recently, with no results reported (www.platinumgroupmetals.net).

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 GEOLOGY

The Project area is underlain by mafic to ultramafic rocks of the Archean LDI-IC. The LDI-IC has intruded granites and greenstones of the Wabigoon Subprovince of the Superior Province. The LDI-IC is the largest of a series of mafic and ultramafic intrusions that occur along the boundary of the Wabigoon-Quetico Subprovince and which collectively define a 30 km in diameter circular pattern. The Project lies in the southern portion of the LDI-IC, in a roughly elliptical intrusive package measuring 3 km long by 1.5 km wide, termed the MBI.

The MBI hosts a number of PGE deposits; the most important of these are the Roby Zone (including its three subzones: the North Roby Zone, the High Grade Zone, and the Breccia Zone) and the Offset Zone.

25.2 MINERAL RESOURCE ESTIMATE

A mineral resource was generated using the criteria of 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. A total of 59,995 palladium assay intervals were used in the estimation process. Grade capping was reviewed and applied were necessary and compositing was completed post-capping on a 1 m downhole length interval.

Grade estimations were completed using OK, ID^2, and NN processes for six elements on a multi-pass search parameter into 5 m by 5 m by 5 m blocks.

All drilling, logging, sampling, and analytical procedures meet industry standards and are acceptable to use in the resource estimation.

Using a 2.5 g/t palladium cut-off, the Measured Resource is 10.4 Mt at 4.67 g/t palladium. The Indicated Resource is 11.6 Mt at 4.18 g/t palladium. The Measured and Indicated resources are inclusive of any reserves. The Inferred Resource is 13.7 Mt at 3.59 g/t palladium.

25.3 METALLURGY

The Lac des Iles Mine is currently running on an intermittent operating schedule, resulting from an intermittent ore supply to the mill. Lac des Iles Mine currently runs fourteen day campaigns at 13,000 t/d followed by fourteen days off. Until the tonnage from the mine is sufficient to meet the full capacity of the mill, this arrangement seems to be working.

The conclusions from the XPS test work in 2010 (XPS 2010) determined that a grind size of 80% passing 38 µm would achieve the optimum palladium flotation recovery. The changes required to the Vertimill™ circuit to increase capacity should be made as quickly as possible to optimize the flotation recoveries.

25.4 MINING

NAP provided a stope wireframe model for the area of the Offset Zone Phase I above the 990 Level to arrive at their interpretation of the reserves.

At a cut-off grade of 2.5 g/t palladium, the mineral reserves identified for the Offset Zone Phase I above the 990 Level (elevation 4,490) consist of 5.63 Mt of Proven Reserves and 2.11 Mt of Probable Reserves for a total of 7.74 Mt grading 4.30 g/t palladium, 0.30 g/t platinum, 0.30 g/t gold, 0.08% copper and 0.11% nickel. This includes dilutions ranging from 14% for the stopes, to 25% for the crowns, and mining recoveries ranging from 60% for crowns, to 90% for the stopes. With 16.05 Mt of Measured and Indicated Resources, at a similar cut-off grade of 2.5 g/t palladium, available above the 990 Level, the 7.74 Mt of mineable reserves for Phase I represent 48% of the resources available to be mined.

The reserves will be mined using the longhole stoping with cemented rockfill mining method. The planned nominal mining rate is 1.26 Mt/a. This can be exceeded in 2014 and 2015 when development ore is added to the annual stope production rate totals. The Phase I reserves will provide production for six years, starting in 2013 with 1.2 Mt, and ending in 2018 with 1.1 Mt.

The main ramp from surface has been extended by contractors from the Roby Zone across the Offset Fault into the Offset Zone and down to the 825 Level. Development above the 735 Level has been completed. Production from this level has commenced using the longhole stoping method. This operation demonstrates the viability of the method at the full production rate. Ore from this horizon is truck hauled to the surface ore pad in the open pit.

Concurrently, a contractor is sinking the No. 1 Shaft and has completed 533 m of the planned 825 m in depth. Once the shaft has been commissioned, ore delivery to surface will switch to hoisting.

Much of the surface infrastructure such as the hoist building and the headframe are now in place. The No. 1 Shaft is scheduled to be operational by the end of Q3 2013. Using contractors to continue underground development and construct underground infrastructure, the target production of 1.2 Mt in 2013 from Phase I of the Offset Zone will likely be met.

The existing processing plant, tailings management facility and mine site infrastructure can continue to accommodate the Offset Zone mining project in their existing configurations.

Employees stay at a camp for their shift rotations, generally accessing the site from Thunder Bay and beyond.

25.5 ECONOMIC EVALUATION

The economic evaluation for the Lac Des Iles Mine generates a cash flow of $171 million, a NPV at a 5% discount of $138 million, an IRR of 63.7%, and a payback period of 2.69 years or 32 months.

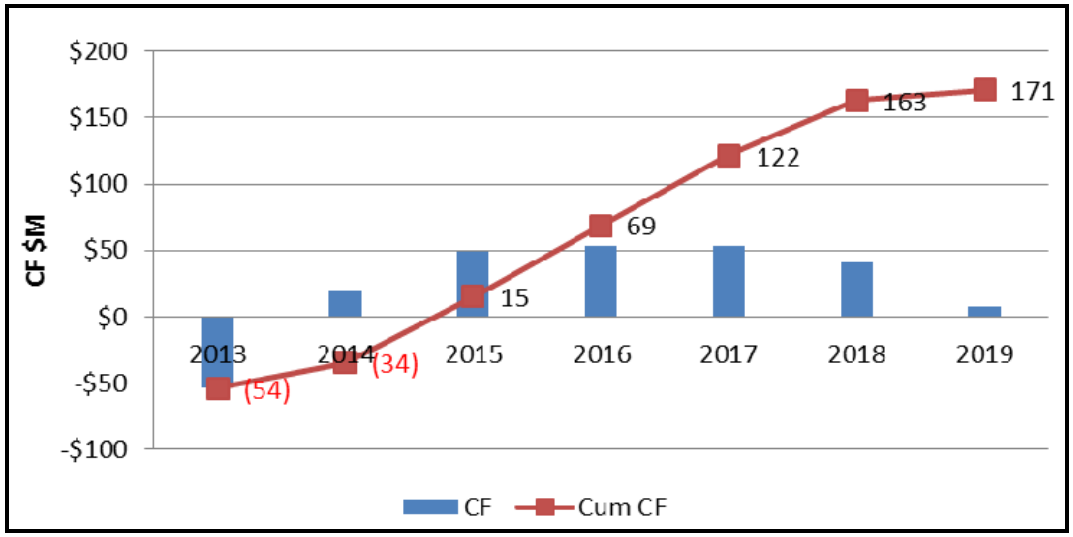
Table 25.1 shows a summary of payable metal in relation to the recovered metal in concentrates over Phase I.

Table 25.1 Payable Metals by Concentrate

Metal	Units	Metal Recovered to Concentrates Phase I	Payable Metal Phase I
Pd	oz	860,107	788,397
Pt	oz	53,764	48,388
Au	oz	61,663	54,880
Cu	t	5,631	5,068
Ni	t	3,706	3,335

The undiscounted annual cash flows are illustrated in Figure 25.1.

Figure 25.1 Annual Undiscounted Cash Flow



The average metal prices for three cases were identified with their corresponding NPVs at discount rates of 8%, 5%, and 0%. The results are shown in Table 25.2.

Table 25.2 NPV by Metal Price Scenario

Case	Average Metal Prices					NPV ($million)		
	Pd (US$/oz)	Pt (US$/oz)	Au (US$/oz)	Cu (US$/lb)	Ni (US$/lb)	@ 8%	@ 5%	@ 0%
Base +20%	810	2,010	2,100	4.20	9.60	248	273	323
Base +10%	743	1,843	1,925	3.85	8.80	185	206	247
Base	675	1,675	1,750	3.50	8.00	121	138	171
Base -10%	608	1,508	1,575	3.15	7.20	58	71	95
Base -20%	540	1,340	1,400	2.80	6.40	(5)	3	19

It can be seen from Table 25.1 and Table 25.2 above that the Lac Des Iles Mine Phase I Offset Zone reserves are economically viable.

26.0 RECOMMENDATIONS

26.1 GEOLOGY

It is recommended that LDIM conduct a detailed mapping program underground with particular emphasis placed on delineating the EGAB contact. As this unit is a barren marker and in close proximity to the higher-grade potions of the Offset Zone it is important to understand the geometry of this unit.

It is strongly recommended for LDIM to continue to conduct specific gravity measurements in order to generate a reasonable specific gravity value for each rock unit and potential by-grades. A minimum of 500 to 1,000 specific gravity measurements would be required in order to generate a regression formula based on rock types and grades. Complete documentation of the procedure should be created including QA/QC, in order to ensure there is no instrument drift.

Exploration on the Offset Zone is recommended to focus on infilling the current resource at approximately 20 to 25 m centers, in order to improve the understanding of the deposit. Continuing to implement the current drilling and sampling procedures would allow diamond drill spacing on the range of 45 to 50 m centers to be adequate for upgrading material from the Inferred to Indicated category. Additional exploration should concentrate on expanding the mineral resource by testing the strike extension to the south of the shaft as well as both the up dip and down dip extensions at a spacing of 90 to 100 m centers.

A vast majority of the drilling will require underground exploration drifts and stations in order to provide suitable drilling platforms. Table 26.1 summarizes the exploration drilling budget.

Table 26.1 Exploration Drilling Budget

Project	Category	Drilling (m)	Direct Drilling Costs ($)	Assays ($)	Labour ($)	Total ($)
Offset Zone	Extension Drilling (>30 m centres)	35,000	3,500,000	1,750,000	1,750,000	**7,000,000**
Offset Zone	Infill Drilling (15 m centres)	15,000	1,500,000	375,000	750,000	**2,625,000**
MBI Surface Targets	Exploration and Resource Definition	20,000	2,000,000	1,000,000	1,000,000	**4,000,000**
Total		**70,000**	**7,000,000**	**3,125,000**	**3,500,000**	**13,625,000**

26.2 METALLURGY

It is recommended that the optimization be completed on the Vertimills™ to achieve the final grind size of P_{80} -38 µm prior to flotation, as recommended in the test work by XPS (XPS 2010). The current flowsheet also has the feed to flotation split after the ball mill hydrocyclones such that 20% of the ball mill overflow goes to the flotation conditioning tank and 80% to the Vertimills™. There are plans in place to create the sump and Vertimill™ capacity to accept the entire ball mill hydrocyclone overflow into the Vertimills™ rather than divert 20% of the flow to flotation. It is recommended these plans be implemented so the optimum grind size can be achieved on all of the feed prior to flotation. This should help optimize the flotation recoveries.

Further test work should be done to determine how the Lac des Iles Mine feed would react in a MF2 flowsheet, as is commonly used in South Africa for platinum flotation. The estimated cost for some sighter tests to determine if further test work is warranted would be approximately $50,000.

26.3 MINING

The following are mining recommendations for the Lac des Iles Mine:

- The Offset Zone Phase I Project should continue detailed engineering and construction focused on ensuring the ramp up of production from the Offset Zone at the Lac des Iles Mine continues as planned.
- Additional infill drilling successes should be expected in the north and south limbs. Accordingly, detailed engineering of the level designs should be cognizant of this expectation with ventilation raises, ore passes, footwall accesses, and cross-cuts positioned in a manner that these opportunities are not sterilized (in-house, included in operating costs).
- Approximately 48% of the available Measured and Indicated resource above the 990 Level became reserves under the current Phase I mining plan (this becomes 30% with the inclusion of Inferred material). A PEA should be conducted on all future phase resources above the 990 Level. These resources have been sufficiently delineated to allow identification of mining options ($100,000).
- A PEA should be conducted on all future phase resources below the 990 Level. These resources have been sufficiently delineated to allow identification of mining and access options ($100,000).
- A comprehensive rock mechanics assessment initiated between NAP and Itasca in 2013 should be extended to define the crown pillar requirement for the 990 Level and to define whether the sequence above is applicable as mining continues with future phases at depth. Further to this Itasca should be tasked with assessing the potential for any fault slip once mining

extraction across the total Offset Zone (including removal of the 990 Level, 840 Level and 750 Level crown pillars) nears maturity ($75,000).

- The method applied to determine ventilation quantities in this report is of a prefeasibility study nature and accordingly requires improvement at a feasibility study stage. With the Offset Zone Phase I Project being well advanced, NAP should carry out further detailed ventilation design to confirm the ventilation quantities (confirmatory study $25,000).
- NAP should evaluate continuing shaft sinking with the establishment of an ore handling system immediately below the 990 Level that allows all ore above the 990 Level to be gravity fed by ore passes to a crusher thereby negating the need for trucking Phase I ore up ramp to the 645 Level grizzly and rock breaker location (trade-off study $50,000).
- As part of the expectation for an increase in strike length of the Offset Zone mining envelope, NAP should assess whether the planned cemented rock fill system is as conducive to horizontal distribution as a backfill that can be placed through a pipeline such as paste fill. The assessment would be expected to include consideration for capital and operating costs including the impact on overall trucking/conveying needs and the associated ventilation demands were the paste fill system found to be the option of choice (backfill trade-off study $50,000).
- A prefeasibility study that looks at the resources above and below the 990 Level should follow completion of the PEA's and other mentioned trade-off studies (plus $250,000).
- Continue cavity monitoring and survey pickups of all openings with the intent to properly document and identify solutions for achievement of a reduction in their historical dilution values that for this report come from the mining of the larger stopes in the Roby Zone (in-house).
- The potential for oscillations in production may limit operating throughput. Recommend the enterprise constraints (i.e. grizzly, ore passes and hoist capacity, ventilation, etc.) be studied and contingencies identified on an annual basis ($25/a).
- Given the relatively short Phase I production plan based on initial reserve estimation of the Offset Zone, a sustaining capital study for the mining equipment should be performed to see the fleet through Phase I (forward sustaining capital study $20,000).
- Similarly, an efficient operating and maintenance discipline must be established to keep the hoisting plant (hoists, conveyances, headframe) in good shape, especially for the potential increase in the mineable reserves.

The total estimated cost of mining recommendations identified above is $695,000 over a 1.3-year period.

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WEBSITES

Accurassay Laboratories
http://www.accurassay.com/quality/content.php?id=32

Activation Laboratories Ltd.
http://www.actlabs.com/page.aspx?menu=60&app=206&cat1=731&tp=2&lk=no

Houston Lake Mining Inc.
http://www.houstonlakemining.com/index.php/news-mainmenu-54/2012-news-releases/311-hlm-enters-definitive-agreement-with-north-american-palladium-with-the-option-to-purchase-its-tib-lake-pge-property

North American Palladium
http://www.nap.com

Pacific Northwest Capital Corp.
http://pfncapital.com/s/ProjectSubmittals.asp?ReportID=535289

Platinum Group Metals Ltd.
http://www.platinumgroupmetals.net/projects/canada/lac_des_iles/

Stockhouse.com
http://www.stockhouse.com/natural-resources-news/2011/mar/11/stockhouse-movers---shakers--pdac-and-a-nod-to-pal.aspx

The Weather Network
http://www.theweathernetwork.com/statistics/CL6049443/caon0688?ref=topnav_historical_statistics

City of Thunder Bay
http://www.thunderbay.ca/CEDC/Reports_and_Publications/Community_Profile_Facts_and_Statistics/Demographics.htm

US Geological Survey
http://pubs.usgs.gov/bul/b1693/html/bullfrms.htm

Yr.no
http://www.yr.no/place/Canada/Ontario/Lac_des_Iles/statistics.html

Platinex Inc.
http://platinex.com/wp-content/uploads/2012/10/PTX-Sells-Tib-Lake-Claim-Group1.pdf

APPENDIX A

CERTIFICATES OF QUALIFIED PERSONS

DAVID N. PENNA, P.GEO.

I, David N. Penna, P.Geo., of Thunder Bay, Ontario, do hereby certify:

- I am a Professional Geoscientist 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 Technical Report Lac des Iles Mine, Ontario, Incorporating Prefeasibility Study Offset Zone Phase I, dated January 31, 2013 (the "Technical Report").
- I am a graduate of Laurentian University (B.Sc. (Hon), 1986). I am a member in good standing of the Association of Professional Geoscientists of Ontario (#112). My relevant experience includes underground grade control, open pit planning and design, reserve and resource estimations, budgeting/forecasting and exploration planning. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have completed a personal inspection of the Property as I am currently employed at the Property.
- I am responsible for Section 15.2 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 Professional Geoscientist at the mine on the Property that is the subject of this Technical Report for since March 2005.
- 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 dated this 13th day of February, 2013 at Thunder Bay, Ontario.

"Original document signed and sealed by David N. Penna, P.Geo."

David N. Penna, P.Geo.
Principal Geologist
Lac Des Iles Mines Ltd.

KEVIN SMALL, P.ENG.

I, Kevin Small, P.Eng., of Thunder Bay, Ontario, do hereby certify:

- I am a Director, Technical Services with North American Palladium Ltd. with a business address at Royal Bank Plaza, South Tower, 200 Bay Street, Suite 2350, Toronto Ontario, M5J 2J2.
- This certificate applies to the technical report entitled Technical Report Lac des Iles Mine, Ontario, Incorporating Prefeasibility Study, Offset Zone Phase I, dated January 31, 2013 (the "Technical Report").

  I am a graduate of Queen's University (B.Sc. Mining Engineering, 1990). I am a member in good standing of the Association of Professional Engineers of Ontario (#90376674). My relevant experience includes the following positions Technical Director for North American Palladium Ltd.; Engineering Manager for DMC Mine Services; Engineering Manager and Area Manager for Dumas; Engineering Manager for Hatch Consultants; Superintendent Project Management Office for Inco-Coleman/Creighton Mines; Underground General Foreman-Garson, North, Stobie Mines; Chief Engineer for Inco -Garson Mine; Underground Mine Foreman-Stobie Mine; Engineering Supervisor for Inco – Stobie Mine and Garson Mines; Mine Planning Engineering for Inco –Little Stobie, Stobie, Creighton Mines. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was January 28, 2013 for four days.
- I am responsible for Section 14.2 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 Director, Technical Service at the mine on the Property that is the subject of this Technical Report Since July 2011.
- 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 dated this 13th day of February, 2013 at Thunder Bay, Ontario.

"Original document signed and sealed by Kevin Small, P.Eng."

Kevin Small, P.Eng.
Director, Technical Services
North American Palladium Ltd.

PHILIP BRIDSON, P.ENG.

I, Philip Bridson, P.Eng., of Lively, Ontario, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled Technical Report Lac des Iles Mine, Ontario, Incorporating Prefeasibility Study Offset Zone Phase I, dated January 31, 2013 (the "Technical Report").
- I am a graduate of the Michigan Technological University (B.Sc., Mining Engineering, 1972 and B.Sc., Engineering Administration, 1972). I am a member in good standing of the Association of Professional Engineers of Ontario (#5181011). My relevant experience includes expertise in underground long range and short range mine planning and scheduling, financial evaluations, life of mine development and analysis, project evaluations and strategic planning. From 1972 to 2008 I worked at Canadian and US underground mines holding positions as a Miner, Blasthole Engineer, First Line Supervisor, Design Engineer, Sr. Development Engineer, Senior Project Engineer, Senior Long Range Planning Engineer, Senior Planning Supervisor and Senior Business Planner. Since 2009 I have worked as a consultant mining engineer. As a Mining Engineer I have completed a variety of projects for mining companies involving scoping studies, prefeasibility studies, feasibility studies, project evaluations, and life-of-mine studies for projects in Canada, US, and Australia. My work experience includes base metal deposits (copper, zinc, nickel, gold, silver, platinum, palladium), uranium deposits, iron deposits and potash deposits. 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 1.10, 1.11, 1.12, 1.13.4, 19.0, 21.0, 22.0, and 25.5 of the Technical Report.
- I am independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- 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 dated this 13th day of February, 2013 at Lively, Ontario.

"Original document signed and sealed by Philip Bridson, P.Eng."

Philip Bridson, P.Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.

WILLIAM RICHARD MCBRIDE, P. ENG.

I, William Richard McBride, P.Eng., of Sudbury, Ontario, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at 101-957 Cambrian Heights, Sudbury, Ontario, P3C 5M6.
- This certificate applies to the technical report entitled Technical Report Lac des Iles Mine, Ontario, Incorporating Prefeasibility Study Offset Zone Phase I, dated January 31, 2013 (the "Technical Report").
- I am a graduate of Queen's University (B.Sc., Mining Engineering, 1973). I am a member in good standing of the Association of Professional Engineers of Ontario (#29888013). My relevant experience includes 40 years of experience in mines engineering and operations, including long-range and short-range mine planning and the managing of projects from concept through to start-up. From 1970 to 2008, I worked at Canadian and Central American mines holding positions as a Miner, Certified Open Pit Blaster, Planning Engineer, open pit and underground, Certified Hard Rock Underground Supervisor, Engineering Supervisor, Chief Mine Engineer, Superintendent Technical Services and Senior Projects Manager. In 2009, I worked as a consultant Project Manager. Since 2010 I have worked as a Senior Mining Engineer. I have completed scoping studies, prefeasibility studies, feasibility studies, project evaluations, and life-of-mine studies for underground mines that employ bulk mining techniques in nickel sulphide PGE type 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 5 to 8, 2012.
- I am responsible for Sections 1.5, 1.7, 1.8, 1.13.3, 15.1, 16.0, 18.0, 25.4, 26.3 and 27.0 of the Technical Report.
- I am independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- 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 dated this 13th day of February, 2013 at Sudbury, Ontario.

"Original document signed and sealed by
William Richard McBride, P.Eng."

William Richard McBride, P. Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.

TODD KANHAI, P.ENG.

I, Todd Kanhai, P.Eng., of Sudbury, Ontario, do hereby certify:

- I am a Metallurgical Engineer with Tetra Tech WEI Inc. with a business address at 101-957 Cambrian Heights, Sudbury, Ontario, P3C 5M6.
- This certificate applies to the technical report entitled Technical Report Lac des Iles Mine, Ontario, Incorporating Prefeasibility Study Offset Zone Phase I, dated January 31, 2013 (the "Technical Report").
- I am a graduate of Laurentian University, (M. A. Sc., 2000). I am a member in good standing of the Professional Engineers of Ontario (#90471020). My relevant experience is with previous conceptual, preliminary economic assessment, prefeasibility, and feasibility studies while working with Tetra Tech WEI Inc. Included in these studies were nickel sulphide platinum group element 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 5 to 7, 2012.
- I am responsible for Sections 1.6, 1.13.2, 13.0, 17.0, 25.3, 26.2, and 27.0 of the Technical Report.
- I am independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- 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 dated this 13th day of February, 2013 at Sudbury, Ontario.

"Original document signed and sealed by Todd Kanhai, P.Eng."

Todd Kanhai, P.Eng.
Metallurgical Engineer
Tetra Tech WEI Inc.

TODD MCCRACKEN, P.GEO.

I, Todd McCracken, P.Geo., of Sudbury, Ontario, do hereby certify:

- I am a Principal Geologist with Tetra Tech WEI Inc. with a business address at 101-957 Cambrian Heights, Sudbury, Ontario, P3C 5M6.
- This certificate applies to the technical report entitled Technical Report Lac des Iles Mine, Ontario, Incorporating Prefeasibility Study Offset Zone Phase I, dated January 31, 2013 (the "Technical Report").
- I am a graduate of the University of Waterloo, (B.Sc. Honours, 1992). I am a member in good standing of the Association of Professional Geoscientists of Ontario (#0631). My relevant experience includes 20 years of experience in exploration and operations, including several years working magmatic nickel sulphides and PGE deposits. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was May 23 to 25, 2012.
- I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.9, 1.13.1, 2.0, 3.0, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.1, 20.0, 23.0, 24.0, 25.1, 25.2, 26.1, and 27.0 of the Technical Report.
- I am independent of North American Palladium Ltd. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- 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.

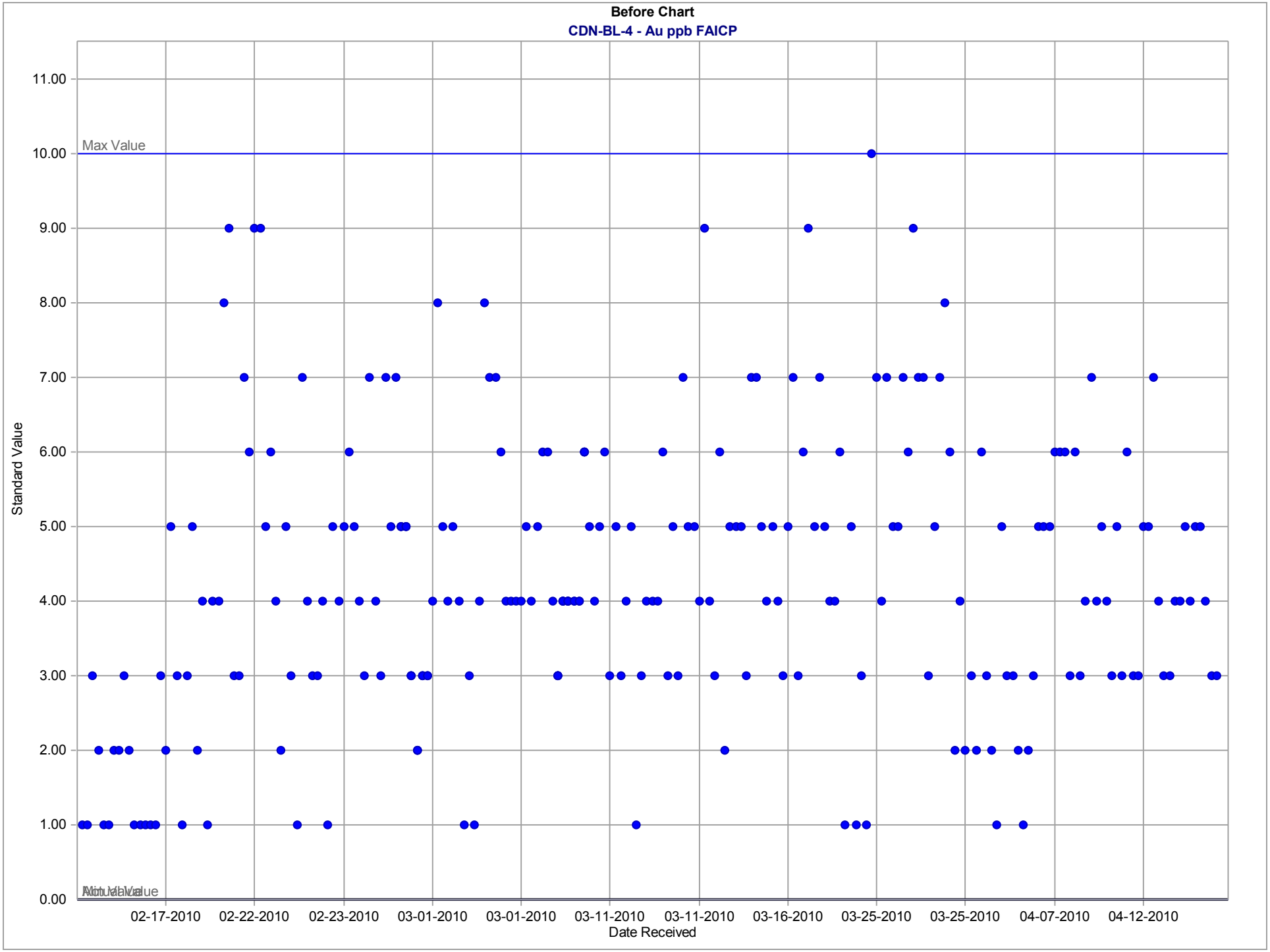
Signed and dated this 13th day of February, 2013 at Sudbury, Ontario.

"Original document signed and sealed by Todd McCracken, P.Geo."

Todd McCracken, P.Geo.
Principal Geologist
Tetra Tech WEI Inc.

APPENDIX B

STANDARDS RESULTS GRAPHS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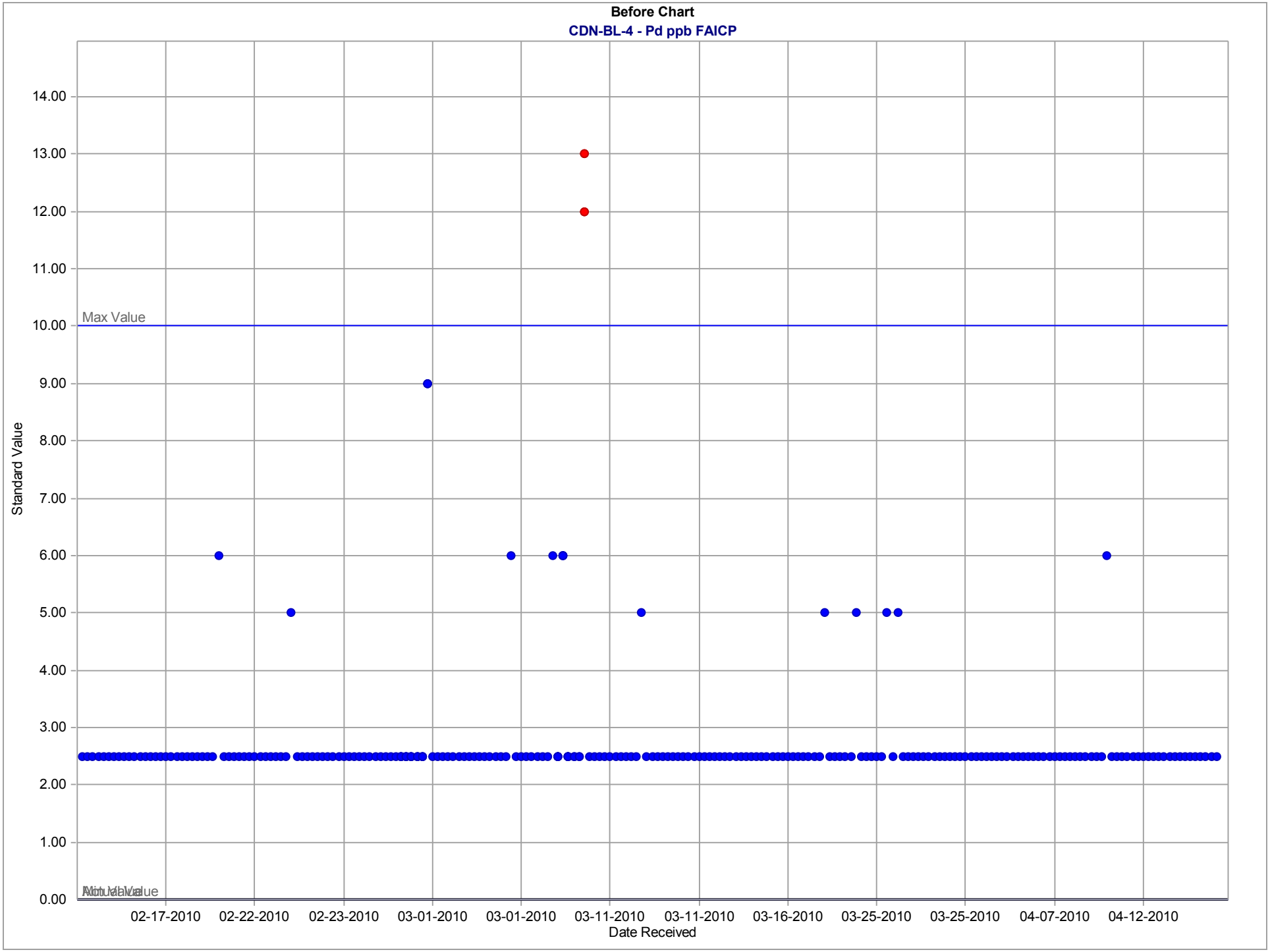
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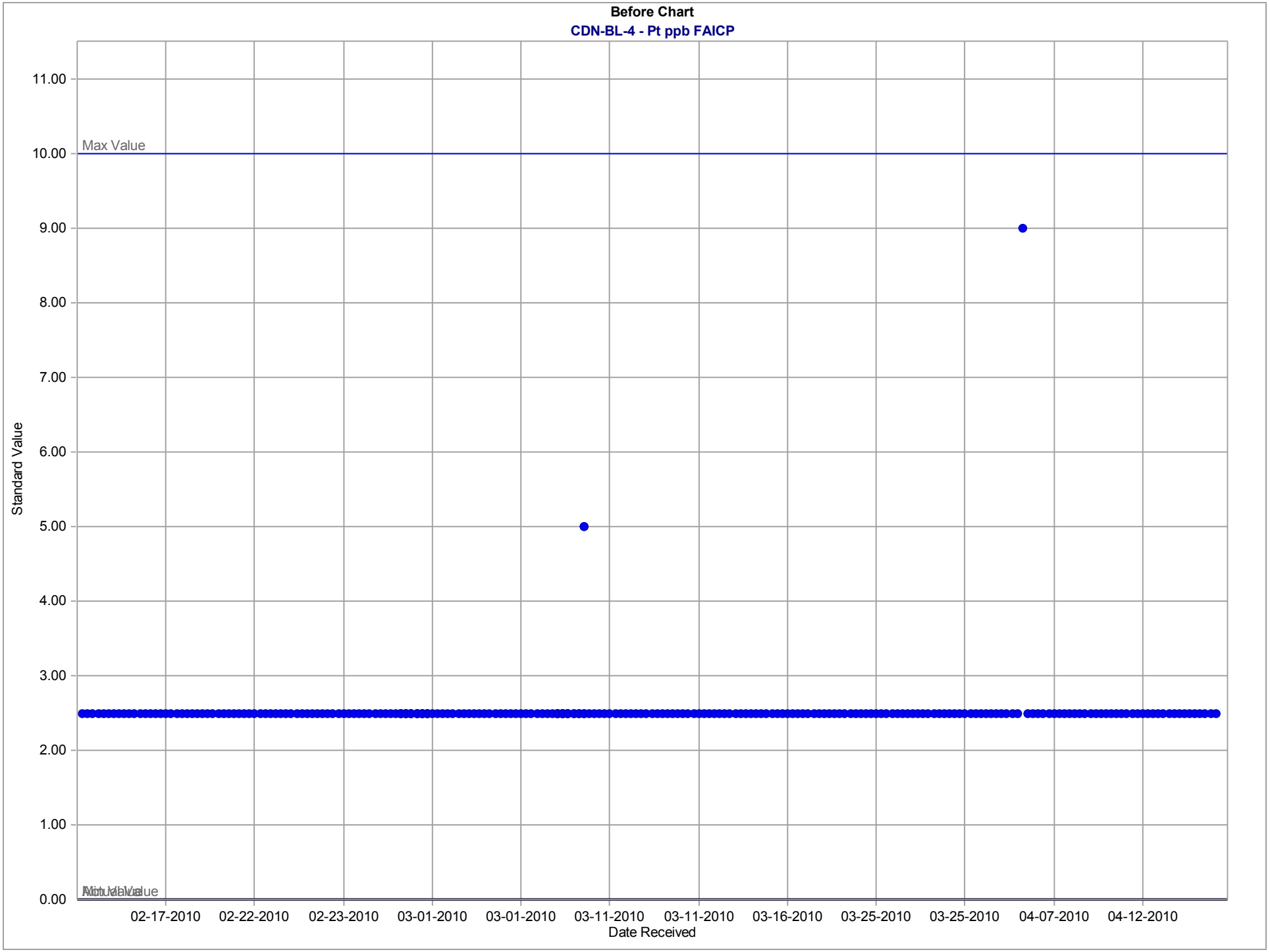
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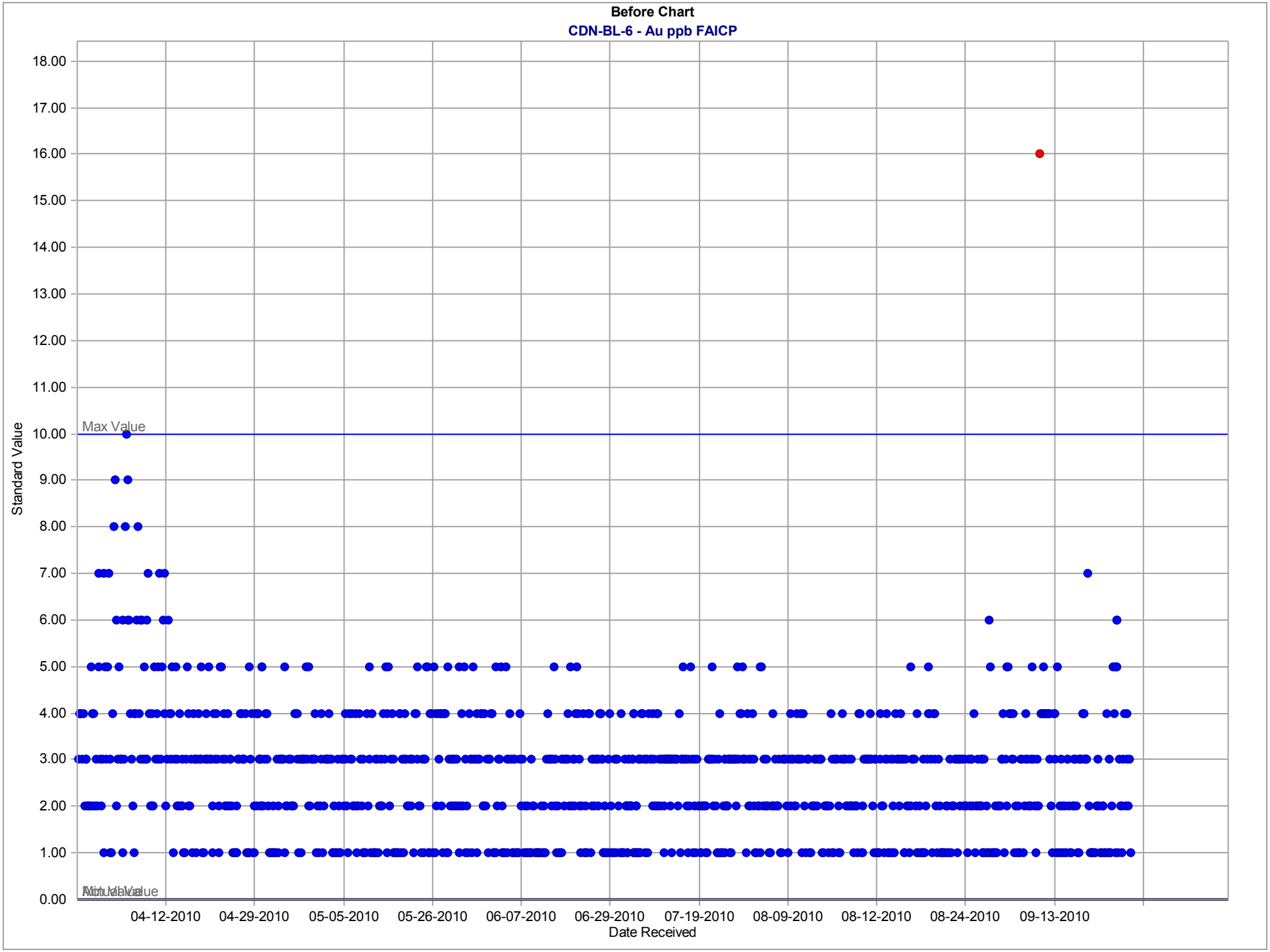
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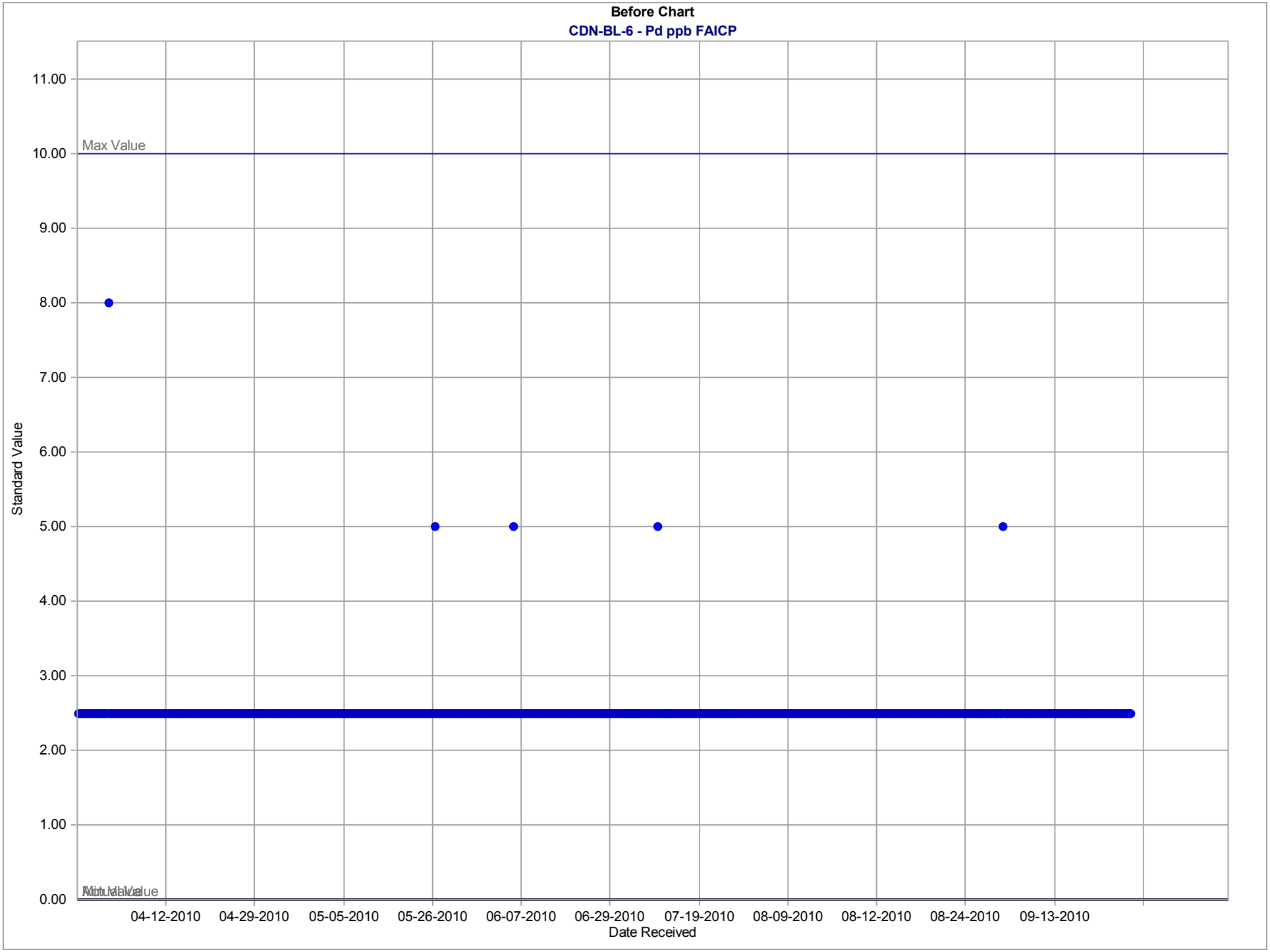
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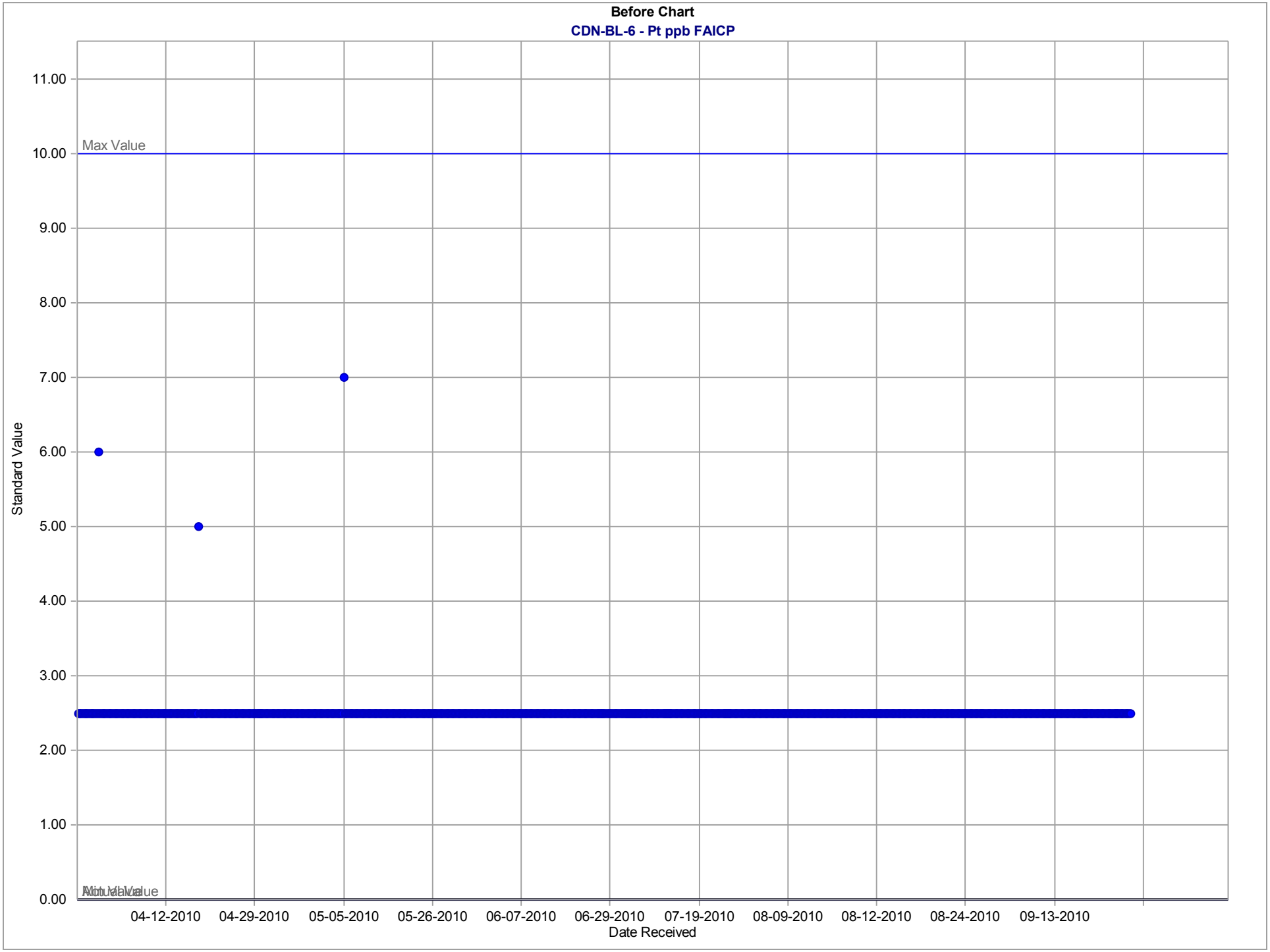
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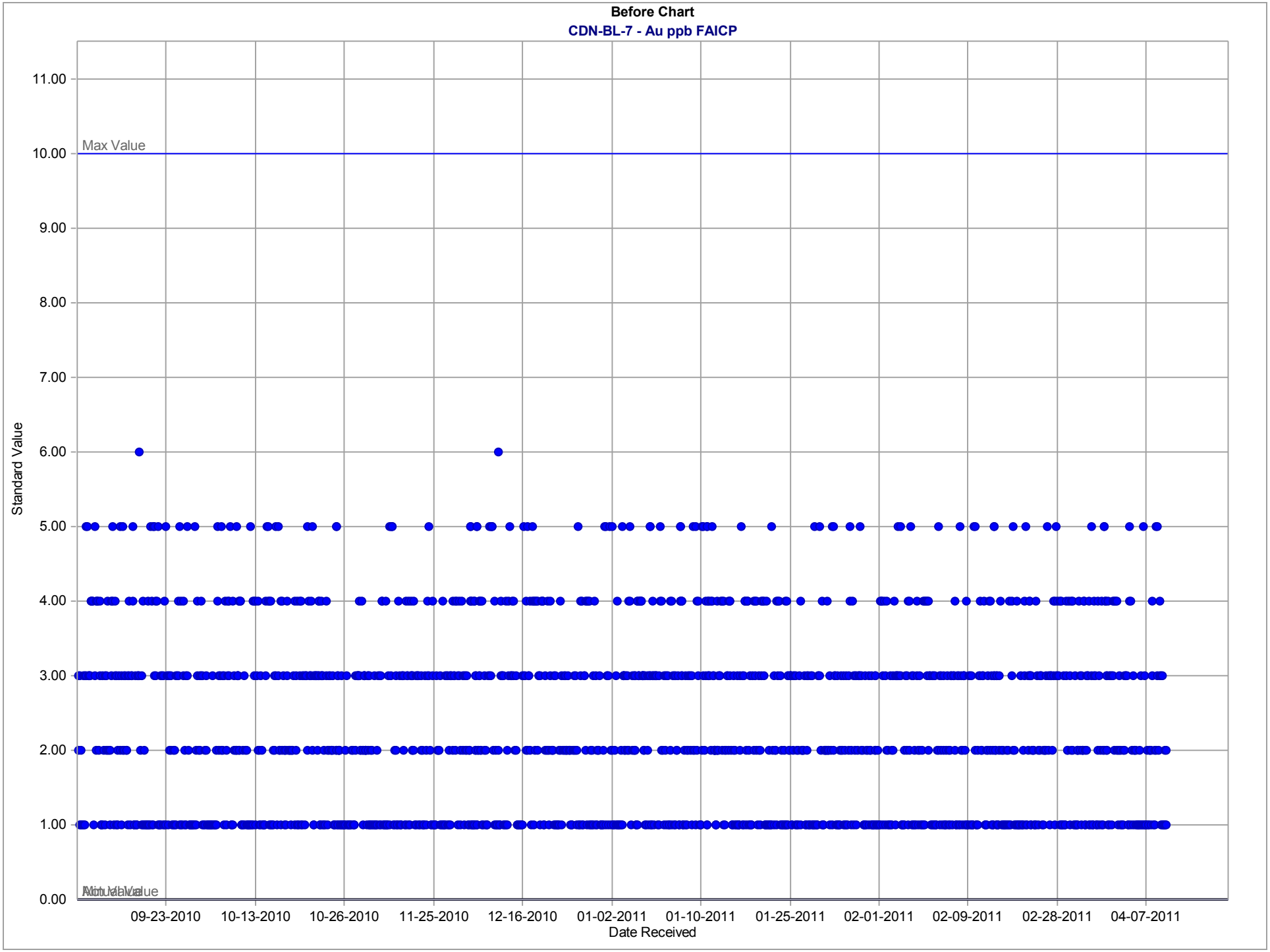
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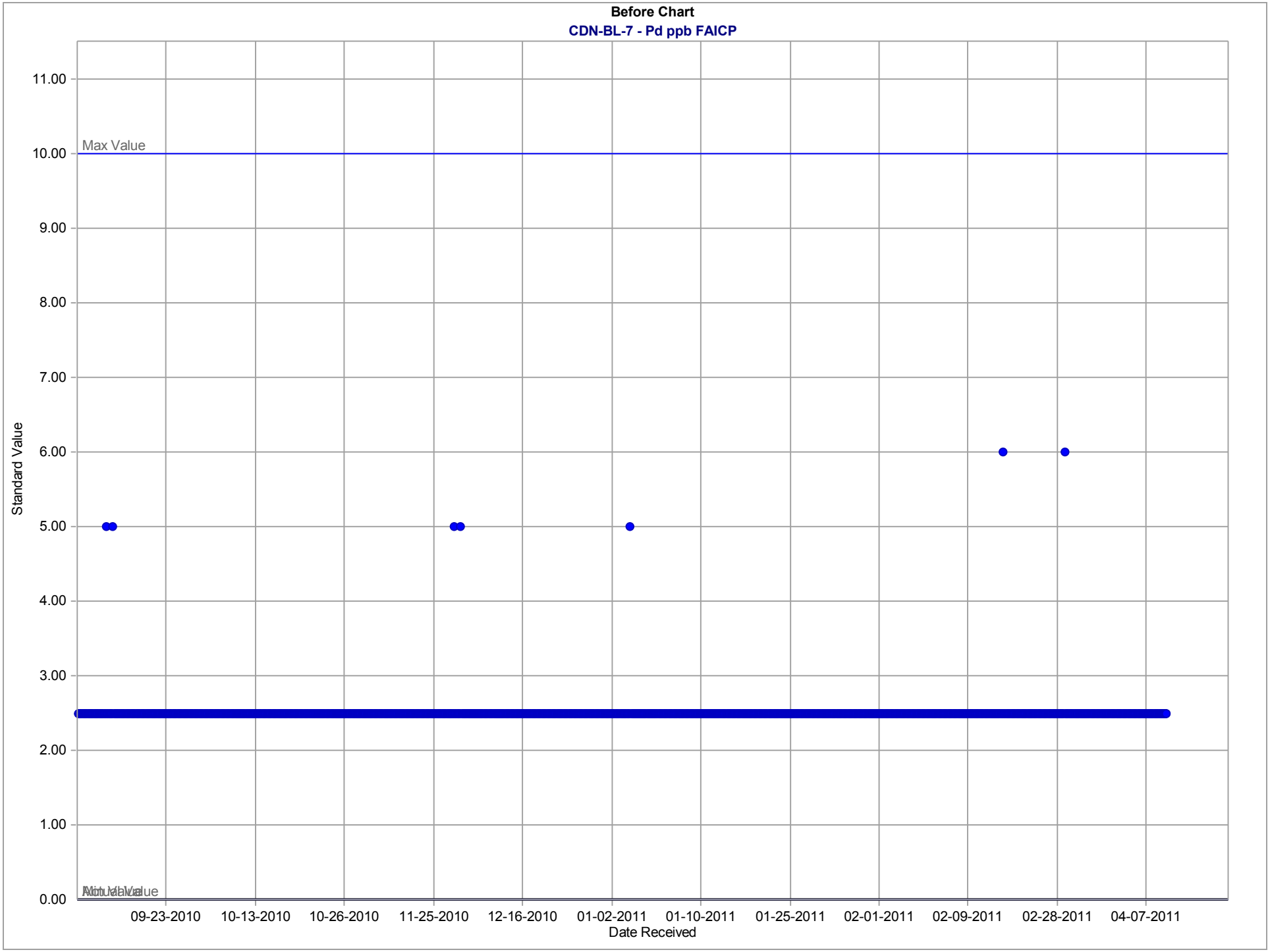
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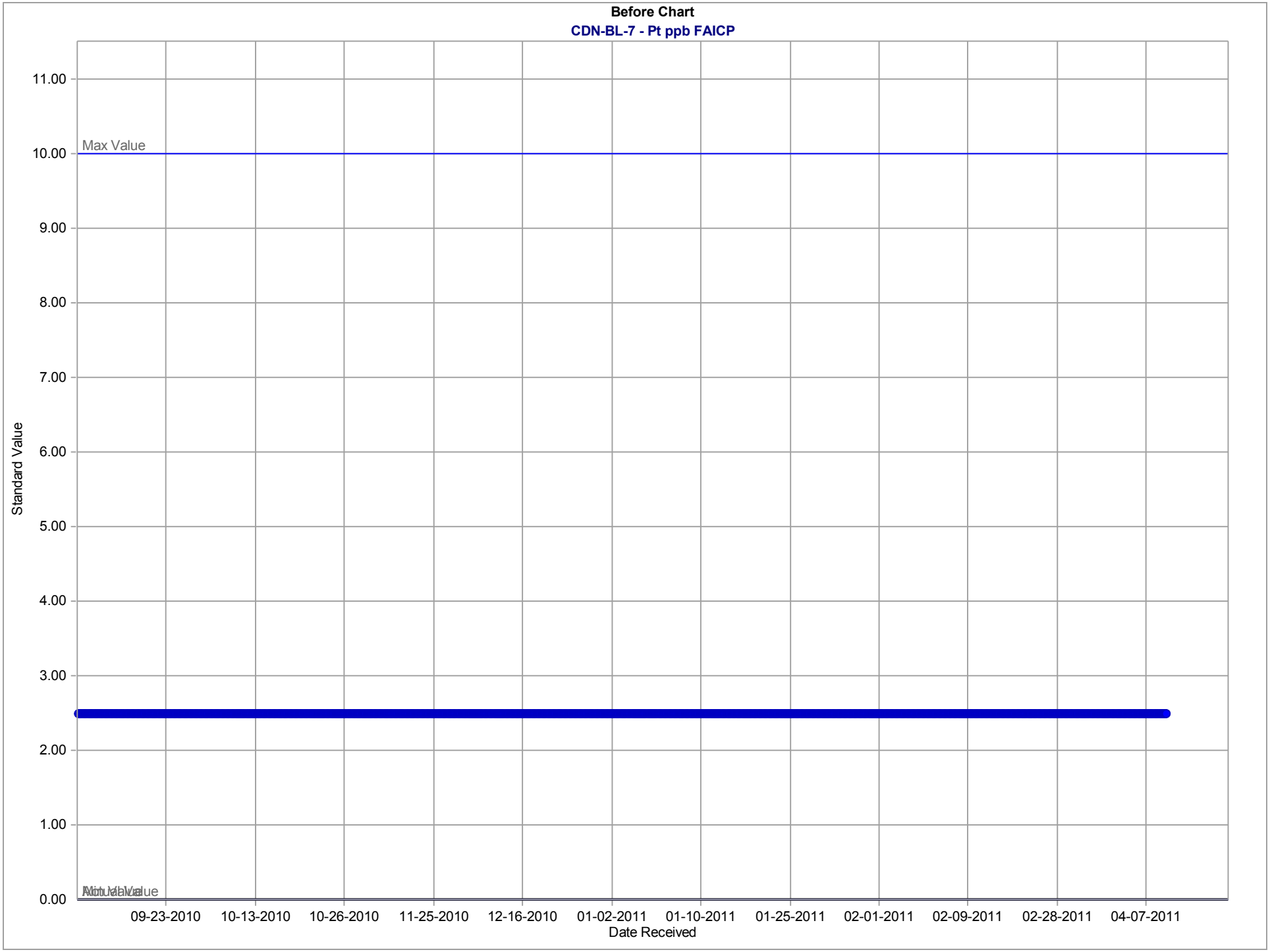
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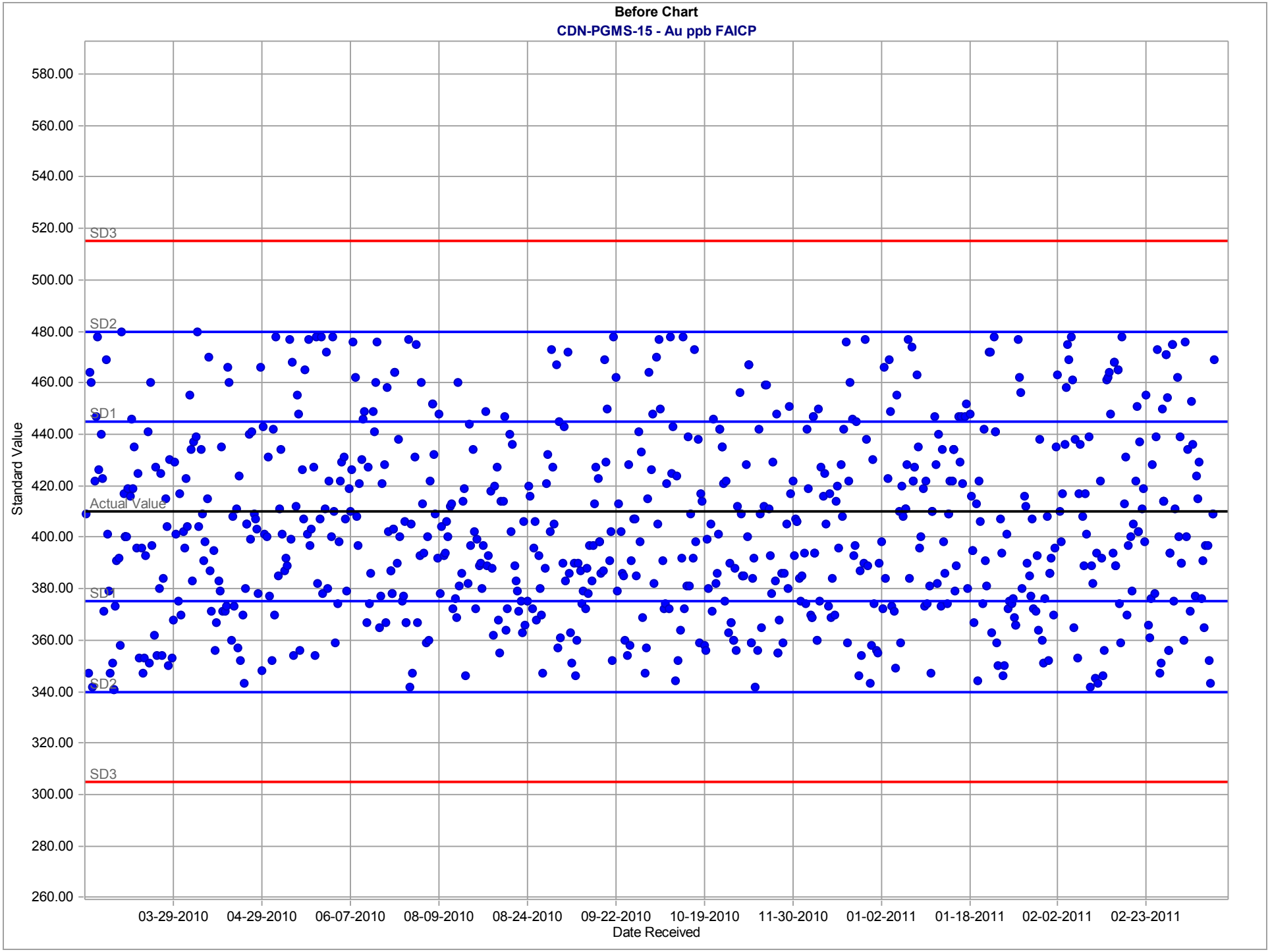
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280.00
260.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
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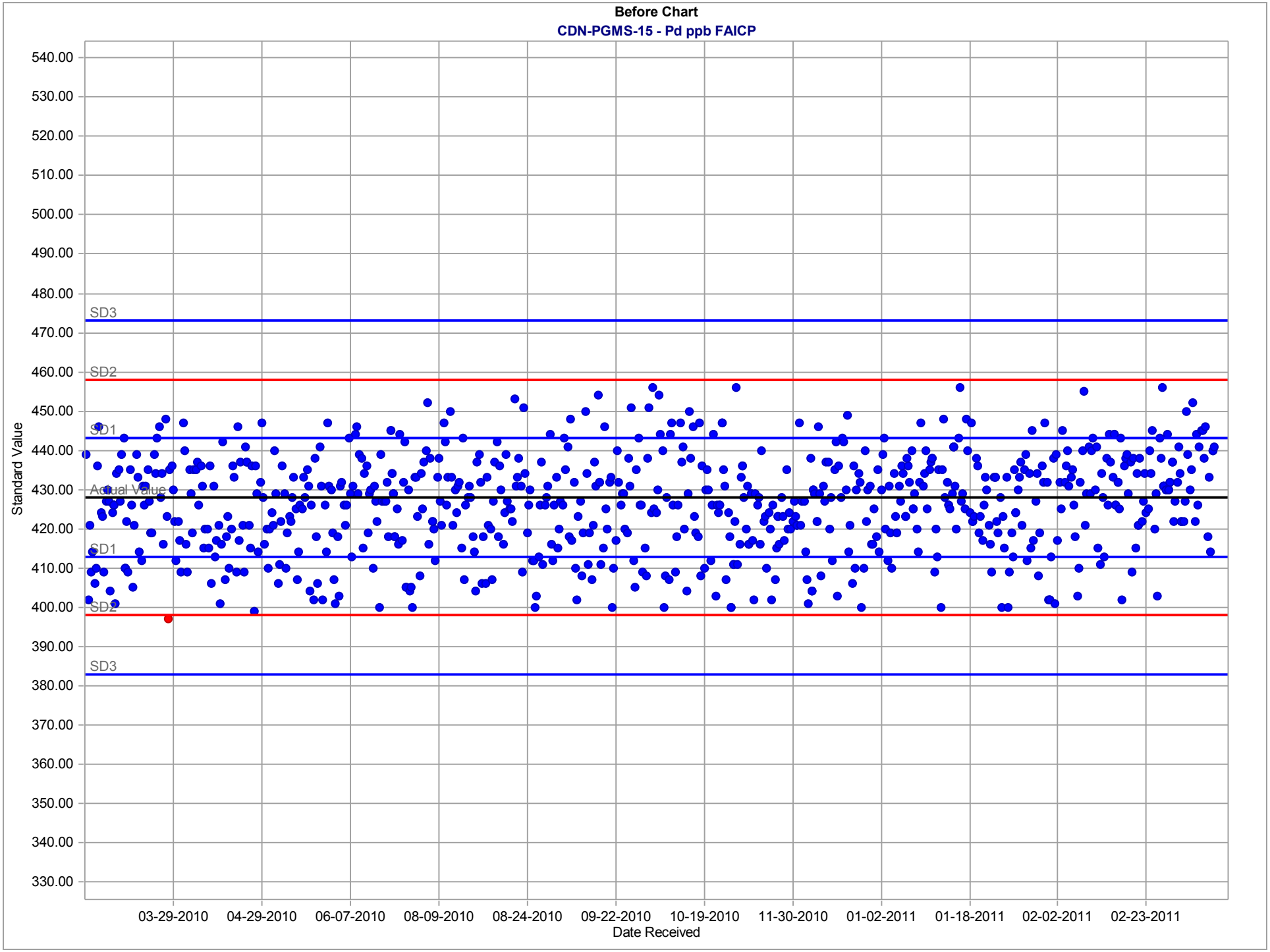
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SD1
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Actual Value
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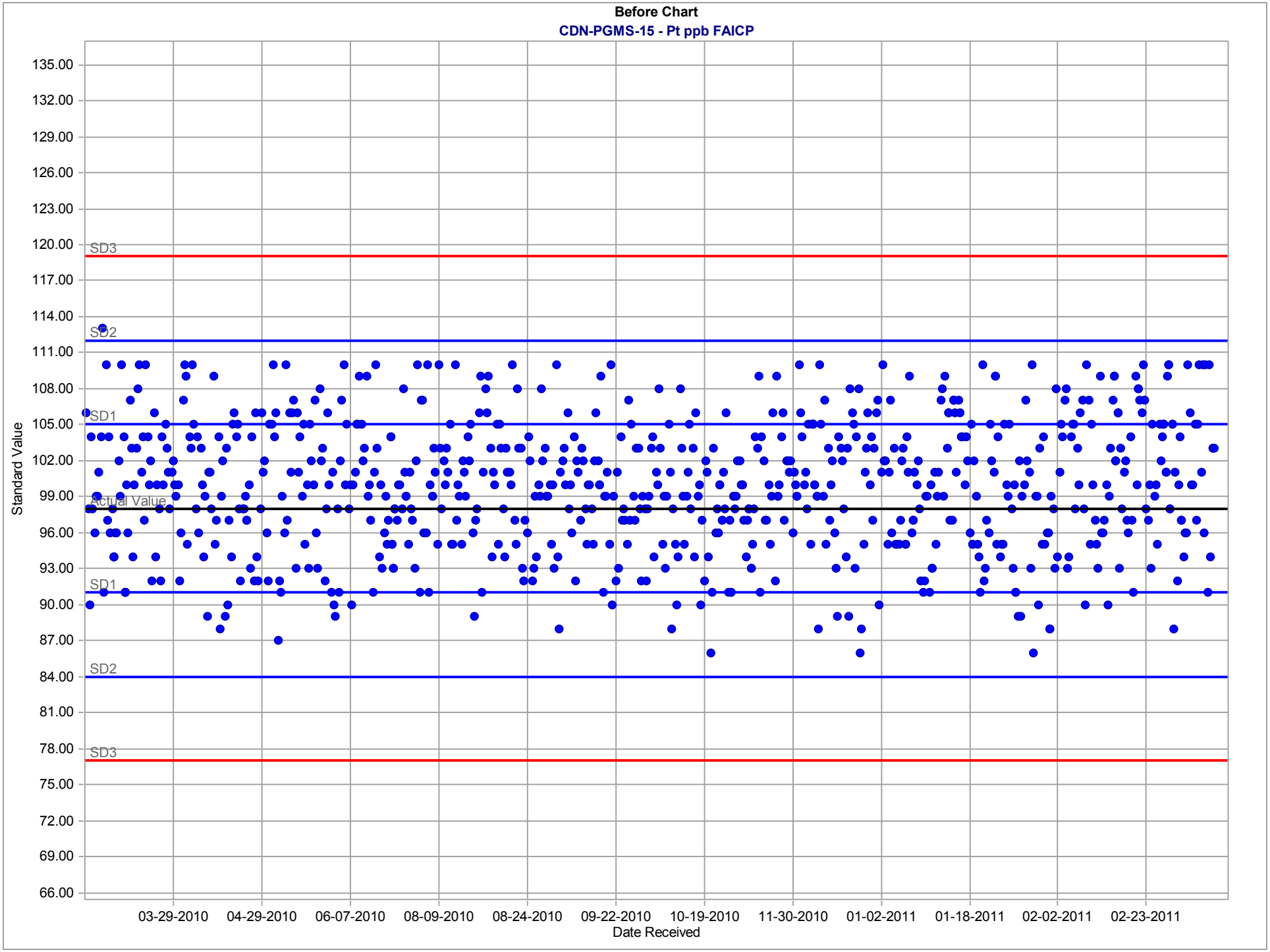
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SD1
SD2
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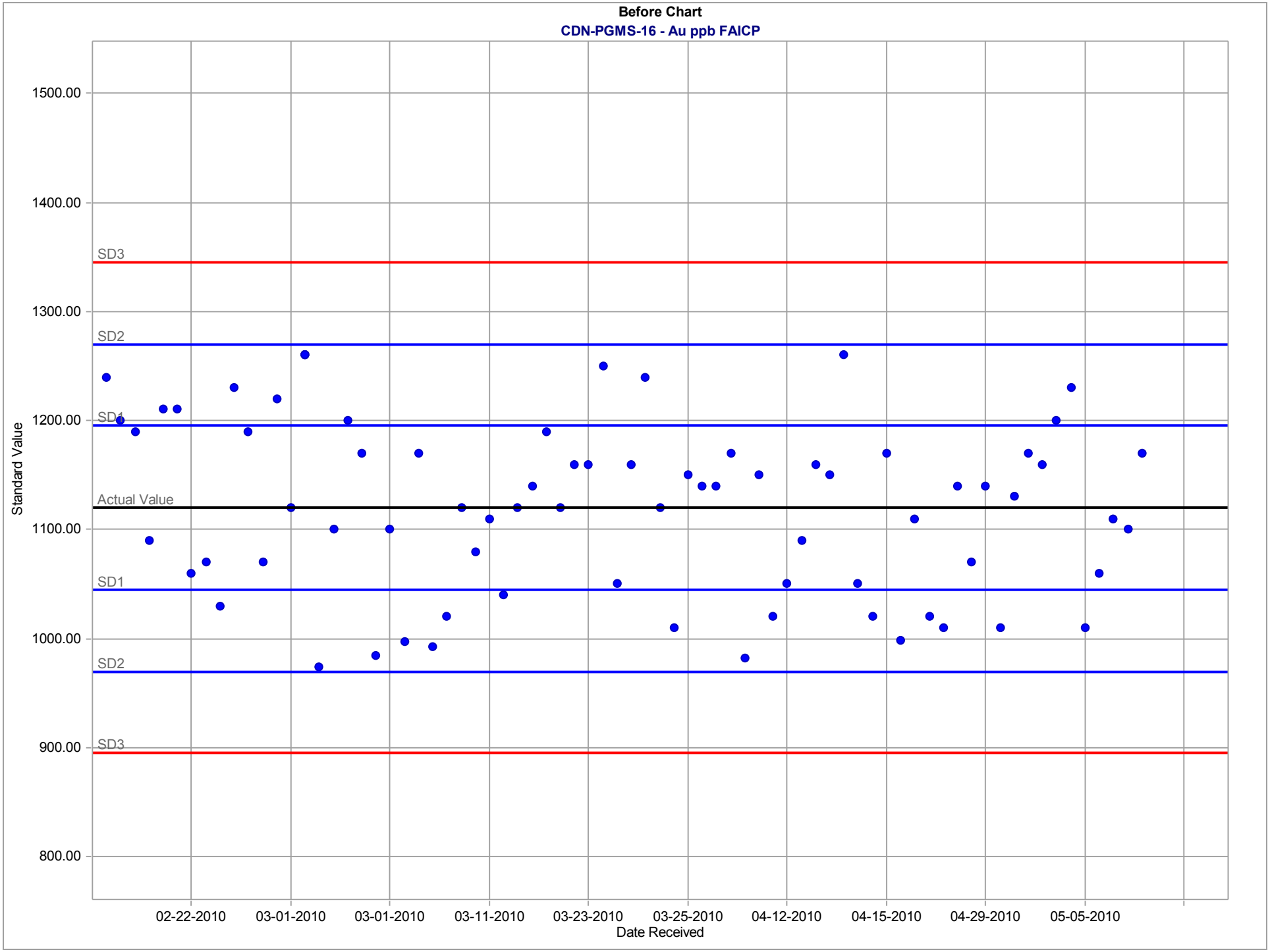
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SD1
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SD1
SD2
SD3
Standard Value
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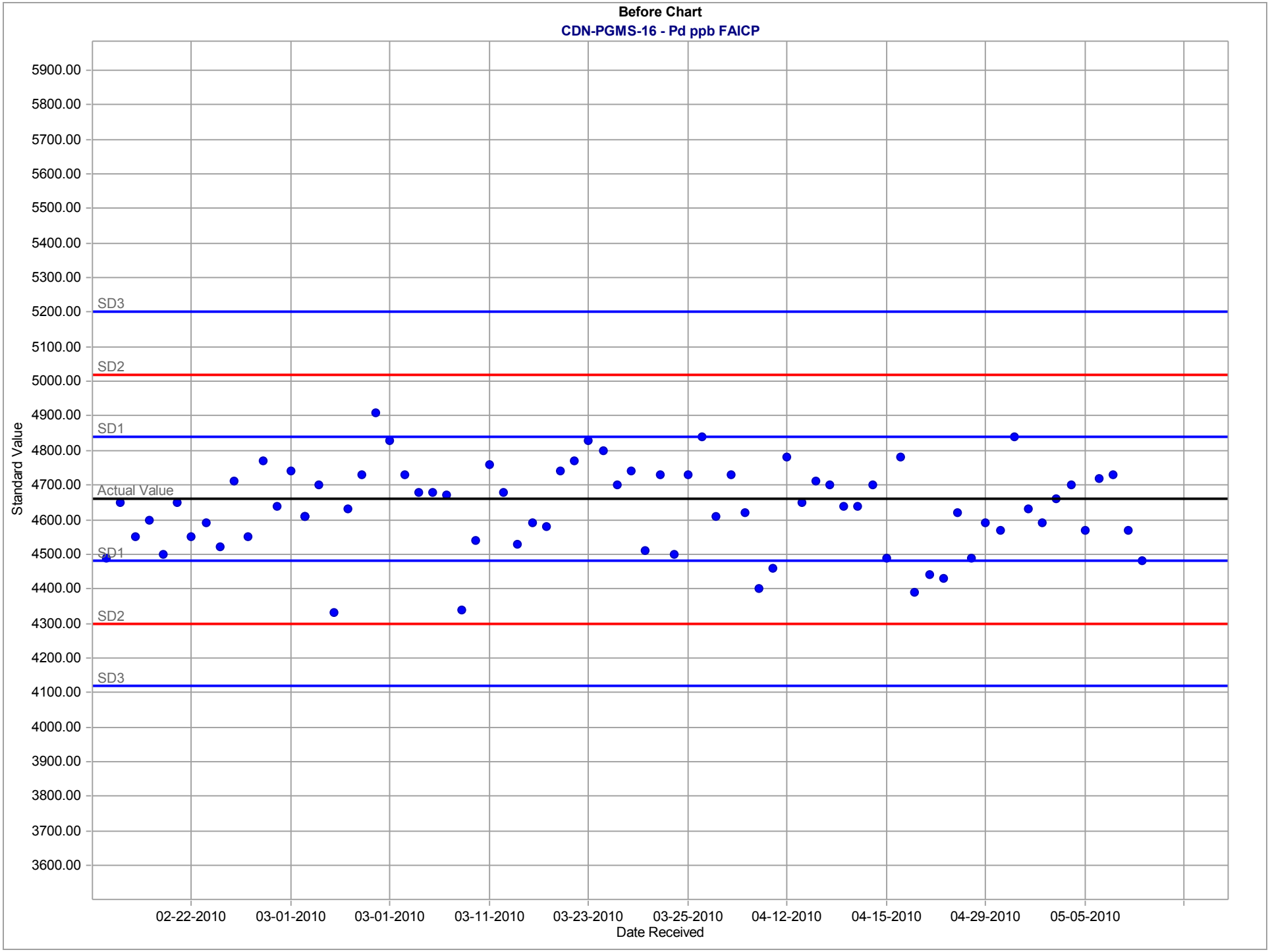
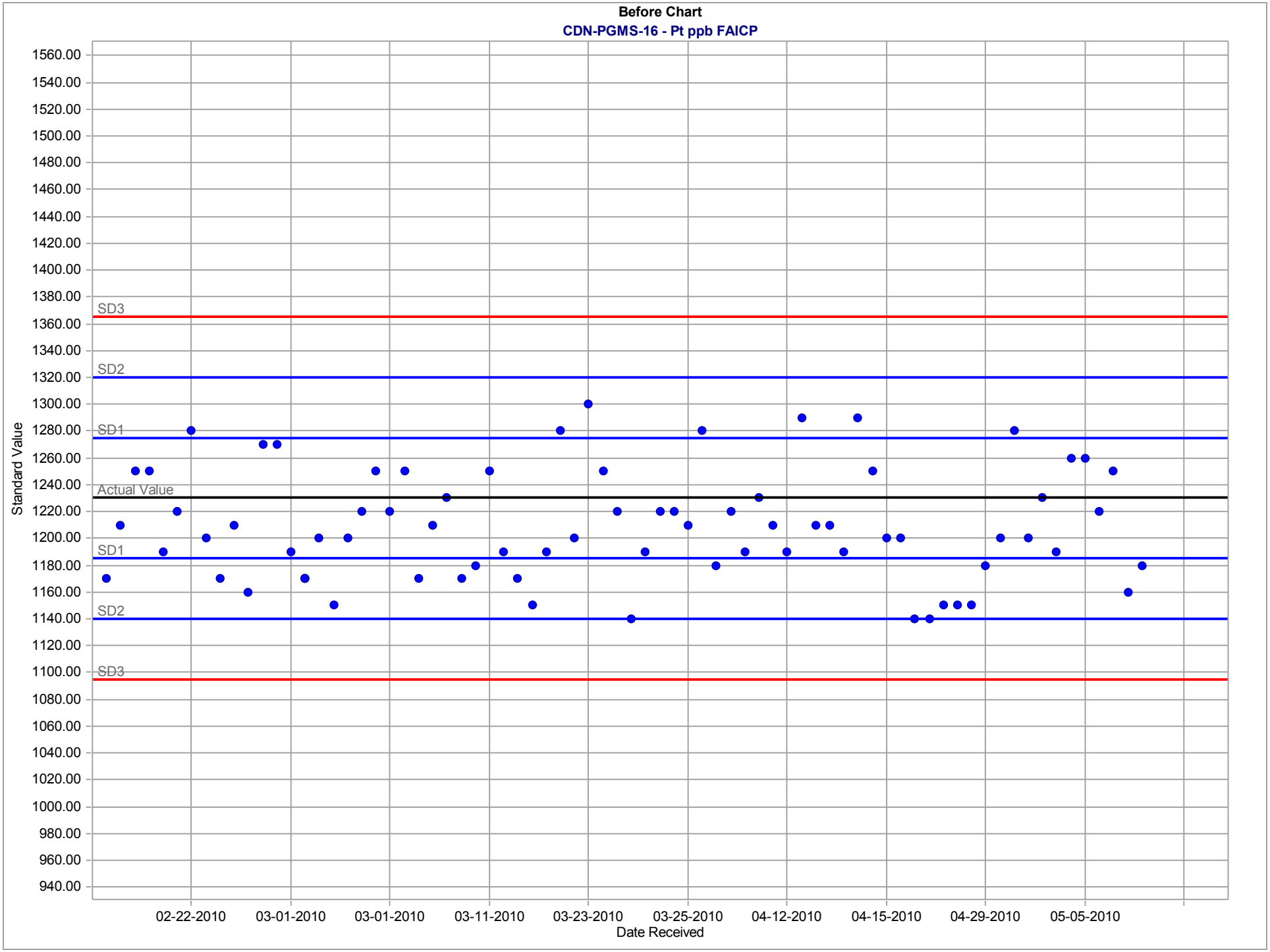
05-05-2010
Date Received


Before Chart
CDN-PGMS-16 - Pd ppb FAICP
Standard Value
5900.00
5800.00
5700.00
5600.00
5500.00
5400.00
5300.00
5200.00
5100.00
5000.00
4900.00
4800.00
4700.00
4600.00
4500.00
4400.00
4300.00
4200.00
4100.00
4000.00
3900.00
3800.00
3700.00
3600.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
02-22-2010
03-01-2010
03-01-2010
03-11-2010
03-23-2010
03-25-2010
04-12-2010
04-15-2010
04-29-2010
05-05-2010
Date Received


Before Chart
CDN-PGMS-16 - Pt ppb FAICP
Standard Value
1560.00
1540.00
1520.00
1500.00
1480.00
1460.00
1440.00
1420.00
1400.00
1380.00
1360.00
1340.00
1320.00
1300.00
1280.00
1260.00
1240.00
1220.00
1200.00
1180.00
1160.00
1140.00
1120.00
1100.00
1080.00
1060.00
1040.00
1020.00
1000.00
980.00
960.00
940.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
02-22-2010
03-01-2010
03-01-2010
03-11-2010
03-23-2010
03-25-2010
04-12-2010
04-15-2010
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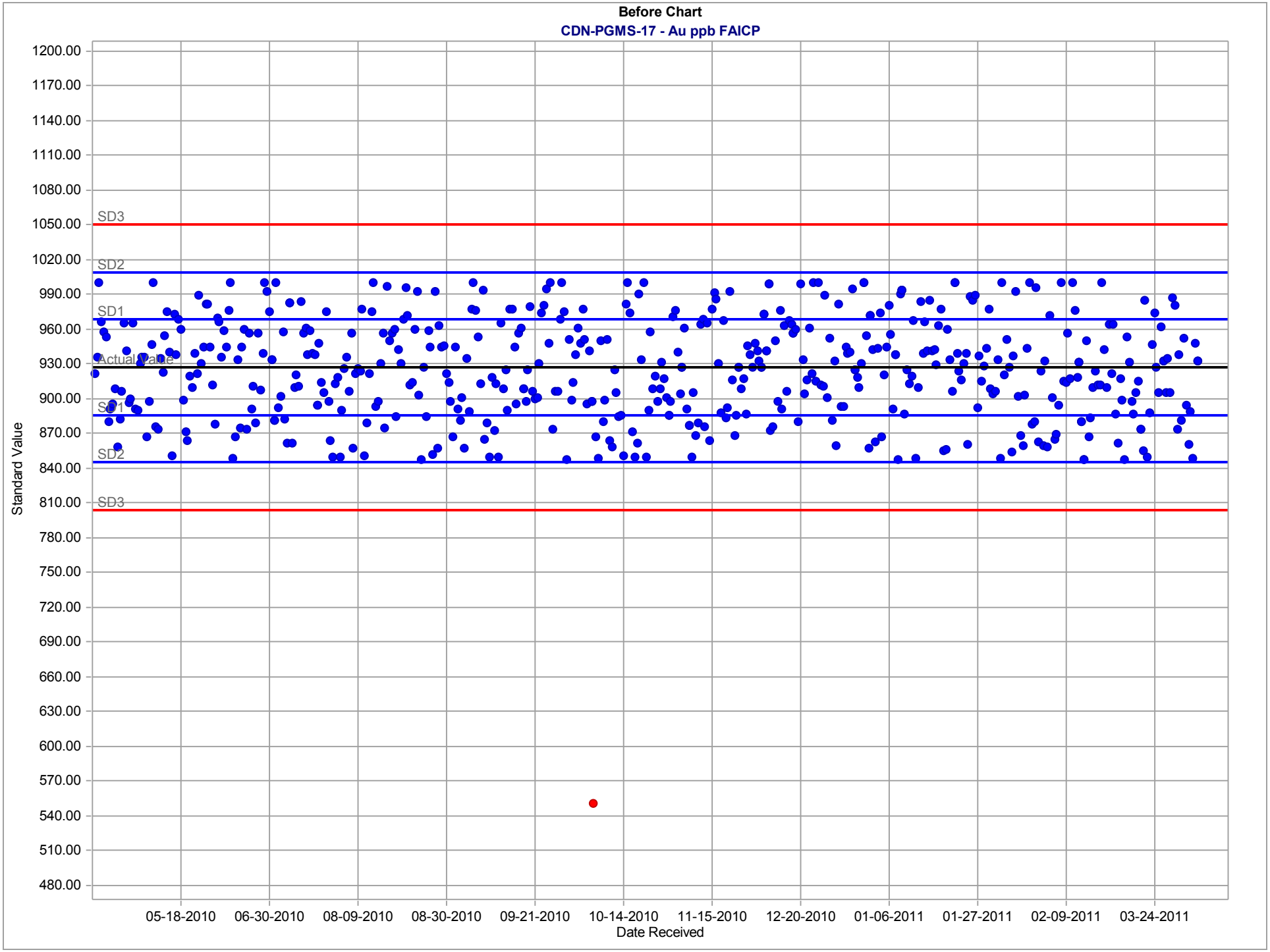
05-05-2010
Date Received


Before Chart
CDN-PGMS-17 - Au ppb FAICP
1200.00
1170.00
1140.00
1110.00
1080.00
1050.00
1020.00
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930.00
900.00
870.00
840.00
810.00
780.00
750.00
720.00
690.00
660.00
630.00
600.00
570.00
540.00
510.00
480.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
05-18-2010
06-30-2010
08-09-2010
08-30-2010
09-21-2010
10-14-2010
11-15-2010
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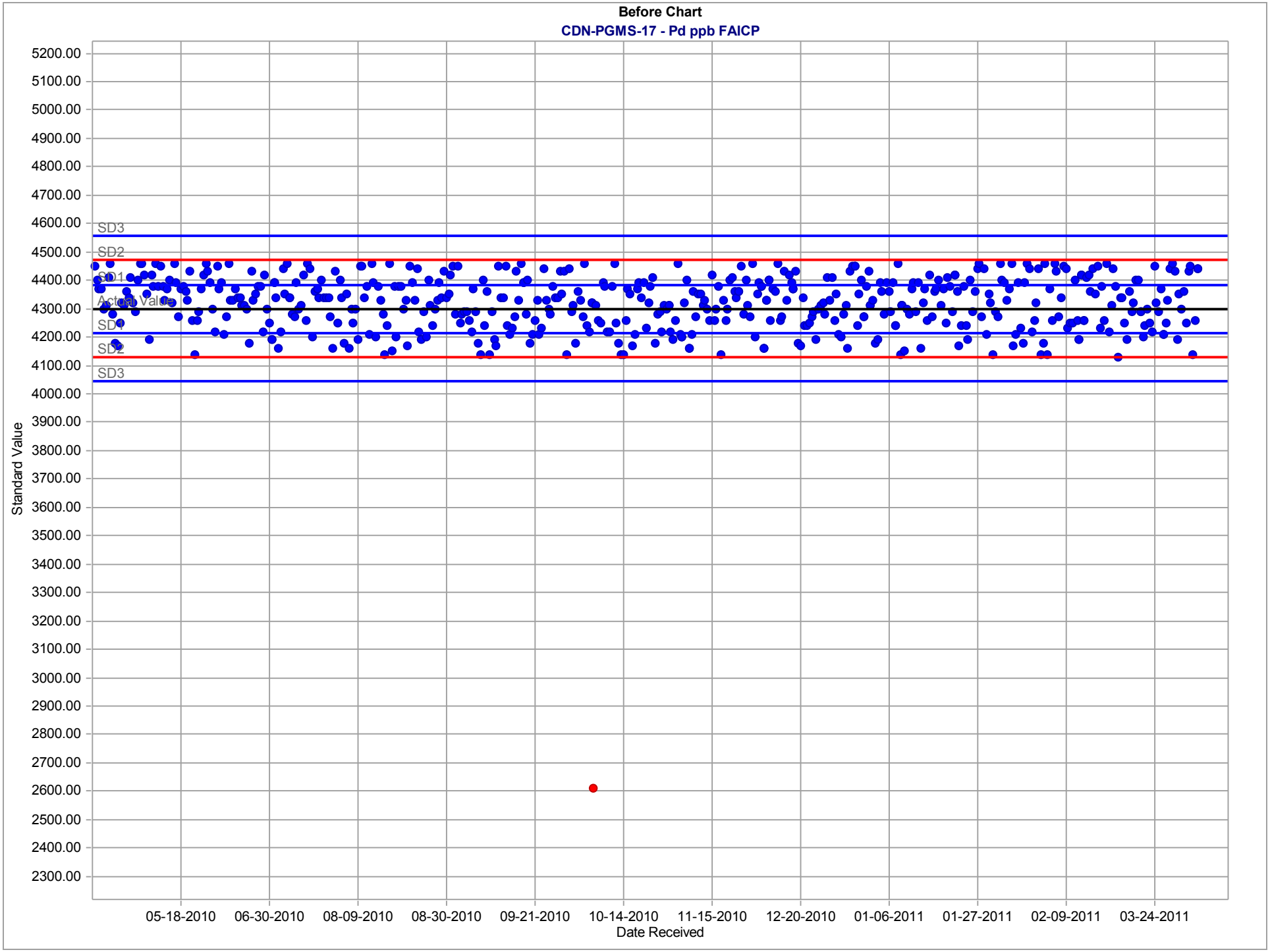
01-06-2011
01-27-2011
02-09-2011
03-24-2011
Date Received


Before Chart
CDN-PGMS-17 - Pd ppb FAICP
5200.00
5100.00
5000.00
4900.00
4800.00
4700.00
4600.00
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4400.00
4300.00
4200.00
4100.00
4000.00
3900.00
3800.00
3700.00
3600.00
3500.00
3400.00
3300.00
3200.00
3100.00
3000.00
2900.00
2800.00
2700.00
2600.00
2500.00
2400.00
2300.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
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06-30-2010
08-09-2010
08-30-2010
09-21-2010
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11-15-2010
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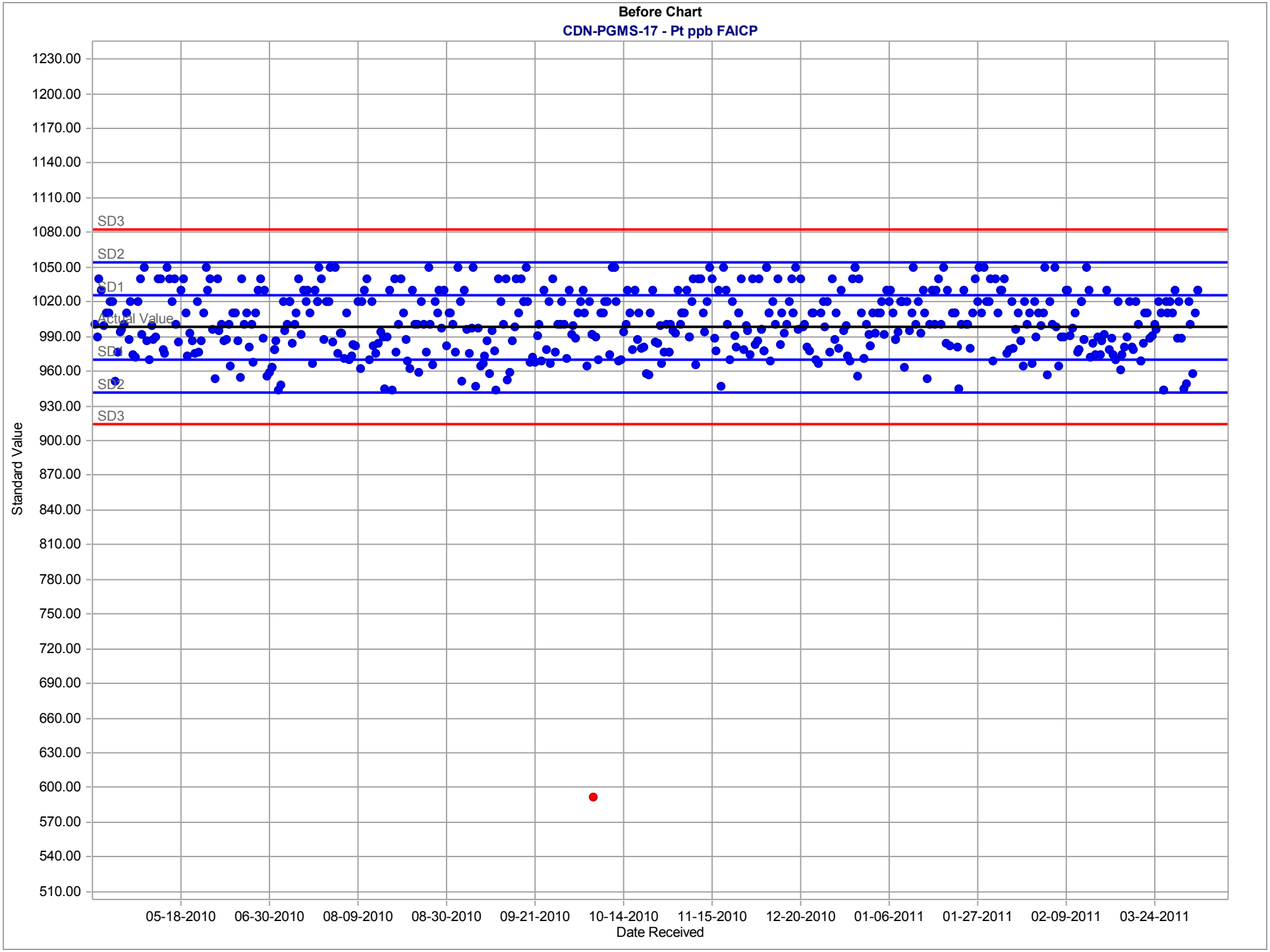
01-06-2011
01-27-2011
02-09-2011
03-24-2011
Date Received


Before Chart
CDN-PGMS-17 - Pt ppb FAICP
1230.00
1200.00
1170.00
1140.00
1110.00
1080.00
1050.00
1020.00
990.00
960.00
930.00
900.00
870.00
840.00
810.00
780.00
750.00
720.00
690.00
660.00
630.00
600.00
570.00
540.00
510.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
05-18-2010
06-30-2010
08-09-2010
08-30-2010
09-21-2010
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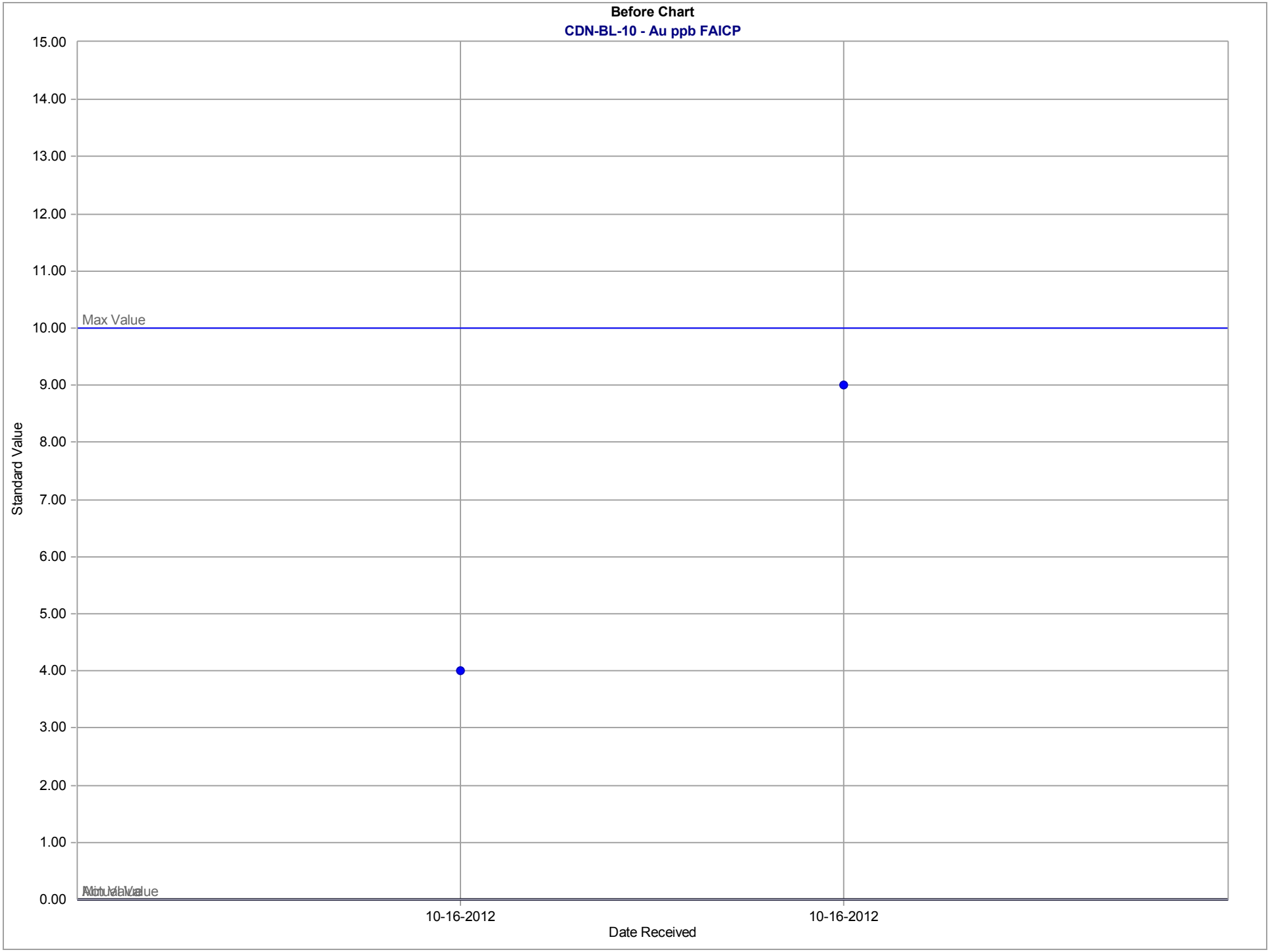
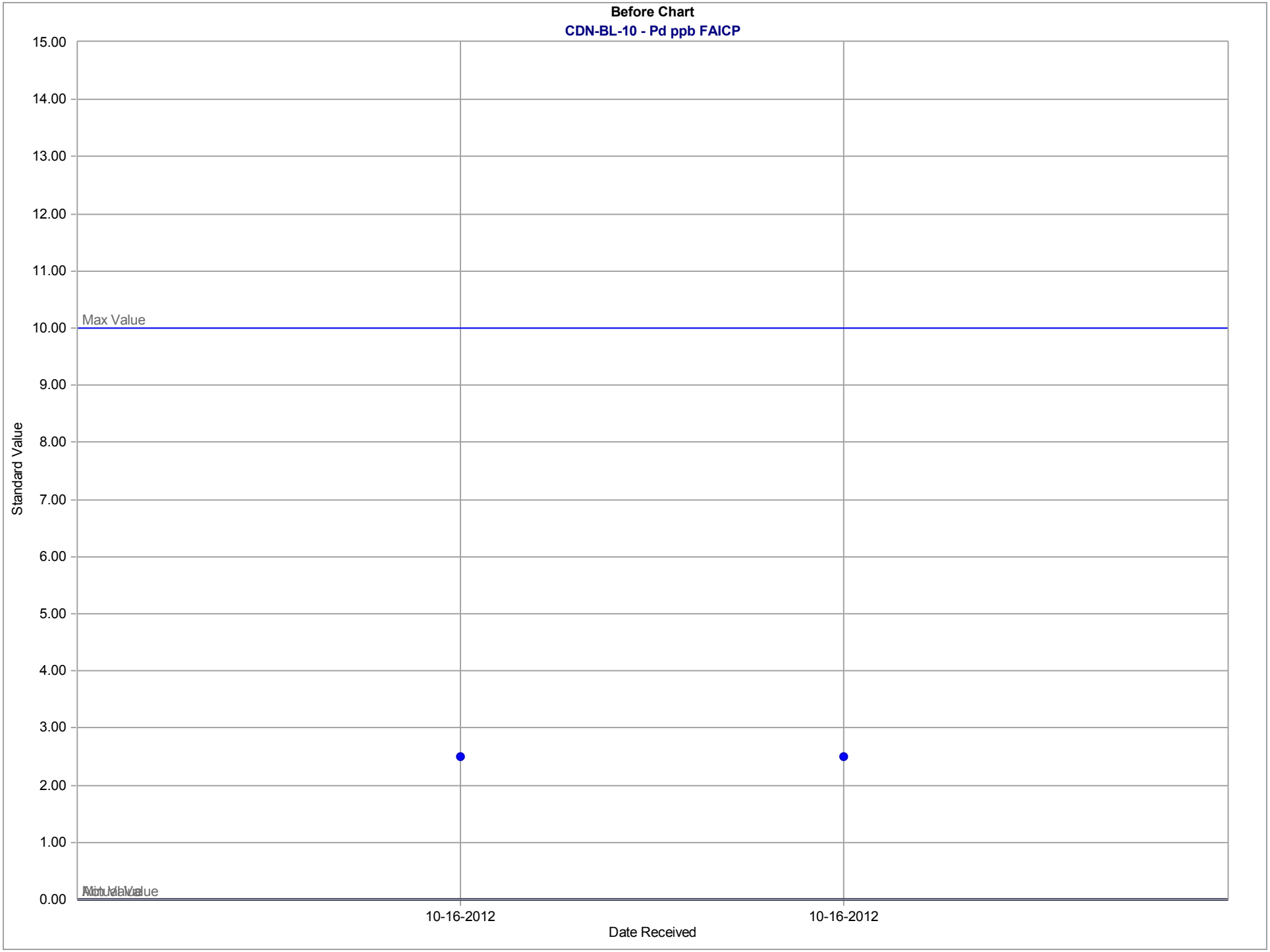
11-15-2010
12-20-2010
01-06-2011
01-27-2011
02-09-2011
03-24-2011
Date Received


Before Chart
CDN-BL-10 - Au ppb FAICP
15.00
14.00
13.00
12.00
11.00
Max Value
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Standard Value
10-16-2012
10-16-2012
Date Received


Before Chart
CDN-BL-10 - Pd ppb FAICP
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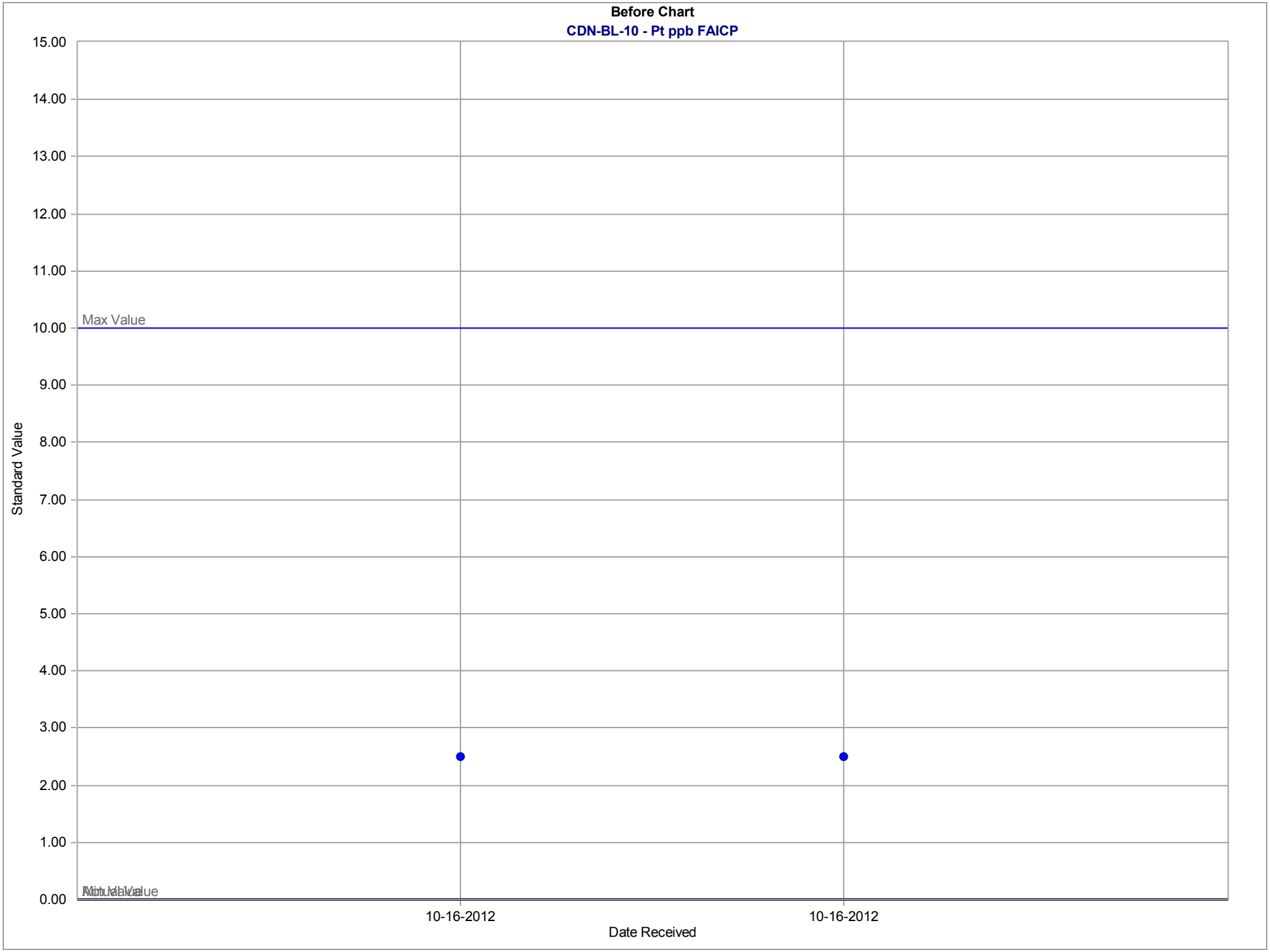
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
10-16-2012
10-16-2012
Date Received


Before Chart
CDN-BL-10 - Pt ppb FAICP
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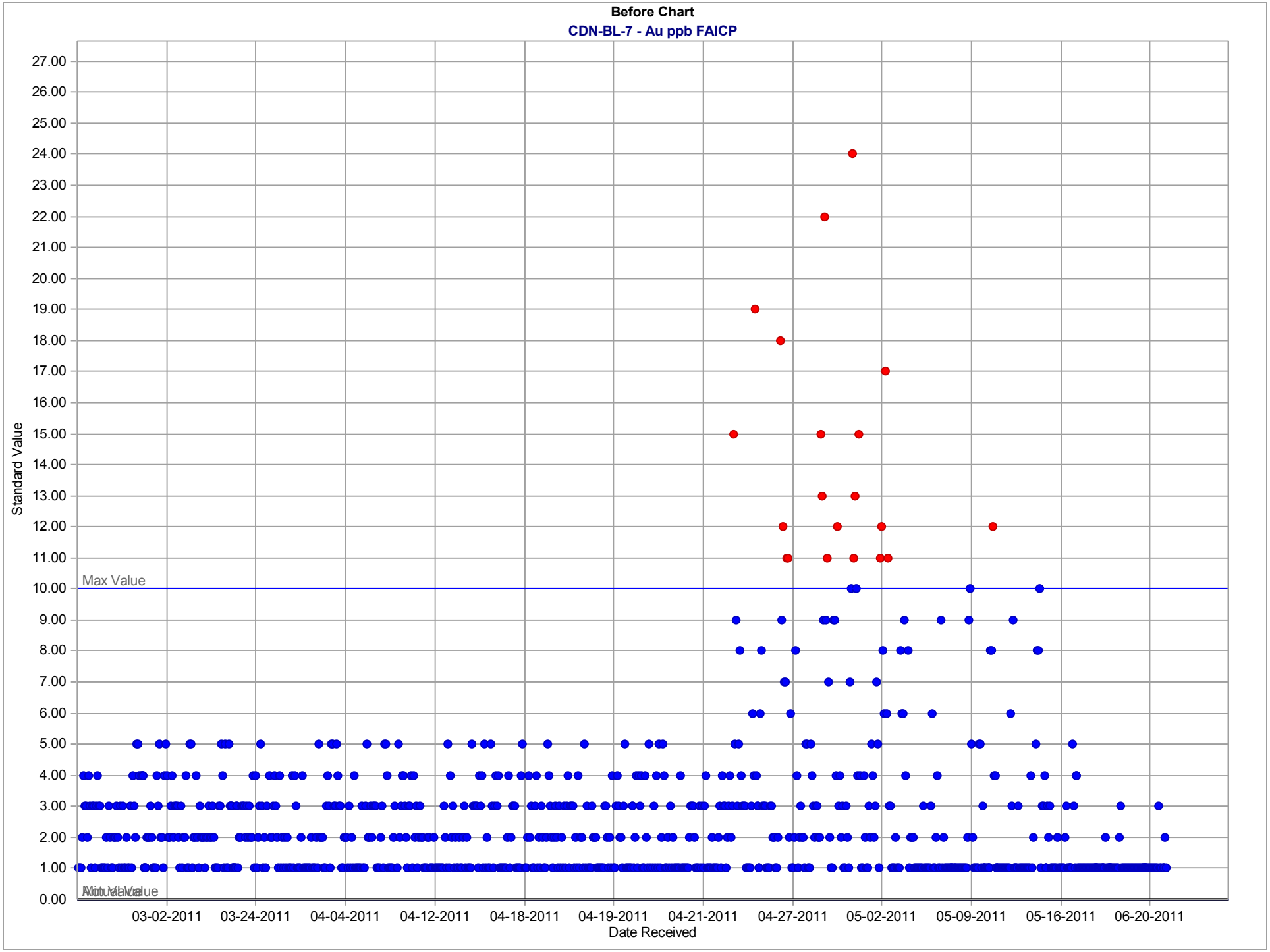
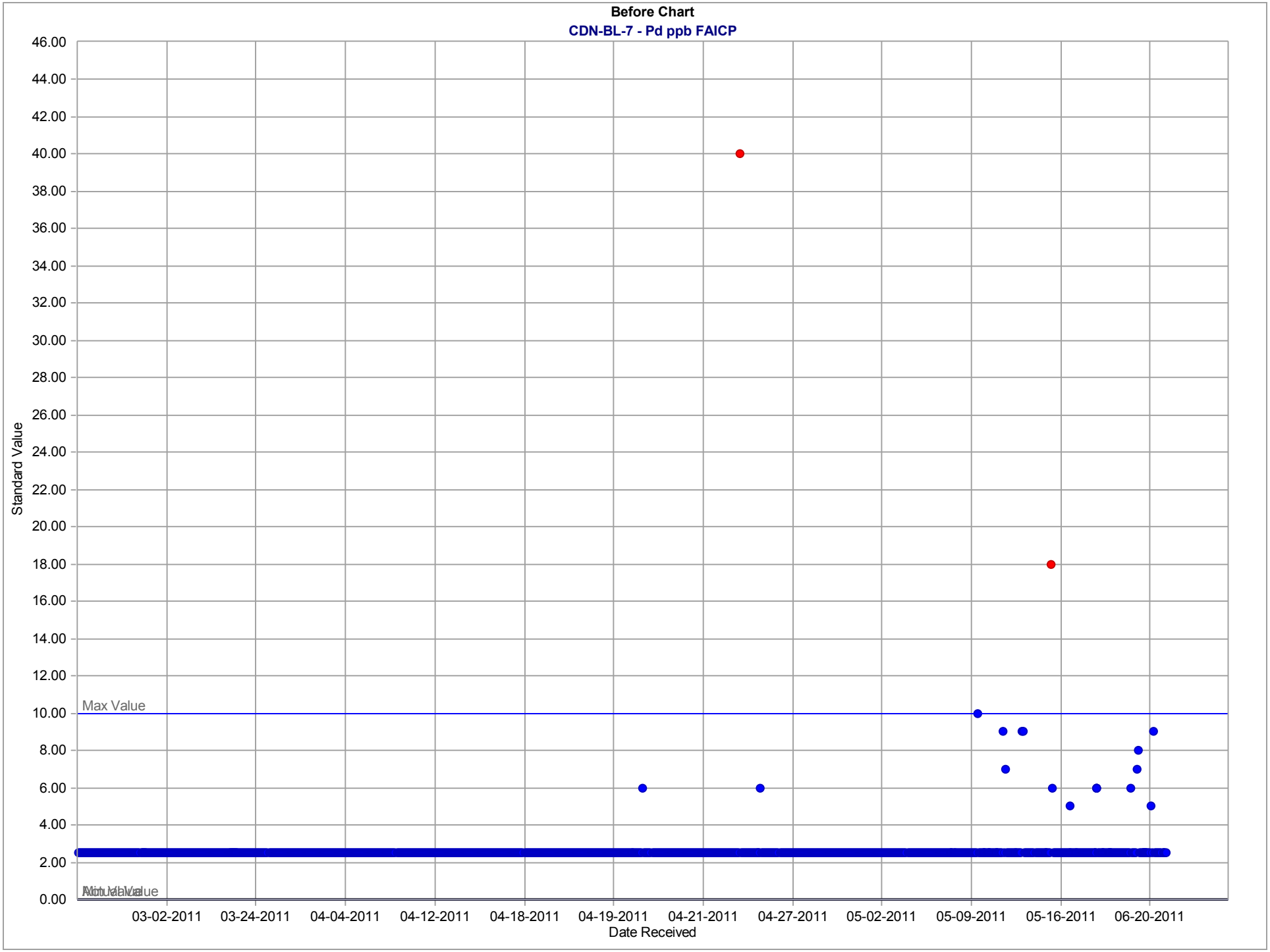
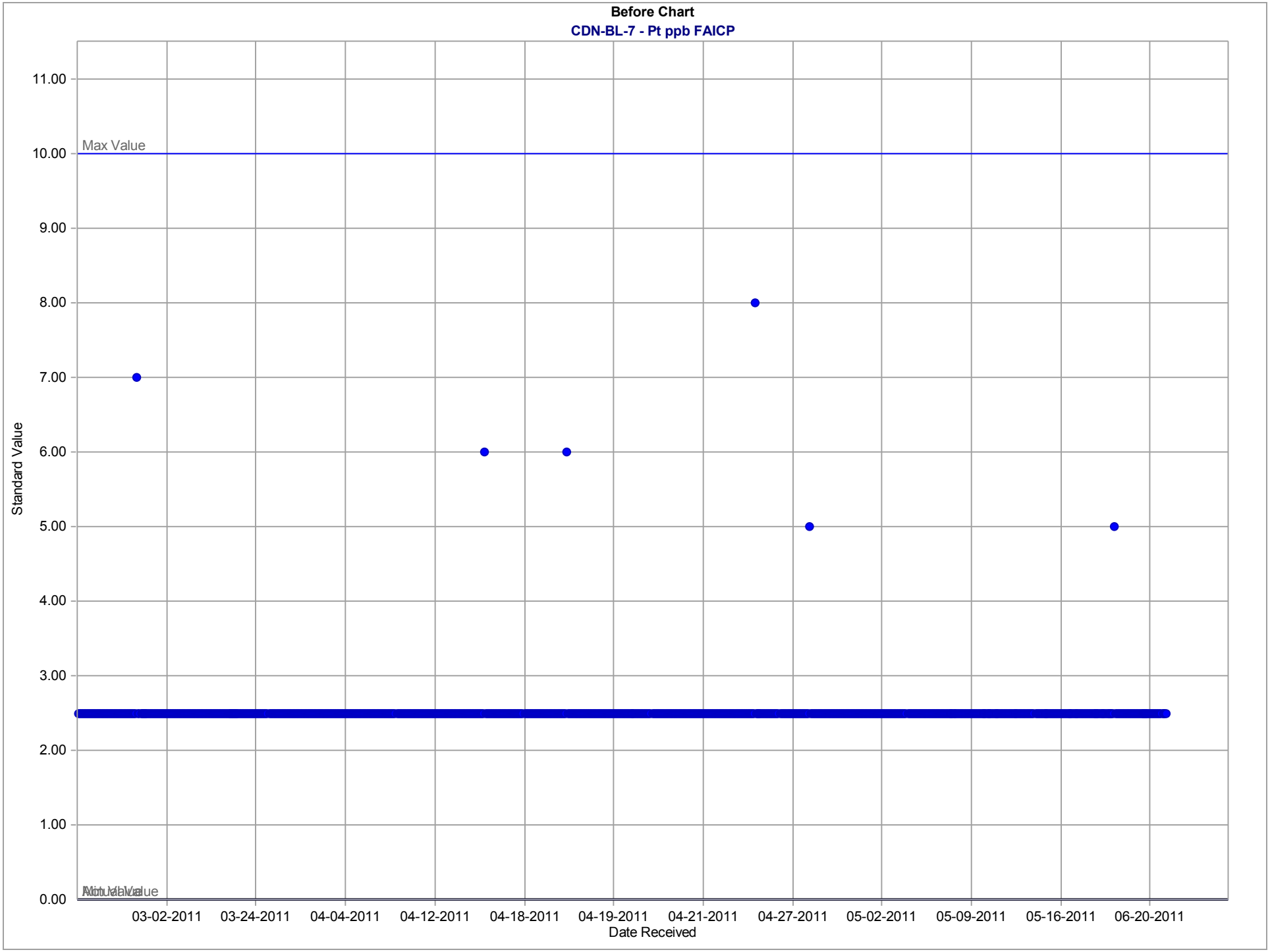
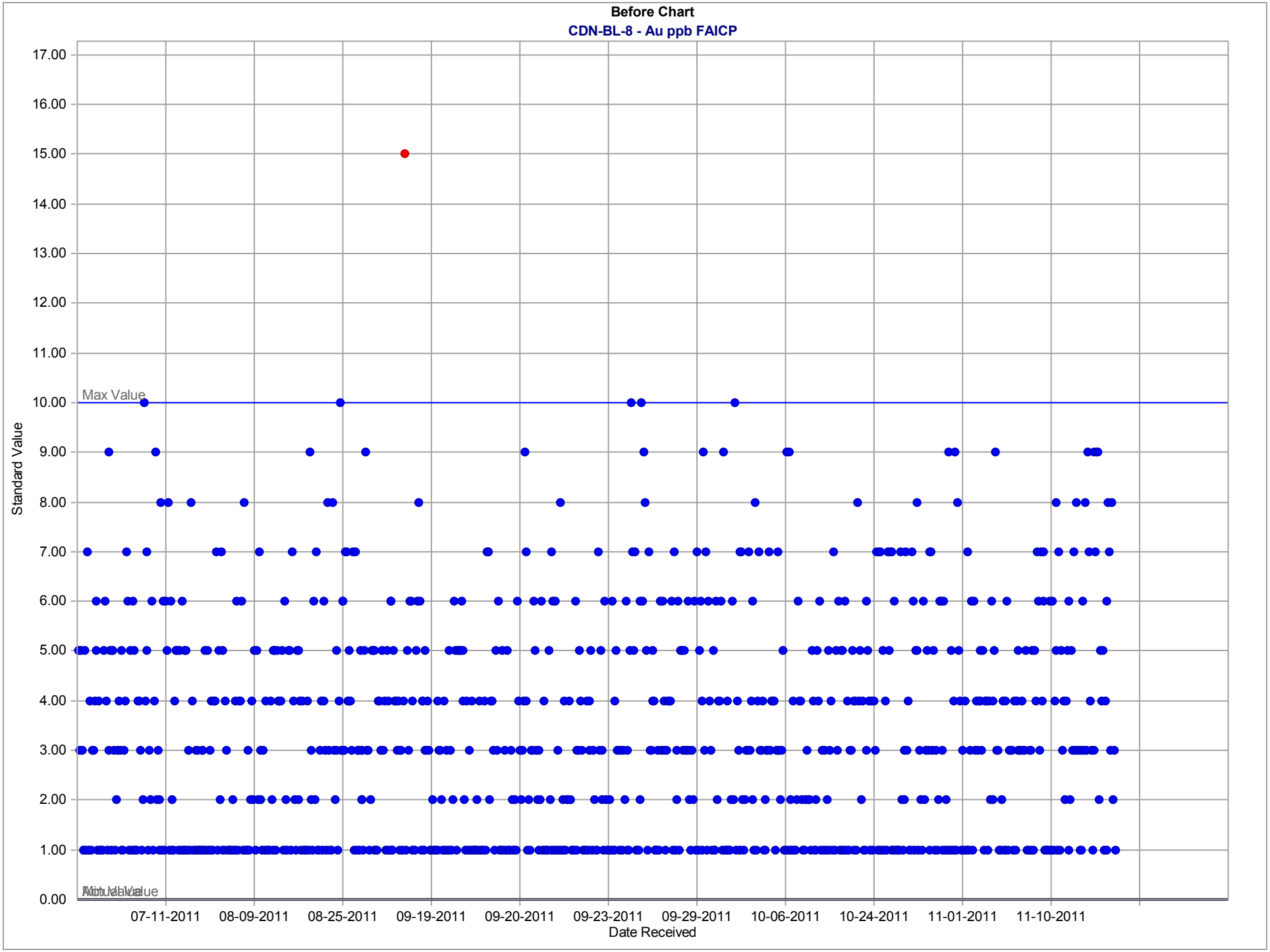
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
10-16-2012
10-16-2012
Date Received


Before Chart
CDN-BL-7 - Au ppb FAICP
Standard Value
Date Received
Max Value
27.00
26.00
25.00
24.00
23.00
22.00
21.00
20.00
19.00
18.00
17.00
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
03-02-2011
03-24-2011
04-04-2011
04-12-2011
04-18-2011
04-19-2011
04-21-2011
04-27-2011
05-02-2011
05-09-2011
05-16-2011
06-20-2011


Before Chart
CDN-BL-7 - Pd ppb FAICP
46.00
44.00
42.00
40.00
38.00
36.00
34.00
32.00
30.00
28.00
26.00
24.00
22.00
20.00
18.00
16.00
14.00
12.00
10.00
8.00
6.00
4.00
2.00
0.00
Standard Value
Max Value
03-02-2011
03-24-2011
04-04-2011
04-12-2011
04-18-2011
04-19-2011
04-21-2011
04-27-2011
05-02-2011
05-09-2011
05-16-2011
06-20-2011
Date Received


Before Chart
CDN-BL-7 - Pt ppb FAICP
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
03-02-2011
03-24-2011
04-04-2011
04-12-2011
04-18-2011
04-19-2011
04-21-2011
04-27-2011
05-02-2011
05-09-2011
05-16-2011
06-20-2011
Date Received


Before Chart
CDN-BL-8 - Au ppb FAICP
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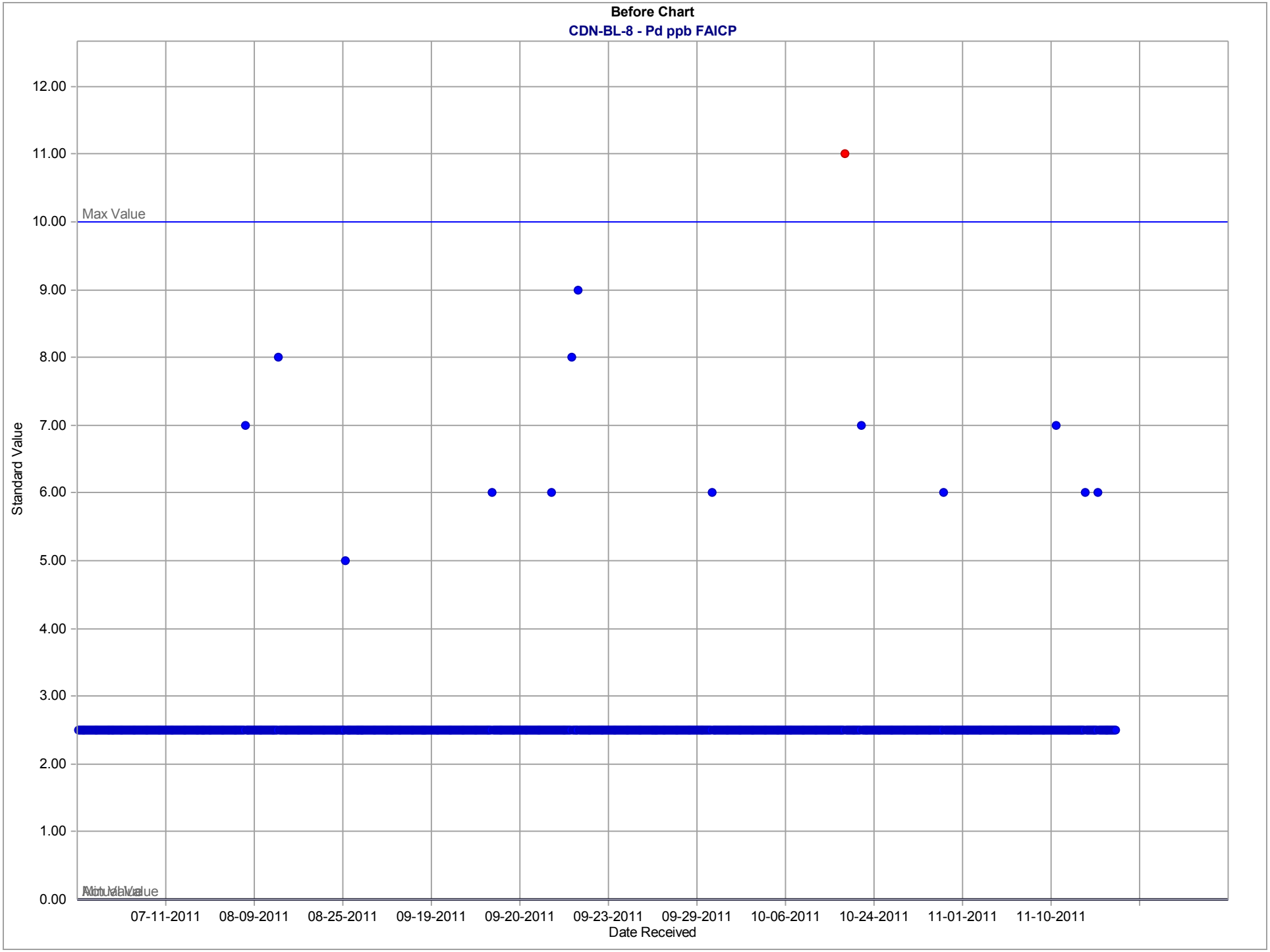
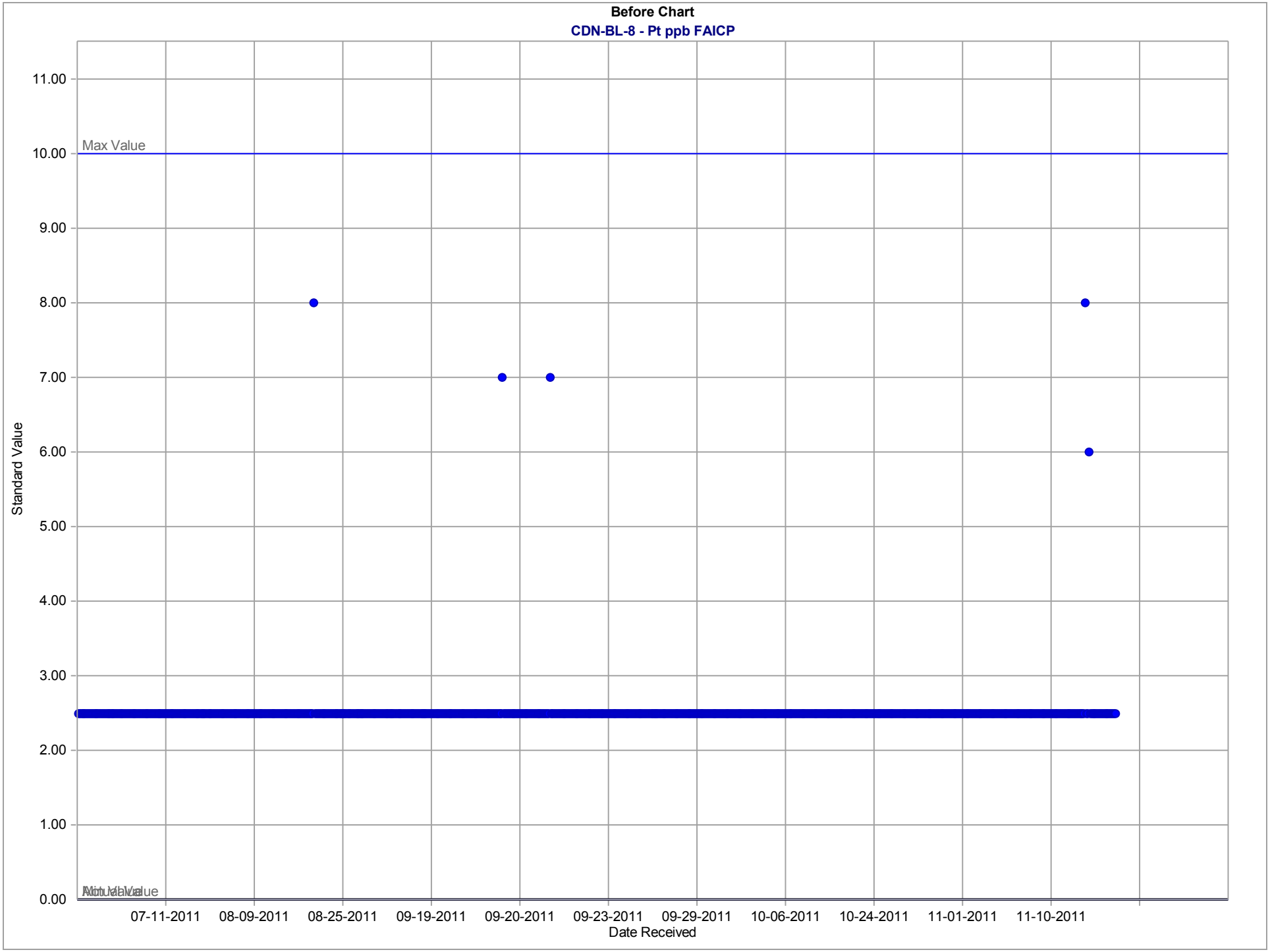
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
07-11-2011
08-09-2011
08-25-2011
09-19-2011
09-20-2011
09-23-2011
09-29-2011
10-06-2011
10-24-2011
11-01-2011
11-10-2011
Date Received


Before Chart
CDN-BL-8 - Pd ppb FAICP
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
07-11-2011
08-09-2011
08-25-2011
09-19-2011
09-20-2011
09-23-2011
09-29-2011
10-06-2011
10-24-2011
11-01-2011
11-10-2011
Date Received


Before Chart
CDN-BL-8 - Pt ppb FAICP
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10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
07-11-2011
08-09-2011
08-25-2011
09-19-2011
09-20-2011
09-23-2011
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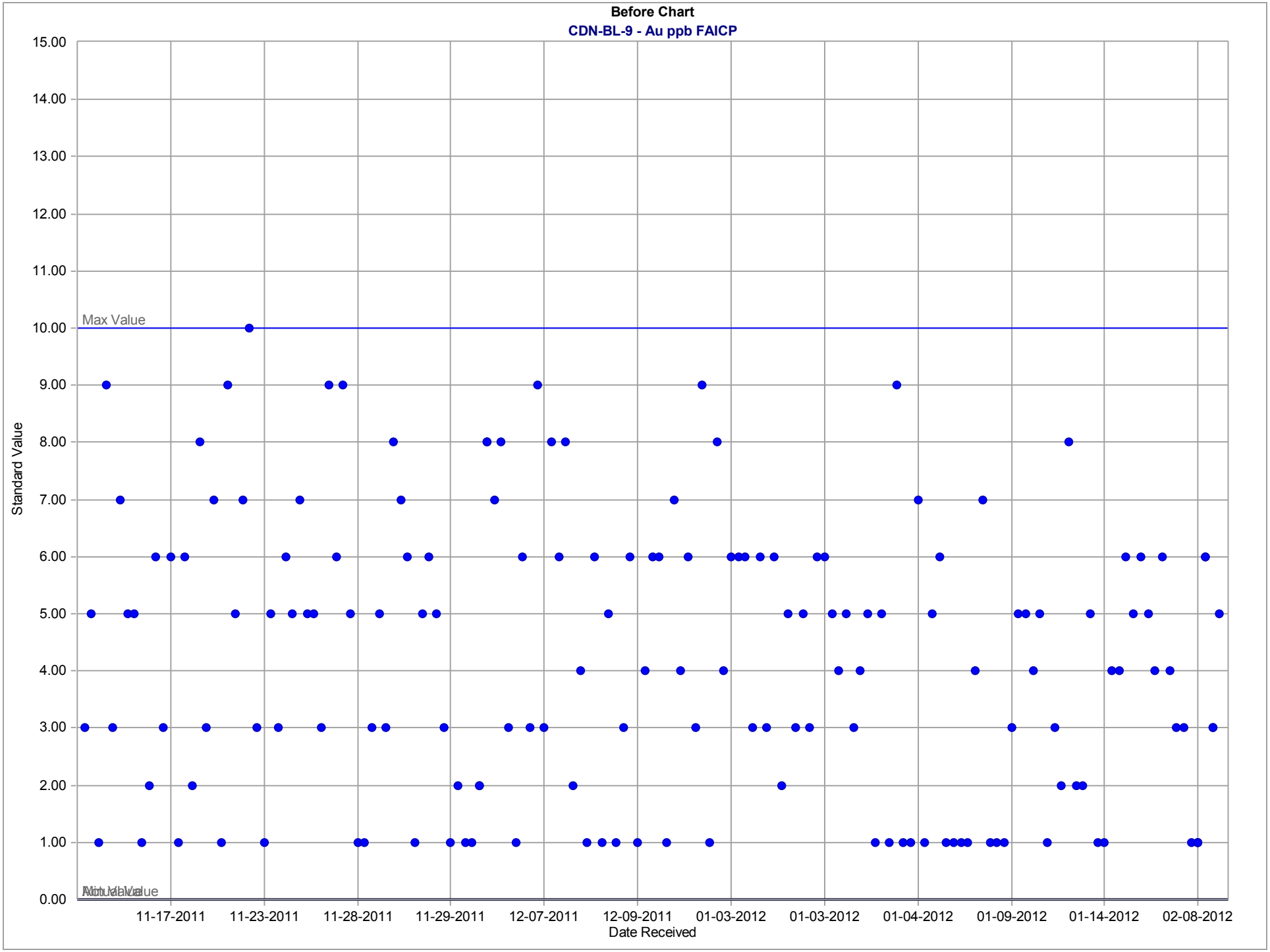
11-10-2011
Date Received


Before Chart
CDN-BL-9 - Au ppb FAICP
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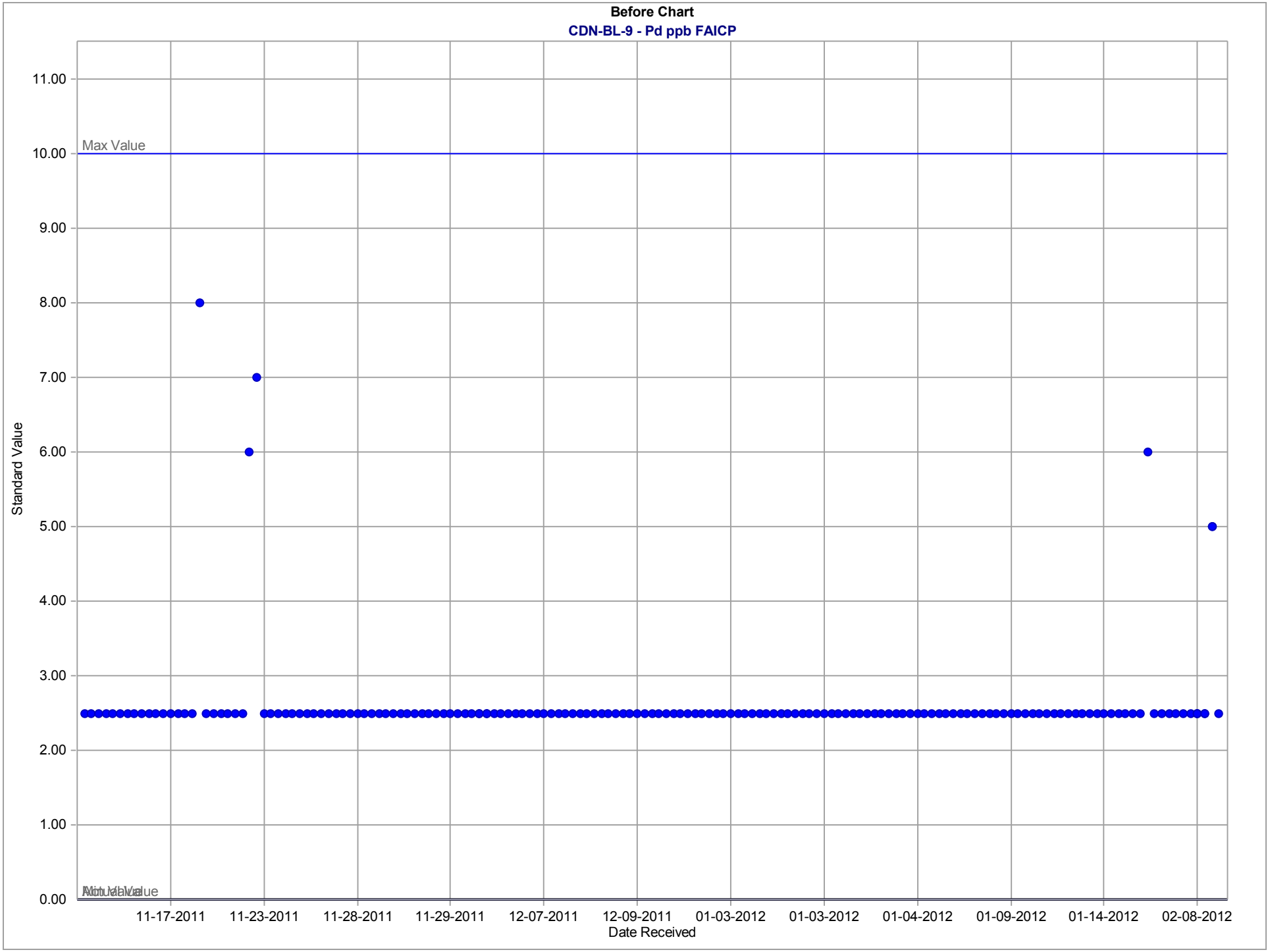
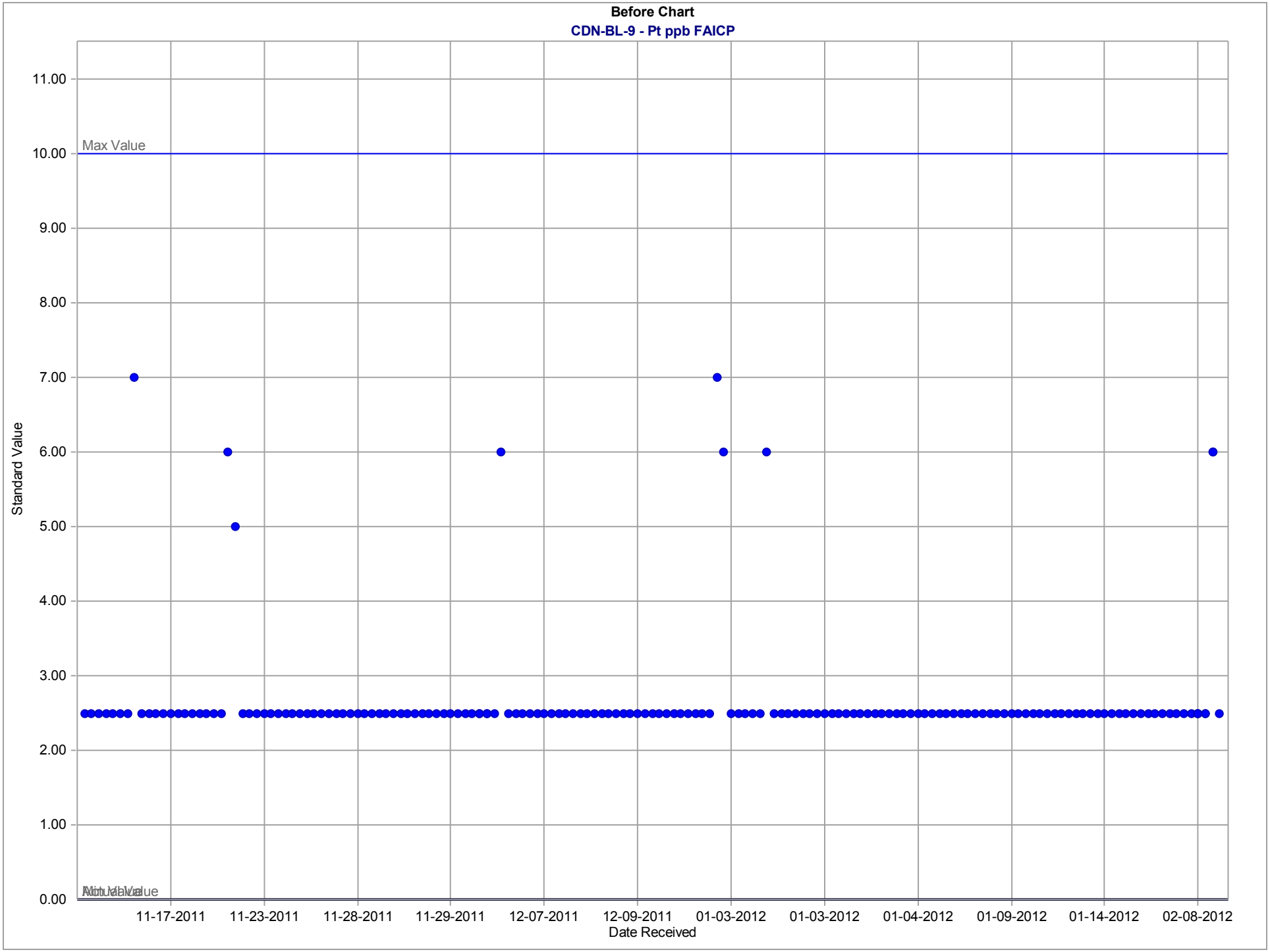
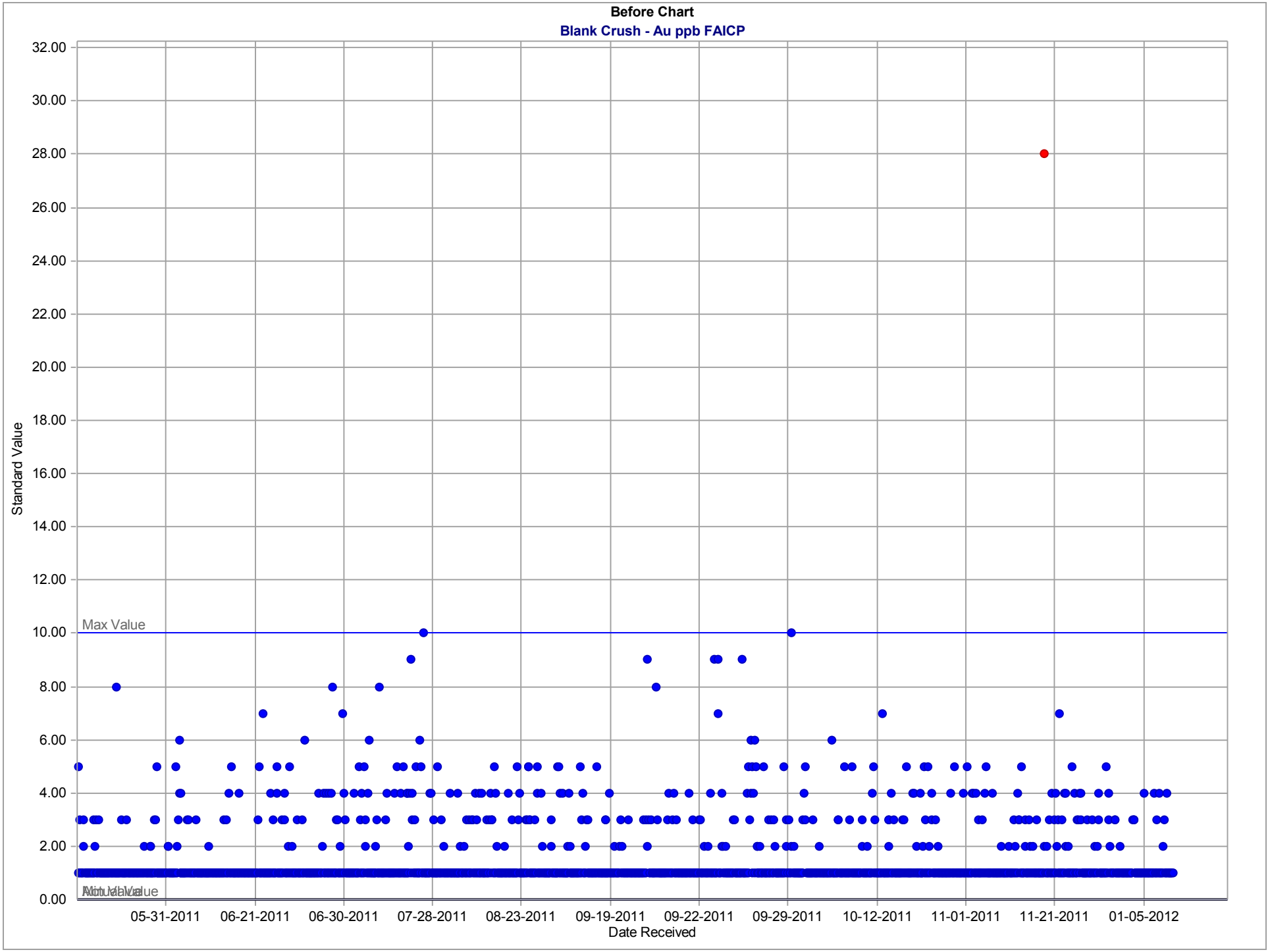
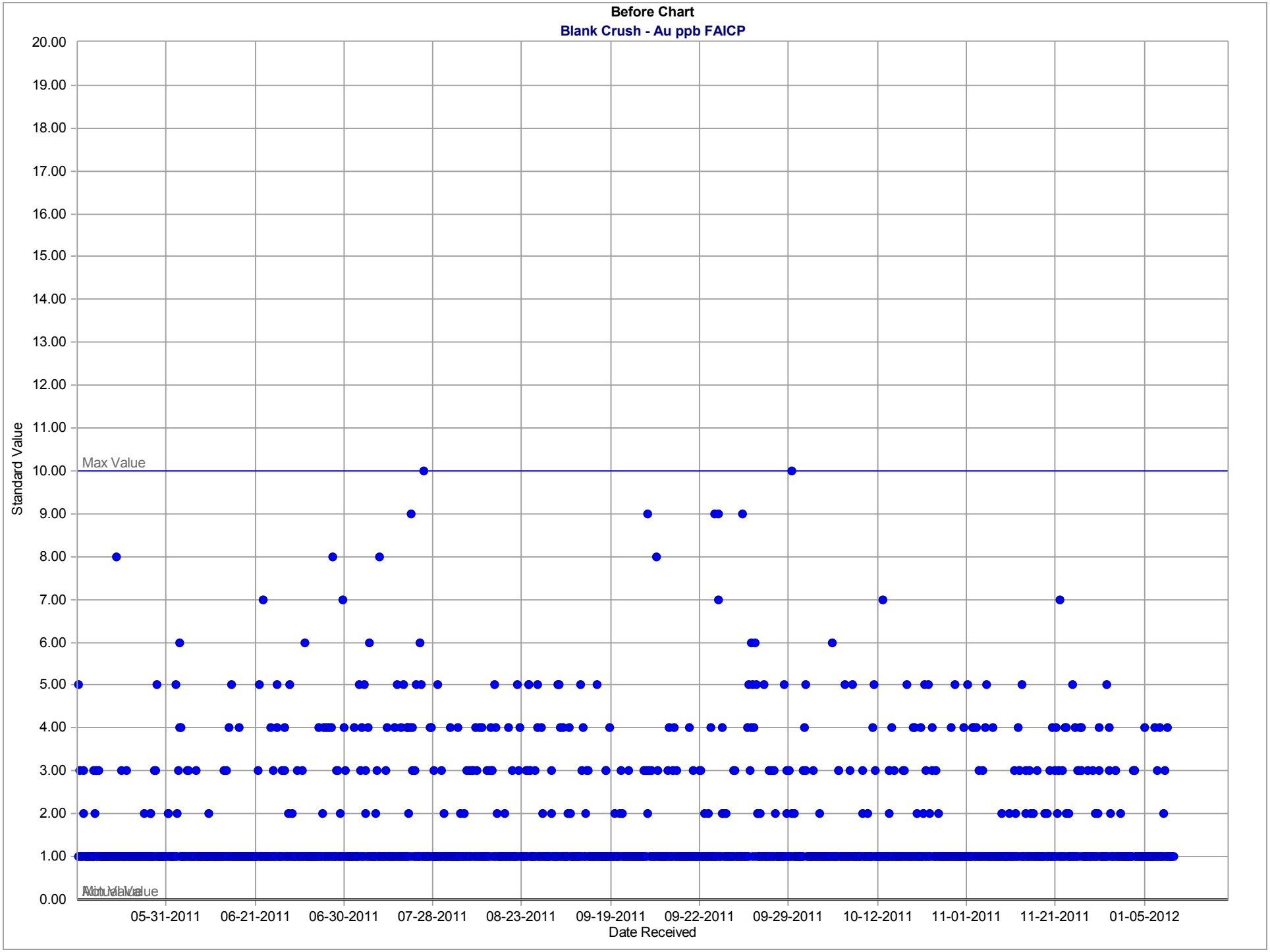
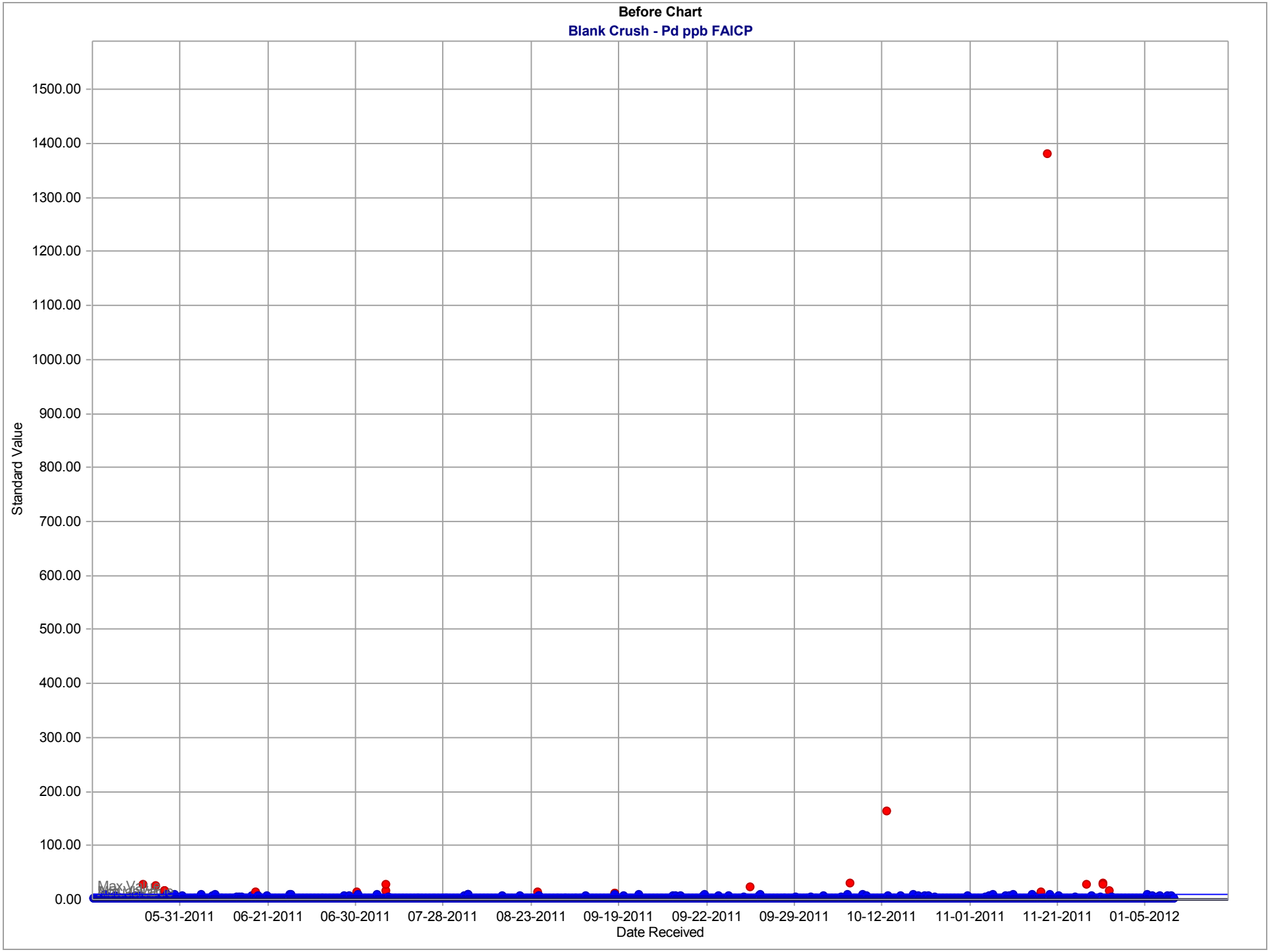
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
11-17-2011
11-23-2011
11-28-2011
11-29-2011
12-07-2011
12-09-2011
01-03-2012
01-03-2012
01-04-2012
01-09-2012
01-14-2012
02-08-2012
Date Received

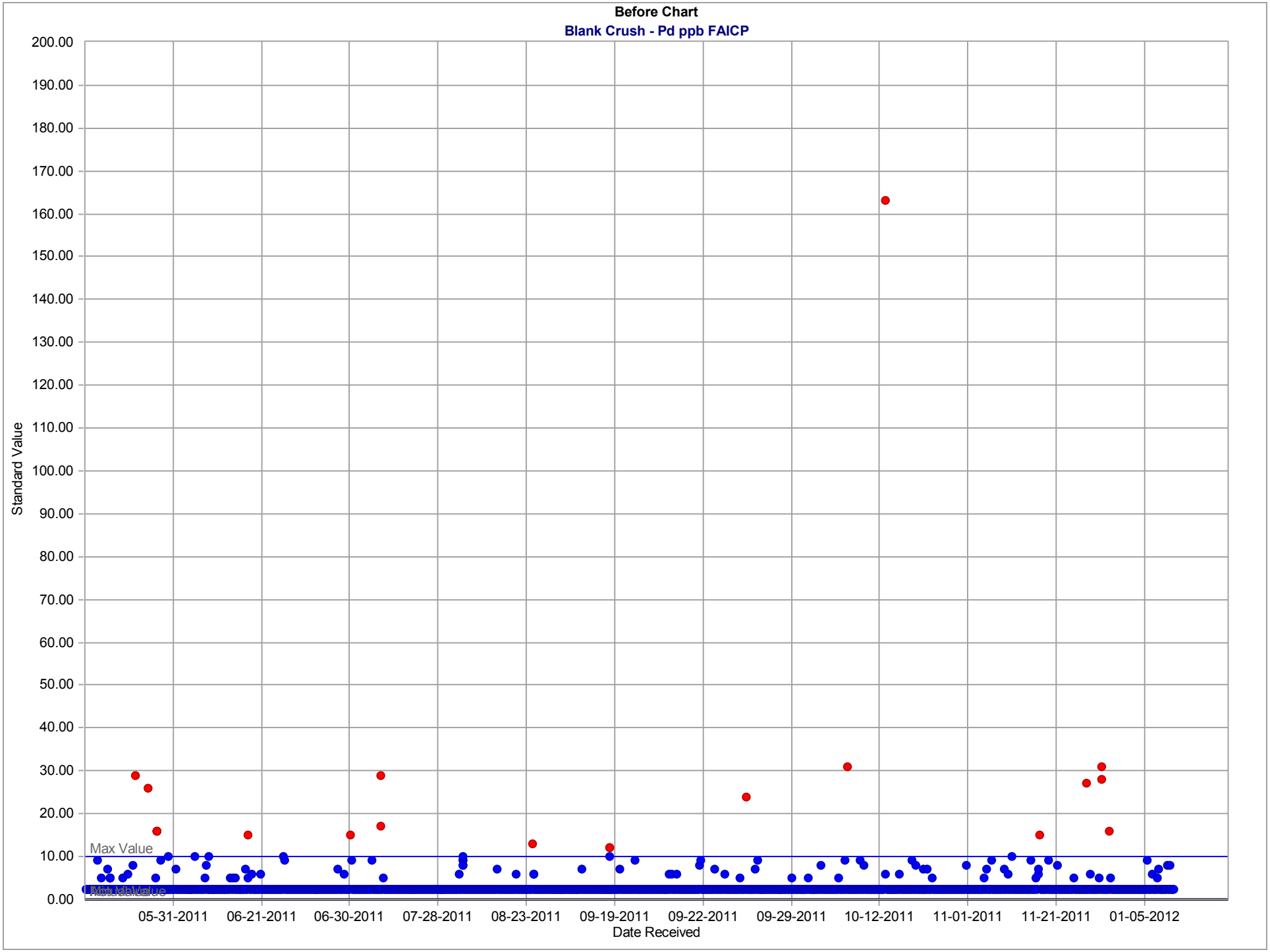

Before Chart
CDN-BL-9 - Pd ppb FAICP
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
11-17-2011
11-23-2011
11-28-2011
11-29-2011
12-07-2011
12-09-2011
01-03-2012
01-03-2012
01-04-2012
01-09-2012
01-14-2012
02-08-2012
Date Received

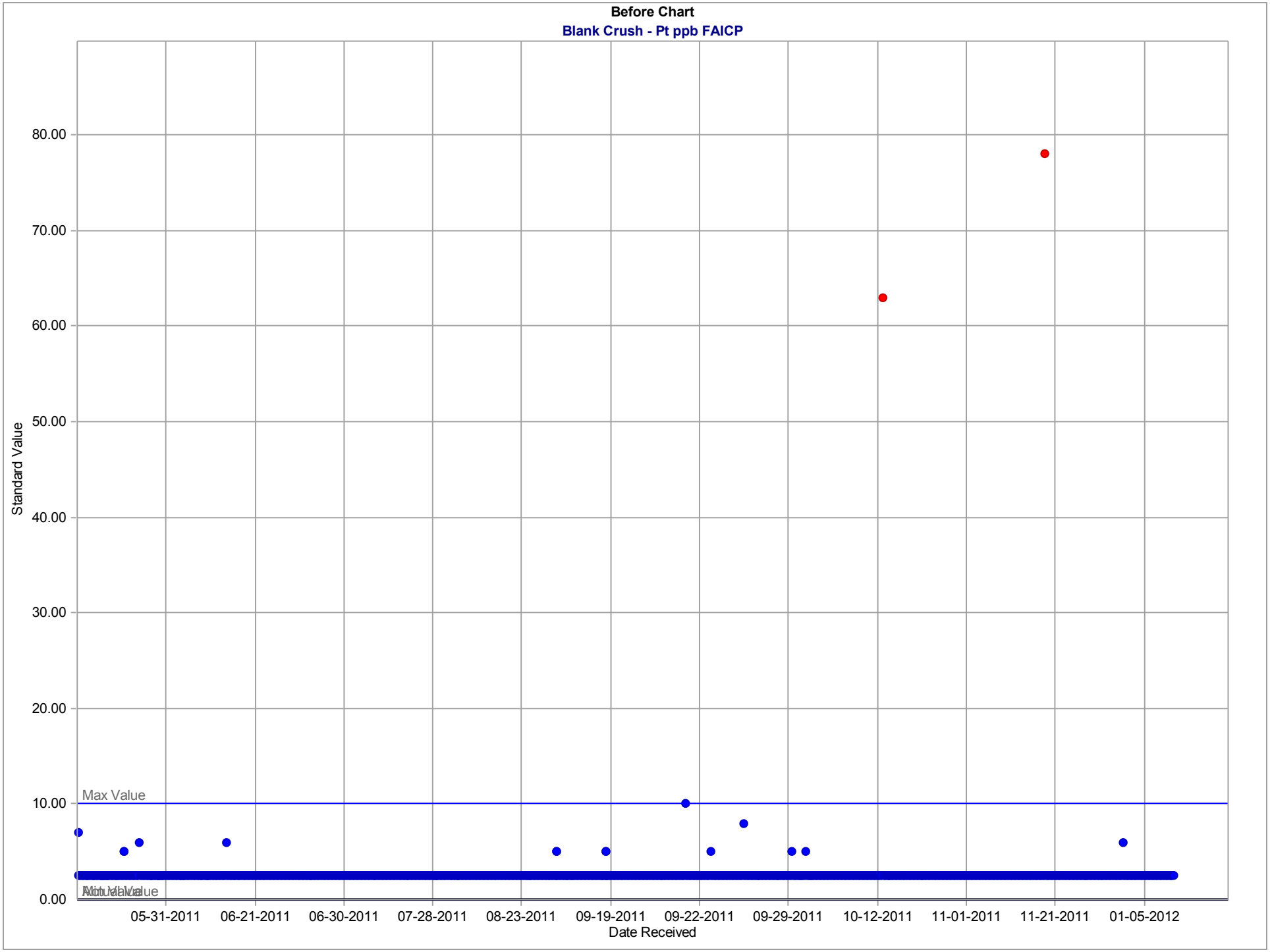

Before Chart
CDN-BL-9 - Pt ppb FAICP
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
11-17-2011
11-23-2011
11-28-2011
11-29-2011
12-07-2011
12-09-2011
01-03-2012
01-03-2012
01-04-2012
01-09-2012
01-14-2012
02-08-2012
Date Received


Before Chart
Blank Crush - Au ppb FAICP
32.00
30.00
28.00
26.00
24.00
22.00
20.00
18.00
16.00
14.00
12.00
10.00
8.00
6.00
4.00
2.00
0.00
Standard Value
Max Value
05-31-2011
06-21-2011
06-30-2011
07-28-2011
08-23-2011
09-19-2011
09-22-2011
09-29-2011
10-12-2011
11-01-2011
11-21-2011
01-05-2012
Date Received


Before Chart
Blank Crush - Au ppb FAICP
20.00
19.00
18.00
17.00
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Standard Value
Max Value
05-31-2011
06-21-2011
06-30-2011
07-28-2011
08-23-2011
09-19-2011
09-22-2011
09-29-2011
10-12-2011
11-01-2011
11-21-2011
01-05-2012
Date Received


Before Chart
Blank Crush - Pd ppb FAICP
1500.00
1400.00
1300.00
1200.00
1100.00
1000.00
900.00
800.00
700.00
600.00
500.00
400.00
300.00
200.00
100.00
0.00
Standard Value
05-31-2011
06-21-2011
06-30-2011
07-28-2011
08-23-2011
09-19-2011
09-22-2011
09-29-2011
10-12-2011
11-01-2011
11-21-2011
01-05-2012
Date Received


Before Chart
Blank Crush - Pd ppb FAICP
200.00
190.00
180.00
170.00
160.00
150.00
140.00
130.00
120.00
110.00
100.00
90.00
80.00
70.00
60.00
50.00
40.00
30.00
20.00
10.00
0.00
Standard Value
Max Value
05-31-2011
06-21-2011
06-30-2011
07-28-2011
08-23-2011
09-19-2011
09-22-2011
09-29-2011
10-12-2011
11-01-2011
11-21-2011
01-05-2012
Date Received


Before Chart
Blank Crush - Pt ppb FAICP
80.00
70.00
60.00
50.00
40.00
30.00
20.00
10.00
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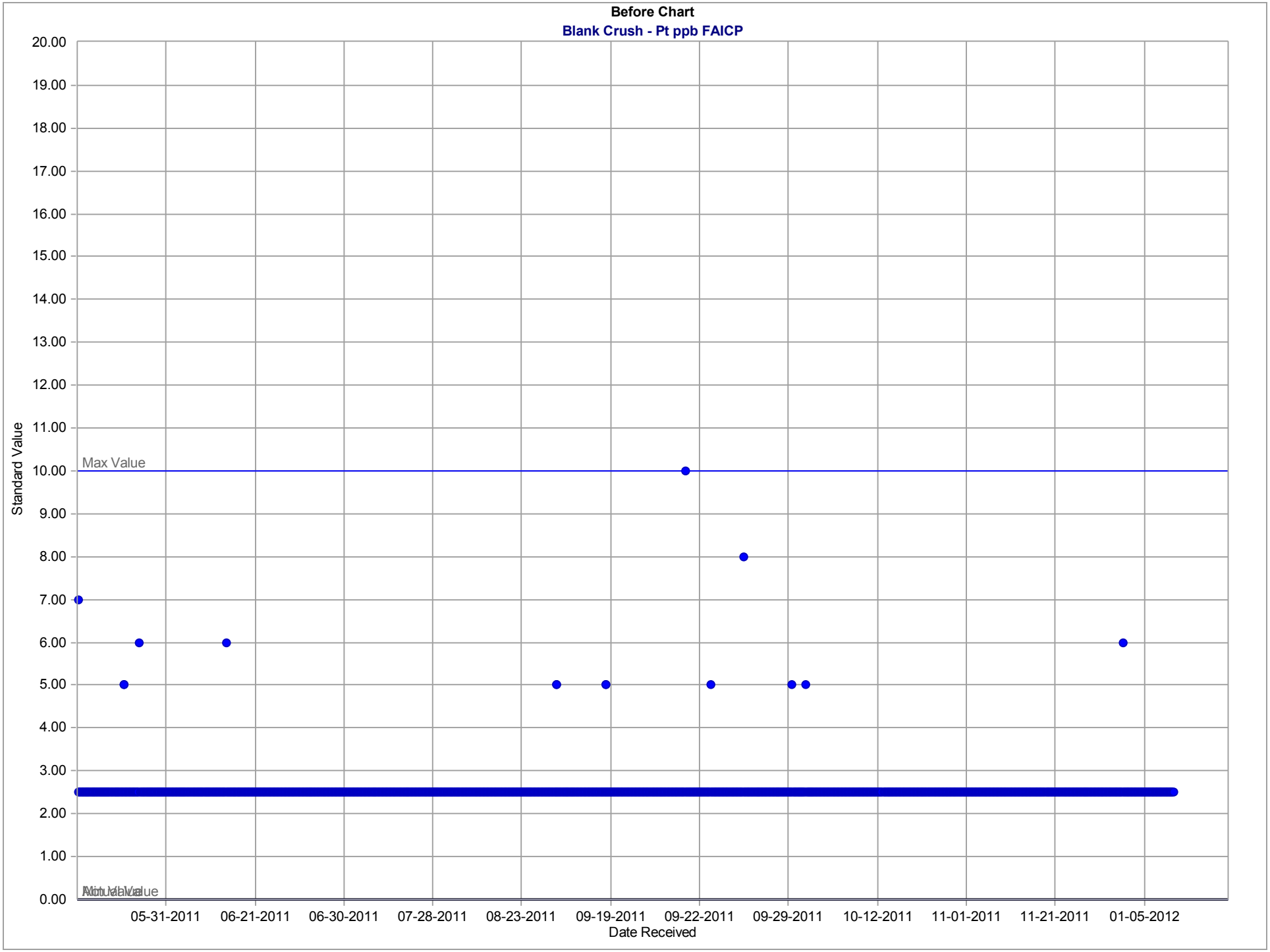
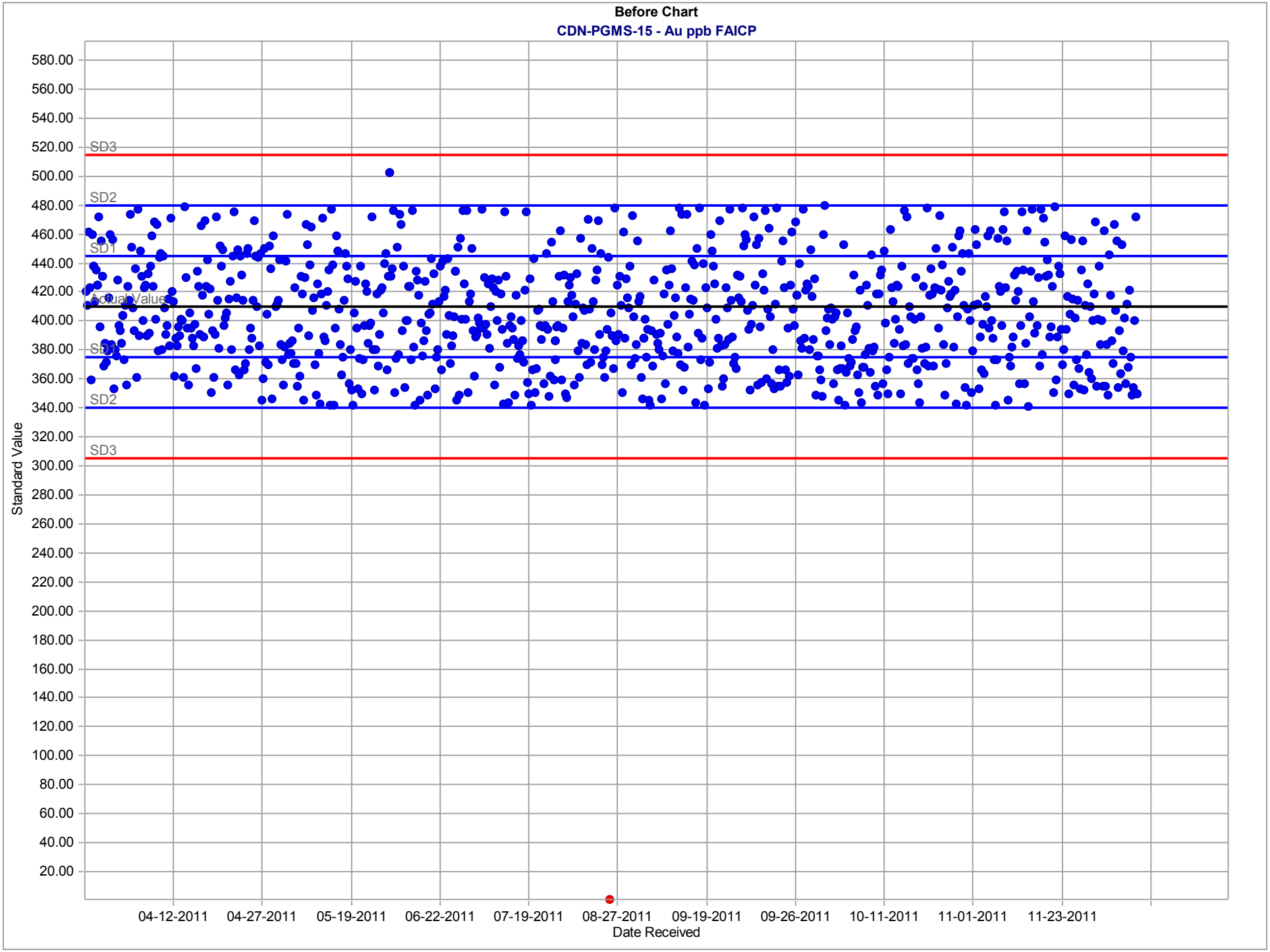
Standard Value
Max Value
05-31-2011
06-21-2011
06-30-2011
07-28-2011
08-23-2011
09-19-2011
09-22-2011
09-29-2011
10-12-2011
11-01-2011
11-21-2011
01-05-2012
Date Received


Before Chart
Blank Crush - Pt ppb FAICP
Standard Value
Max Value
20.00
19.00
18.00
17.00
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
05-31-2011
06-21-2011
06-30-2011
07-28-2011
08-23-2011
09-19-2011
09-22-2011
09-29-2011
10-12-2011
11-01-2011
11-21-2011
01-05-2012
Date Received


Before Chart
CDN-PGMS-15 - Au ppb FAICP
580.00
560.00
540.00
520.00
500.00
480.00
460.00
440.00
420.00
400.00
380.00
360.00
340.00
320.00
300.00
280.00
260.00
240.00
220.00
200.00
180.00
160.00
140.00
120.00
100.00
80.00
60.00
40.00
20.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
04-12-2011
04-27-2011
05-19-2011
06-22-2011
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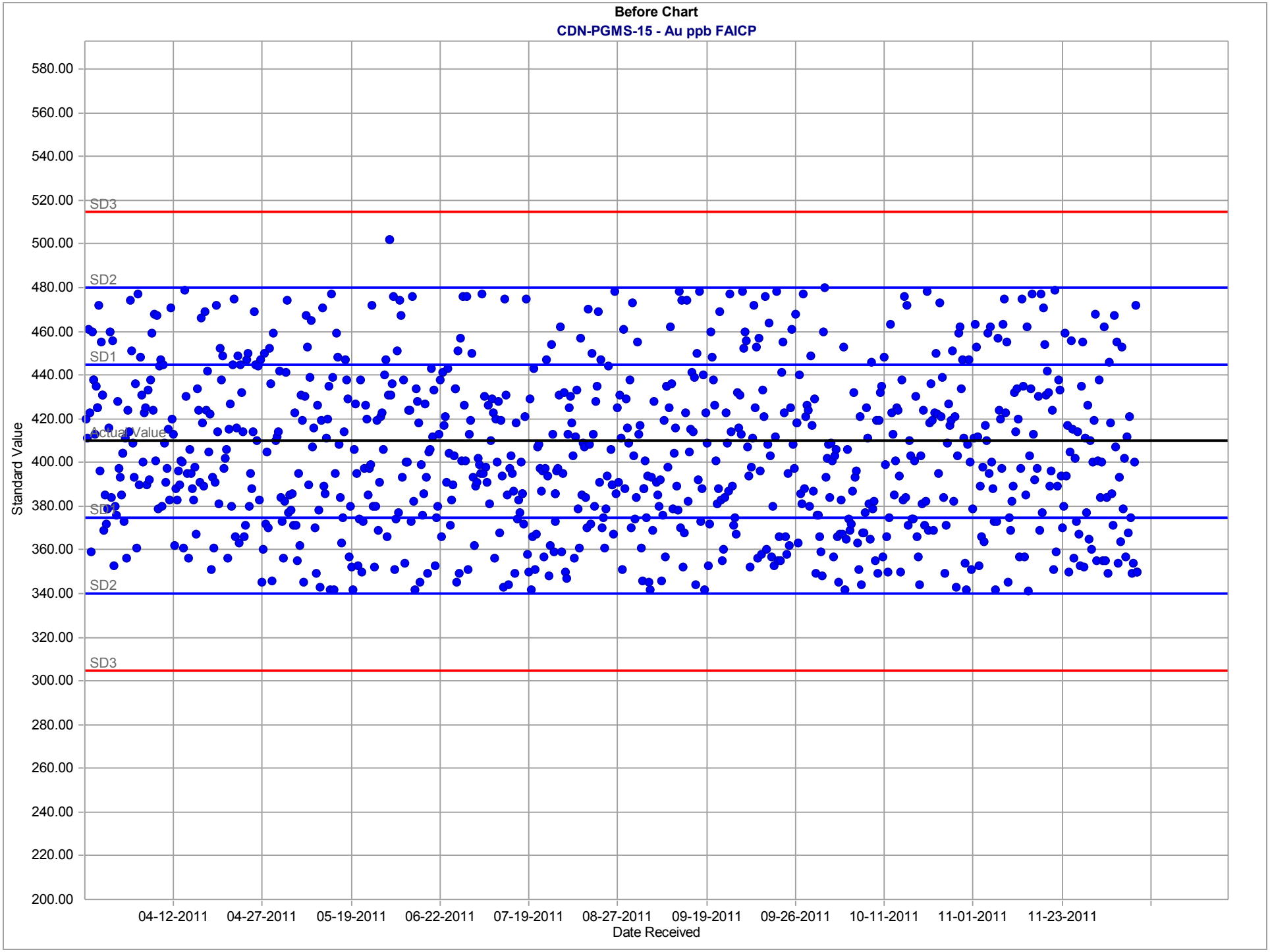
08-27-2011
09-19-2011
09-26-2011
10-11-2011
11-01-2011
11-23-2011
Date Received


Before Chart
CDN-PGMS-15 - Au ppb FAICP
580.00
560.00
540.00
520.00
500.00
480.00
460.00
440.00
420.00
400.00
380.00
360.00
340.00
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280.00
260.00
240.00
220.00
200.00
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SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
04-12-2011
04-27-2011
05-19-2011
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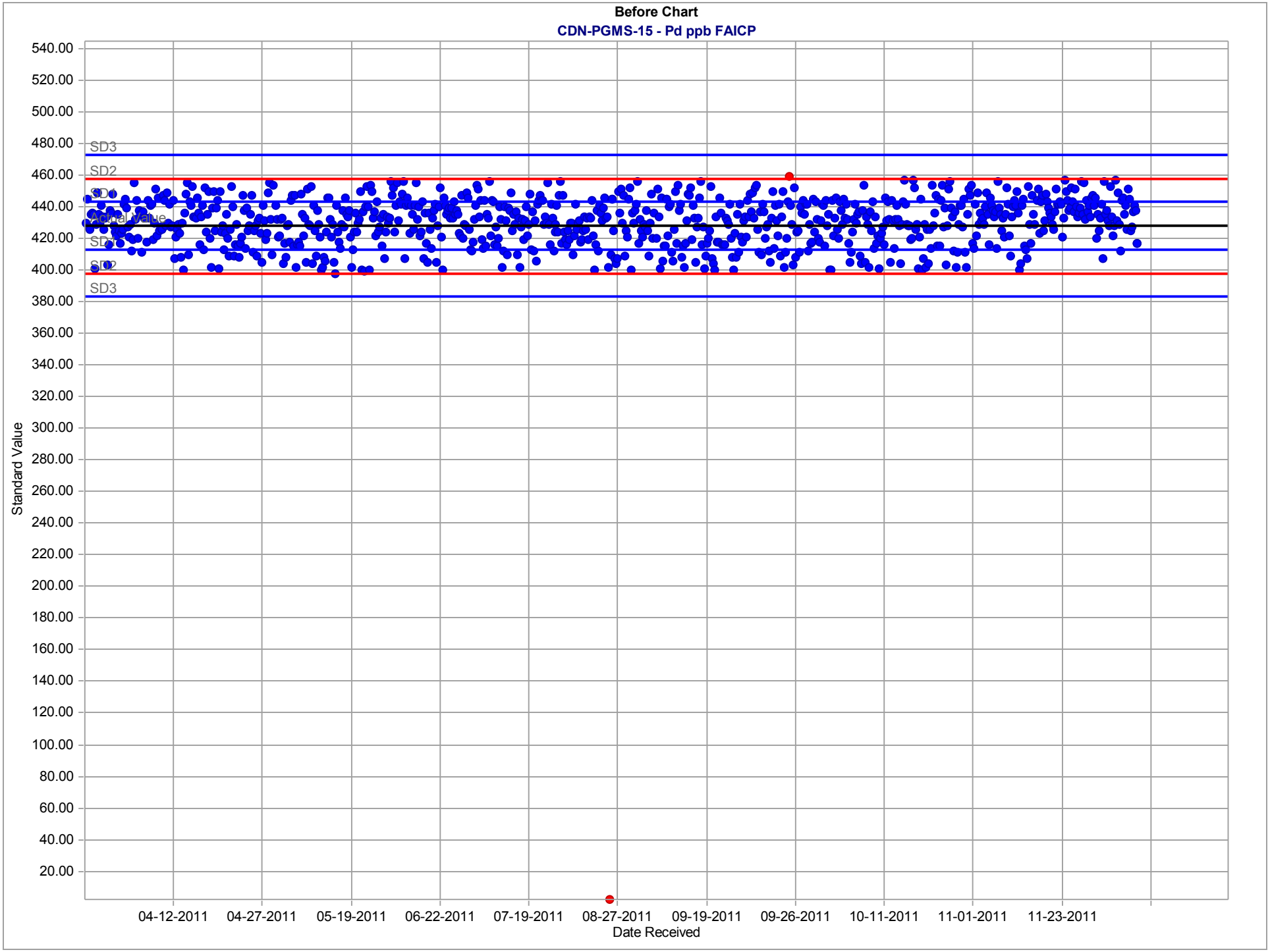
Date Received


Before Chart
CDN-PGMS-15 - Pd ppb FAICP
540.00
520.00
500.00
480.00
460.00
440.00
420.00
400.00
380.00
360.00
340.00
320.00
300.00
280.00
260.00
240.00
220.00
200.00
180.00
160.00
140.00
120.00
100.00
80.00
60.00
40.00
20.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
04-12-2011
04-27-2011
05-19-2011
06-22-2011
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08-27-2011
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09-26-2011
10-11-2011
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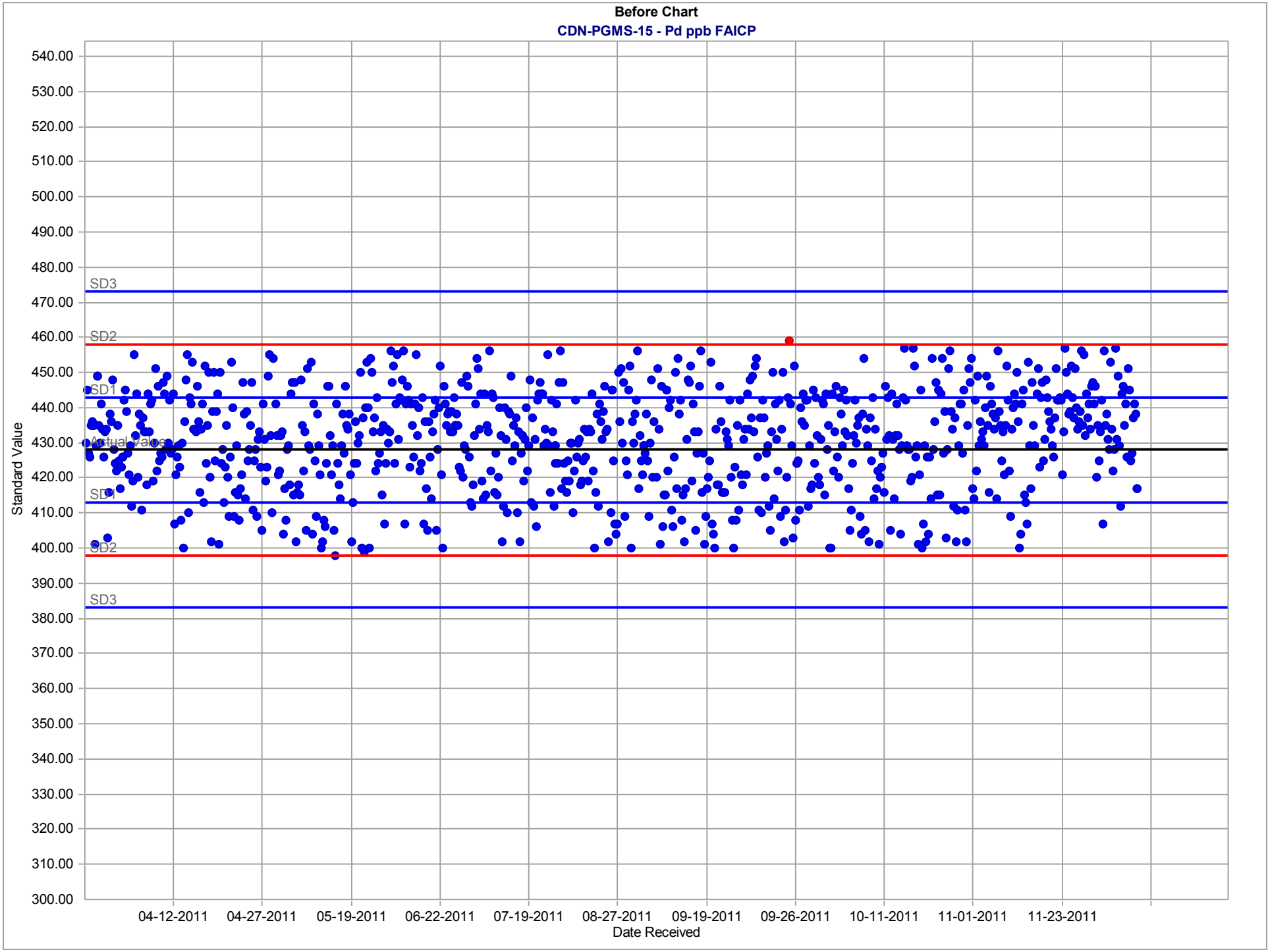
11-23-2011
Date Received


Before Chart
CDN-PGMS-15 - Pd ppb FAICP
Standard Value
540.00
530.00
520.00
510.00
500.00
490.00
480.00
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460.00
450.00
440.00
430.00
420.00
410.00
400.00
390.00
380.00
370.00
360.00
350.00
340.00
330.00
320.00
310.00
300.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
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04-27-2011
05-19-2011
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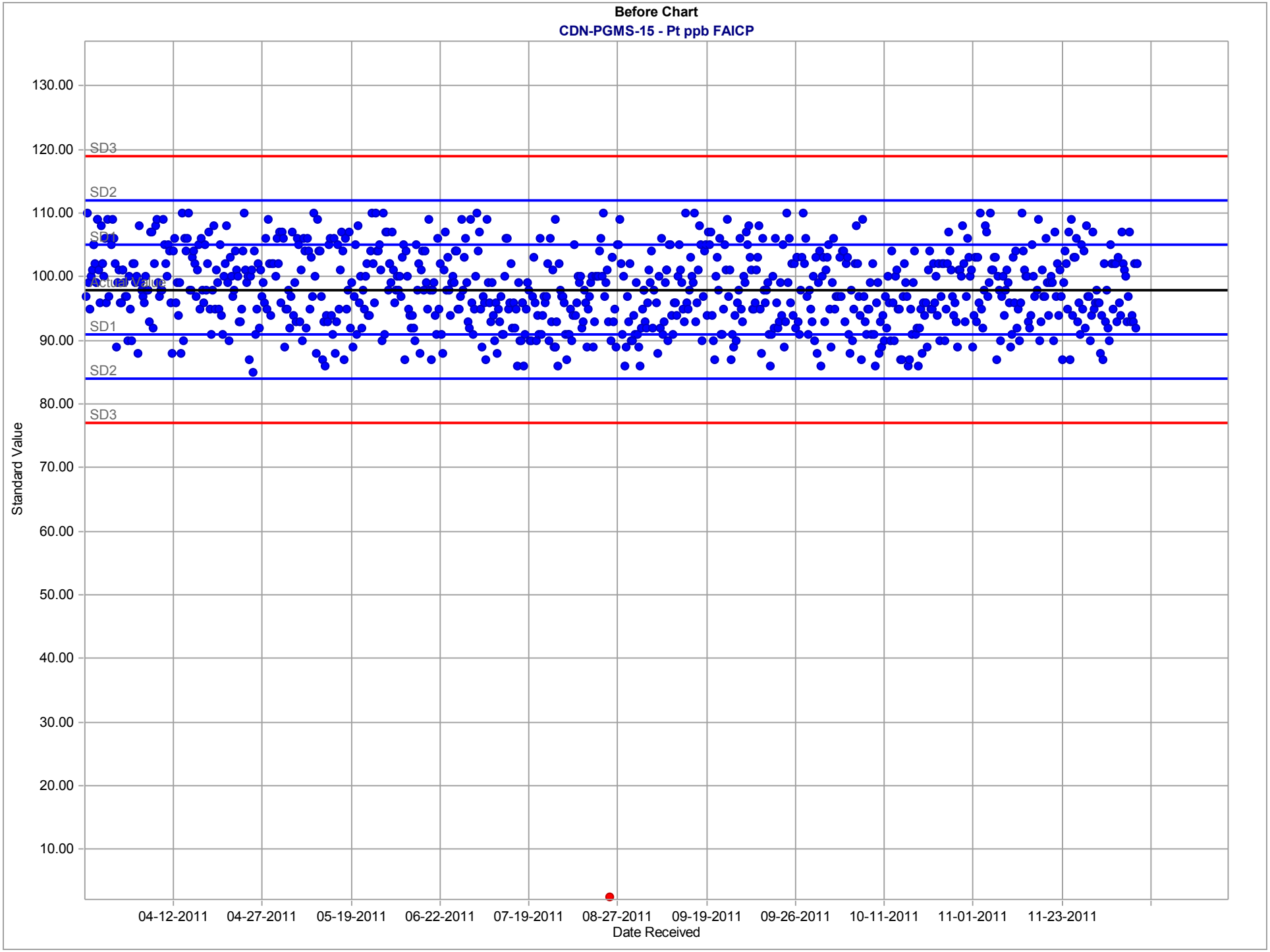
Date Received


Before Chart
CDN-PGMS-15 - Pt ppb FAICP
130.00
120.00
110.00
100.00
90.00
80.00
70.00
60.00
50.00
40.00
30.00
20.00
10.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
04-12-2011
04-27-2011
05-19-2011
06-22-2011
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08-27-2011
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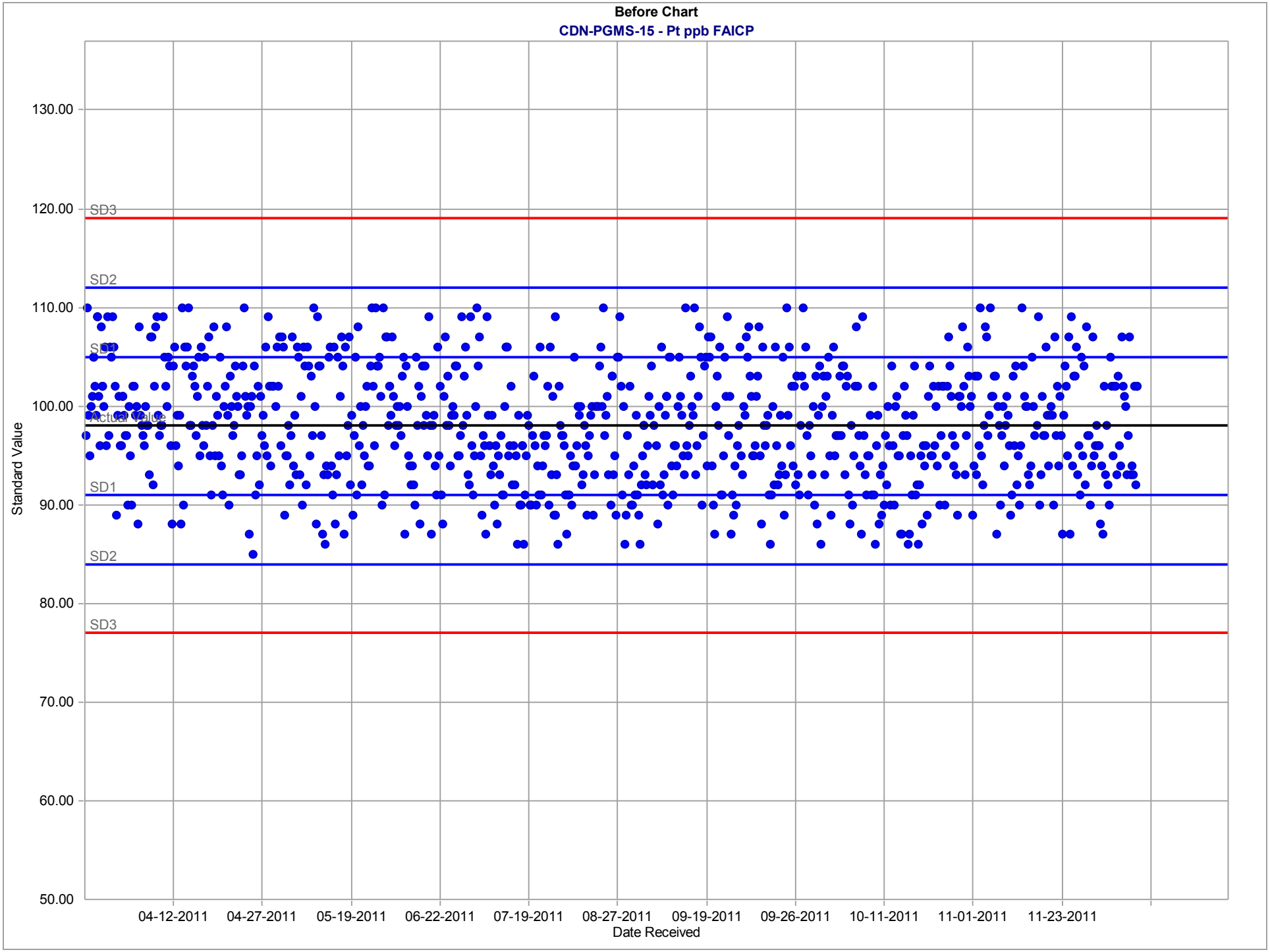
09-26-2011
10-11-2011
11-01-2011
11-23-2011
Date Received


Before Chart
CDN-PGMS-15 - Pt ppb FAICP
130.00
120.00
110.00
100.00
90.00
80.00
70.00
60.00
50.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
04-12-2011
04-27-2011
05-19-2011
06-22-2011
07-19-2011
08-27-2011
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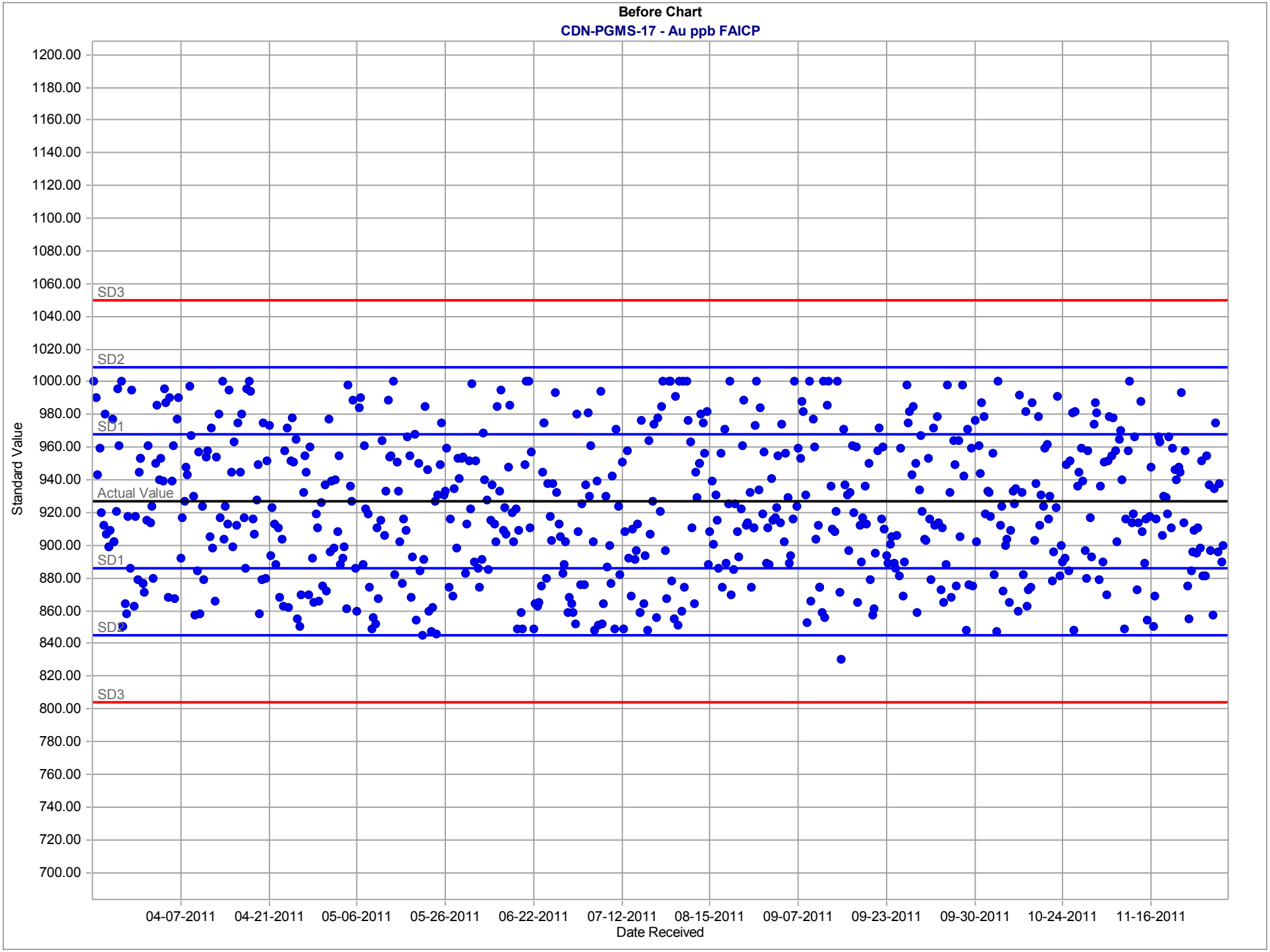
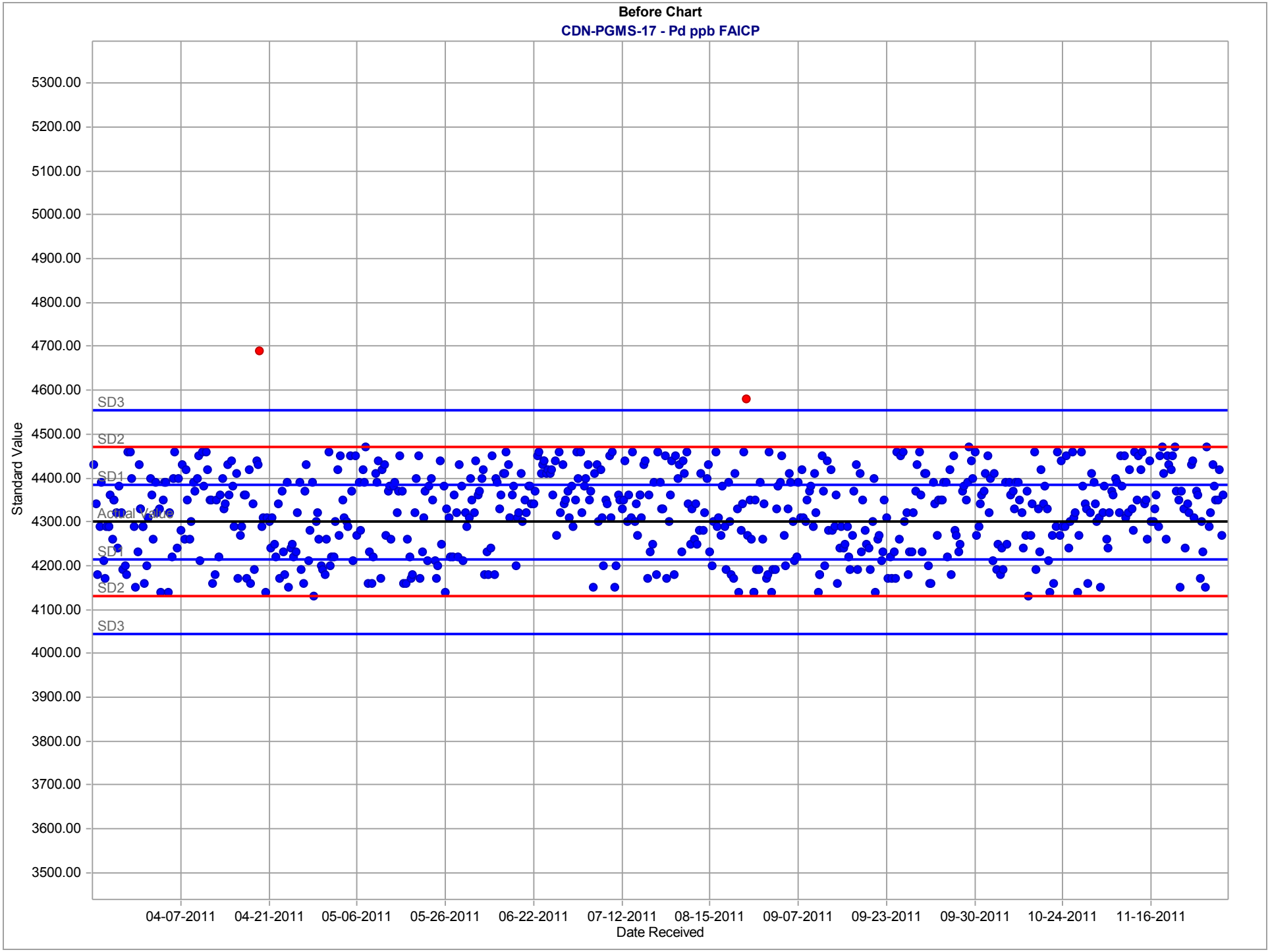
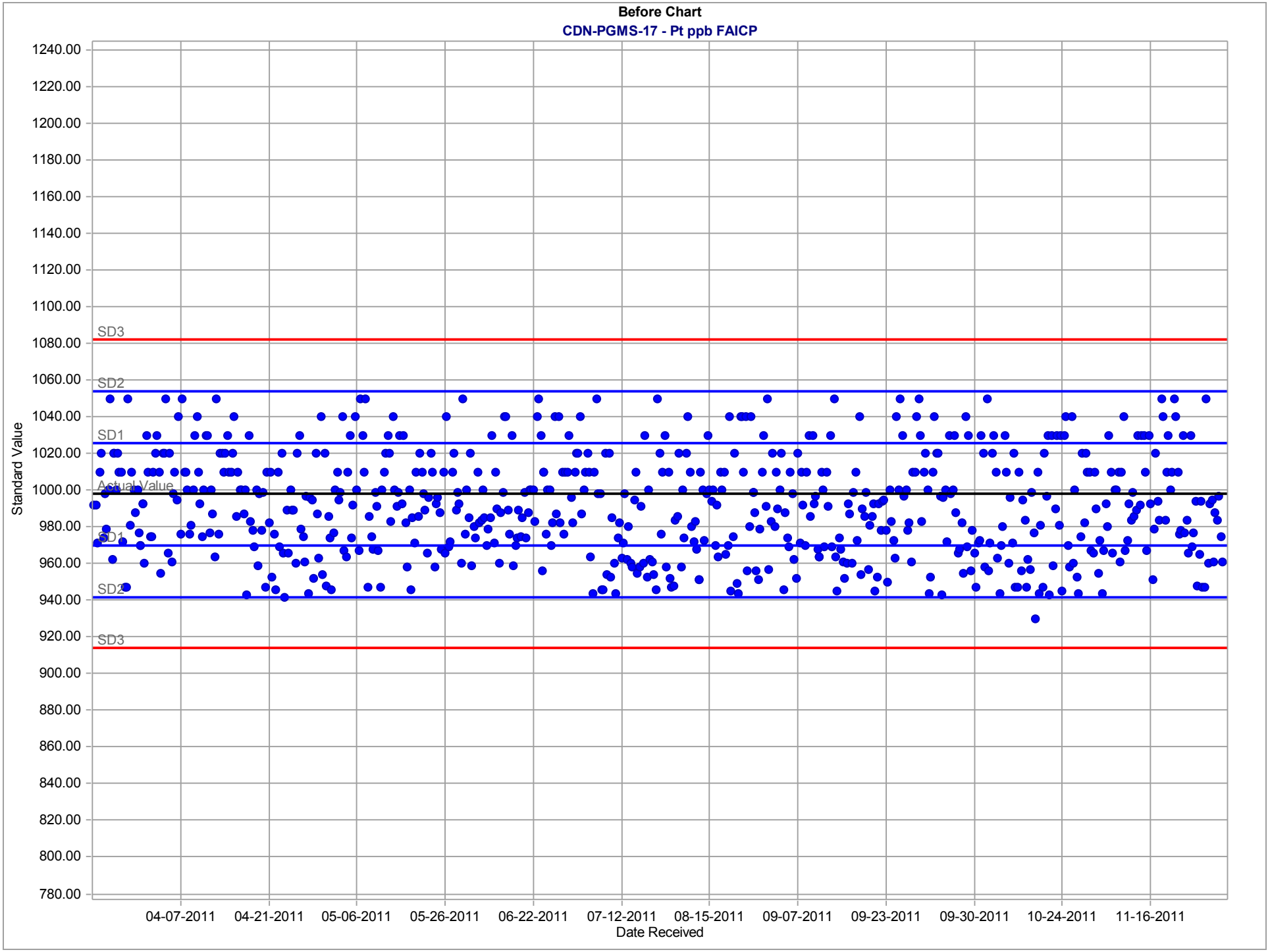
09-26-2011
10-11-2011
11-01-2011
11-23-2011
Date Received


Before Chart
CDN-PGMS-17 - Au ppb FAICP
Standard Value
1200.00
1180.00
1160.00
1140.00
1120.00
1100.00
1080.00
1060.00
1040.00
1020.00
1000.00
980.00
960.00
940.00
920.00
900.00
880.00
860.00
840.00
820.00
800.00
780.00
760.00
740.00
720.00
700.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
04-07-2011
04-21-2011
05-06-2011
05-26-2011
06-22-2011
07-12-2011
08-15-2011
09-07-2011
09-23-2011
09-30-2011
10-24-2011
11-16-2011
Date Received


Before Chart
CDN-PGMS-17 - Pd ppb FAICP
Standard Value
5300.00
5200.00
5100.00
5000.00
4900.00
4800.00
4700.00
4600.00
4500.00
4400.00
4300.00
4200.00
4100.00
4000.00
3900.00
3800.00
3700.00
3600.00
3500.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
04-07-2011
04-21-2011
05-06-2011
05-26-2011
06-22-2011
07-12-2011
08-15-2011
09-07-2011
09-23-2011
09-30-2011
10-24-2011
11-16-2011
Date Received


Before Chart
CDN-PGMS-17 - Pt ppb FAICP
Standard Value
1240.00
1220.00
1200.00
1180.00
1160.00
1140.00
1120.00
1100.00
1080.00
1060.00
1040.00
1020.00
1000.00
980.00
960.00
940.00
920.00
900.00
880.00
860.00
840.00
820.00
800.00
780.00
SD3
SD2
SD1
ActualValue
SD1
SD2
SD3
04-07-2011
04-21-2011
05-06-2011
05-26-2011
06-22-2011
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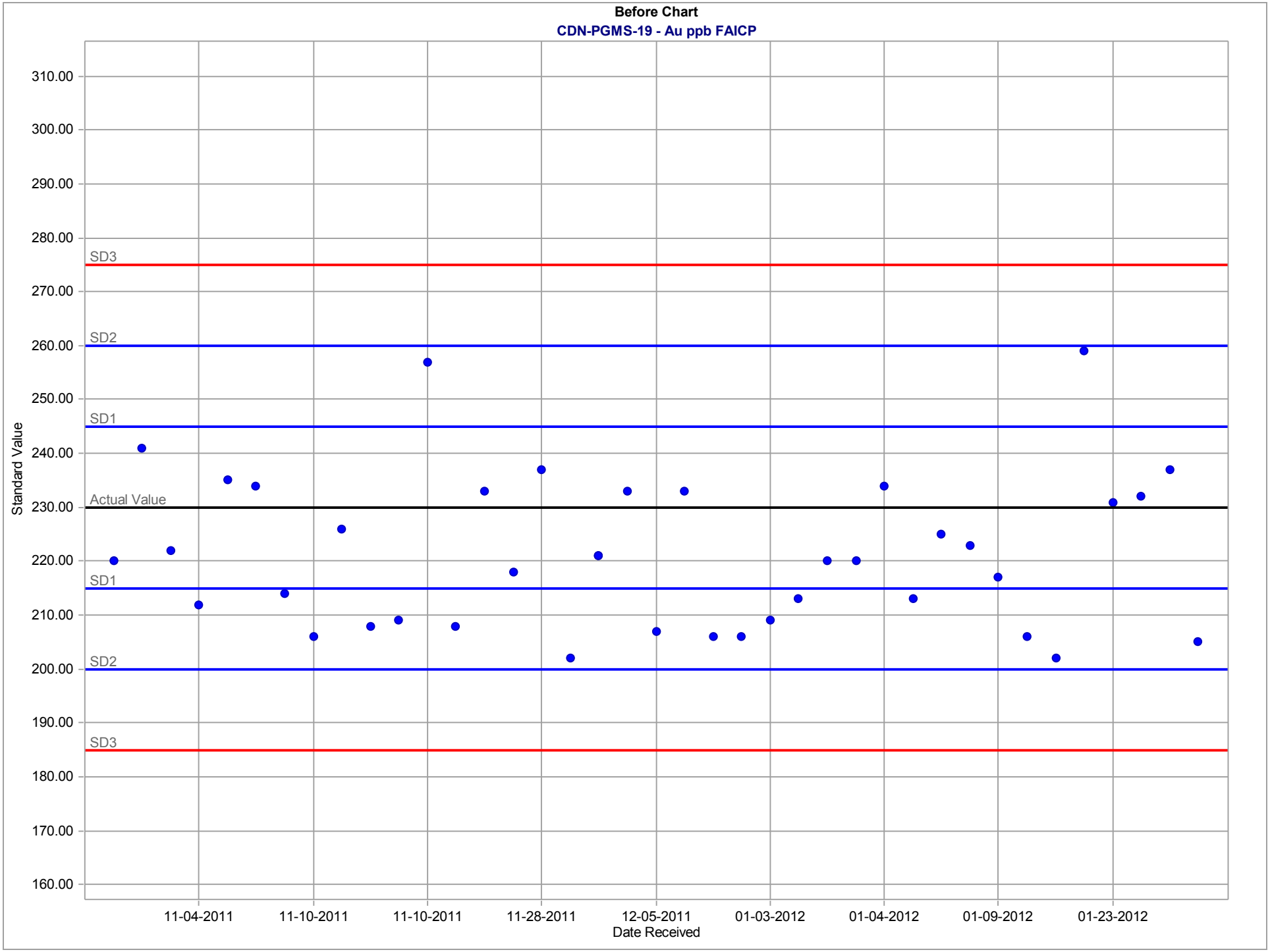
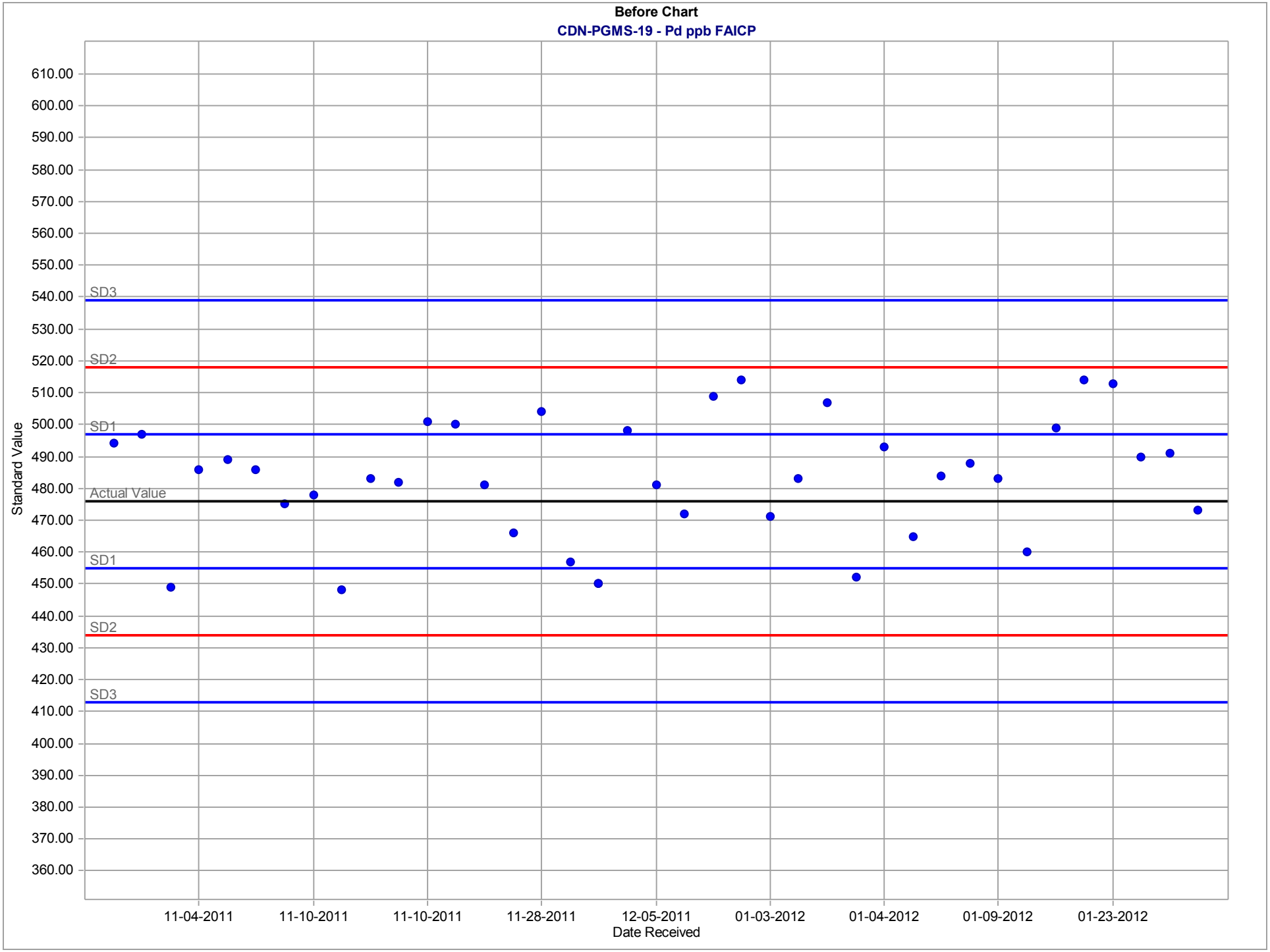
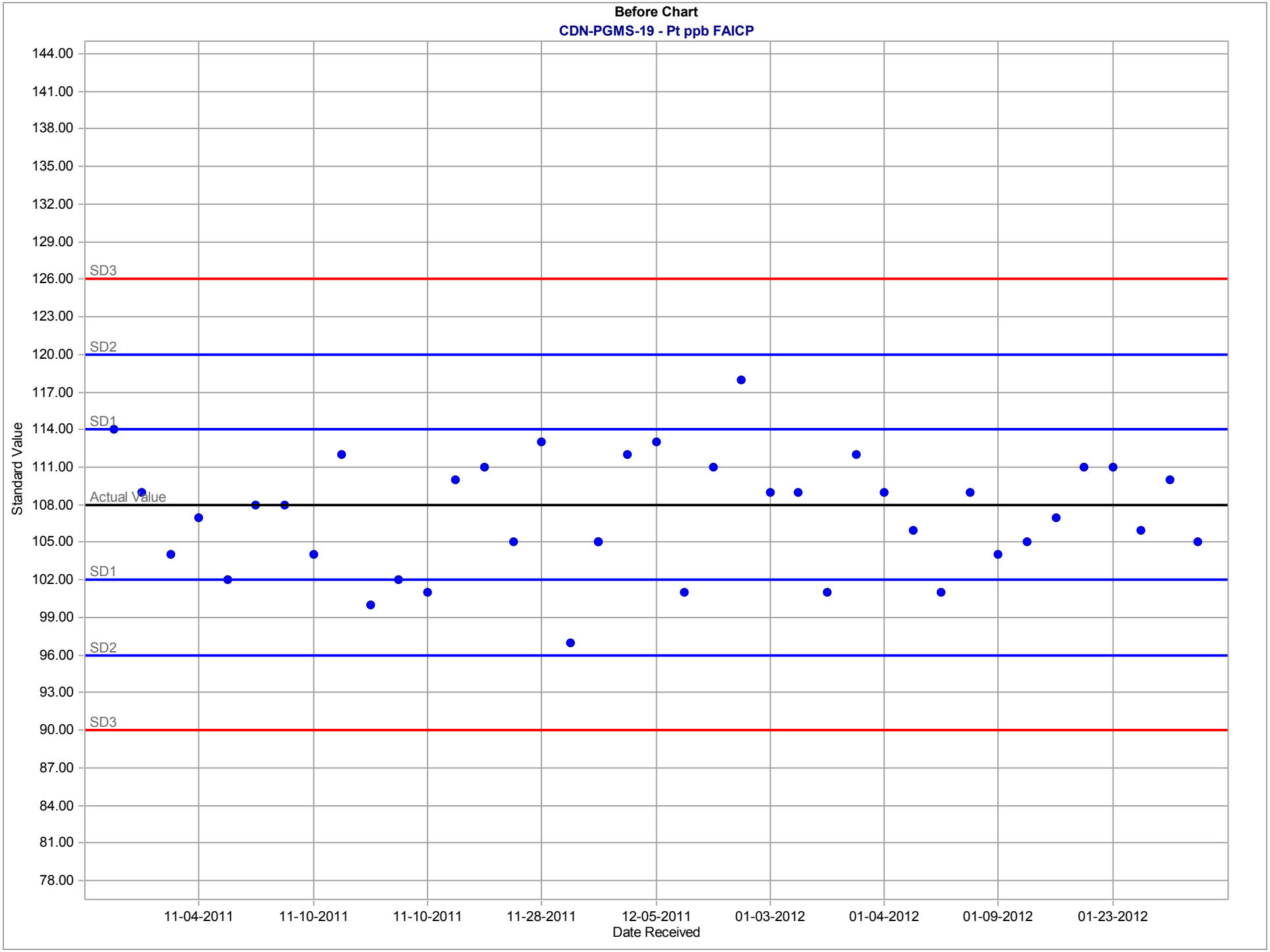
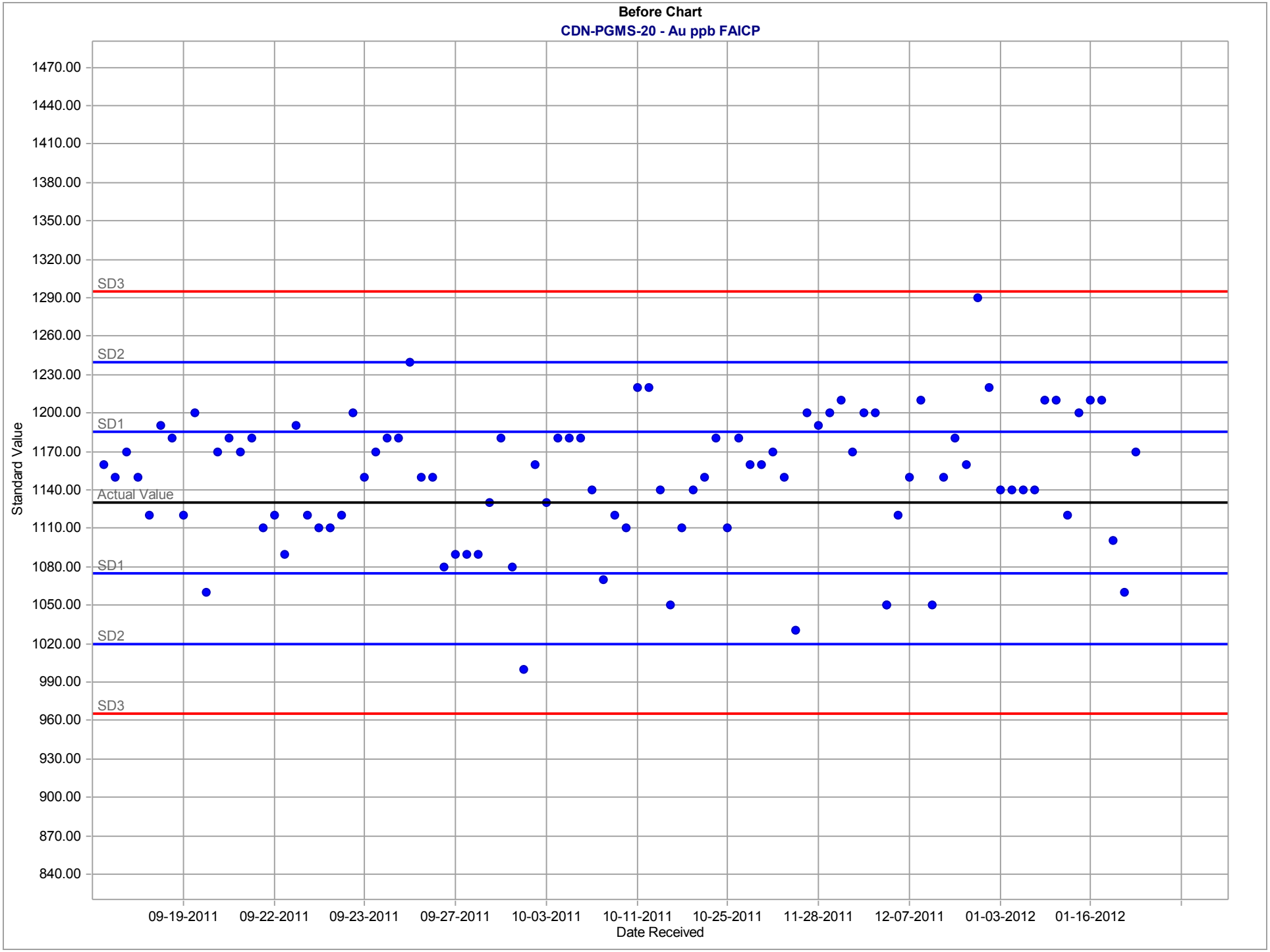
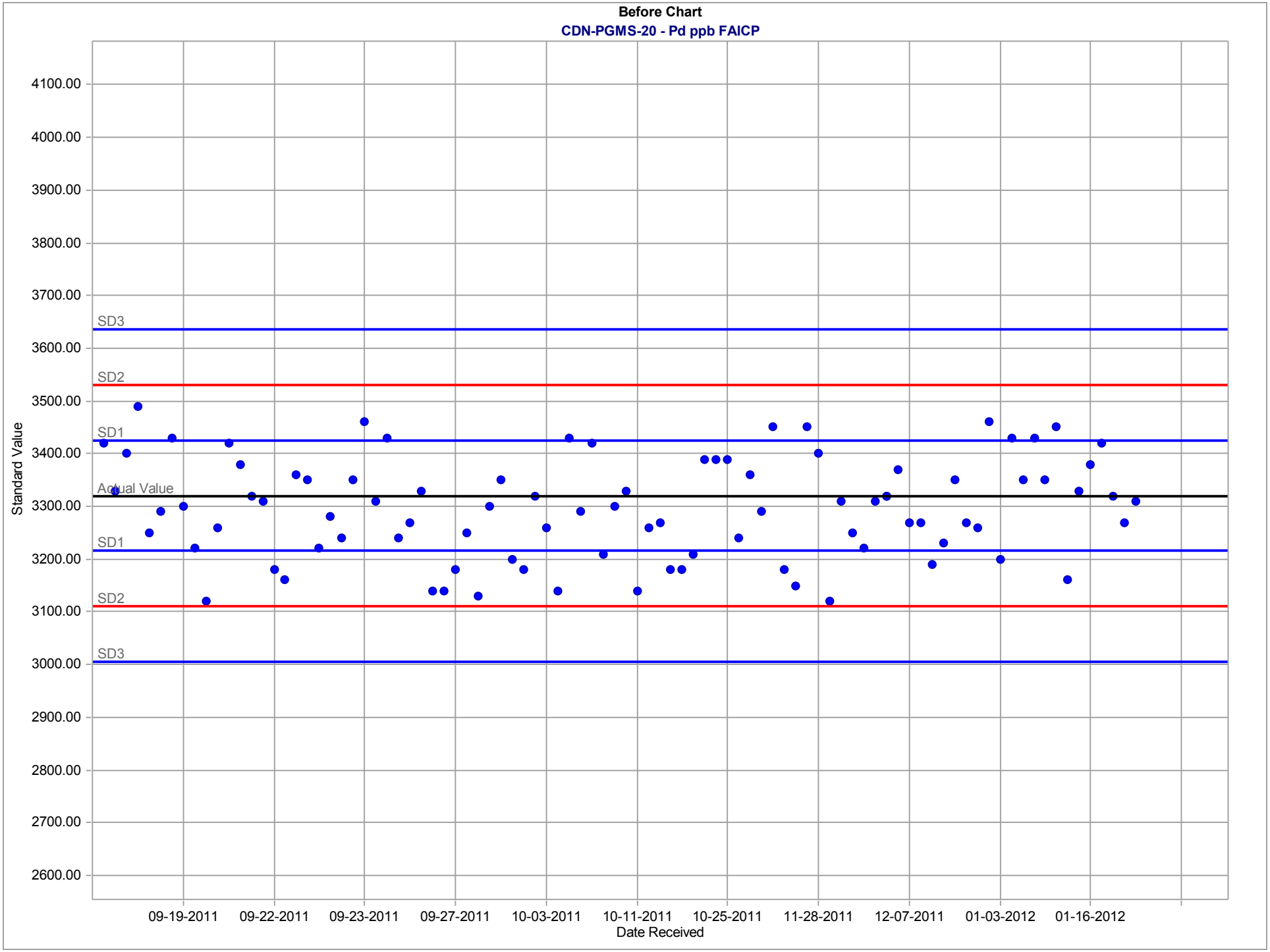
08-15-2011
09-07-2011
09-23-2011
09-30-2011
10-24-2011
11-16-2011
Date Received

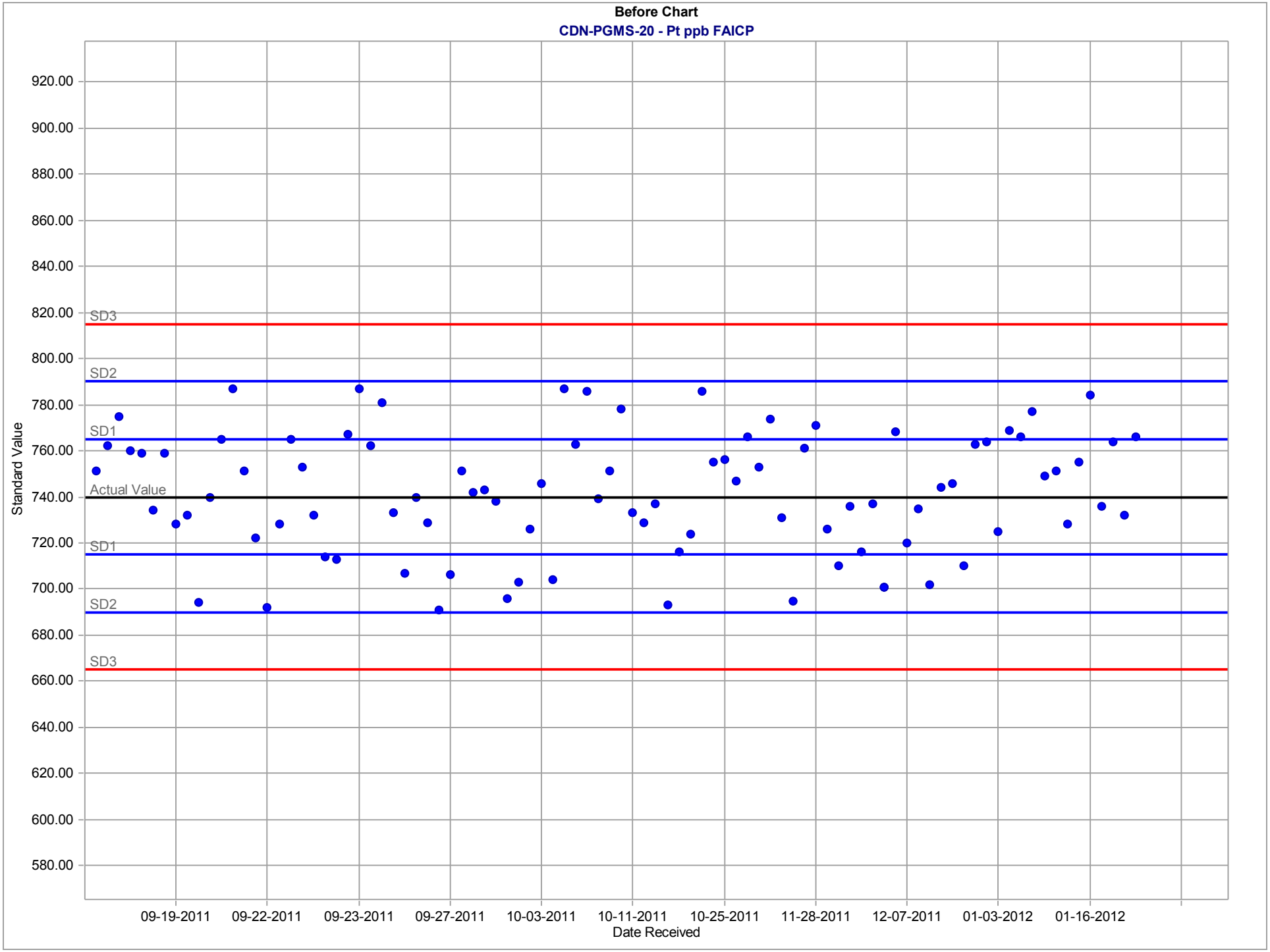

Before Chart
CDN-PGMS-19 - Au ppb FAICP
Standard Value
310.00
300.00
290.00
280.00
270.00
260.00
250.00
240.00
230.00
220.00
210.00
200.00
190.00
180.00
170.00
160.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
11-04-2011
11-10-2011
11-10-2011
11-28-2011
12-05-2011
01-03-2012
01-04-2012
01-09-2012
01-23-2012
Date Received


Before Chart
CDN-PGMS-19 - Pd ppb FAICP
610.00
600.00
590.00
580.00
570.00
560.00
550.00
540.00
530.00
520.00
510.00
500.00
490.00
480.00
470.00
460.00
450.00
440.00
430.00
420.00
410.00
400.00
390.00
380.00
370.00
360.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
11-04-2011
11-10-2011
11-10-2011
11-28-2011
12-05-2011
01-03-2012
01-04-2012
01-09-2012
01-23-2012
Date Received


Before Chart
CDN-PGMS-19 - Pt ppb FAICP
144.00
141.00
138.00
135.00
132.00
129.00
126.00
123.00
120.00
117.00
114.00
111.00
108.00
105.00
102.00
99.00
96.00
93.00
90.00
87.00
84.00
81.00
78.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
11-04-2011
11-10-2011
11-10-2011
11-28-2011
12-05-2011
01-03-2012
01-04-2012
01-09-2012
01-23-2012
Date Received


Before Chart
CDN-PGMS-20 - Au ppb FAICP
Standard Value
1470.00
1440.00
1410.00
1380.00
1350.00
1320.00
SD3
1290.00
1260.00
SD2
1230.00
1200.00
SD1
1170.00
1140.00
Actual Value
1110.00
1080.00
SD1
1050.00
1020.00
SD2
990.00
960.00
SD3
930.00
900.00
870.00
840.00
09-19-2011
09-22-2011
09-23-2011
09-27-2011
10-03-2011
10-11-2011
10-25-2011
11-28-2011
12-07-2011
01-03-2012
01-16-2012
Date Received


Before Chart
CDN-PGMS-20 - Pd ppb FAICP
Standard Value
4100.00
4000.00
3900.00
3800.00
3700.00
3600.00
3500.00
3400.00
3300.00
3200.00
3100.00
3000.00
2900.00
2800.00
2700.00
2600.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
09-19-2011
09-22-2011
09-23-2011
09-27-2011
10-03-2011
10-11-2011
10-25-2011
11-28-2011
12-07-2011
01-03-2012
01-16-2012
Date Received


Before Chart
CDN-PGMS-20 - Pt ppb FAICP
920.00
900.00
880.00
860.00
840.00
820.00
800.00
780.00
760.00
740.00
720.00
700.00
680.00
660.00
640.00
620.00
600.00
580.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
09-19-2011
09-22-2011
09-23-2011
09-27-2011
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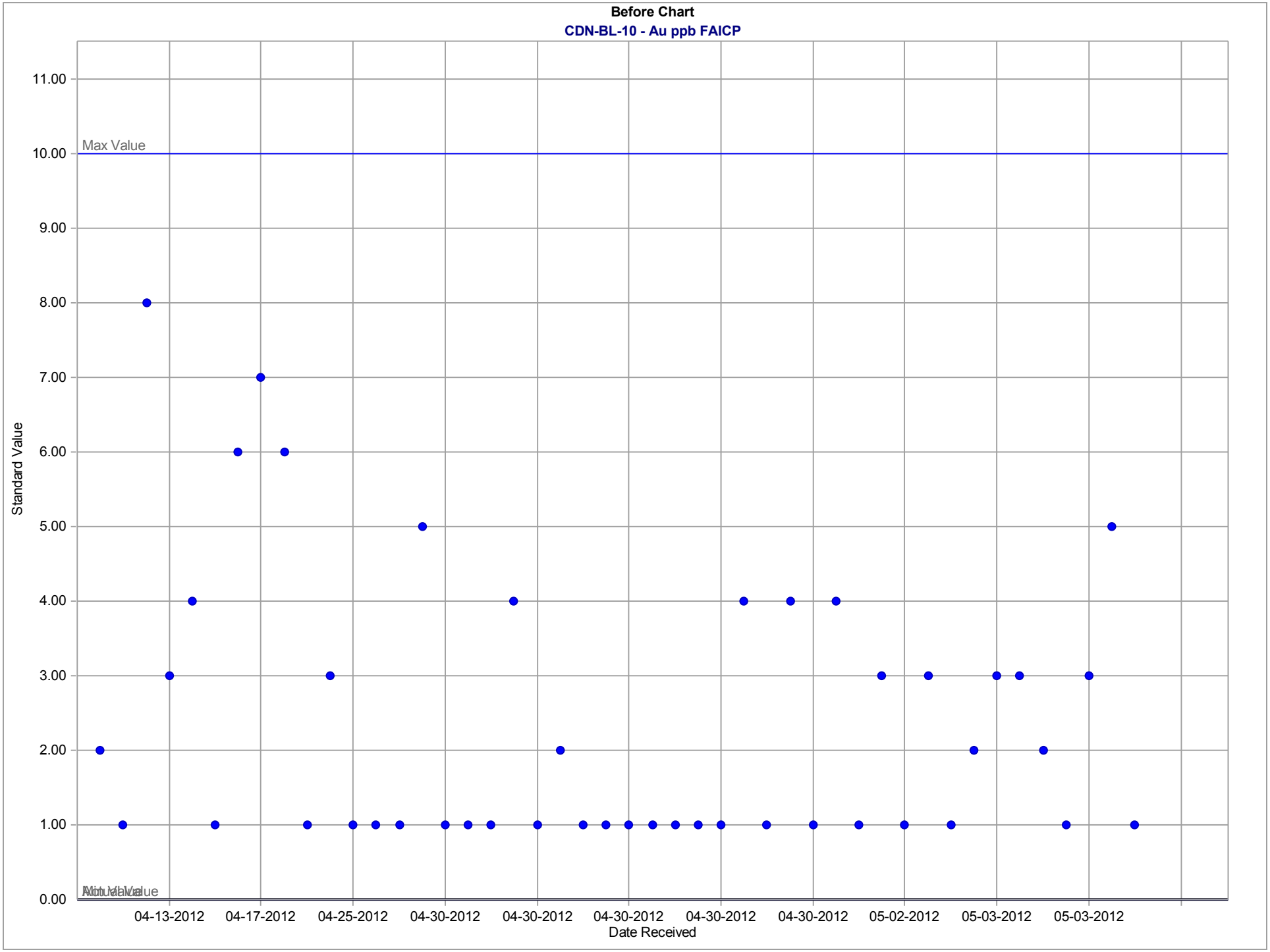
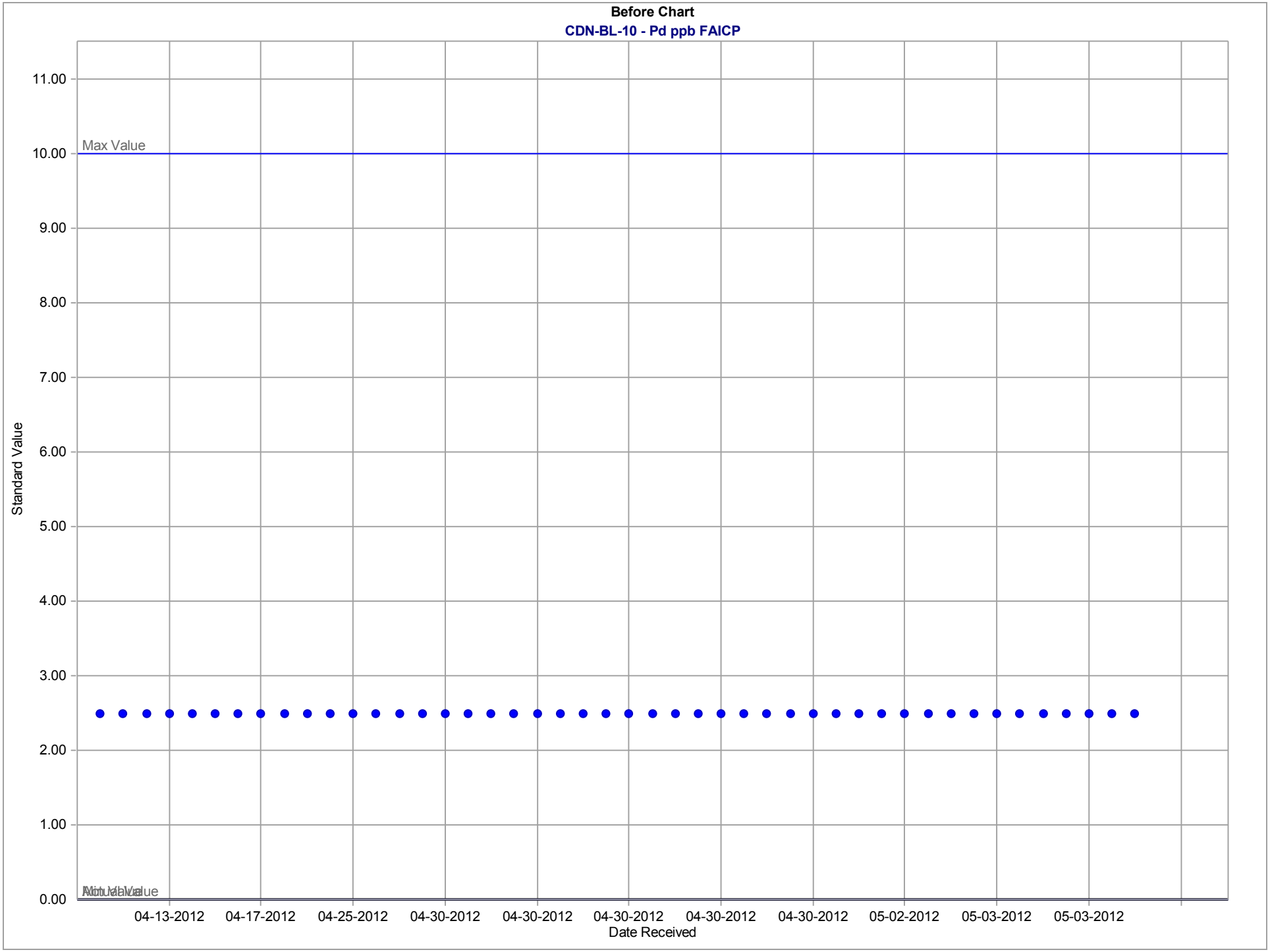
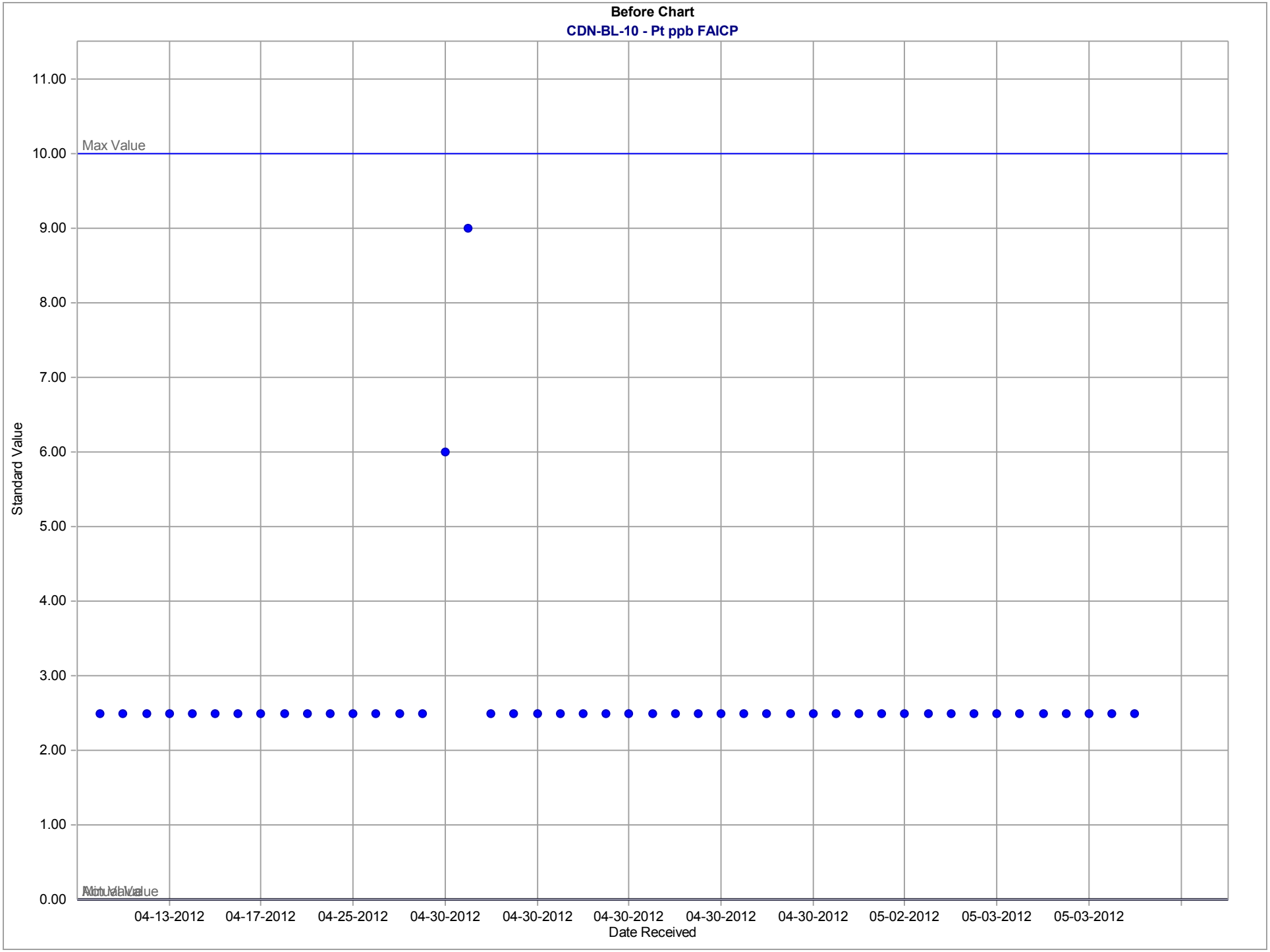
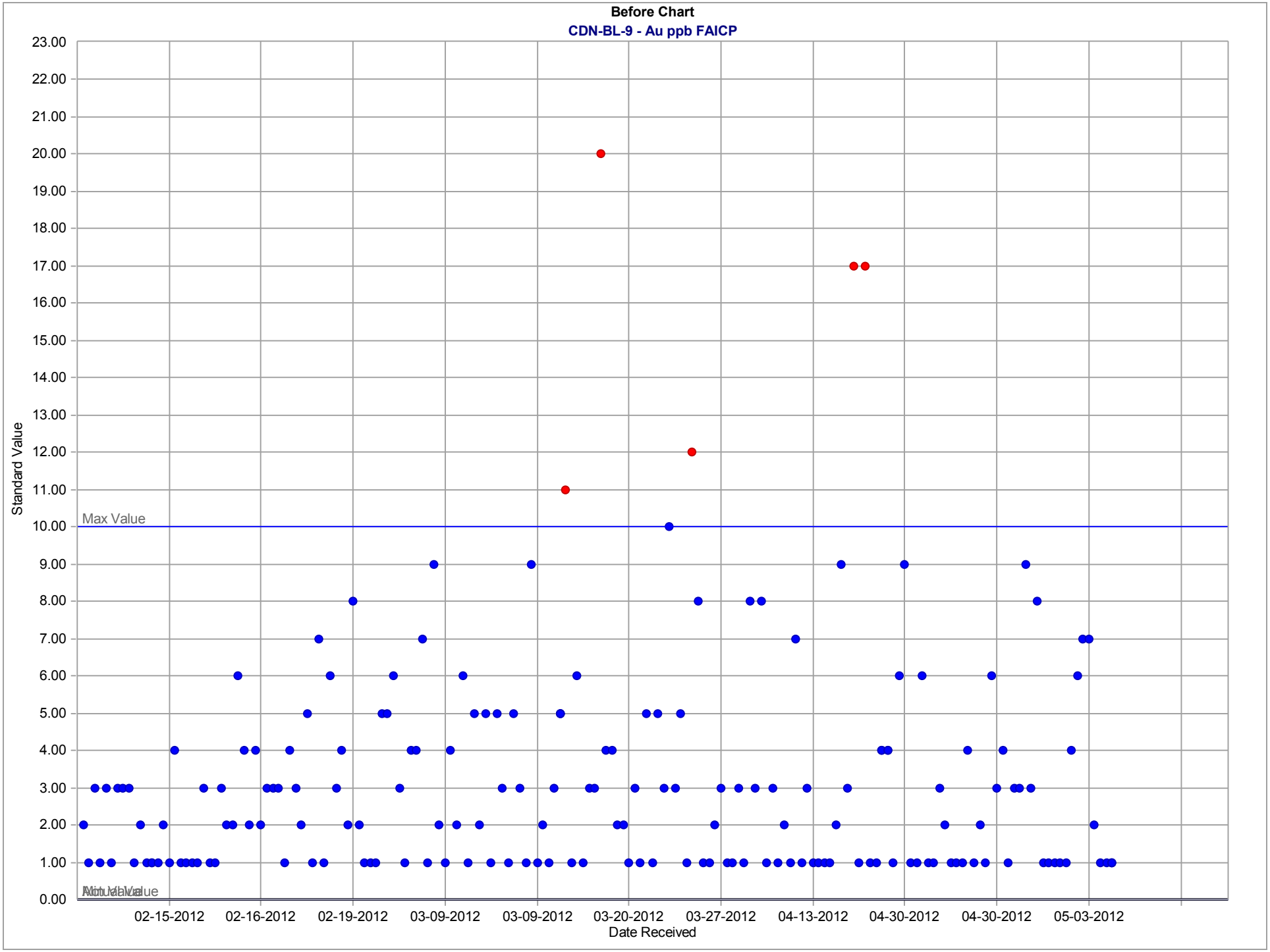
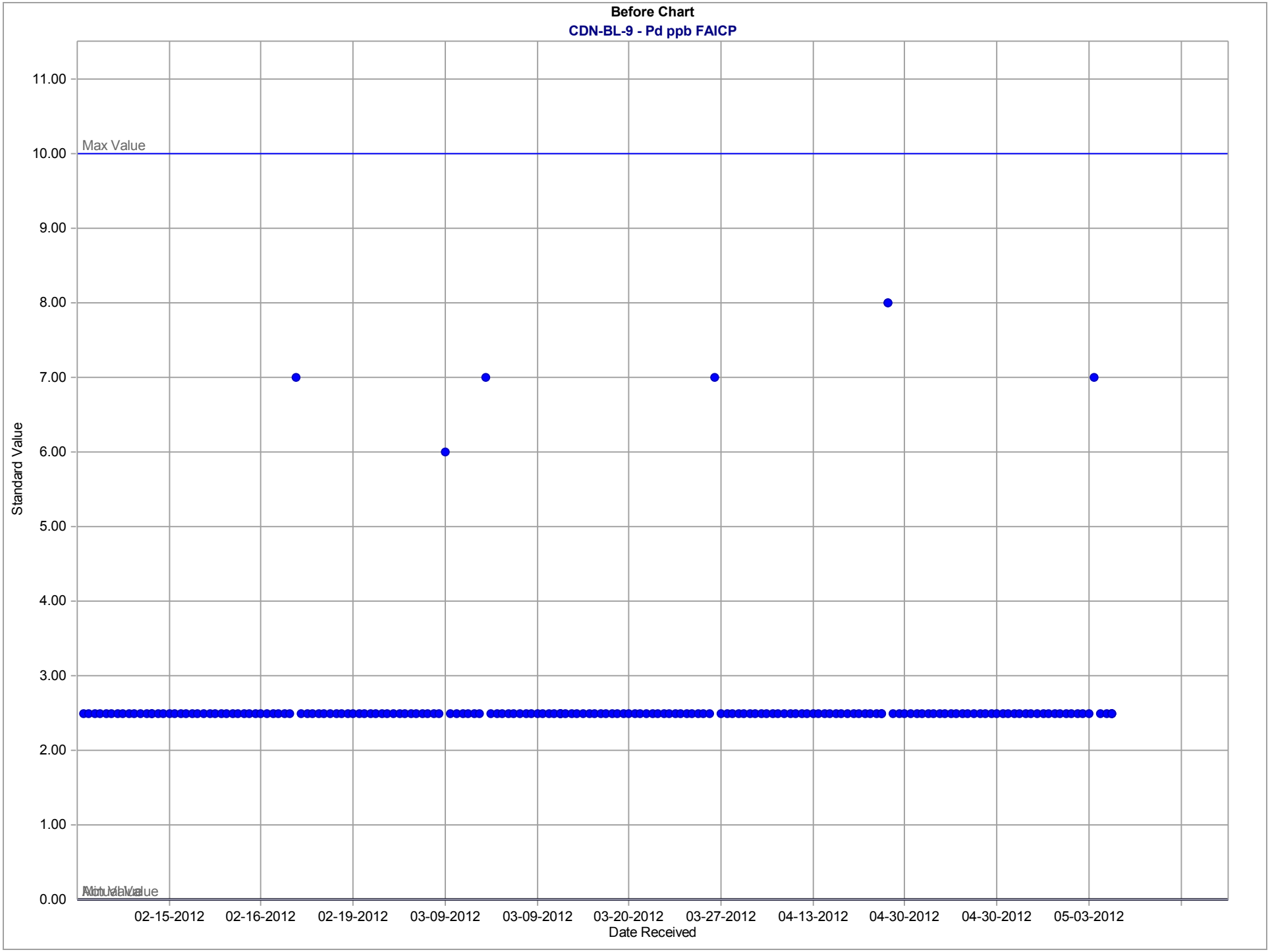
10-11-2011
10-25-2011
11-28-2011
12-07-2011
01-03-2012
01-16-2012
Date Received

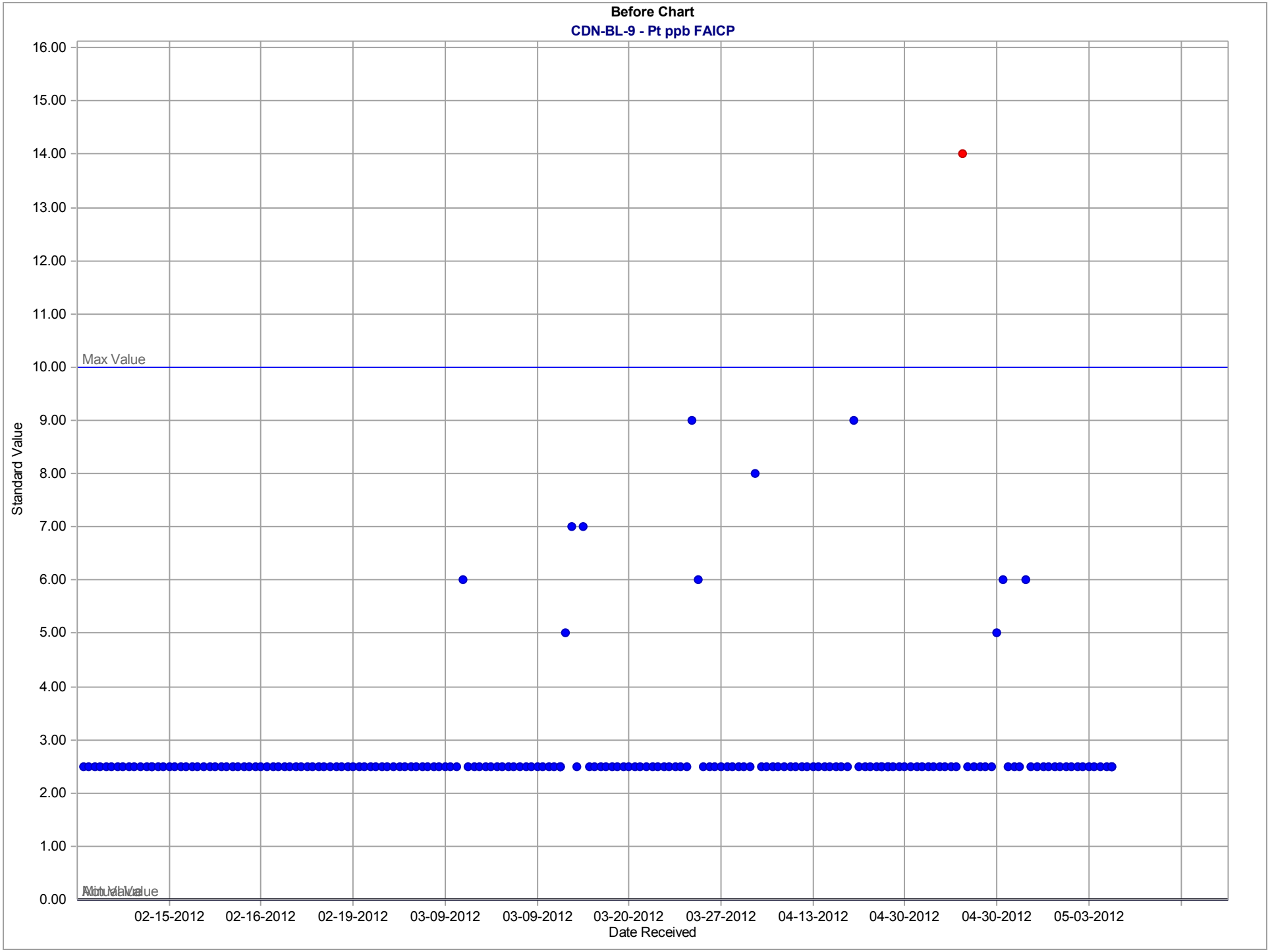

Before Chart
CDN-BL-10 - Au ppb FAICP
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
04-13-2012
04-17-2012
04-25-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
05-02-2012
05-03-2012
05-03-2012
Date Received

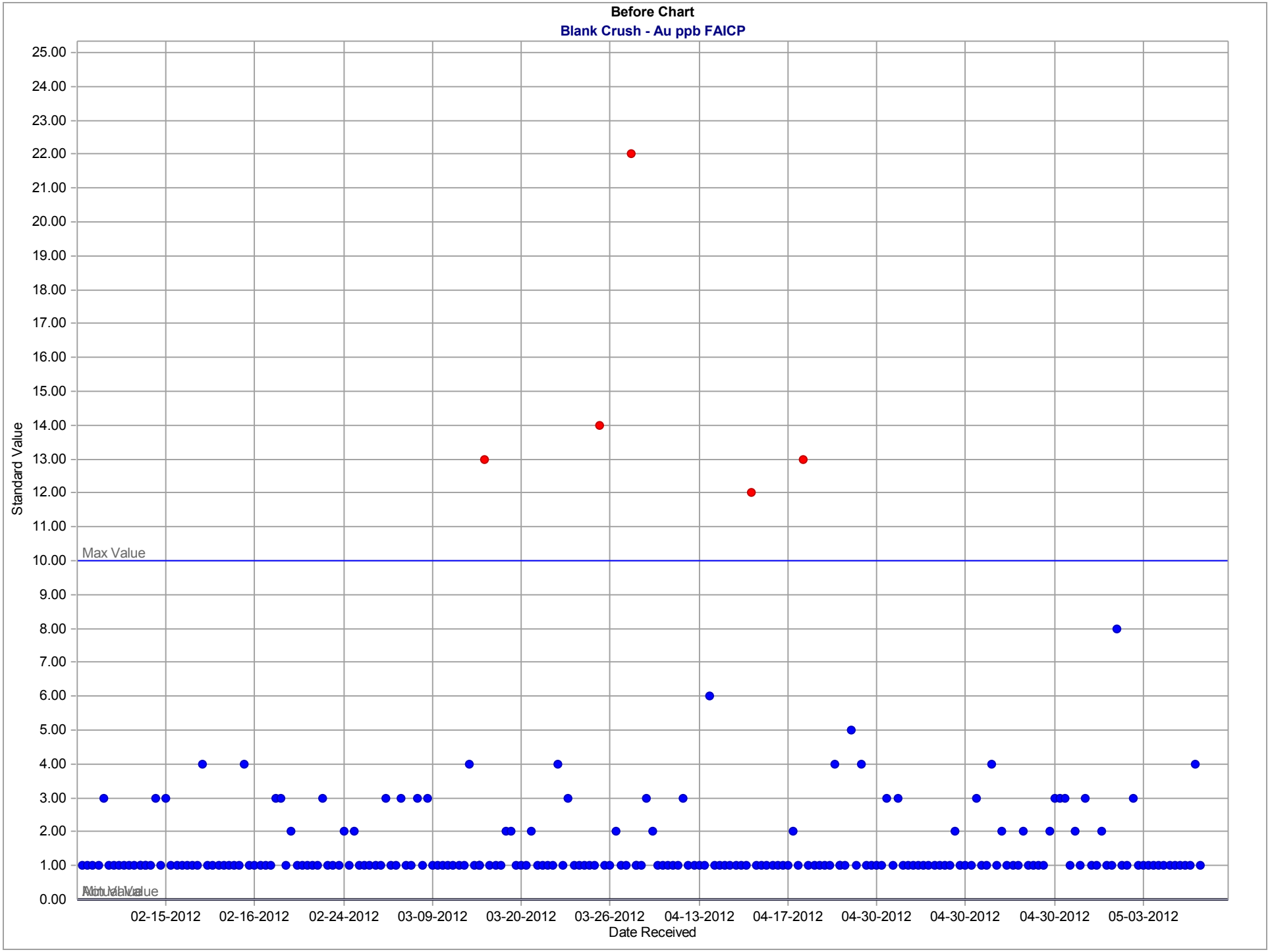

Before Chart
CDN-BL-10 - Pd ppb FAICP
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
04-13-2012
04-17-2012
04-25-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
05-02-2012
05-03-2012
05-03-2012
Date Received


Before Chart
CDN-BL-10 - Pt ppb FAICP
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
04-13-2012
04-17-2012
04-25-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
05-02-2012
05-03-2012
05-03-2012
Date Received


Before Chart
CDN-BL-9 - Au ppb FAICP
Standard Value
Max Value
23.00
22.00
21.00
20.00
19.00
18.00
17.00
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
02-15-2012
02-16-2012
02-19-2012
03-09-2012
03-09-2012
03-20-2012
03-27-2012
04-13-2012
04-30-2012
04-30-2012
05-03-2012
Date Received


Before Chart
CDN-BL-9 - Pd ppb FAICP
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
02-15-2012
02-16-2012
02-19-2012
03-09-2012
03-09-2012
03-20-2012
03-27-2012
04-13-2012
04-30-2012
04-30-2012
05-03-2012
Date Received


Before Chart
CDN-BL-9 - Pt ppb FAICP
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
Standard Value
02-15-2012
02-16-2012
02-19-2012
03-09-2012
03-09-2012
03-20-2012
03-27-2012
04-13-2012
04-30-2012
04-30-2012
05-03-2012
Date Received


Before Chart
Blank Crush - Au ppb FAICP
Standard Value
25.00
24.00
23.00
22.00
21.00
20.00
19.00
18.00
17.00
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
02-15-2012
02-16-2012
02-24-2012
03-09-2012
03-20-2012
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04-13-2012
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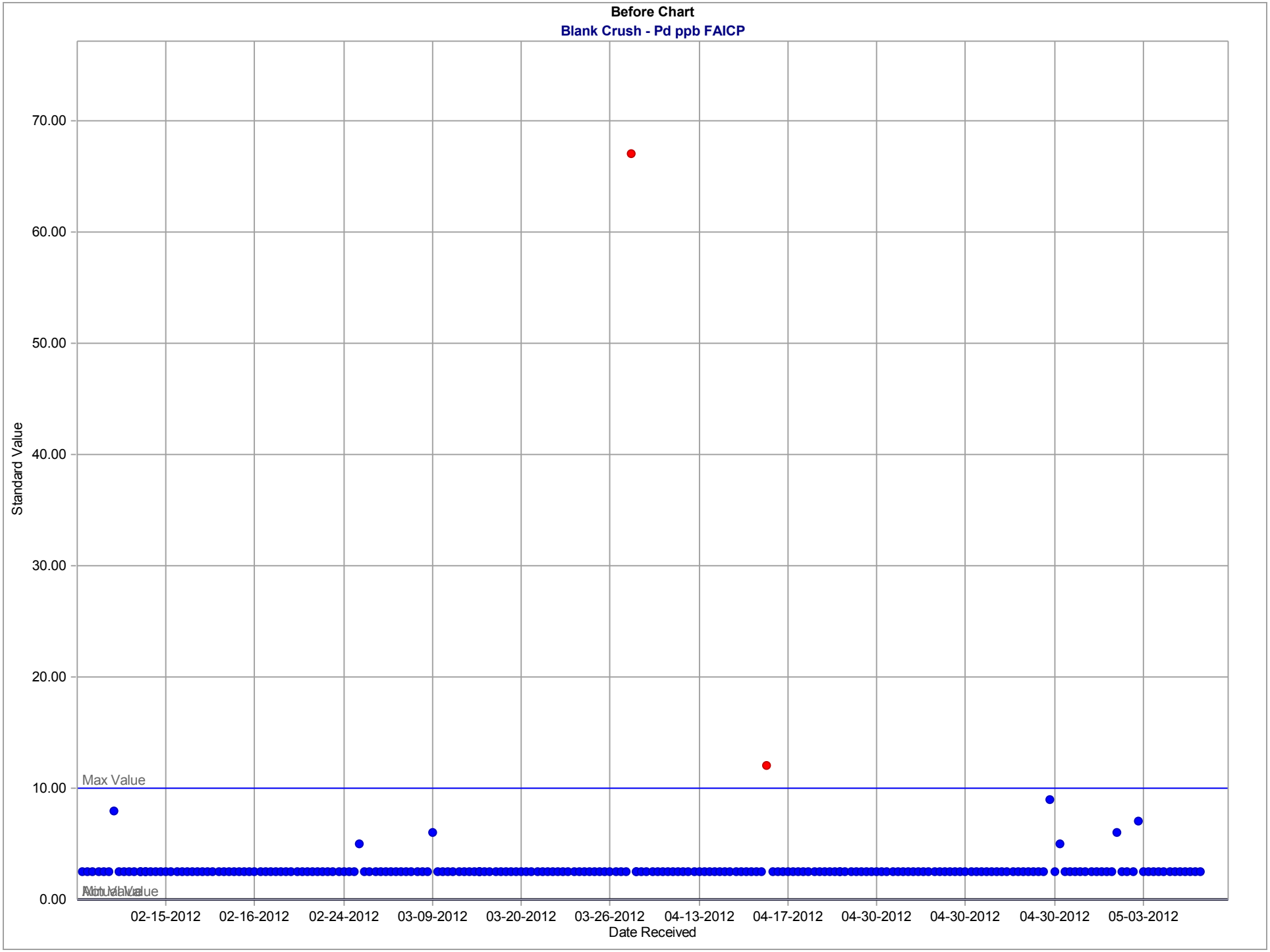
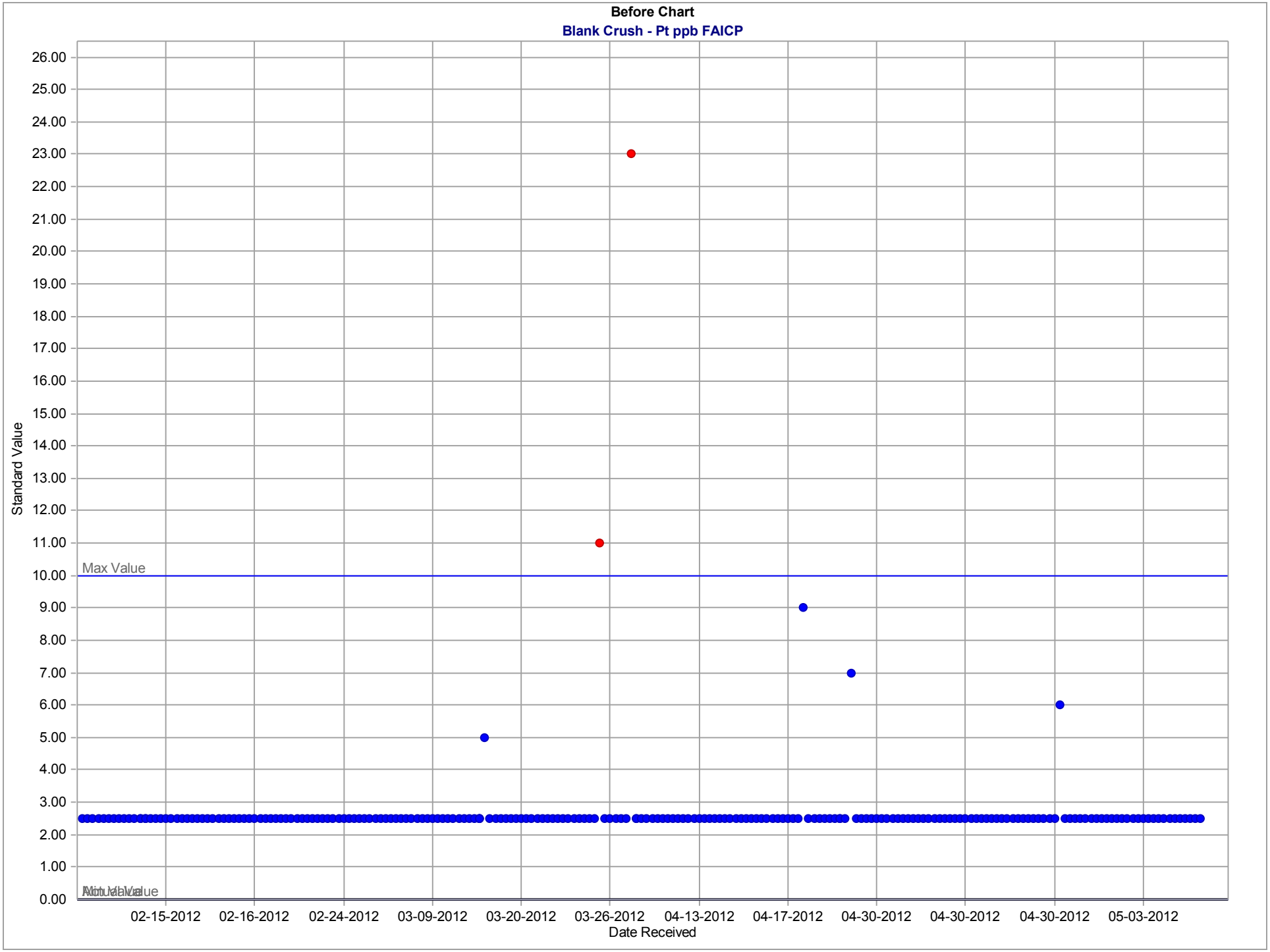
04-30-2012
04-30-2012
05-03-2012
Date Received


Before Chart
Blank Crush - Pd ppb FAICP
70.00
60.00
50.00
40.00
30.00
20.00
10.00
0.00
Standard Value
Max Value
02-15-2012
02-16-2012
02-24-2012
03-09-2012
03-20-2012
03-26-2012
04-13-2012
04-17-2012
04-30-2012
04-30-2012
04-30-2012
05-03-2012
Date Received


Before Chart
Blank Crush - Pt ppb FAICP
Standard Value
26.00
25.00
24.00
23.00
22.00
21.00
20.00
19.00
18.00
17.00
16.00
15.00
14.00
13.00
12.00
11.00
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Max Value
02-15-2012
02-16-2012
02-24-2012
03-09-2012
03-20-2012
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04-30-2012
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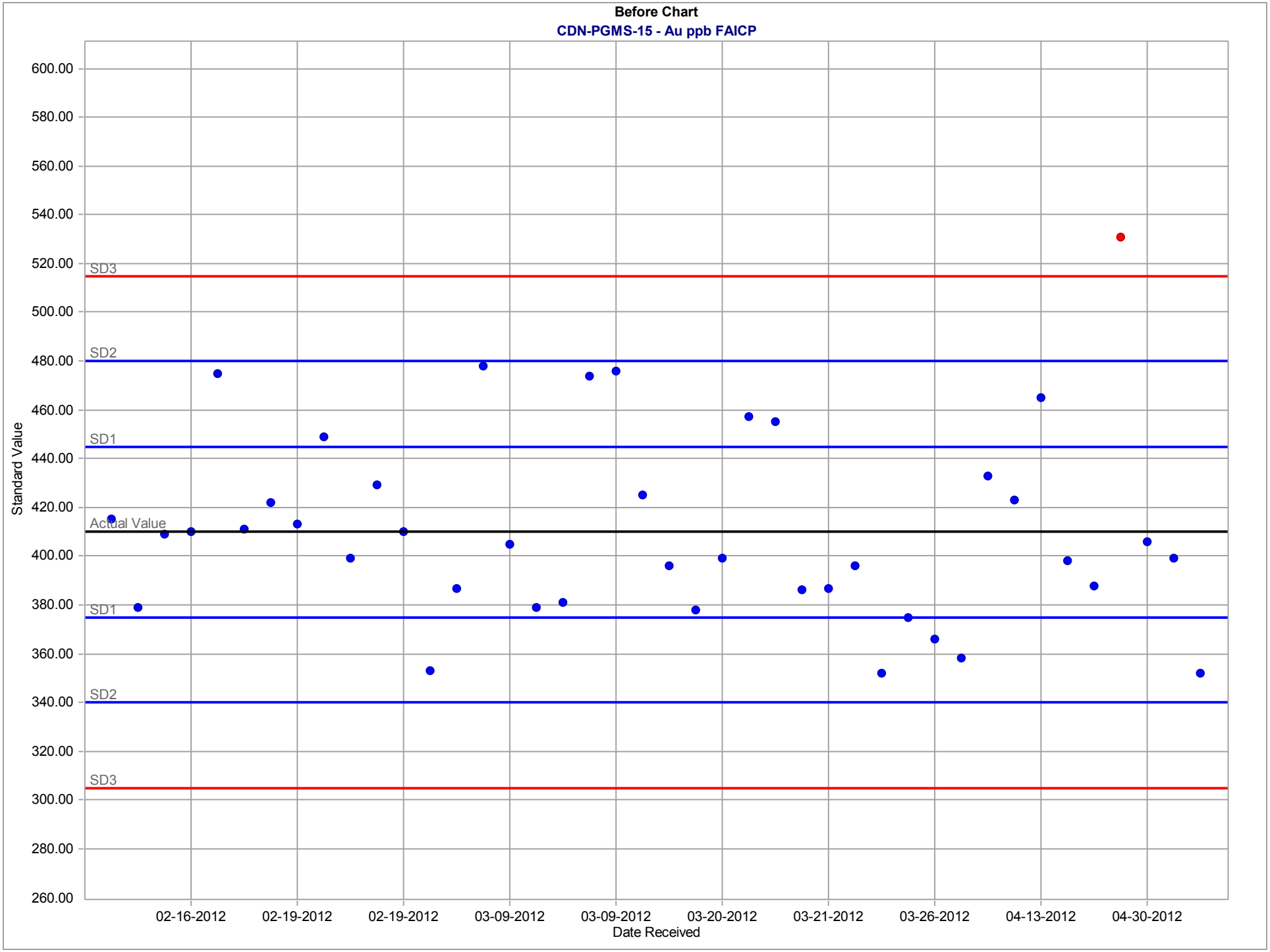
Date Received


Before Chart
CDN-PGMS-15 - Au ppb FAICP
600.00
580.00
560.00
540.00
520.00
500.00
480.00
460.00
440.00
420.00
400.00
380.00
360.00
340.00
320.00
300.00
280.00
260.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-16-2012
02-19-2012
02-19-2012
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03-09-2012
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03-21-2012
03-26-2012
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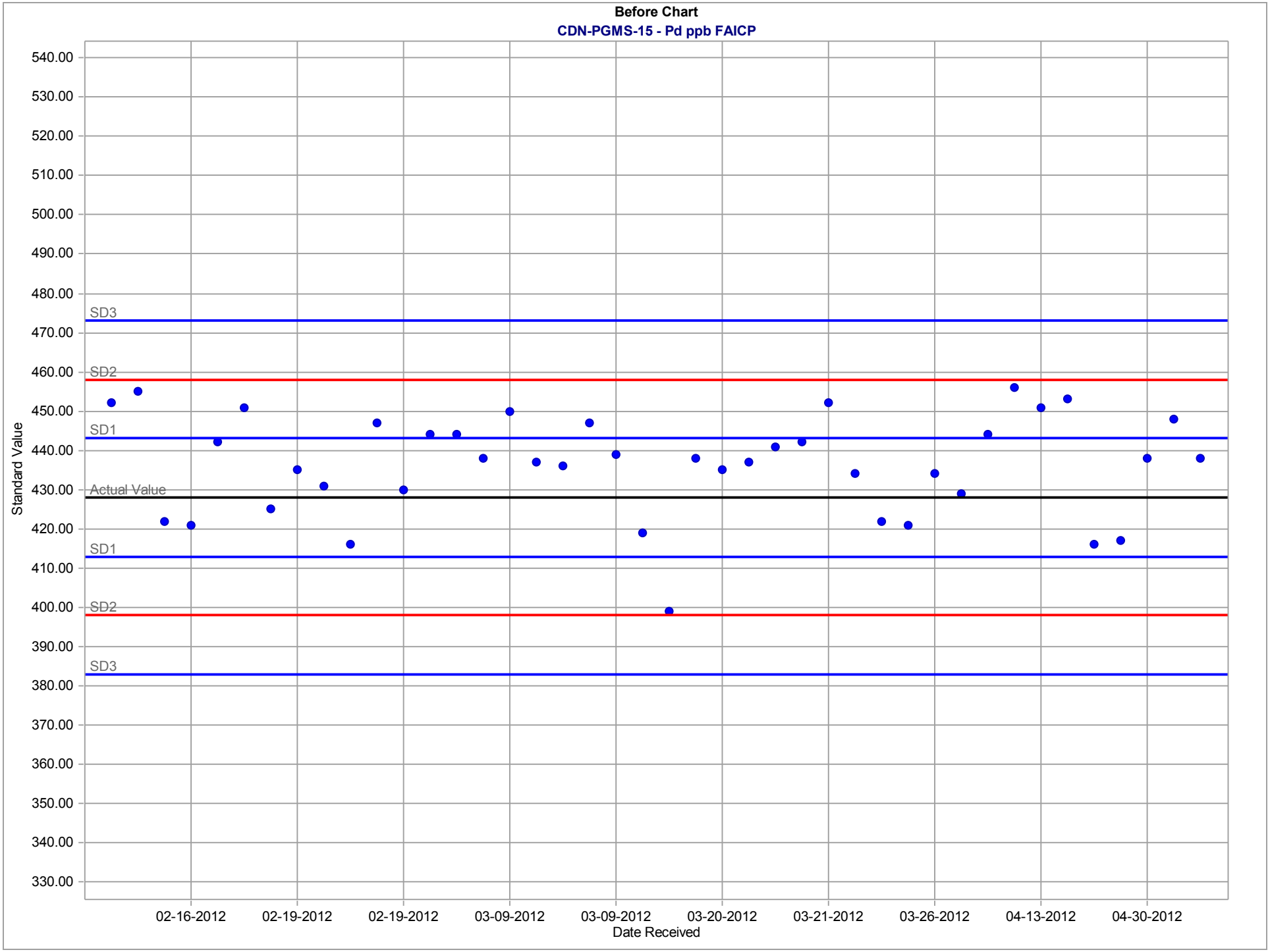
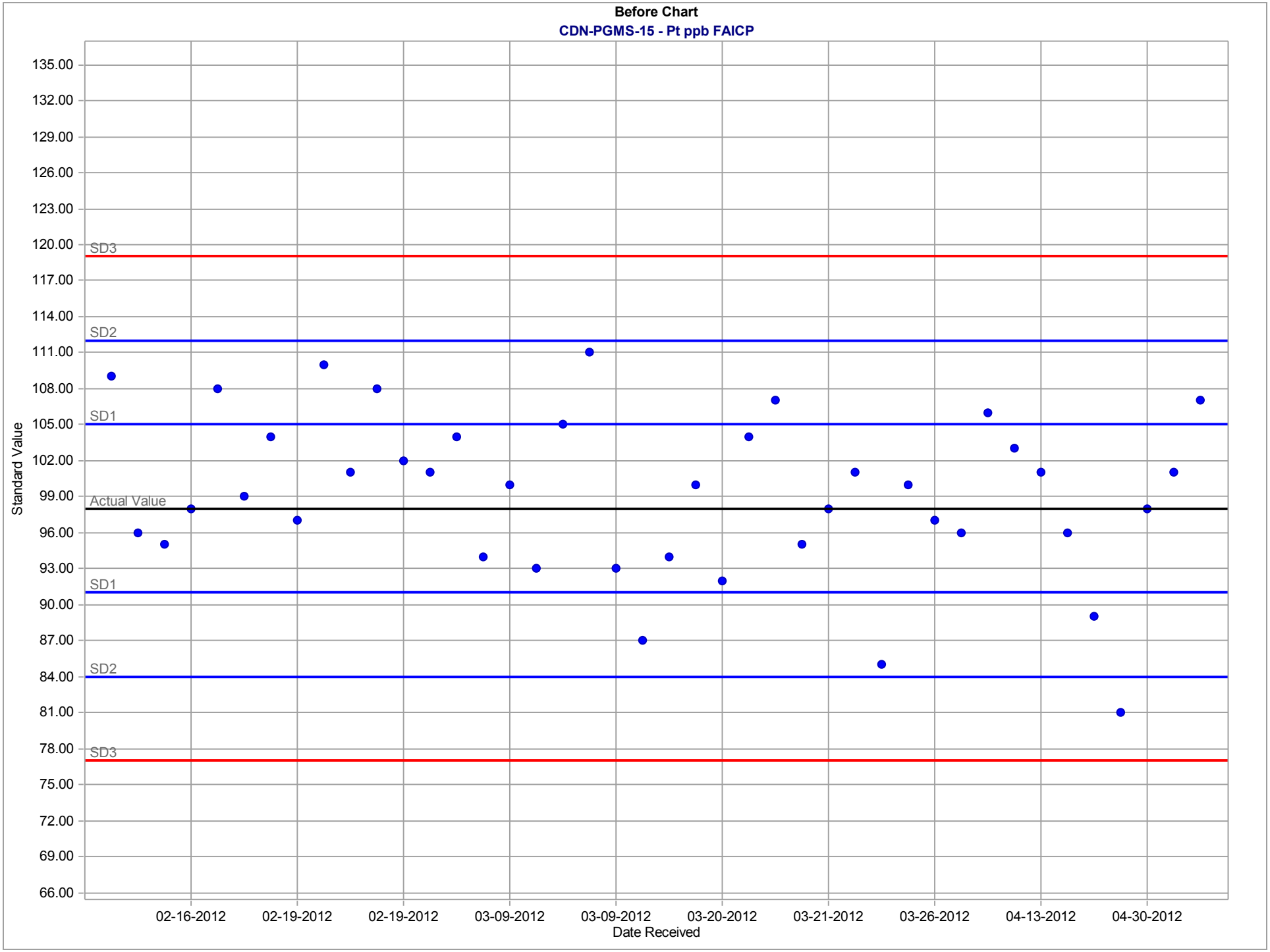
Date Received


Before Chart
CDN-PGMS-15 - Pd ppb FAICP
Standard Value
Date Received
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
540.00
530.00
520.00
510.00
500.00
490.00
480.00
470.00
460.00
450.00
440.00
430.00
420.00
410.00
400.00
390.00
380.00
370.00
360.00
350.00
340.00
330.00
02-16-2012
02-19-2012
02-19-2012
03-09-2012
03-09-2012
03-20-2012
03-21-2012
03-26-2012
04-13-2012
04-30-2012


Before Chart
CDN-PGMS-15 - Pt ppb FAICP
Standard Value
135.00
132.00
129.00
126.00
123.00
120.00
117.00
114.00
111.00
108.00
105.00
102.00
99.00
96.00
93.00
90.00
87.00
84.00
81.00
78.00
75.00
72.00
69.00
66.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
02-16-2012
02-19-2012
02-19-2012
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03-09-2012
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03-26-2012
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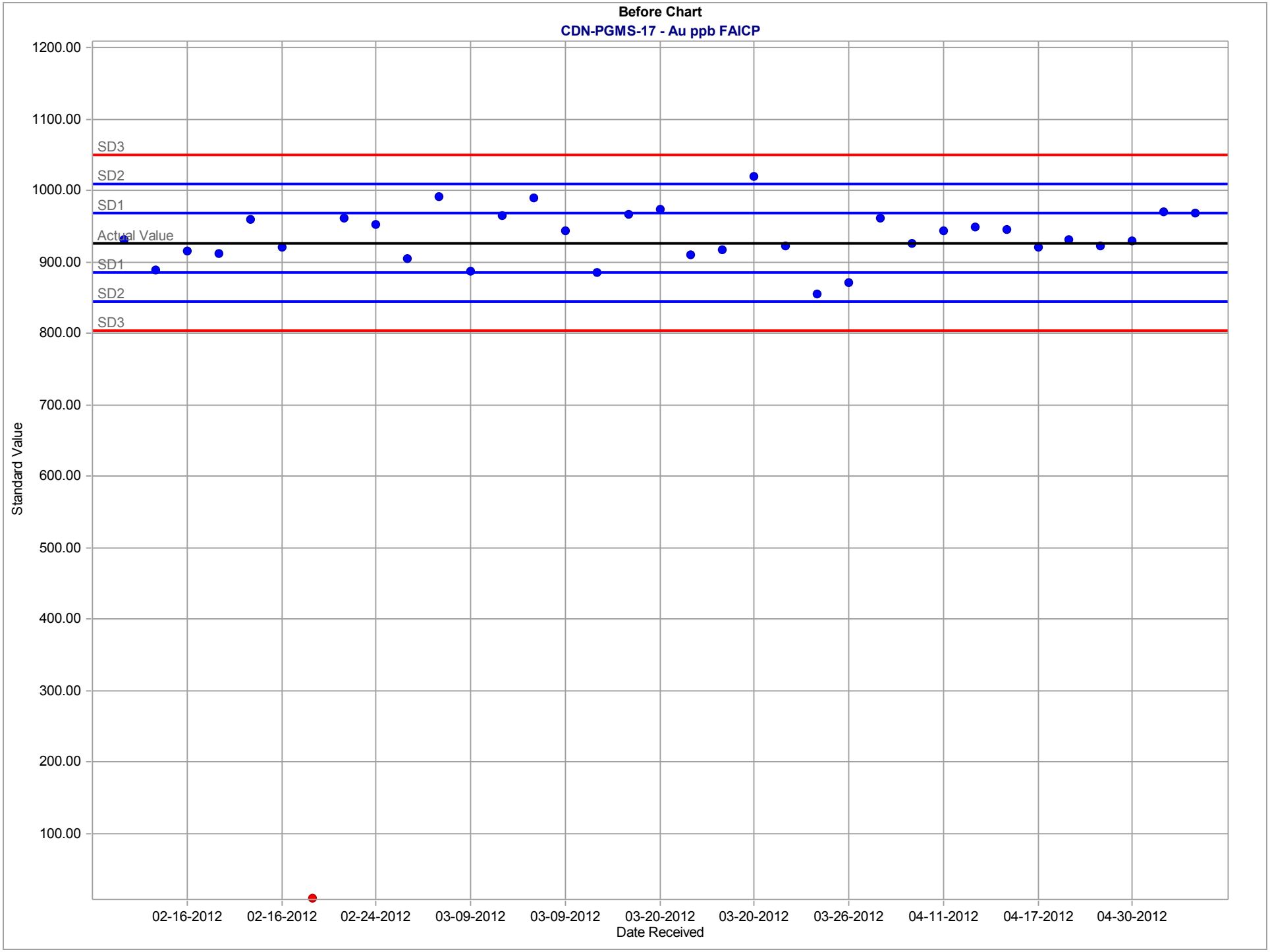
04-30-2012
Date Received


Before Chart
CDN-PGMS-17 - Au ppb FAICP
1200.00
1100.00
1000.00
900.00
800.00
700.00
600.00
500.00
400.00
300.00
200.00
100.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-16-2012
02-16-2012
02-24-2012
03-09-2012
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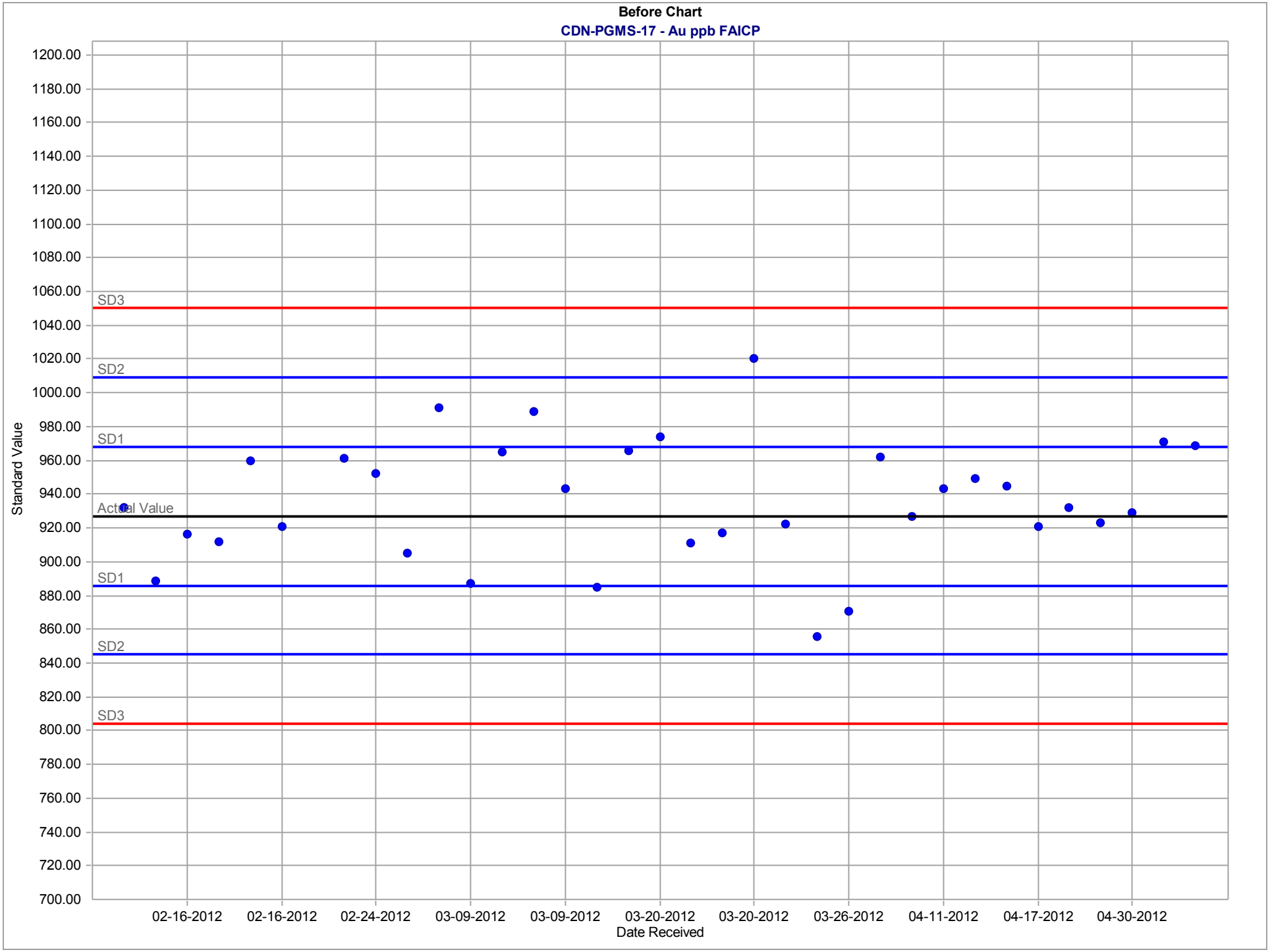
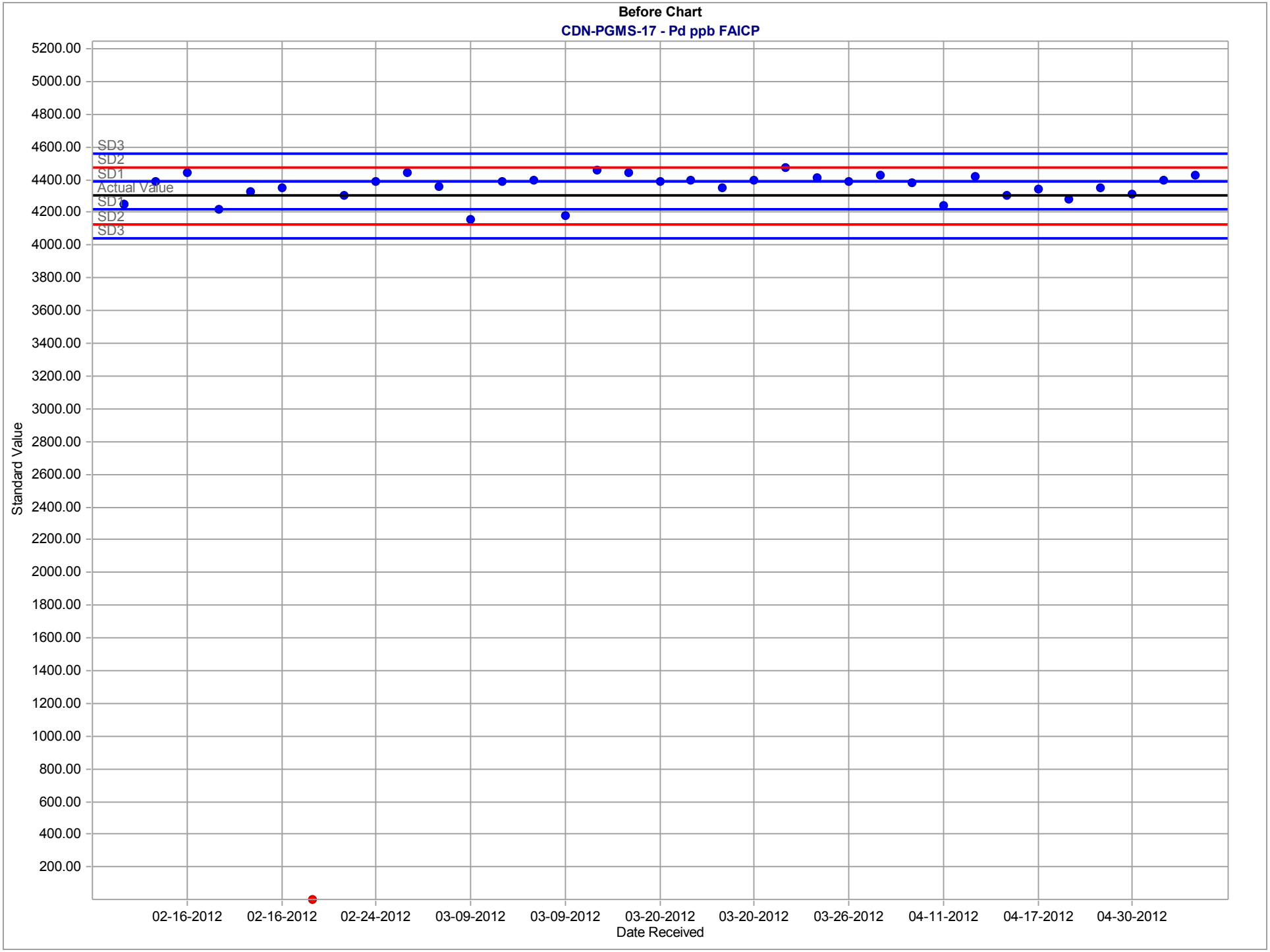
03-26-2012
04-11-2012
04-17-2012
04-30-2012
Date Received


Before Chart
CDN-PGMS-17 - Au ppb FAICP
Standard Value
1200.00
1180.00
1160.00
1140.00
1120.00
1100.00
1080.00
1060.00
1040.00
1020.00
1000.00
980.00
960.00
940.00
920.00
900.00
880.00
860.00
840.00
820.00
800.00
780.00
760.00
740.00
720.00
700.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
02-16-2012
02-16-2012
02-24-2012
03-09-2012
03-09-2012
03-20-2012
03-20-2012
03-26-2012
04-11-2012
04-17-2012
04-30-2012
Date Received


Before Chart
CDN-PGMS-17 - Pd ppb FAICP
5200.00
5000.00
4800.00
4600.00
4400.00
4200.00
4000.00
3800.00
3600.00
3400.00
3200.00
3000.00
2800.00
2600.00
2400.00
2200.00
2000.00
1800.00
1600.00
1400.00
1200.00
1000.00
800.00
600.00
400.00
200.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-16-2012
02-16-2012
02-24-2012
03-09-2012
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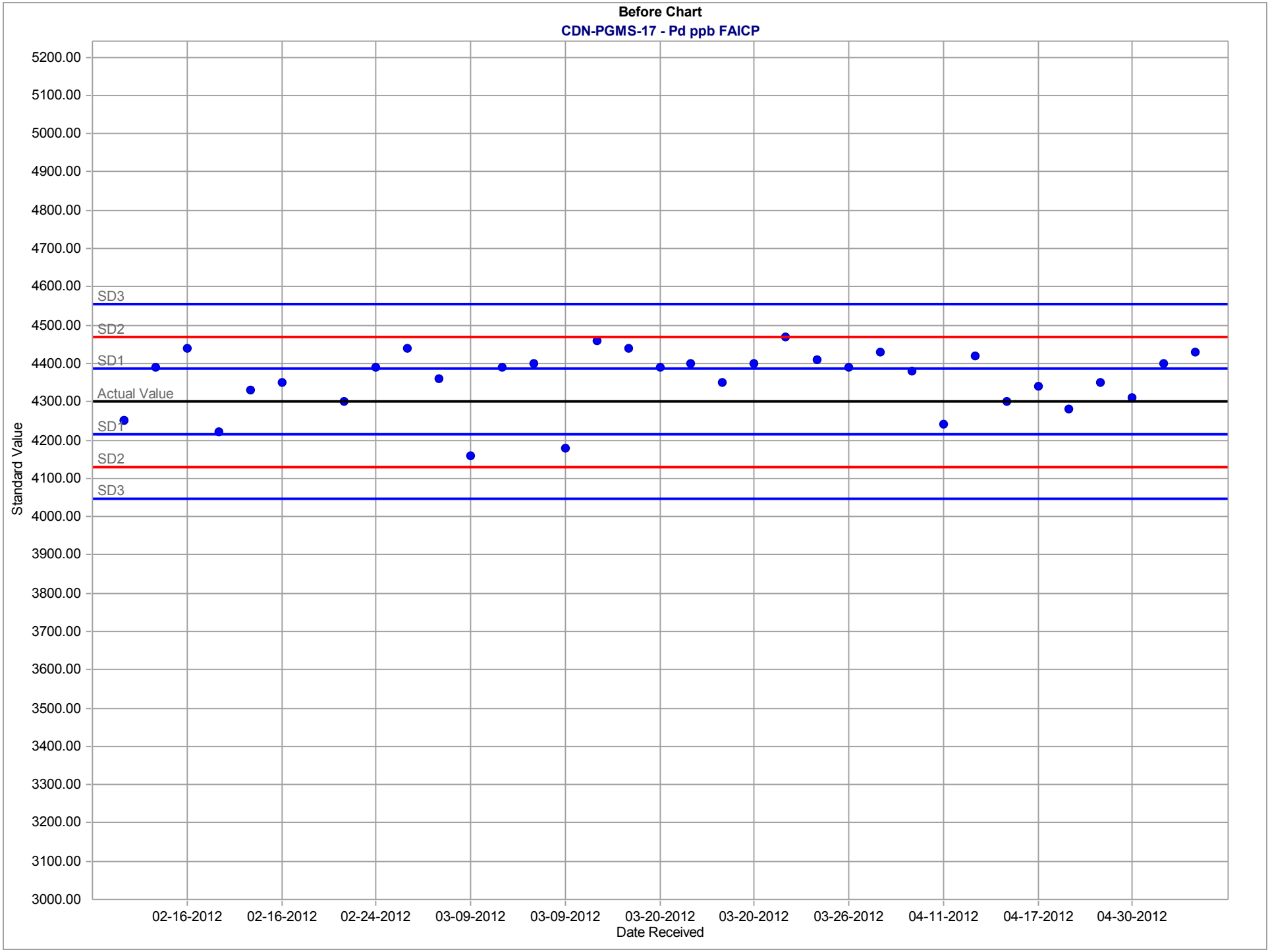
04-17-2012
04-30-2012
Date Received


Before Chart
CDN-PGMS-17 - Pd ppb FAICP
5200.00
5100.00
5000.00
4900.00
4800.00
4700.00
4600.00
4500.00
4400.00
4300.00
4200.00
4100.00
4000.00
3900.00
3800.00
3700.00
3600.00
3500.00
3400.00
3300.00
3200.00
3100.00
3000.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-16-2012
02-16-2012
02-24-2012
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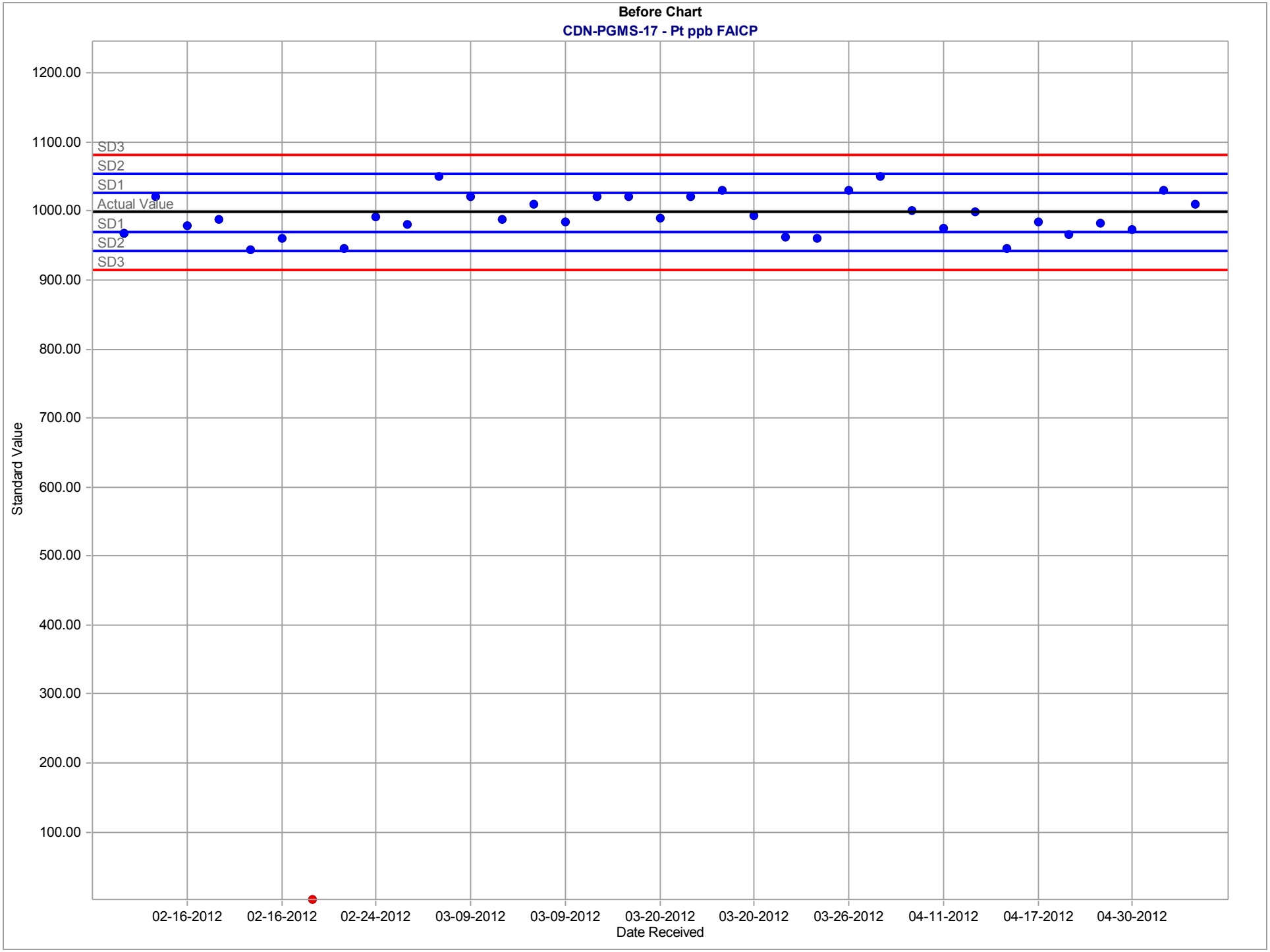
03-20-2012
03-20-2012
03-26-2012
04-11-2012
04-17-2012
04-30-2012
Date Received


Before Chart
CDN-PGMS-17 - Pt ppb FAICP
1200.00
1100.00
1000.00
900.00
800.00
700.00
600.00
500.00
400.00
300.00
200.00
100.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-16-2012
02-16-2012
02-24-2012
03-09-2012
03-09-2012
03-20-2012
03-20-2012
03-26-2012
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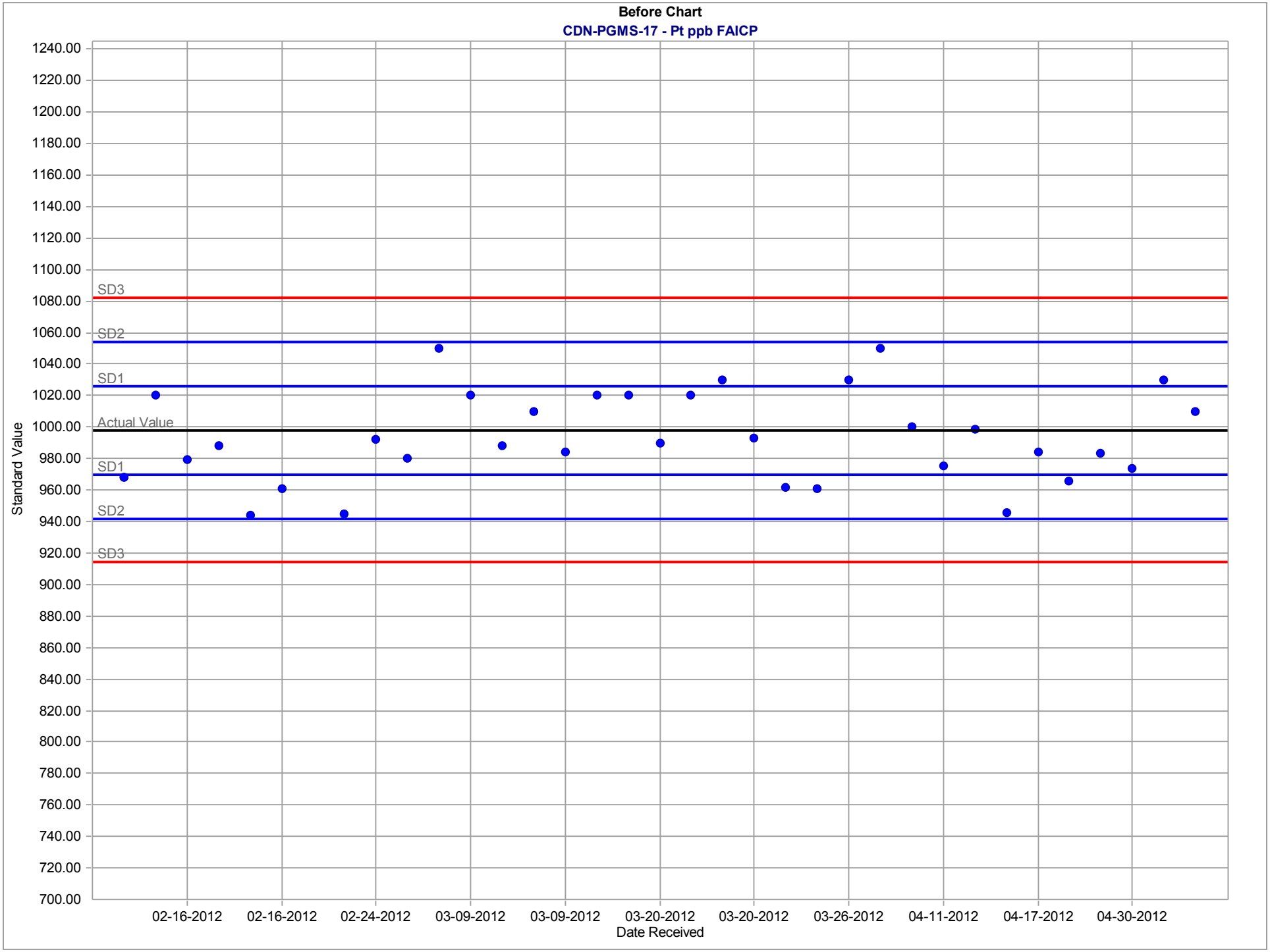
04-17-2012
04-30-2012
Date Received


Before Chart
CDN-PGMS-17 - Pt ppb FAICP
1240.00
1220.00
1200.00
1180.00
1160.00
1140.00
1120.00
1100.00
1080.00
1060.00
1040.00
1020.00
1000.00
980.00
960.00
940.00
920.00
900.00
880.00
860.00
840.00
820.00
800.00
780.00
760.00
740.00
720.00
700.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-16-2012
02-16-2012
02-24-2012
03-09-2012
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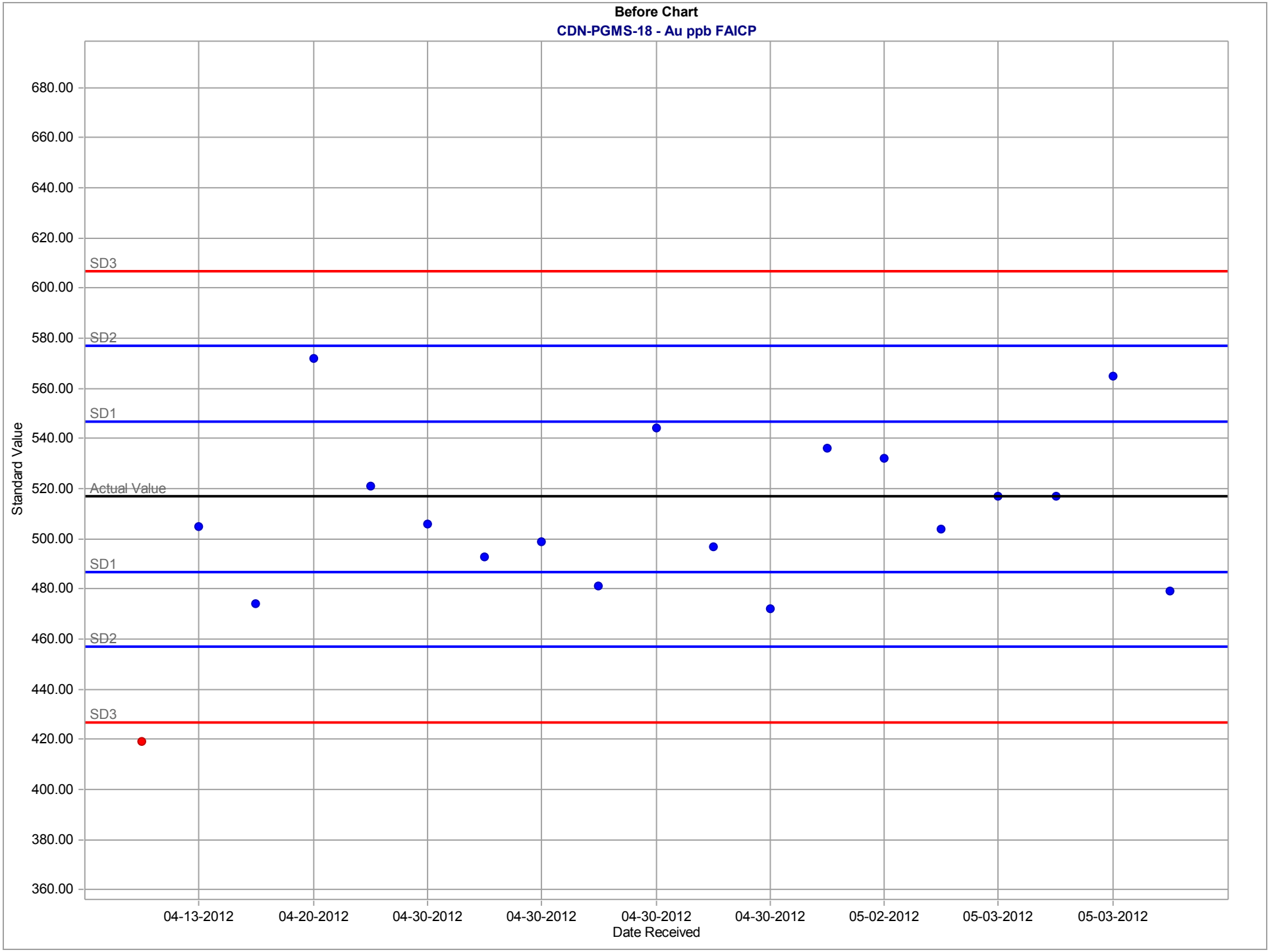
03-20-2012
03-20-2012
03-26-2012
04-11-2012
04-17-2012
04-30-2012
Date Received


Before Chart
CDN-PGMS-18 - Au ppb FAICP
680.00
660.00
640.00
620.00
SD3
600.00
580.00
SD2
560.00
SD1
540.00
520.00
Actual Value
500.00
SD1
480.00
460.00
SD2
440.00
SD3
420.00
400.00
380.00
360.00
Standard Value
04-13-2012
04-20-2012
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04-30-2012
04-30-2012
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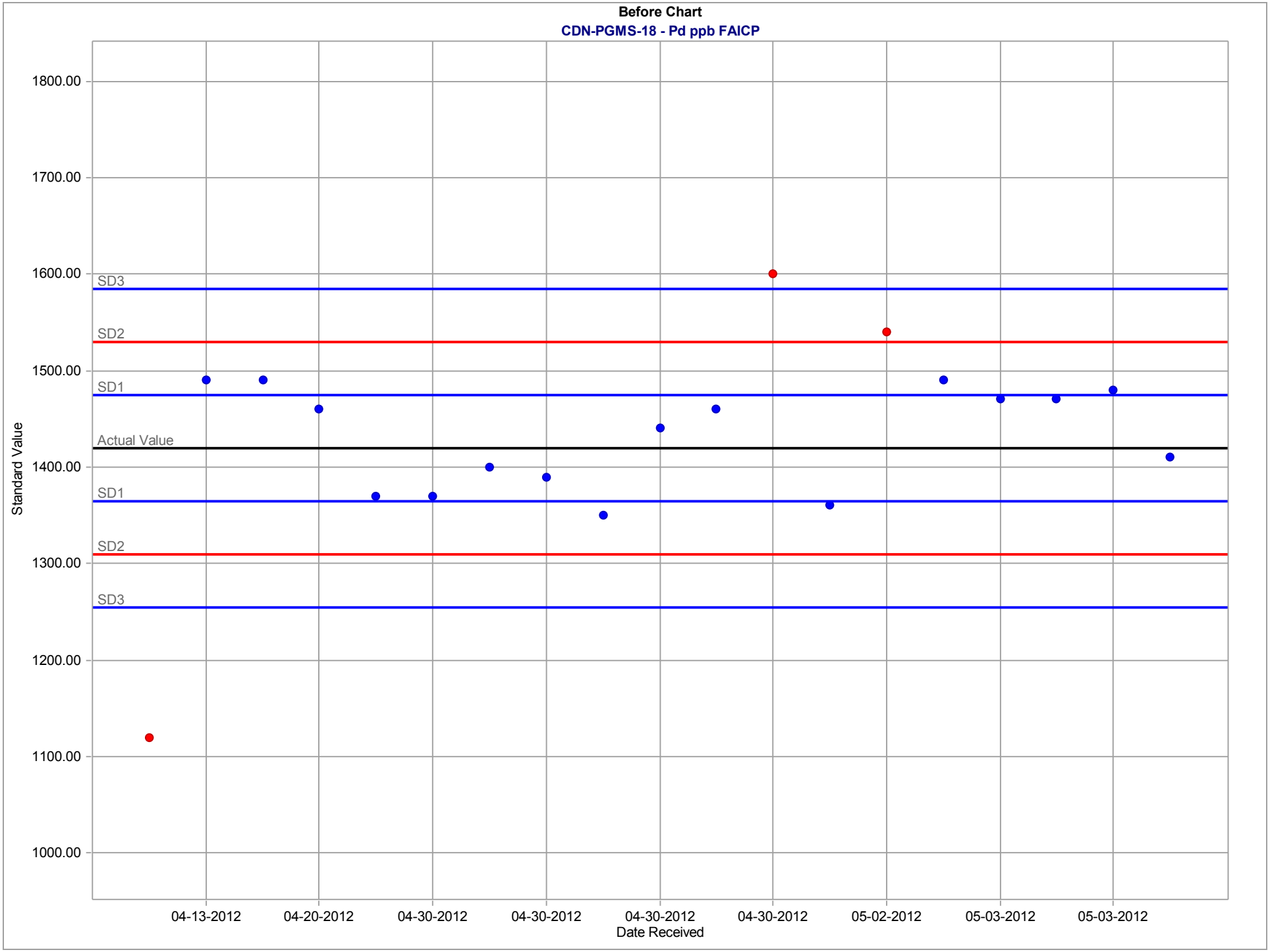
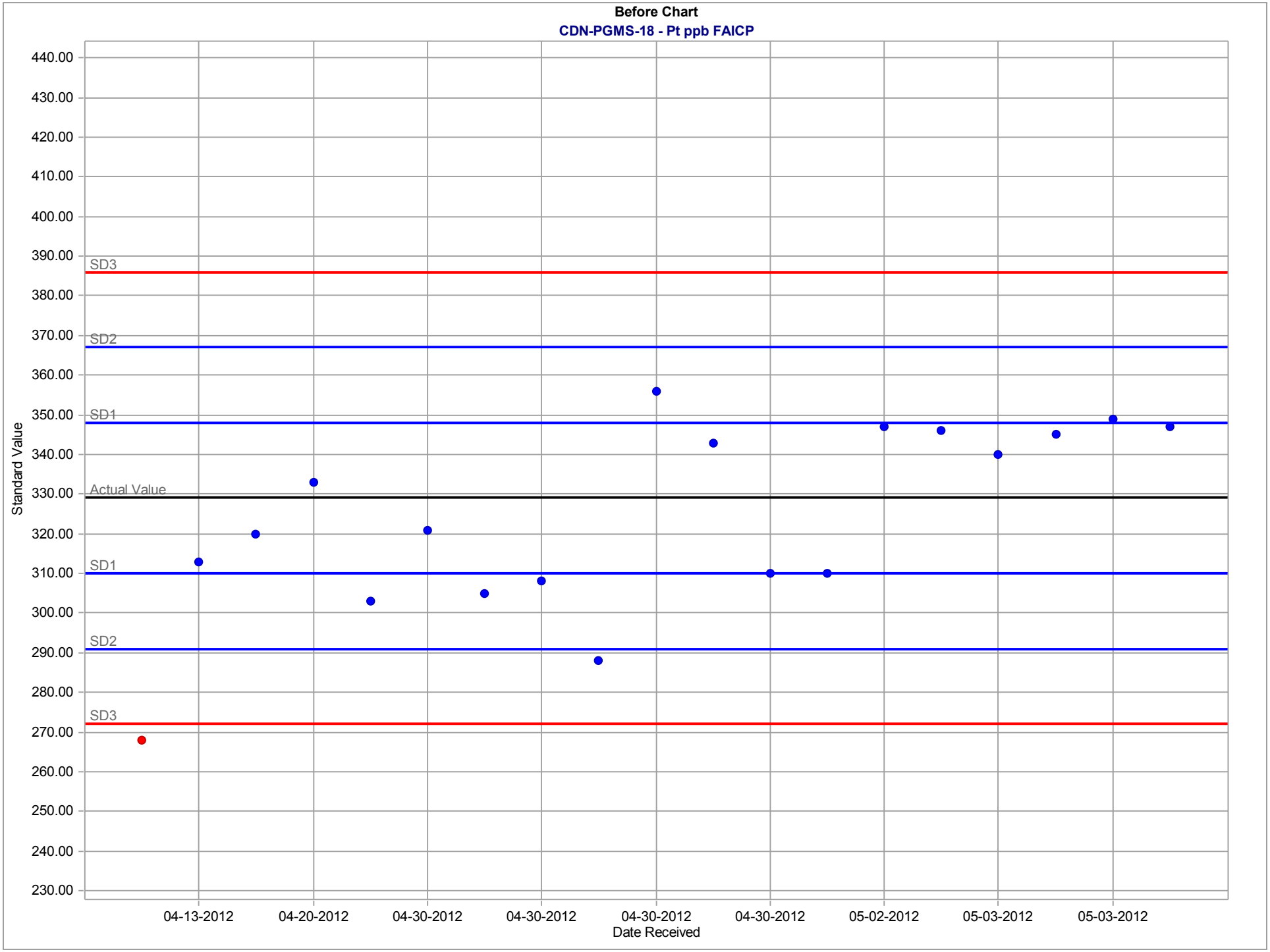
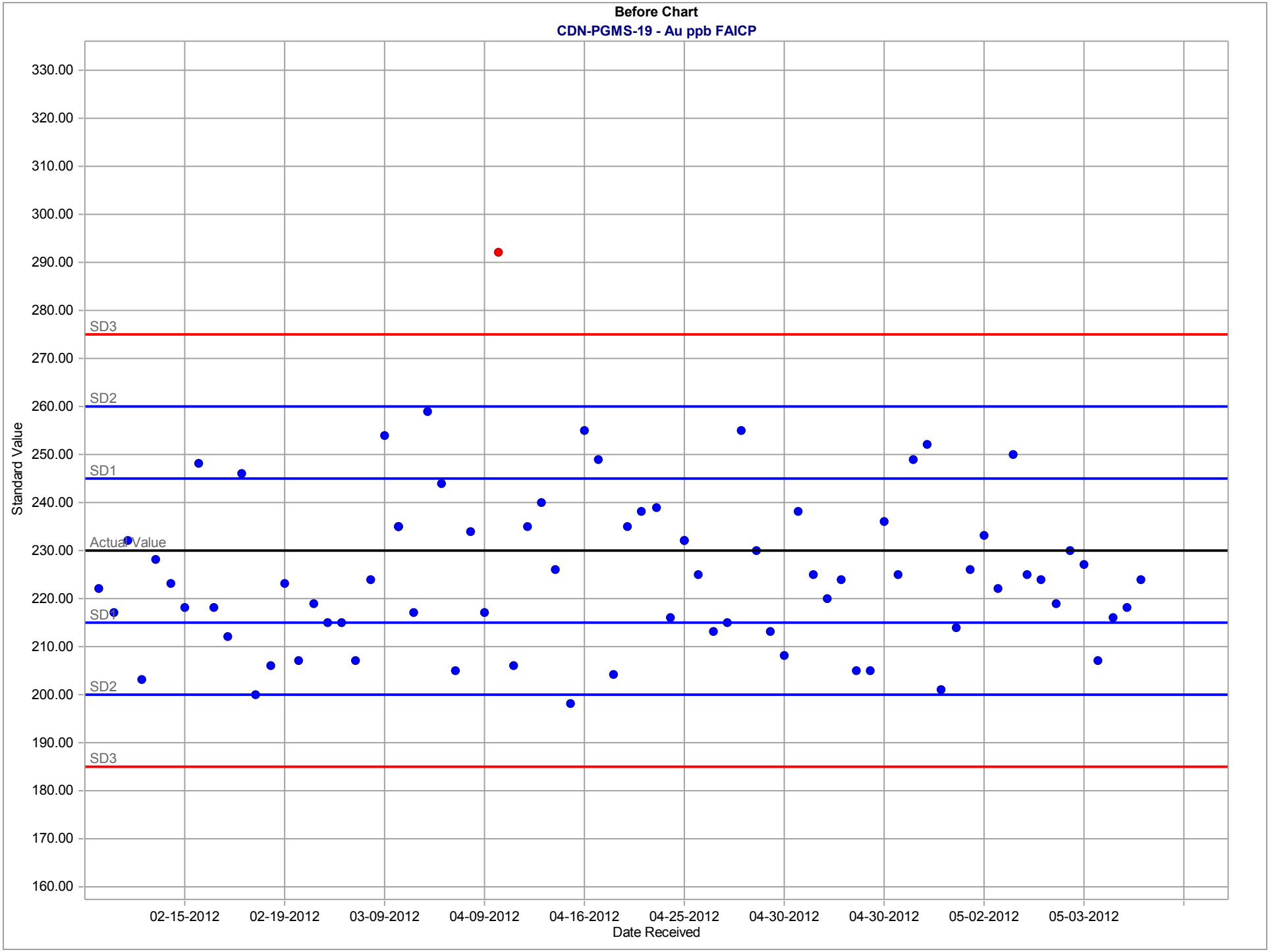
05-02-2012
05-03-2012
05-03-2012
Date Received


Before Chart
CDN-PGMS-18 - Pd ppb FAICP
1800.00
1700.00
1600.00
1500.00
1400.00
1300.00
1200.00
1100.00
1000.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
04-13-2012
04-20-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
05-02-2012
05-03-2012
05-03-2012
Date Received


Before Chart
CDN-PGMS-18 - Pt ppb FAICP
440.00
430.00
420.00
410.00
400.00
390.00
380.00
370.00
360.00
350.00
340.00
330.00
320.00
310.00
300.00
290.00
280.00
270.00
260.00
250.00
240.00
230.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
04-13-2012
04-20-2012
04-30-2012
04-30-2012
04-30-2012
04-30-2012
05-02-2012
05-03-2012
05-03-2012
Date Received


Before Chart
CDN-PGMS-19 - Au ppb FAICP
330.00
320.00
310.00
300.00
290.00
280.00
270.00
260.00
250.00
240.00
230.00
220.00
210.00
200.00
190.00
180.00
170.00
160.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
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02-19-2012
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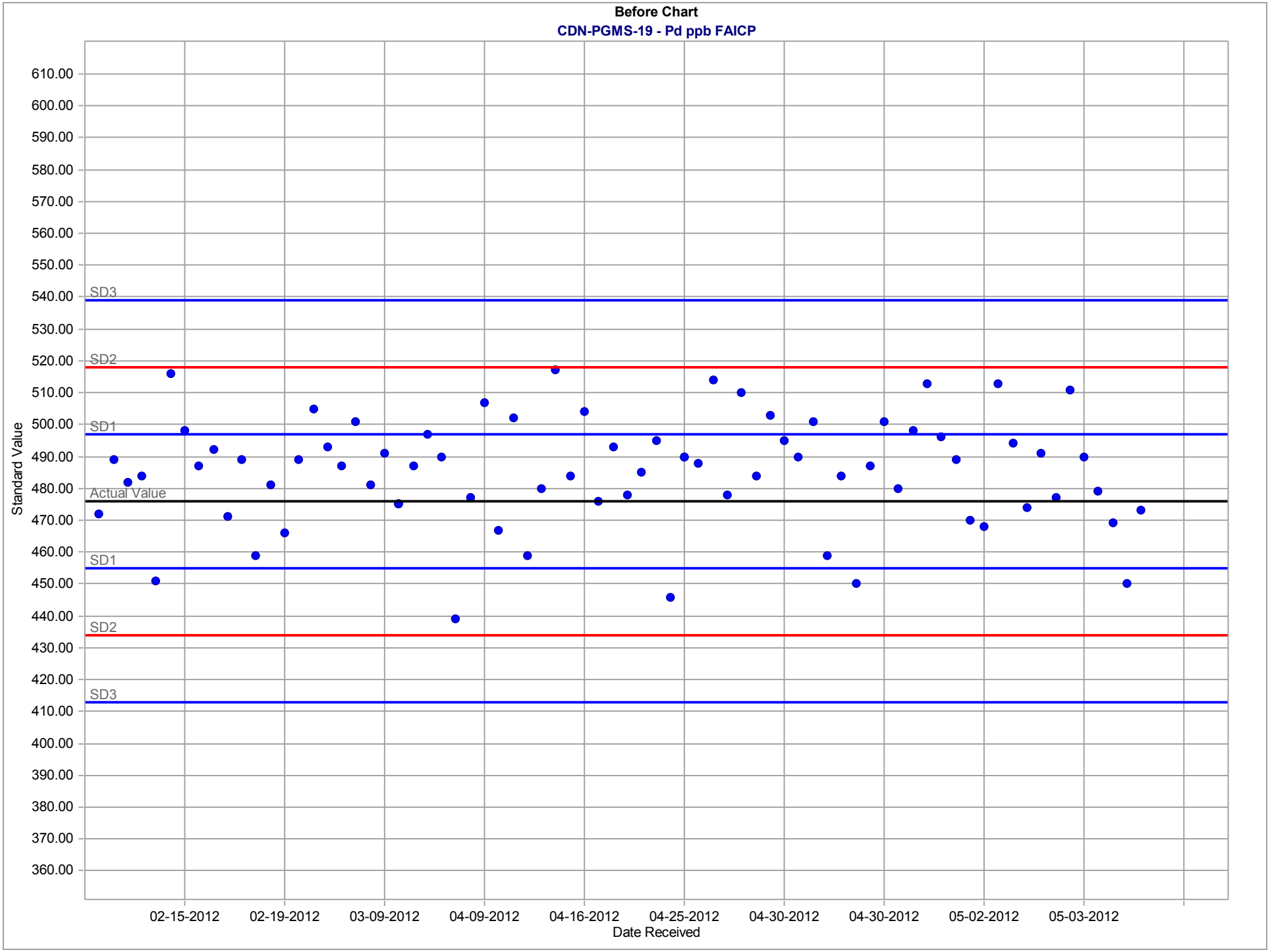
Date Received


Before Chart
CDN-PGMS-19 - Pd ppb FAICP
610.00
600.00
590.00
580.00
570.00
560.00
550.00
540.00
530.00
520.00
510.00
500.00
490.00
480.00
470.00
460.00
450.00
440.00
430.00
420.00
410.00
400.00
390.00
380.00
370.00
360.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-15-2012
02-19-2012
03-09-2012
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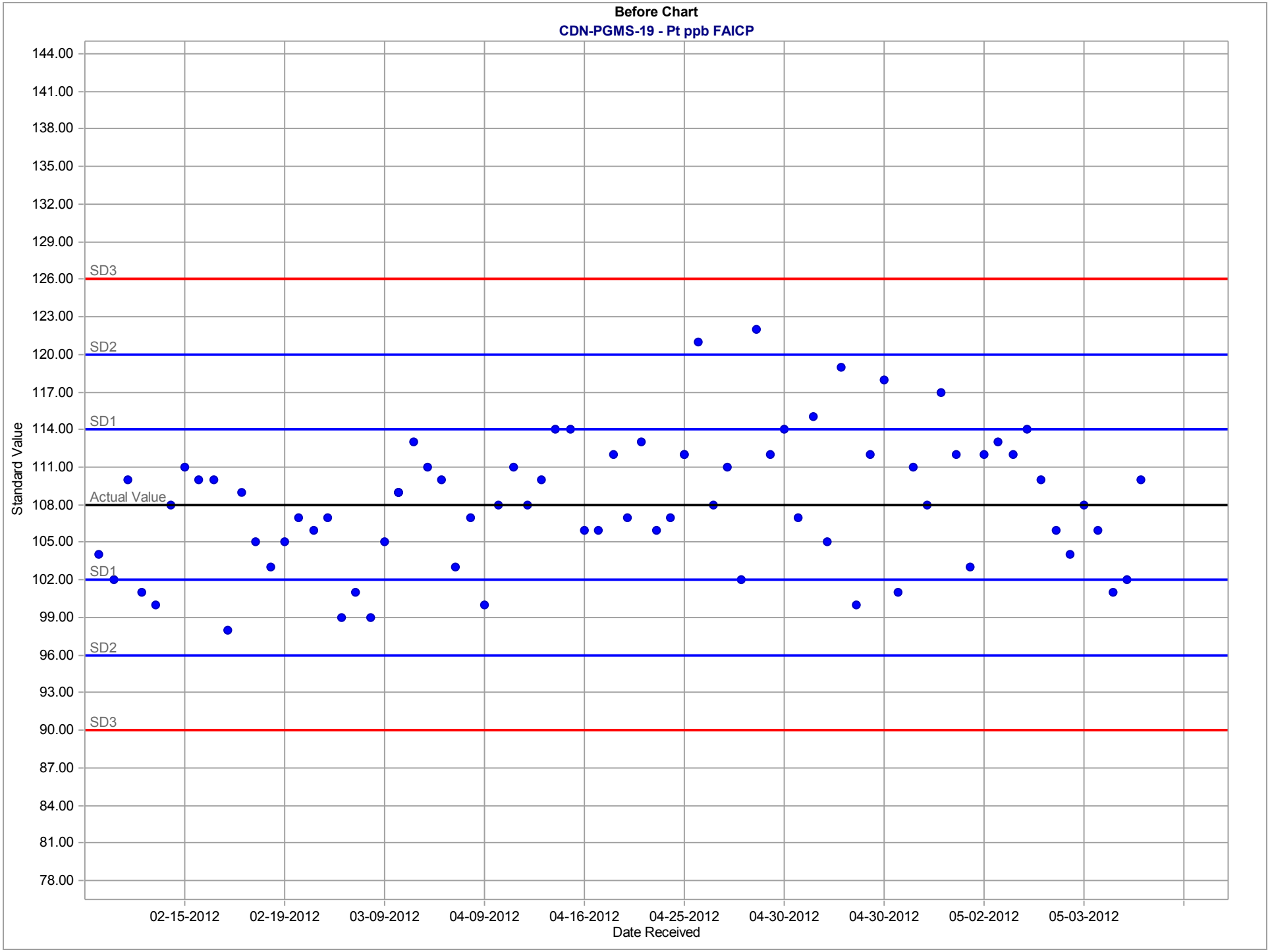
05-03-2012
Date Received


Before Chart
CDN-PGMS-19 - Pt ppb FAICP
144.00
141.00
138.00
135.00
132.00
129.00
126.00
123.00
120.00
117.00
114.00
111.00
108.00
105.00
102.00
99.00
96.00
93.00
90.00
87.00
84.00
81.00
78.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
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02-19-2012
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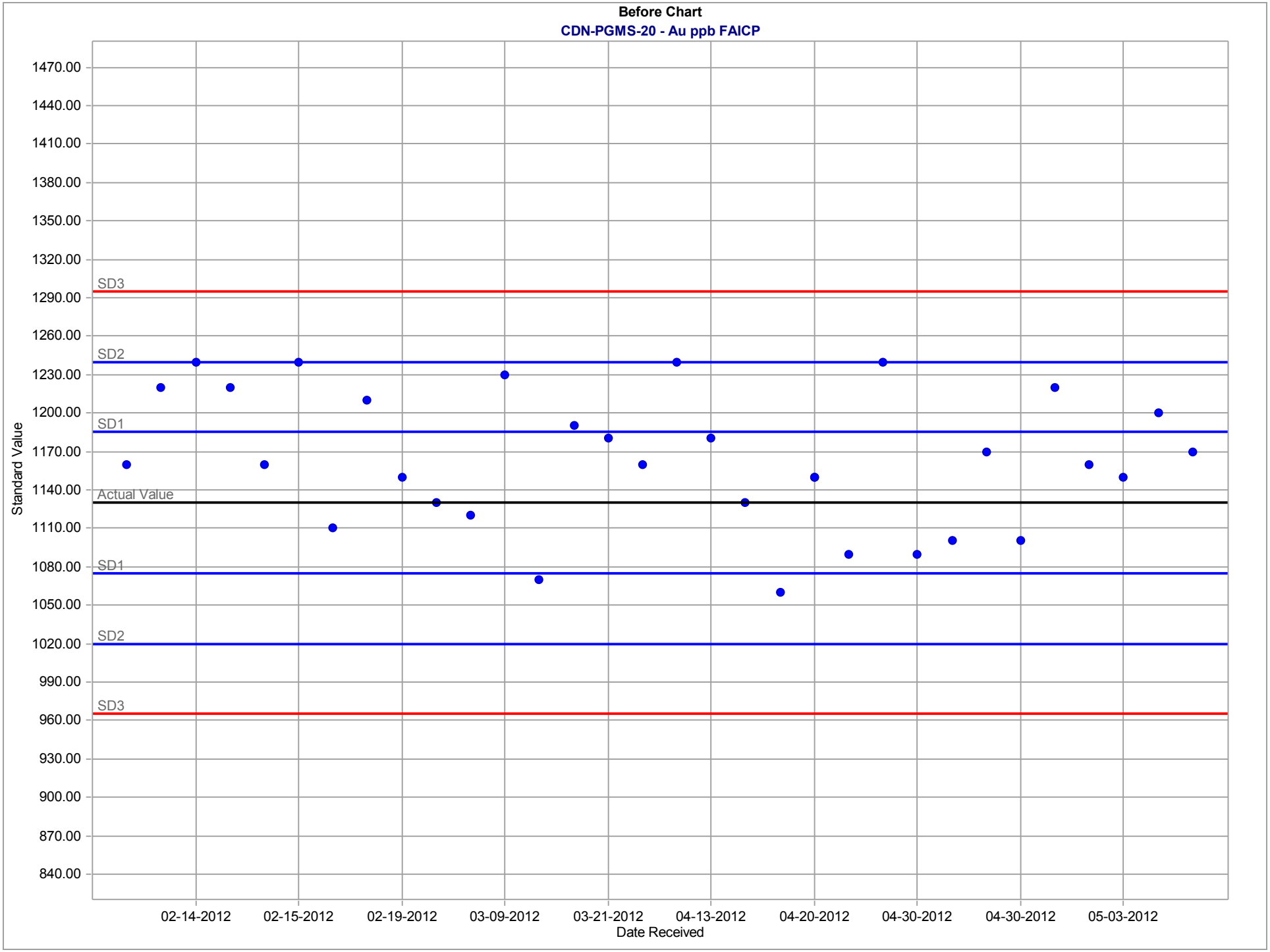
Date Received


Before Chart
CDN-PGMS-20 - Au ppb FAICP
1470.00
1440.00
1410.00
1380.00
1350.00
1320.00
SD3
1290.00
1260.00
SD2
1230.00
1200.00
SD1
1170.00
1140.00
Actual Value
1110.00
1080.00
SD1
1050.00
1020.00
SD2
990.00
SD3
960.00
930.00
900.00
870.00
840.00
Standard Value
02-14-2012
02-15-2012
02-19-2012
03-09-2012
03-21-2012
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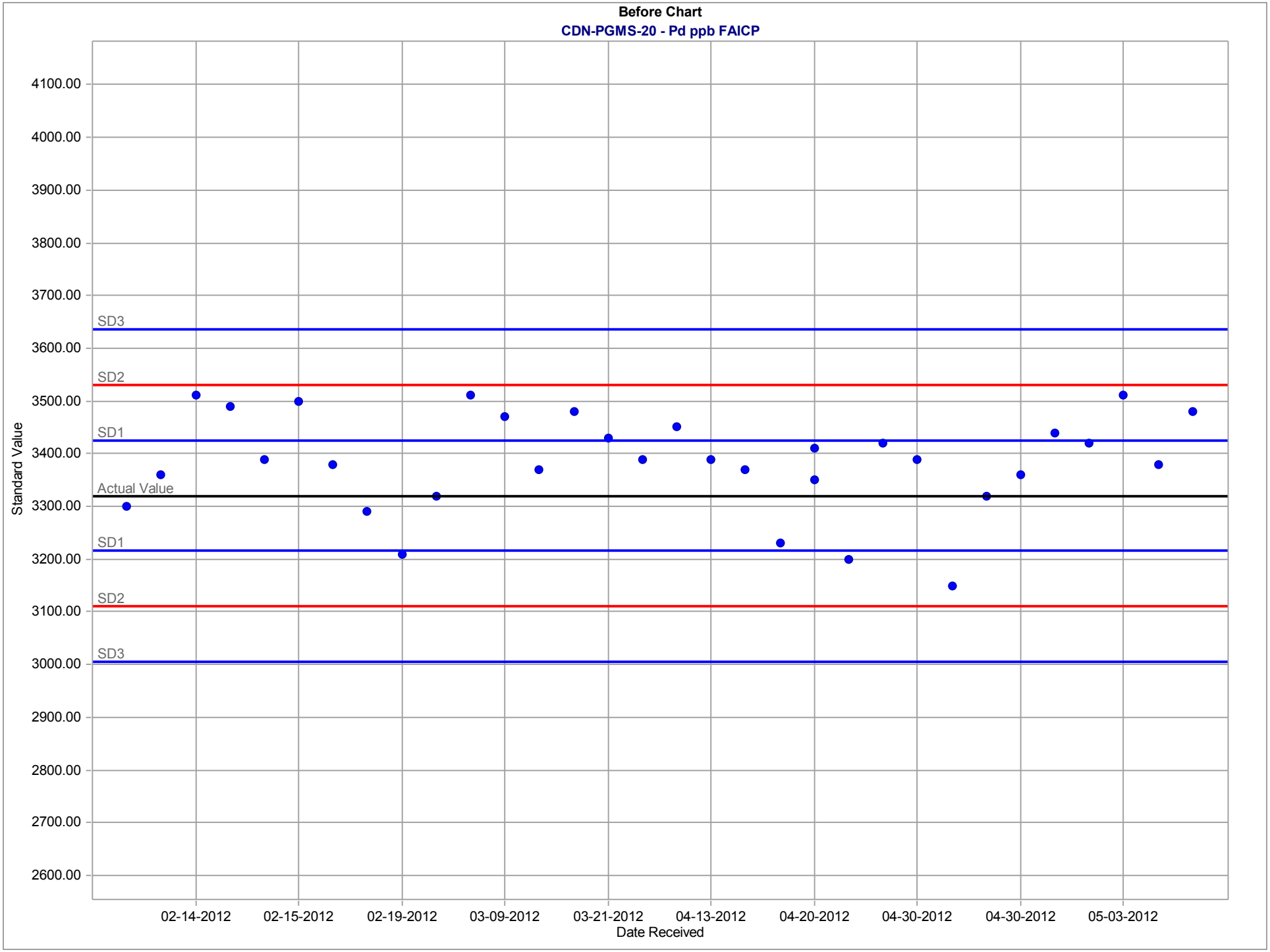
04-20-2012
04-30-2012
04-30-2012
05-03-2012
Date Received


Before Chart
CDN-PGMS-20 - Pd ppb FAICP
Standard Value
4100.00
4000.00
3900.00
3800.00
3700.00
3600.00
3500.00
3400.00
3300.00
3200.00
3100.00
3000.00
2900.00
2800.00
2700.00
2600.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
02-14-2012
02-15-2012
02-19-2012
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03-21-2012
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04-30-2012
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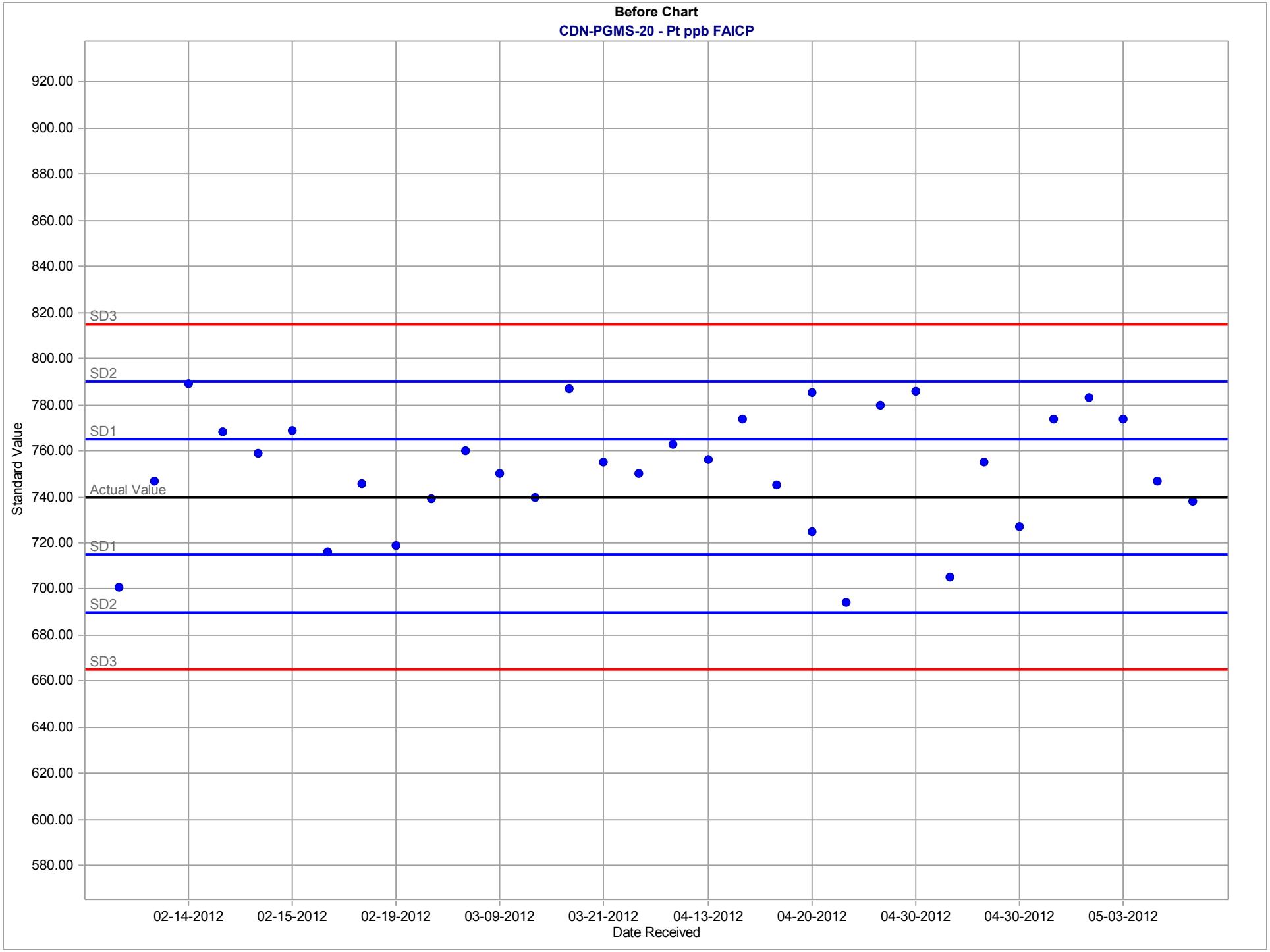
05-03-2012
Date Received


Before Chart
CDN-PGMS-20 - Pt ppb FAICP
920.00
900.00
880.00
860.00
840.00
820.00
800.00
780.00
760.00
740.00
720.00
700.00
680.00
660.00
640.00
620.00
600.00
580.00
SD3
SD2
SD1
Actual Value
SD1
SD2
SD3
Standard Value
02-14-2012
02-15-2012
02-19-2012
03-09-2012
03-21-2012
04-13-2012
04-20-2012
04-30-2012
04-30-2012
05-03-2012
Date Received

APPENDIX C

STANDARD REFERENCE MATERIAL CERTIFICATES

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466
(www.cdnlabs.com)

STANDARD REFERENCE MATERIAL: CDN-BL-10

Recommended values:
Gold concentration: ***< 0.01 g/t***
Platinum concentration: ***< 0.01 g/t***
Palladium concentration: ***< 0.01 g/t***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph. D., P. Geo.
DATE OF CERTIFICATION: November 25, 2011

ORIGIN OF REFERENCE MATERIAL:

Standard CDN-BL-10 was prepared using a blank granitic material.

METHOD OF PREPARATION:

The granitic material was dried, crushed, pulverized and then passed through a 270 mesh screen. The +270 material was discarded. The -270 (<53 micron) material was mixed for 5 days in a double-cone blender. Splits were taken and sent to 12 commercial laboratories for round robin assaying. Round robin results are displayed on the next page.

APPROXIMATE CHEMICAL COMPOSITION (by whole rock analysis):

	Percent			Percent
SiO2	69.7		Na2O	3.1
Al2O3	12.3		MgO	2.3
Fe2O3	5.2		K2O	0.9
CaO	3.8		TiO2	0.6
MnO	0.1		LOI	1.9
			S	<0.1

Statistical Procedures: There was no statistical analysis performed on the data.

Participating Laboratories: (not in same order as table of assays)

Acme Analytical Laboratories Ltd., Vancouver
Actlabs, Ancaster, Ontario
Actlabs, Thunder Bay, Ontario
ALS Chemex Laboratories, North Vancouver
AGAT, Mississauga, Ontario
AHK, Alaska, USA
Alex Stewart, Mendosa, Argentina
TSL Laboratories, Saskatoon
Genalysis, Perth, Australia
Labtium, Finland
SGS, Lima, Peru
Ultra Trace, Perth, Australia

Assay Procedure: assays were fire assay, AA or ICP finish on 30g samples.

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12
Sample	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm
CDN-BL-10-1	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-2	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-3	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-4	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-5	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-6	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-7	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-8	<0.01	<0.01	<0.01	0.01	<0.01	<0.01	<0.01	<0.01	0.02	<0.01	<0.01	<0.01
CDN-BL-10-9	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-10	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm
CDN-BL-10-1	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.06	<0.01	<0.01	<0.01
CDN-BL-10-2	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.02	<0.01	<0.01	<0.01
CDN-BL-10-3	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.03	<0.01	<0.01	<0.01
CDN-BL-10-4	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.03	<0.01	<0.01	<0.01
CDN-BL-10-5	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.03	<0.01	<0.01	<0.01
CDN-BL-10-6	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.04	<0.01	<0.01	<0.01
CDN-BL-10-7	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.04	<0.01	<0.01	<0.01
CDN-BL-10-8	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.06	<0.01	<0.01	<0.01
CDN-BL-10-9	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.02	<0.01	<0.01	<0.01
CDN-BL-10-10	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.02	<0.01	<0.01	<0.01
	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm
CDN-BL-10-1	<0.01	<0.01	<0.01	0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-2	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-3	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-4	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-5	<0.01	<0.01	<0.01	0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-6	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-7	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-8	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-9	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01
CDN-BL-10-10	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	< 0.01	<0.01	<0.01	<0.01

Legal Notice:

This certificate and the reference material described in it have been prepared with due care and attention. However CDN Resource Laboratories Ltd. nor Barry Smee accept any liability for any decisions or actions taken following the use of the reference material. Our liability is limited solely to the cost of the reference material.

Certified by Duncan Sanderson

Duncan Sanderson, Certified Assayer of B.C.

Geochemist BW Smee

Dr. Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

10945-B River Road, Delta, B.C., V4C 2R8, Ph: 604-540-2233, Fax: 604-540-2233 (www.cdnlabs.com)

STANDARD REFERENCE MATERIAL: CDN-BL-4

Recommended values:

Gold concentration: ***< 0.01 g/t***
Platinum concentration: ***< 0.01 g/t***
Palladium concentration: ***< 0.01 g/t***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph. D., P. Geo.
DATE OF CERTIFICATION: November 8, 2006

ORIGIN OF REFERENCE MATERIAL:

Standard CDN-BL-4 was prepared using a blank granitic material.

METHOD OF PREPARATION:

The granitic material was dried, crushed, pulverized and then passed through a 200 mesh screen. The +200 material was discarded. The -200 (<75 micron) material was mixed for 5 days in a double-cone blender. Splits were taken and sent to 10 commercial laboratories for round robin assaying. Round robin results are displayed below:

APPROXIMATE CHEMICAL COMPOSITION:

	Percent			Percent
SiO2	65.3		Na2O	3.8
Al2O3	13.9		MgO	2.2
Fe2O3	6.0		K2O	1.2
CaO	3.8		TiO2	0.6
MnO	0.1		LOI	1.4

Statistical Procedures: There was no statistical analysis performed on the data.

Participating Laboratories: (not in same order as table of assays)

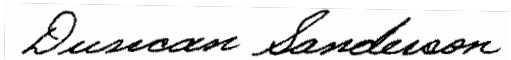
Acme Analytical Laboratories Ltd., Vancouver
Actlabs, Ontario, Canada
Alex Stewart Assayers Argentina Ltd.
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Genalysis Lab. Services, Australia
Omac Laboratory Ltd., Ireland
Skyline Laboratory, Arizona, USA
Teck Cominco - Global Discovery Laboratory, Vancouver
TSL Laboratories, Saskatoon

Assay Procedure: assays were fire assay, AA or ICP finish on 30g samples.

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10
Sample	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb
GS-BL4-1	10	4	6	<10	<10	9	4	7	<10	7
GS-BL4-2	10	6	5	<10	10	7	4	6	<10	5
GS-BL4-3	10	5	3	<10	10	9	5	6	<10	6
GS-BL4-4	<10	6	5	<10	<10	6	6	6	<10	6
GS-BL4-5	10	6	4	<10	10	6	6	6	<10	6
GS-BL4-6	10	5	4	<10	10	7	5	6	10	5
GS-BL4-7	<10	5	4	<10	<10	9	6	6	10	6
GS-BL4-8	<10	5	4	<10	<10	7	9	7	<10	5
GS-BL4-9	-	6	4	<10	10	7	5	17	<10	9
GS-BL4-10	10	6	1	<10	10	6	7	9	<10	5
	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb
GS-BL4-1	<10	< 5	8	<10	<10	<5	4	5	<10	4
GS-BL4-2	<10	< 5	<5	<10	<10	<5	4	4	<10	3
GS-BL4-3	<10	5	6	<10	<10	<5	4	4	<10	3
GS-BL4-4	<10	< 5	9	<10	<10	<5	4	5	<10	3
GS-BL4-5	<10	< 5	6	<10	10	<5	4	3	<10	3
GS-BL4-6	<10	5	<5	<10	10	<5	4	3	<10	4
GS-BL4-7	<10	< 5	<5	<10	10	<5	7	3	<10	2
GS-BL4-8	<10	9	5	<10	10	<5	4	4	<10	8
GS-BL4-9	-	< 5	5	<10	10	<5	4	4	<10	4
GS-BL4-10	<10	< 5	5	<10	<10	<5	4	3	<10	2
	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb
GS-BL4-1	10	5	5	<10	< 10	6	4	6	<10	6
GS-BL4-2	10	5	5	<10	< 10	6	4	6	<10	5
GS-BL4-3	10	5	5	<10	< 10	5	4	5	<10	6
GS-BL4-4	10	5	5	<10	10	6	4	5	<10	6
GS-BL4-5	<10	5	5	<10	< 10	7	4	5	<10	5
GS-BL4-6	<10	5	5	<10	< 10	6	4	5	<10	5
GS-BL4-7	<10	5	8	<10	< 10	6	4	6	<10	6
GS-BL4-8	10	5	5	<10	< 10	6	4	6	<10	6
GS-BL4-9	-	5	6	<10	< 10	6	4	6	<10	6
GS-BL4-10	10	5	4	<10	10	5	6	6	<10	5

Legal Notice:

This certificate and the reference material described in it have been prepared with due care and attention. However CDN Resource Laboratories Ltd. nor Barry Smee accept any liability for any decisions or actions taken following the use of the reference material. Our liability is limited solely to the cost of the reference material.

Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Dr. Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466
(www.cdnlabs.com)

STANDARD REFERENCE MATERIAL: CDN-BL-6

Recommended values:
Gold concentration: ***< 0.01 g/t***
Platinum concentration: ***< 0.01 g/t***
Palladium concentration: ***< 0.01 g/t***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph. D., P. Geo.
DATE OF CERTIFICATION: January 15, 2010

ORIGIN OF REFERENCE MATERIAL:

Standard CDN-BL-6 was prepared using a blank granitic material.

METHOD OF PREPARATION:

The granitic material was dried, crushed, pulverized and then passed through a 270 mesh screen. The +270 material was discarded. The -270 (<53 micron) material was mixed for 5 days in a double-cone blender. Splits were taken and sent to 10 commercial laboratories for round robin assaying. Round robin results are displayed below:

APPROXIMATE CHEMICAL COMPOSITION:

	Percent			Percent
SiO2	72.4		Na2O	2.9
Al2O3	10.9		MgO	1.7
Fe2O3	5.1		K2O	1.2
CaO	2.6		TiO2	0.3
MnO	0.1		LOI	1.2
C	0.1		S	0.1

Statistical Procedures: There was no statistical analysis performed on the data.

Participating Laboratories: (not in same order as table of assays)

Acme Analytical Laboratories Ltd., Vancouver
Actlabs, Ontario, Canada
Actlabs, Thunder Bay, Ontario
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Bourlamaque Laboratory, Quebec, Canada
Eco Tech Laboratory Ltd., Kamloops, B.C., Canada
International Plasma Labs, Richmond, B.C., Canada
Inspectorate America, Sparks, Nevada, USA
TSL Laboratories, Saskatoon

Assay Procedure: assays were fire assay, AA or ICP finish on 30g samples.
Assay results on the following page

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10
Sample	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb
BL-6-1	8	3	3	3	<5	<5	< 2	11	<5	< 10
BL-6-2	9	10	3	3	<5	<5	2.9	17	<5	< 10
BL-6-3	4	2	3	2	<5	<5	2.7	11	<5	< 10
BL-6-4	8	3	3	2	<5	<5	2.6	15	<5	< 10
BL-6-5	8	< 2	5	< 2	<5	<5	2.6	< 2	<5	< 10
BL-6-6	4	4	3	2	<5	<5	< 2	< 2	<5	< 10
BL-6-7	6	3	8	4	<5	<5	2.2	7	<5	< 10
BL-6-8	3	< 2	3	< 2	<5	<5	2.3	5	<5	< 10
BL-6-9	5	< 2	3	< 2	<5	<5	2.1	7	<5	< 10
BL-6-10	4	4	4	4	<5	<5	2.2	9	<5	< 10
	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb
BL-6-1	<3	< 5	<5	< 5	<5	<5	<10	7	<10	< 30
BL-6-2	<3	< 5	<5	< 5	<5	<5	<10	< 5	<10	< 30
BL-6-3	<3	< 5	<5	< 5	<5	<5	<10	< 5	<10	< 30
BL-6-4	<3	< 5	<5	< 5	<5	<5	<10	< 5	<10	< 30
BL-6-5	<3	< 5	<5	< 5	<5	<5	<10	< 5	<10	< 30
BL-6-6	<3	< 5	<5	< 5	<5	<5	<10	< 5	<10	< 30
BL-6-7	<3	< 5	<5	< 5	<5	<5	<10	< 5	<10	< 30
BL-6-8	<3	< 5	<5	< 5	5	<5	<10	< 5	<10	< 30
BL-6-9	<3	< 5	<5	< 5	<5	<5	<10	< 5	<10	< 30
BL-6-10	<3	< 5	<5	< 5	<5	<5	<10	5	<10	< 30
	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb
BL-6-1	2	< 5	2	< 5	<5	<5	<10	6	<5	< 20
BL-6-2	3	< 5	1	< 5	<5	<5	<10	<5	<5	< 20
BL-6-3	3	< 5	1	< 5	<5	<5	<10	13	<5	< 20
BL-6-4	2	< 5	1	< 5	<5	<5	<10	<5	<5	< 20
BL-6-5	4	< 5	2	< 5	<5	<5	<10	9	<5	< 20
BL-6-6	3	< 5	1	< 5	<5	<5	<10	<5	<5	< 20
BL-6-7	4	< 5	1	< 5	<5	<5	<10	<5	<5	< 20
BL-6-8	3	< 5	2	< 5	<5	<5	<10	10	<5	< 20
BL-6-9	2	< 5	2	< 5	<5	<5	<10	<5	<5	< 20
BL-6-10	2	< 5	2	< 5	<5	<5	<10	<5	<5	< 20

Legal Notice:

This certificate and the reference material described in it have been prepared with due care and attention. However CDN Resource Laboratories Ltd. nor Barry Smee accept any liability for any decisions or actions taken following the use of the reference material. Our liability is limited solely to the cost of the reference material.

Certified by

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Dr. Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466
(www.cdnlabs.com)

STANDARD REFERENCE MATERIAL: CDN-BL-7

Recommended values:
Gold concentration: ***< 0.01 g/t***
Platinum concentration: ***< 0.01 g/t***
Palladium concentration: ***< 0.01 g/t***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph. D., P. Geo.
DATE OF CERTIFICATION: June 22, 2010

ORIGIN OF REFERENCE MATERIAL:

Standard CDN-BL-7 was prepared using a blank granitic material.

METHOD OF PREPARATION:

The granitic material was dried, crushed, pulverized and then passed through a 270 mesh screen. The +270 material was discarded. The -270 (<53 micron) material was mixed for 5 days in a double-cone blender. Splits were taken and sent to 6 commercial laboratories for round robin assaying. Round robin results are displayed on the next page.

APPROXIMATE CHEMICAL COMPOSITION (whole rock analysis):

	Percent			Percent
SiO2	73.2		Na2O	2.9
Al2O3	10.7		MgO	1.6
Fe2O3	4.0		K2O	1.3
CaO	2.6		TiO2	0.4
MnO	0.1		LOI	1.5
C	0.1		S	0.1

Statistical Procedures: There was no statistical analysis performed on the data.

Participating Laboratories: (not in same order as table of assays)

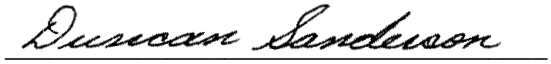
Acme Analytical Laboratories Ltd., Vancouver
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Eco Tech Laboratory Ltd., Kamloops, B.C., Canada
International Plasma Labs, Richmond, B.C., Canada
TSL Laboratories, Saskatoon

Assay Procedure: assays were fire assay, AA or ICP finish on 30g samples.
Assay results on the following page

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6
Sample	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb	Au ppb
CDN-BL-7-1	6	<1	<5	<5	8	<5
CDN-BL-7-2	3	1	<5	<5	<5	<5
CDN-BL-7-3	<2	5	<5	<5	<5	<5
CDN-BL-7-4	<2	2	<5	<5	8	<5
CDN-BL-7-5	5	2	<5	<5	5	<5
CDN-BL-7-6	<2	1	<5	<5	<5	<5
CDN-BL-7-7	<2	2	<5	<5	11	<5
CDN-BL-7-8	3	1	<5	<5	5	<5
CDN-BL-7-9	<2	1	<5	<5	8	<5
CDN-BL-7-10	<2	2	<5	<5	<5	<5
	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb	Pt ppb
CDN-BL-7-1	<3	5	<5	<5	<5	<10
CDN-BL-7-2	<3	<5	<5	<5	<5	<10
CDN-BL-7-3	<3	<5	<5	<5	<5	<10
CDN-BL-7-4	<3	<5	<5	<5	<5	<10
CDN-BL-7-5	<3	<5	<5	<5	<5	<10
CDN-BL-7-6	<3	<5	<5	<5	<5	<10
CDN-BL-7-7	<3	<5	<5	<5	<5	<10
CDN-BL-7-8	<3	<5	<5	<5	<5	<10
CDN-BL-7-9	<3	<5	<5	<5	<5	<10
CDN-BL-7-10	4	<5	<5	<5	<5	<10
	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb	Pd ppb
CDN-BL-7-1	<2	1	<5	<5	10	<5
CDN-BL-7-2	<2	2	<5	<5	<5	<5
CDN-BL-7-3	<2	2	<5	<5	14	<5
CDN-BL-7-4	<2	1	<5	<5	7	<5
CDN-BL-7-5	<2	1	<5	<5	<5	<5
CDN-BL-7-6	<2	1	<5	<5	8	<5
CDN-BL-7-7	<2	2	<5	<5	<5	<5
CDN-BL-7-8	<2	1	<5	<5	6	<5
CDN-BL-7-9	<2	1	<5	<5	9	<5
CDN-BL-7-10	<2	1	<5	<5	8	<5

Legal Notice:

This certificate and the reference material described in it have been prepared with due care and attention. However CDN Resource Laboratories Ltd. nor Barry Smee accept any liability for any decisions or actions taken following the use of the reference material. Our liability is limited solely to the cost of the reference material.

Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Dr. Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466
(www.cdnlabs.com)

STANDARD REFERENCE MATERIAL: CDN-BL-8

Recommended values:
Gold concentration: ***< 0.01 g/t***
Platinum concentration: ***< 0.01 g/t***
Palladium concentration: ***< 0.01 g/t***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph. D., P. Geo.
DATE OF CERTIFICATION: May 15, 2011

ORIGIN OF REFERENCE MATERIAL:

Standard CDN-BL-8 was prepared using a blank granitic material.

METHOD OF PREPARATION:

The granitic material was dried, crushed, pulverized and then passed through a 270 mesh screen. The +270 material was discarded. The -270 (<53 micron) material was mixed for 5 days in a double-cone blender. Splits were taken and sent to 6 commercial laboratories for round robin assaying. Round robin results are displayed on the next page.

APPROXIMATE CHEMICAL COMPOSITION (whole rock analysis):

	Percent			Percent
SiO2	75.0		Na2O	2.7
Al2O3	10.1		MgO	1.3
Fe2O3	3.5		K2O	1.0
CaO	2.5		TiO2	0.4
MnO	0.1		LOI	2.4
C	0.1		S	0.1

Statistical Procedures: There was no statistical analysis performed on the data.

Participating Laboratories: (not in same order as table of assays)

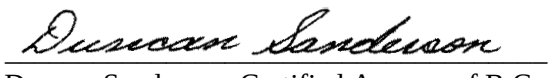
Acme Analytical Laboratories Ltd., Vancouver
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Eco Tech Laboratory Ltd., Kamloops, B.C., Canada
International Plasma Labs, Richmond, B.C., Canada
TSL Laboratories, Saskatoon

Assay Procedure: assays were fire assay, AA or ICP finish on 30g samples.
Assay results on the following page

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6
Sample	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm
CDN-BL-8-1	< 0.002	< 0.002	0.001	<0.005	0.001	<1
CDN-BL-8-2	< 0.002	0.006	0.001	<0.005	0.001	<1
CDN-BL-8-3	< 0.002	0.002	0.001	<0.005	0.001	<1
CDN-BL-8-4	< 0.002	0.002	0.001	<0.005	0.001	<1
CDN-BL-8-5	< 0.002	< 0.002	0.002	<0.005	0.001	<1
CDN-BL-8-6	< 0.002	0.003	0.002	<0.005	0.001	<1
CDN-BL-8-7	< 0.002	0.002	0.002	<0.005	0.002	<1
CDN-BL-8-8	< 0.002	< 0.002	0.001	<0.005	0.001	<1
CDN-BL-8-9	< 0.002	< 0.002	0.001	<0.005	0.002	<1
CDN-BL-8-10	< 0.002	< 0.002	0.001	<0.005	0.005	1
	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm
CDN-BL-8-1	< 0.005	< 0.005	<0.005	<0.010	<.001	0.2
CDN-BL-8-2	< 0.005	< 0.005	<0.005	<0.010	<.001	0.3
CDN-BL-8-3	< 0.005	< 0.005	<0.005	<0.010	<.001	0.5
CDN-BL-8-4	< 0.005	< 0.005	<0.005	<0.010	<.001	0.4
CDN-BL-8-5	< 0.005	< 0.005	<0.005	<0.010	<.001	0.4
CDN-BL-8-6	< 0.005	< 0.005	<0.005	<0.010	<.001	0.4
CDN-BL-8-7	< 0.005	< 0.005	<0.005	<0.010	<.001	0.4
CDN-BL-8-8	< 0.005	< 0.005	<0.005	<0.010	<.001	0.4
CDN-BL-8-9	< 0.005	< 0.005	<0.005	<0.010	<.001	0.5
CDN-BL-8-10	< 0.005	< 0.005	<0.005	<0.010	<.001	0.5
	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm
CDN-BL-8-1	< 0.005	< 0.005	<0.001	<0.005	<.001	<0.5
CDN-BL-8-2	< 0.005	< 0.005	<0.001	<0.005	<.001	<0.5
CDN-BL-8-3	< 0.005	< 0.005	0.001	<0.005	<.001	<0.5
CDN-BL-8-4	< 0.005	< 0.005	<0.001	<0.005	<.001	<0.5
CDN-BL-8-5	< 0.005	< 0.005	<0.001	<0.005	<.001	<0.5
CDN-BL-8-6	< 0.005	< 0.005	0.001	<0.005	<.001	<0.5
CDN-BL-8-7	< 0.005	< 0.005	<0.001	<0.005	<.001	<0.5
CDN-BL-8-8	< 0.005	< 0.005	<0.001	<0.005	<.001	0.6
CDN-BL-8-9	< 0.005	< 0.005	<0.001	<0.005	<.001	<0.5
CDN-BL-8-10	< 0.005	< 0.005	<0.001	<0.005	<.001	<0.5

Legal Notice:

This certificate and the reference material described in it have been prepared with due care and attention. However CDN Resource Laboratories Ltd. nor Barry Smee accept any liability for any decisions or actions taken following the use of the reference material. Our liability is limited solely to the cost of the reference material.

Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Dr. Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466
(www.cdnlabs.com)

STANDARD REFERENCE MATERIAL: CDN-BL-9

Recommended values:
Gold concentration: ***< 0.01 g/t***
Platinum concentration: ***< 0.01 g/t***
Palladium concentration: ***< 0.01 g/t***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph. D., P. Geo.
DATE OF CERTIFICATION: June 17, 2011

ORIGIN OF REFERENCE MATERIAL:

Standard CDN-BL-9 was prepared using a blank granitic material.

METHOD OF PREPARATION:

The granitic material was dried, crushed, pulverized and then passed through a 270 mesh screen. The +270 material was discarded. The -270 (<53 micron) material was mixed for 5 days in a double-cone blender. Splits were taken and sent to 8 commercial laboratories for round robin assaying. Round robin results are displayed on the next page.

APPROXIMATE CHEMICAL COMPOSITION (by whole rock analysis):

	Percent			Percent
SiO2	75.0		Na2O	2.7
Al2O3	10.1		MgO	1.3
Fe2O3	3.6		K2O	1.0
CaO	2.6		TiO2	0.4
MnO	0.1		LOI	1.8
C	0.1		S	<0.1

Statistical Procedures: There was no statistical analysis performed on the data.

Participating Laboratories: (not in same order as table of assays)

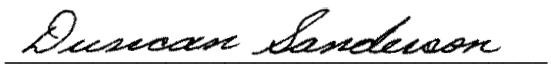
Acme Analytical Laboratories Ltd., Vancouver
Actlabs, Ancaster, Ontario
Actlabs, Thunder Bay, Ontario
ALS Chemex Laboratories, North Vancouver
Eco Tech Laboratory Ltd., Kamloops, B.C., Canada
TSL Laboratories, Saskatoon
Genalysis, Perth, Australia
Ultra Trace, Perth, Australia

Assay Procedure: assays were fire assay, AA or ICP finish on 30g samples.
Assay results on the following page

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8
Sample	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm	Au ppm
CDN-BL-9-1	0.003	0.004	<.005	0.005	<0.001	<.001	<.005	0.002
CDN-BL-9-2	0.002	0.002	<.005	0.005	<0.001	<.001	0.005	0.001
CDN-BL-9-3	< 0.002	0.002	<.005	0.004	0.002	<.001	<.005	0.001
CDN-BL-9-4	< 0.002	0.003	<.005	0.005	<0.001	<.001	<.005	0.001
CDN-BL-9-5	< 0.002	< 0.002	<.005	0.007	<0.001	0.001	<.005	0.001
CDN-BL-9-6	< 0.002	< 0.002	<.005	0.004	<0.001	0.002	<.005	0.002
CDN-BL-9-7	0.002	0.002	<.005	0.003	0.001	0.001	<.005	0.001
CDN-BL-9-8	< 0.002	0.003	<.005	0.004	<.001	0.003	0.005	0.001
CDN-BL-9-9	< 0.002	< 0.002	<.005	0.002	0.002	0.002	<.005	0.004
CDN-BL-9-10	< 0.002	< 0.002	<.005	0.005	<.001	<.001	<.005	0.003
	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm	Pt ppm
CDN-BL-9-1	< 0.005	< 0.005	<0.010	< 0.005	0.003	<.001	<.005	<0.005
CDN-BL-9-2	< 0.005	< 0.005	<0.010	< 0.005	0.002	<.001	<.005	<0.005
CDN-BL-9-3	< 0.005	< 0.005	<0.010	< 0.005	0.004	<.001	<.005	<0.005
CDN-BL-9-4	< 0.005	< 0.005	<0.010	< 0.005	0.003	<.001	<.005	<0.005
CDN-BL-9-5	< 0.005	< 0.005	<0.010	< 0.005	0.003	<.001	<.005	<0.005
CDN-BL-9-6	< 0.005	< 0.005	<0.010	< 0.005	0.005	<.001	<.005	<0.005
CDN-BL-9-7	< 0.005	< 0.005	<0.010	< 0.005	0.003	<.001	<.005	<0.005
CDN-BL-9-8	< 0.005	< 0.005	<0.010	< 0.005	0.004	<.001	<.005	<0.005
CDN-BL-9-9	< 0.005	< 0.005	<0.010	< 0.005	0.004	<.001	<.005	<0.005
CDN-BL-9-10	< 0.005	< 0.005	<0.010	< 0.005	0.003	<.001	<.005	<0.005
	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm	Pd ppm
CDN-BL-9-1	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	0.001
CDN-BL-9-2	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	0.001
CDN-BL-9-3	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	<0.001
CDN-BL-9-4	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	0.001
CDN-BL-9-5	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	0.001
CDN-BL-9-6	< 0.005	0.005	<.005	< 0.005	<0.001	<.001	<.005	<0.001
CDN-BL-9-7	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	<0.001
CDN-BL-9-8	< 0.005	0.007	<.005	< 0.005	<0.001	<.001	<.005	<0.001
CDN-BL-9-9	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	0.001
CDN-BL-9-10	< 0.005	< 0.005	<.005	< 0.005	<0.001	<.001	<.005	0.001

Legal Notice:

This certificate and the reference material described in it have been prepared with due care and attention. However CDN Resource Laboratories Ltd. nor Barry Smee accept any liability for any decisions or actions taken following the use of the reference material. Our liability is limited solely to the cost of the reference material.

Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Dr. Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

10945-B River Road, Delta, B.C., V4C 2R8, 604 596-2245, Fax: 604 588-3960

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-10

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration:	0.307 ± 0.044 g/t
Platinum concentration:	2.93 ± 0.20 g/t
Palladium concentration:	10.99 ± 0.73 g/t

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.

METHOD OF PREPARATION:

The ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining a quantity of the Stillwater ore (screened to –325) with a quantity of gold-bearing ore from the Misty Mountain Specogna deposit (screened to –200). The material was mixed for 6 days in a rotary mixer. Splits were sent to 11 laboratories for round robin assaying .

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	44.7		MgO	9.8
Al2O3	20.8		K2O	0.1
Fe2O3	7.4		TiO2	0.1
CaO	11.4		LOI	3.2
Na2O	1.4			

Statistical Procedures:

The mean and standard deviation for all data was calculated. Outliers were defined as samples beyond the mean ± 2 Standard Deviations from all data. These outliers were removed from the data and a new mean and standard deviation was determined. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Certified Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedure: 30g fire assay, AA or ICP finish.

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-10

	Lab. 1	Lab. 2	Lab. 3	Lab. 4	Lab. 5	Lab. 6	Lab. 7	Lab. 8	Lab. 9	Lab. 10	Lab. 11
	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
1	0.29	0.33	0.38	0.28	0.33	0.27	0.31	0.33	0.26	0.30	0.33
2	0.30	0.31	0.30	0.28	0.30	0.29	0.27	0.36	0.27	0.31	0.29
3	0.31	0.31	0.40	0.29	0.33	0.32	0.29	0.34	0.27	I.S.	0.32
4	0.40	0.30	0.33	0.27	0.31	0.32	0.30	0.32	0.25	0.32	0.30
5	0.32	0.34	0.29	0.32	0.29	0.28	0.29	0.31	0.30	0.32	0.27
6	0.34	0.34	0.36	0.30	0.31	0.32	0.31	0.30	0.28	I.S.	0.29
7	0.29	0.31	0.32	0.28	0.30	0.29	0.30	0.31	0.30	I.S.	0.34
8	0.30	0.30	0.36	0.29	0.29	0.31	0.32	0.35	0.31	I.S.	0.38
9	0.31	0.31	0.33	0.27	0.33	0.32	0.30	0.33	0.29	0.30	0.31
10	0.30	0.31	0.35	0.28	0.34	0.33	0.32	0.31	0.29	0.31	0.37
Mean	0.32	0.32	0.34	0.28	0.31	0.31	0.30	0.33	0.28	0.31	0.32
Std. Dev'n	0.0331	0.0151	0.0346	0.0134	0.0183	0.0207	0.0132	0.0196	0.0193	0.0089	0.0376
%RSD	10.46	4.76	10.11	4.71	5.84	6.78	4.41	6.00	6.85	2.89	11.78
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
1	2.70	3.00	2.95	2.95	2.92	2.97	2.97	2.88	2.88	2.88	3.01
2	2.71	2.90	2.97	2.98	2.84	2.96	2.88	2.84	2.94	2.88	2.82
3	2.72	2.96	2.93	3.00	2.87	3.01	2.91	2.87	3.04	3.00	3.01
4	2.78	2.96	2.98	3.06	2.88	3.15	3.03	2.85	2.87	3.00	2.95
5	2.76	3.11	2.83	2.90	2.72	3.02	2.90	2.79	2.99	3.05	2.83
6	2.67	2.92	3.13	3.05	2.88	3.10	2.82	2.97	2.91	2.97	2.89
7	2.84	3.16	2.97	2.95	2.92	3.07	2.43	2.92	3.01	2.78	2.91
8	2.83	2.95	2.84	2.94	2.88	2.86	2.98	2.92	3.07	2.84	2.87
9	2.70	3.08	2.95	3.10	2.74	2.98	3.01	2.93	2.87	3.01	2.95
10	2.71	2.94	3.02	3.00	2.83	2.97	2.93	2.97	3.14	3.01	2.87
Mean	2.74	3.00	2.96	2.99	2.85	3.01	2.89	2.89	2.97	2.94	2.91
Std. Dev'n	0.0581	0.0880	0.0853	0.0613	0.0686	0.0820	0.1725	0.0583	0.0933	0.0899	0.0677
%RSD	2.12	2.94	2.88	2.05	2.41	2.72	5.98	2.02	3.14	3.06	2.33
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
1	9.6	11.2	11.5	11.0	10.6	11.2	11.1	10.7	11.2	10.3	11.6
2	10.4	11.3	11.5	11.0	10.7	11.3	10.6	10.8	11.2	10.7	11.4
3	10.5	11.2	11.4	10.9	10.8	10.9	10.7	10.8	11.2	11.7	10.9
4	10.5	11.3	11.4	11.0	10.9	10.7	11.2	11.0	11.4	10.8	11.1
5	10.4	11.5	11.2	10.7	10.2	11.1	10.6	10.8	11.1	11.2	11.1
6	10.6	11.3	11.7	10.8	10.7	10.8	10.4	11.6	11.4	10.7	11.1
7	10.9	11.6	11.4	10.6	10.7	11.3	8.9	11.9	11.1	10.1	11.3
8	10.9	11.1	11.5	10.9	10.6	10.8	11.2	11.3	11.0	10.5	11.7
9	10.5	11.5	11.5	10.9	10.5	11.0	10.9	12.0	11.2	10.9	11.2
10	10.4	11.5	11.6	10.7	10.8	10.8	10.8	11.1	10.8	10.6	11.2
Mean	10.45	11.35	11.47	10.84	10.64	10.99	10.63	11.20	11.16	10.75	11.26
Std. Dev'n	0.3440	0.1260	0.1337	0.1273	0.1849	0.2234	0.6777	0.4807	0.1776	0.4528	0.2459
%RSD	3.29	1.11	1.17	1.17	1.74	2.03	6.37	4.29	1.59	4.21	2.18

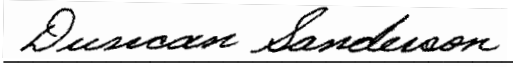
PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-10

Participating Laboratories:
(not in same order as listed in table of results)

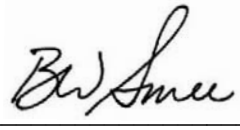
Acme Analytical Laboratories Ltd., Vancouver
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Genalysis Laboratory Services Pty Ltd., Perth
GTK Laboratory, Geological Survey of Finland
OMAC Laboratories Ltd., Ireland
SGS-Lakefield, Ontario
SGS-Welshpool, Australia
SGS-XRAL Labs, Toronto
Stillwater Mining Corporation, Montana
TSL Laboratories, Saskatoon

Legal Notice:

This certificate and the reference material described in it have been prepared with due care and attention. However CDN Resource Laboratories Ltd. or Barry Smee accept no liability for any decisions or actions taken following the use of the reference material. Our liability is limited solely to the cost of the reference material.

Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist 

Dr. Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

10945-B River Road, Delta, B.C., V4C 2R8, 604-540-2233, Fax: 604-540-2237

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-15

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration: 0.41 ± 0.07 g/t ***(Au is provisional value only, RSD = 8.96%)***
Platinum concentration: 0.098 ± 0.014 g/t
Palladium concentration: 0.428 ± 0.030 g/t

**** Note ****

Standards with an RSD of near or less than 5% are certified, RSD's of between 5% and 15% are Provisional, and RSD's over 15% are Indicated. Provisional and Indicated values cannot be used to monitor accuracy with a high degree of certainty.

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: October 25, 2008

METHOD OF PREPARATION:

The ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining a quantity of the Stillwater ore (screened to –325) with a quantity of gold-bearing ore from the Eleonore (Les Mines Opinaca) deposit (screened to –200) and diluting with a blank granitic material that had been screened to –200 mesh. The material was mixed for 6 days in a double-cone mixer. Splits were sent to 12 laboratories for round robin assaying .

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	64.2		MgO	2.7
Al2O3	14.0		K2O	1.1
Fe2O3	6.4		TiO2	0.6
CaO	4.8		LOI	2.0
Na2O	2.9		S	0.2

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedure: 30g fire assay, AA or ICP finish.

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-15

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12
SAMPLE	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS-15-1	0.561	0.44	0.35	0.432	0.38	0.354	0.43	0.377	0.429	0.45	0.45	0.372
PGMS-15-2	0.428	0.40	0.33	0.403	0.43	0.337	0.44	0.356	0.398	0.45	0.35	0.361
PGMS-15-3	0.487	0.45	0.42	0.401	0.39	0.388	0.40	0.474	0.359	0.44	0.49	0.368
PGMS-15-4	0.342	0.42	0.34	0.424	0.57	0.398	0.41	0.447	0.392	0.42	0.51	0.387
PGMS-15-5	0.410	0.43	0.33	0.367	0.33	0.422	0.37	0.433	0.398	0.38	0.49	0.365
PGMS-15-6	0.427	0.42	0.33	0.374	0.38	0.376	0.38	0.393	0.429	0.39	0.38	0.380
PGMS-15-7	0.391	0.41	0.36	0.377	0.41	0.440	0.44	0.411	0.398	0.45	0.39	0.356
PGMS-15-8	0.371	0.39	0.44	0.392	0.40	0.443	0.39	0.407	0.395	0.48	0.44	0.368
PGMS-15-9	0.429	0.41	0.36	0.373	0.37	0.380	0.37	0.428	0.439	0.48	0.36	0.369
PGMS-15-10	0.351	0.44	0.37	0.435	0.48	0.388	0.43	0.364	0.432	0.46	0.45	0.370
Mean	0.420	0.421	0.363	0.398	0.414	0.393	0.406	0.409	0.407	0.439	0.429	0.370
Std. Dev'n	0.0656	0.0191	0.0383	0.0255	0.0675	0.0345	0.0280	0.0375	0.0248	0.0324	0.0566	0.0088
%RSD	15.63	4.54	10.55	6.42	16.31	8.80	6.89	9.18	6.09	7.39	13.17	2.39
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS-15-1	0.098	0.11	0.08	0.109	0.10	0.099	0.11	0.111	0.097	0.09	0.09	0.080
PGMS-15-2	0.102	0.11	0.08	0.099	0.08	0.091	0.10	0.109	0.100	0.08	0.09	0.085
PGMS-15-3	0.103	0.09	0.07	0.094	0.09	0.103	0.10	0.109	0.098	0.10	0.09	0.079
PGMS-15-4	0.095	0.09	0.07	0.105	0.10	0.104	0.10	0.100	0.090	0.09	0.09	0.078
PGMS-15-5	0.102	0.10	0.07	0.087	0.10	0.094	0.10	0.103	0.099	0.09	0.09	0.083
PGMS-15-6	0.100	0.10	0.08	0.098	0.09	0.100	0.11	0.101	0.109	0.08	0.09	0.076
PGMS-15-7	0.096	0.10	0.08	0.089	0.08	0.090	0.10	0.103	0.092	0.10	0.09	0.077
PGMS-15-8	0.105	0.10	0.08	0.095	0.08	0.090	0.11	0.097	0.102	0.10	0.09	0.077
PGMS-15-9	0.104	0.11	0.07	0.095	0.09	0.099	0.11	0.096	0.103	0.08	0.09	0.079
PGMS-15-10	0.092	0.10	0.07	0.098	0.09	0.096	0.11	0.104	0.102	0.09	0.09	0.082
Mean	0.100	0.101	0.075	0.097	0.090	0.097	0.105	0.103	0.099	0.091	0.089	0.080
Std. Dev'n	0.0043	0.0074	0.0053	0.0066	0.0082	0.0052	0.0053	0.0052	0.0056	0.0077	0.0029	0.0029
%RSD	4.31	7.31	7.03	6.84	9.07	5.39	5.02	5.01	5.64	8.47	3.24	3.66
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS-15-1	0.418	0.43	0.32	0.411	0.44	0.428	0.42	0.427	0.441	0.45	0.44	0.356
PGMS-15-2	0.429	0.44	0.38	0.411	0.45	0.404	0.42	0.425	0.435	0.44	0.43	0.373
PGMS-15-3	0.429	0.43	0.32	0.400	0.46	0.427	0.43	0.440	0.435	0.44	0.44	0.363
PGMS-15-4	0.417	0.43	0.36	0.403	0.45	0.412	0.43	0.432	0.448	0.44	0.44	0.366
PGMS-15-5	0.426	0.42	0.34	0.383	0.43	0.412	0.43	0.330	0.428	0.44	0.44	0.361
PGMS-15-6	0.426	0.43	0.32	0.397	0.43	0.403	0.41	0.433	0.457	0.44	0.42	0.356
PGMS-15-7	0.413	0.44	0.34	0.397	0.42	0.409	0.42	0.426	0.426	0.45	0.43	0.363
PGMS-15-8	0.420	0.44	0.36	0.403	0.44	0.404	0.44	0.415	0.442	0.43	0.42	0.348
PGMS-15-9	0.428	0.41	0.33	0.410	0.44	0.406	0.39	0.405	0.441	0.45	0.45	0.358
PGMS-15-10	0.424	0.44	0.33	0.418	0.46	0.412	0.41	0.425	0.435	0.45	0.44	0.368
Mean	0.423	0.431	0.340	0.403	0.442	0.412	0.420	0.416	0.439	0.443	0.434	0.361
Std. Dev'n	0.0056	0.0099	0.0205	0.0099	0.0132	0.0090	0.0141	0.0316	0.0092	0.0062	0.0095	0.0071
%RSD	1.33	2.31	6.04	2.45	2.98	2.19	3.37	7.60	2.09	1.40	2.20	1.97

Note: Au data: all data from Lab 3 removed for failing the "t" test.

Pd, Pt data: all data from Lab 3 & Lab 12 removed for failing the "t" test.

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-15

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Actlabs, Ancaster, Ontario
ASA Argentina
Genalysis Laboratory Services Pty Ltd., Perth
International Plasma Labs. Ltd., Richmond, B.C.
Labtium Inc., Finland
OMAC Laboratories Ltd., Ireland
Skyline Laboratories Ltd., Arizona, USA
Ultra Trace Pty Ltd., Australia
TSL Laboratories, Saskatoon

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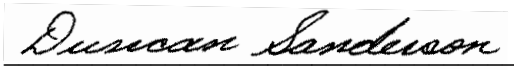
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

10945-B River Road, Delta, B.C., V4C 2R8, 604-540-2233, Fax: 604-540-2237

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-16

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration:	1.12 ± 0.15 g/t
Platinum concentration:	1.23 ± 0.09 g/t
Palladium concentration:	4.66 ± 0.36 g/t

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: October 25, 2008

METHOD OF PREPARATION:

The ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining a quantity of the Stillwater ore (screened to –325) with a quantity of gold-bearing ore from the Meliadine (Comaplex Minerals) deposit (screened to –270) and diluting with a blank granitic material that had been screened to –200 mesh. The material was mixed for 6 days in a double-cone mixer. Splits were sent to 12 laboratories for round robin assaying .

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	54.2		MgO	5.3
Al2O3	16.8		K2O	1.0
Fe2O3	8.1		TiO2	0.3
CaO	8.0		LOI	4.4
Na2O	1.6		S	0.6

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual “between-laboratory” standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedure: 30g fire assay, AA or ICP finish.

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-16

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12
SAMPLE	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS-16-1	1.17	1.14	1.10	1.03	1.06	0.93	1.12	1.04	1.21	1.17	1.18	1.24
PGMS-16-2	1.09	0.97	1.08	1.19	1.11	0.89	1.04	1.19	1.20	1.06	1.16	1.12
PGMS-16-3	0.94	1.20	1.12	1.05	1.16	1.17	1.02	1.15	1.13	1.04	1.33	1.18
PGMS-16-4	1.06	1.04	1.08	1.10	1.01	1.09	1.16	1.06	1.19	1.12	1.37	1.05
PGMS-16-5	1.12	1.02	1.17	1.17	1.00	1.01	1.14	1.02	1.13	1.08	1.41	0.95
PGMS-16-6	1.23	1.00	1.00	1.13	1.17	0.92	1.12	1.17	1.25	1.20	1.14	1.01
PGMS-16-7	1.20	1.05	1.12	1.16	1.13	1.08	1.17	1.12	1.21	1.09	1.05	1.03
PGMS-16-8	1.13	0.99	1.22	1.13	1.25	0.93	1.06	1.27	1.14	1.15	1.18	1.31
PGMS-16-9	1.25	1.12	1.16	1.20	1.11	1.09	1.08	1.17	1.16	0.99	1.34	1.24
PGMS-16-10	1.29	1.29	1.00	1.10	1.12	1.24	1.22	1.04	1.14	1.08	1.28	1.12
Mean	1.15	1.08	1.11	1.12	1.11	1.03	1.11	1.12	1.18	1.10	1.24	1.12
Std. Dev'n	0.1043	0.1035	0.0701	0.0592	0.0751	0.1190	0.0629	0.0814	0.0417	0.0636	0.1179	0.1167
%RSD	9.11	9.56	6.35	5.26	6.75	11.51	5.65	7.25	3.54	5.79	9.48	10.38
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS-16-1	1.23	1.26	1.09	1.21	1.21	1.18	1.30	1.23	1.29	1.27	1.22	1.17
PGMS-16-2	1.22	1.19	1.16	1.26	1.22	1.22	1.16	1.26	1.31	1.23	1.21	1.13
PGMS-16-3	1.15	1.26	1.04	1.25	1.16	1.20	1.18	1.24	1.30	1.22	1.19	1.19
PGMS-16-4	1.31	1.30	1.13	1.26	1.20	1.24	1.24	1.25	1.31	1.19	1.20	1.17
PGMS-16-5	1.23	1.23	1.09	1.28	1.20	1.21	1.24	1.20	1.28	1.22	1.24	1.07
PGMS-16-6	1.29	1.19	1.16	1.18	1.27	1.16	1.26	1.29	1.32	1.22	1.26	1.09
PGMS-16-7	1.18	1.25	1.09	1.30	1.27	1.18	1.26	1.33	1.30	1.21	1.24	1.10
PGMS-16-8	1.21	1.22	1.10	1.30	1.25	1.17	1.28	1.23	1.27	1.26	1.20	1.17
PGMS-16-9	1.20	1.28	1.16	1.28	1.17	1.18	1.24	1.23	1.27	1.28	1.23	1.16
PGMS-16-10	1.21	1.29	1.09	1.25	1.22	1.20	1.18	1.21	1.23	1.20	1.21	1.13
Mean	1.22	1.25	1.11	1.26	1.22	1.19	1.23	1.25	1.29	1.23	1.22	1.14
Std. Dev'n	0.0473	0.0389	0.0401	0.0377	0.0377	0.0243	0.0462	0.0386	0.0266	0.0301	0.0221	0.0405
%RSD	3.88	3.12	3.61	3.00	3.10	2.04	3.75	3.10	2.06	2.45	1.81	3.56
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS-16-1	4.63	4.57	4.36	4.65	4.44	4.61	4.79	4.81	4.97	4.87	4.50	4.59
PGMS-16-2	4.69	4.14	4.33	4.90	4.45	4.75	4.61	4.73	4.94	4.77	4.45	4.41
PGMS-16-3	4.31	4.50	4.47	4.84	4.40	4.74	4.75	4.64	4.81	4.78	4.50	4.55
PGMS-16-4	4.63	4.26	4.38	5.16	4.52	4.78	4.63	4.72	5.00	4.90	4.47	4.46
PGMS-16-5	4.59	4.65	4.03	4.90	4.37	4.81	4.87	4.87	4.73	4.84	4.48	4.24
PGMS-16-6	4.49	4.46	4.28	4.66	4.55	4.67	4.75	4.76	5.00	4.76	4.44	4.23
PGMS-16-7	4.53	4.68	4.01	5.06	4.50	4.79	4.59	4.80	4.92	4.69	4.55	4.45
PGMS-16-8	4.51	4.24	3.99	4.93	4.61	4.77	4.71	4.80	4.76	4.84	4.46	4.45
PGMS-16-9	4.63	4.68	4.19	4.98	4.42	4.81	4.81	4.78	4.84	4.78	4.45	4.49
PGMS-16-10	4.49	4.33	4.16	4.89	4.44	4.79	4.63	4.73	4.99	4.78	4.47	4.37
Mean	4.55	4.45	4.22	4.90	4.47	4.75	4.71	4.76	4.90	4.80	4.48	4.42
Std. Dev'n	0.1091	0.1982	0.1703	0.1571	0.0738	0.0658	0.0956	0.0631	0.1028	0.0621	0.0327	0.1179
%RSD	2.40	4.45	4.04	3.21	1.65	1.38	2.03	1.32	2.10	1.29	0.73	2.66

Note: Au data: all data from Lab 6 & Lab 11 removed for failing the "t" test.

Pd & Pt data: all data from Lab 3 removed for failing the "t" test.

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-16

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Actlabs, Ancaster, Ontario
ASA Argentina
Genalysis Laboratory Services Pty Ltd., Perth
International Plasma Labs. Ltd., Richmond, B.C.
Labtium Inc., Finland
OMAC Laboratories Ltd., Ireland
Skyline Laboratories Ltd., Arizona, USA
Ultra Trace Pty Ltd., Australia
TSL Laboratories, Saskatoon

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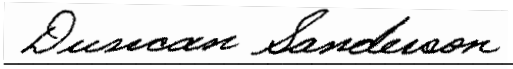
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466 (www.cdnlabs.com)

REFERENCE MATERIAL: CDN-PGMS-17

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration: 0.927 ± 0.082 g/t
Platinum concentration: 0.998 ± 0.056 g/t
Palladium concentration: 4.30 ± 0.17 g/t

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: November 4, 2009

METHOD OF PREPARATION:

The ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining 700kg of the Stillwater ore (screened to –325) with 10 kg of a gold concentrate (screened to –270). The material was mixed for 5 days in a double-cone mixer. Splits were sent to 14 laboratories for round robin assaying .

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	43.3		MgO	6.1
Al2O3	25.2		K2O	0.2
Fe2O3	5.5		TiO2	0.1
CaO	15.5		LOI	2.1
Na2O	1.4		S	0.6

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedure: 30g fire assay, AA or ICP finish.

REFERENCE MATERIAL: CDN-PGMS-17

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12	Lab 13	Lab 14
	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS17-1	0.890	0.98	0.89	0.914	0.90	0.930	0.92	0.971	0.944	1.050	0.927	0.914	0.932	0.984
PGMS17-2	0.957	0.94	0.88	0.876	0.92	0.935	0.97	0.957	0.923	0.976	0.909	0.835	0.924	0.895
PGMS17-3	0.928	0.87	0.89	0.921	0.96	0.925	0.87	0.948	0.905	0.815	0.972	0.906	0.970	0.878
PGMS17-4	0.815	1.00	0.82	1.014	0.92	0.965	0.94	1.089	0.942	0.826	0.969	0.940	0.957	0.995
PGMS17-5	1.000	1.00	0.76	0.976	0.92	0.882	0.90	0.981	0.965	0.924	0.926	0.820	0.895	0.963
PGMS17-6	0.934	0.93	0.93	0.911	0.88	0.927	0.88	0.878	0.926	0.937	0.997	0.926	0.951	0.908
PGMS17-7	0.895	0.96	0.89	0.891	0.97	0.947	0.93	0.961	0.950	0.894	1.100	0.976	0.984	0.904
PGMS17-8	0.887	0.95	0.90	0.902	0.91	0.937	0.94	0.994	0.888	0.936	0.955	0.949	0.933	0.897
PGMS17-9	0.942	0.86	0.83	0.976	0.90	0.921	0.86	0.892	0.913	0.951	0.903	0.862	0.948	0.914
PGMS17-10	0.886	0.86	0.85	0.936	0.95	0.967	0.96	0.905	0.881	0.888	0.885	0.967	0.923	0.792
Mean	0.913	0.935	0.864	0.932	0.923	0.934	0.917	0.958	0.924	0.920	0.954	0.910	0.942	0.913
Std. Dev'n	0.0504	0.0546	0.0495	0.0437	0.0287	0.0241	0.0380	0.0604	0.0273	0.0692	0.0620	0.0541	0.0257	0.06
%RSD	5.52	5.84	5.73	4.69	3.11	2.59	4.15	6.31	2.96	7.53	6.50	5.95	2.73	6.41
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS17-1	0.966	1.02	1.05	1.033	0.99	0.97	1.00	1.002	0.97	1.04	1.00	1.060	0.999	0.813
PGMS17-2	0.965	1.00	1.05	0.967	1.00	1.03	0.97	1.039	1.01	1.03	1.02	0.941	0.995	0.809
PGMS17-3	0.980	0.98	1.03	0.983	1.05	1.03	0.95	1.007	0.98	1.00	0.98	0.989	0.987	0.830
PGMS17-4	0.941	1.01	1.01	0.996	0.99	1.08	1.01	1.000	1.00	1.00	0.99	1.010	1.030	0.830
PGMS17-5	0.949	0.99	1.06	0.997	0.99	1.07	0.98	1.027	1.01	1.03	0.95	0.975	0.997	0.804
PGMS17-6	0.956	1.00	1.10	1.007	1.12	1.04	0.97	1.031	1.00	1.01	1.01	1.050	1.010	0.776
PGMS17-7	0.957	1.02	1.04	0.991	1.01	1.05	0.98	1.059	0.96	1.06	0.99	0.994	1.030	0.808
PGMS17-8	0.946	0.99	1.11	0.975	1.01	0.98	0.96	0.996	1.01	1.05	1.00	0.986	1.010	0.790
PGMS17-9	1.000	0.95	1.04	0.982	1.05	1.00	0.99	0.998	0.96	1.04	1.00	1.010	1.040	0.797
PGMS17-10	0.972	1.00	1.03	1.005	0.97	1.04	0.99	0.995	0.94	1.05	0.99	0.979	0.968	0.779
Mean	0.963	0.996	1.052	0.994	1.018	1.029	0.980	1.015	0.984	1.031	0.992	0.999	1.007	0.804
Std. Dev'n	0.0177	0.0207	0.0312	0.0188	0.0442	0.0360	0.0183	0.0222	0.0255	0.0213	0.0177	0.0353	0.0221	0.0186
%RSD	1.83	2.07	2.97	1.89	4.34	3.50	1.86	2.18	2.59	2.07	1.78	3.53	2.19	2.32
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS17-1	4.22	4.25	4.76	4.32	4.23	4.13	4.32	4.47	4.44	4.04	4.25	4.33	4.27	3.70
PGMS17-2	4.20	4.38	4.72	4.23	4.25	4.35	4.28	4.52	4.34	4.34	4.30	4.05	4.33	3.67
PGMS17-3	4.28	4.45	4.64	4.36	4.42	4.21	4.20	4.47	4.21	4.22	4.30	4.24	4.33	3.75
PGMS17-4	4.22	4.20	4.60	4.32	4.35	4.31	4.26	4.46	4.25	4.19	4.26	4.24	4.45	3.68
PGMS17-5	4.19	4.52	4.36	4.35	4.28	4.14	4.38	4.42	4.31	4.17	4.28	4.30	4.35	3.62
PGMS17-6	4.21	4.20	4.84	4.38	4.71	4.23	4.33	4.38	4.26	4.18	4.31	4.36	4.37	3.52
PGMS17-7	4.30	4.20	4.84	4.31	4.27	4.29	4.38	4.45	4.27	4.30	4.25	4.28	4.38	3.48
PGMS17-8	4.21	4.24	4.72	4.29	4.37	4.14	4.31	4.39	4.31	4.56	4.30	4.26	4.36	3.40
PGMS17-9	4.21	4.44	4.76	4.31	4.35	4.40	4.25	4.32	4.24	4.33	4.23	4.18	4.49	3.51
PGMS17-10	4.21	4.34	4.72	4.44	4.21	4.34	4.32	4.36	4.21	4.37	4.17	4.29	4.26	3.38
Mean	4.23	4.32	4.70	4.33	4.34	4.25	4.30	4.42	4.28	4.27	4.27	4.25	4.36	3.57
Std. Dev'n	0.0357	0.1201	0.1401	0.0569	0.1452	0.0979	0.0568	0.0610	0.0696	0.1428	0.0430	0.0873	0.0711	0.1303
%RSD	0.84	2.78	2.98	1.31	3.34	2.30	1.32	1.38	1.63	3.34	1.01	2.05	1.63	3.65

Note: Pt & Pd: data from Labs 3 and 14 removed for failing the "t" test.

REFERENCE MATERIAL: CDN-PGMS-17

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver, B.C., Canada
Activation Laboratories Ltd., Ancaster, Ontario, Canada
Activation Laboratories Ltd., Thunder Bay, Ontario, Canada
Assayers Canada Ltd., Vancouver, B.C., Canada
ALS Chemex Laboratories, North Vancouver, B.C., Canada
EcoTech, Kamloops, B.C., Canada
SGS Toronto, Ontario, Canada
Genalysis Laboratory Services Pty. Ltd., Perth, Australia
Inspectorate America Assay Labs, Nevada, USA
Labtium, Finland
OMAC Laboratories Ltd., Ireland
Skyline Assayers & Laboratories, Arizona, USA
TSL Laboratories, Saskatoon, SK, Canada
Ultra Trace Analytical Laboratories, Perth, Australia

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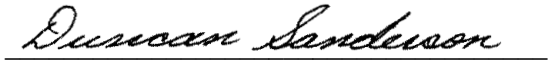
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466 (www.cdnlabs.com)

REFERENCE MATERIAL: CDN-PGMS-18

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration: 0.517 ± 0.060 g/t
Platinum concentration: 0.329 ± 0.038 g/t
Palladium concentration: 1.42 ± 0.11 g/t

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: November 13, 2009

METHOD OF PREPARATION:

The standard was made by combining two ores: one supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana and the other supplied by Pacific Sentinel from their Casino property in the Yukon.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.
Copper-gold-molybdenum mineralization in the Casino ore is genetically related to a breccia and microbreccia pipe of fine grained quartz monzonites, intrusion breccias, and plagioclase-porphyritic intrusions that may be subvolcanic in origin, comprising part of the 72-74 Ma Casino Intrusive Complex. Roughly centred on the microbreccia pipe, both the alteration and mineralization are zoned. Innermost is the potassic alteration suite consisting of K-feldspar, biotite, magnetite, anhydrite, gypsum, and pyrite, chalcopyrite, molybdenite, and gold.

This standard was prepared by combining 180kg of the Stillwater ore (screened to –325) with 360 kg of the Casino ore (screened to –270). The material was mixed for 5 days in a double-cone mixer. Splits were sent to 15 laboratories for round robin assaying.

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	51.5		MgO	3.8
Al2O3	17.1		K2O	2.1
Fe2O3	7.1		TiO2	0.4
CaO	8.5		LOI	5.7
Na2O	1.7		S	1.5

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual “between-laboratory” standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

REFERENCE MATERIAL: CDN-PGMS-18

Assay Procedure: 30g fire assay, AA or ICP finish.

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12	Lab 13	Lab 14	Lab 15
	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS18-1	0.517	0.53	0.54	0.453	0.44	0.508	0.49	0.474	0.528	0.54	0.51	0.544	0.542	0.457	0.62
PGMS18-2	0.513	0.56	0.50	0.479	0.50	0.531	0.49	0.501	0.568	0.54	0.56	0.547	0.552	0.475	0.52
PGMS18-3	0.523	0.53	0.49	0.518	0.42	0.437	0.55	0.550	0.523	0.49	0.55	0.510	0.539	0.502	0.52
PGMS18-4	0.607	0.56	0.51	0.503	0.52	0.462	0.54	0.467	0.541	0.49	0.50	0.522	0.487	0.497	0.52
PGMS18-5	0.536	0.58	0.54	0.471	0.48	0.502	0.48	0.516	0.528	0.53	0.53	0.587	0.546	0.463	0.50
PGMS18-6	0.517	0.54	0.56	0.512	0.49	0.494	0.51	0.554	0.586	0.52	0.52	0.618	0.462	0.486	0.50
PGMS18-7	0.526	0.58	0.49	0.496	0.53	0.504	0.53	0.543	0.536	0.51	0.52	0.524	0.486	0.460	0.50
PGMS18-8	0.543	0.58	0.48	0.499	0.41	0.547	0.47	0.484	0.524	0.51	0.53	0.515	0.556	0.557	0.50
PGMS18-9	0.491	0.53	0.54	0.523	0.52	0.500	0.48	0.537	0.503	0.48	0.61	0.567	0.532	0.461	0.50
PGMS18-10	0.594	0.53	0.52	0.515	0.44	0.500	0.54	0.509	0.532	0.48	0.55	0.591	0.514	0.482	0.56
Mean	0.537	0.552	0.517	0.497	0.475	0.499	0.508	0.513	0.537	0.509	0.537	0.553	0.522	0.484	0.524
Std. Dev'n	0.0365	0.0225	0.0271	0.0227	0.0443	0.0311	0.0297	0.0321	0.0238	0.0237	0.0307	0.0369	0.0327	0.03	0.04
%RSD	6.80	4.08	5.24	4.57	9.32	6.23	5.85	6.24	4.43	4.66	5.72	6.67	6.26	6.23	7.37
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS18-1	0.342	0.30	0.32	0.308	0.28	0.317	0.36	0.338	0.322	0.38	0.33	0.365	0.33	0.294	0.30
PGMS18-2	0.334	0.30	0.32	0.342	0.35	0.329	0.33	0.333	0.290	0.33	0.34	0.367	0.35	0.306	0.30
PGMS18-3	0.348	0.31	0.33	0.324	0.33	0.299	0.33	0.334	0.299	0.32	0.36	0.362	0.34	0.283	0.30
PGMS18-4	0.344	0.32	0.37	0.332	0.37	0.325	0.36	0.340	0.316	0.32	0.34	0.375	0.32	0.313	0.33
PGMS18-5	0.352	0.31	0.33	0.339	0.31	0.309	0.34	0.359	0.315	0.36	0.34	0.360	0.34	0.279	0.29
PGMS18-6	0.343	0.32	0.35	0.333	0.34	0.305	0.32	0.337	0.304	0.36	0.34	0.342	0.32	0.294	0.31
PGMS18-7	0.363	0.31	0.34	0.338	0.35	0.315	0.32	0.324	0.326	0.34	0.35	0.356	0.33	0.311	0.28
PGMS18-8	0.350	0.34	0.33	0.338	0.27	0.318	0.33	0.334	0.325	0.35	0.34	0.333	0.31	0.316	0.27
PGMS18-9	0.352	0.32	0.34	0.343	0.34	0.302	0.32	0.338	0.295	0.32	0.33	0.372	0.32	0.283	0.29
PGMS18-10	0.332	0.32	0.34	0.352	0.30	0.328	0.34	0.325	0.330	0.31	0.32	0.340	0.33	0.306	0.30
Mean	0.346	0.315	0.336	0.335	0.324	0.315	0.335	0.336	0.312	0.339	0.339	0.357	0.327	0.299	0.297
Std. Dev'n	0.0091	0.0118	0.0141	0.0120	0.0327	0.0107	0.0151	0.0096	0.0142	0.0228	0.0114	0.0143	0.0107	0.0137	0.0164
%RSD	2.64	3.74	4.19	3.59	10.10	3.41	4.51	2.86	4.55	6.73	3.37	4.00	3.29	4.59	5.51
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS18-1	1.48	1.41	1.48	1.30	1.25	1.43	1.47	1.38	1.36	1.41	1.40	1.56	1.45	1.39	1.31
PGMS18-2	1.44	1.30	1.50	1.37	1.47	1.41	1.44	1.47	1.34	1.41	1.46	1.59	1.49	1.45	1.34
PGMS18-3	1.49	1.32	1.48	1.39	1.46	1.38	1.44	1.42	1.39	1.38	1.46	1.54	1.46	1.35	1.34
PGMS18-4	1.46	1.30	1.54	1.41	1.50	1.40	1.46	1.45	1.37	1.41	1.46	1.56	1.45	1.43	1.33
PGMS18-5	1.45	1.34	1.48	1.44	1.41	1.42	1.45	1.40	1.33	1.48	1.46	1.56	1.50	1.40	1.32
PGMS18-6	1.46	1.33	1.46	1.44	1.50	1.40	1.43	1.43	1.39	1.43	1.44	1.54	1.41	1.42	1.35
PGMS18-7	1.47	1.24	1.48	1.43	1.48	1.45	1.45	1.41	1.38	1.40	1.46	1.52	1.40	1.38	1.30
PGMS18-8	1.48	1.34	1.48	1.47	1.24	1.41	1.43	1.44	1.37	1.44	1.48	1.55	1.40	1.45	1.29
PGMS18-9	1.46	1.27	1.50	1.49	1.41	1.40	1.42	1.46	1.28	1.42	1.45	1.57	1.45	1.41	1.31
PGMS18-10	1.44	1.24	1.52	1.46	1.30	1.41	1.46	1.39	1.34	1.36	1.44	1.54	1.46	1.40	1.34
Mean	1.46	1.31	1.49	1.42	1.40	1.41	1.45	1.42	1.36	1.41	1.45	1.55	1.45	1.41	1.32
Std. Dev'n	0.0165	0.0515	0.0235	0.0554	0.1017	0.0194	0.0158	0.0297	0.0337	0.0327	0.0213	0.0195	0.0347	0.0306	0.0200
%RSD	1.13	3.94	1.57	3.90	7.26	1.37	1.09	2.08	2.49	2.31	1.47	1.25	2.40	2.17	1.51

Note: Pd: data from Lab 12 removed for failing the "t" test.

REFERENCE MATERIAL: CDN-PGMS-18

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver, B.C., Canada
Activation Laboratories Ltd., Ancaster, Ontario, Canada
Activation Laboratories Ltd., Thunder Bay, Ontario, Canada
Assayers Canada Ltd., Vancouver, B.C., Canada
ALS Chemex Laboratories, North Vancouver, B.C., Canada
EcoTech, Kamloops, B.C., Canada
International Plasma Labs. Ltd., Richmond, B.C., Canada
SGS Toronto, Ontario, Canada
Genalysis Laboratory Services Pty. Ltd., Perth, Australia
Inspectorate America Assay Labs, Nevada, USA
Labtium, Finland
OMAC Laboratories Ltd., Ireland
Skyline Assayers & Laboratories, Arizona, USA
TSL Laboratories, Saskatoon, SK, Canada
Ultra Trace Analytical Laboratories, Perth, Australia

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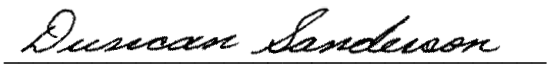
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466 (www.cdnlabs.com)

REFERENCE MATERIAL: CDN-PGMS-19

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration:	0.230 ± 0.030 g/t
Platinum concentration:	0.108 ± 0.012 g/t
Palladium concentration:	0.476 ± 0.042 g/t

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: November 19, 2009

METHOD OF PREPARATION:

The Pt/Pd ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining 75kg of the Stillwater ore (screened to –325) with 525 kg of a low grade Au ore (screened to –270). The material was mixed for 5 days in a double-cone mixer. Splits were sent to 15 laboratories for round robin assaying .

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	54.4		MgO	3.0
Al2O3	14.7		K2O	2.2
Fe2O3	7.1		TiO2	0.5
CaO	7.6		LOI	6.5
Na2O	2.2		S	1.4

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedure: 30g fire assay, AA or ICP finish.

REFERENCE MATERIAL: CDN-PGMS-19

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12	Lab 13	Lab 14	Lab 15
SAMPLE	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS-19-1	0.246	0.26	0.242	0.230	0.234	0.205	0.22	0.221	0.223	0.204	0.225	0.199	0.250	0.239	0.190
PGMS-19-2	0.229	0.24	0.240	0.256	0.264	0.216	0.24	0.239	0.218	0.228	0.218	0.245	0.251	0.238	0.216
PGMS-19-3	0.230	0.25	0.260	0.254	0.230	0.222	0.24	0.196	0.220	0.233	0.238	0.225	0.247	0.218	0.192
PGMS-19-4	0.229	0.25	0.242	0.217	0.225	0.255	0.22	0.232	0.238	0.214	0.265	0.190	0.253	0.252	0.212
PGMS-19-5	0.234	0.24	0.250	0.210	0.230	0.249	0.23	0.219	0.263	0.223	0.230	0.205	0.235	0.253	0.188
PGMS-19-6	0.258	0.21	0.238	0.243	0.229	0.215	0.22	0.200	0.236	0.255	0.207	0.232	0.229	0.236	0.206
PGMS-19-7	0.219	0.22	0.223	0.212	0.221	0.214	0.22	0.239	0.227	0.226	0.268	0.258	0.227	0.236	0.184
PGMS-19-8	0.209	0.22	0.235	0.253	0.231	0.211	0.23	0.214	0.214	0.248	0.242	0.198	0.225	0.234	0.193
PGMS-19-9	0.225	0.24	0.270	0.206	0.254	0.227	0.24	0.211	0.224	0.221	0.225	0.216	0.220	0.242	0.188
PGMS-19-10	0.275	0.24	0.248	0.207	0.233	0.209	0.22	0.218	0.258	0.218	0.229	0.233	0.211	0.242	0.222
Mean	0.235	0.237	0.245	0.229	0.235	0.222	0.228	0.219	0.232	0.227	0.235	0.220	0.235	0.239	0.199
Std. Dev'n	0.0194	0.0157	0.0132	0.0209	0.0134	0.0169	0.0092	0.0147	0.0167	0.0153	0.0194	0.0223	0.0147	0.01	0.01
%RSD	8.23	6.61	5.38	9.14	5.68	7.60	4.03	6.73	7.21	6.72	8.26	10.13	6.27	4.11	6.84
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS-19-1	0.103	0.12	0.113	0.106	0.117	0.108	0.10	0.106	0.117	0.10	0.104	0.111	0.109	0.107	0.10
PGMS-19-2	0.110	0.13	0.106	0.110	0.127	0.105	0.12	0.108	0.106	0.11	0.102	0.112	0.103	0.104	0.09
PGMS-19-3	0.112	0.11	0.100	0.115	0.112	0.118	0.10	0.114	0.111	0.12	0.110	0.106	0.103	0.101	0.09
PGMS-19-4	0.107	0.12	0.107	0.101	0.095	0.112	0.11	0.112	0.115	0.11	0.115	0.114	0.112	0.105	0.10
PGMS-19-5	0.112	0.11	0.111	0.101	0.118	0.111	0.10	0.108	0.119	0.11	0.100	0.108	0.098	0.106	0.08
PGMS-19-6	0.102	0.11	0.120	0.099	0.110	0.112	0.10	0.101	0.114	0.11	0.103	0.113	0.117	0.109	0.09
PGMS-19-7	0.107	0.14	0.107	0.098	0.104	0.104	0.10	0.111	0.103	0.11	0.110	0.117	0.117	0.106	0.09
PGMS-19-8	0.108	0.12	0.113	0.102	0.113	0.106	0.12	0.108	0.105	0.10	0.105	0.103	0.114	0.095	0.09
PGMS-19-9	0.105	0.11	0.114	0.108	0.112	0.105	0.12	0.109	0.106	0.12	0.101	0.113	0.103	0.108	0.10
PGMS-19-10	0.108	0.12	0.117	0.094	0.117	0.107	0.10	0.104	0.108	0.11	0.099	0.109	0.107	0.102	0.09
Mean	0.107	0.119	0.111	0.103	0.113	0.109	0.107	0.108	0.110	0.110	0.105	0.111	0.108	0.104	0.092
Std. Dev'n	0.0034	0.0099	0.0059	0.0063	0.0086	0.0044	0.0095	0.0038	0.0056	0.0067	0.0052	0.0041	0.0066	0.0041	0.0063
%RSD	3.17	8.36	5.31	6.09	7.65	4.04	8.87	3.53	5.06	6.06	4.96	3.74	6.08	3.94	6.87
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS-19-1	0.469	0.47	0.480	0.475	0.540	0.485	0.46	0.478	0.441	0.438	0.481	0.507	0.473	0.513	0.43
PGMS-19-2	0.473	0.44	0.495	0.489	0.541	0.472	0.47	0.477	0.452	0.450	0.478	0.508	0.481	0.522	0.45
PGMS-19-3	0.468	0.42	0.491	0.491	0.481	0.485	0.50	0.481	0.426	0.452	0.466	0.505	0.467	0.508	0.44
PGMS-19-4	0.468	0.43	0.485	0.470	0.445	0.478	0.48	0.492	0.458	0.454	0.502	0.524	0.498	0.505	0.47
PGMS-19-5	0.480	0.45	0.485	0.464	0.484	0.488	0.45	0.486	0.451	0.461	0.481	0.483	0.472	0.506	0.44
PGMS-19-6	0.469	0.42	0.483	0.467	0.482	0.472	0.46	0.466	0.432	0.451	0.475	0.519	0.504	0.528	0.46
PGMS-19-7	0.453	0.42	0.490	0.459	0.487	0.471	0.47	0.479	0.477	0.465	0.475	0.517	0.496	0.515	0.43
PGMS-19-8	0.471	0.43	0.478	0.450	0.492	0.467	0.49	0.483	0.469	0.459	0.470	0.485	0.487	0.514	0.42
PGMS-19-9	0.467	0.46	0.488	0.457	0.491	0.460	0.48	0.471	0.461	0.447	0.475	0.505	0.474	0.514	0.42
PGMS-19-10	0.463	0.45	0.497	0.452	0.493	0.467	0.46	0.479	0.424	0.454	0.480	0.512	0.476	0.514	0.45
Mean	0.468	0.439	0.487	0.467	0.494	0.475	0.472	0.479	0.449	0.453	0.478	0.507	0.483	0.514	0.441
Std. Dev'n	0.0069	0.0179	0.0062	0.0142	0.0283	0.0092	0.0155	0.0073	0.0180	0.0076	0.0096	0.0134	0.0127	0.0070	0.0166
%RSD	1.48	4.08	1.27	3.04	5.73	1.94	3.28	1.52	4.02	1.68	2.01	2.65	2.64	1.37	3.77

Note:
1) **Au data from Labs 3 and 15 removed for failing the "t" test.**
2) **Pt data from Labs 2 and 15 removed for failing the "t" test.**
3) **Pd data from Lab 2 removed for failing the "t" test.**

REFERENCE MATERIAL: CDN-PGMS-19

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver, B.C., Canada
Activation Laboratories Ltd., Ancaster, Ontario, Canada
Activation Laboratories Ltd., Thunder Bay, Ontario, Canada
Assayers Canada Ltd., Vancouver, B.C., Canada
ALS Chemex Laboratories, North Vancouver, B.C., Canada
EcoTech, Kamloops, B.C., Canada
International Plasma Labs, Richmond, B.C., Canada
SGS Toronto, Ontario, Canada
Genalysis Laboratory Services Pty. Ltd., Perth, Australia
Inspectorate America Assay Labs, Nevada, USA
Labtium, Finland
OMAC Laboratories Ltd., Ireland
Skyline Assayers & Laboratories, Arizona, USA
TSL Laboratories, Saskatoon, SK, Canada
Ultra Trace Analytical Laboratories, Perth, Australia

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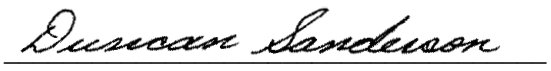
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466 (www.cdnlabs.com)

REFERENCE MATERIAL: CDN-PGMS-20

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration: 1.13 ± 0.11 g/t
Platinum concentration: 0.74 ± 0.05 g/t
Palladium concentration: 3.32 ± 0.21 g/t

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: May 2, 2011

METHOD OF PREPARATION:

The ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining 240kg of the Stillwater ore (screened to –325) with 1 kg of a gold concentrate (screened to –325). The material was mixed for 5 days in a double-cone mixer. Splits were sent to 15 laboratories for round robin assaying .

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	46.7		MgO	6.1
Al2O3	24.0		K2O	0.1
Fe2O3	6.6		TiO2	0.1
CaO	13.7		LOI	1.5
Na2O	1.4		S	0.1

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedure: 30g fire assay, AA or ICP finish.

REFERENCE MATERIAL: CDN-PGMS-20

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12	Lab 13	Lab 14	Lab 15
SAMPLE	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS-20-1	1.18	1.11	1.15	1.17	1.12	1.17	1.17	1.27	1.11	1.04	1.10	1.14	1.14	1.18	1.10
PGMS-20-2	1.06	1.13	1.03	1.18	1.08	1.19	1.14	1.15	1.12	1.12	1.17	1.19	1.19	1.18	1.08
PGMS-20-3	1.17	.	1.06	1.15	1.11	1.10	1.26	1.22	1.07	1.06	1.16	1.17	1.14	1.20	1.11
PGMS-20-4	1.12	1.17	1.14	1.22	1.13	1.10	1.19	1.24	1.03	1.09	1.27	1.13	1.04	1.22	1.08
PGMS-20-5	1.10	1.12	1.15	1.18	1.10	1.15	1.14	1.22	1.03	1.04	1.30	1.12	1.15	1.17	1.10
PGMS-20-6	1.05	1.10	1.19	1.25	1.14	1.10	1.16	1.16	1.07	1.03	1.18	1.18	1.14	1.19	1.08
PGMS-20-7	1.14	1.01	1.11	1.14	1.06	1.12	1.11	1.19	1.06	1.14	1.19	1.19	1.16	1.22	1.08
PGMS-20-8	1.19	0.97	1.07	1.17	1.06	1.20	1.15	1.18	1.02	1.05	1.24	1.16	1.18	1.17	1.11
PGMS-20-9	1.07	1.07	1.08	1.14	1.07	1.10	1.19	1.21	1.03	1.17	1.13	1.19	1.18	1.18	1.13
PGMS-20-10	1.14	1.15	1.04	1.18	1.04	1.20	1.17	1.19	1.02	1.12	1.31	1.18	1.15	1.23	1.10
Mean	1.12	1.09	1.10	1.18	1.09	1.14	1.17	1.20	1.05	1.09	1.21	1.17	1.15	1.20	1.10
Std. Dev'n	0.0518	0.0658	0.0539	0.0344	0.0346	0.0428	0.0405	0.0373	0.0353	0.0490	0.0717	0.0264	0.0419	0.02	0.02
%RSD	4.63	6.03	4.89	2.92	3.17	3.74	3.47	3.10	3.35	4.52	5.95	2.26	3.65	1.88	1.55
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS-20-1	0.740	0.74	0.650	0.784	0.769	0.761	0.74	0.730	0.936	0.87	0.73	0.754	0.713	0.73	0.755
PGMS-20-2	0.731	0.77	0.653	0.819	0.711	0.753	0.77	0.724	0.990	0.89	0.78	0.725	0.702	0.73	0.783
PGMS-20-3	0.736	.	0.662	0.801	0.764	0.731	0.79	0.754	0.984	0.86	0.79	0.743	0.732	0.75	0.78
PGMS-20-4	0.735	0.71	0.641	0.819	0.747	0.680	0.76	0.764	1.020	0.83	0.76	0.722	0.726	0.72	0.741
PGMS-20-5	0.737	0.73	0.646	0.784	0.738	0.764	0.76	0.743	0.941	0.89	0.76	0.726	0.735	0.73	0.771
PGMS-20-6	0.739	0.70	0.675	0.825	0.735	0.705	0.77	0.719	0.941	0.86	0.82	0.706	0.691	0.73	0.786
PGMS-20-7	0.737	0.71	0.661	0.802	0.710	0.715	0.75	0.726	0.986	0.90	0.79	0.737	0.731	0.73	0.754
PGMS-20-8	0.765	0.71	0.648	0.814	0.786	0.747	0.78	0.732	0.966	0.93	0.76	0.736	0.721	0.73	0.737
PGMS-20-9	0.770	0.72	0.643	0.797	0.694	0.648	0.77	0.730	0.974	0.92	0.76	0.740	0.750	0.73	0.787
PGMS-20-10	0.748	0.71	0.660	0.835	0.740	0.735	0.76	0.744	1.007	0.89	0.78	0.734	0.711	0.73	0.746
Mean	0.74	0.72	0.65	0.81	0.74	0.72	0.77	0.74	0.97	0.88	0.77	0.73	0.72	0.73	0.76
Std. Dev'n	0.0133	0.0217	0.0105	0.0172	0.0288	0.0375	0.0143	0.0143	0.0287	0.0299	0.0226	0.0132	0.0173	0.0081	0.0196
%RSD	1.78	3.00	1.61	2.13	3.89	5.18	1.87	1.94	2.94	3.38	2.92	1.81	2.41	1.11	2.56
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS-20-1	3.27	3.12	3.36	3.44	3.25	3.28	3.20	3.41	3.97	3.30	3.28	3.21	3.17	3.46	3.39
PGMS-20-2	3.23	3.41	3.28	3.59	3.24	3.33	3.29	3.29	4.07	3.48	3.33	3.29	3.29	3.44	3.37
PGMS-20-3	3.19	.	3.28	3.49	3.31	3.11	3.33	3.36	4.03	3.59	3.35	3.31	3.21	3.51	3.42
PGMS-20-4	3.17	2.97	3.28	3.61	3.22	3.09	3.25	3.35	4.06	3.65	3.35	3.27	3.05	3.45	3.36
PGMS-20-5	3.27	3.14	3.23	3.44	3.28	3.23	3.26	3.34	4.02	3.59	3.37	3.30	3.20	3.46	3.39
PGMS-20-6	3.22	3.40	3.44	3.63	3.20	3.17	3.34	3.40	3.94	3.51	3.40	3.23	3.13	3.48	3.39
PGMS-20-7	3.18	3.35	3.36	3.49	3.26	3.10	3.33	3.35	4.10	3.28	3.35	3.22	3.01	3.43	3.40
PGMS-20-8	3.27	3.38	3.26	3.56	3.26	3.38	3.29	3.34	4.09	3.49	3.37	3.29	3.04	3.44	3.37
PGMS-20-9	3.26	3.18	3.40	3.51	3.27	3.01	3.27	3.40	3.96	3.39	3.33	3.32	3.22	3.49	3.43
PGMS-20-10	3.24	3.26	3.32	3.68	3.26	3.19	3.27	3.42	4.09	3.56	3.38	3.33	3.08	3.45	3.35
Mean	3.23	3.25	3.32	3.54	3.25	3.19	3.28	3.37	4.03	3.48	3.35	3.28	3.14	3.46	3.39
Std. Dev'n	0.0389	0.1530	0.0671	0.0838	0.0301	0.1161	0.0430	0.0399	0.0613	0.1251	0.0331	0.0430	0.0925	0.0258	0.0243
%RSD	1.20	4.71	2.02	2.36	0.93	3.64	1.31	1.19	1.52	3.59	0.99	1.31	2.95	0.75	0.72

Note: Pt data from Labs 3, 9 and 10 was excluded for failing the t test.
Pd data from Lab 9 was excluded for failing the t test.

REFERENCE MATERIAL: CDN-PGMS-20

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver, B.C., Canada
Activation Laboratories Ltd., Ancaster, Ontario, Canada
Activation Laboratories Ltd., Thunder Bay, Ontario, Canada
Alex Stewart, Mendosa, Argentina
ALS Chemex Laboratories, North Vancouver, B.C., Canada
Alfred Knight, Alaska, USA
Alfred Knight, Kamloops, B.C., Canada
Genalysis Laboratory Services Pty. Ltd., Perth, Australia
Inspectorate, Richmond, B.C., Canada
Labtium, Finland
OMAC Laboratories Ltd., Ireland
SGS, Lima, Peru
SGS Vancouver, Canada
TSL Laboratories, Saskatoon, SK, Canada
Ultra Trace Analytical Laboratories, Perth, Australia

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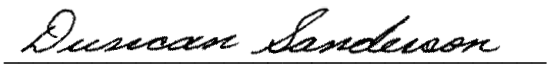
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466 (www.cdnlabs.com)

REFERENCE MATERIAL: CDN-PGMS-21

Recommended values and the "Between Lab" Two Standard Deviations

Gold	***3.42 g/t ± 0.41 g/t***	***Certified value***	***30g FA / ICP or AA***
Platinum	***0.293 g/t ± 0.026 g/t***	***Certified value***	***30g FA / ICP or AA***
Palladium	***2.00 g/t ± 0.18 g/t***	***Certified value***	***30g FA / ICP or AA***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: November 25, 2011

METHOD OF PREPARATION:

Standard CDN-PGMS-21 was prepared using a variety of Au-Pt-Pd bearing ores of a siliceous nature.

Reject ore material was dried, crushed, pulverized and then passed through a 270 mesh screen. The +270 material was discarded. The -270 material was mixed for 5 days in a double-cone mixer. Splits were taken and sent to 15 laboratories for round robin assaying.

Approximate chemical composition (by whole rock analysis) is as follows:

	Percent			Percent
SiO2	55.4		MgO	4.4
Al2O3	12.3		K2O	1.8
Fe2O3	11.9		TiO2	0.4
CaO	4.1		LOI	6.7
Na2O	1.2		S	3.4

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual “between-laboratory” standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

REFERENCE MATERIAL: CDN-PGMS-21

Assay Procedure: 30g fire assay, AA or ICP finish.

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12	Lab 13	Lab 14	Lab 15
SAMPLE	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS-21-1	3.79	3.54	3.12	3.57	3.25	3.22	3.36	3.44	3.03	3.77	3.84	3.35	3.31	3.81	3.68
PGMS-21-2	3.67	3.53	3.10	3.55	3.27	3.22	3.36	3.47	2.96	3.54	3.78	3.31	3.29	4.04	3.73
PGMS-21-3	3.42	3.47	2.96	3.74	3.41	3.29	3.30	3.41	3.25	3.23	3.78	3.06	3.56	3.39	3.65
PGMS-21-4	3.38	3.63	3.16	3.53	3.20	3.34	3.28	3.41	2.95	3.62	4.26	3.17	3.46	3.92	3.63
PGMS-21-5	3.44	3.60	3.09	3.70	3.16	3.33	3.43	3.51	3.09	3.57	3.95	3.13	3.52	3.84	3.72
PGMS-21-6	3.47	3.47	3.04	3.87	3.43	3.46	3.30	3.50	3.12	3.39	3.76	3.27	3.29	3.53	3.56
PGMS-21-7	3.44	3.44	3.15	3.82	3.43	3.31	3.41	3.50	3.26	3.46	3.86	3.05	3.36	3.84	3.55
PGMS-21-8	3.65	3.63	2.98	3.62	3.24	3.37	3.29	3.45	3.16	3.38	4.04	3.07	3.25	3.57	3.63
PGMS-21-9	3.72	3.54	3.21	3.79	3.51	3.59	3.43	3.52	3.02	3.51	4.08	3.18	3.43	3.72	3.65
PGMS-21-10	3.49	3.57	3.16	3.65	3.22	3.48	3.33	3.48	3.12	3.35	3.91	3.28	3.39	3.66	3.56
Mean	3.55	3.54	3.10	3.69	3.31	3.36	3.35	3.47	3.10	3.48	3.93	3.19	3.39	3.73	3.64
Std. Dev'n	0.1456	0.0671	0.0815	0.1191	0.1211	0.1191	0.0582	0.0418	0.1087	0.1543	0.1608	0.1103	0.1049	0.20	0.06
%RSD	4.11	1.90	2.63	3.23	3.66	3.55	1.74	1.21	3.51	4.43	4.09	3.46	3.10	5.32	1.77
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS-21-1	0.301	0.296	0.288	0.290	0.283	0.311	0.29			0.37	0.299	0.302	0.271	0.281	0.318
PGMS-21-2	0.304	0.295	0.279	0.296	0.288	0.305	0.28			0.36	0.292	0.297	0.257	0.292	0.293
PGMS-21-3	0.292	0.298	0.278	0.296	0.298	0.296	0.28			0.34	0.317	0.286	0.285	0.256	0.308
PGMS-21-4	0.299	0.302	0.280	0.293	0.294	0.317	0.27			0.34	0.314	0.268	0.283	0.283	0.317
PGMS-21-5	0.297	0.311	0.292	0.299	0.278	0.307	0.30			0.34	0.300	0.264	0.278	0.275	0.307
PGMS-21-6	0.281	0.305	0.280	0.304	0.293	0.318	0.30			0.36	0.310	0.299	0.269	0.288	0.324
PGMS-21-7	0.300	0.310	0.285	0.292	0.287	0.287	0.29			0.37	0.307	0.269	0.285	0.303	0.318
PGMS-21-8	0.297	0.309	0.282	0.297	0.288	0.313	0.27			0.39	0.293	0.289	0.273	0.301	0.295
PGMS-21-9	0.292	0.305	0.272	0.306	0.288	0.310	0.28			0.34	0.281	0.278	0.279	0.289	0.322
PGMS-21-10	0.289	0.304	0.275	0.298	0.286	0.291	0.27			0.33	0.293	0.284	0.273	0.303	0.290
Mean	0.295	0.304	0.281	0.297	0.288	0.306	0.283			0.354	0.301	0.284	0.275	0.287	0.309
Std. Dev'n	0.0068	0.0057	0.0060	0.0050	0.0057	0.0108	0.0116			0.0190	0.0113	0.0136	0.0086	0.0146	0.0126
%RSD	2.30	1.88	2.12	1.69	1.97	3.52	4.10			5.36	3.77	4.79	3.14	5.09	4.09
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS-21-1	1.99	2.06	2.00	2.03	1.90	1.98	1.89			2.16	2.18	2.02	1.87	2.08	2.10
PGMS-21-2	2.00	1.98	2.05	2.06	1.94	1.99	1.90			2.13	2.29	1.86	1.83	2.10	2.04
PGMS-21-3	1.96	1.95	2.10	2.02	1.95	1.99	1.84			2.04	2.09	1.85	1.92	1.88	2.12
PGMS-21-4	2.00	1.97	2.05	2.08	1.94	1.97	1.87			2.06	2.25	1.82	1.91	2.09	2.13
PGMS-21-5	1.98	2.01	1.95	2.07	1.90	2.02	1.90			2.06	2.14	1.80	1.86	2.08	2.07
PGMS-21-6	1.99	2.04	2.00	2.05	2.00	1.92	1.91			2.04	1.99	1.88	1.91	2.10	2.12
PGMS-21-7	2.00	2.03	2.10	2.07	1.97	1.93	1.90			2.11	2.02	1.82	1.89	2.08	2.06
PGMS-21-8	2.04	2.04	2.00	2.06	2.03	1.98	1.86			2.11	2.13	1.86	1.82	2.07	2.04
PGMS-21-9	2.03	2.05	2.00	2.10	2.00	2.00	1.88			2.13	2.08	1.76	1.87	2.09	2.08
PGMS-21-10	1.99	1.97	2.05	2.04	1.95	1.97	1.86			2.10	2.15	1.81	1.88	2.11	2.05
Mean	2.00	2.01	2.03	2.06	1.96	1.97	1.88			2.09	2.13	1.85	1.88	2.07	2.08
Std. Dev'n	0.0230	0.0394	0.0483	0.0231	0.0413	0.0291	0.0228			0.0417	0.0959	0.0699	0.0334	0.0676	0.0340
%RSD	1.15	1.96	2.38	1.12	2.11	1.47	1.21			1.99	4.50	3.78	1.78	3.27	1.63

Note: **Au data from Lab 11 was excluded for failing the t test.**
Pt data from Lab 10 was excluded for failing the t test.

Two laboratories did not supply Pt, Pd data.

REFERENCE MATERIAL: CDN-PGMS-21

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver, B.C., Canada
Activation Laboratories Ltd., Ancaster, Ontario, Canada
Activation Laboratories Ltd., Thunder Bay, Ontario, Canada
AGAT, Mississauga, Ontario
Alex Stewart, Mendosa, Argentina
ALS Chemex Laboratories, North Vancouver, B.C., Canada
Alfred Knight, Alaska, USA
Alfred Knight, Kamloops, B.C., Canada
CIMM, Lima, Peru
Genalysis Laboratory Services Pty. Ltd., Perth, Australia
Inspectorate, Richmond, B.C., Canada
SGS, Lima, Peru
Skyline Assayers & Laboratories, Arizona, USA
TSL Laboratories, Saskatoon, SK, Canada
Ultra Trace Analytical Laboratories, Perth, Australia

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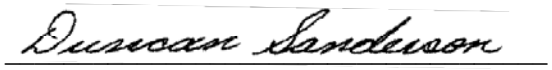
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

#2, 20148 - 102nd Avenue, Langley, B.C., Canada, V1M 4B4, Ph: 604-882-8422 Fax: 604-882-8466 (www.cdnlabs.com)

REFERENCE MATERIAL: CDN-PGMS-22

Recommended values and the "Between Lab" Two Standard Deviations

Gold	***1.23 g/t ± 0.15 g/t***	***Certified value***	***30g FA / ICP or AA***
Platinum	***1.36 g/t ± 0.17 g/t***	***Certified value***	***30g FA / ICP or AA***
Palladium	***6.15 g/t ± 0.56 g/t***	***Certified value***	***30g FA / ICP or AA***

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.
DATE OF CERTIFICATION: April 12, 2012

METHOD OF PREPARATION:

The ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining 593 kg of the Stillwater ore (screened to –325) with 7 kg of a gold ore concentrate (screened to –325). The material was mixed for 5 days in a double-cone mixer. Splits were sent to 15 laboratories for round robin assaying .

Approximate chemical composition (by whole rock analysis) is as follows:

	Percent			Percent
SiO2	45.3		MgO	8.9
Al2O3	21.8		K2O	0.1
Fe2O3	6.9		TiO2	0.1
CaO	12.7		LOI	2.4
Na2O	1.3		S	0.1

Statistical Procedures:

The final limits were calculated after first determining if all data was compatible within a spread normally expected for similar analytical methods done by reputable laboratories. Data from any one laboratory was removed from further calculations when the mean of all analyses from that laboratory failed a t test of the global means of the other laboratories. The means and standard deviations were calculated using all remaining data. Any analysis that fell outside of the mean ±2 standard deviations was removed from the ensuing data base. The mean and standard deviations were again calculated using the remaining data. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Confidence Limits published on other standards.

Results from round-robin assaying are presented on the following page:

REFERENCE MATERIAL: CDN-PGMS-22

Assay Procedure: 30g fire assay, AA or ICP finish.

	Lab 1	Lab 2	Lab 3	Lab 4	Lab 5	Lab 6	Lab 7	Lab 8	Lab 9	Lab 10	Lab 11	Lab 12	Lab 13	Lab 14	Lab 15
SAMPLE	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
PGMS-22-1	1.22	1.16	1.29	1.27	1.33	1.19	1.10	1.32	1.07	1.27	1.29	1.12	1.13	1.37	1.27
PGMS-22-2	1.14	1.21	1.32	1.26	1.24	1.24	1.18	1.30	1.11	1.23	1.40	1.18	1.24	1.35	1.28
PGMS-22-3	1.31	1.25	1.29	1.28	1.26	1.16	1.18	0.83	1.16	1.43	1.31	1.15	1.18	1.32	1.26
PGMS-22-4	1.26	1.22	1.26	1.27	1.21	1.18	1.16	0.98	1.24	1.27	1.25	1.11	1.15	1.27	1.36
PGMS-22-5	1.15	1.24	1.19	1.19	1.29	1.19	1.20	1.15	1.23	1.26	1.33	1.09	1.13	1.39	1.32
PGMS-22-6	1.22	1.19	1.23	1.24	1.23	1.07	1.18	1.43	1.23	1.33	1.34	1.12	1.17	1.28	1.33
PGMS-22-7	1.18	1.20	1.36	1.23	1.20	1.14	1.22	1.21	1.16	1.31	1.29	1.12	1.20	1.31	1.29
PGMS-22-8	1.20	1.26	1.31	1.23	1.34	1.19	1.22	1.18	1.17	1.24	1.30	1.15	1.10	1.28	1.40
PGMS-22-9	1.18	1.28	1.30	1.31	1.27	1.04	1.21	1.09	1.25	1.31	1.15	1.12	1.17	1.33	1.37
PGMS-22-10	1.21	1.21	1.24	1.33	1.18	1.03	1.18	1.27	1.19	1.25	1.31	1.17	1.13	1.42	1.35
Mean	1.20	1.22	1.28	1.26	1.26	1.14	1.18	1.17	1.18	1.29	1.30	1.13	1.16	1.33	1.32
Std. Dev'n	0.0501	0.0358	0.0495	0.0401	0.0536	0.0730	0.0353	0.1744	0.0579	0.0591	0.0648	0.0283	0.0408	0.0503	0.0454
%RSD	4.16	2.93	3.87	3.19	4.27	6.39	2.98	14.85	4.91	4.58	5.00	2.50	3.52	3.78	3.43
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
PGMS-22-1	1.32	1.40	1.51	1.27	1.60	1.29	1.17	1.32		1.43	1.52	1.21	1.32	1.44	1.48
PGMS-22-2	1.31	1.32	1.50	1.28	1.42	1.26	1.24	1.33		1.44	1.43	1.23	1.35	1.36	1.47
PGMS-22-3	1.33	1.34	1.44	1.25	1.44	1.30	1.27	0.85		1.41	1.56	1.28	1.35	1.46	1.52
PGMS-22-4	1.37	1.35	1.47	1.31	1.39	1.33	1.26	1.05		1.39	1.45	1.29	1.35	1.36	1.53
PGMS-22-5	1.35	1.33	1.41	1.33	1.42	1.30	1.35	1.18		1.42	1.55	1.23	1.37	1.41	1.50
PGMS-22-6	1.34	1.34	1.43	1.26	1.36	1.24	1.22	1.33		1.41	1.35	1.27	1.31	1.41	1.53
PGMS-22-7	1.33	1.39	1.46	1.26	1.40	1.27	1.37	1.34		1.39	1.38	1.30	1.39	1.40	1.52
PGMS-22-8	1.29	1.41	1.45	1.27	1.51	1.26	1.33	1.29		1.39	1.64	1.30	1.28	1.44	1.54
PGMS-22-9	1.37	1.44	1.49	1.37	1.45	1.28	1.39	1.14		1.40	1.62	1.25	1.26	1.42	1.51
PGMS-22-10	1.34	1.38	1.46	1.39	1.43	1.20	1.26	1.24		1.45	1.59	1.25	1.27	1.37	1.54
Mean	1.33	1.37	1.46	1.30	1.44	1.27	1.29	1.20		1.41	1.51	1.26	1.33	1.41	1.51
Std. Dev'n	0.0258	0.0397	0.0316	0.0490	0.0683	0.0339	0.0711	0.1590		0.0216	0.1001	0.0318	0.0443	0.0350	0.0250
%RSD	1.94	2.90	2.16	3.77	4.74	2.66	5.53	13.20		1.53	6.64	2.52	3.34	2.49	1.65
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
PGMS-22-1	6.05	5.98	6.67	6.05	6.76	6.03	5.57	6.01		6.24	6.00	5.75	5.82	6.50	6.90
PGMS-22-2	6.22	6.05	6.66	6.13	6.15	6.13	5.88	5.94		6.37	6.12	5.69	5.71	6.44	6.91
PGMS-22-3	6.21	5.94	6.67	6.07	6.37	6.28	5.83	3.90		6.36	6.31	5.82	6.15	6.51	7.06
PGMS-22-4	6.27	5.94	6.81	6.12	6.35	6.20	6.11	4.75		6.24	6.20	5.76	5.88	6.43	7.10
PGMS-22-5	6.21	6.03	6.73	6.03	6.29	6.04	6.29	5.31		6.30	6.41	5.65	6.00	6.61	7.08
PGMS-22-6	6.17	5.92	6.70	6.10	6.24	5.91	5.96	5.64		6.24	6.13	5.73	6.06	6.58	7.21
PGMS-22-7	6.17	5.89	6.77	6.08	6.12	6.12	6.43	5.87		6.20	6.16	5.91	6.07	6.55	7.02
PGMS-22-8	6.16	5.92	6.90	6.06	6.58	6.07	6.42	5.83		6.22	6.42	5.87	5.75	6.57	7.03
PGMS-22-9	6.19	5.94	6.79	6.08	6.45	6.09	6.28	5.06		6.14	6.23	5.73	5.73	6.52	7.08
PGMS-22-10	6.18	6.04	6.54	6.31	6.35	5.86	6.25	5.72		6.25	6.03	5.79	5.77	6.50	7.16
Mean	6.18	5.97	6.72	6.10	6.37	6.07	6.10	5.40		6.26	6.20	5.77	5.89	6.52	7.06
Std. Dev'n	0.0568	0.0566	0.0996	0.0781	0.1936	0.1239	0.2837	0.6688		0.0703	0.1460	0.0796	0.1626	0.0582	0.0966
%RSD	0.92	0.95	1.48	1.28	3.04	2.04	4.65	12.38		1.12	2.35	1.38	2.76	0.89	1.37

Note: **Pt data from Lab 11 was excluded for failing the t test.**
Pd data from Lab 15 was excluded for failing the t test.

One laboratory did not supply Pt, Pd data.

REFERENCE MATERIAL: CDN-PGMS-22

Participating Laboratories:

(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver, B.C., Canada
Activation Laboratories Ltd., Ancaster, Ontario, Canada
Activation Laboratories Ltd., Thunder Bay, Ontario, Canada
AGAT, Mississauga, Ontario
Alex Stewart Argentina SA
ALS Chemex Laboratories, North Vancouver, B.C., Canada
ALS Brisbane, Australia
American Assay Laboratories, Nevada, USA
Genalysis Laboratory Services Pty. Ltd., Perth, Australia
Labtium, Finland
SGS, Lima, Peru
SGS, Vancouver, B.C., Canada
Skyline Assayers & Laboratories, Arizona, USA
TSL Laboratories, Saskatoon, SK, Canada
Ultra Trace Analytical Laboratories, Perth, Australia

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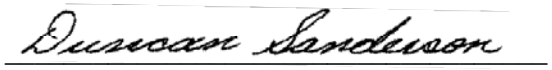
Certified by 

Duncan Sanderson, Certified Assayer of B.C.

Geochemist

Barry Smee, Ph.D., P. Geo.

CDN Resource Laboratories Ltd.

10945-B River Road, Delta, B.C., V4C 2R8, 604 596-2245, Fax: 604 588-3960

PLATINUM GROUP ORE REFERENCE STANDARD: CDN-PGMS-8

Recommended values and the "Between Lab" Two Standard Deviations

Gold concentration:	0.82 ± 0.14 g/t
Platinum concentration:	0.44 ± 0.07 g/t
Palladium concentration:	1.50 ± 0.12 g/t

PREPARED BY: CDN Resource Laboratories Ltd.
CERTIFIED BY: Duncan Sanderson, B.Sc., Licensed Assayer of British Columbia
INDEPENDENT GEOCHEMIST: Dr. Barry Smee., Ph.D., P. Geo.

METHOD OF PREPARATION:

The ore was supplied by Stillwater Mining Corporation from the Stillwater Complex in Montana.
The mineralogy of the Stillwater Pt/Pd ore consists of up to 1 % sulphides comprising chalcopyrite, pentlandite , pyrrhotite, ± pyrite hosted by a chromite-rich ultramafic layer. The main platinum–bearing minerals are Braggite (Pt,Pd,Ni)S, Cooperite (Pt, Pd ,Ni)S as well as Isoferroplatinum (PtFe3) and Moncheite (Pt,Pd)(Te,Bi)2. The majority of the palladium is hosted as solid solution within the pendlandite ((Fe,Ni)9S8); less then 15 % as Vysotskite (Pd,Ni,Pt)S, Bragite, Cooperite and Moncheite.

This standard was prepared by combining a quantity of the Stillwater ore (screened to –325) with a quantity of gold-bearing ore from the Misty Mountain Specogna deposit (screened to –200) and diluting with a blank granitic material that had been screened to –200 mesh. The material was mixed for 6 days in a rotary mixer. Splits were sent to 12 laboratories for round robin assaying .

Approximate chemical composition is as follows:

	Percent			Percent
SiO2	60.7		MgO	3.0
Al2O3	16.1		K2O	1.4
Fe2O3	7.6		TiO2	0.6
CaO	5.4		LOI	0.5
Na2O	3.3			

Statistical Procedures:

The mean and standard deviation for all data was calculated. Outliers were defined as samples beyond the mean ± 2 Standard Deviations from all data. These outliers were removed from the data and a new mean and standard deviation was determined. This method is different from that used by Government agencies in that the actual "between-laboratory" standard deviation is used in the calculations. This produces upper and lower limits that reflect actual individual analyses rather than a grouped set of analyses. The limits can therefore be used to monitor accuracy from individual analyses, unlike the Certified Limits published on other standards.

Results from round-robin assaying are presented on the following page:

Assay Procedure: 30g fire assay, AA or ICP finish.

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	Lab. 1	Lab. 2	Lab. 3	Lab. 4	Lab. 5	Lab. 6	Lab. 7	Lab. 8	Lab. 9	Lab. 10	Lab. 11	Lab. 12
SAMPLE	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t
CDN-PG8-1	0.82	0.80	0.82	0.84	0.74	0.90	0.89	0.85	0.55	0.84	0.89	0.72
CDN-PG8-2	0.75	0.88	0.72	0.79	0.74	0.85	0.81	0.76	0.73	0.84	0.86	0.69
CDN-PG8-3	0.74	0.87	0.79	0.95	0.81	0.80	0.81	0.83	0.69	0.90	0.90	0.71
CDN-PG8-4	0.75	0.93	0.84	0.82	0.86	0.82	0.93	0.77	0.71	0.82	0.92	0.69
CDN-PG8-5	0.77	0.90	0.88	0.83	0.73	0.91	0.92	0.69	0.72	0.79	0.88	0.71
CDN-PG8-6	0.89	0.79	0.86	0.93	0.73	0.80	0.89	0.90	0.81	0.81	0.90	0.72
CDN-PG8-7	0.72	0.91	0.71	0.88	0.76	0.80	0.91	0.83	0.87	0.86	0.92	0.73
CDN-PG8-8	0.72	0.82	0.73	0.80	0.70	0.84	0.94	0.79	0.73	0.79	0.90	0.71
CDN-PG8-9	0.77	0.83	0.80	0.93	0.81	0.85	0.80	0.90	0.77	0.83	0.88	0.76
CDN-PG8-10	0.80	0.96	0.65	0.86	0.75	0.87	0.85	0.88	0.87	0.83	0.92	0.73
Mean	0.77	0.87	0.78	0.86	0.76	0.84	0.88	0.82	0.75	0.83	0.90	0.72
Std. Dev'n	0.0527	0.0574	0.0745	0.0574	0.0469	0.0403	0.0534	0.0691	0.0940	0.0328	0.0200	0.0216
%RSD	6.81	6.61	9.56	6.65	6.14	4.78	6.10	8.44	12.61	3.95	2.23	3.02
	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t	Pt g/t
CDN-PG8-1	0.47	0.42	0.61	0.42	0.48	0.44	0.44	0.45	0.31	0.50	0.51	0.40
CDN-PG8-2	0.41	0.43	0.41	0.45	0.44	0.48	0.40	0.50	0.35	0.45	0.49	0.37
CDN-PG8-3	0.42	0.41	0.41	0.42	0.44	0.43	0.45	0.54	0.38	0.46	0.48	0.41
CDN-PG8-4	0.44	0.43	0.47	0.42	0.44	0.46	0.46	0.46	0.36	0.49	0.47	0.40
CDN-PG8-5	0.40	0.44	0.43	0.45	0.43	0.42	0.44	0.44	0.38	0.46	0.52	0.39
CDN-PG8-6	0.43	0.41	0.44	0.42	0.42	0.51	0.45	0.43	0.48	0.47	0.49	0.41
CDN-PG8-7	0.43	0.42	0.42	0.47	0.45	0.42	0.45	0.50	0.44	0.47	0.53	0.41
CDN-PG8-8	0.43	0.44	0.43	0.42	0.44	0.44	0.47	0.47	0.40	0.49	0.51	0.39
CDN-PG8-9	0.44	0.45	0.40	0.44	0.45	0.51	0.44	0.54	0.46	0.46	0.50	0.41
CDN-PG8-10	0.42	0.42	0.38	0.42	0.42	0.43	0.45	0.45	0.42	0.47	0.48	0.39
Mean	0.43	0.43	0.44	0.43	0.44	0.45	0.45	0.48	0.40	0.47	0.50	0.40
Std. Dev'n	0.0193	0.0134	0.0645	0.0183	0.0170	0.0347	0.0184	0.0406	0.0527	0.0162	0.0193	0.0137
%RSD	4.52	3.13	14.65	4.22	3.88	7.64	4.14	8.50	13.23	3.43	3.88	3.45
	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t	Pd g/t
CDN-PG8-1	1.52	1.54	1.48	1.54	1.46	1.55	1.51	1.53	1.07	1.62	1.54	1.38
CDN-PG8-2	1.42	1.52	1.54	1.52	1.44	1.53	1.47	1.52	1.24	1.60	1.56	1.37
CDN-PG8-3	1.45	1.55	1.54	1.51	1.44	1.54	1.53	1.52	1.32	1.65	1.56	1.42
CDN-PG8-4	1.53	1.57	1.48	1.57	1.47	1.48	1.48	1.50	1.22	1.60	1.53	1.46
CDN-PG8-5	1.42	1.56	1.54	1.55	1.45	1.50	1.50	1.47	1.35	1.60	1.54	1.47
CDN-PG8-6	1.51	1.56	1.47	1.46	1.42	1.51	1.48	1.55	1.39	1.58	1.52	1.46
CDN-PG8-7	1.42	1.55	1.48	1.48	1.48	1.49	1.46	1.54	1.50	1.60	1.53	1.37
CDN-PG8-8	1.45	1.54	1.48	1.51	1.50	1.49	1.48	1.53	1.41	1.62	1.50	1.34
CDN-PG8-9	1.47	1.56	1.40	1.53	1.44	1.53	1.51	1.51	1.45	1.55	1.54	1.41
CDN-PG8-10	1.53	1.55	1.45	1.52	1.39	1.52	1.50	1.54	1.48	1.57	1.53	1.34
Mean	1.47	1.55	1.49	1.52	1.45	1.51	1.49	1.52	1.34	1.60	1.54	1.40
Std. Dev'n	0.0452	0.0141	0.0445	0.0321	0.0313	0.0237	0.0215	0.0235	0.1343	0.0281	0.0178	0.0484
%RSD	3.08	0.91	3.00	2.12	2.16	1.56	1.44	1.55	10.00	1.76	1.16	3.45

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Participating Laboratories:
(not in same order as listed in table of results)

Acme Analytical Laboratories Ltd., Vancouver
Assayers Canada Ltd., Vancouver
ALS Chemex Laboratories, North Vancouver
Genalysis Laboratory Services Pty Ltd., Perth
GTK Laboratory, Geological Survey of Finland
International Plasma Laboratories Ltd., Vancouver
OMAC Laboratories Ltd., Ireland
SGS-Perth, Australia
SGS-XRAL Labs, Toronto
Stillwater Mining Corporation, Montana
TSL Laboratories, Saskatoon
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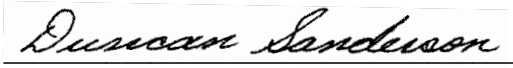
Certified by 
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APPENDIX D

METALLURGICAL TEST WORK

xstrata
process support

Xstrata Process Support

Final Report – 5009817.00

Prepared For: Mr. Erin Legault
Copy to: D. Fragomeni

Lac des Iles Roby and Offset Zone Ore Characterisation Report

Adding value.
Reducing risk…

Prepared By:

Simon Yu Jorge Oliveira Ryan Salemink Kevin Chisholm

Final Report Submitted: June 4th 2010

Executive Summary

Xstrata Process Support (XPS) undertook metallurgical and mineralogical testwork on Lac des Iles Roby Zone and Offset Zone composites. Samples were received at XPS in Feburary 2010 and work commenced shortly thereafter using methods and practices [1] developed at the Xstrata Process Support Centre focused on providing quality and bankable mineralogical and metallurgical data. The following salient conclusions were drawn from this testwork program. These are discussed in more detail in the main body of the report:

Mineralogical Analysis of Feeds

- The modal mineralogical analysis indicates that the Roby Zone and the Offset Zone composites are very similar.
- Talc content ranges from 0.9% by mass in the Roby composite, to 1.1% by mass in the Offset composite indicating only marginally higher talc content in the Offset zone. It is recognized that due to localised faulting and alteration that talc content can be significantly higher in localised regions within these zones
- At a P_{80} of 75 µm, Ni-Sulphide liberation (pentlandite and millerite combined) is moderate at 76% and 79% for Roby and Offset respectively while chalcopyrite is 64% and 63% liberated indicating minimal variation in sulphide liberation between the Roby and Offset composites.
- PGM phases identified within the feed are Pd-sulphides (vysotskite), Pd-tellurides (kotulskite, merenskyite), Pd-antimonides (mertieite), Pd-arsenides (arsenopaladinite) with lesser amounts of Pt-arsenides (sperrylite) and Au/Ag species (electrum).
- The majority of PGMs by mass are Pd-tellurides and Pd-antimonides followed by Pd sulphides.
- Average PGM grain size is very fine grained and the combined average for both Roby and Offset zone are similar at 7 µm and 8 µm respectively.
- Although based on limited PGM particle statistics in the feed, 70% of PGM by mass in both Roby and Offset are liberated at a target grind size of 80% passing 75 µm.
- PGM association and locking data indicates that the majority of remaining PGM which is not liberated is largely locked with non-sulphide gangue or other complex gangue dominated associations.
- Mineral composition analysis by EPMA indicates solid solution levels of Ni in pyrrhotite and pyrite as well as low levels of Pd in solution within pentlandite. Silicate analyses indicate low levels of solid solution Ni in talc, chlorite, orthopyroxene, amphiboles and biotite.
- Ni deportment data indicates that for Roby and Offset composites, only 52% and 60% of total Ni is carried by Ni-sulphides respectively. The remaining nickel occurs predominantly in Fe-Mg gangue. Ni recovery is heavily impact by gangue rejection in the circuit resulting in lower realized Ni recoveries.

Metallurgical Study Results:

Roby:

- Roby Zone Composite contains: 0.12% Ni, 0.07% Cu, 0.18% S, 6.12g/t Pd, 0.31g/t Pt, 0.37 g/t Au, and 12.20% MgO.
- A series of rougher-scavenger flotation tests were undertaken to determine an appropriate primary grind size and assess staged-grind vs. fine-grinding as well as to select and optimise dosages for MgO depressants, and PGM promoters.
- Tests showed that a primary grind of 80% passing 38 microns, 26 minutes of rougher flotation time, 92 g/t CMC (*Depramin C*), 26 g/t PAX, 16 g/t Aero 3477 and 36 g/t MIBC gave optimum rougher-scavenger flotation response.
- Rougher flotation testing indicates no clear advantage to staged grinding which is supported by the fine PGM grain size and lack of coarse bi-modal PGM grain size distribution.
- A series of open circuit cleaner tests were undertaken to determine appropriate MgO depressant and cleaning stage requirements. Tests showed that 25 g/t CMC for MgO depressant and 3 stages of cleaning gave optimum cleaning performance.
- Locked-Cycle Tests (LCTs) were undertaken to provide steady state metallurgical projections of Roby ore. LCT results highlight the good upgrade characteristics of the Roby ore with average 35.9% Ni recovery, 86.1% Cu recovery, 86.2% Pd recovery and 78.6% Pt recovery were achieved at 5.2 % Ni and 7.3% Cu grade in the final concentrate. Due to reduced CMC addition in the 1st Clnr and 2nd Clnr, and high circulating load in the 1st Clnr, MgO content of the final concentrate was 8.97%, slightly above the desired limit. However, based on open circuit cleaning flotation tests, the MgO content in the final cleaner concentrate was as low as 0.85% and should be controlled in the plant by judicious use of CMC and control of circulating loads.
- Performance of the locked cycle test was a challenge, due to the low mass pulls. Feed of 10 kg/cycle were used and after 6 cycles, mass, metal balance and stability was acceptable for the Roby Zone.

Offset:

- Offset Composite contains: 0.14% Ni, 0.10% Cu, 0.28% S, 5.52g/t Pd, 0.35g/t Pt, 0.29 g/t Au, and 11.60% MgO.
- A series of rougher-scavenger flotation tests were undertaken to determine an appropriate primary grind size, staged-grind vs. fine-grinding, MgO depressant, and reagent dosage. Consistent with that of Roby composite, tests showed that a primary grind of 80% passing 38 microns, 26 minutes of rougher flotation time, 92 g/t CMC (*Depramin C*), 26 g/t PAX, 16 g/t Aero 3477 and 36 g/t MIBC gave optimum rougher-scavenger flotation response.
- A series of open circuit concentrate tests were undertaken to determine an appropriate MgO depressant and cleaning stages requirement. Consistent with that of Roby composite, cleaning test results showed that 50 g/t CMC for MgO depressant and 3 stages of cleaning gave optimum cleaning performance.
- Locked-Cycle Tests (LCTs) were undertaken to provide a steady state metallurgical projection of Offset ore. The LCT results highlight the good upgrade characteristics of the Offset ore with average 27.3% Ni, 90.5% Cu recovery, 83.1% Pd recovery and 86.8% Pt recovery were achieved at 3.96 % Ni and 10.0% Cu grade in the final concentrate. Better Cu metallurgy is likely due to higher Cu head grades.
- Mass, metal balance and stability was poor for the Offset Zone locked cycle test. Cu and Pt metal balance was acceptable however mass, Ni, Pd and MgO stability were below targets

with unaccounted loss. Over 35% of Pd was present in the Offset Zone intermediate tests versus only 12% for the Roby Zone. This contributed to a lack of steady state conditions.

- Although this robust method of calculating LCT projections highlights a range of performance, 2-3 of the methods are in agreement. This in combination with open-circuit cleaner tests performed during the course of the program indicated acceptable recovery and MgO contents. More cycles may have been required to achieve steady state and the test can be repeated under these conditions if requested.

Mineralogical Analysis of Offset LCT Concentrate:

- Sulphides make up 62% of the concentrate by mass while pyrite alone constitutes 24%.
- Gangue diluents in concentrate are dominated by Fe-Mg silicates with noticeable upgrading of talc in the concentrate to 6.5% by mass. Orthopyroxene makes up approximately 10% of the concentrate by mass.
- Overall Ni-sulphide liberation (pentlandite + millerite) in the concentrate is 82% while total chalcopyrite liberation is 91%.
- The majority of PGMs by mass are Pd-tellurides and Pd-antimonides followed by Pd sulphides which is consistent with the Offset feed measurements.
- Average PGM grain size data per fraction indicates that the PGM population is very fine grained and the combined average for the Offset LCT concentrate at 8µm which is also consistent with the Offset feed measurements.
- Total PGM liberation data indicates that approximately 84% of PGMs by mass in the Offset LCT concentrate are liberated at a target grind size of 80% passing 38 µm while 15% by mass is middling and 2% is locked.
- Association and locking data indicates that the majority of remaining PGM which is not liberated is largely locked with non sulphide gangue or other complex gangue dominated associations, similar to analysis of the feed.

Ore Hardness:

- Ore hardness tests show that both Roby and Offset ore are characterized as 'hard' with a Bond ball mill work index (BWI) of 17.9 and 17.0 kWh/t respectively.

Gravity Recovery Tests (GRG):

- Based on the extended GRG tests at a P_{80} of 54 µm, both platinum and palladium bearing minerals do not respond well to gravity recovery with only 31.9% Pt and 28.8% Pd recovery after three processing stages at 12.0 g/t and 176.5 g/t respectively in concentrate . This is consistent with the fine grain size of PGM minerals identified from QEMSCAN mineralogical analysis. Gravity concentration of PGM does not appear to be a viable process option to improve grade and recovery of PGE. MgO in final gravity concentrates was not assayed and could present an alternative for MgO rejection if MgO penalties become excessive.

Recommendations:

Based on the testwork program undertaken at XPS, the following recommendations should be considered by LDI for Phase 2 Metallurgical Testwork:

- Grinding JKSimMet Simulation Study is to access the current grinding capacity and configuration for the proposed primary grind target of P_{80} 38µm.
- Zone composites help provide robust and long term metallurgical projection data however, a series of variability tests are recommended which test higher talc contents as well as a range of grades through the proposed flowsheet. This can help define the range of metallurgical variability which could be expected during processing of the ores.
- Flotation testing using soda ash as a pH modifier should be investigated in order to determine any benefits of higher than natural pH on gangue depression.

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1.0 – Introduction

Xstrata Process Support (XPS) undertook metallurgical and mineralogical testwork on the Roby Zone and Offset Zone from Lac des Iles Mine. Samples were received at XPS in Feburary 2010 and work commenced shortly thereafter using methods and practices [1] developed at the Xstrata Process Support Centre focused on providing quality and bankable mineralogical and metallurgical data. Phases of work completed are in accordance with the agreed scope of work issued Nov 12th 2009 and include the following:

- Modal mineralogy of mill feed composites and one LCT concentrate for QEMSCAN
- Flowsheet development: including optimal primary grind size, reagent screening, staged grind/flotation evaluation, cleaning and MgO depressant tests
- LCT tests for metallurgical performance projections for Roby and Offset zone
- Bond's Work Index data for each composite
- Gravity Recovery Tests (GRG)
- Data interpretation and reporting

The objective of this project is to provide mineralogical information (QEMSCAN) on each composite, help define optimal flowsheet conditions related to grind size and reagents for processing Roby and Offset ores in the LDI concentrator, perform Locked-Cycle tests for the overall metallurgical projections as well as provide ore hardness characterisation data[2].

2.0 –Composite Selection

A site visit was completed in December of 2009 by XPS personnel to assist in the selection of appropriate sample material for the metallurgical composites in order to represent the zones in question. Meetings and discussions took place regarding the issues of variability testing vs. zone composites from a logistics, cost and scheduling perspective. Based on discussion and consensus with the LDI teem, zone composites were selected as the most feasible method of proceeding with this phase of testing. The following parameters and conditions were applied to each zone composite in order to build the metallurgical composites:

Roby Composite

- The 4.5 g/t resource shell was determined as the guideline for the sample population and the corresponding in-situ resource grades were used as the target grade for this population model.
- The sample population in the mined out areas were checked against the rest of the Roby population which was deemed to be similar. Therefore, all samples were incorporated from this region in order to ensure enough mass could be collected.
- Target grade and grade distributions (Pd, Pt, Au, Cu, Ni) specific to this region were modelled and used as guideline to ensure the appropriate range in grades were captured in the composite
- Individual samples were chosen from across the Roby zone to ensure that the spatial distribution is reasonable and to minimize the possible effect of clustering bias.
- Lithological distribution was also modelled to ensure the rock types captured in the composite fit with the overall population.

Offset Composite

- The 4.0 g/t resource shell was determined as the guideline for the sample population and the corresponding in-situ resource grades were used as the target grade for this population model.
- The sample population was limited to the upper phase of the current Offset mine plan
- Target grade and grade distributions (Pd, Pt, Au, Cu, Ni) specific to this region were modeled and used as guideline to ensure the appropriate range in grades were captured in the composite
- Individual samples were chosen from across the upper Offset zone to ensure that the spatial distribution is reasonable and to minimize the possible effect of clustering bias.
- Lithological distribution was also modelled to ensure the rock types captured in the composite fit with the overall population.

During phases of sample collection, a minor proportion of the sample intervals were missing and replacement samples were selected in multiple phases. Attempts were made to maintain the grade, lithology and spatial distribution of the composite whenever possible.

3.0– Mineralogical Results - Feed

3.1- Introduction and Methodology

Mineral analyses were completed using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscope) and EPMA (Electron Probe Microanalysis). The objectives of the study were to quantify modal analysis, grain size, Ni deportment, and compositional analyses of the gangue phases and paymetal phases within the samples.

Mineralogy was conducted on a subsample of rougher float feed from the Roby and Offset composites. The rougher float feed was ground to a P_{80} of 75 µm. The two samples were screened; cyclosized into a series of size fractions and each fraction prepared as polished sections for mineralogical analysis by QEMSCAN. Up to six polished sections were prepared for each fraction for increased counting statistics on PGM-bearing particles. The mineralogy for the CS7 fraction was estimated from assay and no PGM data is available.

QEMSCAN is an automated system that produces particle maps (colour coded by mineral), through collection of rapidly acquired energy dispersive x-rays. These maps quantify the modal mineralogy, texture, grain size, element deportment and liberation in each of the samples.

Quantitative compositional analysis was completed using a Cameca SX-100 Electron Microprobe. Electron Probe Microanalysis (EPMA) produces higher electron beam currents and increased beam stability, coupled with higher resolution wavelength dispersive spectrometry (WDS). These features allow for improved detection limits and accuracy of the resulting analysis. Detection limits can be as low as 200 ppm which provides detailed trace element compositions within the various mineral species in this study. Resulting detailed compositional data is then input back into the QEMSCAN software, in order to refine the final elemental deportment calculations. Compositional data from microprobe analysis can also be used to update the "Species Identification Program" within

QEMSCAN. Measurements can be reclassified in order to update overall modal and deportment data.

3.2- Modal Mineralogy

Figure 1 shows the overall modal mineralogy for the rougher float feed of the Roby Zone composite and the Offset zone composite. Detailed data tables are included in the Appendix.

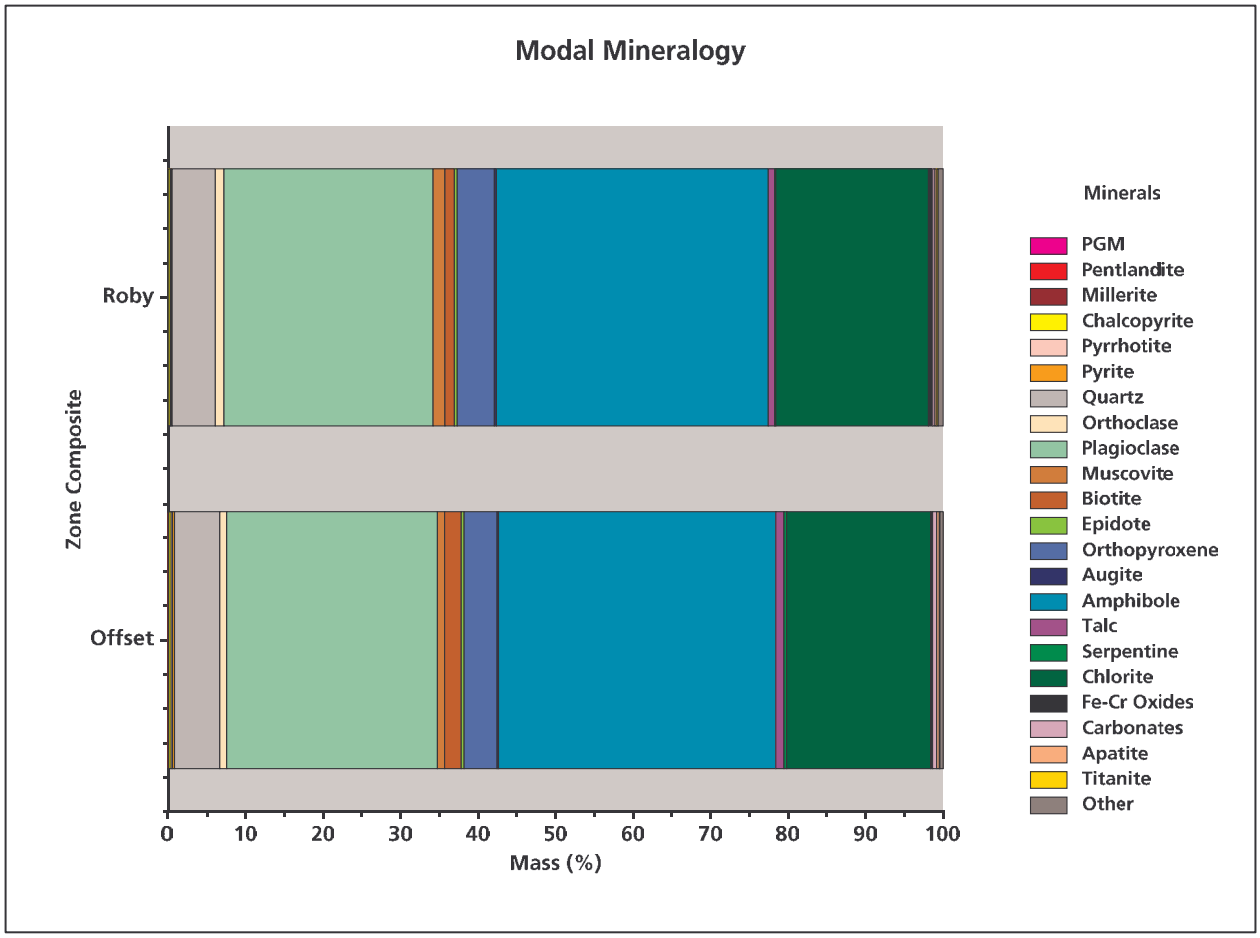

Figure 1: Modal Mineralogy Comparing Roby Zone and Offset Zone

The modal analysis indicates that the Roby Zone and the Offset Zone composites are very similar with minimal variation in overall mineralogical breakdown. Roby and Offset Zone mineralogy is dominated by amphibole (actinolite), plagioclase feldspar, chlorite, quartz and orthopyroxene in decreasing order of abundance. Minor amounts of mica (muscovite, biotite), talc and orthoclase are present with trace amounts sulphide phases (pentlandite, chalcopyrite, pyrite, pyrrhotite, millerite) as well as trace Fe-oxides, apatite and titanite.

Talc is noted as a gangue phase of significant importance in processing of these ores and talc content ranges from 0.9% by mass in the Roby composite, to 1.1% by mass in the Offset

composite indicating only marginally higher talc content in the Offset zone. The decision to construct global zone composites versus a variability composite approach can mute the effect or significance of talc. It is recognized that due to localised faulting and alteration that talc content can be significantly higher in localised regions within these zones.

Although sulphides make up less than 1% of the mineralogy by mass Figure 2 highlights the detailed breakdown of sulphide content in the zone composites. Although the Offset composite contains marginally more overall sulphide content, the relative proportions of sulphide phases are similar between the Roby and Offset Zone composite.

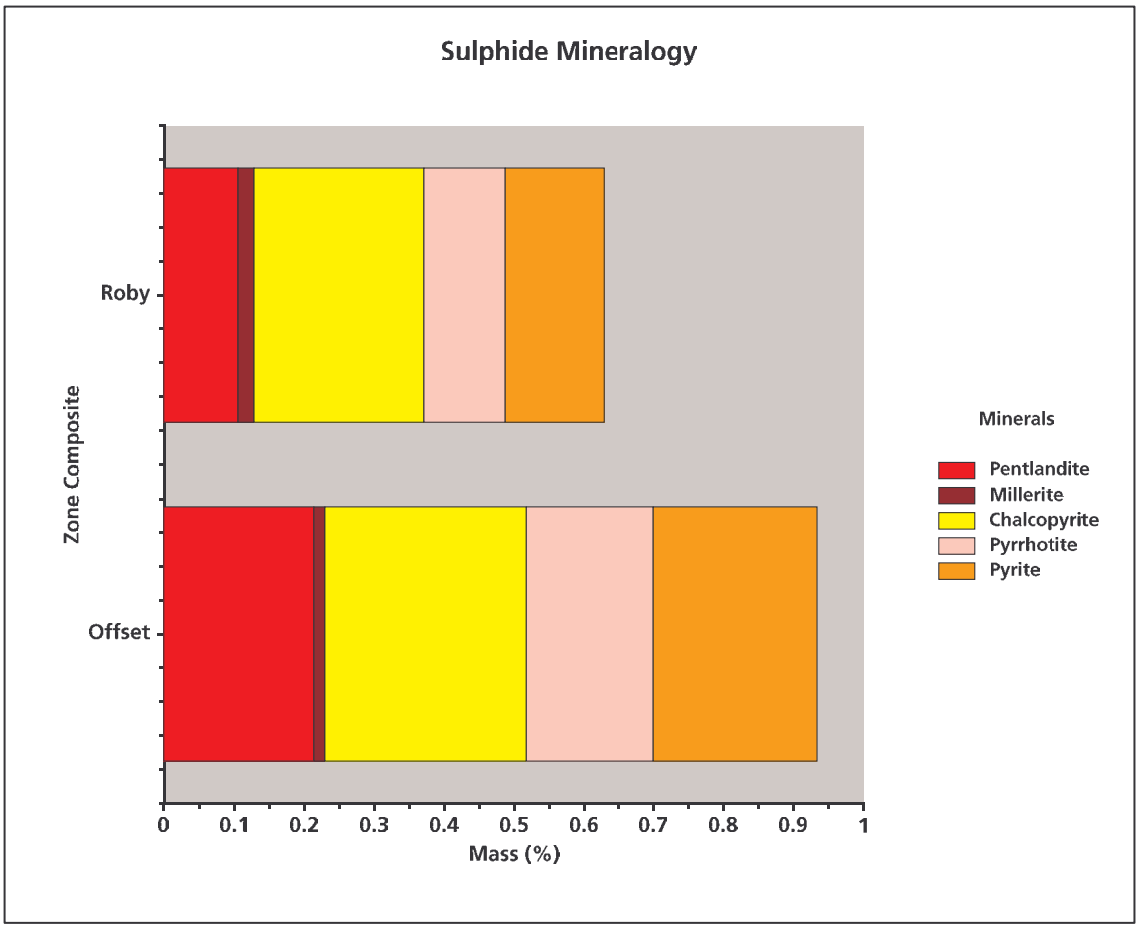

Figure 2: Sulphide Mineralogy Comparing Roby Zone and Offset Zone

Figure 3 and Figure 4 summarise the size-by size mineralogical breakdown of the feed composites which also indicate similar mineralogical breakdown from composite to composite.

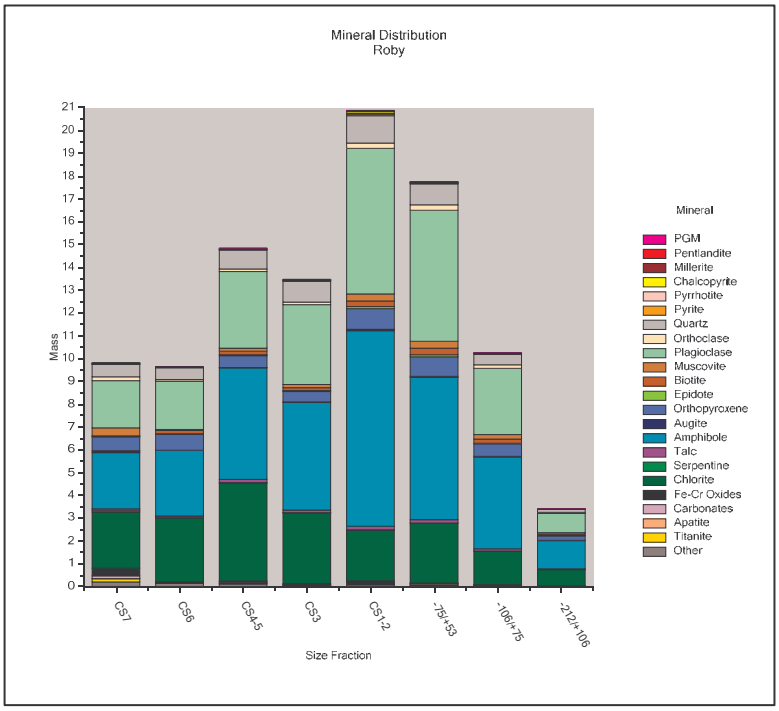

Figure 3: Roby Zone Size-by-size Mineralogy

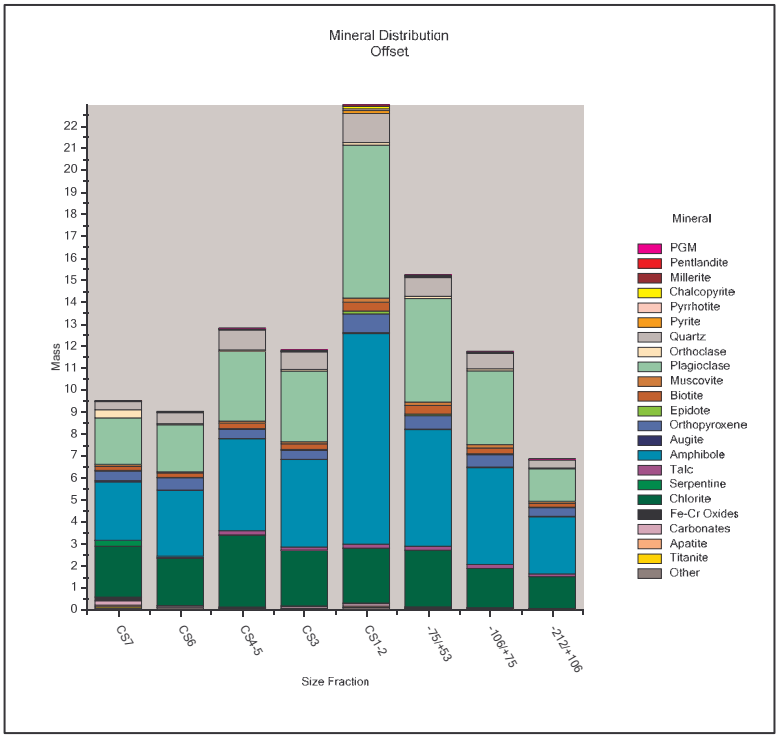

Figure 4: Offset Zone Size-by-Size Mineralogy

3.3- Ni and Cu-Sulphide Liberation

Due to the opportunity and impact of Ni and Cu recoveries on the overall economics of LDI Mine, a mineral liberation assessment for Ni and Cu sulphides was completed in order to provide insight into Cu and Ni recoveries in the plant. It is important to recall that the feed samples measured here are at a P_{80} of 75µm.

Figure 5 summarises the total liberation of Ni-sulphides (millerite and pentlandite) and Cu-sulphides (chalcopyrite) in both composites. For this exercise, millerite and pentlandite are grouped together into one Ni-sulphide category due to their close textural association.

This plot represents the total mass distribution of the mineral/group of interest divided into three liberation states. Liberated particles are defined as any particle containing greater than 90% by area of the mineral of interest (red). Middling particles are defined as those containing between 30% and 90% (green), and locked particles represent those containing less than 30% by area (yellow). Detailed size-by-size liberation tables are in included in the Appendix.

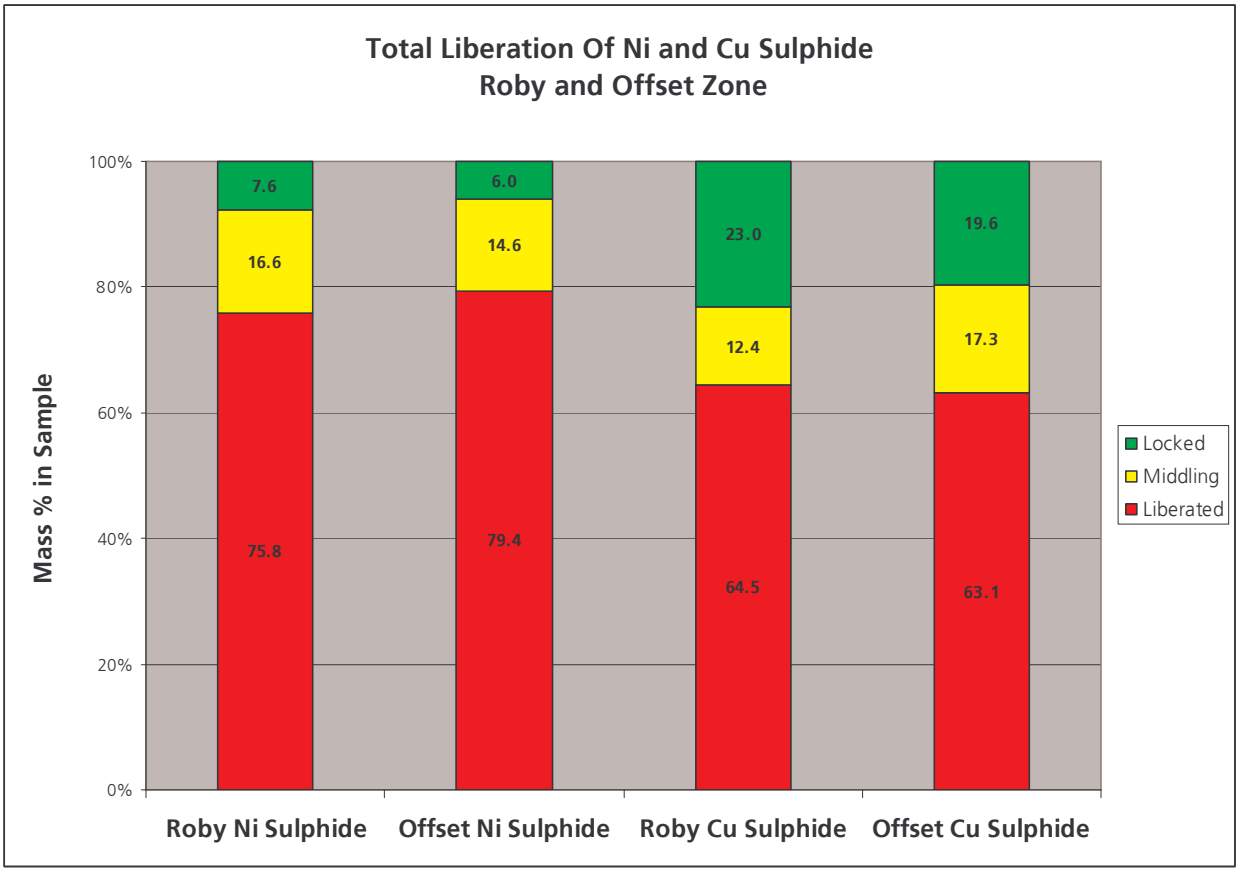

Figure 5: Total Ni and Cu Sulphide Liberation in Roby and Offset Feed

Total liberation data indicates that at a P_{80} of 75 µm, Ni-sulphides are moderately well liberated while Cu-sulphides have proportionally more locked textures which reduce overall liberation. Liberation data between the two composite is consistent indicating minimal variation in sulphide liberation between the Roby and Offset composites.

Further investigation into the nature of the locked and middling Ni-sulphides was completed to provide additional insight based on mineralogical textures in the feed. Figure 6 summarises the middling and locked mineral association characteristics of Ni-sulphides within the Roby and Offset Zone composites.

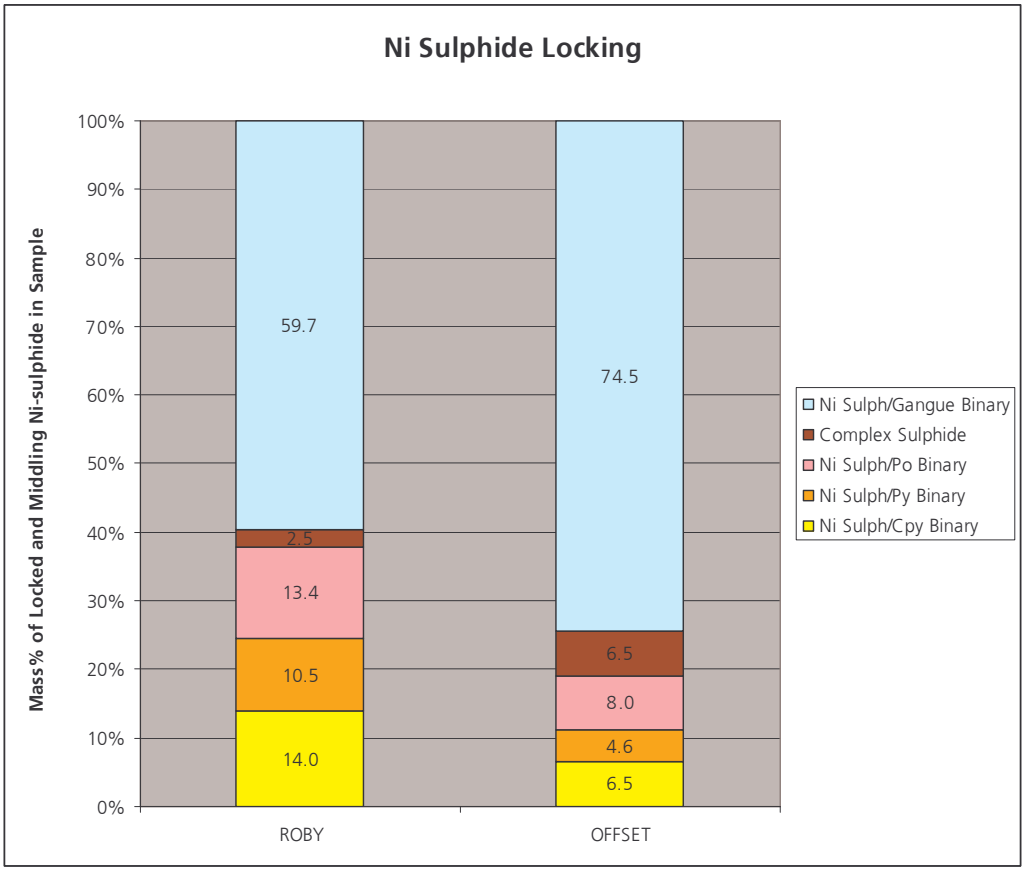

Figure 6: Ni-Sulphide Locking Characteristics in Roby and Offset Feed

Data indicates that a higher proportion of locked and middling Ni-sulphide is associated with non-sulphide gangue in the Offset Zone. This suggests that Ni losses may be marginally higher in the Offset Zone due to this textural relationship.

3.4- PGM Mineralogy

In order to help improve PGM counting statistics for mineralogical analysis of feeds, up to 6 replicate polished sections were measured. Given the low grade nature of the feed sample, limited PGMs were analysed in the feeds. The data presented is based on limited PGM observations which should be considered when interpreting results.

PGM phases identified within the feed can be categorised as Pd-sulphides (vysotskite), Pd-tellurides (kotulskite, merenskyite), Pd-antimonides (mertieite), Pd-arsenides (arsenopaladinite) with lesser amounts of Pt-arsenides (sperrylite) and Au/Ag species (electrum). Due to overlaps in the X-ray spectra, Pd antimonides and tellurides cannot be easily isolated and are grouped together. Table 1 summarises the PGM occurrences by grouping based on the mass % of total PGM in each sample. Table 2 breaks down the average PGM grain size and PGM occurrences based on mass% in each fraction within broad speciation categories.

Table 1: PGM Occurrence by Category for Roby and Offset

Mass (%) PGM Norm 100%	Zone Composite	
PGM Groupings	**Offset**	**Roby**
Au/Ag	4%	1%
Pd Tellurides-Antimonides	50%	56%
Pd-Arsenides/Other	0%	0%
Pd Sulphides	41%	39%
Pt Arsenides	3%	0%
Pt PGM	1%	4%
Other Te/Bi	1%	0%
# of PGM Grains	**53**	**63**

Table 2: PGM Occurrence and Average Grain Size Based on Mass% Total PGM/Fraction

ROBY Feed PGM Distribution								
Name	**Combined**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**
Average PGM Grain Size (µm)	7	5	9	10	4	20	10	6
PGM Distribution Mass%								
% Au/Ag Alloys	2.4	4.7	3.4	3.0	-	0.1	-	-
% Pd PGM	88.1	66.2	96.6	95.9	61.0	99.7	100.0	100.0
% Pt PGM	9.5	29.1	-	1.1	39.0	0.2	-	-
% Other Te/Bi	-	-	-	-	-	-	-	-
	100.0	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*

OFFSET Feed PGM Distribution								
Name	**Combined**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**
Average PGM Grain Size (µm)	8	4	9	13	16	9	7	5
PGM Distribution Mass%								
% Au/Ag Alloys	3.8	-	20.0	0.3	-	-	3.0	-
% Pd PGM	93.1	93.8	74.0	99.2	100.0	61.5	90.1	81.2
% Pt PGM	3.1	-	6.0	0.6	-	38.5	6.9	18.8
% Other Te/Bi	0.0	6.2	-	-	-	-	-	-
	100.0	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*

Table 1 indicates that the majority of PGMs by mass are Pd-tellurides and Pd antomonides followed by Pd sulphides. Average PGM grain size data per fraction indicates that the PGM population is very fine grained and the combined average for both Roby and Offset zone are similar at 7 µm and 8 µm respectively.

A PGM grain size distribution is presented in Figure 7. Although based on limited particle statistics, grain size distribution data supports the average PGM grain size data which indicates a slightly coarser PGM grain size in the Offset sample. The Offset sample also exhibits a bimodal PGM population however; in both Roby and Offset composites the PGMs are almost exclusively less than 25 µm indicating limited opportunity for staged grinding strategies.

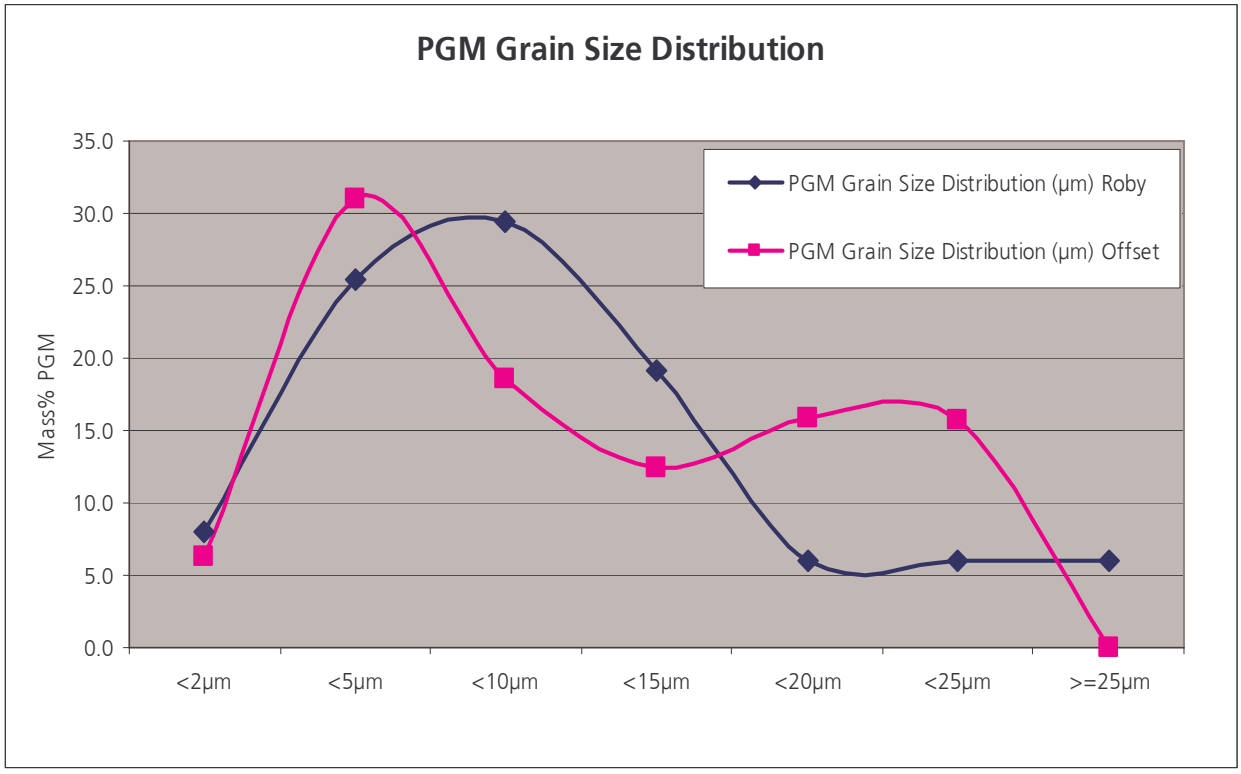

Figure 7: PGM Grain Size Distribution Based on Mass% PGM

PGM liberation analysis is limited by the number of particles observed in the feed; therefore size-by-size liberation data for PGMs are not presented here. Table 3 summarises the total PGM liberation of each sample broken down by mineral association and locking characteristics which provide insight into general PGM textures which could affect processing of these ores. Figure 8 shows the corresponding particle images of the PGM population measured in each feed composite. Particle size fraction analyses are completed at different magnifications and particle images are not to scale.

Table 3: PGM Liberation and Locking/Association Based on Mass% PGM

PGM Occurrence Mode	Roby		Offset	
	Mass % PGM	# Particles	Mass % PGM	# Particles
Liberated PGM	**72.2**	25	**71.6**	18
Locked with Cpy	0.0	1	0.0	0
Locked with Pn	0.0	0	0.0	0
Locked in Gangue	11.6	30	9.9	27
Complex Locked with Gangue and Sulphide	0.3	2	0.0	2
Other	15.8	5	18.5	6
	100.0	63	100.0	53

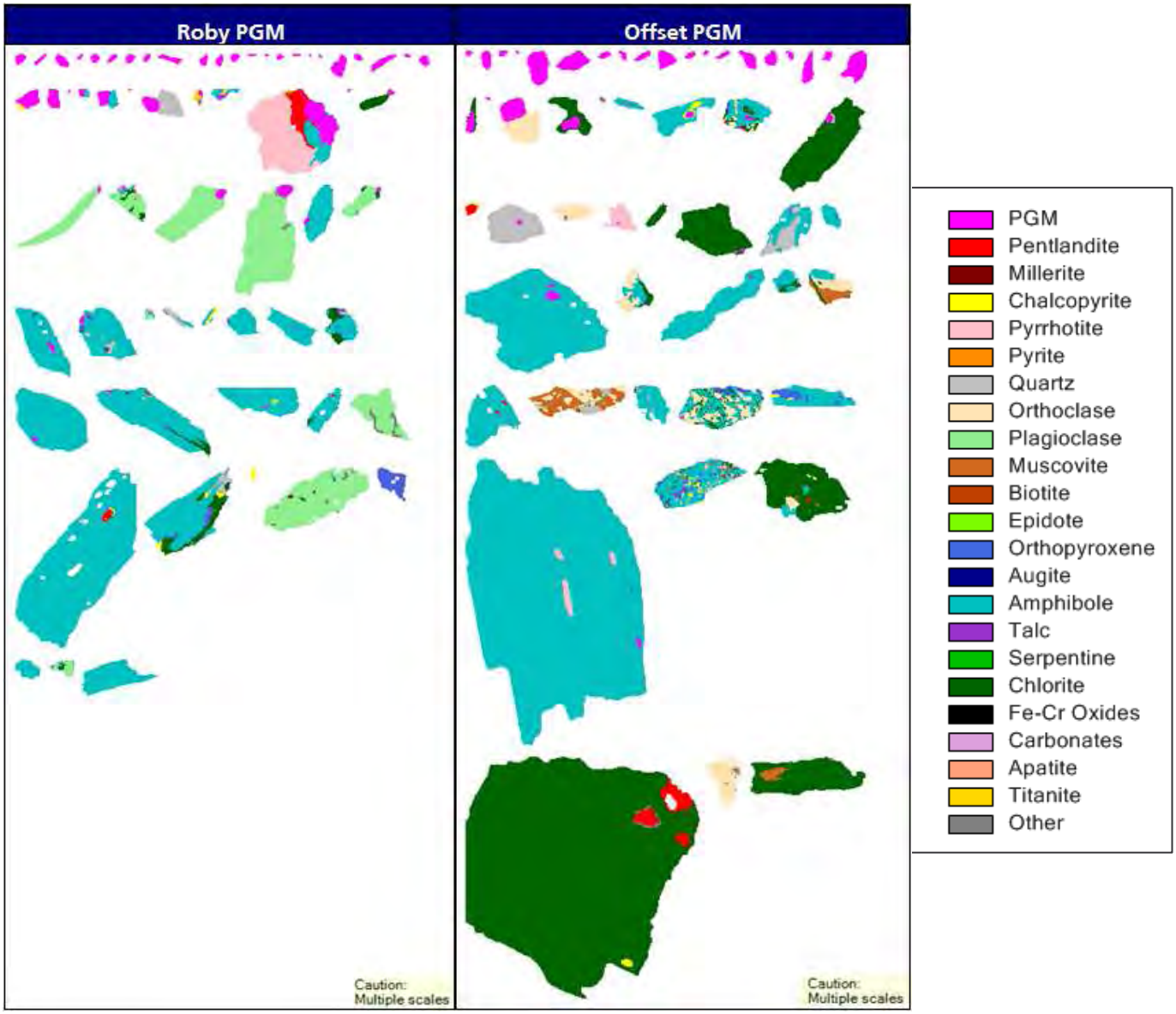

Figure 8: PGM Particle Images Sorted on Area % PGM.

Although based on limited particles, total PGM liberation data indicates that approximately 70% of PGM by mass in both Roby and Offset are liberated at a target grind size of 80% passing 75 µm.

Association and locking data indicates that the majority of remaining PGMs which are not liberated are largely locked with non sulphide gangue or other complex gangue dominated associations.

3.5- Mineral Composition Analysis

Mineral composition analysis was completed via Electron Probe Micro Analysis (EPMA) in order to determine the mineral chemistry of sulphides, silicate gangue, and PGM phases, assess solid solution levels and assist in mineral identification in conjunction with QEMSCAN species identification.

A subsample of polished sections was selected from Roby and Offset feeds, as well as from the Offset Locked Cycle Test final concentrate in order to locate required phases for analysis. In all cases, there is no clear chemical variation between mineral phases from Roby or Offset. Therefore, mineral composition analyses were averaged together to produce a more robust average to apply to the database in order to complete QEMSCAN assay reconciliation and deportment calculations.

A summary of sulphide, silicate and PGM mineral compositions are presented in Tables 4, 5 and 6 respectively. Data is presented as weight% and oxygen is a calculated value. Silicate analyses do not include the hydrous component resulting in depressed totals.

Table 4: Mean wt% EPMA Mineral Composition Analyses for Sulphides

	S	*Fe*	*Co*	*Ni*	*Cu*	*Pd*	*Pt*	*Total*	n=
Chalcopyrite	34.76	29.57	-0.01	0.01	34.66	0.00	-0.02	99.02	22
Pyrrhotite	39.53	58.31	-0.01	**0.53**	0.01	0.00	0.03	98.41	3
Pyrite	53.06	45.56	0.61	**0.08**	0.03	0.00	0.03	99.38	15
Millerite	34.96	1.37	0.44	62.75	0.00	0.00	0.01	99.55	6
Pentlandite	33.30	28.51	0.79	37.00	0.01	**0.12**	0.03	99.79	12

Table 5: Mean wt% EPMA Mineral Composition Analyses for Silicate Gangue Phases

	Na	Mg	Al	Si	K	Ca	Ti	Cr	Mn	Fe	Ni	O	Total	n=
Actinolite	0.28	11.23	1.76	25.49	0.04	8.86	0.06	0.07	0.17	6.85	**0.07**	43.75	98.6	31
Hornblende	0.92	7.85	6.33	20.82	1.03	7.76	0.32	0.03	0.09	10.48	**0.07**	41.42	97.1	1
OPX-Enstatite-Ferrosilite	0.09	13.68	0.98	25.51	0.01	1.89	0.07	0.07	0.30	12.97	**0.08**	43.63	99.3	22
Talc	0.06	15.80	0.85	27.44	0.17	0.47	0.04	0.08	0.05	5.98	**0.17**	44.50	95.6	9
Chlorite	0.01	14.21	11.14	13.13	0.07	0.04	0.03	0.02	0.12	10.89	**0.13**	37.48	87.3	33
Plagioclase	2.35	0.00	17.13	23.11	0.03	10.99	0.00	0.00	0.01	0.11	0.01	46.82	100.6	9
Albite-Plag	8.33	0.01	11.43	30.95	0.10	1.54	0.00	0.00	-0.01	0.06	0.01	48.99	101.4	3
Muscovite	0.14	0.36	18.26	22.12	10.01	0.14	0.01	0.00	0.01	1.07	0.01	44.15	96.3	4
Biotite	0.05	9.32	9.11	16.80	6.72	0.17	0.68	0.05	0.09	12.14	**0.06**	38.83	94.0	5
Epidote	0.01	2.84	12.33	16.66	0.01	13.12	0.00	0.02	0.14	10.41	0.01	40.10	95.7	2

Table 6: Mean wt% EPMA Mineral Composition Analyses for PGM Phases

	S	*Fe*	*Ni*	*Cu*	*Pd*	*Pt*	*As*	*Sb*	*Te*	*Bi*	*Total*	n=
Vysotskite	22.32	0.56	4.26	0.13	66.87	3.14	-0.04	0.01	0.00	-0.05	97.3	4
Mertieite	0.01	0.24	0.07	0.13	72.79	0.09	7.91	15.05	1.86	-0.18	98.2	3
Kotulskite	0.02	0.44	0.05	0.15	41.66	0.30	-0.09	0.01	39.56	16.24	98.4	5
Palladoarsenide	0.01	0.33	0.08	0.16	73.74	0.05	26.03	0.33	0.80	-0.23	101.5	1
Merenskyite	0.01	0.37	0.11	0.08	27.20	0.59	-0.11	0.01	52.88	17.20	98.5	1
Sperrylite	0.25	0.41	0.07	0.17	-0.01	55.17	45.20	0.04	-0.02	-0.05	101.3	1

Sulphide analyses indicate solid solution levels of Ni in pyrrhotite and pyrite as well as low levels of Pd in solution within pentlandite. Silicate analyses indicate low levels of solid solution Ni in various Fe-Mg gangue phases including talc, chlorite, orthopyroxene, amphiboles and biotite. Limited PGM analyses were completed due to limited PGM grains and grain size constraints; however validation of dominant PGM species was completed.

3.6- Ni Deportment

Due to the potential impact of improved Ni recoveries on mine economics and the apparent low total Ni recoveries typically achieved during processing, compositional analyses by EPMA have been combined with the modal analysis produced in QEMSCAN to determine the overall Ni deportment by species for each composite sample. The deportment results are presented in Figure 9.

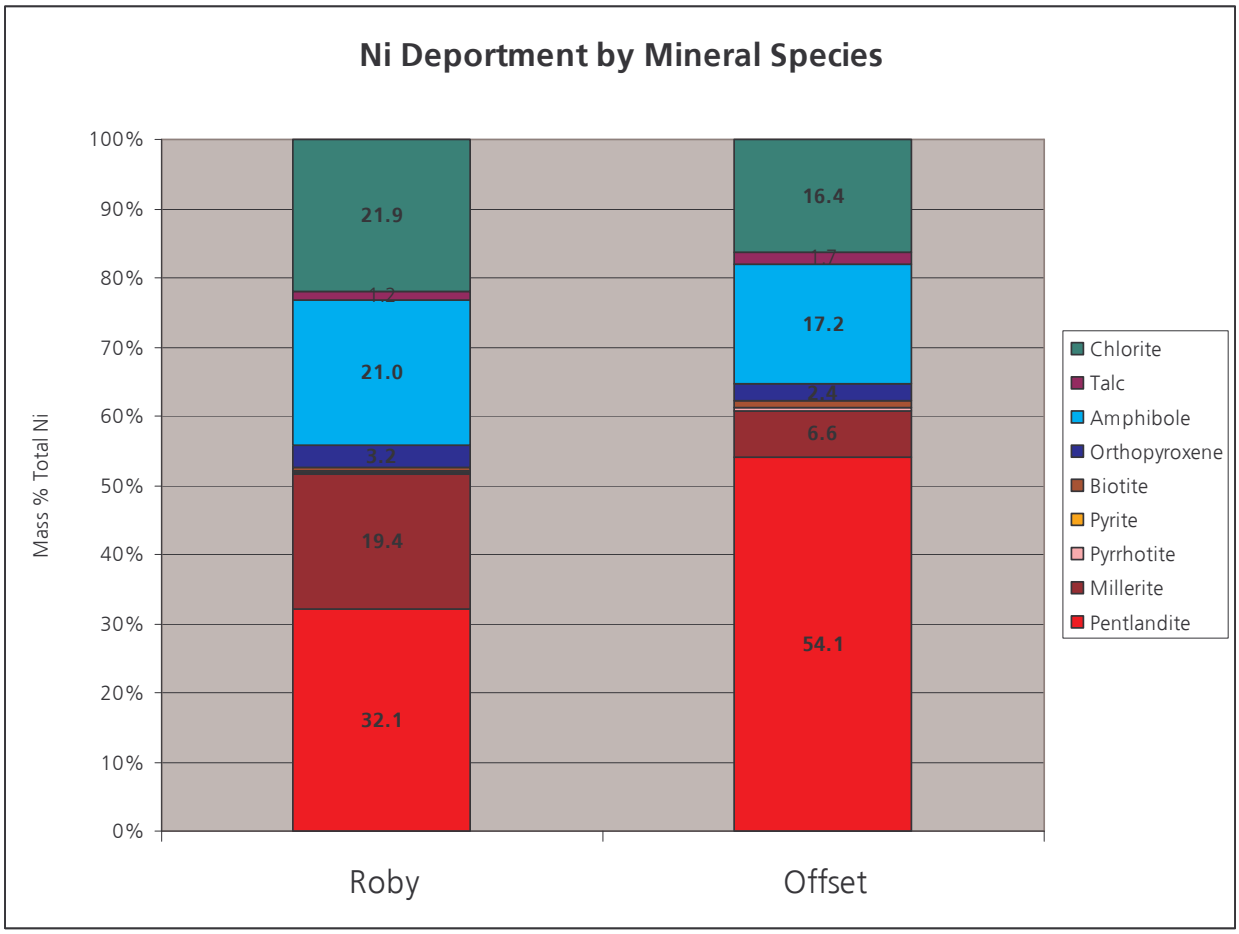

Figure 9: Ni Deportment- Total Mass% Ni Deported by Mineral Species

Ni deportment data clearly indicates that for Roby and Offset composites, only 52% and 60% of total Ni is carried by Ni sulphides respectively. The remaining nickel occurs predominantly in Fe-Mg gangue species as indicated by low solid solution levels of Ni identified by EPMA. Ni recovery is heavily impact by gangue rejection in the circuit resulting in lower realised Ni recoveries.

4.0 – Metallurgical Testwork Results

4.1- Feed Preparation and Head Assay

Approximately 200 kg of drillcore each from the LDI Roby and Offset zone was received by XPS in February 2010. A fifteen (15) kg sample was extracted from Roby and Offset zone and sent to SGS for ore hardness testing. The remaining drillcore was stage crushed down to 100% passing 10 mesh (1.7mm) and underwent *Odds and Evens Blending* to produce two homogenised sample lots. These lots were split into representative 2.5kg charges.

The following table summarises the feed grades for the LDI Roby and Offset composite:

Table 7: 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	12.20	0.37	0.31	6.12
Offset Zone	0.14	0.10	0.28	11.60	0.29	0.35	5.52

4.2- Lab Grind Time Calibration

In order to establish lab grind times for different primary grind targets, the 2.5 kg flotation charges were ground at the lab rod mill at 15, 21, 26, 36 and 43 minutes. The ground products were sized and the cumulative passing vs. size curves at different grind times were plotted (Figure 10).

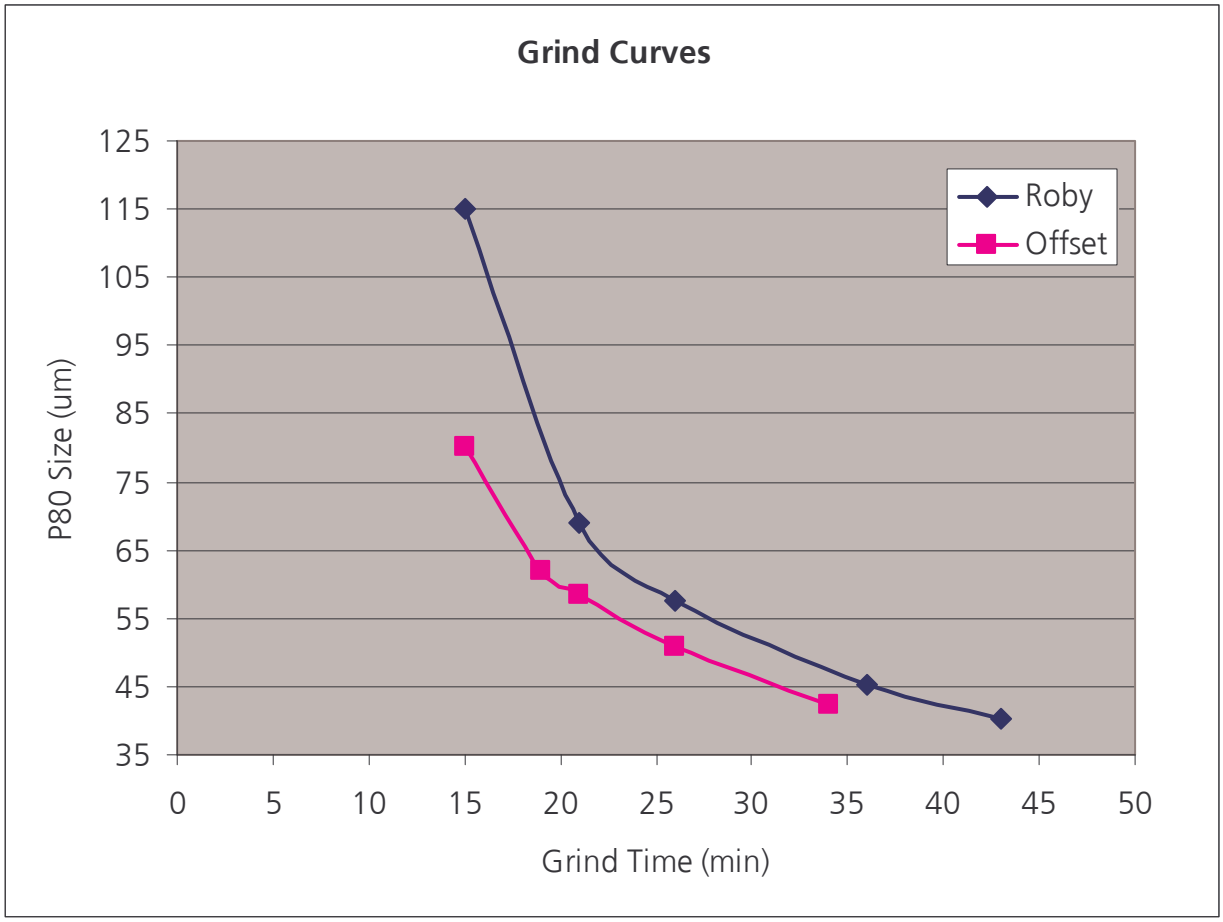

Figure 10: Lab Grind Time Determination

The grind time (at the lab rod mill) relationship curve vs. the P80 size for the 2.5 kg flotation charges was established as seen in Table 8.

Table 8: Lab Rod Mill Grind Curve Calibration

Target Grind P_{80} Size (µm)	Lab Rod Mill Grind Time (min)	
	Roby	Offset
75	20	16.5
53	28	25
38	45	37

4.3- Roby Composite Metallurgical Results

4.3.1 – Rougher Flotation Tests

Primary Grind Sensitivity

A series of rougher-scavenger flotation tests were conducted at different primary grind sizes to define the optimal primary grind size in an aim to achieve better overall rougher-scavenger recovery. The detailed metallurgical mass balances are shown in the Appendix. The comparison of Pd grade/recovery at different primary grind sizes is displayed in Figure 11.

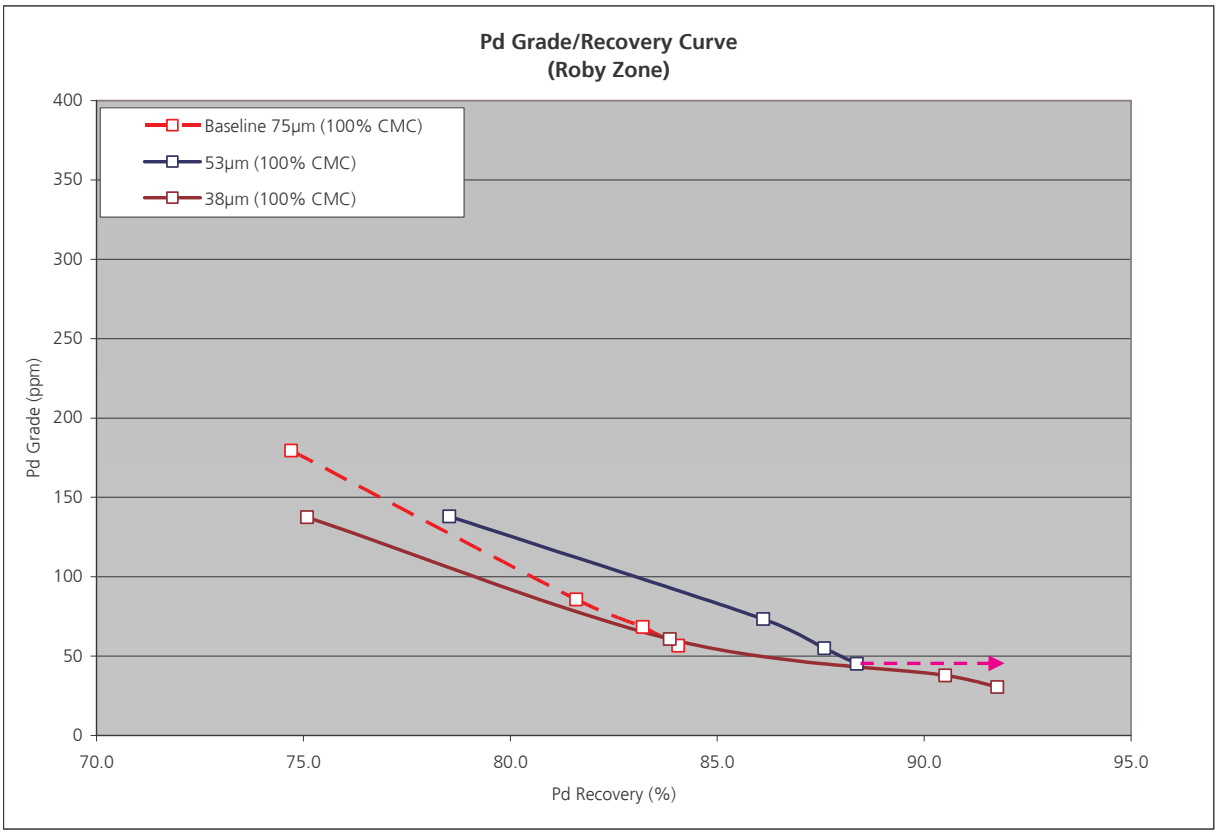

Figure 11: Rougher-Scavenger Pd Grade/Recovery vs. Primary Grind Size.

From the test work, it is clear that the finer the primary grinding size (38 µm), the better the overall rougher-scavenger Pd recovery performance. This is consistent with mineralogical data which indicates the fine nature of the PGM population.

Effect of CMC Dosage on Rougher Pd Recovery

A series of rougher-scavenger flotation tests were conducted to investigate the effect of CMC (*Depramin C- MgO depressant*) addition on overall Pd rougher-scavenger recovery. A baseline of 75 µm was used as a primary grind in order to isolate the CMC dosage rate as the key variable. The detailed metallurgical mass balances are shown in the Appendix. The comparison of Pd grade/recovery at 2 different CMC dosages is displayed in Figure 12.

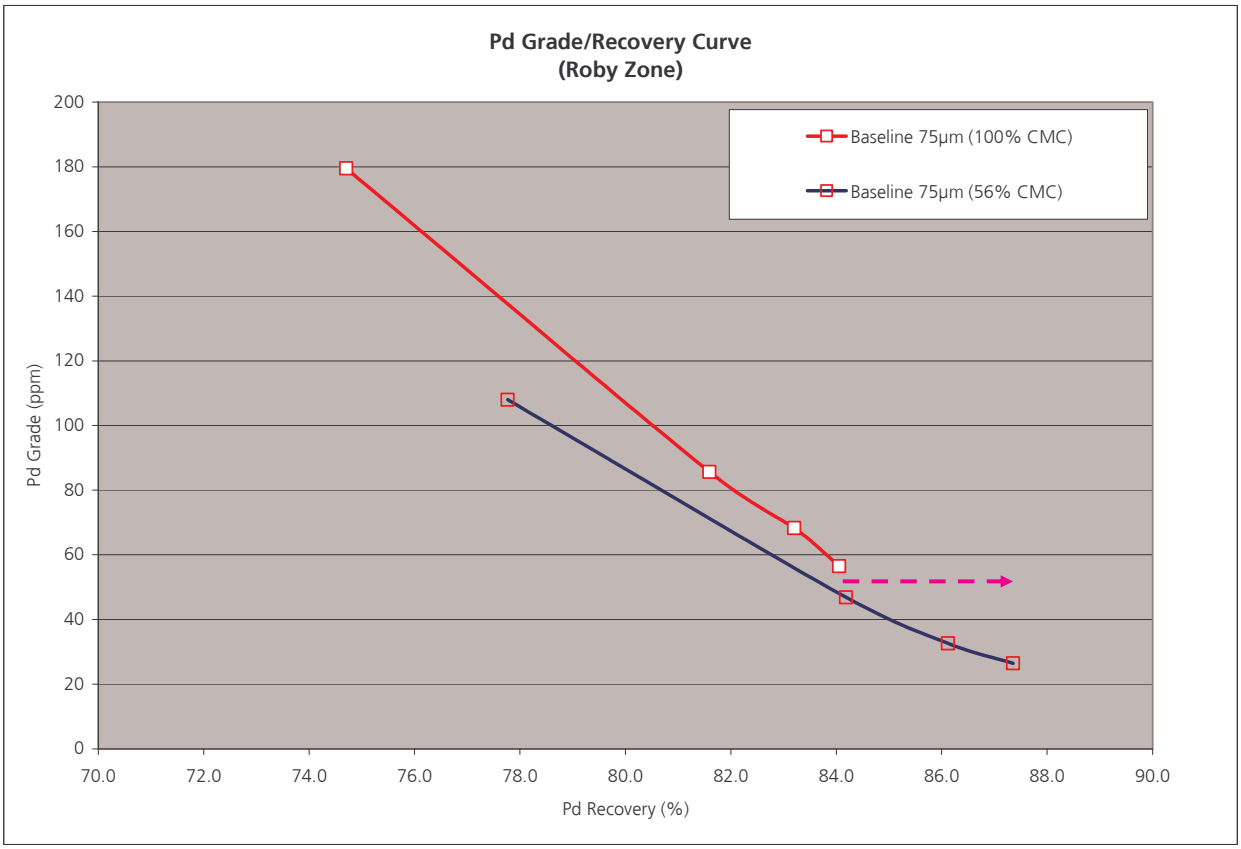

Figure 12: Rougher-Scavenger Pd Grade/Recovery vs. CMC Dosage at Primary Grind of 75µm.

It was noted that CMC addition can have a negative impact on Pd recovery. Figure 12 shows that at 100% of the initially prescribed CMC dosage (~172 g/t addition) compared to 56% of the CMC dosage (~92 g/t addition) that an increase in Pd rougher recovery is observed at lower CMC dosages. Figure 13 also highlights a similar trend of improved recovery with reduced CMC dosage at the finer primary grind (53 µm). The overall increase in Pd recovery is not as significant at the 53 µm grind as compared to the 75 µm grind at 0.7% vs. 3.3% overall Pd recovery improvement respectively.

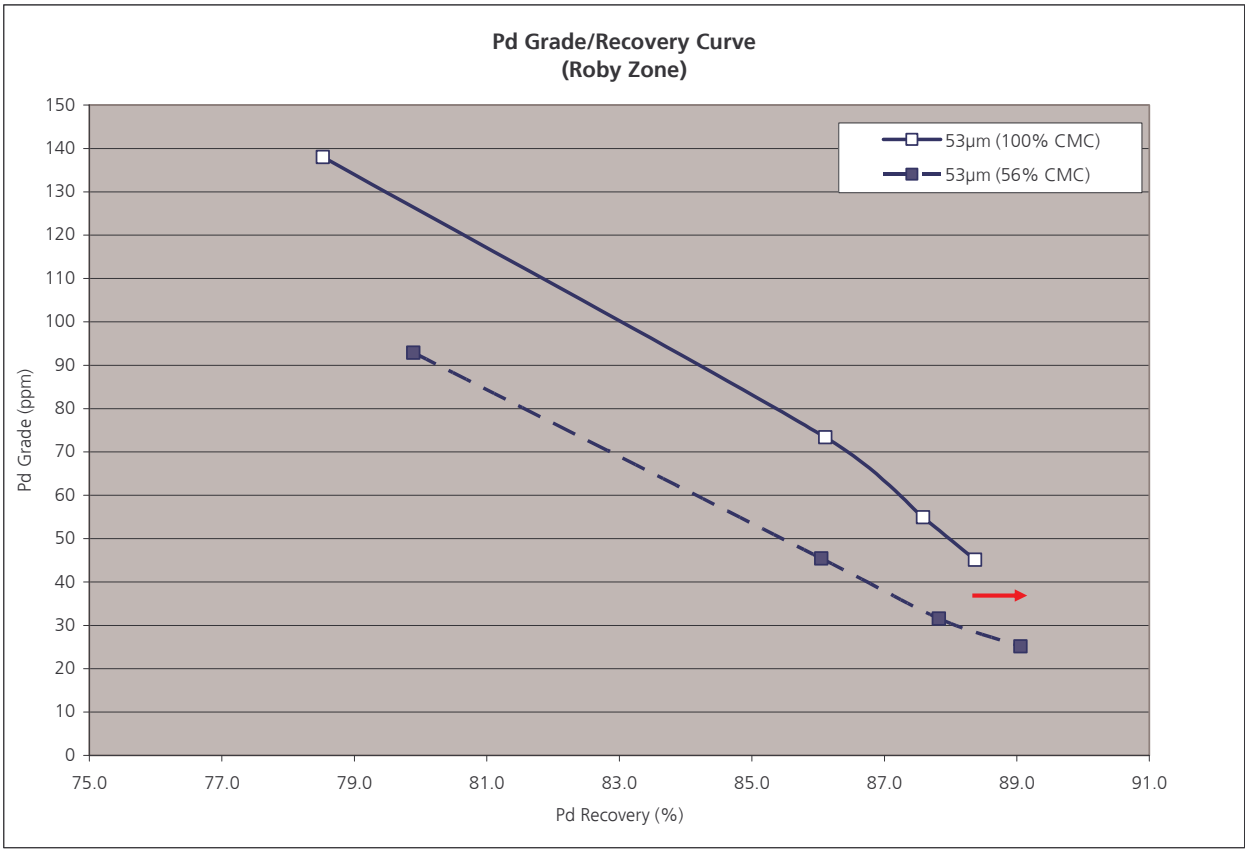

Figure 13: Rougher-Scavenger Pd Grade/Recovery vs. CMC dosage at Primary Grind of 53µm.

Effect of Aero 3477 on Rougher Pd Recovery

A series of rougher-scavenger flotation tests were conducted to investigate the effect of Aero 3477 PGM promoter addition on overall Pd rougher-scavenger recovery. A baseline of 75 µm was used as the primary grind. The detailed metallurgical mass balances are shown in the Appendix. The baseline comparison of Pd grade/recovery vs. the addition of Aero 3477 is displayed in Figure 14.

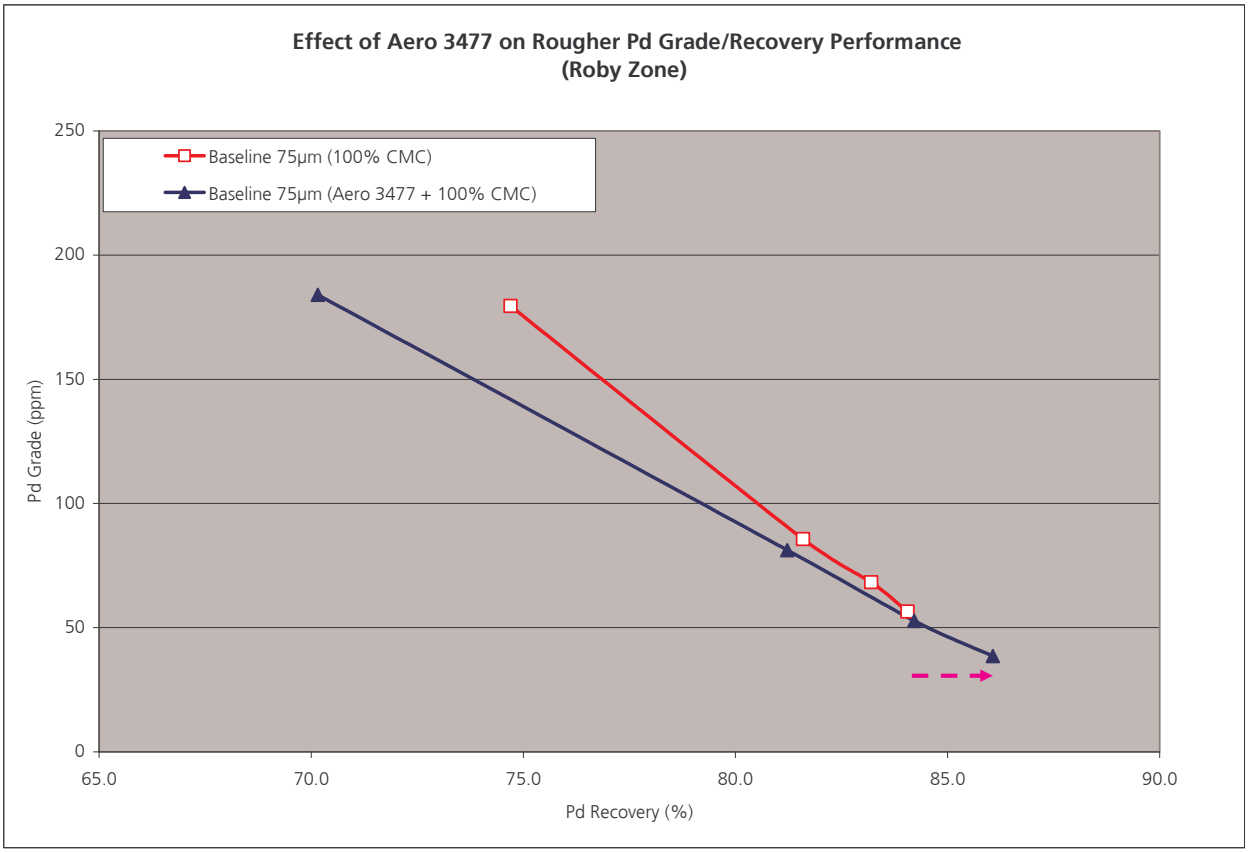

Figure 14: Rougher-Scavenger Pd Grade/Recovery vs. CMC Dosage at Primary Grind of 75µm.

Aero 3477 addition was tested on a baseline ore at 75 µm and at a prescribed addition rate of 16 g/t (8 g/t in the secondary rougher, and 8 g/t in the scavenger circuit). Figure 14 highlights the addition of Aero 3477 resulting in a 2% improvement in overall Pd recovery in the rougher-scavenger circuit.

Fine Primary Grinding vs. Staged Grinding

A series of rougher-scavenger flotation tests were conducted to investigate the effect of staged grinding compared to fine primary grinding. Staged grinding can have beneficial effects in many cases as a result of optimising the liberation of minerals of interest in some ore types.

A final grind of 53 µm and 38 µm was targeted for these tests while maintaining the lower CMC dosage (92 g/t) for all tests. The detailed metallurgical mass balances are shown in the Appendix. The baseline comparisons of Pd grade/recovery at fine primary grinds vs. corresponding staged grinding to a final grind size of 53 µm and 38 µm are displayed in Figure 15 and Figure 16 respectively.

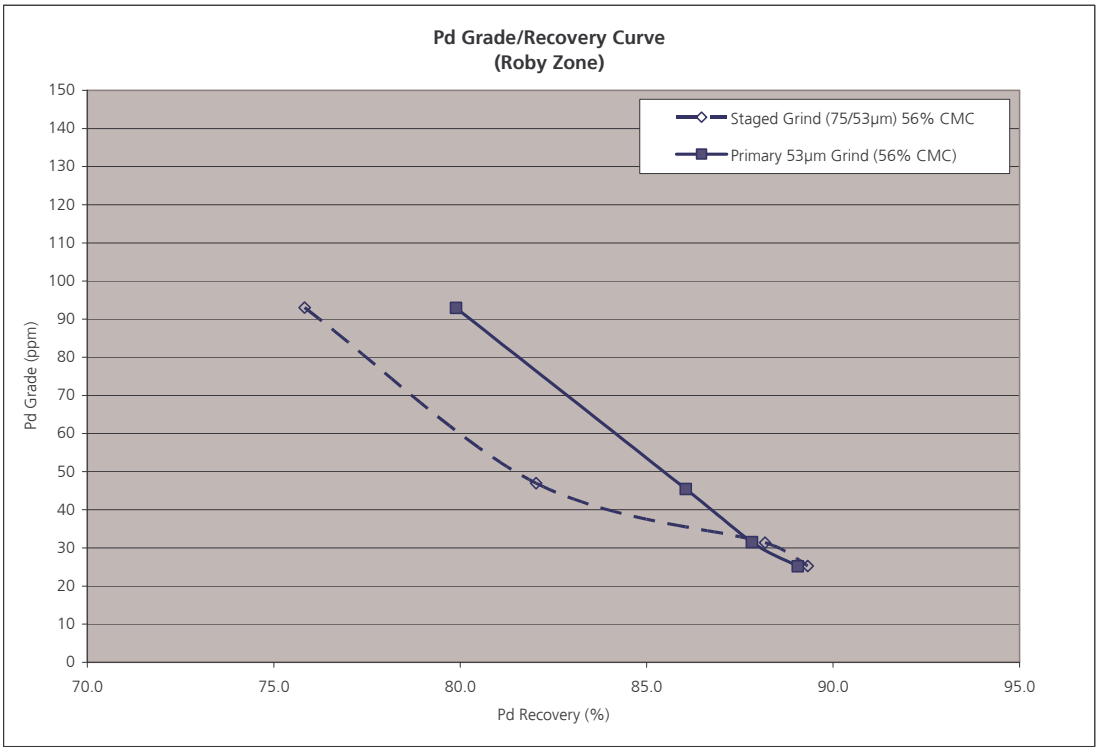

Figure 15: Rougher-Scavenger Pd Grade/Recovery at 53 µm Primary Grind vs. Staged Grind

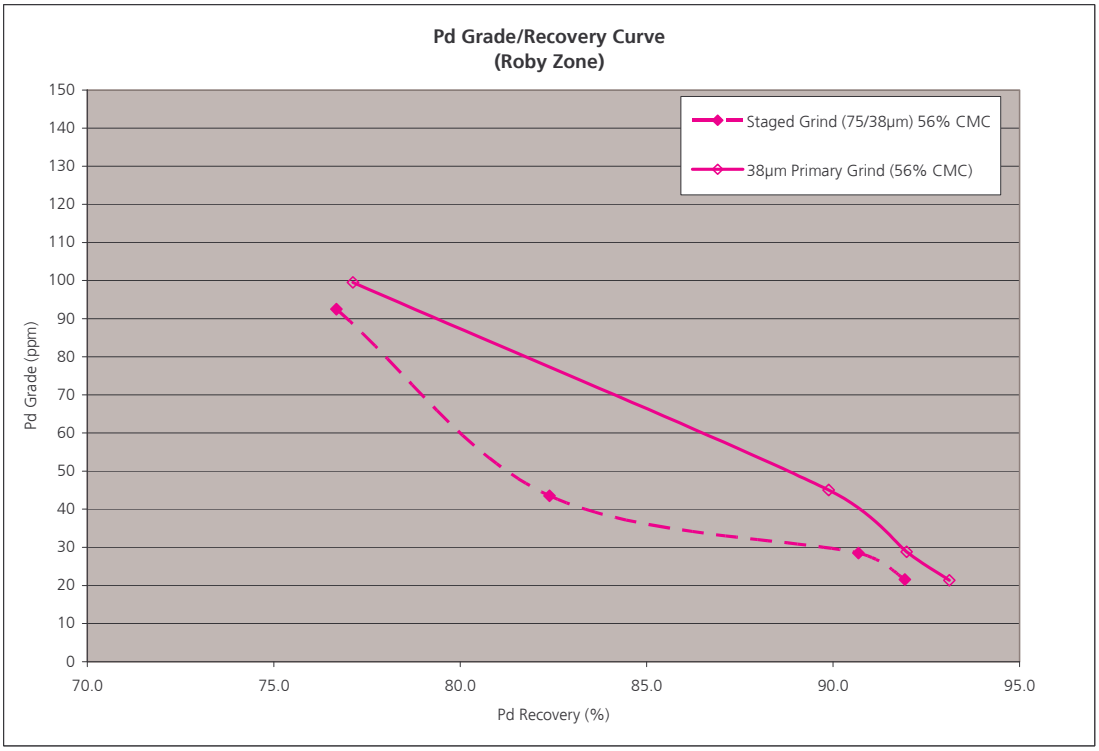

Figure 16: Rougher-Scavenger Pd Grade/Recovery at 38 µm Primary Grind vs. Staged Grind

Although results are consistent with previously reported data which indicates that finer grinding in general has added benefits, results from both target final grinds (53 µm and 38 µm) indicate no clear advantage to staged grinding over fine primary grinding with respect to Pd recovery in the rougher-scavenger circuit.

This result may be related to the mineralogical data which shows no clear coarse bimodal PGM grain size population which would benefit from coarser primary grinding followed by finer regrinding stages. The effect of a finer primary grind on the largely fine PGM population appears sufficient to effectively liberate the PGM as compared to a staged grinding approach.

Optimal Rougher Flotation Conditions

Based on results from multiple stages of rougher-scavenger flotation testing, the following parameters yielded superior results:

- Finer primary grind of 80% passing 38µm (no advantage to staged grinding is evident)
- 26 minutes of rougher flotation time
- 92 g/t CMC dosage with addition in the rougher by-pass (20 g/t), secondary rougher (36 g/t) and in the scavenger (36 g/t)
- 26 g/t PAX dosage with addition in the rougher by-pass (10.8 g/t), secondary rougher (8 g/t) and in the scavenger (8 g/t)
- 16 g/t Aero 3477 dosage with addition in the secondary rougher (8 g/t) and in the scavenger (8 g/t)
- 56 g/t MIBC with addition in the rougher by-pass (28 g/t), secondary rougher (14 g/t) and in the scavenger (14 g/t)

Figure 17 summarises the Pd grade/recovery results for various stages of grind size and reagent testing.

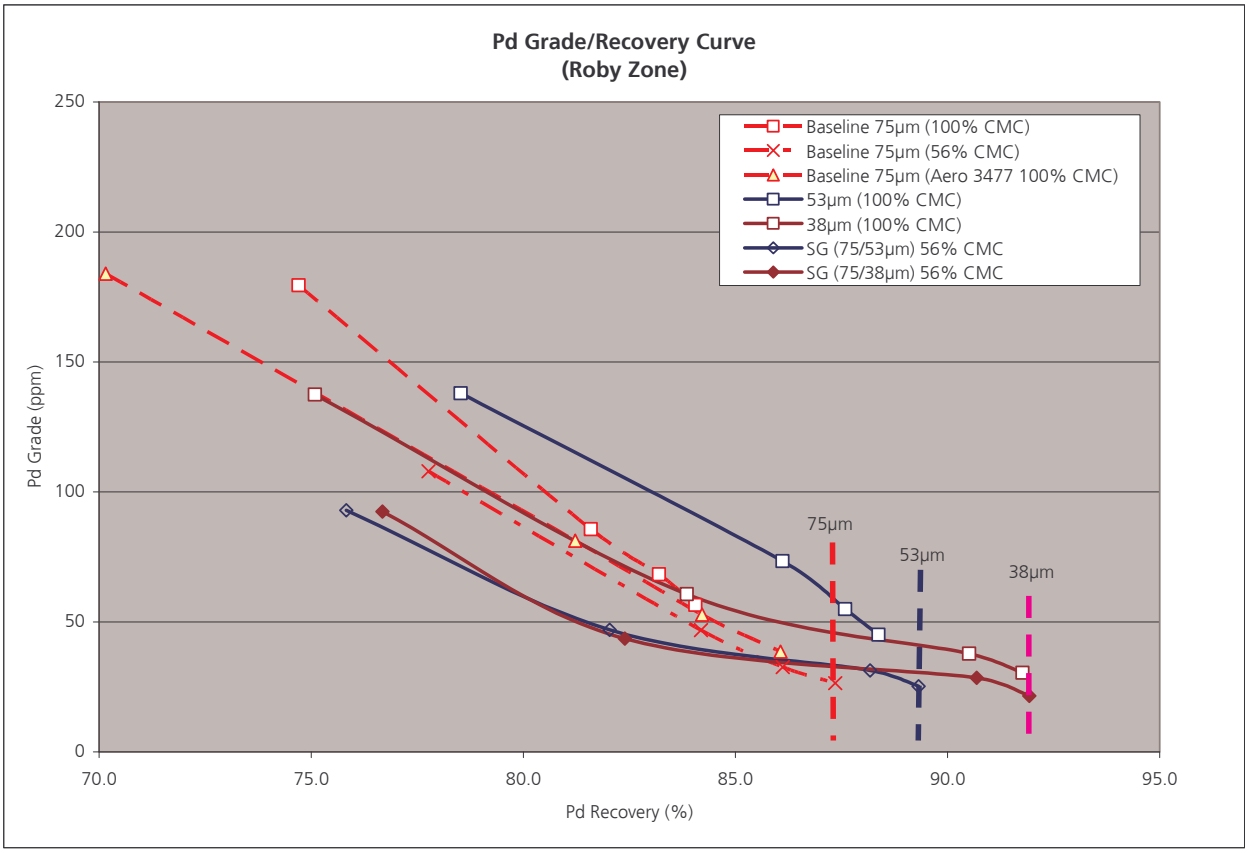

Figure 17: Rougher-Scavenger Pd Grade/Recovery at Various Grind and Reagent Conditions

4.3.2 – Open-Circuit Cleaning Flotation Tests

Fine Grind (38 µm) vs. Baseline (75 µm)

Phases of open circuit flotation testing were also completed in order to determine the optimal conditions and the effect of grind and reagent dosage on the cleaning circuit.

Open-circuit cleaner flotation tests were completed in order to investigate the effect of finer primary grinding on the cleaner circuit. The detailed metallurgical mass balances are shown in the Appendix. The comparison of Pd grade/recovery at different primary grind sizes is displayed in Figure 18.

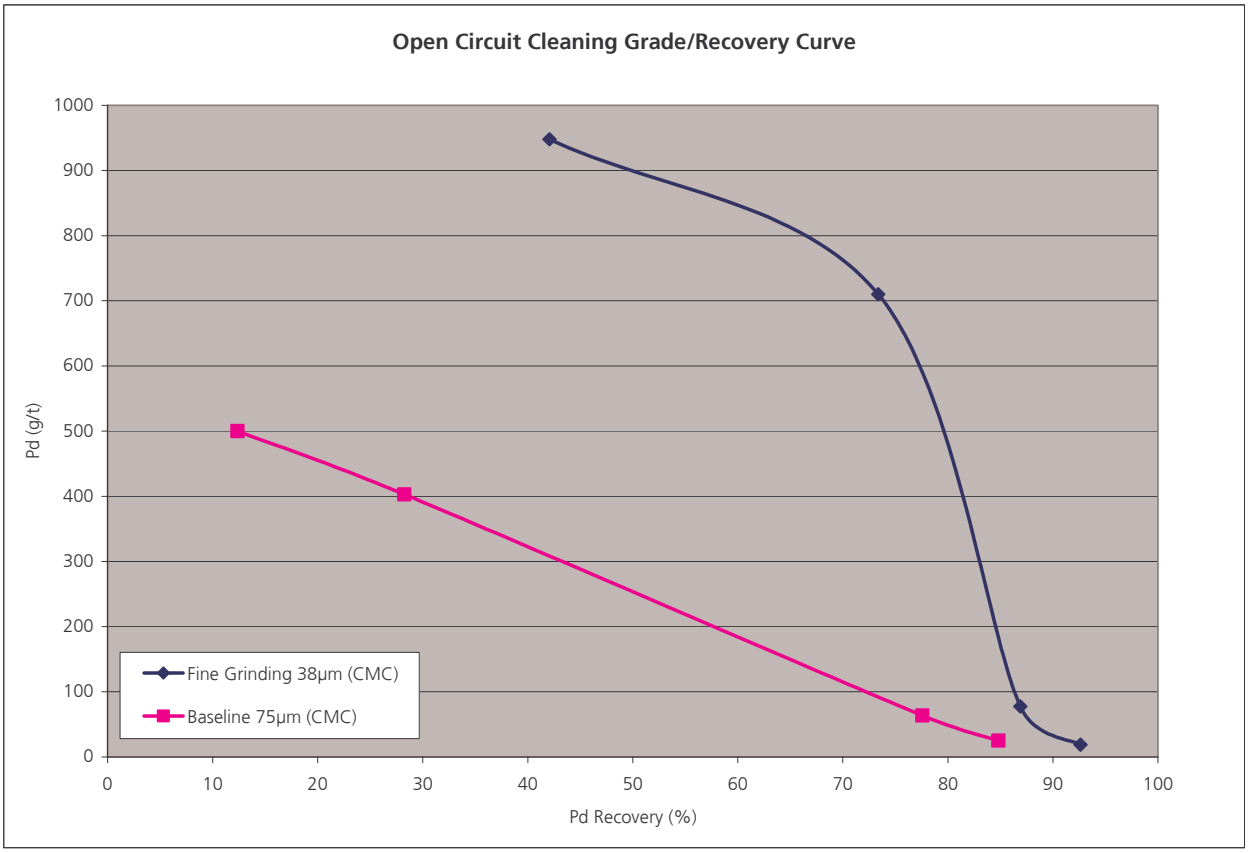

Figure 18: Cleaner Pd Grade/Recovery Comparing 38 µm Primary Grind vs. 75 µm Baseline Grind

Similar to the rougher-scavenger circuit, the positive effect of finer primary grinding is evident in the cleaners which indicate superior Pd grade and recovery performance. The fine primary grinding effectively liberates the PGM.

Fine Primary Grinding vs. Staged Grinding

A series of open-circuit flotation tests were conducted to investigate the effect of staged grinding compared to fine primary grinding.

A final grind of 38 µm was targeted for these tests while maintaining a consistent CMC dosage. The detailed metallurgical mass balances are shown in the Appendix. The baseline comparisons of Pd grade/recovery at a fine primary grind vs. corresponding staged grinding to a final grind size of 38 µm is displayed in Figure 19. Two fine grinding iterations were completed with different sizes of flotation charges (2.5 kg and 5 kg) in order to account for potential low mass pull issues in the cleaner.

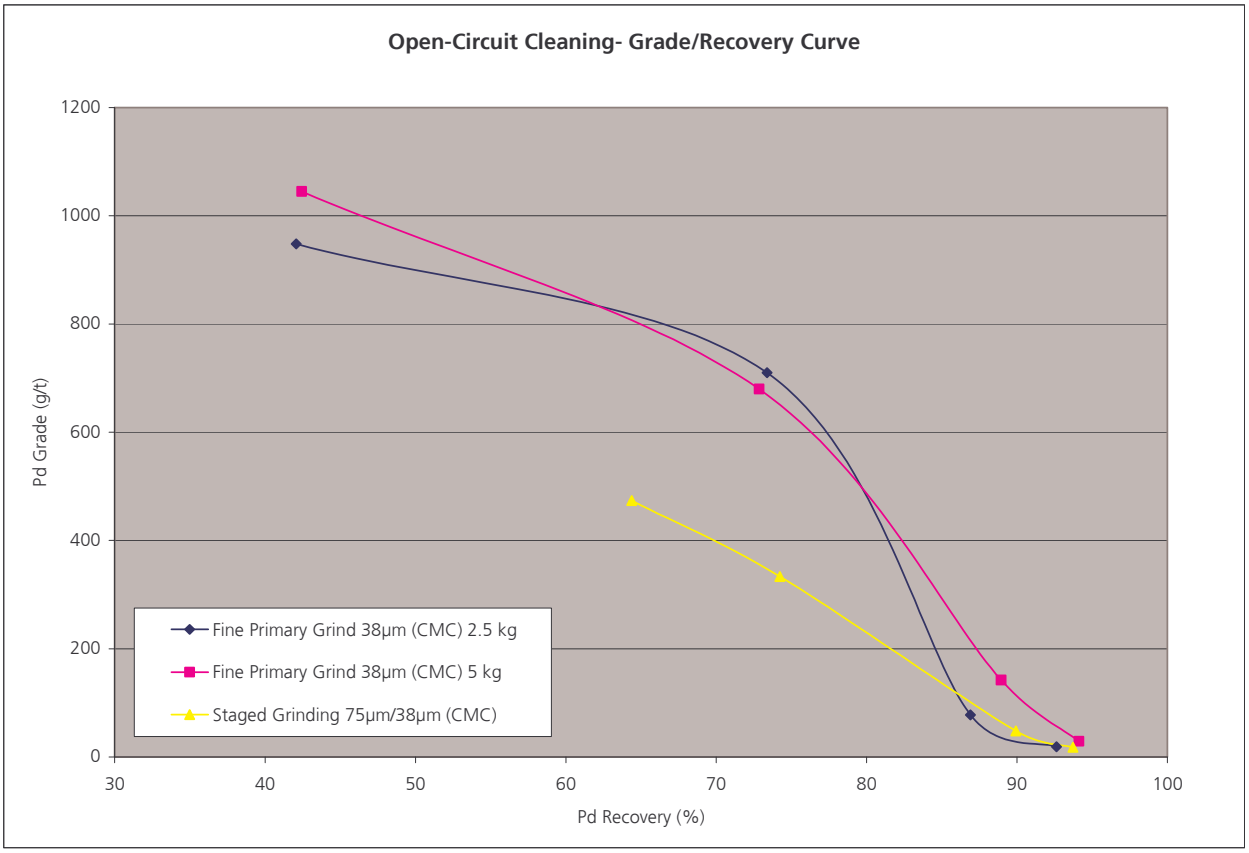

Figure 19: Cleaner Pd Grade/Recovery Comparing 38 µm Primary Grind vs. Staged Grind.

Similar to the response observed in the rougher-scavenger circuit, there is no clear advantage to staged grinding as compared to fine primary grinding for Pd grade/recovery in the cleaners.

MgO Depressant Tests

A series of open-circuit flotation tests were conducted to investigate the effect of various MgO depressants (CMC, Dextrin, Guar Gum) on the Pd grade/recovery response. For these tests, a fine primary grind of 38 µm was used. The detailed metallurgical mass balances are shown in the Appendix. The comparison of Pd grade/recovery with the different MgO depressants is displayed in Figure 20.

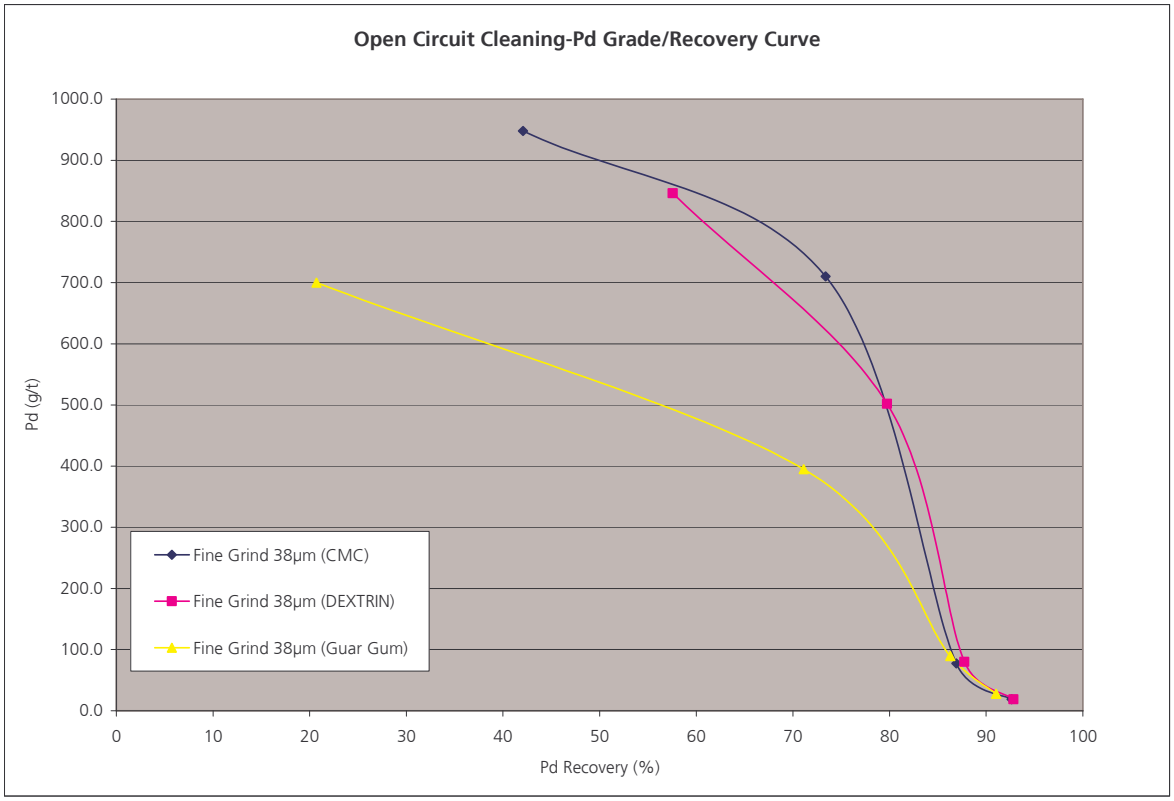

Figure 20: Cleaner Pd Grade/Recovery Comparing MgO Depressants

Results indicate that while Guar may be an effective MgO depressant, there is a negative overall effect on Pd grade/recovery performance compared to the other MgO depressants tested. This is likely due to the impact of Gaur on the flotation kinetics in the circuit which impacts the grade/recovery response given the low mass pulls. CMC and Dextrin can provide sufficient MgO depression with less impact on Pd grade/recovery. CMC exhibits only slightly better overall performance as compared to Dextrin.

4.3.3- Locked-Cycle Tests (LCT)

A series of Locked Cycle Tests were completed in order to validate and provide steady state metallurgical projections for the Roby ore composite.

A Locked Cycle Test is a repetitive batch test used to simulate a continuous circuit [2] or flowsheet. The flowsheet is repeated for a given number of cycles with the recirculated streams being re-introduced back into the circuit until steady state is reached. The number of cycles required to reach steady state generally varies from between 2 and 8 cycles, depending on the characteristics of the ore and the complexity of the flowsheet being tested. Steady state is deemed to be achieved when equivalent final product masses in the three cycles are consistent to within 5% of each other.

The Locked Cycle Test flowsheet used for this series of test, including the recirculating streams is shown in Figure 21.

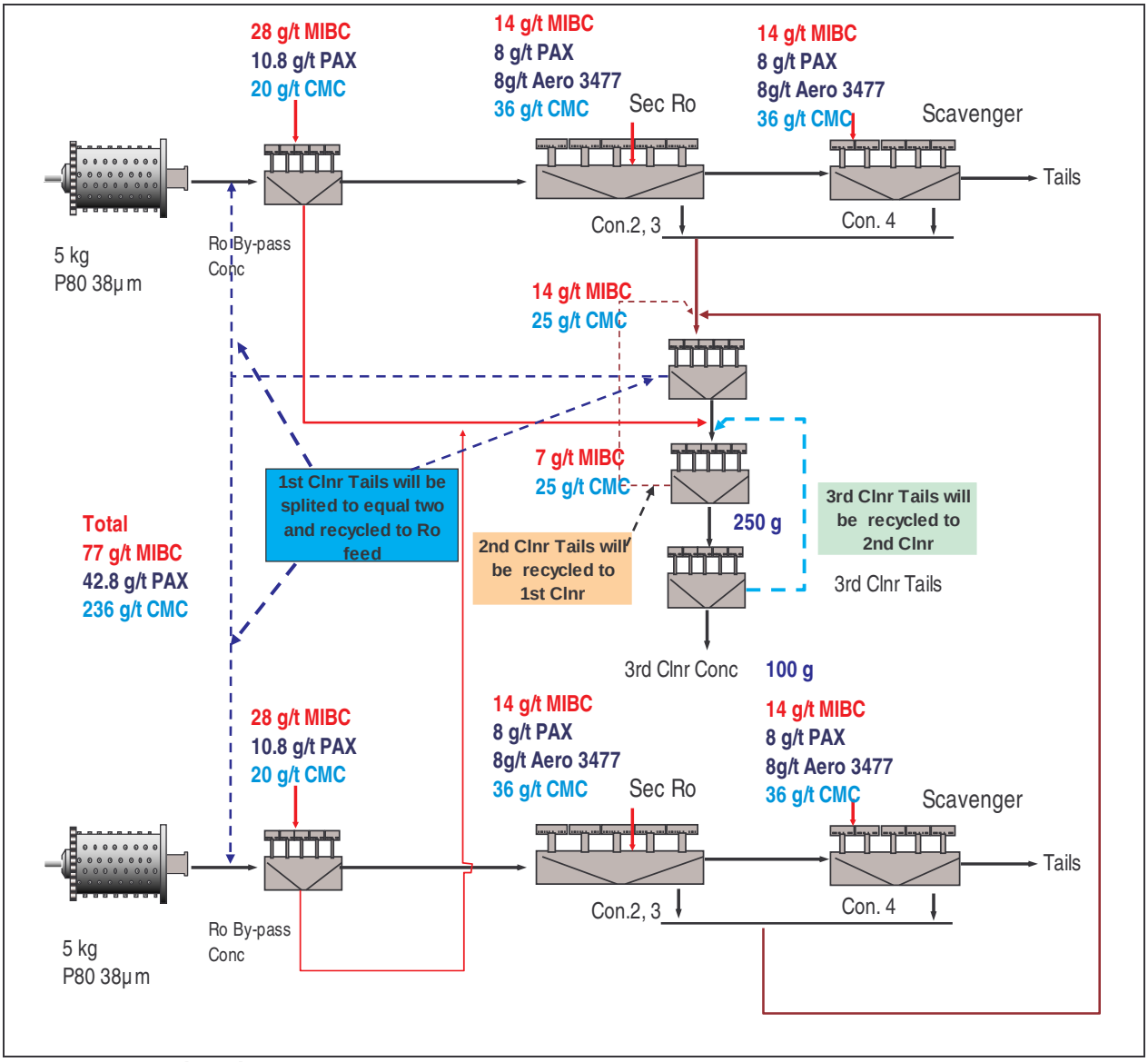

Figure 21: ***LCT Flowsheet.***

There are four accepted methods used to calculate the metal and mass balance for LCTs as described in the literature from the 2001 *Canadian Mineral Processors* (CMP) proceedings [2]. These are:

1. 2 product formula (balance on assays from final products)
2. SME Handbook (balance on final product weights and assays)
3. Concentration Production – Assumes unaccounted metal is in the tailings
4. Alternative – Assumes unaccounted metal is in the concentrate

It is important to note that no procedure is ideal for a test that has not reached steady state. However, for a test that has reached steady state, all procedures should produce a similar metallurgical projection. The four metallurgical projections for each test are summarised in Table 9 below.

Table 9: Metallurgical Projections Using the Four Methods for LCT – Roby Zone

2P Formula	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	10269.3	100.00	0.12	0.07	6.56	0.36	12.36	100.00	100.00	100.00	100.00	100.00
Concentrate	89.93	0.88	**4.81**	**7.13**	**640**	**31.46**	**8.92**	**36.58**	**87.44**	**88.07**	**80.61**	**0.63**
Tails	10179.3	99.12	0.07	0.01	0.77	0.07	12.40	63.42	12.56	11.93	19.39	99.37

SME Handbook	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	10269.3	100.00	0.11	0.07	5.82	0.31	12.37	100.00	100.00	100.00	100.00	100.00
Concentrate	81.1	0.79	**4.81**	**7.13**	**640.41**	**31.46**	**8.92**	**34.20**	**86.26**	**86.94**	**78.93**	**0.57**
Tails	10188.1	99.21	0.07	0.01	0.77	0.07	12.40	65.80	13.74	13.06	21.07	99.43

Concentrate Production	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	10269.3	100.00	0.12	0.07	6.56	0.36	12.36	100.00	100.00	100.00	100.00	100.00
Concentrate	81.1	0.79	**4.81**	**7.13**	**640.41**	**31.46**	**8.92**	**31.26**	**78.89**	**77.18**	**76.91**	**0.57**
Tails	10188.1	99.21	0.08	0.02	1.51	0.08	12.39	68.74	21.11	22.82	23.09	99.43

ALT	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	10269.3	100.00	0.12	0.07	6.56	0.36	12.36	100.00	100.00	100.00	100.00	100.00
Concentrate	81.1	0.79	**6.14**	**7.90**	**733.53**	**36.77**	**7.55**	**39.88**	**87.43**	**88.40**	**81.41**	**0.48**
Tails	10188.1	99.21	0.07	0.01	0.77	0.07	12.40	60.12	12.57	11.60	18.59	99.52

The average metallurgical projections for the Roby LCT are 35.9% Ni recovery, 86.1% Cu recovery, 86.2% Pd recovery and 80.5% Pt recovery at 5.2% Ni grade, 7.3% Cu grade and 664 g/t Pd and 8.58% MgO in concentrate. Au assays on the concentrate were not possible due to a shortage of sample mass and are estimated based on the tailings grade and assay head grade.

Table 10: Metallurgical Projections for Roby LCT#1 (Average)

Roby LCT Projections	Ni	Cu	Pd	Pt	Au	MgO
Grade (% or g/t)	5.15	7.32	664	32.8		8.58
Recovery (%)	35.9	86.1	86.2	78.6	87.5	0.57

** Au recovery is based on tails assays and assay head grade*

The LCT test for Roby Zone achieved an acceptable level of steady state as evidenced by the range of recovery projections produced and shown graphically in Figure 22. Pd recoveries ranged from 83.2% to 87.4% at between 640 g/t to 733 g/t Pd. More consistent results could perhaps be achieved with a higher number of cycles but were not warranted in this case.

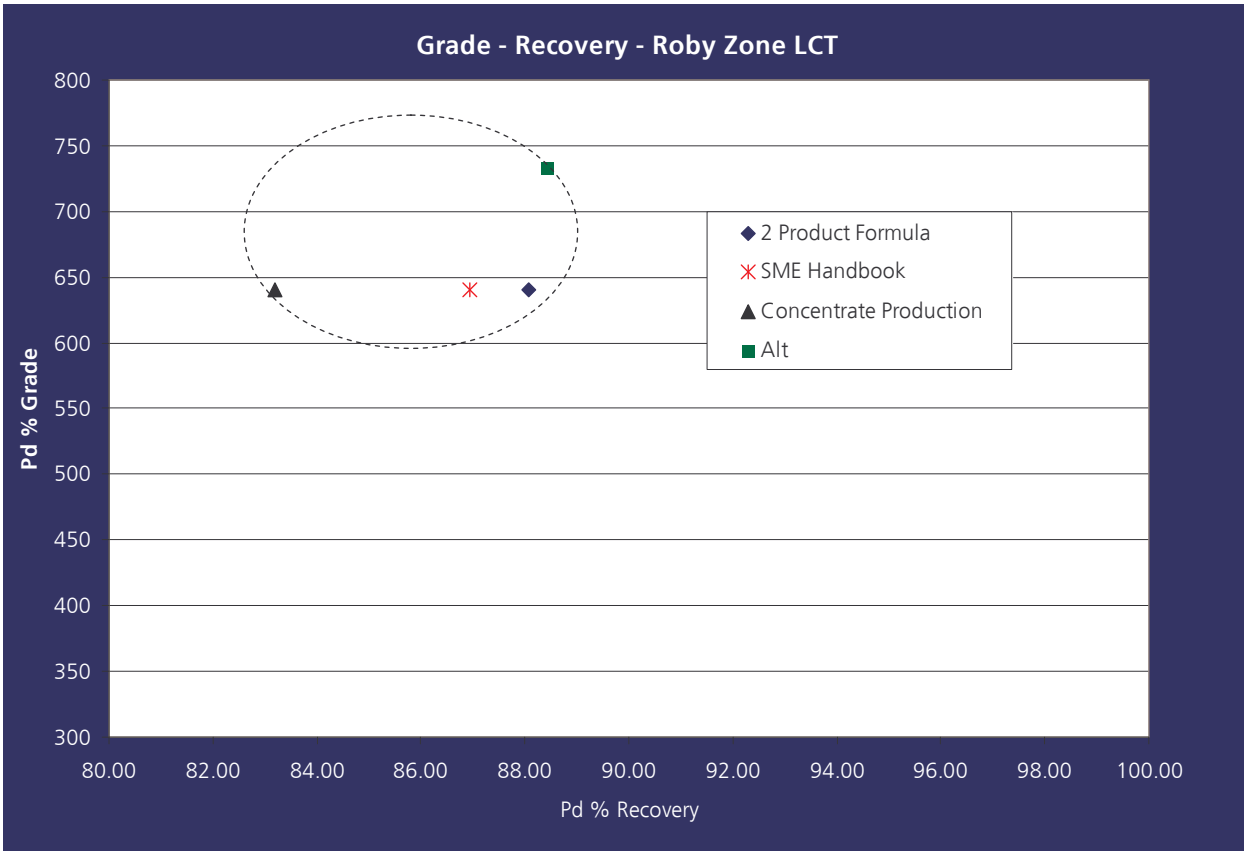

Figure 22: **LCT Pd Projections – Roby Zone**

Ni recoveries averaged 35.9% which is consistent with the Ni deportment data in previous sections which showed only 56% of the total Ni is carried in Ni sulphides of which only 76% of nickel is attributed to liberated Ni-sulphides.

It is important to note that the Roby LCT was the first one completed and that the CMC dosage was initially reduced to a lower level based on the indicated results of the rougher CMC dosage tests. The un-optimised CMC dosage resulted in higher circulating loads which increased the overall MgO grades in the concentrate to 8.58% MgO.

This highlights the impact of CMC optimisation on the LDI circuit as key to controlling MgO in concentrate. As an example, during the experimentation phases of open circuit cleaning, one test was completed which used 50 g/t CMC addition in the 1st cleaner and 36 g/t in the second cleaner which resulted in an MgO grade in the final concentrate of only 0.84% MgO. Therefore, data suggests that reagent optimisation can effectively control MgO levels below penalty thresholds while maintaining Pd recoveries. It is recommended that the potential impact of on-stream analysis for MgO be investigated in order to assist with CMC dosage optimisation in the plant.

4.4- Offset Composite Metallurgical Results

4.4.1 – Rougher Flotation Tests

Primary Grind Sensitivity Validation

A series of rougher-scavenger flotation tests were conducted on Offset ores at the baseline primary grind of 75 µm and at a finer primary grind of 38 µm to define the optimal primary grind size in an aim to achieve better overall rougher-scavenger recovery. Testing on the Offset ores was aimed at validating the key results from the more extensive Roby testing. The detailed metallurgical mass balances are shown in the Appendix. The comparison of Pd grade/recovery at the different primary grind sizes is displayed in Figure 23.

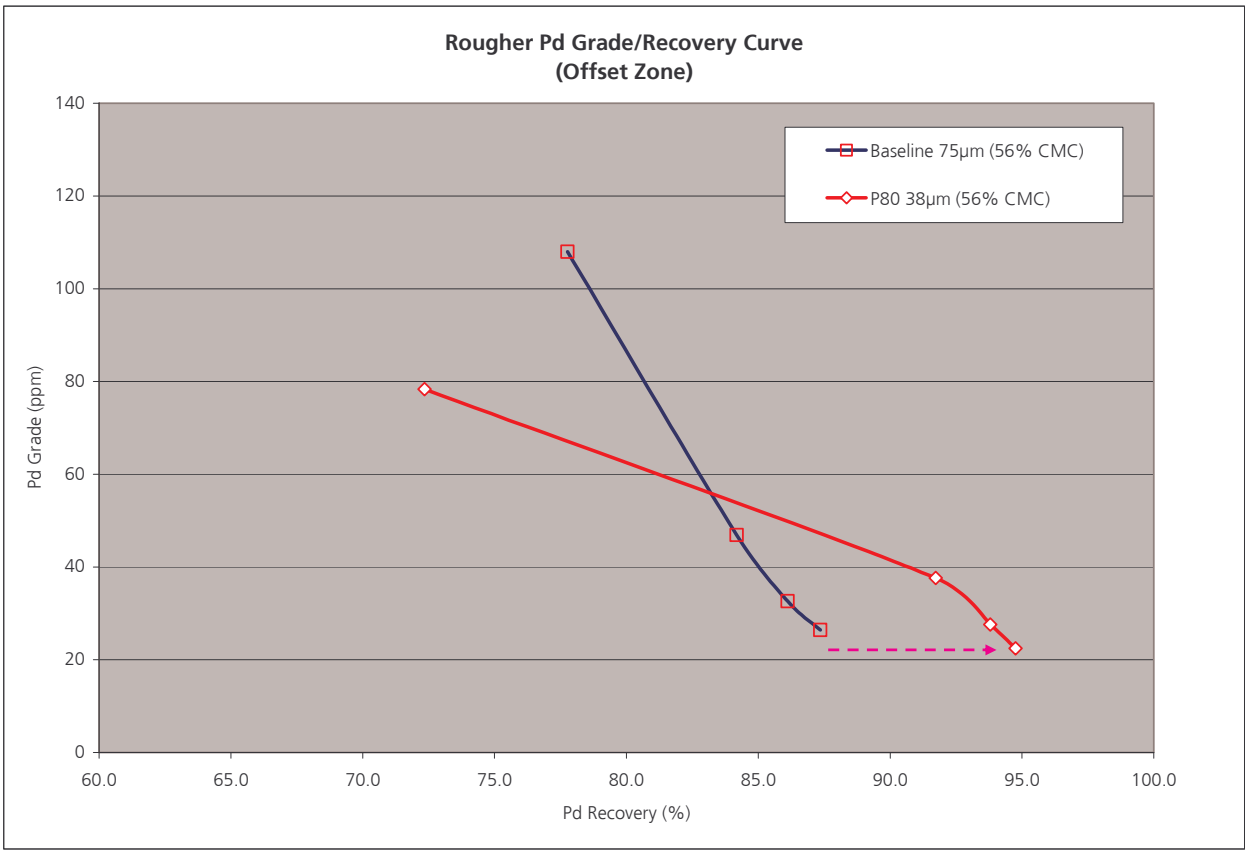

Figure 23: Rougher-Scavenger Pd Grade/Recovery vs. Primary Grind Size

From the test work, it is clear that the finer the primary grinding size (38 µm), the better the overall rougher-scavenge Pd recovery performance (7.5% increase in total Pd recovery). This is consistent with the results from the Roby tests.

CMC Dosage Validation

A series of rougher-scavenger flotation tests were conducted on the Offset composite to investigate the effect of CMC addition (MgO depressant) on overall Pd rougher-scavenger recovery and to validate the results from the Roby testing. A baseline of 75 µm was used as a primary grind in order to isolate the CMC dosage rate as the key variable. The detailed metallurgical mass balances are shown in the Appendix. The comparison of Pd grade/recovery at 2 different CMC dosages is displayed in Figure 24.

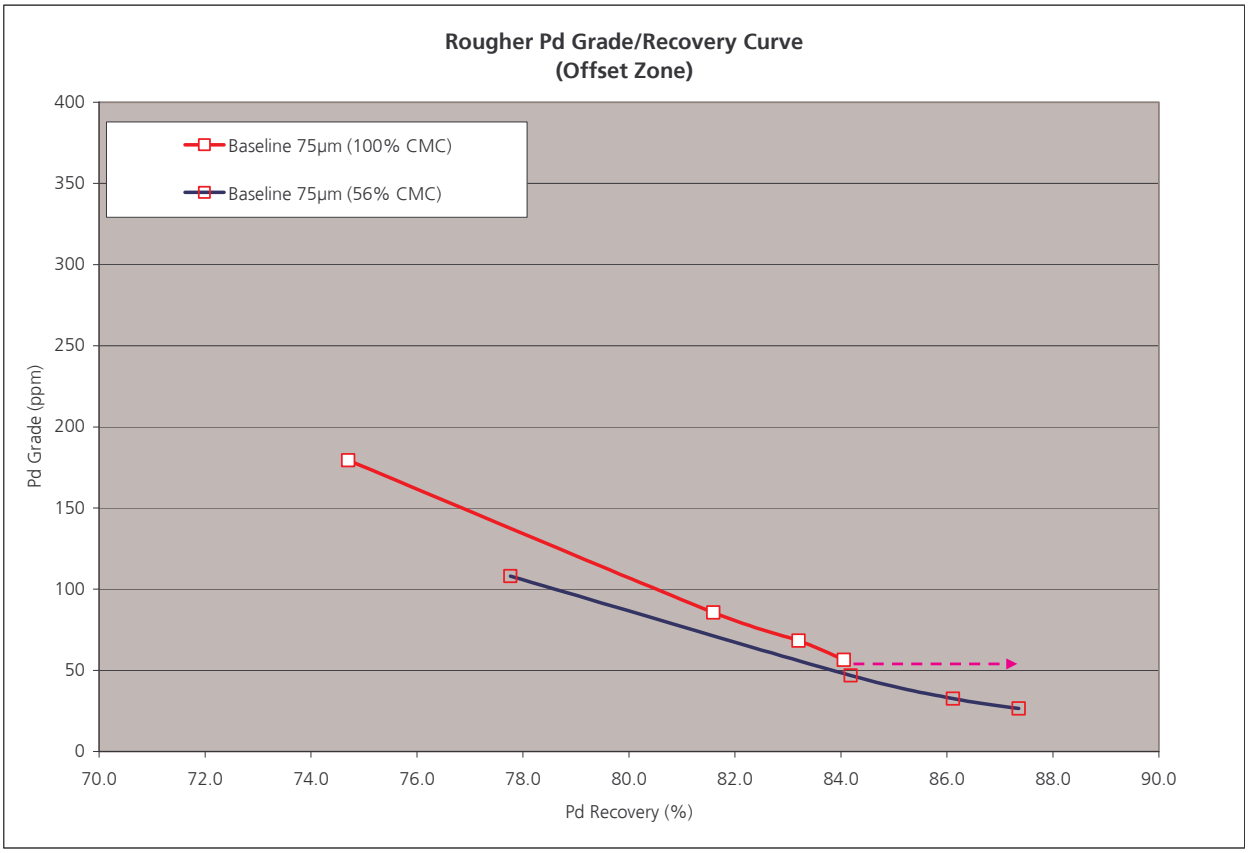

Figure 24: Rougher-Scavenger Pd Grade/Recovery vs. CMC Dosage

Figure 24 shows similar results to the Roby tests. At 100% of the initially prescribed CMC dosage (~172 g/t addition) compared to 56% of the CMC dosage (~92 g/t addition), an overall increase in Pd rougher recovery of 3.3% is observed at the lower CMC dosages.

4.4.2 – Open Circuit Cleaning Flotation Tests

Primary Grind Size Validation

Phases of open circuit flotation testing were also completed for the Offset ore in order to validate the results from the Roby tests.

Open-circuit cleaner flotation tests were completed in order to investigate the effect of finer primary grinding on the cleaner circuit. The detailed metallurgical mass balances are shown in the Appendix. The comparison of Pd grade/recovery at different primary grind sizes is displayed in Figure 25.

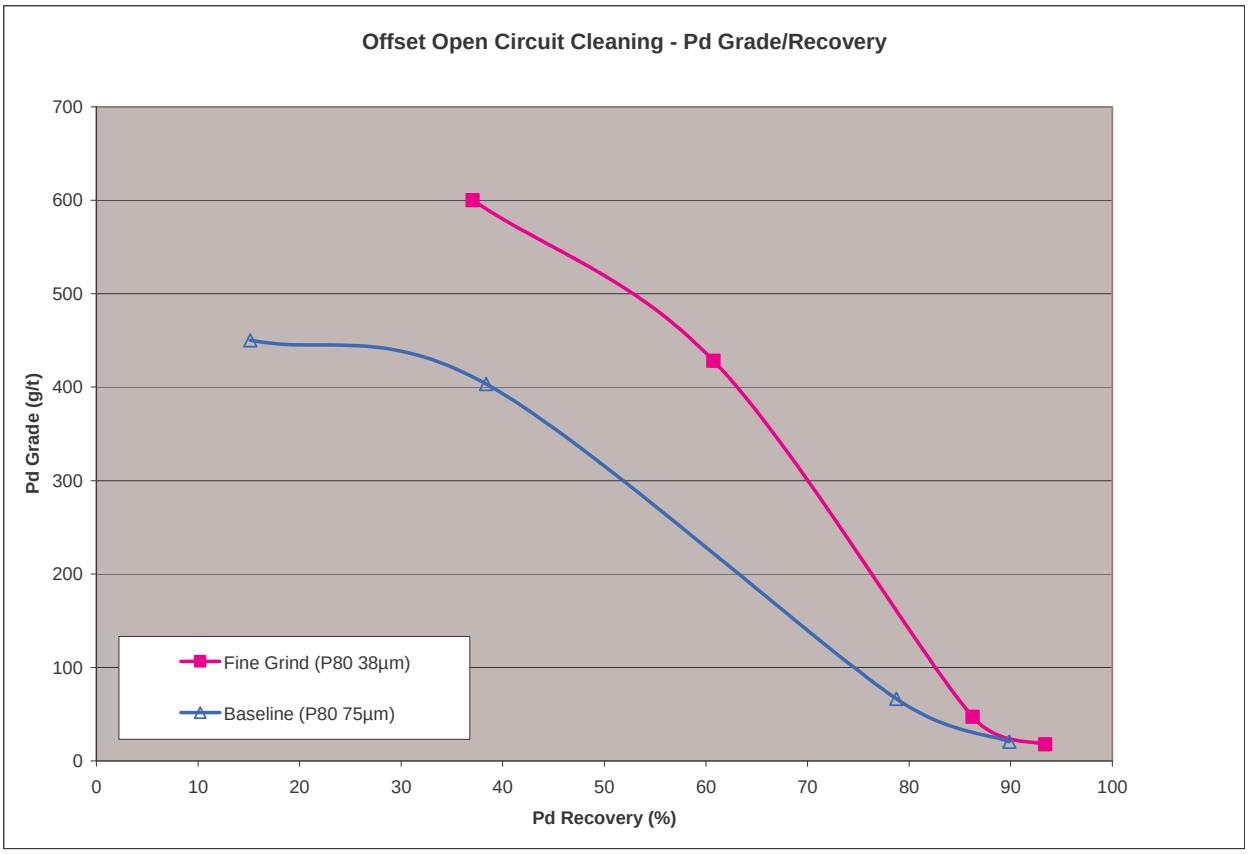

Figure 25: Cleaner Pd Grade/Recovery vs. Primary Grind Size.

Similar to the rougher-scavenger circuit and consistent with the Roby open-circuit testing results, the positive effect of finer primary grinding is evident in the cleaners which indicate superior Pd grade and recovery performance. The fine primary grinding up front effectively liberates the PGM minerals.

Fine Primary Grinding vs. Staged Grinding Validation

A series of open-circuit flotation tests were conducted to validate the effect of staged grinding compared to fine primary grinding observed in the Roby open-circuit cleaning tests.

A final grind of 38 µm was targeted for these tests. The detailed metallurgical mass balances are shown in the Appendix. The baseline comparisons of Pd grade/recovery at a fine primary grind vs. corresponding staged grinding to a final grind size of 38 µm is displayed in Figure 26.

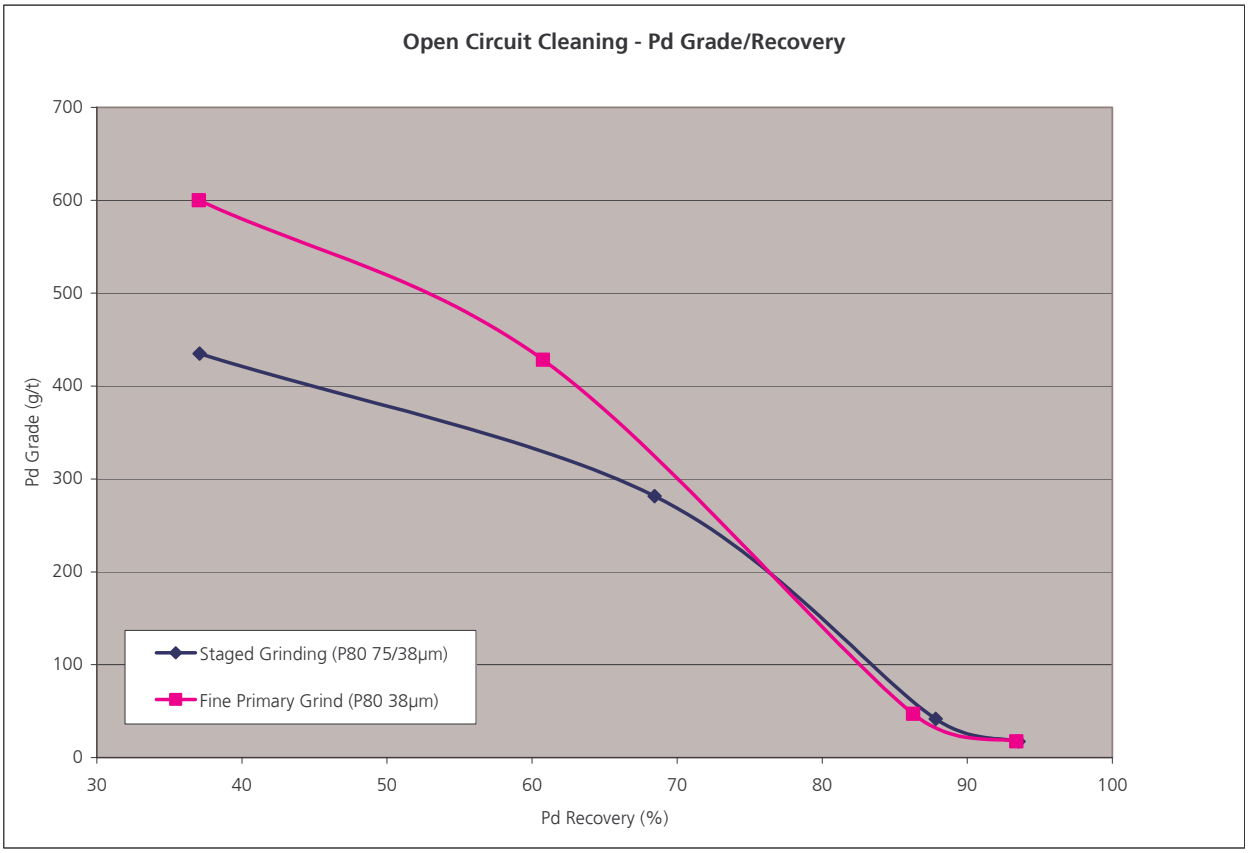

Figure 26: Cleaner Pd Grade/Recovery Comparing Fine Primary Grind to Staged Grinding

Similar to the response observed in the rougher-scavenger circuit and in the Roby tests, there is no clear advantage to staged grinding as compared to fine primary grinding for Pd grade/recovery in the cleaners for Offset ore.

4.4.3- Locked-Cycle Tests (LCT)

A series of Locked Cycle Tests were completed in order to validate and provide steady state metallurgical projections for the Offset ore composite. A locked cycle test is a repetitive batch test used to simulate a continuous circuit [2] or flowsheet. The flowsheet is repeated for a given number of cycles with the recirculating streams being re-introduced back into the circuit until steady state is reached. The number of cycles required to reach steady state generally varies from between 2 and 8 cycles, depending on the characteristics of the ore and the complexity of the flowsheet being

tested. Steady state is deemed to be achieved when equivalent final product masses in the three cycles are consistent to within 5% of each other.

The Locked Cycle Test flowsheet, with recirculating streams included is shown in the following diagram.

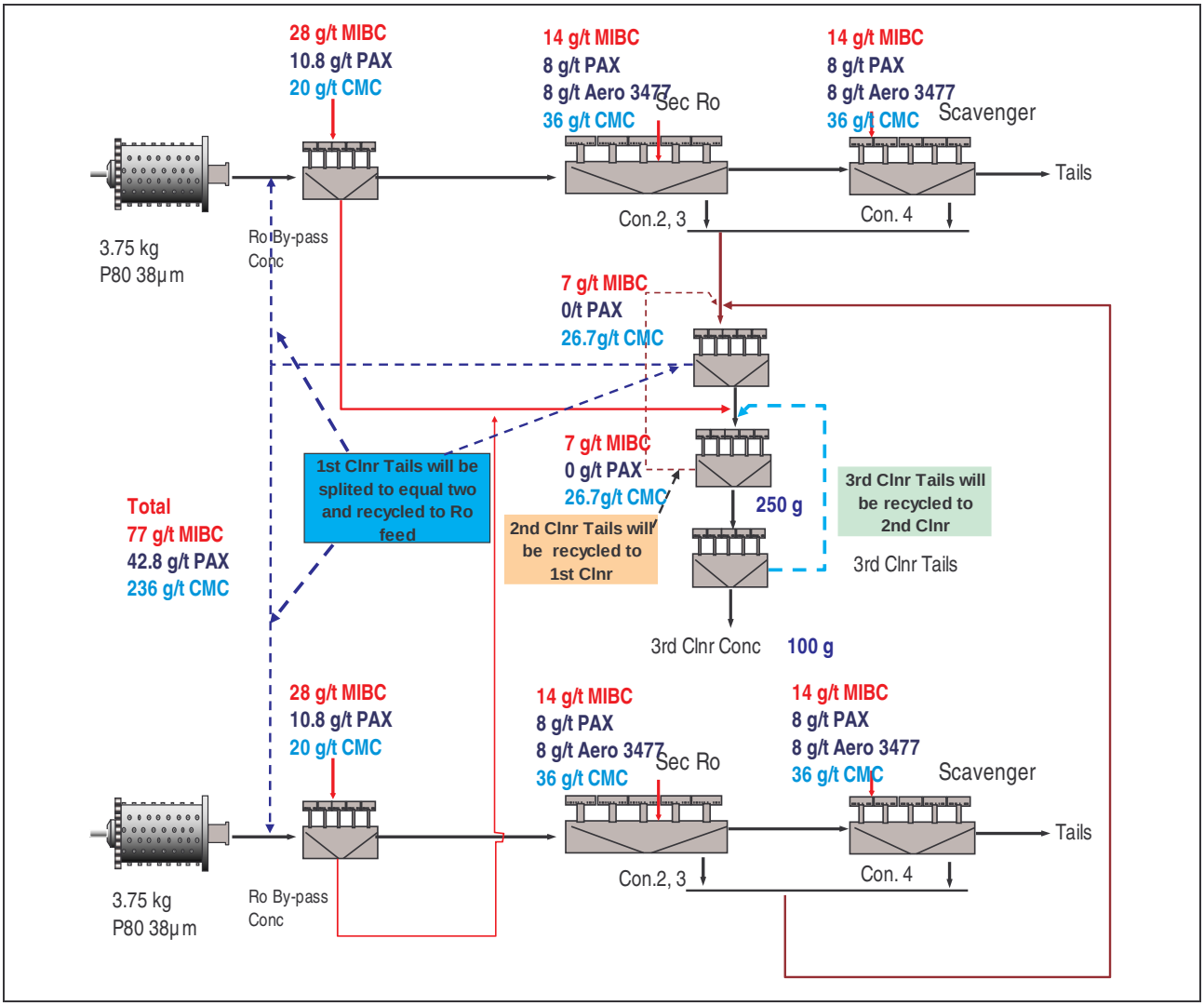

Figure 27: ***LCT Flowsheet.***

There are four accepted methods used to calculate the metal and mass balance for LCTs as described in the literature from the 2001 *Canadian Mineral Processors* (CMP) proceedings [2]. These are:

1. 2 product formula (balance on assays from final products)
2. SME Handbook (balance on final product weights and assays)
3. Concentration Production – Assumes unaccounted metal is in the tailings
4. Alternative – Assumes unaccounted metal is in the concentrate.

It is important to note that no procedure is ideal for a test that has not reached steady state. However, for a test that has reached steady state, all procedures should produce a similar metallurgical projection.

The four metallurgical projections for each test are summarised in the tables below.

Table 11: **Metallurgical Projections Using the Four Methods for LCT – Offset Zone**

2P Formula - Cu	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	7411.2	100.00	0.11	0.10	4.09	0.35	11.95	100.00	100.00	100.00	100.00	100.00
Concentrate	67.97	0.92	**2.80**	**10.13**	**390**	**33.16**	**7.32**	**23.11**	**90.58**	**87.44**	**86.60**	**0.56**
Tails	7343.2	99.08	0.09	0.01	0.52	0.05	11.99	76.89	9.42	12.56	13.40	99.44

SME Handbook	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	7091.4	100.00	0.11	0.11	4.22	0.36	11.96	100.00	100.00	100.00	100.00	100.00
Concentrate	67.5	0.95	**2.80**	**10.13**	**390.04**	**33.16**	**7.32**	**23.97**	**90.96**	**87.95**	**88.03**	**0.58**
Tails	7023.9	99.05	0.09	0.01	0.51	0.04	12.01	76.03	9.04	12.05	11.97	99.42

Concentrate Production - Unaccounted in Tails	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	7411.2	100.00	0.16	0.10	5.33	0.36	12.07	100.00	100.00	100.00	100.00	100.00
Concentrate	67.5	0.91	**2.80**	**10.13**	**390.04**	**33.16**	**7.32**	**16.40**	**89.93**	**66.67**	**84.69**	**0.55**
Tails	7343.7	99.09	0.13	0.01	1.79	0.06	12.11	83.60	10.07	33.33	15.31	99.45

ALTERNATIVE Unaccounted in the Conc	Weight		Assay, %					Distribution, %				
	g	%	Ni	Cu	Pd	Pt	MgO	Ni	Cu	Pd	Pt	MgO
Feed	7091.4	100.00	0.16	0.10	5.33	0.36	12.07	100.00	100.00	100.00	100.00	100.00
Concentrate	67.5	0.95	**7.45**	**9.77**	**506.35**	**32.96**	**18.30**	**45.63**	**90.66**	**90.46**	**87.96**	**1.44**
Tails	7023.9	99.05	0.09	0.01	0.51	0.04	12.01	54.37	9.34	9.54	12.04	98.56

When examining the mass balance and stability data it is clear that there is an imbalance in the test. The test did not reach steady state with 95.7% mass accountability and 76.1% Pd accountability in the last 3 cycles of the test. Pd Recoveries range from 66.7% to 90.7% Pd across the methods of calculation. Ni, Pd and MgO are the most effected, while Pt and Cu appear to produce consistent results and have the best metal accountability. Over 36% of the Pd was identified in the intermediate tails in the Offset Zone versus only 12.6% in the Roby Zone which contributed to the poor metal accountability. The test was run with 7 cycles and samples were re-assayed in an attempt to resolve the metal imbalances with limited improvement identified.

Recovery calculations based on the 'Concentrate Production' methodology appear to underestimate Pd recoveries. As expected, the 'Alt'method appears to overestimate MgO recovery and subsequent MgO grade in the concentrate. General stability/mass conservation values are lower than expected. Although this robust method of calculating LCT projections highlights a range of performance, 2-3 of the methods are in agreement. This in combination with open-circuit cleaner tests performed during the course of the program indicated acceptable recovery and MgO contents. More cycles may have been required to achieve steady state. The Pd grade/recovery points are shown graphically in Figure 28.

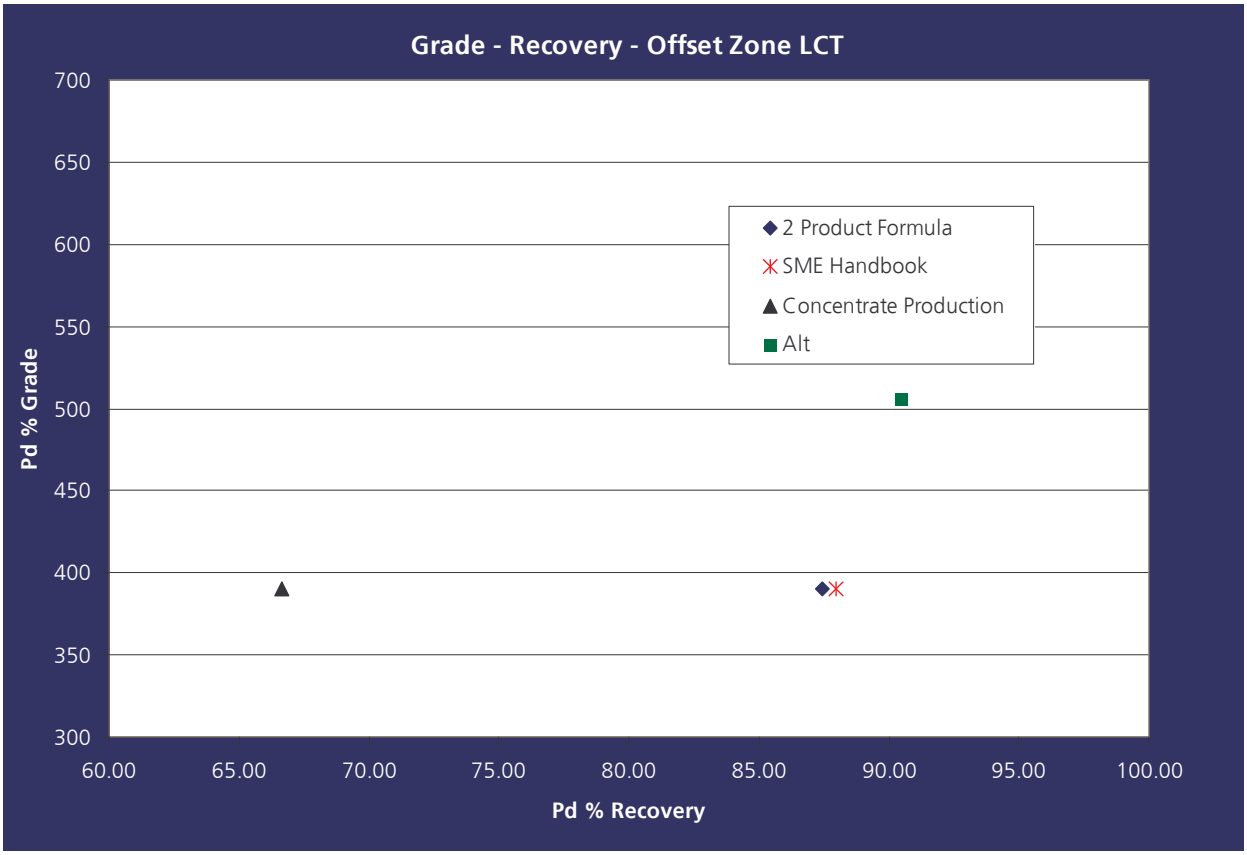

Figure 28: LCT Projections – Offset Zone

Based on the four calculation methods, the average metallurgical projections for Offset LCT are 27.3% Ni recovery, 90.5% Cu recovery, 83.1% Pd recovery and 86.8% Pt recovery at 3.96% Ni grade, 10.04% Cu grade and 419 g/t Pd. Pd in concentrate (419 g/t) is generally lower for Offset than for Roby Zone ore tested at 664 g/t Pd.

Table 12: Metallurgical projections for Offset LCT (Average)

Offset LCT Projections	Ni	Cu	Pd	Pt	Au	MgO
Grade (% or g/t)	3.96	10.04	419	33.1		10.06
Recovery (%)	27.3	90.5	83.1	86.8	85.3	0.8

** Au recovery is based on tails assays and assay head grade*

Mineralogical Analysis of Offset LCT Final Concentrate

As part of the scope of work, one LCT concentrate with a P_{80} of 38 µm was selected for mineralogical analysis. The Offset Zone LCT 3rd cleaner concentrate was selected as it represents a future ore.

Figure 29 highlights the modal mineralogical breakdown of the concentrate. Detailed data tables are included in the Appendix.

Figure 30 summarises the size-by-size mineralogical breakdown.

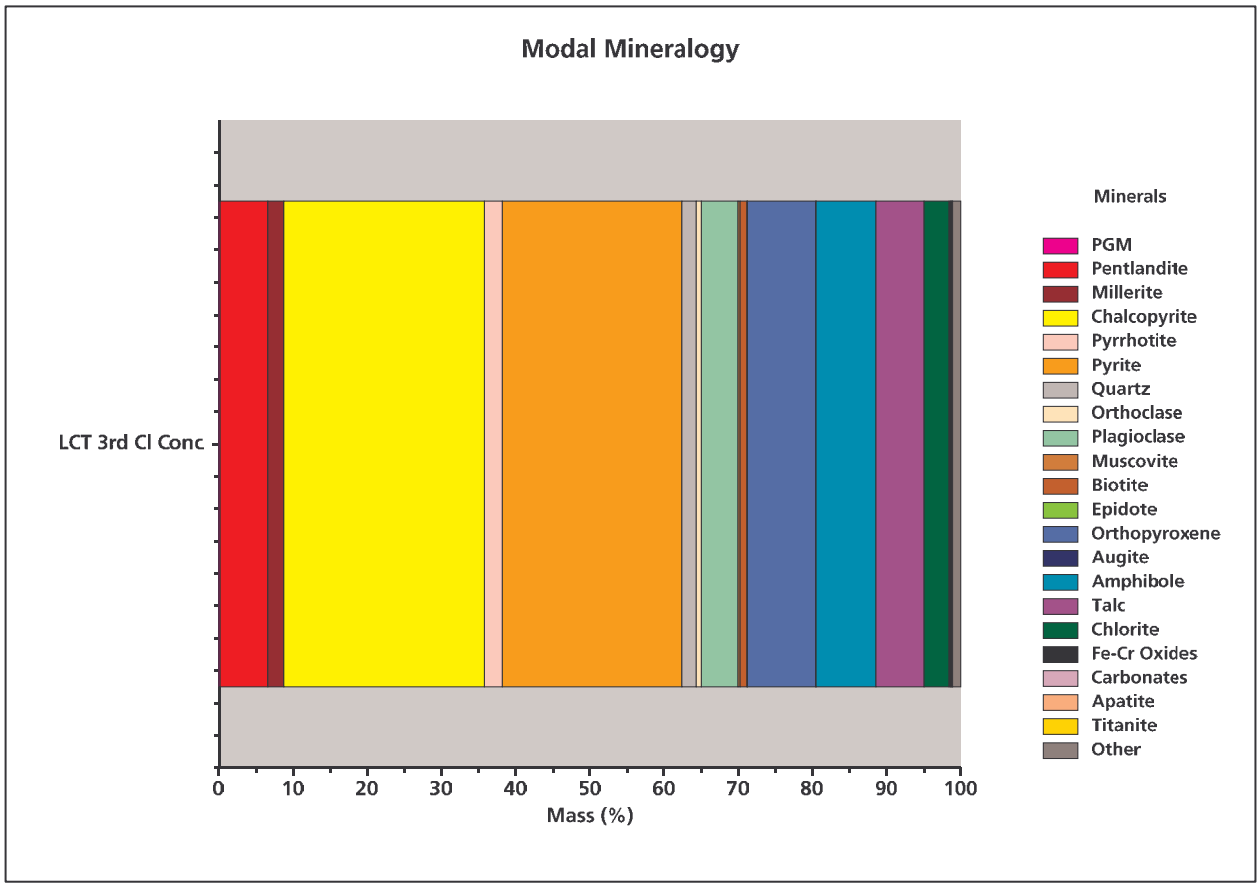

Figure 29*:* Modal mineralogy, Offset LCT 3rd Cleaner Concentrate.

Sulphides make up 62% of the concentrate by mass while pyrite alone constitutes 24%. Gangue diluents in concentrate are dominated by Fe-Mg silicates with noticeable upgrading of talc in the concentrate to 6.5% by mass. Orthopyroxene makes up approximately 10% of the concentrate by mass.

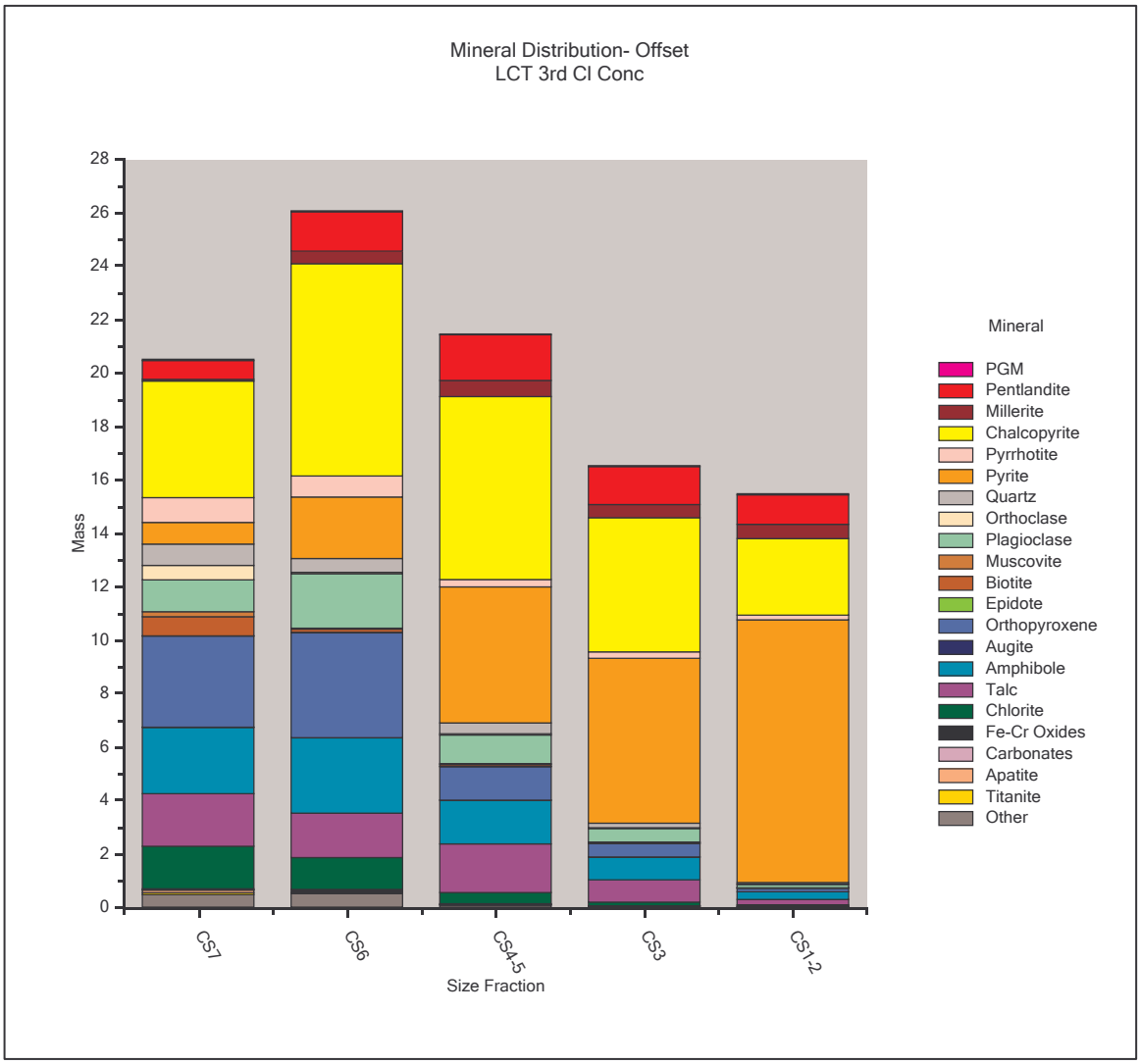

Figure 30: **Size-by-Size mineralogy of Offset LCT 3rd Cleaner Concentrate**

Pyrite dominates much of the mass in the coarser fractions while silicate gangue dominates the distribution in the finer fractions. Note-CS7 is a calculated fraction based on assays and mineralogical extrapolation from CS6.

Overall Ni-sulphide liberation (pentlandite + millerite) in the concentrate is 82% while total chalcopyrite liberation is 91%. Locked and middling Ni–sulphide and chalcopyrite are largely associated with non-sulphide gangue as observed in the feed.

Although the PGM levels in concentrate are in the order of 400 ppm, balancing calculated Pd grades from the mineralogical analysis is not feasible. PGM results presented are based on the TMS (Trace Mineral Search) QEMSCAN dataset which is a measurement mode which measures and targets only bright phase (PGM) particles for analysis.

Table 13 summarises the PGM occurrences by grouping based on the mass % of total PGM in the sample. Table 14 breaks down the average PGM grain size and PGM occurrences based on mass% in each fraction within broad speciation categories.

Table 13: PGM Occurrence by Category for Offset LCT Concentrate

Mass (%) PGM Norm 100%	Offset
PGM Groupings	**LCT Conc**
Au/Ag	6%
Pd Tellurides-Antimonides	54%
Pd-Arsenides/Other	0%
Pd Sulphides	28%
Pt Arsenides	4%
Pt PGM	7%
Other Te/Bi	0%
# of PGM Particles	2257

Table 14: PGM Occurrence and Average Grain Size Based on Mass% Total PGM/ Fraction

Offset- LCT Concentrate PGM Distribution					
Name	**Combined**	**CS6**	**CS4-5**	**CS3**	**CS1-2**
Mass Size Distribution (%)		26.1	21.5	16.5	15.5
Average PGM Grain Size (µm)	**8**	5	8	10	17
# of Precious Metal Bearing Particles	2257	584	626	615	432
PGM Distribution Mass%					
% Au/Ag Alloys	2.4	4.7	3.4	3.0	-
% Pd PGM	88.1	66.2	96.6	95.9	61.0
% Pt PGM	9.5	29.1	-	1.1	39.0
% Other Te/Bi	-	-	-	-	-
	100.0	*100.0*	*100.0*	*100.0*	*100.0*

Table 13 indicates that the majority of PGMs by mass are Pd-tellurides and antimonides followed by Pd sulphides which is consistent with the Offset feed measurements. Average PGM grain size data per fraction indicates that the PGM population is very fine grained with a combined average for the Offset LCT concentrate at 8 µm which is also consistent with the Offset feed measurements. Note that the feed P_{80} was 75 µm as compared to the LCT P_{80} of 38 µm.

Grain size distribution data for the Offset LCT concentrate supports the average PGM grain size data. Figure 31 shows the PGM grain size distribution of the LCT concentrate which was ground to a P_{80} of 38 µm.

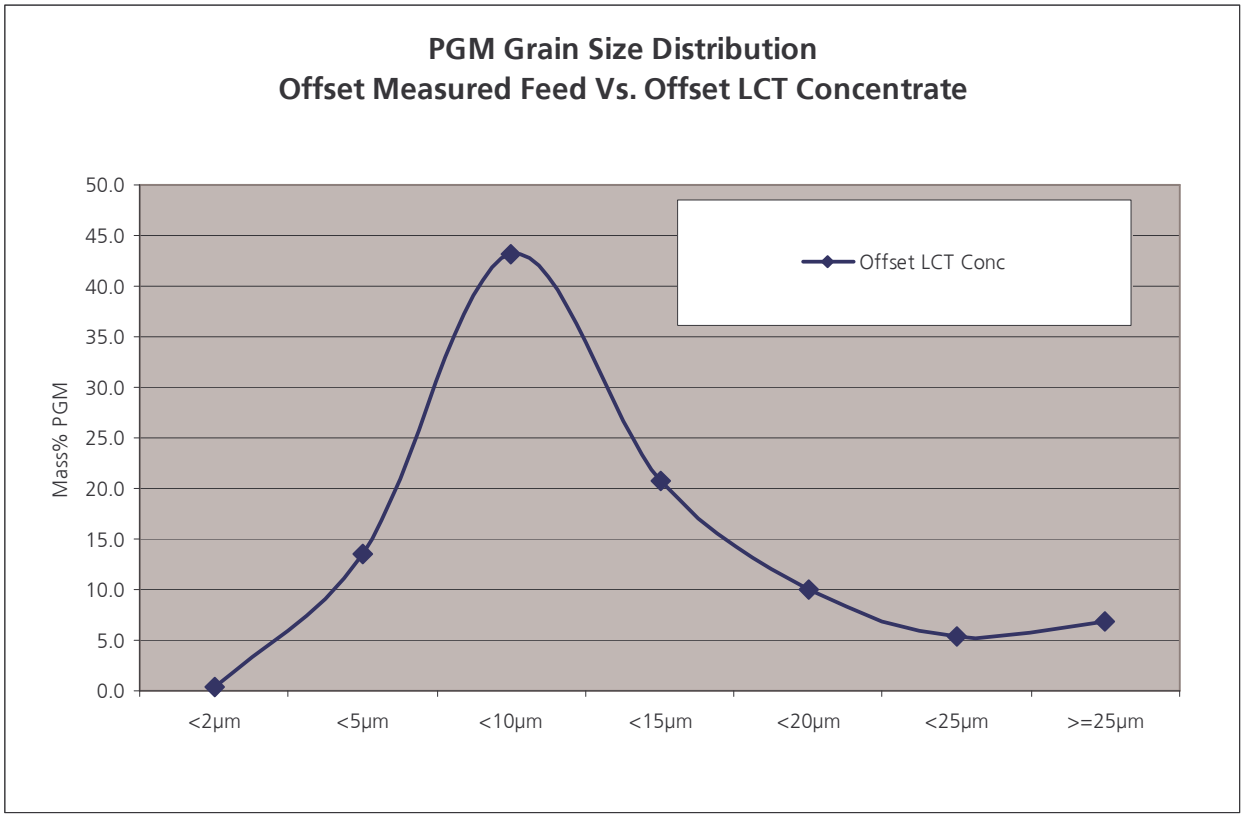

Figure 31: PGM Grain Size Distribution Comparing Measured Feed to Offset LCT Concentrate

The PGM grain size distribution is somewhat normal with greater than 90% by mass of PGm less than 25 µm.

PGM liberation data is presented in Table 15 which summarises the total PGM liberation broken down by mineral association and locking characteristics which provide insight into general PGM textures which relate to processing of these ores. Figure 32 summarises the size-by-size PGM liberation data.

Table 15: PGM Liberation and Locking/Association - Offset LCT Concentrate

PGM Occurrence Mode	Offset LCT Conc	
	Mass % PGM	# Particles
Liberated PGM	**83.9**	1496
Locked with Cpy	2.7	145
Locked with Pn	2.1	113
Locked in Gangue	3.3	111
Complex Locked with Gangue	0.7	49
Other	7.2	343
Totals	100.0	2257

Total PGM liberation data indicates that approximately 84% of PGM by mass in the Offset LCT concentrate are liberated at a target grind size of 80% passing 38 µm. Association and locking data indicates that the majority of remaining PGM which is not liberated is largely locked with non sulphide gangue or other complex gangue dominated associations, similar to analysis of the feed.

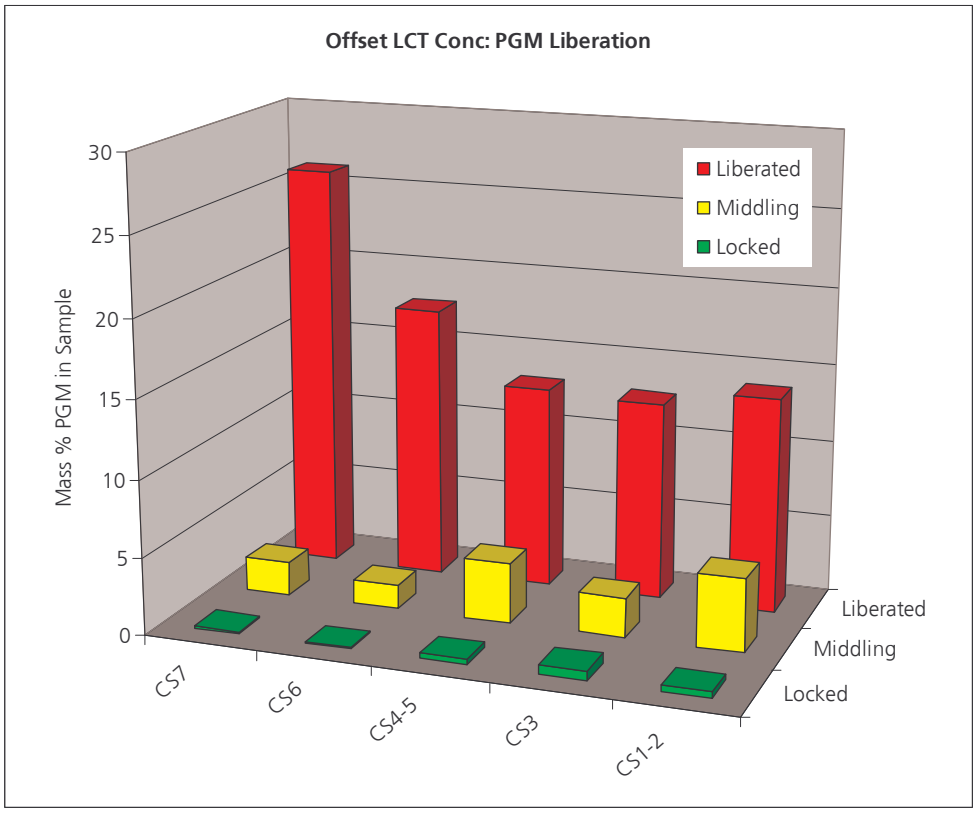

Figure 32: Size-by-Size PGM Liberation Offset LCT Concentrate Based on Mass% PGM

Total PGM liberation in concentrate is 84% while 15% by mass is middling and 2% is locked.

5.0- Ore Hardness Testing

Fifteen (15) kg of ½ inch LDI Roby and 15 kg of ½ inch Offset samples were extracted during the crushing and blending process, and were sent to SGS Lakefield for Bond Work Index testing in Feburary 2010. The testing procedure and results were summarized in a separate SGS report [Appended].

The Bond ball mill grindability test was performed at 200 mesh (75 microns) on the two LDI samples. The test results are summarized in Figure 33 and compared to SGS database. The Roby and Offset samples were categorised as hard, with a Bond ball mill work index (BWI) of 17.9 and 17.0 kWh/t respectively.

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

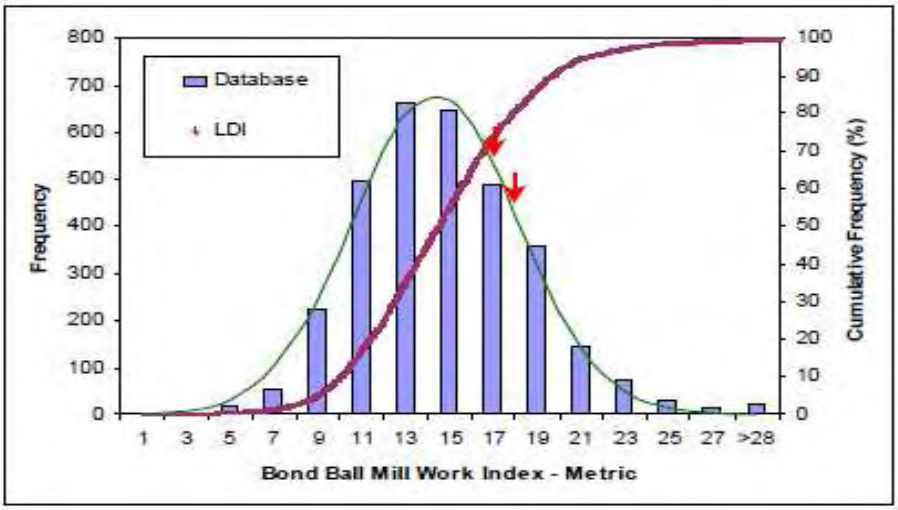

Figure 33: LDI Ore Bond Ball Mill Work Index

6.0- Gravity Recovery Tests (GRG)

25 kg of Roby and Offset ore hardness testing reject material was sent to Knelson Research and Technology Centre for an Extended Gravity Recoverable Gold, Silver, Platinum and Palladium test (E-GRG).

E-GRG Test Program Methodology

The GRG value of an ore sample provides an indication of the amenability of the ore to gravity concentration. The E-GRG testing scheme is based on progressive particle size reduction, which allows recovery of gold, silver, platinum and palladium as liberated while minimizing over grinding and smearing. The E-GRG test consists of sequential liberation and recovery stages. The Extended Gravity Recoverable Gold, Silver, Platinum and Palladium test includes screen analysis on the concentrates and last-stage tailings with assays on each size fraction. The results of these tests are summarised in Table 16.

Table 16: Results of GRG testing

Extended Gravity Recovery Test								
Grind Size (µm)	Product	Mass (%)	Assay (g/t)			Recovery (%)		
			Pd	Pt	Au	Pd	Pt	Au
P80 840 µm	Stage 1 Conc	0.4	183	12.9	47.9	11.5	13.1	33.8
P80 206 µm	Stage 2 Conc	0.4	185.2	11.7	13.5	11.3	11.6	9.3
P80 54 µm	Stage 3 Conc	0.2	152.5	11.2	18.9	6	7.2	8.4
Knelson Conc	**Stage 1~3 Conc**	1.0	**176.5**	12.0	0.54	**28.8**	**31.9**	**51.5**
Totals (Head)		100	6.06	0.37	0.54	100	100	100

The overall GRG Au recovery after three stages was 51.5%. The calculated gold feed grade of the sample was 0.54 g/t Au. First stage recovery attributed to 33.8% of the GRG, the second stage provided 9.3% and the third stage recovery contributed 8.4% with a final grind P_{80} of 54 microns.

The overall gravity recoverable platinum after three stages was 31.9% at 12 g/t Pt. The calculated platinum feed grade of the sample was 0.37 g/t Pt. Gravity palladium recovery was 28.8% at 176.5 g/t Pd after three processing stages. The calculated palladium feed grade of the sample was 6.06 g/t Pd.

Based on the extended GRG tests at a P_{80} of 54 µm, both platinum and palladium bearing minerals do not respond well to gravity recovery. This is consistent with the fine grain size of PGM minerals identified from QEMSCAN mineralogical analysis. Au recovery in GRG was moderate. Generally, as high grade concentrates cannot be produced directly by gravity recovery even at lower recoveries, gravity recovery is not feasible in terms of PGE recovery improvement for Roby and Offset ores. Gravity concentrates were not assayed for MgO. A portion of recovery in very low MgO concentrates could present an option if MgO penalties become excessive.

7.0- Conclusions and Recommendations

Mineralogical Analysis of Feeds

- The modal mineralogical analysis indicates that the Roby Zone and the Offset Zone composites are very similar.
- Talc content ranges from 0.9% by mass in the Roby composite, to 1.1% by mass in the Offset composite indicating only marginally higher talc content in the Offset zone. It is recognized that due to localised faulting and alteration that talc content can be significantly higher in localised regions within these zones
- At a P_{80} of 75 µm, Ni-Sulphide liberation (pentlandite and millerite combined) is moderate at 76% and 79% for Roby and Offset respectively while chalcopyrite is 64% and 63%

liberated indicating minimal variation in sulphide liberation between the Roby and Offset composites.

- PGM phases identified within the feed are Pd-sulphides (vysotskite), Pd-tellurides (kotulskite, merenskyite), Pd-antimonides (mertieite), Pd-arsenides (arsenopaladinite) with lesser amounts of Pt-arsenides (sperrylite) and Au/Ag species (electrum).
- The majority of PGMs by mass are Pd-tellurides and Pd-antimonides followed by Pd sulphides.
- Average PGM grain size is very fine grained and the combined average for both Roby and Offset zone are similar at 7 µm and 8 µm respectively.
- Although based on limited PGM particle statistics in the feed, 70% of PGM by mass in both Roby and Offset are liberated at a target grind size of 80% passing 75 µm.
- PGM association and locking data indicates that the majority of remaining PGM which is not liberated is largely locked with non-sulphide gangue or other complex gangue dominated associations.
- Mineral composition analysis by EPMA indicates solid solution levels of Ni in pyrrhotite and pyrite as well as low levels of Pd in solution within pentlandite. Silicate analyses indicate low levels of solid solution Ni in talc, chlorite, orthopyroxene, amphiboles and biotite.
- Ni deportment data indicates that for Roby and Offset composites, only 52% and 60% of total Ni is carried by Ni-sulphides respectively. The remaining nickel occurs predominantly in Fe-Mg gangue. Ni recovery is heavily impact by gangue rejection in the circuit resulting in lower realized Ni recoveries.

Metallurgical Study Results:

Roby:

- Roby Zone Composite contains: 0.12% Ni, 0.07% Cu, 0.18% S, 6.12g/t Pd, 0.31g/t Pt, 0.37 g/t Au, and 12.20% MgO.
- A series of rougher-scavenger flotation tests were undertaken to determine an appropriate primary grind size and assess staged-grind vs. fine-grinding as well as to select and optimise dosages for MgO depressants, and PGM promoters.
- Tests showed that a primary grind of 80% passing 38 microns, 26 minutes of rougher flotation time, 92 g/t CMC (*Depramin C*), 26 g/t PAX, 16 g/t Aero 3477 and 36 g/t MIBC gave optimum rougher-scavenger flotation response.
- Rougher flotation testing indicates no clear advantage to staged grinding which is supported by the fine PGM grain size and lack of coarse bi-modal PGM grain size distribution.
- A series of open circuit cleaner tests were undertaken to determine appropriate MgO depressant and cleaning stage requirements. Tests showed that 25 g/t CMC for MgO depressant and 3 stages of cleaning gave optimum cleaning performance.
- Locked-Cycle Tests (LCTs) were undertaken to provide steady state metallurgical projections of Roby ore. LCT results highlight the good upgrade characteristics of the Roby ore with average 35.9% Ni recovery, 86.1% Cu recovery, 86.2% Pd recovery and 78.6% Pt recovery were achieved at 5.2 % Ni and 7.3% Cu grade in the final concentrate. Due to reduced CMC addition in the 1st Clnr and 2nd Clnr, and high circulating load in the 1st Clnr, MgO content of the final concentrate was 8.97%, slightly above the desired limit. However, based on open circuit cleaning flotation tests, the MgO content in the final cleaner concentrate was as low as 0.85% and should be controlled in the plant by judicious use of CMC and control of circulating loads.

- Performance of the locked cycle test was a challenge, due to the low mass pulls. Feed of 10 kg/cycle were used and after 6 cycles, mass, metal balance and stability was acceptable for the Roby Zone.

Offset:

- Offset Composite contains: 0.14% Ni, 0.10% Cu, 0.28% S, 5.52g/t Pd, 0.35g/t Pt, 0.29 g/t Au, and 11.60% MgO.
- A series of rougher-scavenger flotation tests were undertaken to determine an appropriate primary grind size, staged-grind vs. fine-grinding, MgO depressant, and reagent dosage. Consistent with that of Roby composite, tests showed that a primary grind of 80% passing 38 microns, 26 minutes of rougher flotation time, 92 g/t CMC (*Depramin C*), 26 g/t PAX, 16 g/t Aero 3477 and 36 g/t MIBC gave optimum rougher-scavenger flotation response.
- A series of open circuit concentrate tests were undertaken to determine an appropriate MgO depressant and cleaning stages requirement. Consistent with that of Roby composite, cleaning test results showed that 50 g/t CMC for MgO depressant and 3 stages of cleaning gave optimum cleaning performance.
- Locked-Cycle Tests (LCTs) were undertaken to provide a steady state metallurgical projection of Offset ore. The LCT results highlight the good upgrade characteristics of the Offset ore with average 27.3% Ni, 90.5% Cu recovery, 83.1% Pd recovery and 86.8% Pt recovery were achieved at 3.96 % Ni and 10.0% Cu grade in the final concentrate. Better Cu metallurgy is likely due to higher Cu head grades.
- Mass, metal balance and stability was poor for the Offset Zone locked cycle test. Cu and Pt metal balance was acceptable however mass, Ni, Pd and MgO stability were below targets with unaccounted loss. Over 35% of Pd was present in the Offset Zone intermediate tests versus only 12% for the Roby Zone. This contributed to a lack of steady state conditions.
- Although this robust method of calculating LCT projections highlights a range of performance, 2-3 of the methods are in agreement. This in combination with open-circuit cleaner tests performed during the course of the program indicated acceptable recovery and MgO contents. More cycles may have been required to achieve steady state and the test can be repeated under these conditions if requested.

Mineralogical Analysis of Offset LCT Concentrate:

- Sulphides make up 62% of the concentrate by mass while pyrite alone constitutes 24%.
- Gangue diluents in concentrate are dominated by Fe-Mg silicates with noticeable upgrading of talc in the concentrate to 6.5% by mass. Orthopyroxene makes up approximately 10% of the concentrate by mass.
- Overall Ni-sulphide liberation (pentlandite + millerite) in the concentrate is 82% while total chalcopyrite liberation is 91%.
- The majority of PGMs by mass are Pd-tellurides and Pd-antimonides followed by Pd sulphides which is consistent with the Offset feed measurements.
- Average PGM grain size data per fraction indicates that the PGM population is very fine grained and the combined average for the Offset LCT concentrate at 8µm which is also consistent with the Offset feed measurements.
- Total PGM liberation data indicates that approximately 84% of PGMs by mass in the Offset LCT concentrate are liberated at a target grind size of 80% passing 38 µm while 15% by mass is middling and 2% is locked.

- Association and locking data indicates that the majority of remaining PGM which is not liberated is largely locked with non sulphide gangue or other complex gangue dominated associations, similar to analysis of the feed.

Ore Hardness:

- Ore hardness tests show that both Roby and Offset ore are characterized as 'hard' with a Bond ball mill work index (BWI) of 17.9 and 17.0 kWh/t respectively.

Gravity Recovery Tests (GRG):

- Based on the extended GRG tests at a P_{80} of 54 µm, both platinum and palladium bearing minerals do not respond well to gravity recovery with only 31.9% Pt and 28.8% Pd recovery after three processing stages at 12.0 g/t and 176.5 g/t respectively in concentrate . This is consistent with the fine grain size of PGM minerals identified from QEMSCAN mineralogical analysis. Gravity concentration of PGM does not appear to be a viable process option to improve grade and recovery of PGE. MgO in final gravity concentrates was not assayed and could present an alternative for MgO rejection if MgO penalties become excessive.

Recommendations:

Based on the testwork program undertaken at XPS, the following recommendations should be considered by LDI for Phase 2 Metallurgical Testwork:

- Grinding JKSimMet Simulation Study is to access the current grinding capacity and configuration for the proposed primary grind target of P_{80} 38µm.
- Zone composites help provide robust and long term metallurgical projection data however, a series of variability tests are recommended which test higher talc contents as well as a range of grades through the proposed flowsheet. This can help define the range of metallurgical variability which could be expected during processing of the ores.
- Flotation testing using soda ash as a pH modifier should be investigated in order to determine any benefits of higher than natural pH on gangue depression.

8.0- References

[1] Lotter, N. Middleditch, D. Xstrata Process Support – Process Mineralogy Laboratory Standard Manual. Updated August 2006.

[2] Ounpuu, M., Was that locked Cycle Test Any Good?, Paper 27, pages 389-404, 33rd Annual Meeting of the Canadian Mineral Processors, Ottawa, Ontario, 2001.

9.0 - Appendix

Appendix A- QEMSCAN Modal Mineralogy

Offset

			Min Size	0.0	3.0	7.0	15.0	25.0	53.0	75.0	106.0
			Max Size	3.0	8.0	18.0	27.0	53.0	75.0	106.0	212.0
			Mass Size Distribution (%)	9.50	9.02	12.81	11.82	23.00	15.24	11.75	6.85
	Element	**Head**	**Combined**	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**
Chemical vs QEMSCAN Assay	Cu		0.10	0.07	0.09	0.09	0.11	0.18	0.07	0.07	0.05
	(Chemical)	0.10	0.07	0.07	0.08	0.08	0.08	0.11	0.05	0.04	0.04
	Fe		6.12	6.84	6.85	6.28	5.98	5.85	5.73	5.78	6.49
	(Chemical)		5.79	6.85	6.73	6.44	5.81	5.50	4.97	5.37	5.37
	Mg		7.45	8.07	8.05	7.98	7.44	6.75	7.06	7.27	8.40
	(Chemical)	6.99	7.35	8.07	8.27	8.25	7.51	6.69	6.61	7.22	7.22
	Ni		0.15	0.14	0.17	0.15	0.13	0.21	0.16	0.11	0.11
	(Chemical)	0.14	0.13	0.14	0.14	0.12	0.12	0.18	0.09	0.08	0.08
	S		0.36	0.13	0.22	0.24	0.28	0.73	0.33	0.33	0.16
	(Chemical)	0.28	0.23	0.13	0.16	0.18	0.20	0.41	0.21	0.14	0.14
	Si		23.06	21.30	22.49	22.88	22.99	23.73	23.26	23.65	23.01
	(Chemical)		23.12	21.30	21.00	22.30	23.20	24.20	23.70	23.80	23.80
	Mineral		**Combined**	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**
Mineral Mass	PGM		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Pentlandite		0.2	0.1	0.2	0.2	0.2	0.4	0.2	0.1	0.1
	Millerite		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Chalcopyrite		0.3	0.2	0.3	0.3	0.3	0.5	0.2	0.2	0.2
	Pyrrhotite		0.2	0.0	0.2	0.1	0.1	0.3	0.2	0.2	0.1
	Pyrite		0.2	0.0	0.0	0.1	0.1	0.6	0.2	0.3	0.1
	Quartz		5.8	3.9	5.6	7.1	6.8	5.8	5.6	5.9	5.4
	Orthoclase		0.9	3.9	0.5	0.5	0.6	0.5	0.6	0.8	0.6
	Plagioclase		27.2	22.3	23.7	24.9	27.2	30.3	30.9	28.6	21.6
	Muscovite		0.9	0.9	0.5	0.7	0.8	0.8	1.0	1.3	1.2
	Biotite		2.1	2.0	2.3	2.0	2.2	1.7	2.6	2.3	2.7
	Epidote		0.4	0.3	0.2	0.2	0.4	0.6	0.4	0.4	0.3
	Orthopyroxene		4.2	4.6	6.2	3.3	3.4	3.7	4.0	4.8	5.8
	Augite		0.2	0.6	0.0	0.0	0.1	0.2	0.2	0.3	0.3
	Amphibole		35.7	28.0	33.3	32.7	33.7	41.7	34.8	37.4	38.0
	Talc		1.1	0.1	0.8	1.5	1.2	0.8	1.2	1.5	1.8
	Serpentine		0.4	2.8	0.4	0.2	0.1	0.0	0.0	0.1	0.0
	Chlorite		18.5	24.4	23.8	25.5	21.4	10.8	16.9	15.2	21.1
	Fe-Cr Oxides		0.2	1.8	0.2	0.0	0.1	0.1	0.1	0.1	0.0
	Carbonates		0.5	2.0	0.7	0.3	0.7	0.5	0.2	0.1	0.1
	Apatite		0.3	0.8	0.1	0.2	0.3	0.3	0.3	0.3	0.4
	Titanite		0.1	0.7	0.1	0.1	0.0	0.0	0.0	0.0	0.0
	Other		0.4	0.7	0.9	0.4	0.3	0.4	0.3	0.2	0.2

Roby

			Min Size	0.0	3.0	7.0	15.0	25.0	53.0	75.0	106.0
			Max Size	3.0	8.0	18.0	27.0	53.0	75.0	106.0	212.0
			Mass Size Distribution (%)	9.80	9.64	14.83	13.46	20.87	17.74	10.24	3.40
	Element	**Head**	**Combined**	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**
Chemical vs QEMSCAN Assay	Cu		0.09	0.12	0.10	0.08	0.06	0.12	0.06	0.07	0.04
	(Chemical)	0.07	0.12	0.12	0.14	0.12	0.11	0.18	0.07	0.05	0.08
	Fe		6.20	7.44	7.33	6.87	6.06	5.67	5.39	5.70	6.19
	(Chemical)		6.39	7.45	7.20	6.74	6.26	6.03	5.59	5.73	8.45
	Mg		7.54	7.60	8.57	8.43	7.78	6.84	6.85	7.35	8.04
	(Chemical)	7.36	7.03	7.60	7.78	7.66	7.20	6.31	6.42	6.99	7.70
	Ni		0.13	0.18	0.14	0.13	0.11	0.18	0.10	0.08	0.08
	(Chemical)	0.12	0.19	0.18	0.18	0.17	0.16	0.29	0.16	0.12	0.17
	S		0.24	0.21	0.17	0.20	0.20	0.44	0.20	0.20	0.09
	(Chemical)	0.18	0.32	0.21	0.26	0.29	0.31	0.59	0.23	0.22	0.18
	Si		23.08	21.41	22.08	22.17	23.24	24.14	23.65	23.66	22.92
	(Chemical)		23.04	21.40	21.30	22.70	23.00	24.10	23.60	23.90	22.40
	Mineral		**Combined**	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**
Mineral Mass	PGM		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Pentlandite		0.1	0.1	0.1	0.1	0.1	0.2	0.1	0.1	0.0
	Millerite		0.0	0.1	0.0	0.0	0.0	0.1	0.0	0.0	0.0
	Chalcopyrite		0.2	0.3	0.3	0.2	0.2	0.3	0.2	0.2	0.1
	Pyrrhotite		0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
	Pyrite		0.1	0.0	0.0	0.1	0.1	0.3	0.2	0.1	0.0
	Quartz		5.6	5.6	5.4	5.6	6.8	5.7	5.2	4.6	4.3
	Orthoclase		1.1	1.7	0.8	0.7	0.8	1.1	1.3	1.5	1.4
	Plagioclase		26.9	21.2	22.0	22.6	26.0	30.6	32.4	28.4	24.8
	Muscovite		1.5	3.6	0.5	0.9	1.0	1.5	1.7	1.9	1.9
	Biotite		1.2	0.1	1.5	1.0	1.0	1.2	1.6	1.8	1.9
	Epidote		0.4	0.5	0.2	0.3	0.3	0.4	0.5	0.3	0.3
	Orthopyroxene		4.8	6.1	7.3	3.5	3.4	4.3	4.8	5.3	5.8
	Augite		0.3	0.8	0.0	0.1	0.1	0.3	0.3	0.3	0.4
	Amphibole		35.0	25.1	29.9	33.0	35.1	41.1	35.2	39.3	36.4
	Talc		0.8	0.7	0.5	0.9	0.8	0.8	0.8	1.0	1.0
	Serpentine		0.2	0.7	0.3	0.1	0.1	0.0	0.0	0.0	0.0
	Chlorite		19.7	25.1	28.9	28.9	23.0	10.7	14.8	14.3	20.9
	Fe-Cr Oxides		0.6	3.5	0.4	0.3	0.2	0.4	0.1	0.1	0.1
	Carbonates		0.3	1.2	0.3	0.3	0.2	0.1	0.1	0.1	0.0
	Apatite		0.2	0.1	0.1	0.2	0.2	0.3	0.3	0.3	0.3
	Titanite		0.2	1.4	0.1	0.0	0.0	0.0	0.1	0.0	0.0
	Other		0.6	2.0	1.4	0.7	0.3	0.4	0.3	0.3	0.3

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Offset- LCT Concentrate

		Min Size	0.0	3.0	7.0	15.0	25.0
		Max Size	3.0	8.0	18.0	27.0	53.0
		Mass Size Distribution (%)	20.50	26.06	21.45	16.52	15.47
	Element	**Combined**	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**
Chemical vs QEMSCAN Assay	Cu	9.31	7.18	10.53	11.03	10.50	6.39
	(Chemical)	9.63	7.18	9.70	11.70	11.20	8.24
	Fe	24.27	14.60	19.84	25.07	30.13	37.19
	(Chemical)	24.22	14.60	18.80	25.80	30.30	37.40
	Mg	4.11	7.26	5.39	3.63	2.10	0.62
	(Chemical)	4.27	7.25	5.98	3.51	2.14	0.74
	Ni	3.59	1.25	3.11	4.58	4.91	4.70
	(Chemical)	3.30	1.25	2.57	4.12	4.65	4.64
	S	25.60	11.20	18.49	27.68	34.68	44.07
	(Chemical)	25.03	11.20	18.30	27.40	34.10	41.70
	Si	9.68	17.11	12.54	8.44	5.11	1.63
	(Chemical)	10.30	17.10	14.10	8.80	5.62	1.99
Mineral Mass	**Mineral**	**Combined**	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**
	PGM	0.2	0.2	0.1	0.2	0.2	0.3
	Pentlandite	6.4	3.4	5.7	8.0	8.6	7.1
	Millerite	2.2	0.3	1.8	2.8	3.0	3.5
	Chalcopyrite	27.0	21.3	30.5	31.9	30.4	18.5
	Pyrrhotite	2.4	4.5	3.0	1.3	1.4	1.2
	Pyrite	24.2	4.0	8.8	23.7	37.3	63.5
	Quartz	1.9	3.9	2.0	1.9	1.0	0.4
	Orthoclase	0.7	2.6	0.2	0.2	0.2	0.1
	Plagioclase	4.9	5.8	7.8	5.0	3.1	0.8
	Muscovite	0.3	0.9	0.1	0.2	0.1	0.0
	Biotite	0.9	3.5	0.5	0.3	0.2	0.1
	Epidote	0.1	0.0	0.1	0.1	0.1	0.1
	Orthopyroxene	9.2	16.7	15.0	5.8	3.0	0.8
	Augite	0.0	0.0	0.0	0.0	0.0	0.0
	Amphibole	8.1	12.1	10.8	7.6	5.2	1.9
	Talc	6.5	9.6	6.4	8.5	5.1	1.3
	Chlorite	3.4	7.8	4.6	1.9	0.8	0.3
	Fe-Cr Oxides	0.2	0.2	0.4	0.1	0.1	0.1
	Carbonates	0.1	0.0	0.1	0.1	0.0	0.0
	Apatite	0.1	0.5	0.0	0.0	0.0	0.0
	Titanite	0.1	0.3	0.0	0.0	0.0	0.0
	Other	1.1	2.3	1.9	0.4	0.2	0.1

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Appendix B- Ni-Sulphide and Cu Sulphide Liberation

ROBY

SMS Corrected as % in SAMPLE			
Liberation of Ni Sulphide			
	Locked	Middling	Liberated
-212/+106	0.3	0.0	0.6
-106/+75	1.0	0.5	3.0
-75/+53	1.4	2.3	10.7
CS1-2	3.6	6.7	34.5
CS3	0.8	2.9	9.3
CS4-5	0.2	0.7	6.5
CS6	0.1	1.7	5.4
CS7	0.1	1.8	5.7
Combined	7.6	16.6	75.8

SMS Corrected as % in SAMPLE			
Liberation of Chalcopyrite			
	Locked	Middling	Liberated
-212/+106	1.1	0.2	0.2
-106/+75	5.3	1.0	2.2
-75/+53	6.0	1.7	6.0
CS1-2	7.3	3.9	18.8
CS3	1.7	1.3	6.4
CS4-5	1.1	1.0	11.5
CS6	0.3	1.6	9.5
CS7	0.3	1.7	10.0
Combined	23.0	12.4	64.5

OFFSET

SMS Corrected as % in SAMPLE			
Liberation of Ni Sulphide			
	Locked	Middling	Liberated
-212/+106	1.1	0.8	2.2
-106/+75	1.3	0.5	5.5
-75/+53	1.0	3.1	12.4
CS1-2	1.7	5.4	31.3
CS3	0.6	1.1	7.9
CS4-5	0.2	1.0	9.0
CS6	0.1	1.4	6.2
CS7	0.1	1.2	5.0
Combined	6.0	14.6	79.4

SMS Corrected as % in SAMPLE			
Liberation of Chalcopyrite			
	Locked	Middling	Liberated
-212/+106	1.8	0.4	1.4
-106/+75	4.9	1.4	2.3
-75/+53	3.0	2.0	4.9
CS1-2	6.8	8.9	25.4
CS3	1.8	2.0	9.3
CS4-5	0.9	0.9	9.4
CS6	0.2	1.1	6.9
CS7	0.1	0.6	3.5
Combined	19.6	17.3	63.1

OFFSET LCT Conc

Mass% Ni Sulphide in SAMPLE			
Liberation of Ni Sulphide			
	Locked	Middling	Liberated
CS7	0.2	2.1	6.7
CS6	0.5	5.2	16.9
CS4-5	0.6	2.7	23.7
CS3	0.9	3.1	18.3
CS1-2	0.7	2.5	15.9
Combined	2.9	15.6	81.5

Mass % Chalcopyrite in SAMPLE			
Liberation of Chalcopyrite			
	Locked	Middling	Liberated
CS7	0.3	1.1	14.7
CS6	0.6	2.1	26.7
CS4-5	0.4	1.1	23.8
CS3	0.5	1.6	16.4
CS1-2	0.3	0.9	9.4
Combined	2.2	6.9	91.0

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Appendix C- Ni-Sulphide Locking Characteristics- Feed

NOTE- PMA and SMS are different QEMSCAN measurement modes whereby SMS is a bright phase sear method which results in more sulphide particle statistics.

ROBY

Mass % Locked and Middling Ni Sulphide in Sample		
Ni Sulphide Locking	**Roby SMS**	**Roby PMA**
Ni Sulph/Cpy Binary	**13.98**	21.71
Ni Sulph/Py Binary	**10.45**	12.58
Ni Sulph/Po Binary	**13.42**	4.45
Complex Sulphide	**2.45**	2.72
Ni Sulph/Chlorite Binary	**31.13**	10.88
Ni Sulph/Amphibole Binary	**9.21**	9.25
Ni Sulph/Plag Binary	**13.98**	18.16
Ni Sulph/Opx	**0.50**	0.87
Complex Silicates	**2.90**	8.35
Other	**1.97**	11.01

Ni Sulphide Liberation-Grain Size			
Fraction	**Name**	**SMS Grain Size**	**PMA Grain Size**
-212/+106	**Liberated**	**83.89**	84.73
	Middling		
	Liberated	**10.44**	8.57
-106/+75	**Liberated**	**75.74**	67.93
	Middling	**47.66**	
	Liberated	**12.30**	10.47
-75/+53	**Liberated**	**53.19**	45.25
	Middling	**29.59**	39.76
	Liberated	**10.71**	10.79
CS1-2	**Liberated**	**27.91**	29.29
	Middling	**19.00**	19.63
	Liberated	**8.45**	6.31
CS3	**Liberated**	**15.05**	14.00
	Middling	**11.67**	12.98
	Liberated	**5.30**	3.52
CS4-5	**Liberated**	**10.21**	7.74
	Middling	**6.87**	2.53
	Liberated	**4.53**	2.56
CS6	**Liberated**	**4.55**	3.75
	Middling	**3.54**	2.01
	Liberated	**2.21**	1.32

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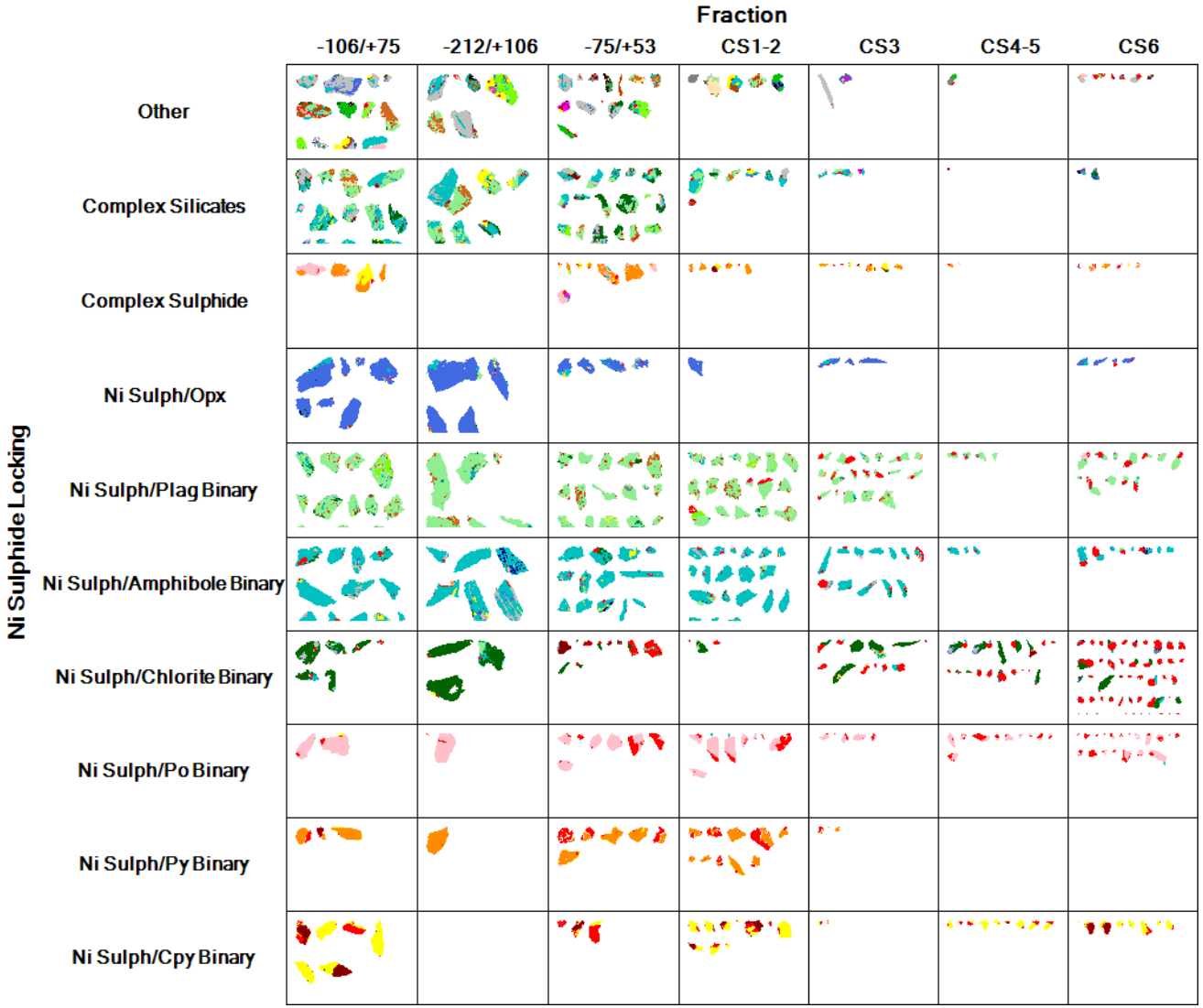

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OFFSET

Mass % Locked and Middling Ni Sulphide in Sample		
Ni Sulphide Locking	**Offset SMS**	**Offset PMA**
Ni Sulph/Cpy Binary	**6.50**	3.22
Ni Sulph/Py Binary	**4.63**	0.54
Ni Sulph/Po Binary	**7.96**	5.69
Complex Sulphide	**6.46**	5.54
Ni Sulph/Chlorite Binary	**43.50**	22.46
Ni Sulph/Amphibole Binary	**13.43**	26.75
Ni Sulph/Plag Binary	**10.41**	26.69
Ni Sulph/Opx	**1.97**	2.08
Complex Silicates	**2.41**	5.20
Other	**2.73**	1.82

Ni Sulphide Liberation-Grain Size			
Fraction	**Name**	**SMS Grain Size**	**PMA Grain Size**
-212/+106	**Liberated**	**98.33**	95.13
	Middling	**36.31**	41.53
	Locked	**12.64**	12.73
-106/+75	**Liberated**	**68.59**	73.84
	Middling	**26.80**	41.56
	Locked	**12.60**	12.32
-75/+53	**Liberated**	**43.92**	52.06
	Middling	**23.17**	40.20
	Locked	**12.03**	8.94
CS1-2	**Liberated**	**29.18**	27.32
	Middling	**18.30**	16.52
	Locked	**10.36**	9.97
CS3	**Liberated**	**15.31**	12.65
	Middling	**11.43**	12.47
	Locked	**6.35**	4.95
CS4-5	**Liberated**	**9.11**	7.57
	Middling	**6.91**	7.75
	Locked	**3.39**	1.89
CS6	**Liberated**	**4.76**	4.28
	Middling	**3.78**	2.75
	Locked	**2.51**	1.83

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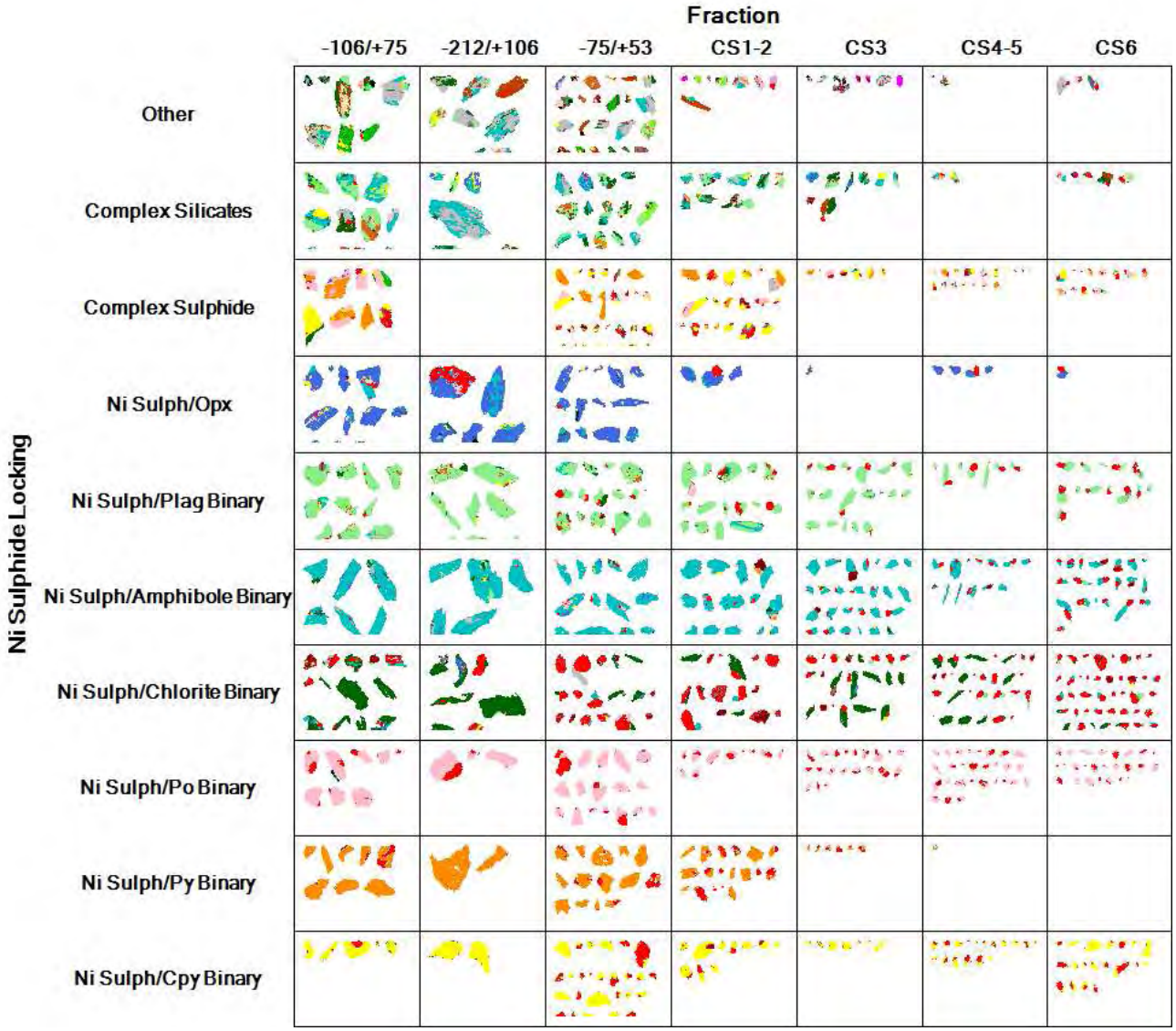

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Appendix D- Size-by-Size Ni Deportment- Feed

	Roby Feed Ni Deportment - Mass % in Fraction								
Mineral	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**	**Combined**
Pentlandite	**19.11**	**32.89**	**23.91**	**40.87**	**44.62**	**24.11**	**24.06**	**15.82**	**32.14**
Millerite	**48.72**	**1.32**	**2.41**	**2.54**	**26.42**	**23.77**	**1.14**	**0.17**	**19.42**
Pyrrhotite	0.00	0.34	0.71	0.53	0.33	0.23	0.81	0.24	**0.37**
Pyrite	0.01	0.01	0.08	0.08	0.16	0.12	0.14	0.05	**0.10**
Biotite	0.11	0.88	0.68	0.55	0.42	1.02	1.47	1.58	**0.64**
Orthopyroxene	2.52	5.77	3.13	2.55	2.06	3.97	5.93	6.45	**3.24**
Amphibole	10.09	20.70	25.62	23.22	17.00	25.44	38.24	35.42	**21.00**
Talc	0.79	0.92	1.73	1.35	0.76	1.48	2.30	2.48	**1.20**
Chlorite	18.65	37.16	41.73	28.31	8.23	19.85	25.92	37.78	**21.91**
	100.00	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**

	OFFSET Feed Ni Deportment - Mass % in Fraction								
Mineral	**CS7**	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**-75/+53**	**-106/+75**	**-212/+106**	**Combined**
Pentlandite	**55.10**	**50.58**	**40.17**	**53.49**	**65.66**	**52.85**	**43.70**	**41.84**	**54.06**
Millerite	**0.00**	**0.00**	**13.72**	**0.29**	**8.69**	**11.46**	**4.40**	**0.34**	**6.61**
Pyrrhotite	0.00	0.54	0.37	0.37	0.79	0.54	0.71	0.33	**0.53**
Pyrite	0.00	0.01	0.06	0.08	0.23	0.12	0.22	0.05	**0.13**
Biotite	1.17	1.07	0.87	1.03	0.52	1.02	1.28	1.48	**0.91**
Orthopyroxene	3.82	3.93	1.95	2.18	1.50	2.13	3.62	4.18	**2.45**
Amphibole	14.23	18.39	16.74	18.83	14.79	16.08	24.87	24.13	**17.22**
Talc	4.88	1.01	1.86	1.57	0.66	1.30	2.50	2.71	**1.67**
Chlorite	20.80	24.46	24.25	22.16	7.14	14.50	18.70	24.94	**16.41**
	100.00	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**

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Appendix E- PGM Mineralogy Data

Grain Size Distribution

PGM Grain Size Distribution (µm)			
Mass% PGM in Each Size Class	**Roby Feed**	**Offset Feed**	**Offset LCT Conc**
<2µm	7.9	6.3	0.4
<5µm	25.4	31.0	13.6
<10µm	29.4	18.6	43.2
<15µm	19.1	12.4	20.8
<20µm	6.1	15.9	9.9
<25µm	6.0	15.8	5.3
>=25µm	6.1	0.0	6.8
Total	*100.0*	*100.0*	*100.0*

Size-By Size Offset LCT PGM Liberation

	Offset LCT Conc - PGM Liberation (Mass% PGM)					
	CS7	**CS6**	**CS4-5**	**CS3**	**CS1-2**	**TOTAL**
Liberated	26.2	17.6	13.1	12.8	13.9	84
Middling	2.2	1.6	3.9	2.6	4.7	15
Locked	0.1	0.1	0.3	0.6	0.4	2
Total	*28.5*	*19.3*	*17.3*	*16.0*	*19.0*	*100*

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Appendix F- Roby Grind Time Size Validation

15 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Robbie Zone
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 2

Measured: Wed Mar 3 2010 1:16PM
Analysed: Wed Mar 3 2010 1:16PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 53.2 %
Residual: 0.257 %

Result Statistics

Distribution Type: Volume

Concentration = 0.1094 %Vol
Span = 4.503E+00

Density = 1.000 g / cub. cm
Uniformity = 1.375E+00

Specific S.A. = 0.5233 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	67.29	27.77	13.05	7.85
D[3, N]		11.47	5.75	3.84
D[2, N]			2.88	2.22
D[1, N]				1.71

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	67.29	76.983	1.7746	3.1321
Surface	11.47	25.298	5.9500	49.6145
Length	2.88	4.974	16.1364	521.9683
Number	1.71	1.416	14.5234	862.0726

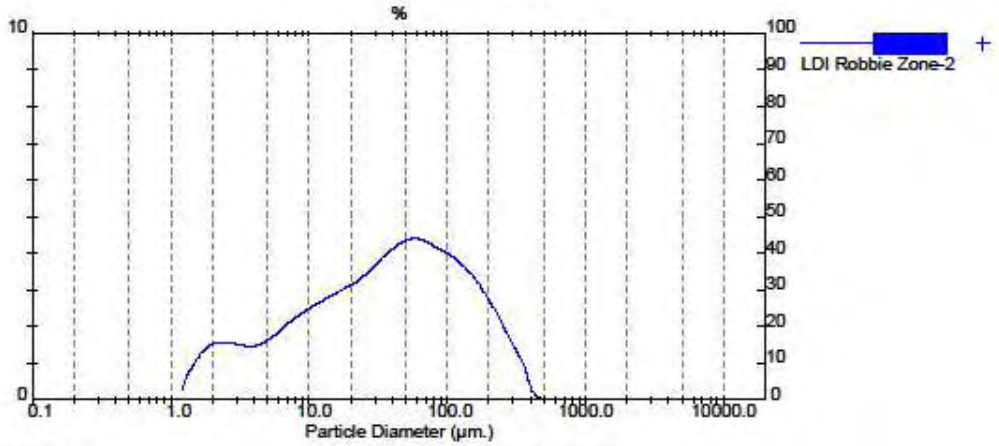
Distribution Percentiles (um) – Volume

Percentile	Size	Percentile	Size
10.0 %	3.81	75.0 %	94.09
20.0 %	8.76	80.0 %	114.13
50.0 %	38.43	90.0 %	176.87

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	58.39	2	2.38

LDI Robbie Zone



Malvern Instruments Ltd.
Malvern, UK
Tel:=+[44] (0)1684-892456 Fax:+[44] (0)1684-892789

Mastersizer X long bed Ver. 2.19
Serial Number: 6555

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21 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Robbie Zone
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 2

Measured: Wed Mar 3 2010 1:28PM
Analysed: Wed Mar 3 2010 1:29PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 65.3 %
Residual: 0.257 %

Result Statistics

Distribution Type: Volume

Concentration = 0.1181 %Vol
Span = 4.126E+00

Density = 1.000 g / cub. cm
Uniformity = 1.349E+00

Specific S.A. = 0.6744 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	44.32	19.86	10.17	6.44
D[3, N]		8.90	4.87	3.39
D[2, N]			2.66	2.09
D[1, N]				1.64

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	44.32	53.845	2.2838	5.9961
Surface	8.90	17.753	6.1044	57.4533
Length	2.66	4.075	13.2846	377.5727
Number	1.64	1.296	12.0475	535.2108

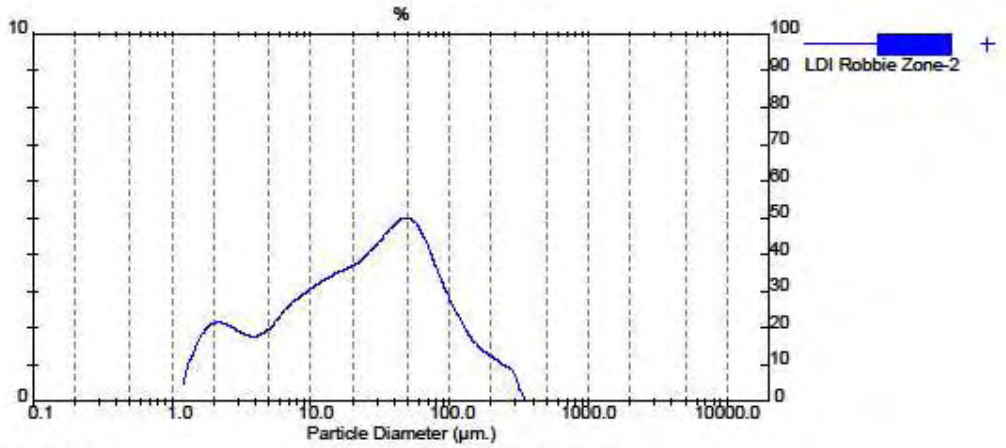
Distribution Percentiles (um) – Volume

Percentile	Size	Percentile	Size
10.0 %	2.85	75.0 %	58.20
20.0 %	6.26	80.0 %	68.77
50.0 %	25.54	90.0 %	108.27

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	48.57	2	2.18

LDI Robbie ZOne



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Mastersizer X long bed Ver. 2.19
Serial Number: 6555

p. 107
03 Mar 10 13:29

26 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Robbie Zone
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 3

Measured: Wed Mar 3 2010 1:42PM
Analysed: Wed Mar 3 2010 1:42PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 45.8 %
Residual: 0.198 %

Result Statistics

Distribution Type: Volume

Concentration = 0.0687 %Vol
Span = 3.569E+00

Density = 1.000 g / cub. cm
Uniformity = 1.284E+00

Specific S.A. = 0.6725 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	39.45	18.76	9.94	6.38
D[3, N]		8.92	4.99	3.47
D[2, N]			2.79	2.17
D[1, N]				1.68

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	39.45	52.067	3.2108	13.0995
Surface	8.92	16.504	6.6899	81.1697
Length	2.79	4.137	11.5487	325.8520
Number	1.68	1.366	10.8857	407.2520

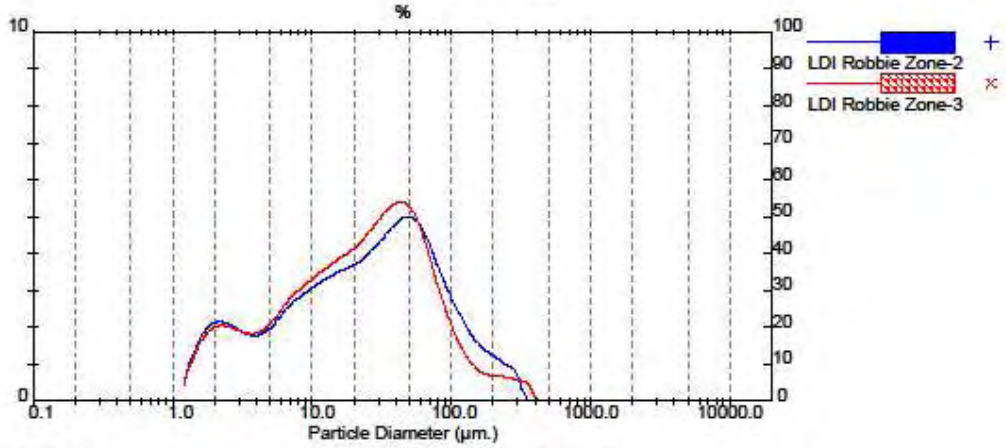
Distribution Percentiles (um) – Volume

Percentile	Size	Percentile	Size
10.0 %	2.98	75.0 %	49.67
20.0 %	6.34	80.0 %	57.71
50.0 %	23.29	90.0 %	86.10

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	43.13	2	2.26

LDI Robbie ZOne



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Mastersizer X long bed Ver. 2.19
Serial Number: 6555

p. 109
03 Mar 10 13:43

36 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Robbie Zone
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 4

Measured: Wed Mar 3 2010 1:54PM
Analysed: Wed Mar 3 2010 1:54PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 44.8 %
Residual: 0.176 %

Result Statistics

Distribution Type: Volume
Concentration = 0.0602 %Vol
Span = 3.300E+00
Density = 1.000 g / cub. cm
Uniformity = 1.285E+00
Specific S.A. = 0.7436 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	32.27	16.14	8.98	5.91
D[3, N]		8.07	4.74	3.36
D[2, N]			2.78	2.17
D[1, N]				1.69

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	32.27	47.553	4.1035	20.4288
Surface	8.07	13.975	7.8398	123.5542
Length	2.78	3.835	10.2855	306.4351
Number	1.69	1.359	9.4602	297.0978

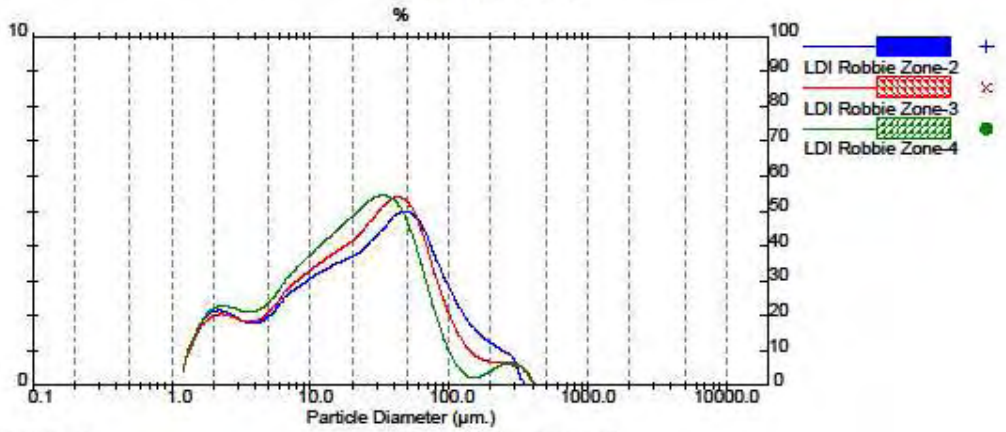
Distribution Percentiles (um) – Volume

Percentile	Size	Percentile	Size
10.0 %	2.77	75.0 %	39.15
20.0 %	5.52	80.0 %	45.31
50.0 %	18.82	90.0 %	64.88

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	33.46	3	278.01
2	2.34		

LDI Robbie ZOne



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Mastersizer X long bed Ver. 2.19
Serial Number: 6555

p. 111
03 Mar 10 13:57

43 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Robbie Zone
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 5

Measured: Wed Mar 3 2010 2:04PM
Analysed: Wed Mar 3 2010 2:04PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 46.8 %
Residual: 0.174 %

Result Statistics

Distribution Type: Volume
Concentration = 0.0596 %Vol
Span = 3.325E+00
Density = 1.000 g / cub. cm
Uniformity = 1.322E+00
Specific S.A. = 0.7966 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	29.32	14.86	8.46	5.65
D[3, N]		7.53	4.54	3.26
D[2, N]			2.74	2.15
D[1, N]				1.68

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	29.32	46.048	4.5224	24.3345
Surface	7.53	12.811	8.7082	156.9869
Length	2.74	3.623	10.0177	319.7043
Number	1.68	1.333	8.8632	262.8358

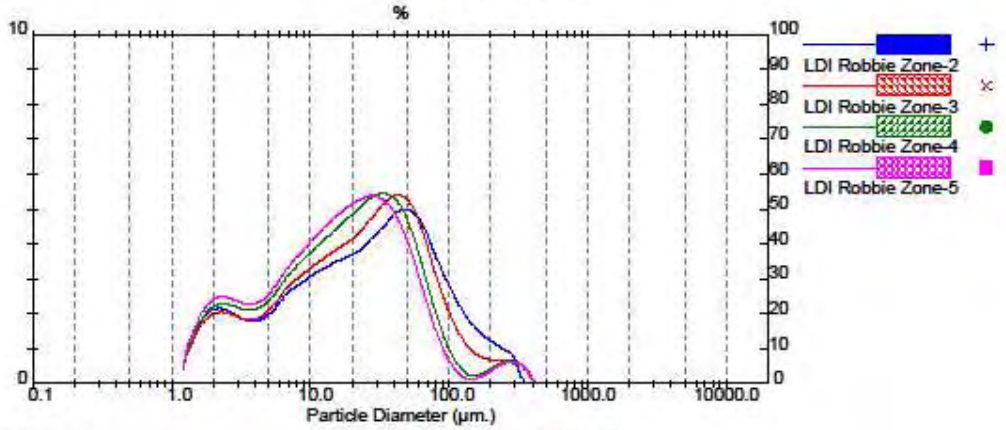
Distribution Percentiles (um) -- Volume

Percentile	Size	Percentile	Size
10.0 %	2.61	75.0 %	34.65
20.0 %	5.00	80.0 %	40.29
50.0 %	16.69	90.0 %	58.10

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	27.64	3	283.54
2	2.32		

LDI Robbie ZOne



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Serial Number: 6555

p. 113
03 Mar 10 14:04

Appendix G- Offset Grind Time Size Validation

15 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Offset 5min
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 4

Measured: Mon Feb 22 2010 1:41PM
Analysed: Mon Feb 22 2010 1:41PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 37.3 %
Residual: 0.463 %

Result Statistics

Distribution Type: Volume

Concentration = 0.0605 %Vol
Span = 3.910E+00

Density = 1.000 g / cub. cm
Uniformity = 1.295E+00

Specific S.A. = 0.5813 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	52.66	23.31	11.61	7.20
D[3, N]		10.32	5.45	3.71
D[2, N]			2.88	2.23
D[1, N]				1.72

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	52.66	64.787	2.4765	7.4699
Surface	10.32	20.903	6.2747	63.2292
Length	2.88	4.628	13.6705	411.7343
Number	1.72	1.411	12.7303	603.2554

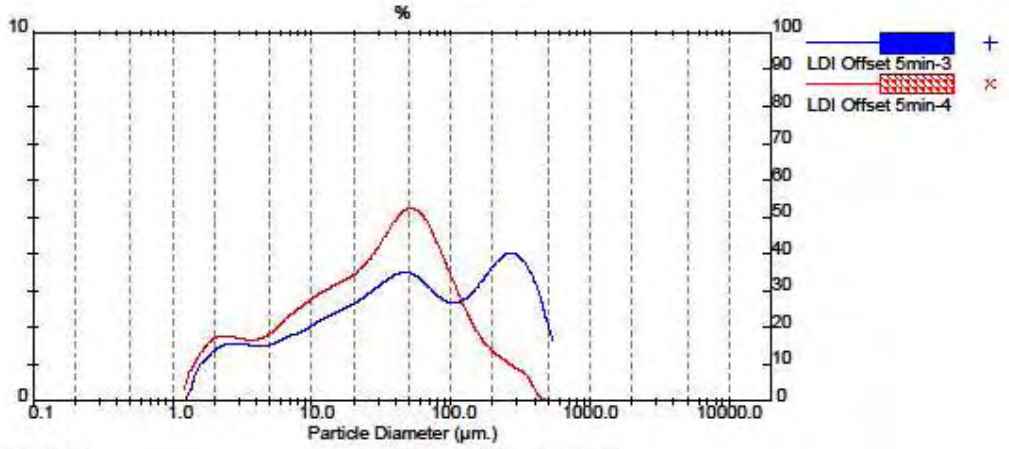
Distribution Percentiles (um) -- Volume

Percentile	Size	Percentile	Size
10.0 %	3.41	75.0 %	67.78
20.0 %	7.55	80.0 %	80.16
50.0 %	31.33	90.0 %	125.90

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	52.10	2	2.42

LDI Offset



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Mastersizer X long bed Ver. 2.19
Serial Number: 6555

p. 7
22 Feb 10 13:43

19 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Offset 5min
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 4

Measured: Mon Feb 22 2010 1:41PM
Analysed: Mon Feb 22 2010 1:41PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 37.3 %
Residual: 0.463 %

Result Statistics

Distribution Type: Volume
Concentration = 0.0605 %Vol
Span = 3.910E+00
Density = 1.000 g / cub. cm
Uniformity = 1.295E+00
Specific S.A. = 0.5813 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	52.66	23.31	11.61	7.20
D[3, N]		10.32	5.45	3.71
D[2, N]			2.88	2.23
D[1, N]				1.72

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	52.66	64.787	2.4765	7.4699
Surface	10.32	20.903	6.2747	63.2292
Length	2.88	4.628	13.6705	411.7343
Number	1.72	1.411	12.7303	603.2554

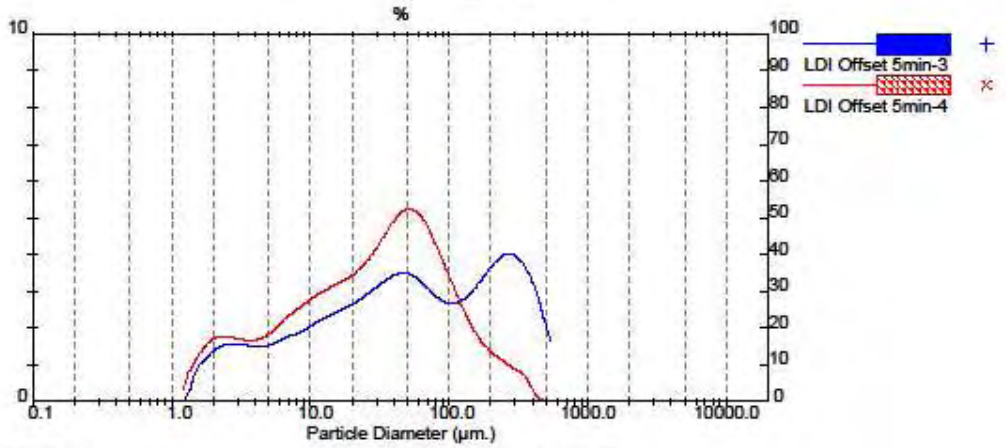
Distribution Percentiles (um) – Volume

Percentile	Size	Percentile	Size
10.0 %	3.41	75.0 %	67.78
20.0 %	7.55	80.0 %	80.16
50.0 %	31.33	90.0 %	125.90

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	52.10	2	2.42

LDI Offset



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Mastersizer X long bed Ver. 2.19
Serial Number: 6555

p. 7
22 Feb 10 13:43

21 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Offset 5min
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 7

Measured: Mon Feb 22 2010 2:04PM
Analysed: Mon Feb 22 2010 2:04PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 58.2 %
Residual: 0.181 %

Result Statistics

Distribution Type: Volume

Concentration = 0.0906 %Vol
Span = 3.827E+00

Density = 1.000 g / cub. cm
Uniformity = 1.481E+00

Specific S.A. = 0.7246 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	42.25	18.70	9.71	6.22
D[3, N]		8.28	4.66	3.28
D[2, N]			2.62	2.07
D[1, N]				1.63

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	42.25	67.557	4.4594	25.1181
Surface	8.28	16.771	9.5431	188.9774
Length	2.62	3.851	13.6917	582.9121
Number	1.63	1.271	11.2690	499.0948

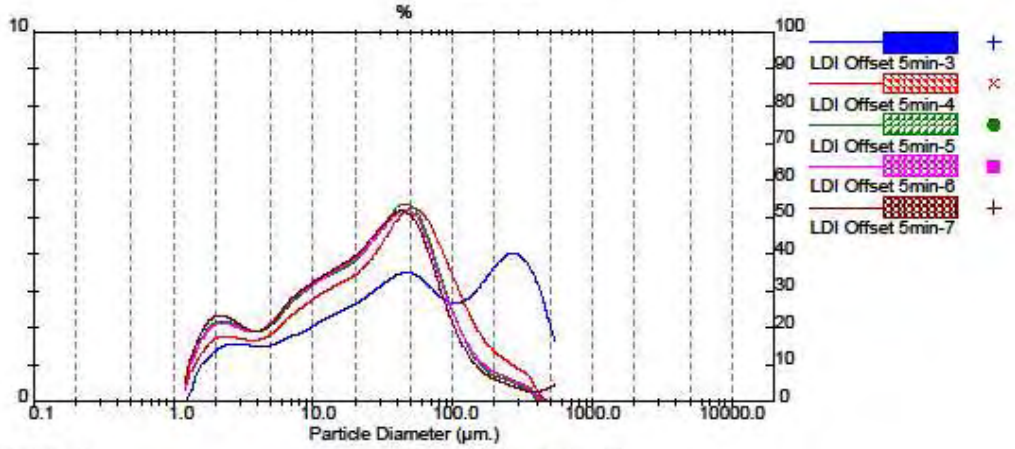
Distribution Percentiles (um) -- Volume

Percentile	Size	Percentile	Size
10.0 %	2.67	75.0 %	50.03
20.0 %	5.64	80.0 %	58.53
50.0 %	22.53	90.0 %	88.88

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	43.32	2	2.17

LDI Offset



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Serial Number: 6555

p. 11
22 Feb 10 14:04

26 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Offset 5min
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 8

Measured: Mon Feb 22 2010 2:13PM
Analysed: Mon Feb 22 2010 2:13PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 53.3 %
Residual: 0.171 %

Result Statistics

Distribution Type: Volume
Concentration = 0.0756 %Vol
Span = 3.616E+00
Density = 1.000 g / cub. cm
Uniformity = 1.387E+00
Specific S.A. = 0.7581 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	35.72	16.81	9.09	5.93
D[3, N]		7.91	4.58	3.26
D[2, N]			2.65	2.09
D[1, N]				1.65

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	35.72	51.939	3.6501	16.0470
Surface	7.91	14.835	7.8802	116.6486
Length	2.65	3.737	11.8941	388.1135
Number	1.65	1.288	10.2812	381.7861

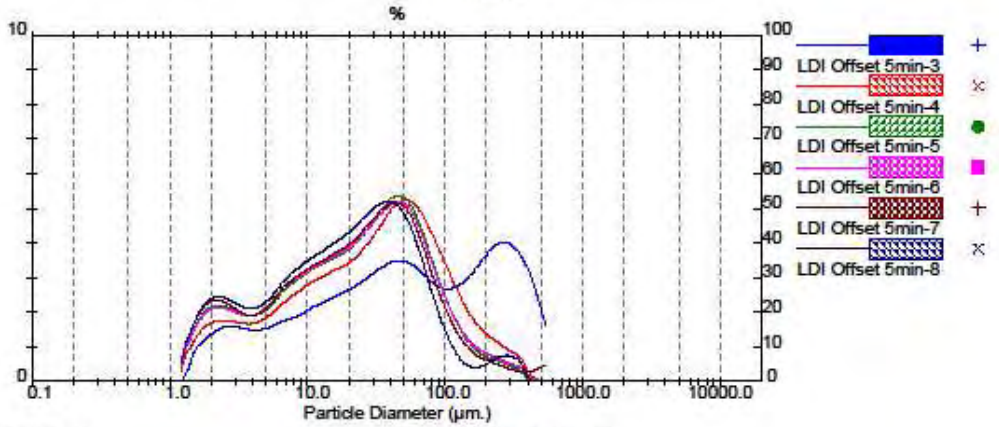
Distribution Percentiles (um) -- Volume

Percentile	Size	Percentile	Size
10.0 %	2.62	75.0 %	43.64
20.0 %	5.25	80.0 %	50.86
50.0 %	19.92	90.0 %	74.64

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	39.32	3	283.52
2	2.26		

LDI Offset



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Serial Number: 6555

p. 13
22 Feb 10 14:17

34 Minutes

MALVERN MASTERSIZER

Result Derived Diameters Report

Sample Details

Sample ID: LDI Offset 5min
Sample File: (Result Not Saved)
Sample Path: C:\SIZERX\DATA\
Sample Notes:

Run Number: 8

Measured: Mon Feb 22 2010 2:13PM
Analysed: Mon Feb 22 2010 2:13PM
Result Source: Analysed

System Details

Range Lens: 300 mm
Presentation: 2OHD
Analysis Model: Polydisperse
Modifications: None

Beam Length: 2.40 mm
[Particle R.I. = (1.5295, 0.1000); Dispersant R.I. = 1.3300]

Sampler: MS17

Obscuration: 53.3 %
Residual: 0.171 %

Result Statistics

Distribution Type: Volume

Concentration = 0.0756 %Vol
Span = 3.616E+00

Density = 1.000 g / cub. cm
Uniformity = 1.367E+00

Specific S.A. = 0.7581 sq. m / g

A.S.T.M Derived Diameters (um)

N	3	2	1	0
D[4, N]	35.72	16.81	9.09	5.93
D[3, N]		7.91	4.58	3.26
D[2, N]			2.65	2.09
D[1, N]				1.65

Distribution Moments

	Mean	Stan. Dev.	Skewness	Kurtosis
Volume	35.72	51.939	3.6501	16.0470
Surface	7.91	14.835	7.8802	116.6486
Length	2.65	3.737	11.8941	388.1135
Number	1.65	1.288	10.2812	381.7861

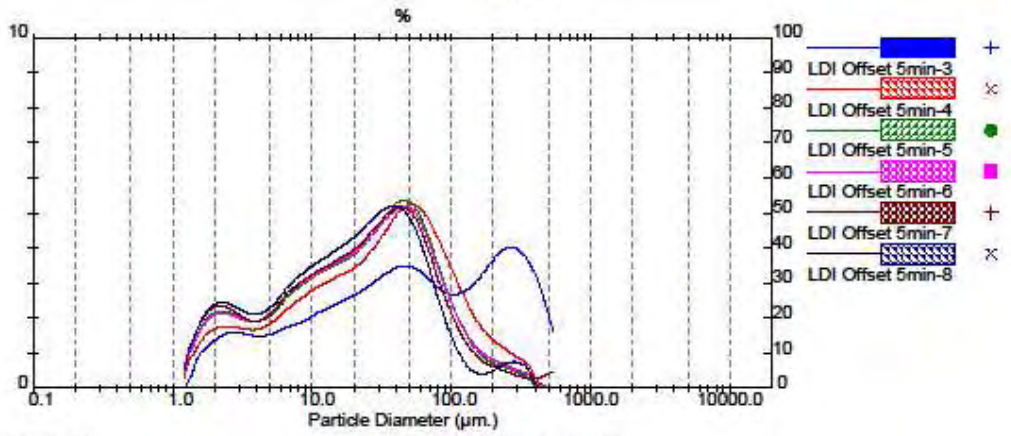
Distribution Percentiles (um) -- Volume

Percentile	Size	Percentile	Size
10.0 %	2.62	75.0 %	43.64
20.0 %	5.25	80.0 %	50.86
50.0 %	19.92	90.0 %	74.64

Distribution Modal Sizes (um)

Mode	Size	Mode	Size
1	39.32	3	283.52
2	2.26		

LDI Offset



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Serial Number: 6555

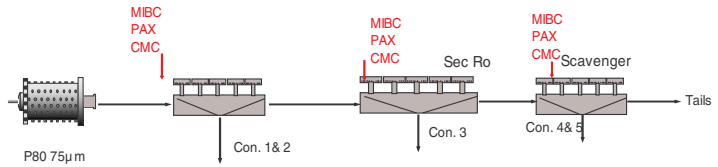
p. 13
22 Feb 10 14:17

Appendix H- Roby Rougher-Scavenger Mass Balances

Baseline 75µm and 100% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet- Baseline (@ 100% CMC)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork		
Project #	5009817		
Feed Sample	Roby Zone Composite		
Objectives			
Target Grind	P80 75 microns	100% CMC	
MDL #s			
Charge Mass	2500	grams	
No. of Charges	1	charges	



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2498.62	99.94	0.12	0.07	9.57	0.43	6.72	0.31	101.9	102.6	78.4	138.0	109.8	83.8
Call Factor			101.92	102.57	78.41	137.99	109.84	83.79						
Rougher Con 1	69.86	2.80	1.52	2.00	16.10	10.45	179.50	7.60	35.76	77.73	4.71	68.34	74.70	68.58
Rougher Con 2	90.02	3.60	0.23	0.14	14.50	0.89	12.85	0.47	7.00	6.88	5.46	7.50	6.89	5.46
Rougher Con 3	44.61	1.79	0.15	0.07	14.00	0.46	6.05	0.30	2.28	1.74	2.61	1.92	1.61	1.73
Rougher Con 4	45.13	1.81	0.13	0.04	13.15	0.28	3.18	0.17	1.93	1.11	2.48	1.18	0.85	0.99
Rougher Tail	2249.00	90.01	0.07	0.01	9.00	0.10	1.19	0.08	53.02	12.54	84.73	21.05	15.94	23.24
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	69.86	2.80	1.52	2.00	16.10	10.45	179.50	7.60	35.8	77.7	4.7	68.3	74.7	68.6
Rougher Con 2	159.88	6.40	0.79	0.95	15.20	5.07	85.67	3.59	42.8	84.6	10.2	75.8	81.6	74.0
Rougher Con 3	204.49	8.18	0.65	0.76	14.94	4.06	68.30	2.87	45.0	86.3	12.8	77.8	83.2	75.8
Rougher Con 4	**249.62**	**9.99**	**0.56**	**0.63**	**14.61**	**3.38**	**56.53**	**2.38**	**47.0**	**87.5**	**15.3**	**78.9**	**84.1**	**76.8**
Ro 1 Tail	2428.76	97.15	0.08	0.02	9.38	0.14	1.75	0.10	64.2	22.3	95.3	31.7	25.3	31.4
Ro 2 Tail	2338.74	93.55	0.07	0.01	9.18	0.11	1.32	0.09	57.2	15.4	89.8	24.2	18.4	26.0
Ro 3 Tail	2294.13	91.77	0.07	0.01	9.09	0.10	1.23	0.08	55.0	13.7	87.2	22.2	16.8	24.2
Ro 4 Tail	2249.00	89.96	0.07	0.01	9.00	0.10	1.19	0.08	53.0	12.5	84.7	21.1	15.9	23.2

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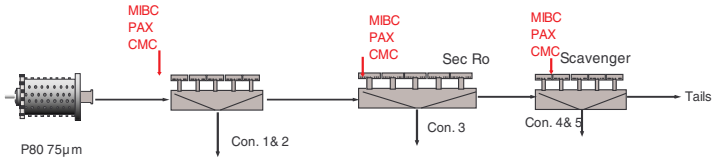
75µm and 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (@ 56% of CMC)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Objectives		
Target Grind	P80 75 microns	56% CMC
MDL #s		
Charge Mass	2500	grams
No. of Charges	1	charges



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2491.42	99.66	0.12	0.07	9.69	0.38	6.24	0.36	102.0	96.2	79.4	122.2	102.0	98.1
Call Factor			101.97	96.24	79.43	122.21	101.98	98.08						
Rougher Con 1	111.58	4.48	0.99	1.19	15.25	6.01	108.00	6.01	37.32	79.39	7.07	71.29	77.76	74.42
Rougher Con 2	166.64	6.69	0.15	0.07	13.80	0.43	5.97	0.43	8.35	6.58	9.56	7.62	6.42	7.95
Rougher Con 3	130.82	5.25	0.10	0.03	13.45	0.18	2.29	0.18	4.56	2.03	7.31	2.50	1.93	2.61
Rougher Con 4	103.06	4.14	0.10	0.02	13.15	0.16	1.86	0.16	3.52	1.36	5.63	1.75	1.24	1.83
Rougher Tail	1979.32	79.45	0.07	0.01	8.56	0.08	0.99	0.06	46.24	10.65	70.42	16.83	12.65	13.18
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	111.58	4.48	0.99	1.19	15.25	6.01	108.00	6.01	37.3	79.4	7.1	71.3	77.8	74.4
Rougher Con 2	278.22	11.17	0.48	0.52	14.38	2.67	46.89	2.67	45.7	86.0	16.6	78.9	84.2	82.4
Rougher Con 3	409.04	16.42	0.36	0.36	14.08	1.87	32.63	1.87	50.2	88.0	23.9	81.4	86.1	85.0
Rougher Con 4	**512.10**	**20.55**	**0.31**	**0.29**	**13.90**	**1.53**	**26.43**	**1.53**	**53.8**	**89.3**	**29.6**	**83.2**	**87.4**	**86.8**
Ro 1 Tail	2379.84	95.19	0.08	0.01	9.43	0.11	1.45	0.10	62.7	20.6	92.9	28.7	22.2	25.6
Ro 2 Tail	2213.20	88.53	0.07	0.01	9.09	0.09	1.11	0.07	54.3	14.0	83.4	21.1	15.8	17.6
Ro 3 Tail	2082.38	83.30	0.07	0.01	8.82	0.08	1.04	0.07	49.8	12.0	76.1	18.6	13.9	15.0
Ro 4 Tail	1979.32	79.17	0.07	0.01	8.59	0.08	0.99	0.06	46.2	10.7	70.4	16.8	12.6	13.2

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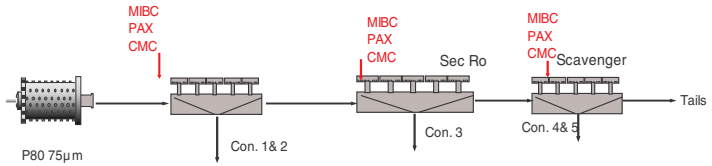
75µm with Aero 3477 and 100% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (Aero 3477)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Objectives		
Target Grind	P80 75 microns	Aero 3477
MDL #s		100% CMC
Charge Mass	2500 grams	
No. of Charges	1 charges	



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2505.84	100.23	0.12	0.07	9.76	0.34	6.28	0.27	102.9	94.4	80.0	109.2	102.5	74.3
Call Factor			102.95	94.43	80.00	109.25	102.54	74.29						
Rougher Con 1	60.10	2.40	1.60	2.15	16.10	8.98	184.00	7.14	31.78	77.83	3.95	63.45	70.16	62.15
Rougher Con 2	97.44	3.89	0.32	0.17	14.50	1.06	17.90	0.61	10.47	10.15	5.76	12.14	11.07	8.61
Rougher Con 3	93.89	3.75	0.12	0.06	13.40	0.37	5.02	0.33	3.80	3.17	5.13	4.08	2.99	4.49
Rougher Con 4	100.55	4.01	0.10	0.04	12.25	0.22	2.90	0.20	3.27	2.36	5.02	2.60	1.85	2.91
Rougher Tail	2153.86	85.95	0.07	0.01	9.12	0.07	1.02	0.07	50.69	6.49	80.13	17.72	13.94	21.84
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Rougher Con 1	60.10	2.40	1.60	2.15	16.10	8.98	184.00	7.14	31.8	77.8	3.9	63.4	70.2	62.2
Rougher Con 2	157.54	6.29	0.81	0.93	15.11	4.08	81.27	3.10	42.2	88.0	9.7	75.6	81.2	70.8
Rougher Con 3	251.43	10.03	0.55	0.60	14.47	2.70	52.79	2.07	46.0	91.2	14.8	79.7	84.2	75.2
Rougher Con 4	**351.98**	**14.05**	**0.42**	**0.44**	**13.84**	**1.99**	**38.54**	**1.53**	**49.3**	**93.5**	**19.9**	**82.3**	**86.1**	**78.2**
Ro 1 Tail	2445.74	97.83	0.08	0.02	9.60	0.13	1.92	0.11	68.2	22.2	96.1	36.6	29.8	37.8
Ro 2 Tail	2348.30	93.93	0.07	0.01	9.40	0.09	1.26	0.09	57.8	12.0	90.3	24.4	18.8	29.2
Ro 3 Tail	2254.41	90.18	0.07	0.01	9.24	0.08	1.10	0.08	54.0	8.8	85.2	20.3	15.8	24.8
Ro 4 Tail	2153.86	86.15	0.07	0.00	9.10	0.07	1.02	0.07	50.7	6.5	80.1	17.7	13.9	21.8

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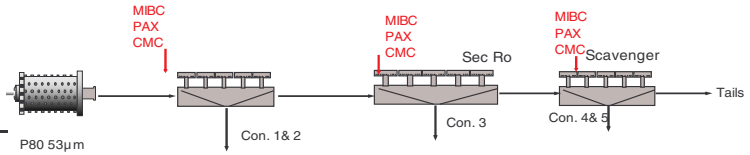
53µm with 100% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (53 µm)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork		
Project #	5009817		
Feed Sample	Roby Zone Composite		
Objectives			
Target Grind	P80 53 microns	100% CMC	
MDL #s			
Charge Mass	2500	grams	
No. of Charges	1	charges	



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2503.46	100.14	0.12	0.07	10.25	0.38	6.52	0.30	104.7	98.6	84.0	124.1	106.5	80.9
Call Factor			104.69	98.60	83.98	124.10	106.46	80.86						
Rougher Con 1	92.94	3.71	1.22	1.52	15.50	7.50	138.00	6.19	36.88	81.65	5.61	72.27	78.53	76.71
Rougher Con 2	98.70	3.94	0.25	0.12	14.10	0.83	12.55	0.46	8.12	6.96	5.42	8.49	7.58	6.05
Rougher Con 3	68.81	2.75	0.13	0.04	13.70	0.31	3.50	0.18	2.90	1.63	3.67	2.21	1.47	1.65
Rougher Con 4	59.56	2.38	0.11	0.03	13.35	0.19	2.15	0.13	2.18	0.93	3.10	1.17	0.78	1.03
Rougher Tail	2183.45	87.22	0.07	0.01	9.67	0.07	0.87	0.05	49.92	8.83	82.21	15.85	11.63	14.56

Cumulative	Cum Weight		Cumulative Grade						Cum. Recovery					
Product	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Rougher Con 1	92.94	3.71	1.22	1.52	15.50	7.50	138.00	6.19	36.9	81.6	5.6	72.3	78.5	76.7
Rougher Con 2	191.64	7.66	0.72	0.80	14.78	4.06	73.39	3.24	45.0	88.6	11.0	80.8	86.1	82.8
Rougher Con 3	260.45	10.40	0.56	0.60	14.49	3.07	54.93	2.43	47.9	90.2	14.7	83.0	87.6	84.4
Rougher Con 4	**320.01**	**12.78**	**0.48**	**0.49**	**14.28**	**2.54**	**45.10**	**2.00**	**50.1**	**91.2**	**17.8**	**84.2**	**88.4**	**85.4**
Ro 1 Tail	2410.52	96.42	0.08	0.01	10.04	0.11	1.45	0.07	63.1	18.4	94.4	27.7	21.5	23.3
Ro 2 Tail	2311.82	92.47	0.07	0.01	9.87	0.08	0.98	0.06	55.0	11.4	89.0	19.2	13.9	17.2
Ro 3 Tail	2243.01	89.72	0.07	0.01	9.75	0.07	0.90	0.05	52.1	9.8	85.3	17.0	12.4	15.6
Ro 4 Tail	2183.45	87.34	0.07	0.01	9.66	0.07	0.87	0.05	49.9	8.8	82.2	15.8	11.6	14.6

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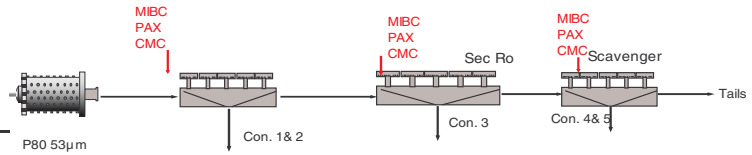
53µm with 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (53 µm @ 56% of CMC)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork		
Project #	5009817		
Feed Sample	Roby Zone Composite		
Objectives			
Target Grind	P80 53 microns	56% CMC	
MDL #s			
Charge Mass	2500	grams	
No. of Charges	1	charges	



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2498.92	99.96	0.13	0.07	12.32	0.35	6.21	0.29	114.3	105.4	101.0	114.1	101.4	79.6
Call Factor			114.34	105.37	100.99	114.14	101.41	79.64						
Rougher Con 1	133.31	5.33	0.93	1.15	14.70	4.91	92.90	4.16	37.25	83.21	6.37	74.06	79.89	75.35
Rougher Con 2	160.24	6.41	0.16	0.06	13.75	0.42	5.96	0.25	7.69	5.57	7.16	7.61	6.16	5.44
Rougher Con 3	138.24	5.53	0.12	0.03	13.30	0.19	1.99	0.10	5.06	2.48	5.97	2.97	1.77	1.88
Rougher Con 4	116.92	4.68	0.11	0.02	13.40	0.16	1.63	0.09	3.75	1.33	5.09	2.12	1.23	1.43
Rougher Tail	1950.21	78.04	0.08	0.01	11.90	0.06	0.87	0.06	46.24	7.41	75.41	13.24	10.95	15.90
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	133.31	5.33	0.93	1.15	14.70	4.91	92.90	4.16	37.2	83.2	6.4	74.1	79.9	75.4
Rougher Con 2	293.55	11.75	0.51	0.56	14.18	2.46	45.44	2.03	44.9	88.8	13.5	81.7	86.1	80.8
Rougher Con 3	431.79	17.28	0.39	0.39	13.90	1.73	31.53	1.41	50.0	91.3	19.5	84.6	87.8	82.7
Rougher Con 4	**548.71**	**21.96**	**0.33**	**0.31**	**13.79**	**1.40**	**25.16**	**1.13**	**53.8**	**92.6**	**24.6**	**86.8**	**89.1**	**84.1**
Ro 1 Tail	2365.61	94.62	0.09	0.01	12.19	0.10	1.32	0.08	62.8	16.8	93.6	25.9	20.1	24.6
Ro 2 Tail	2205.37	88.21	0.08	0.01	12.07	0.07	0.98	0.06	55.1	11.2	86.5	18.3	13.9	19.2
Ro 3 Tail	2067.13	82.69	0.08	0.01	11.99	0.07	0.91	0.06	50.0	8.7	80.5	15.4	12.2	17.3
Ro 4 Tail	1950.21	78.01	0.08	0.01	11.91	0.06	0.87	0.06	46.2	7.4	75.4	13.2	10.9	15.9

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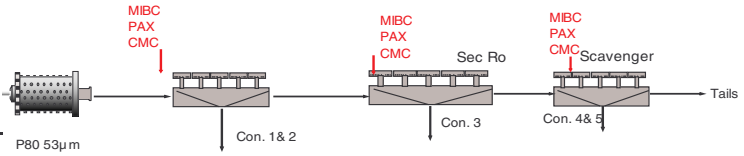
38µm with 100% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (38 µm @ 100% of CMC)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork		
Project #	5009817		
Feed Sample	Roby Zone Composite		
Objectives			
Target Grind	P80 **38** microns	100% CMC	
MDL #s			
Charge Mass	2500	grams	
No. of Charges	1	charges	



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2510.44	100.42	0.12	0.07	10.86	0.38	6.11	0.29	105.4	95.2	89.0	123.4	99.8	79.7
Call Factor			105.36	95.20	89.03	123.41	99.84	79.72						
Rougher Con 1	84.12	3.35	1.15	1.54	15.10	7.83	137.50	6.69	31.22	76.86	4.64	68.30	75.09	75.69
Rougher Con 2	128.63	5.12	0.26	0.09	13.35	0.77	10.50	0.36	10.75	6.81	6.27	10.27	8.77	6.23
Rougher Con 3	156.00	6.21	0.16	0.08	12.40	0.44	6.57	0.27	7.90	7.71	7.06	7.12	6.65	5.66
Rougher Con 4	95.65	3.81	0.11	0.02	12.35	0.16	2.02	0.11	3.24	1.31	4.31	1.59	1.25	1.42
Rougher Tail	2046.04	81.50	0.07	0.01	10.40	0.06	0.62	0.04	46.88	7.31	77.71	12.73	8.24	11.01
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	84.12	3.35	1.15	1.54	15.10	7.83	137.50	6.69	31.2	76.9	4.6	68.3	75.1	75.7
Rougher Con 2	212.75	8.47	0.61	0.66	14.04	3.56	60.71	2.86	42.0	83.7	10.9	78.6	83.9	81.9
Rougher Con 3	368.75	14.69	0.42	0.42	13.35	2.24	37.81	1.77	49.9	91.4	18.0	85.7	90.5	87.6
Rougher Con 4	**464.40**	**18.50**	**0.35**	**0.34**	**13.14**	**1.81**	**30.44**	**1.42**	**53.1**	**92.7**	**22.3**	**87.3**	**91.8**	**89.0**
Ro 1 Tail	2426.32	97.05	0.09	0.02	10.72	0.13	1.57	0.07	68.8	23.1	95.4	31.7	24.9	24.3
Ro 2 Tail	2297.69	91.91	0.08	0.01	10.57	0.09	1.08	0.06	58.0	16.3	89.1	21.4	16.1	18.1
Ro 3 Tail	2141.69	85.67	0.07	0.01	10.44	0.06	0.68	0.04	50.1	8.6	82.0	14.3	9.5	12.4
Ro 4 Tail	2046.04	81.84	0.07	0.01	10.36	0.06	0.62	0.04	46.9	7.3	77.7	12.7	8.2	11.0

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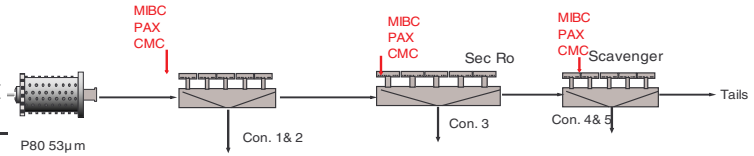
38µm with 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (38 µm @ 56% of CMC)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Objectives		
Target Grind	P80 **38** microns	100% CMC
MDL #s		
Charge Mass	2500	grams
No. of Charges	1	charges



xstrata process support

	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2482.34	99.29	0.13	0.07	12.64	0.39	6.44	0.39	111.6	106.2	103.6	124.3	105.2	104.9
Call Factor			111.62	106.19	103.59	124.34	105.19	104.86						
Rougher Con 1	122.99	4.95	0.93	1.15	14.30	5.48	99.50	6.29	35.71	76.86	5.65	70.94	77.12	80.89
Rougher Con 2	193.74	7.80	0.20	0.11	13.05	0.72	10.45	0.38	12.13	11.53	8.12	14.68	12.76	7.70
Rougher Con 3	189.49	7.63	0.11	0.02	12.60	0.16	1.75	0.09	6.38	2.38	7.66	3.19	2.09	1.78
Rougher Con 4	185.87	7.49	0.10	0.01	12.25	0.09	0.98	0.11	5.62	1.42	7.31	1.76	1.15	2.14
Rougher Tail	1790.25	72.12	0.07	0.01	12.40	0.05	0.61	0.04	40.16	7.82	71.26	9.42	6.88	7.49
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Rougher Con 1	122.99	4.95	0.93	1.15	14.30	5.48	99.50	6.29	35.7	76.9	5.6	70.9	77.1	80.9
Rougher Con 2	316.73	12.76	0.48	0.51	13.54	2.57	45.03	2.67	47.8	88.4	13.8	85.6	89.9	88.6
Rougher Con 3	506.22	20.39	0.34	0.33	13.19	1.67	28.83	1.71	54.2	90.8	21.4	88.8	92.0	90.4
Rougher Con 4	**692.09**	**27.88**	**0.28**	**0.24**	**12.93**	**1.24**	**21.35**	**1.28**	**59.8**	**92.2**	**28.7**	**90.6**	**93.1**	**92.5**
Ro 1 Tail	2359.35	94.37	0.09	0.02	12.55	0.12	1.55	0.08	64.3	23.1	94.4	29.1	22.9	19.1
Ro 2 Tail	2165.61	86.62	0.08	0.01	12.49	0.06	0.75	0.05	52.2	11.6	86.2	14.4	10.1	11.4
Ro 3 Tail	1976.12	79.04	0.07	0.01	12.47	0.05	0.65	0.05	45.8	9.2	78.6	11.2	8.0	9.6
Ro 4 Tail	1790.25	71.61	0.07	0.01	12.49	0.05	0.61	0.04	40.2	7.8	71.3	9.4	6.9	7.5

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Staged Grinding (75/53µm) with 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Objectives		
Target Grind	P80 **53** microns	**Staged Grinding 75/53µm**
MDL #s		56% CMC
Charge Mass	2500 grams	
No. of Charges	1 charges	

	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2498.45	99.94	0.12	0.07	10.37	0.33	5.75	0.24	104.8	94.6	85.0	105.1	93.9	64.7
Call Factor			104.82	94.62	85.02	105.05	93.93	64.72						
Rougher Con 1	117.03	4.68	0.96	1.12	15.55	4.67	93.00	3.39	36.87	79.25	7.03	67.21	75.83	66.35
Rougher Con 2	133.81	5.36	0.15	0.07	14.00	0.50	6.66	0.29	6.49	5.58	7.23	8.23	6.21	6.49
Rougher Con 3	152.94	6.12	0.11	0.06	12.70	0.43	5.76	0.35	5.31	5.55	7.50	8.09	6.14	8.95
Rougher Con 4	103.27	4.13	0.09	0.02	12.65	0.14	1.59	0.09	3.08	1.19	5.04	1.78	1.14	1.55
Rougher Tail	1991.40	79.71	0.07	0.01	9.52	0.06	0.77	0.05	48.26	8.43	73.20	14.69	10.68	16.65

Cumulative	Cum Weight		Cumulative Grade						Cum. Recovery					
Product	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Rougher Con 1	117.03	4.68	0.96	1.12	15.55	4.67	93.00	3.39	36.9	79.3	7.0	67.2	75.8	66.4
Rougher Con 2	250.84	10.04	0.53	0.56	14.72	2.45	46.94	1.74	43.4	84.8	14.3	75.4	82.0	72.8
Rougher Con 3	403.78	16.16	0.37	0.37	13.96	1.68	31.34	1.21	48.7	90.4	21.8	83.5	88.2	81.8
Rougher Con 4	**507.05**	**20.29**	**0.31**	**0.30**	**13.69**	**1.37**	**25.28**	**0.98**	**51.7**	**91.6**	**26.8**	**85.3**	**89.3**	**83.3**
Ro 1 Tail	2381.42	95.26	0.08	0.01	10.12	0.11	1.46	0.08	63.1	20.7	93.0	32.8	24.2	33.6
Ro 2 Tail	2247.61	89.90	0.08	0.01	9.89	0.09	1.15	0.07	56.6	15.2	85.7	24.6	18.0	27.2
Ro 3 Tail	2094.67	83.79	0.07	0.01	9.68	0.06	0.81	0.05	51.3	9.6	78.2	16.5	11.8	18.2
Ro 4 Tail	1991.40	79.66	0.07	0.01	9.53	0.06	0.77	0.05	48.3	8.4	73.2	14.7	10.7	16.7

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Staged Grinding (75/38µm) with 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Objectives		
Target Grind	P80 **38** microns	**Staged Grinding 75/38µm**
MDL #s		56% CMC
Charge Mass	2500	grams
No. of Charges	1	charges

	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2380.47	95.22	0.12	0.07	11.54	0.34	5.93	0.28	106.0	94.0	94.6	110.5	96.8	75.6
Call Factor			105.96	94.04	94.55	110.47	96.85	75.56						
Rougher Con 1	111.37	4.68	0.92	1.07	15.40	4.84	92.50	3.96	36.45	79.50	6.56	69.44	76.68	69.59
Rougher Con 2	142.97	6.01	0.14	0.06	13.85	0.38	5.37	0.25	7.09	5.65	7.57	7.00	5.71	5.64
Rougher Con 3	173.21	7.28	0.11	0.06	12.15	0.45	6.43	0.43	6.74	7.43	8.05	10.04	8.29	11.75
Rougher Con 4	145.29	6.10	0.09	0.01	11.90	0.10	1.15	0.07	4.56	1.36	6.61	1.87	1.24	1.60
Rougher Tail	1807.63	75.94	0.07	0.01	10.30	0.05	0.60	0.04	45.16	6.06	71.21	11.64	8.07	11.41

Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	111.37	4.68	0.92	1.07	15.40	4.84	92.50	3.96	36.4	79.5	6.6	69.4	76.7	69.6
Rougher Con 2	254.34	10.68	0.48	0.50	14.53	2.33	43.52	1.87	43.5	85.1	14.1	76.4	82.4	75.2
Rougher Con 3	427.55	17.96	0.33	0.32	13.57	1.57	28.50	1.29	50.3	92.6	22.2	86.5	90.7	87.0
Rougher Con 4	**572.84**	**24.06**	**0.27**	**0.24**	**13.14**	**1.20**	**21.56**	**0.98**	**54.8**	**93.9**	**28.8**	**88.4**	**91.9**	**88.6**
Ro 1 Tail	2269.10	90.76	0.08	0.01	11.31	0.11	1.45	0.09	63.6	20.5	93.4	30.6	23.3	30.4
Ro 2 Tail	2126.13	85.05	0.08	0.01	11.09	0.09	1.17	0.08	56.5	14.9	85.9	23.6	17.6	24.8
Ro 3 Tail	1952.92	78.12	0.07	0.01	10.94	0.06	0.67	0.04	49.7	7.4	77.8	13.5	9.3	13.0
Ro 4 Tail	1807.63	72.31	0.07	0.01	10.82	0.05	0.63	0.04	45.2	6.1	71.2	11.6	8.1	11.4

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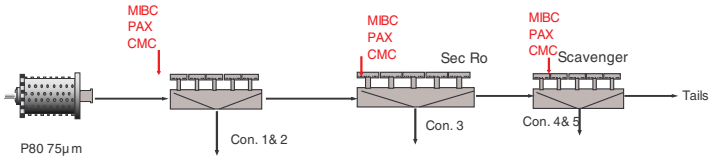
Appendix I- Offset Rougher-Scavenger Mass Balances

Baseline 75µm and 100% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet- Baseline			
Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork		
Project #	5009817		
Feed Sample	Offset Zone Composite		
Objectives			
Target Grind	P80 75 microns	100% CMC	
MDL #s			
Charge Mass	2500	grams	
No. of Charges	1	charges	



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2502.23	100.09	0.15	0.10	8.43	0.39	5.59	0.38	108.3	95.9	72.7	110.2	101.3	132.5
Call Factor			108.28	95.86	72.70	110.23	101.29	132.48						
Rougher Con 1	93.59	3.74	1.34	2.05	15.75	7.23	99.70	7.47	32.91	79.92	6.98	70.03	66.63	72.66
Rougher Con 2	73.17	2.92	0.88	0.26	13.75	1.17	32.10	0.97	16.92	7.92	4.76	8.86	16.77	7.38
Rougher Con 3	68.22	2.73	0.23	0.09	12.85	0.45	6.84	0.34	4.10	2.42	4.15	3.18	3.33	2.41
Rougher Con 4	72.95	2.92	0.14	0.05	12.20	0.27	3.20	0.21	2.73	1.52	4.21	2.04	1.67	1.59
Rougher Tail	2194.30	87.69	0.08	0.01	7.69	0.07	0.74	0.07	43.35	8.23	79.89	15.90	11.60	15.96
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	93.59	3.74	1.34	2.05	15.75	7.23	99.70	7.47	32.9	79.9	7.0	70.0	66.6	72.7
Rougher Con 2	166.76	6.66	1.13	1.26	14.87	4.57	70.04	4.62	49.8	87.8	11.7	78.9	83.4	80.0
Rougher Con 3	234.98	9.39	0.87	0.92	14.29	3.37	51.69	3.38	53.9	90.3	15.9	82.1	86.7	82.4
Rougher Con 4	**307.93**	**12.31**	**0.70**	**0.72**	**13.79**	**2.64**	**40.20**	**2.63**	**56.7**	**91.8**	**20.1**	**84.1**	**88.4**	**84.0**
Ro 1 Tail	2408.64	96.35	0.11	0.02	8.15	0.12	1.94	0.11	67.1	20.1	93.0	30.0	33.4	27.3
Ro 2 Tail	2335.47	93.42	0.08	0.01	7.97	0.09	0.99	0.08	50.2	12.2	88.3	21.1	16.6	20.0
Ro 3 Tail	2267.25	90.69	0.08	0.01	7.83	0.08	0.82	0.07	46.1	9.7	84.1	17.9	13.3	17.6
Ro 4 Tail	2194.30	87.77	0.07	0.01	7.68	0.07	0.74	0.07	43.3	8.2	79.9	15.9	11.6	16.0

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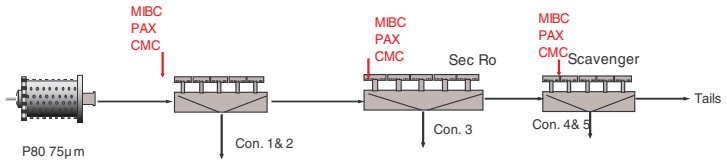
75µm and 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (@ 56% of CMC)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Offset Zone Composite	
Objectives		
Target Grind	P80 75 microns	56% CMC
MDL #s		
Charge Mass	2500	grams
No. of Charges	1	charges



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2501.14	100.05	0.16	0.10	9.87	0.37	5.19	0.35	114.8	95.3	85.1	105.8	94.0	120.0
Call Factor			114.76	95.34	85.10	105.78	93.98	120.04						
Rougher Con 1	129.90	5.19	1.14	1.49	14.85	5.05	70.30	4.93	36.67	81.13	7.81	70.81	70.35	73.52
Rougher Con 2	151.48	6.06	0.40	0.15	13.40	0.68	13.70	0.48	15.15	9.33	8.22	11.12	15.99	8.35
Rougher Con 3	88.50	3.54	0.15	0.06	12.65	0.28	3.29	0.21	3.24	2.04	4.53	2.67	2.24	2.13
Rougher Con 4	82.04	3.28	0.13	0.04	12.65	0.24	2.58	0.20	2.63	1.48	4.20	2.13	1.63	1.88
Rougher Tail	2049.22	81.93	0.08	0.01	9.07	0.06	0.62	0.06	42.31	6.01	75.24	13.27	9.79	14.12
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Rougher Con 1	129.90	5.19	1.14	1.49	14.85	5.05	70.30	4.93	36.7	81.1	7.8	70.8	70.4	73.5
Rougher Con 2	281.38	11.25	0.74	0.77	14.07	2.70	39.83	2.53	51.8	90.5	16.0	81.9	86.3	81.9
Rougher Con 3	369.88	14.79	0.60	0.60	13.73	2.12	31.09	1.98	55.1	92.5	20.6	84.6	88.6	84.0
Rougher Con 4	**451.92**	**18.07**	**0.51**	**0.50**	**13.53**	**1.78**	**25.91**	**1.66**	**57.7**	**94.0**	**24.8**	**86.7**	**90.2**	**85.9**
Ro 1 Tail	2371.24	94.85	0.11	0.02	9.60	0.11	1.62	0.10	63.3	18.9	92.2	29.2	29.6	26.5
Ro 2 Tail	2219.76	88.79	0.09	0.01	9.34	0.08	0.80	0.07	48.2	9.5	84.0	18.1	13.7	18.1
Ro 3 Tail	2131.26	85.25	0.08	0.01	9.20	0.07	0.70	0.07	44.9	7.5	79.4	15.4	11.4	16.0
Ro 4 Tail	2049.22	81.97	0.08	0.01	9.07	0.06	0.62	0.06	42.3	6.0	75.2	13.3	9.8	14.1

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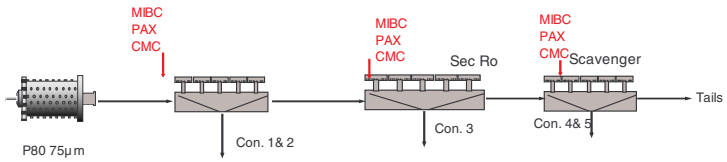
75µm with Aero 3477 and 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Offset Zone Composite	
Objectives		
Target Grind	P80 75 microns	56% CMC
MDL #s		Aero 3477
Charge Mass	2500	grams
No. of Charges	1	charges



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2488.35	99.53	0.15	0.10	8.52	0.38	4.63	0.47	109.0	96.3	73.5	107.3	83.9	160.4
Call Factor			109.05	96.31	73.46	107.30	83.86	160.41						
Rougher Con 1	72.61	2.92	1.93	2.66	16.15	8.96	100.00	12.60	36.97	80.97	5.56	69.95	63.33	79.41
Rougher Con 2	52.90	2.13	0.87	0.30	14.60	1.24	31.80	0.82	12.13	6.70	3.66	7.05	14.67	3.76
Rougher Con 3	50.37	2.02	0.27	0.12	13.40	0.59	8.71	0.34	3.53	2.43	3.20	3.20	3.83	1.49
Rougher Con 4	93.12	3.74	0.16	0.06	12.85	0.31	3.78	0.23	3.94	2.46	5.67	3.10	3.07	1.86
Rougher Tail	2219.35	89.19	0.07	0.01	7.79	0.07	0.78	0.07	43.43	7.44	81.92	16.70	15.10	13.48
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	72.61	2.92	1.93	2.66	16.15	8.96	100.00	12.60	37.0	81.0	5.6	69.9	63.3	79.4
Rougher Con 2	125.51	5.04	1.48	1.67	15.50	5.71	71.26	7.63	49.1	87.7	9.2	77.0	78.0	83.2
Rougher Con 3	175.88	7.07	1.13	1.22	14.90	4.24	53.34	5.55	52.6	90.1	12.4	80.2	81.8	84.7
Rougher Con 4	**269.00**	**10.81**	**0.80**	**0.82**	**14.19**	**2.88**	**36.19**	**3.71**	**56.6**	**92.6**	**18.1**	**83.3**	**84.9**	**86.5**
Ro 1 Tail	2415.74	96.63	0.10	0.02	8.29	0.12	1.75	0.10	63.0	19.0	94.4	30.1	36.7	20.6
Ro 2 Tail	2362.84	94.51	0.08	0.01	8.15	0.09	1.07	0.08	50.9	12.3	90.8	23.0	22.0	16.8
Ro 3 Tail	2312.47	92.50	0.08	0.01	8.03	0.08	0.91	0.08	47.4	9.9	87.6	19.8	18.2	15.3
Ro 4 Tail	2219.35	88.77	0.07	0.01	7.83	0.07	0.78	0.07	43.4	7.4	81.9	16.7	15.1	13.5

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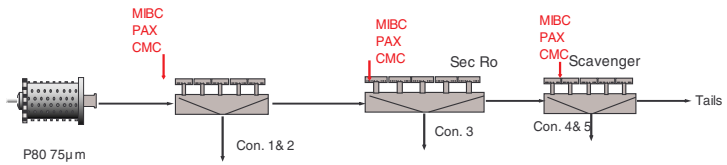
53µm with Aero 3477 and 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (Aero 3477)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Offset Zone Composite	
Objectives		
Target Grind	P80 53 microns	56% CMC
MDL #s		Aero 3477
Charge Mass	2500	grams
No. of Charges	1	charges



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2506.53	100.26	0.15	0.10	9.50	0.42	5.56	0.38	109.9	96.5	81.9	120.1	100.8	132.2
Call Factor			109.89	96.48	81.88	120.10	100.78	132.16						
Rougher Con 1	107.69	4.30	1.39	1.90	15.65	7.22	95.30	6.98	38.58	84.39	7.06	73.60	73.41	78.04
Rougher Con 2	80.62	3.22	0.59	0.15	14.00	0.82	19.65	0.45	12.34	4.95	4.73	6.26	11.33	3.77
Rougher Con 3	58.13	2.32	0.18	0.06	12.80	0.38	4.56	0.22	2.68	1.51	3.12	2.09	1.90	1.33
Rougher Con 4	70.56	2.82	0.12	0.04	12.60	0.22	2.28	0.13	2.23	1.02	3.72	1.47	1.15	0.95
Rougher Tail	2189.53	87.35	0.08	0.01	8.87	0.08	0.78	0.07	44.17	8.13	81.37	16.58	12.22	15.91
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	107.69	4.30	1.39	1.90	15.65	7.22	95.30	6.98	38.6	84.4	7.1	73.6	73.4	78.0
Rougher Con 2	188.31	7.51	1.05	1.15	14.94	4.48	62.91	4.18	50.9	89.3	11.8	79.9	84.7	81.8
Rougher Con 3	246.44	9.83	0.84	0.89	14.44	3.51	49.15	3.25	53.6	90.9	14.9	81.9	86.6	83.1
Rougher Con 4	**317.00**	**12.65**	**0.68**	**0.70**	**14.03**	**2.78**	**38.72**	**2.55**	**55.8**	**91.9**	**18.6**	**83.4**	**87.8**	**84.1**
Ro 1 Tail	2398.84	95.95	0.10	0.02	9.22	0.12	1.55	0.09	61.4	15.6	92.9	26.4	26.6	22.0
Ro 2 Tail	2318.22	92.73	0.08	0.01	9.06	0.09	0.92	0.08	49.1	10.7	88.2	20.1	15.3	18.2
Ro 3 Tail	2260.09	90.40	0.08	0.01	8.96	0.08	0.82	0.07	46.4	9.1	85.1	18.1	13.4	16.9
Ro 4 Tail	2189.53	87.58	0.08	0.01	8.85	0.08	0.78	0.07	44.2	8.1	81.4	16.6	12.2	15.9

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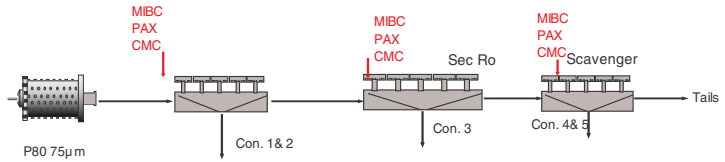
38µm with 56% of CMC Dosage

LDI Flotation Mass Balance Spreadsheet (38 µm @ 56% of CMC)

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Offset Zone Composite	
Objectives		
Target Grind	P80 38 microns	56% CMC
MDL #s		
Charge Mass	2500	grams
No. of Charges	1	charges



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2506.98	100.28	0.16	0.10	10.00	0.32	6.29	0.33	114.5	95.7	86.2	92.6	113.9	112.8
Call Factor			114.47	95.75	86.22	92.63	113.90	112.77						
Rougher Con 1	145.99	5.82	0.835	1.36	13.95	3.87	78.33	4.23	30.26	82.49	8.10	69.32	72.35	75.11
Rougher Con 2	239.39	9.55	0.398	0.112	12.45	0.57	12.8	0.37	23.65	11.14	11.85	16.74	19.39	10.77
Rougher Con 3	152.04	6.06	0.125	0.026	12.2	0.17	2.15	0.09	4.72	1.64	7.38	3.17	2.07	1.66
Rougher Con 4	128.52	5.13	0.108	0.017	11.95	0.11	1.18	0.08	3.45	0.91	6.11	1.73	0.96	1.25
Rougher Tail	1841.04	73.44	0.08	0.01	9.09	0.04	0.45	0.05	37.93	3.82	66.56	9.04	5.24	11.20
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
Rougher Con 1	145.99	5.82	0.84	1.36	13.95	3.87	78.33	4.23	30.3	82.5	8.1	69.3	72.3	75.1
Rougher Con 2	385.38	15.37	0.56	0.58	13.02	1.82	37.62	1.83	53.9	93.6	20.0	86.1	91.7	85.9
Rougher Con 3	537.42	21.44	0.44	0.43	12.79	1.35	27.59	1.34	58.6	95.3	27.3	89.2	93.8	87.6
Rougher Con 4	**665.94**	**26.56**	**0.38**	**0.35**	**12.63**	**1.11**	**22.49**	**1.10**	**62.1**	**96.2**	**33.4**	**91.0**	**94.8**	**88.8**
Ro 1 Tail	2360.99	94.44	0.12	0.02	9.76	0.11	1.85	0.09	69.7	17.5	91.9	30.7	27.7	24.9
Ro 2 Tail	2121.60	84.86	0.09	0.01	9.46	0.05	0.61	0.05	46.1	6.4	80.0	13.9	8.3	14.1
Ro 3 Tail	1969.56	78.78	0.08	0.01	9.25	0.04	0.50	0.05	41.4	4.7	72.7	10.8	6.2	12.4
Ro 4 Tail	1841.04	73.64	0.08	0.00	9.06	0.04	0.45	0.05	37.9	3.8	66.6	9.0	5.2	11.2

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Appendix J- Open Circuit Cleaner Flotation Schedules

ROUGHER/SBAVENGER FLOTATION TEST REPORT **Baseline**

TEST OBJEBTIVES: **Baseline @ p80 38 microns** DATE: TEST FEED:

GRINDING BONDITIONS		**AERATION:**	
(AFTER 10MIN. BLEANING WITH SIL.SAND)		**ROD TYPE:**	**70% MILD STEEL**
MILL TYPE:	**STAINLESS**		**30% STAINLESS**
CHARGE:	**2500 g**	**FEED SIZE:**	**-10 M**
WATER:	**1466 ml**	**WATER pH:**	
GRIND:	**P80 = 38 micron**	**GRIND TIME:**	

FLOTATION BONDITIONS			FLOATED BY: LJB
BELL TYPE:	Denver	BELL VOLUME:	4 liter
IMPELLER rpm:	1600	REPULP WATER:	DOMESTIB
RO GAS RATE:	3 l/min (23)	SBAVENGER GAS RATE:	9 l/min (53)
STROKE RATE : rougher 1/10 seB		SBAVENGER	
	:seB and sBav 1/5 seB		

REAGENT	QUANTITY ml, g	DOSE g/t	STAGE or OPERATION	COND TIME	FLOAT TIME	Stroke Rate		pH / REDOX as is: BOMMENTS
								INITIAL STIR TIME: 1 min.
Dextrin(1%)	5	20		2				
PAX (1%)	1.44	5.76		2				3L/min
MIBC	4 drops (28)	28	**Conc 1**	0	3	1/10		PRI.Ro. CONB.
PAX (1%)	1.26	5.04		0.5	3	1/5		**Change Stroke to 1/5**
PAX (1%)	0.96	4	**Conc 2**	1	3			
PAX (1%)	0.96	4		1	3			
Dextrin (1%)	9	36		2				3L./min
PAX (1%)	0.96	4	**Conc 3**					
MIBC	2 drops (14)	14		0.5	3	1/5		
PAX (1%)	0.96	4		1	3			
Dextrin (1%)	9	36		2				**InBrease Air to 9 l/min**
PAX (1%)	0.96	4		1				
MIBC	2 drops (14)	14	**Conc 4**	0.5	4	1/5		
PAX (1%)	0.96	4		1				
					4	1/5		

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SCAVENGER CONCENTRATE CLEANER FLOTATION TEST REPORT

TEST#	PROJECT #	DATE:						TEST FEED:	Ro-Scav Conc 3-4				
TEST OBJECTIVES:	To reduce the gangue content of the secondary rougher concentrate							TRANSFER TIME:	Test order:				
GRINDING CONDITIONS		**AERATION:**	**N/A**					FLOTATION CONDITIONS		FLOATED BY:			
		ROD TYPE:	**N/A**										
MILL TYPE:	**N/A**							CELL TYPE:	Denver	CELL VOLUME:	**1.5** L		
CHARGE:	**N/A**	**FEED SIZE:**	**N/A**					**IMPELLER rpm:**	**1200**	REPULP WATER:	PROCESS WATER		
WATER:	**N/A**	**WATER pH:**						RO GAS RATE:	2 l/min	SCAVENGER GAS RATE:	L/MIN		
GRIND:		**GRIND TIME:**						STROKE RATE *): rougher	1/10 sec	SCAVENGER			

REAGENT	**QUANTITY ml, g**	**DOSE g/ton**	**STAGE or OPERATION**	**COND TIME**	**FLOAT TIME**			pH as is: / INITIAL STIR TIME: 1 MIN			
								COMMENTS			
Dextrin (1%)	9	36		2							
MIBC	1 drop	7									
PAX (1%)	1.00	4									
			CONC 1		2.0			Assess need for reagent (*xanthate after conc 1)			
			CONC 2		2.0						
			CONC 3		2.0						
CONC 1, 6 STROKES/MIN											

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SECONDARY ROUGHER CLEANER FLOTATION TEST REPORT

TEST#	PROJECT #	DATE:		TEST FEED:	Ro Conc 1-2 plus 1st Clnr Conc
TEST OBJECTIVES:	To reduce the gangue content of the secondary rougher concentrate			TRANSFER TIME:	Test order:

GRINDING CONDITIONS		**AERATION:**	**N/A**
		ROD TYPE:	**N/A**
MILL TYPE:	**N/A**		
CHARGE:	**N/A**	**FEED SIZE:**	**N/A**
WATER:	**N/A**	**WATER pH:**	
GRIND:		**GRIND TIME:**	

FLOTATION CONDITIONS			FLOATED BY:
CELL TYPE:	Denver		CELL VOLUME: **1.5** L
IMPELLER rpm:	**1000**		REPULP WATER: PROCESS WATER
RO GAS RATE:	2 l/min		SCAVENGER GAS RATE: L/MIN
STROKE RATE *): rougher	1/10 sec		SCAVENGER

REAGENT	QUANTITY ml, g	DOSE g/ton	STAGE or OPERATION	COND TIME	FLOAT TIME			pH as is: / INITIAL STIR TIME: 1 MIN			
Dextrin (1%)	9	36		2							
MIBC	1 drop	7									
PAX (1%)	1.00	4									
			CONC 1		2.0			Assess need for reagent (*xanthate after conc 1)			
			CONC 2		2.0						
			CONC 3		2.0						
CONC 1, 6 STROKES/MIN											

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THIRD CLEANER FLOTATION TEST REPORT

TEST#	PROJECT #	DATE:						TEST FEED:	2nd Clnr Conc				
TEST OBJECTIVES:	To reduce the gangue content of the secondary rougher concentrate							TRANSFER TIME:	Test order:				
GRINDING CONDITIONS		**AERATION:**	**N/A**					FLOTATION CONDITIONS		FLOATED BY:			
		ROD TYPE:	**N/A**										
MILL TYPE:	**N/A**							CELL TYPE:	Denver	CELL VOLUME:	**1** L		
CHARGE:	**N/A**	**FEED SIZE:**	**N/A**					**IMPELLER rpm:**	**850**	REPULP WATER:	PROCESS WATER		
WATER:	**N/A**	**WATER pH:**						RO GAS RATE:	2 l/min	SCAVENGER GAS RATE:	L/MIN		
GRIND:		**GRIND TIME:**						STROKE RATE *): rougher	1/10 sec	SCAVENGER			

REAGENT	**QUANTITY ml, g**	**DOSE g/ton**	**STAGE or OPERATION**	**COND TIME**	**FLOAT TIME**			pH as is: / INITIAL STIR TIME: 1 MIN	COMMENTS			
Dextrin (1%)	4.5	18		2								
MIBC	1 drop	7										
PAX (1%)	1.00	4										
			CONC 1		2.0			Assess need for reagent (*xanthate after conc 1)				
			CONC 2		2.0							
			CONC 3		2.0							
CONC 1, 6 STROKES/MIN												

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FOURTH CLEANER FLOTATION TEST REPORT

TEST#	PROJECT #	DATE:				TEST FEED:	2nd Clnr Conc						
TEST OBJECTIVES:	To reduce the gangue content of the secondary rougher concentrate					TRANSFER TIME:		Test order:					
GRINDING CONDITIONS		**AERATION:**	**N/A**			FLOTATION CONDITIONS		FLOATED BY:					
		ROD TYPE:	**N/A**										
MILL TYPE:	**N/A**					CELL TYPE:	Denver	CELL VOLUME:	**0.75** L				
CHARGE:	**N/A**	**FEED SIZE:**	**N/A**			**IMPELLER rpm:**	**850**	REPULP WATER:	PROCESS WATER				
WATER:	**N/A**	**WATER pH:**				RO GAS RATE:	2 l/min	SCAVENGER GAS RATE:	L/MIN				
GRIND:		**GRIND TIME:**				STROKE RATE *): rougher	1/10 sec	SCAVENGER					
REAGENT	**QUANTITY ml, g**	**DOSE g/ton**	**STAGE or OPERATION**	**COND TIME**	**FLOAT TIME**			pH as is: INITIAL STIR TIME:	1 MIN	COMMENTS			
				2									
MIBC	1 drop	7	?										
			CONC 1		2.0			Assess need for reagent (*xanthate after conc 1)					
			CONC 2		2.0								
			CONC 3		2.0								
CONC 1, 6 STROKES/MIN													

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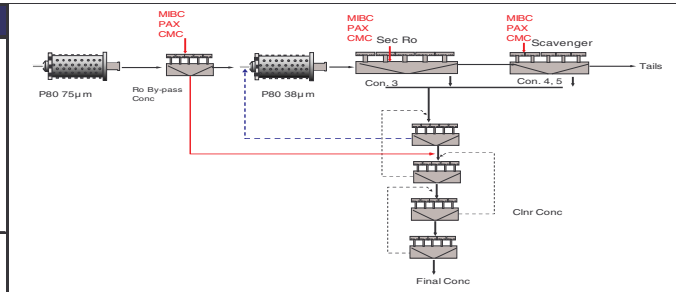
Appendix K- Roby Open Circuit Cleaner Mass Balances

LDI Flotation Mass Balance Spreadsheet		
Project Name	LDI Roby and Offset Zone Ore Characterisation	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Note	Ro 1 and Ro 2 combined and report to 2nd Cleaning	
Target Grind	SG P80 75/38 microns 3 stage-Cleaning	
MDL #s		
Charge Mass	2500	grams
No. of Charges	1	charges



	Weight		Assays							Distribution					
	(g)	(%)	Ni	Cu	Ni+Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	0.19	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2466.96	100.00	0.12	0.07	0.19	12.50	0.33	6.16	0.30	103.7	96.3	102.4	106.7	100.6	80.8
Call Factor			103.70	96.29	100.92	102.43	106.70	100.60	80.84						
3rd Clnr Conc	20.63	0.84	3.88	5.20	9.08	10.70	21.40	474.00	18.45	26.82	64.52	0.72	54.10	64.38	51.58
3rd Clnr Tails	13.18	0.53	1.15	1.15	2.30	13.40	6.05	113.50	17.10	5.08	9.08	0.57	9.77	9.85	30.54
2nd Clnr Tails	250.39	10.15	0.17	0.08	0.25	15.15	0.66	9.52	0.05	14.18	11.75	12.31	20.25	15.69	1.70
1st Clnr Tails	510.26	20.68	0.09	0.02	0.10	12.05	0.09	1.13	0.07	14.70	4.60	19.95	5.63	3.80	4.84
Rougher Tail	1672.50	67.80	0.07	0.01	0.08	12.25	0.05	0.57	0.05	39.22	10.06	66.46	10.25	6.28	11.33
Cumulative	**Cum Weight**		**Cumulative Grade**							**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**Ni+Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
3rd Clnr Conc	20.63	0.84	3.88	5.20	9.08	10.70	21.40	474.00	18.45	26.8	64.5	0.7	54.1	64.4	51.6
2nd Clnr Conc	33.81	1.37	2.82	3.62	6.44	11.75	15.42	333.47	17.92	31.9	73.6	1.3	63.9	74.2	82.1
1st Clnr Conc	284.20	11.52	0.48	0.50	0.98	14.75	2.42	48.06	2.18	46.1	85.3	13.6	84.1	89.9	83.8
1st Clnr Tails	**794.46**	**32.20**	**0.23**	**0.19**	**0.42**	**13.01**	**0.92**	**17.92**	**0.82**	**60.8**	**89.9**	**33.5**	**89.8**	**93.7**	**88.7**
Ro 1 Tail	2446.33	97.85	0.09	0.02	0.11	12.68	0.16	2.24	0.15	73.2	35.5	99.3	45.9	35.6	48.4
Ro 2 Tail	2433.15	97.33	0.08	0.02	0.10	12.67	0.12	1.63	0.05	68.1	26.4	98.7	36.1	25.8	17.9
Ro 3 Tail	2182.76	87.31	0.07	0.01	0.09	12.37	0.06	0.71	0.06	53.9	14.7	86.4	15.9	10.1	16.2
Ro 4 Tail	1672.50	66.90	0.07	0.01	0.08	12.41	0.05	0.58	0.05	39.2	10.1	66.5	10.2	6.3	11.3

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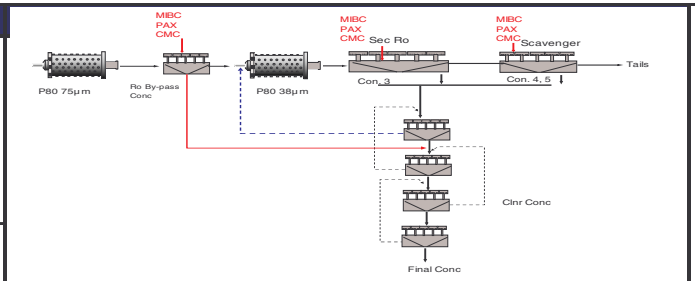
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation
Project #	5009817
Feed Sample	Roby Zone Composite
Note	Ro 1 and Ro 2 combined and report to 2nd Cleaning
Target Grind	Final Grind P80 38 microns 3 stage-Cleaning
MDL #s	Dextrin added in the 1st and 2nd cleaning
Charge Mass	2500 grams
No. of Charges	1 charges



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	Weight		Assays							Distribution					
	(g)	(%)	Ni	Cu	Ni+Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	0.19	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2474.96	100.00	0.13	0.07	0.20	12.38	0.37	6.35	0.33	112.1	101.4	101.4	119.6	103.7	90.5
Call Factor			112.08	101.39	108.07	101.44	119.64	103.71	90.46						
3rd Clnr Conc	10.69	0.43	4.44	10.48	14.92	8.32	42.60	846.00	38.40	14.67	63.78	0.29	49.61	57.57	49.55
3rd Clnr Tails	14.28	0.58	3.34	1.82	5.16	15.30	12.70	244.00	14.30	14.74	14.80	0.71	19.76	22.18	24.65
2nd Clnr Tails	146.81	5.93	0.21	0.07	0.28	13.85	0.69	8.55	0.59	9.48	5.60	6.64	11.04	7.99	10.46
1st Clnr Tails	595.58	24.06	0.10	0.02	0.12	12.95	0.13	1.34	0.07	17.85	6.10	25.18	8.43	5.08	5.03
Rougher Tail	1707.60	69.00	0.08	0.01	0.09	12.05	0.06	0.66	0.05	43.27	9.72	67.18	11.16	7.17	10.31
Cumulative Product	**Cum Weight**		**Cumulative Grade**							**Cum. Recovery**					
	(g)	**(%)**	**Ni**	**Cu**	**Ni+Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
3rd Clnr Conc	10.69	0.43	4.44	10.48	14.92	8.32	42.60	846.00	38.40	14.7	63.8	0.3	49.6	57.6	49.6
2nd Clnr Conc	24.97	1.01	3.81	5.53	9.34	12.31	25.50	501.72	24.62	29.4	78.6	1.0	69.4	79.8	74.2
1st Clnr Conc	171.78	6.94	0.73	0.86	1.59	13.63	4.30	80.24	4.08	38.9	84.2	7.6	80.4	87.7	84.7
1st Clnr Tails	**767.36**	**31.00**	**0.24**	**0.21**	**0.45**	**13.10**	**1.06**	**19.00**	**0.97**	**56.7**	**90.3**	**32.8**	**88.8**	**92.8**	**89.7**
Ro 1 Tail	2464.27	98.57	0.11	0.03	0.14	12.52	0.19	2.73	0.17	85.3	36.2	99.7	50.4	42.4	50.4
Ro 2 Tail	2449.99	98.00	0.09	0.02	0.11	12.50	0.12	1.31	0.09	70.6	21.4	99.0	30.6	20.2	25.8
Ro 3 Tail	2303.18	92.13	0.09	0.01	0.10	12.41	0.08	0.84	0.06	61.1	15.8	92.4	19.6	12.3	15.3
Ro 4 Tail	1707.60	68.30	0.08	0.01	0.09	12.17	0.06	0.67	0.05	43.3	9.7	67.2	11.2	7.2	10.3

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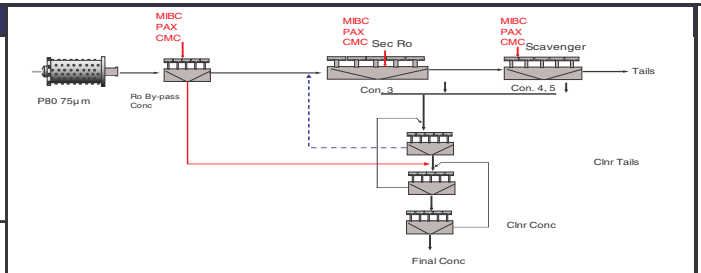
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Note	Ro 1 report to 2nd Cleaning	
Target Grind	Final Grind P80 38 microns 3 stage-Cleaning	
MDL #s	CMC added in the 1st and 2nd cleaning	
Charge Mass	2500	grams
No. of Charges	1	charges



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	Weight		Assays							Distribution					
	(g)	(%)	Ni	Cu	Ni+Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	19.84	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2474.45	100.00	0.12	0.08	0.20	12.27	0.39	6.65	0.30	106.6	111.2	100.6	125.1	108.6	81.6
Call Factor			106.65	111.19	1.02	100.56	125.06	108.63	81.58						
3rd Clnr Conc	7.30	0.30	4.48	15.46	19.94	0.84	42.80	948.00	36.90	10.62	58.60	0.02	32.57	42.07	36.07
3rd Clnr Tails	9.70	0.39	5.53	3.39	8.92	8.29	34.40	531.00	24.40	17.42	17.07	0.26	34.78	31.31	31.69
2nd Clnr Tails	167.32	6.76	0.26	0.11	0.37	12.00	0.85	13.30	0.68	14.18	9.30	6.61	14.83	13.53	15.23
1st Clnr Tails	626.97	25.34	0.10	0.02	0.12	12.45	0.14	1.50	0.07	19.96	5.53	25.71	9.15	5.72	5.88
Rougher Tail	1663.16	67.21	0.07	0.01	0.08	12.30	0.05	0.73	0.05	37.81	9.50	67.39	8.67	7.38	11.13

Cumulative	**Cum Weight**		**Cumulative Grade**							**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**Ni+Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
3rd Clnr Conc	7.30	0.30	4.48	15.46	19.94	0.84	42.80	948.00	36.90	10.6	58.6	0.0	32.6	42.1	36.1
2nd Clnr Conc	17.00	0.69	5.08	8.57	13.65	5.09	38.01	710.06	29.77	28.0	75.7	0.3	67.4	73.4	67.8
1st Clnr Conc	184.32	7.45	0.71	0.89	1.59	11.36	4.28	77.56	3.36	42.2	85.0	6.9	82.2	86.9	83.0
1st Clnr Tails	**811.29**	**32.79**	**0.24**	**0.21**	**0.45**	**12.20**	**1.08**	**18.78**	**0.82**	**62.2**	**90.5**	**32.6**	**91.3**	**92.6**	**88.9**
Ro 1 Tail	2467.15	98.69	0.11	0.03	0.15	12.43	0.26	3.90	0.20	89.4	41.4	100.0	67.4	57.9	63.9
Ro 2 Tail	2457.45	98.30	0.09	0.02	0.11	12.45	0.13	1.80	0.10	72.0	24.3	99.7	32.6	26.6	32.2
Ro 3 Tail	2290.13	91.61	0.08	0.01	0.09	12.47	0.08	0.95	0.06	57.8	15.0	93.1	17.8	13.1	17.0
Ro 4 Tail	1663.16	66.53	0.07	0.01	0.08	12.43	0.05	0.74	0.05	37.8	9.5	67.4	8.7	7.4	11.1

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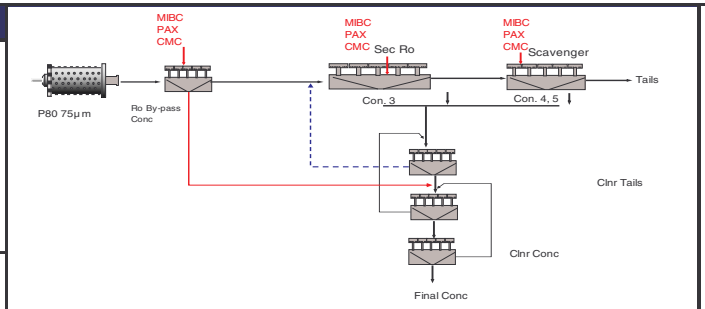
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation
Project #	5009817
Feed Sample	Roby Zone Composite
Note	Ro 1 report to 2nd Cleaning
Target Grind	Final Grind P80 38 micror 3 stage-Cleaning
MDL #s	CMC added in the 1st and 2nd cleaning **(2 charges)**
Charge Mass	2500 grams
No. of Charges	2 charges



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	Weight		Assays							Distribution					
	(g)	(%)	Ni	Cu	Ni+Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	5000.00	100.00	0.12	0.07	19.84	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	5003.94	100.00	0.14	0.09	0.23	11.90	0.41	7.70	0.75	120.7	128.0	97.5	132.3	125.8	203.2
Call Factor			120.66	128.01	1.16	97.55	132.33	125.83	203.16						
3rd Clnr Conc	15.65	0.31	4.16	16.95	21.11	1.67	47.00	1045.00	47.00	9.24	59.16	0.04	35.83	42.44	19.55
3rd Clnr Tails	25.65	0.51	4.19	3.48	7.67	9.38	26.20	457.00	26.20	15.26	19.91	0.40	32.74	30.42	17.87
2nd Clnr Tails	199.85	3.99	0.67	0.17	0.84	11.80	1.53	31.00	0.90	19.01	7.40	3.96	14.90	16.08	4.78
1st Clnr Tails	1012.54	20.23	0.09	0.03	0.13	12.70	0.15	1.97	0.11	13.37	7.68	21.59	7.40	5.18	2.96
Rougher Tail	3750.25	74.95	0.08	0.01	0.09	11.75	0.05	0.61	0.55	43.12	5.85	74.00	9.13	5.89	54.84
Cumulative	**Cum Weight**		**Cumulative Grade**							**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**Ni+Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
3rd Clnr Conc	15.65	0.31	4.16	16.95	21.11	1.67	47.00	1045.00	47.00	9.2	59.2	0.0	35.8	42.4	19.6
2nd Clnr Conc	41.30	0.83	4.18	8.58	12.76	6.46	34.08	679.81	34.08	24.5	79.1	0.4	68.6	72.9	37.4
1st Clnr Conc	241.15	4.82	1.27	1.61	2.88	10.89	7.10	142.12	6.58	43.5	86.5	4.4	83.5	88.9	42.2
1st Clnr Tails	**1253.69**	**25.05**	**0.32**	**0.34**	**0.66**	**12.35**	**1.49**	**28.93**	**1.36**	**56.9**	**94.1**	**26.0**	**90.9**	**94.1**	**45.2**
Ro 1 Tail	4988.29	99.77	0.13	0.04	0.16	11.92	0.26	4.44	0.61	90.8	40.8	100.0	64.2	57.6	80.4
Ro 2 Tail	4962.64	99.25	0.11	0.02	0.13	11.94	0.13	2.11	0.47	75.5	20.9	99.6	31.4	27.1	62.6
Ro 3 Tail	4762.79	95.26	0.08	0.01	0.10	11.94	0.07	0.89	0.46	56.5	13.5	95.6	16.5	11.1	57.8
Ro 4 Tail	3750.25	75.01	0.08	0.01	0.09	11.74	0.05	0.60	0.55	43.1	5.9	74.0	9.1	5.9	54.8

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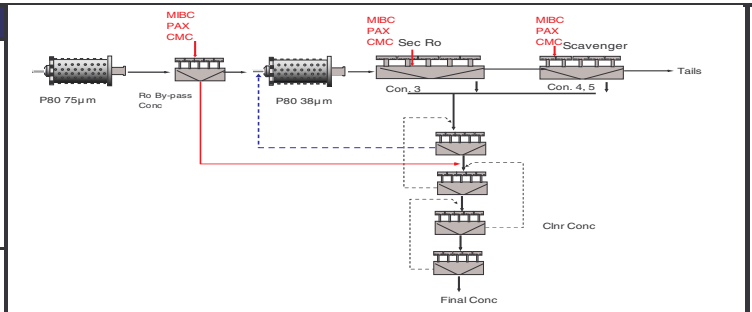
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation	
Project #	5009817	
Feed Sample	Roby Zone Composite	
Note	Ro 1 report to 2nd Cleaning	
Target Grind	Final Grind P80 75 microns	3 stage-Cleaning
MDL #s	CMC added in the 1st and 2nd cleaning	
Charge Mass	2500	grams
No. of Charges	1	charges



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2501.50	100.00	0.13	0.06	12.59	0.37	6.19	0.32	107.4	92.0	103.2	118.0	101.1	86.6
Call Factor			107.35	91.98	103.22	118.03	101.07	86.64						
3rd Clnr Conc	3.83	0.15	2.13	10.92	1.71	29.50	500.00	24.30	2.60	25.97	0.02	12.34	12.38	11.61
3rd Clnr Tails	7.02	0.28	2.81	2.80	10.25	22.60	350.00	30.00	6.30	12.20	0.23	17.33	15.88	26.26
2nd Clnr Tails	178.33	7.13	0.62	0.30	15.55	2.27	42.80	1.53	35.12	33.21	8.80	44.23	49.33	34.02
1st Clnr Tails	336.80	13.46	0.12	0.04	13.80	0.24	3.32	0.20	12.47	8.99	14.75	8.83	7.23	8.40
Rougher Tail	1975.52	78.97	0.07	0.02	12.15	0.08	1.19	0.08	43.51	19.62	76.19	17.27	15.19	19.71
Cumulative	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
Product	**(g)**	**(%)**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**	**Ni**	**Cu**	**MgO**	**Pt**	**Pd**	**Au**
3rd Clnr Conc	3.83	0.15	2.13	10.92	1.71	29.50	500.00	24.30	2.6	26.0	0.0	12.3	12.4	11.6
2nd Clnr Conc	10.85	0.43	2.57	5.67	7.24	25.04	402.95	27.99	8.9	38.2	0.2	29.7	28.3	37.9
1st Clnr Conc	189.18	7.56	0.73	0.61	15.07	3.58	63.46	3.05	44.0	71.4	9.1	73.9	77.6	71.9
1st Clnr Tails	**525.98**	**21.03**	**0.34**	**0.25**	**14.26**	**1.44**	**24.95**	**1.22**	**56.5**	**80.4**	**23.8**	**82.7**	**84.8**	**80.3**
Ro 1 Tail	2497.67	99.91	0.12	0.05	12.60	0.32	5.43	0.28	97.4	74.0	100.0	87.7	87.6	88.4
Ro 2 Tail	2490.65	99.63	0.11	0.04	12.61	0.26	4.45	0.20	91.1	61.8	99.8	70.3	71.7	62.1
Ro 3 Tail	2312.32	92.49	0.08	0.02	12.38	0.10	1.50	0.10	56.0	28.6	90.9	26.1	22.4	28.1
Ro 4 Tail	1975.52	79.02	0.07	0.02	12.14	0.08	1.19	0.08	43.5	19.6	76.2	17.3	15.2	19.7

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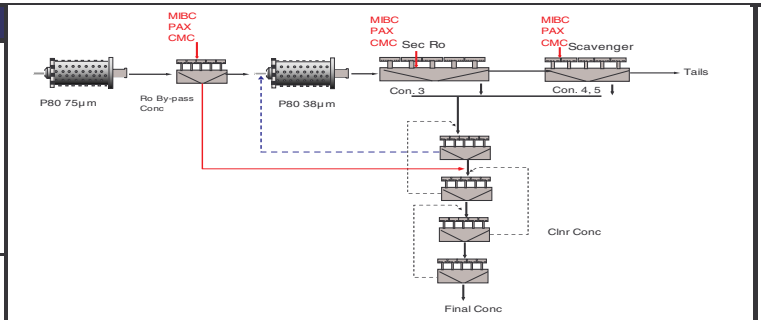
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation
Project #	5009817
Feed Sample	Roby Zone Composite
Note	Ro 1 report to 2nd Cleaning
Target Grind	Final Grind P80 38µm 3 stage-Cleaning
MDL #s	Guar added in the 1st and 2nd cleaning
Charge Mass	2500 grams
No. of Charges	1 charges



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.12	0.07	12.20	0.31	6.12	0.37	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	4963.60	100.00	0.12	0.18	12.72	0.34	5.71	0.36	105.2	255.3	104.2	109.4	93.2	96.6
Call Factor			105.17	255.32	104.23	109.40	93.24	96.58						
3rd Clnr Conc	8.38	0.17	1.99	11.12	3.34	26.40	700.00	36.50	2.74	10.50	0.04	13.14	20.71	17.24
3rd Clnr Tails	42.61	0.86	3.34	2.75	10.85	19.20	335.00	23.60	23.37	13.21	0.73	48.60	50.40	56.69
2nd Clnr Tails	222.01	4.47	0.31	2.75	15.50	1.46	19.35	1.06	11.23	68.82	5.45	19.26	15.17	13.27
1st Clnr Tails	656.06	13.22	0.10	0.03	13.65	0.18	2.05	0.10	10.99	2.46	14.19	7.02	4.75	3.70
Rougher Tail	4034.54	81.28	0.08	0.01	12.45	0.05	0.63	0.04	51.67	5.00	79.58	11.98	8.97	9.10
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
3rd Clnr Conc	8.38	0.17	1.99	11.12	3.34	26.40	700.00	36.50	2.7	10.5	0.0	13.1	20.7	17.2
2nd Clnr Conc	50.99	1.03	3.12	4.13	9.62	20.38	394.99	25.72	26.1	23.7	0.8	61.7	71.1	73.9
1st Clnr Conc	273.00	5.50	0.83	3.01	14.40	4.99	89.51	5.67	37.3	92.5	6.2	81.0	86.3	87.2
1st Clnr Tails	**929.06**	**18.72**	**0.32**	**0.91**	**13.87**	**1.59**	**27.75**	**1.74**	**48.3**	**95.0**	**20.4**	**88.0**	**91.0**	**90.9**
Ro 1 Tail	4955.22	198.21	0.06	0.08	6.41	0.15	2.28	0.15	97.3	89.5	100.0	86.9	79.3	82.8
Ro 2 Tail	4912.61	196.50	0.05	0.07	6.42	0.07	0.84	0.05	73.9	76.3	99.2	38.3	28.9	26.1
Ro 3 Tail	4690.60	187.62	0.04	0.01	6.36	0.03	0.42	0.02	62.7	7.5	93.8	19.0	13.7	12.8
Ro 4 Tail	4034.54	161.38	0.04	0.01	6.27	0.03	0.32	0.02	51.7	5.0	79.6	12.0	9.0	9.1

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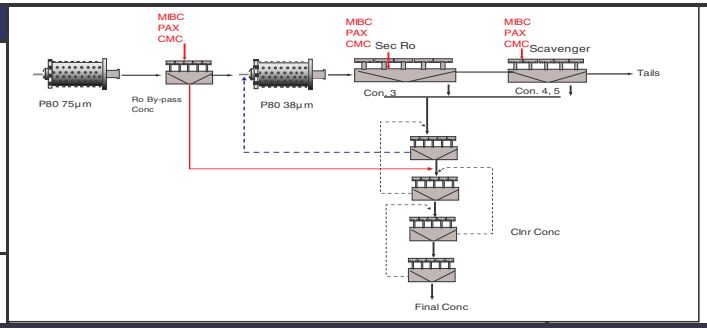
Appendix L- Offset Open Circuit Cleaner Mass Balances

LDI Flotation Mass Balance Spreadsheet		
Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork	
Project #	5009817	
Feed Sample	Offset Zone Composite	
Note	Ro 1 report to 2nd Cleaning	
Target Grind	Final Grind P80 75 microns	3 stage-Cleaning
MDL #s	CMC added in the 1st and 2nd cleaning	
Charge Mass	2500	grams
No. of Charges	1	charges



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2477.28	99.09	0.17	0.08	11.94	0.28	5.12	0.24	120.0	84.9	102.9	80.6	92.7	83.2
Call Factor			119.98	84.93	102.90	80.57	92.74	83.21						
3rd Clnr Conc	4.23	0.17	6.00	12.00	2.00	14.55	450.00	21.50	6.16	24.35	0.03	8.89	15.15	15.35
3rd Clnr Tails	7.72	0.31	5.81	4.15	6.50	22.00	378.00	10.00	10.88	15.37	0.17	24.54	23.22	13.03
2nd Clnr Tails	136.74	5.52	0.92	0.51	15.45	1.91	37.10	1.56	30.54	33.38	7.21	37.73	40.37	36.01
1st Clnr Tails	399.43	16.12	0.12	0.04	13.10	0.21	3.50	0.19	11.72	7.47	17.86	12.12	11.13	12.81
Rougher Tail	1929.16	77.87	0.09	0.02	11.35	0.06	0.66	0.07	40.70	19.43	74.73	16.72	10.13	22.80

Cumulative Product	Cum Weight		Cumulative Grade						Cum. Recovery					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
3rd Clnr Conc	4.23	0.17	6.00	12.00	2.00	14.55	450.00	21.50	6.2	24.3	0.0	8.9	15.1	15.4
3rd Clnr Tails	11.95	0.48	5.88	6.93	4.91	19.36	403.49	14.07	17.0	39.7	0.2	33.4	38.4	28.4
2nd Clnr Tails	148.69	6.00	1.32	1.02	14.60	3.31	66.55	2.57	47.6	73.1	7.4	71.2	78.7	64.4
1st Clnr Tails	**548.12**	**22.13**	**0.45**	**0.31**	**13.51**	**1.05**	**20.60**	**0.83**	**59.3**	**80.6**	**25.3**	**83.3**	**89.9**	**77.2**
Ro 1 Tail	2473.05	98.92	0.16	0.06	11.95	0.26	4.35	0.20	93.8	75.7	100.0	91.1	84.9	84.6
Ro 2 Tail	2465.33	98.61	0.14	0.05	11.97	0.19	3.17	0.17	83.0	60.3	99.8	66.6	61.6	71.6
Ro 3 Tail	2328.59	93.14	0.09	0.02	11.76	0.09	1.16	0.09	52.4	26.9	92.6	28.8	21.3	35.6
Ro 4 Tail	1929.16	77.17	0.09	0.02	11.45	0.06	0.67	0.07	40.7	19.4	74.7	16.7	10.1	22.8

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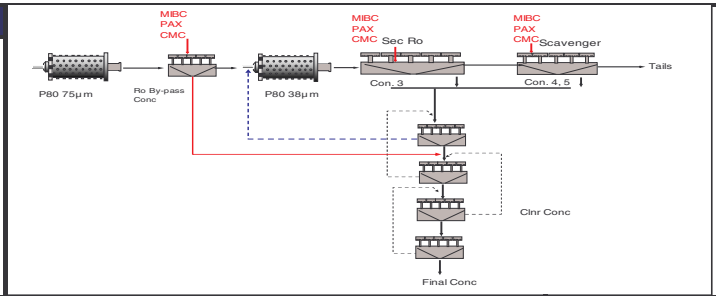
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork
Project #	5009817
Feed Sample	Offset Zone Composite
Note	Ro 1 and Ro 2 combined and report to 2nd Cleaning
Target Grind	SG P80 75/38 microns 3 stage-Cleaning
MDL #s	
Charge Mass	2500 grams
No. of Charges	1 charges



	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2469.43	98.78	0.17	0.10	11.84	0.36	5.54	0.30	120.9	103.3	102.1	102.8	100.4	103.5
Call Factor			120.92	103.34	102.08	102.85	100.35	103.52						
3rd Clnr Conc	11.52	0.47	4.14	13.10	3.36	30.00	435.00	16.00	11.55	59.87	0.13	39.36	37.09	25.17
3rd Clnr Tails	21.35	0.86	4.14	1.78	11.80	10.05	198.50	11.95	21.41	15.08	0.87	24.44	31.36	34.84
2nd Clnr Tails	251.73	10.19	0.29	0.14	14.55	0.62	10.40	0.59	17.62	13.78	12.68	17.78	19.37	20.28
1st Clnr Tails	438.67	17.76	0.11	0.03	11.65	0.13	1.76	0.13	11.37	4.35	17.69	6.49	5.71	7.79
Rougher Tail	1746.16	70.71	0.09	0.01	11.35	0.06	0.50	0.05	38.06	6.93	68.62	11.93	6.46	11.92
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
3rd Clnr Conc	11.52	0.47	4.14	13.10	3.36	30.00	435.00	16.00	11.6	59.9	0.1	39.4	37.1	25.2
3rd Clnr Tails	32.87	1.33	4.14	5.75	8.84	17.04	281.39	13.37	33.0	74.9	1.0	63.8	68.5	60.0
2nd Clnr Tails	284.60	11.52	0.73	0.79	13.89	2.52	41.70	2.07	50.6	88.7	13.7	81.6	87.8	80.3
1st Clnr Tails	**723.27**	**29.29**	**0.35**	**0.32**	**12.53**	**1.07**	**17.48**	**0.89**	**61.9**	**93.1**	**31.4**	**88.1**	**93.5**	**88.1**
Ro 1 Tail	2457.91	98.32	0.15	0.04	11.88	0.22	3.50	0.23	88.4	40.1	99.9	60.6	62.9	74.8
Ro 2 Tail	2436.56	97.46	0.12	0.03	11.88	0.13	1.77	0.12	67.0	25.1	99.0	36.2	31.5	40.0
Ro 3 Tail	2184.83	87.39	0.09	0.01	11.55	0.07	0.76	0.07	49.4	11.3	86.3	18.4	12.2	19.7
Ro 4 Tail	1746.16	69.85	0.09	0.01	11.49	0.06	0.51	0.05	38.1	6.9	68.6	11.9	6.5	11.9

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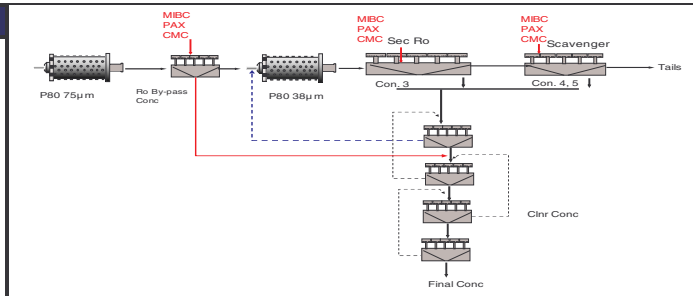
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork
Project #	5009817
Feed Sample	Offset Zone Composite
Note	Ro 1 and Ro 2 combined and report to 2nd Cleaning
Target Grind	Final Grind P80 38 microns 3 stage-Cleaning
MDL #s	CMC added in the 1st and 2nd cleaning
Charge Mass	2500 grams
No. of Charges	1 charges



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2494.96	99.80	0.15	0.10	11.90	0.34	5.56	0.41	107.1	99.6	102.6	97.7	100.8	140.4
Call Factor			107.06	99.57	102.63	97.71	100.78	140.42						
3rd Clnr Conc	8.55	0.34	2.32	14.54	0.64	32.00	600.00	32.00	5.32	50.14	0.02	32.13	37.04	26.98
3rd Clnr Tails	11.10	0.44	5.53	3.27	8.63	17.50	296.00	31.90	16.45	14.64	0.32	22.81	23.72	34.92
2nd Clnr Tails	233.26	9.35	0.45	0.16	13.70	0.80	15.15	0.68	28.31	14.86	10.78	21.91	25.51	15.64
1st Clnr Tails	481.27	19.29	0.13	0.04	12.30	0.19	2.05	0.29	16.89	7.57	19.97	10.74	7.12	13.77
Rougher Tail	1760.78	70.57	0.07	0.02	11.60	0.06	0.52	0.05	33.03	12.78	68.91	12.41	6.61	8.68
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
3rd Clnr Conc	8.55	0.34	2.32	14.54	0.64	32.00	600.00	32.00	5.3	50.1	0.0	32.1	37.0	27.0
3rd Clnr Tails	19.65	0.79	4.13	8.17	5.15	23.81	428.27	31.94	21.8	64.8	0.3	54.9	60.8	61.9
2nd Clnr Tails	252.91	10.14	0.74	0.78	13.04	2.59	47.25	3.11	50.1	79.6	11.1	76.9	86.3	77.6
1st Clnr Tails	**734.18**	**29.43**	**0.34**	**0.29**	**12.55**	**1.02**	**17.62**	**1.26**	**67.0**	**87.2**	**31.1**	**87.6**	**93.4**	**91.3**
Ro 1 Tail	2486.41	99.46	0.14	0.05	11.94	0.23	3.51	0.30	94.7	49.9	100.0	67.9	63.0	73.0
Ro 2 Tail	2475.31	99.01	0.12	0.04	11.96	0.16	2.20	0.16	78.2	35.2	99.7	45.1	39.2	38.1
Ro 3 Tail	2242.05	89.68	0.08	0.02	11.77	0.09	0.85	0.10	49.9	20.4	88.9	23.1	13.7	22.4
Ro 4 Tail	1760.78	70.43	0.07	0.02	11.62	0.06	0.52	0.05	33.0	12.8	68.9	12.4	6.6	8.7

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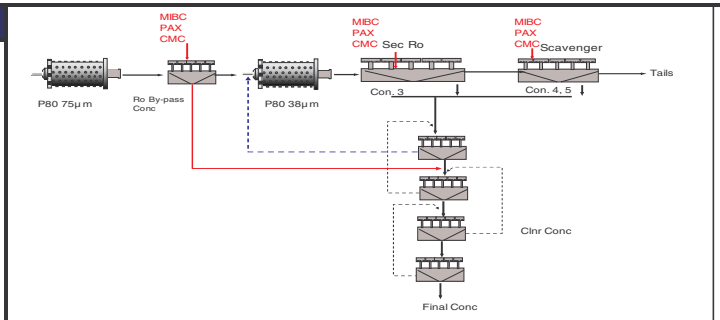
LDI Flotation Mass Balance Spreadsheet

Project Name	LDI Roby and Offset Zone Ore Characterisation Testwork
Project #	5009817
Feed Sample	Offset Zone Composite
Note	Ro 1 report to 2nd Cleaning **(2 CHARGES)**
Target Grind	Final Grind P80 38 microns 3 stage-Cleaning
MDL #s	CMC added in the 1st and 2nd cleaning
Charge Mass	2500 grams
No. of Charges	1 charges



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	Weight		Assays						Distribution					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
Head Assay	2500.00	100.00	0.14	0.10	11.60	0.35	5.52	0.29	100.0	100.0	100.0	100.0	100.0	100.0
Calc Head Assay	2477.28	99.09	0.14	0.14	11.25	0.27	4.40	0.29	102.3	142.2	97.0	76.2	79.7	101.2
Call Factor			102.28	142.23	97.01	76.21	79.65	101.19						
3rd Clnr Conc	4.23	0.17	2.44	19.70	1.75	43.80	750.00	50.80	2.94	23.87	0.03	28.30	29.39	29.83
3rd Clnr Tails	7.72	0.31	4.21	6.48	6.99	26.20	359.00	26.20	9.25	14.33	0.20	30.89	25.68	28.08
2nd Clnr Tails	136.74	5.52	0.63	0.24	14.15	1.01	21.60	1.22	24.35	9.40	7.00	21.09	27.37	23.16
1st Clnr Tails	399.43	16.12	0.15	0.02	12.00	0.13	2.62	0.10	16.82	2.40	17.35	7.93	9.70	5.54
Rougher Tail	1929.16	77.87	0.09	0.09	10.80	0.04	0.44	0.05	46.65	50.00	75.42	11.79	7.86	13.39
Cumulative Product	**Cum Weight**		**Cumulative Grade**						**Cum. Recovery**					
	(g)	(%)	Ni	Cu	MgO	Pt	Pd	Au	Ni	Cu	MgO	Pt	Pd	Au
3rd Clnr Conc	4.23	0.17	2.44	19.70	1.75	43.80	750.00	50.80	2.9	23.9	0.0	28.3	29.4	29.8
3rd Clnr Tails	11.95	0.48	3.58	11.16	5.14	32.43	497.40	34.91	12.2	38.2	0.2	59.2	55.1	57.9
2nd Clnr Tails	148.69	6.00	0.86	1.12	13.43	3.54	59.84	3.93	36.5	47.6	7.2	80.3	82.4	81.1
1st Clnr Tails	**548.12**	**22.13**	**0.34**	**0.32**	**12.39**	**1.05**	**18.14**	**1.14**	**53.4**	**50.0**	**24.6**	**88.2**	**92.1**	**86.6**
Ro 1 Tail	2473.05	98.92	0.14	0.11	11.27	0.19	3.11	0.21	97.1	76.1	100.0	71.7	70.6	70.2
Ro 2 Tail	2465.33	98.61	0.13	0.09	11.28	0.11	1.98	0.12	87.8	61.8	99.8	40.8	44.9	42.1
Ro 3 Tail	2328.59	93.14	0.10	0.08	11.11	0.06	0.82	0.06	63.5	52.4	92.8	19.7	17.6	18.9
Ro 4 Tail	1929.16	77.17	0.09	0.09	10.90	0.04	0.44	0.05	46.6	50.0	75.4	11.8	7.9	13.4

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Appendix M- Roby Locked Cycle Test Flotation Schedules

LDI Roby LCT ROUGHER/SBAVENGER FLOTATION TEST REPORT

DATE: April 21, 2010 — TEST FEED: **5 kg Ground Products + 1/2 1st Clnr Tails**

TEST OBJEBTIVES: **Baseline @ p80 38 microns**

GRINDING BONDITIONS (AFTER 10MIN. BLEANING WITH SIL.SAND)		**AERATION:**	
		ROD TYPE:	70% MILD STEEL 30% STAINLESS
MILL TYPE:	**STAINLESS**		
CHARGE:	**5000 g**	**FEED SIZE:**	**-10 M**
WATER:	**1466 ml**	**WATER pH:**	
GRIND:	**P80 = 38 micron**	**GRIND TIME:**	

FLOTATION BONDITIONS		FLOATED BY: LJB	
BELL TYPE:	Denver	BELL VOLUME:	4 liter
IMPELLER rpm:	1600	REPULP WATER:	DOMESTIB
RO GAS RATE:	5 l/min (23)	SBAVENGER GAS RATE:	9 l/min (53)
STROKE RATE : rougher	1/5 seB	SBAVENGER	
	:sec and scav 1/5 seB		

REAGENT	QUANTITY ml, g	DOSE g/t	STAGE or OPERATION	COND TIME	FLOAT TIME	Stroke Rate		pH / REDOX as is: BOMMENTS
Feed: 5 kg Ground Products + 1/2 1st Clnr Tails								INITIAL STIR TIME 1 min.
CMC(1%)	10	20		2				
PAX (1%)	2.9	5.76		2				5L/min
MIBC	4 drops (28)	28	Conc 1	0	3	1/10		PRI.Ro. CONC.
PAX (1%)	2.5	5.04		0.5	3	1/5		**Change Stroke to 1/5*
Change Pan								
PAX (1%)	1.9	4	Conc 2	1	3			
PAX (1%)	1.9	4		1	3			
CMC (1%)	18	36		2				**Increase Air to 9 l/min**
PAX (1%)	1.0	2	Conc 3					
Aero 3477 (1%)	2.0	4						
				0.5	3	1/5		
PAX (1%)	1.0	2						
Aero 3477 (1%)	2.0	4		1	3			
CMC (1%)	18	36		2				
PAX (1%)	1.0	2						
Aero 3477 (1%)	2.0	4		1				
			Conc 4	0.5	4	1/5		
PAX (1%)	1.0	2						
Aero 3477 (1%)	2.0	4		1	4	1/5		

Collected in the Ro Conc 1 Pan For 2nd Clnr Flotation Record Slurry Weight

Collected in the Ro Conc 2-4 Pan For 1st Clnr Flotation Record Slurry Weight

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LDI Roby LCT 1st Cleaning Flotation REPORT

TEST#	PROJECT #	DATE:	TEST FEED:	**Two Ro-Scav Conc 2-4 + the 2nd Clnr Tails from previous cycle**	
TEST OBJECTIVES:	To reduce the gangue content of the secondary rougher concentrate			TRANSFER TIME:	Test order:

GRINDING CONDITIONS		**AERATION:**	**N/A**
		ROD TYPE:	**N/A**
MILL TYPE:	**N/A**		
CHARGE:	**N/A**	**FEED SIZE:**	**N/A**
WATER:	**N/A**	**WATER pH:**	
GRIND:		**GRIND TIME:**	

FLOTATION CONDITIONS		FLOATED BY:	
CELL TYPE:	Denver	CELL VOLUME:	**4** L
IMPELLER rpm:	**1200**	REPULP WATER:	PROCESS WATER
RO GAS RATE:	2 l/min	SCAVENGER GAS RATE:	L/MIN
STROKE RATE *): rougher	1/10 sec	SCAVENGER	

REAGENT	QUANTITY ml, g	DOSE g/ton	STAGE or OPERATION	COND TIME	FLOAT TIME					
								pH as is:		COMMENTS
								INITIAL STIR TIME:	1 MIN	
FEED: Two Ro-Scav Conc 2-4 + the 2nd Clnr Tails from previous cycle										
CMC (1%)	25	25		2						
MIBC										
PAX (1%)	0.00	0								
Aero 3477 (1%	0.0	0		1						
			CONC 1		2.0					
			CONC 2		2.0					
			CONC 3		2.0					
CONC 1, 6 STROKES/MIN										

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LDI Roby LCT 2nd Cleaning Flotation REPORT

TEST FEED: **Two Ro Conc 1 + the 1st Clnr Conc and the 3rd Clnr Tails from previous cycle**

TEST OBJECTIVES: To reduce the gangue content of the concentrate

TRANSFER TIME: Test order:

GRINDING CONDITIONS		AERATION:	N/A	FLOTATION CONDITIONS	FLOATED BY:
		ROD TYPE:	N/A		
MILL TYPE:	N/A			CELL TYPE: Denver	CELL VOLUME: 2.5 L
CHARGE:	N/A	FEED SIZE:	N/A	IMPELLER rpm: 1200	REPULP WATER: PROCESS WATER
WATER:	N/A	WATER pH:		RO GAS RATE: 2 l/min	SCAVENGER GAS RATE: L/MIN
GRIND:		GRIND TIME:		STROKE RATE *): rougher 1/10 sec	SCAVENGER

REAGENT	QUANTITY ml, g	DOSE g/ton	STAGE or OPERATION	COND TIME	FLOAT TIME			pH as is: / INITIAL STIR TIME: 1 MIN	COMMENTS		
FEED: Two Ro Conc 1 + the 1st Clnr Conc and the 3rd Clnr Tails from previous cycle											
CMC (1%)	25	25		2							
MIBC											
PAX (1%)	0.0	0									
Aero 3477 (1%	0.0	0		1	3						
			CONC 1		2.0			Assess need for reagent (*xanthate after conc 1)			
			CONC 2		2.0						
			CONC 3		2.0						
CONC 1, 6 STROKES/MIN											

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LDI Roby LCT 3rd Cleaning Flotation REPORT

TEST#	PROJECT #	DATE:				TEST FEED:	**2nd Clnr Conc**					
TEST OBJECTIVES:	To reduce the gangue content of the secondary cleaner concentrate					TRANSFER TIME:		Test order:				
GRINDING CONDITIONS			**AERATION:**	**N/A**		FLOTATION CONDITIONS		FLOATED BY:				
			ROD TYPE:	N/A								
MILL TYPE:	**N/A**					CELL TYPE:	Denver	CELL VOLUME:	**1** L			
CHARGE:	**N/A**		**FEED SIZE:**	**N/A**		IMPELLER rpm:	**850**	REPULP WATER:	PROCESS WATER			
WATER:	**N/A**		**WATER pH:**			RO GAS RATE:	2 l/min	SCAVENGER GAS RATE:	L/MIN			
GRIND:			**GRIND TIME:**			STROKE RATE *): rougher	1/10 sec	SCAVENGER				
REAGENT	**QUANTITY ml, g**	**DOSE g/ton**	**STAGE or OPERATION**	**COND TIME**	**FLOAT TIME**			pH as is: INITIAL STIR TIME:	1 MIN	COMMENTS		
Feed: the 2nd Clnr Conc												
			CONC 1		2.0			Assess need for reagent (?)				
			CONC 2		2.0							
			CONC 3		2.0							
CONC 1, 6 STROKES/MIN												

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Appendix N- Offset Locked Cycle Test Flotation Schedules

LDI Offset LCT ROUGHER/SBAVENGER FLOTATION TEST REPORT

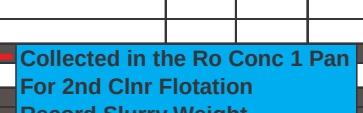
DATE: April 21, 2010 — TEST FEED: **7.5 kg Ground Products + 1/2 1st Clnr Tails**

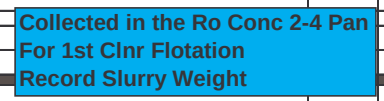
TEST OBJEBTIVES: **Baseline @ p80 38 microns**

GRINDING BONDITIONS (AFTER 10MIN. BLEANING WITH SIL.SAND)		**AERATION:**	
MILL TYPE:	**STAINLESS**	**ROD TYPE:**	70% MILD STEEL 30% STAINLESS
CHARGE:	**7500 g**	**FEED SIZE:**	**-10 M**
WATER:	**1466 ml**	**WATER pH:**	
GRIND:	**P80 = 38 micron**	**GRIND TIME:**	

FLOTATION BONDITIONS		FLOATED BY: LJB	
BELL TYPE:	Denver	BELL VOLUME:	4 liter
IMPELLER rpm:	1600	REPULP WATER:	DOMESTIB
RO GAS RATE:	5 l/min (23)	SBAVENGER GAS RATE:	9 l/min (53)
STROKE RATE : rougher 1/5 seB		SBAVENGER	
	:sec and scav 1/5 seB		

REAGENT	QUANTITY ml, g	DOSE g/t	STAGE or OPERATION	COND TIME	FLOAT TIME	Stroke Rate		pH / REDOX as is: BOMMENTS
Feed: 5 kg Ground Products + 1/2 1st Clnr Tails								INITIAL STIR TIME 1 min.
CMC(1%)	13	20		2				
PAX (1%)	3.7	5.76		2				5L/min
MIBC	4 drops (28)	28	**Conc 1**	0	3	1/10		PRI.Ro. CONC.
PAX (1%)	3.3	5.04		0.5	3	1/5		**Change Stroke to 1/5*
Change Pan								
PAX (1%)	2.5	3.8	**Conc 2**	1	3			
PAX (1%)	2.5	3.8		1	3			
CMC (1%)	18	36.0		2				**Increase Air to 9 l/min**
PAX (1%)	1.3	2.0	**Conc 3**					
Aero 3477 (1%)	2.0	4.0						
				0.5	3	1/5		
PAX (1%)	1.3	2.0						
Aero 3477 (1%)	2.0	4.0		1	3			
CMC (1%)	18	36		2				
PAX (1%)	1.3	2						
Aero 3477 (1%)	2.0	4		1				
			Conc 4	0.5	4	1/5		
PAX (1%)	1.3	2						
Aero 3477 (1%)	2.0	4		1	4	1/5		





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LDI Offset LCT 1st Cleaning Flotation REPORT

TEST#	PROJECT #	DATE:	TEST FEED:	**Two Ro-Scav Conc 2-4 + the 2nd Clnr Tails from previous cycle**
TEST OBJECTIVES:	To reduce the gangue content of the secondary rougher concentrate		TRANSFER TIME:	Test order:

GRINDING CONDITIONS		**AERATION:**	**N/A**	FLOTATION CONDITIONS	FLOATED BY:
		ROD TYPE:	**N/A**		
MILL TYPE:	**N/A**			CELL TYPE: Denver	CELL VOLUME: **4** L
CHARGE:	**N/A**	**FEED SIZE:**	**N/A**	**IMPELLER rpm:** **1200**	REPULP WATER: PROCESS WATER
WATER:	**N/A**	**WATER pH:**		RO GAS RATE: 2 l/min	SCAVENGER GAS RATE: L/MIN
GRIND:		**GRIND TIME:**		STROKE RATE *): rougher 1/10 sec	SCAVENGER

REAGENT	**QUANTITY ml, g**	**DOSE g/ton**	**STAGE or OPERATION**	**COND TIME**	**FLOAT TIME**	START	END	pH as is: INITIAL STIR TIME: 1 MIN	COMMENTS		
FEED: Two Ro-Scav Conc 2-4 + the 2nd Clnr Tails from previous cycle											
CMC (1%)	20	26.7		2							
MIBC											
PAX (1%)	0.00	0									
Aero 3477 (1%)	0.0	0		1							
			CONC 1		2.0						
			CONC 2		2.0						
			CONC 3		2.0						
CONC 1, 6 STROKES/MIN											

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LDI Offset LCT 2nd Cleaning Flotation REPORT

TEST FEED: **Two Ro Conc 1 + the 1st Clnr Conc and the 3rd Clnr Tails from previous cycle**

TEST OBJECTIVES: To reduce the gangue content of the concentrate

TRANSFER TIME:

Test order:

GRINDING CONDITIONS				FLOTATION CONDITIONS			
		AERATION:	N/A			FLOATED BY:	
		ROD TYPE:	N/A				
MILL TYPE:	N/A			CELL TYPE:	Denver	CELL VOLUME:	2.5 L
CHARGE:	N/A	FEED SIZE:	N/A	IMPELLER rpm:	1200	REPULP WATER:	PROCESS WATER
WATER:	N/A	WATER pH:		RO GAS RATE:	2 l/min	SCAVENGER GAS RATE:	L/MIN
GRIND:		GRIND TIME:		STROKE RATE *): rougher	1/10 sec	SCAVENGER	

REAGENT	QUANTITY ml, g	DOSE g/ton	STAGE or OPERATION	COND TIME	FLOAT TIME	pH START	pH END	pH as is: / INITIAL STIR TIME:	COMMENTS
								1 MIN	
FEED: Two Ro Conc 1 + the 1st Clnr Conc and the 3rd Clnr Tails from previous cycle									
CMC (1%)	20	26.7		2					
MIBC									
PAX (1%)	0.0	0							
Aero 3477 (1%	0.0	0		1	3				
			CONC 1		2.0			Assess need for reagent (*xanthate after conc 1)	
			CONC 2		2.0				
			CONC 3		2.0				
CONC 1, 6 STROKES/MIN									

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LDI Offset LCT 3rd Cleaning Flotation REPORT

TEST#	PROJECT #		DATE:					TEST FEED:	**2nd Clnr Conc**			
TEST OBJECTIVES:	To reduce the gangue content of the secondary cleaner concentrate							TRANSFER TIME:	Test order:			
GRINDING CONDITIONS			**AERATION:**	**N/A**				FLOTATION CONDITIONS	FLOATED BY:			
			ROD TYPE:	N/A								
MILL TYPE:	**N/A**							CELL TYPE: Denver	CELL VOLUME: **1** L			
CHARGE:	**N/A**		**FEED SIZE:**	**N/A**				IMPELLER rpm: **850**	REPULP WATER: PROCESS WATER			
WATER:	**N/A**		**WATER pH:**					RO GAS RATE: 2 l/min	SCAVENGER GAS RATE: L/MIN			
GRIND:			**GRIND TIME:**					STROKE RATE *): rougher 1/10 sec	SCAVENGER			
REAGENT	**QUANTITY ml, g**	**DOSE g/ton**	**STAGE or OPERATION**	**COND TIME**	**FLOAT TIME**			pH as is: INITIAL STIR TIME: 1 MIN	COMMENTS			
Feed: the 2nd Clnr Conc												
			CONC 1		2.0			Assess need for reagent (?)				
			CONC 2		2.0							
			CONC 3		2.0							
CONC 1, 6 STROKES/MIN												

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Appendix O- Roby Locked Cycle Test Mass Balance

Product	Weight		Assay % or g/t						Distribution %					
	g		Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Final Clnr Conc -Cycle 1	50.0	0.1%	3.25	8.93	809	43.2		5.59	2.3	10.7	10.6	10.4	10.4	0.0
Final Clnr Conc -Cycle 2	73.7	0.1%	3.68	8.31	665	34		8.84	3.8	14.7	12.8	12.0	12.0	0.0
Final Clnr Conc -Cycle 3	82.2	0.1%	5.21	7.29	644	32		9.18	6.0	14.4	13.8	12.6	12.6	0.0
Final Clnr Conc -Cycle 4	74.3	0.1%	5.33	8.75	763	36		6.77	5.6	15.6	14.8	12.8	12.8	0.0
Final Clnr Conc -Cycle 5	71.5	0.1%	4.35	7.6	642	32.7		8.85	4.4	13.0	12.0	11.2	11.2	0.0
Final Clnr Conc -Cycle 6	97.6	0.2%	4.76	5.55	546.0	27.1		10.6	6.5	13.0	13.9	12.7	12.7	0.0
Ro-Scav Tails- Cycle 1a	3345.7	5.7%	0.072	0.008	0.6	0.05	0.04	12.2	3.4	0.6	0.5	0.8	0.8	3.4
Ro-Scav Tails- Cycle 1b	3378.1	5.8%	0.072	0.007	0.6	0.05	0.04	12.15	3.4	0.6	0.5	0.8	0.8	3.4
Ro-Scav Tails- Cycle 2a	3599.6	6.2%	0.071	0.006	0.6	0.05	0.04	12.1	3.6	0.5	0.6	0.9	0.9	3.7
Ro-Scav Tails- Cycle 2b	3578.9	6.1%	0.072	0.006	0.61	0.05	0.04	12.3	3.6	0.5	0.6	0.9	0.9	3.6
Ro-Scav Tails- Cycle 3a	3798.8	6.5%	0.07	0.007	0.64	0.05	0.04	12.2	3.7	0.6	0.6	0.9	0.9	3.9
Ro-Scav Tails- Cycle 3b	3826.9	6.6%	0.07	0.007	0.6	0.05	0.04	12.1	3.8	0.6	0.6	0.9	0.9	3.9
Ro-Scav Tails- Cycle 4a	5444.9	9.3%	0.074	0.009	0.79	0.06	0.08	12.4	5.7	1.2	1.1	1.6	1.6	11.1
Ro-Scav Tails- Cycle 4b	5279.3	9.0%	0.076	0.011	0.79	0.08	0.06	12.25	5.6	1.4	1.1	2.0	2.0	8.1
Ro-Scav Tails- Cycle 5a	4932.9	8.4%	0.067	0.008	0.75	0.06	0.06	12.35	4.6	0.9	1.0	1.4	1.4	7.5
Ro-Scav Tails- Cycle 5b	4963.4	8.5%	0.071	0.007	0.75	0.06	0.05	12.4	5.0	0.8	1.0	1.4	1.4	6.3
Ro-Scav Tails- Cycle 6a	4740.9	8.1%	0.075	0.008	0.72	0.06	0.05	12.4	5.0	0.9	0.9	1.4	1.4	6.0
Ro-Scav Tails- Cycle 6b	5203.0	8.9%	0.079	0.011	0.79	0.08	0.06	12.6	5.8	1.4	1.1	2.0	2.0	7.9
1st Clnr Tails- Cycle 6	3886.3	6.7%	0.145	0.031	4.02	0.26	0.2	13.45	7.9	2.9	4.1	4.8	4.8	19.8
2nd Clnr Tails- Cycle 6	1708.8	2.9%	0.167	0.052	6.07	0.36	0.24	14	4.0	2.1	2.7	3.0	3.0	10.4
3rd Clnr Tails- Cycle 6	287.6	0.5%	1.535	0.506	77.3	4	0.123	3.79	6.2	3.5	5.8	5.5	5.5	0.9
Head Calculated	58424.3	100.0%	0.12	0.07	6.56	0.36	0.07	12.36	100.0	100.0	100.0	100.0	100.0	100.0
	9737.4													

Combined Products	Weight		Assay %,g/t						Distribution					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Combine Concentrate	449.2	0.8%	4.53	7.55	663.87	33.28	0.00	8.58	28.6	81.3	77.9	71.7	87.5	0.5
Combine Tail	52092.3	89.2%	0.07	0.01	0.70	0.06	0.05	12.31	53.2	10.1	9.5	15.0		88.8
Intermediate Tails	5882.7	10.1%	0.22	0.06	8.20	0.47	0.21	13.14	18.2	8.5	12.6	13.3		10.7
Calc. Head	58424.3	100.0%	0.12	0.07	6.56	0.36		12.36	100.0	100.0	100.0	100.0		100.0
ERD Head	60000.0	97.4%	0.12	0.07	6.12	0.31	0.37	12.20						

* Au Recovery based on assay head and Tails assay

Stability/Mass Conservation

Cycle	wt	Ni	Cu	Pd	MgO	Pt
1	69.6	54.6	71.5	69.7	68.2	71.8
2	74.5	66.2	94.3	83.6	73.3	82.4
3	79.2	81.2	93.8	90.3	77.6	86.6
4	110.9	101.3	108.9	102.1	110.2	98.5
5	102.4	83.9	88.8	83.5	102.3	84.4
6	103.1	103.9	91.6	95.3	104.2	96.3
Average 3-6	105.5	96.4	96.4	93.6	105.6	93.1

2P Formula - Cu	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	9737.4	100.00	0.12	**0.07**	6.37	0.34	0.06	12.37	100.00	100.00	100.00	100.00	100.00	100.00
Concentrate	85.27	0.88	**4.81**	**7.13**	**640**	**31.46**	**0.00**	**8.92**	**36.58**	**87.44**	**88.07**	**80.61**	**0.00**	**0.63**
Tails	9652.1	99.12	0.07	**0.01**	0.77	0.07	0.06	12.40	63.42	12.56	11.93	19.39	100.00	99.37

SME Handbook	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	10269.3	100.00	0.11	0.07	5.82	0.31	0.06	12.37	100.00	100.00	100.00	100.00	100.00	100.00
Concentrate	81.1	0.79	**4.81**	**7.13**	**640.41**	**31.46**	**0.00**	**8.92**	**34.20**	**86.26**	**86.94**	**78.93**	**0.00**	**0.57**
Tails	10188.1	99.21	0.07	0.01	0.77	0.07	0.06	12.40	65.80	13.74	13.06	21.07	100.00	99.43

Concentrate Production - Unaccounted in Tails	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	9737.4	100.00	0.12	0.07	6.56	0.36	0.00	12.36	100.00	100.00	100.00	100.00	#DIV/0!	100.00
Concentrate	81.1	0.83	**4.81**	**7.13**	**640.41**	**31.46**	**0.00**	**8.92**	**32.96**	**83.19**	**81.40**	**73.46**	**0.00**	**0.60**
Tails	9656.2	99.17	0.08	0.01	1.23	0.10	12.47	12.39	67.04	16.81	18.60	26.54	100.00	99.40

ALTERNATIVE Unaccounted in the Conc	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	10269.3	100.00	0.12	0.07	6.56	0.36	0.00	12.36	100.00	100.00	100.00	100.00	#DIV/0!	100.00
Concentrate	81.1	0.79	**6.14**	**7.90**	**733.53**	**36.77**	**36.77**	**7.55**	**39.88**	**87.43**	**88.40**	**81.41**	**81.41**	**0.48**
Tails	10188.1	99.21	0.07	0.01	0.77	0.07	0.06	12.40	60.12	12.57	11.60	18.59	18.59	99.52

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Appendix P- Offset Locked Cycle Test Mass Balance

Product	Weight		Assay % or g/t						Distribution %					
	g		Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Final Clnr Conc -Cycle 1	26.9	0.1%	2.64	17.84	545	37.1		4.04	0.9	9.0	5.3	5.4	5.4	0.0
Final Clnr Conc -Cycle 2	37.5	0.1%	2.89	16.74	480	35.9		4.34	1.3	11.8	6.5	7.3	7.3	0.0
Final Clnr Conc -Cycle 3	38.9	0.1%	2.18	10.7	490	35.1		4.3	1.1	7.8	6.9	7.4	7.4	0.0
Final Clnr Conc -Cycle 4	62.9	0.1%	2.6	8.42	388	26.4		6.08	2.0	9.9	8.8	9.0	9.0	0.0
Final Clnr Conc -Cycle 5	72.9	0.1%	2	10.52	370	39.2		6.58	1.8	14.4	9.8	15.5	15.5	0.0
Final Clnr Conc -Cycle 6	60.3	0.1%	3.63	13.42	444	41.6		9.2	2.7	15.2	9.7	13.6	13.6	0.0
Final Clnr Conc -Cycle 7	69.2	0.1%	3.1	8.42	366	25.6		7.58	2.7	11.0	9.2	9.6	9.6	0.0
Ro-Scav Tails- Cycle 1a	2874.1	5.5%	0.07	0.006	0.41	0.04	0.04	11.7	2.5	0.3	0.4	0.6	0.6	2.7
Ro-Scav Tails- Cycle 1b	3036.2	5.9%	0.088	0.007	0.41	0.04	0.05	11.6	3.3	0.4	0.5	0.7	0.7	3.6
Ro-Scav Tails- Cycle 2a	3536.2	6.8%	0.079	0.009	0.41	0.04	0.04	11.6	3.5	0.6	0.5	0.8	0.8	3.3
Ro-Scav Tails- Cycle 2b	3452.9	6.7%	0.086	0.007	0.44	0.04	0.04	11.7	3.7	0.5	0.5	0.7	0.7	3.2
Ro-Scav Tails- Cycle 3a	3633.4	7.0%	0.084	0.009	0.46	0.03	0.04	11.6	3.8	0.6	0.6	0.6	0.6	3.4
Ro-Scav Tails- Cycle 3b	3576.2	6.9%	0.084	0.011	0.54	0.04	0.05	11.9	3.7	0.7	0.7	0.8	0.8	4.2
Ro-Scav Tails- Cycle 4a	3467.1	6.7%	0.086	0.008	0.49	0.04	0.04	11.95	3.7	0.5	0.6	0.7	0.7	3.3
Ro-Scav Tails- Cycle 4b	3568.1	6.9%	0.084	0.009	0.52	0.04	0.05	12.25	3.7	0.6	0.7	0.8	0.8	4.2
Ro-Scav Tails- Cycle 5a	3490.7	6.7%	0.086	0.01	0.53	0.04	0.05	11.8	3.7	0.7	0.7	0.8	0.8	4.1
Ro-Scav Tails- Cycle 5b	3547.0	6.8%	0.082	0.01	0.52	0.04	0.05	12	3.6	0.7	0.7	0.8	0.8	4.2
Ro-Scav Tails- Cycle 6a	3494.1	6.7%	0.085	0.009	0.51	0.05	0.04	12.05	3.7	0.6	0.6	0.9	0.9	3.3
Ro-Scav Tails- Cycle 6b	3575.4	6.9%	0.089	0.012	0.51	0.05	0.05	12	3.9	0.8	0.7	1.0	1.0	4.2
Ro-Scav Tails- Cycle 7a	3457.4	6.7%	0.089	0.008	0.5	0.05	0.05	11.95	3.8	0.5	0.6	0.9	0.9	4.1
Ro-Scav Tails- Cycle 7b	3507.2	6.8%	0.089	0.012	0.57	0.07	0.05	11.9	3.9	0.8	0.7	1.3	1.3	4.1
1st Clnr Tails- Cycle 7	1972.2	3.8%	0.395	0.081	9.78	0.41	0.27	15.55	9.7	3.0	7.0	4.4	4.4	12.5
2nd Clnr Tails- Cycle 7	1135.5	2.2%	1.39	0.234	44.6	1.37	1.34	16.25	19.6	5.0	18.3	8.4	8.4	35.7
3rd Clnr Tails- Cycle 7	186.4	0.4%	3.36	1.315	148.5	8.18		14.5	7.8	4.6	10.0	8.2	8.2	0.0
Head Calculated	51878.5	100.0%	0.16	0.10	5.33	0.36	0.08	12.07	100.0	100.0	100.0	100.0	100.0	100.0
	8646.4													

Combined Products	Weight		Assay %, g/t						Distribution					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Combine Concentrate	368.6	0.7%	2.73	11.43	421.04	33.93	0.00	6.46	12.5	79.1	56.2	67.6	85.34	0.38
Combine Final Tail	48215.8	92.9%	0.08	0.01	0.49	0.04	0.05	11.86	50.5	8.3	8.5	11.4		91.3
Intermediate Tails	3294.1	6.3%	0.91	0.20	29.63	1.18	0.62	15.73	37.0	12.6	35.3	21.0		8.3
Calc. Head	51878.5	100.0%	0.16	0.10	5.33	0.36		12.07	100.0	100.0	100.0	100.0		100.0
ERD Head	52500.0	98.8%	0.14	0.10	5.52	0.35	0.29	11.60						

*Au recovery based on average tailings assay

Stability/Mass Conservation

Cycle	wt	Ni	Cu	Pd	MgO	Pt
1	80.1	46.8	68.2	43.3	77.1	46.7
2	94.8	59.4	89.8	53.1	85.4	60.8
3	97.8	59.9	64.2	57.4	91.2	62.2
4	95.8	66.1	77.5	70.8	92.7	72.1
5	95.9	64.0	110.2	77.7	95.2	117.7
6	96.2	72.4	116.2	76.9	94.5	105.6
7	94.9	72.4	85.8	73.6	95.8	77.7
Average 4-7	95.7	69.6	104.1	76.1	95.2	100.3

2P Formula - Cu	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	7411.2	100.00	0.11	0.10	4.09	0.35	0.05	11.95	100.00	100.00	100.00	100.00	100.00	100.00
Concentrate	67.97	0.92	**2.80**	**10.13**	**390**	**33.16**	**0.00**	**7.32**	**23.11**	**90.58**	**87.44**	**86.60**	**0.00**	**0.56**
Tails	7343.2	99.08	0.09	0.01	0.52	0.05	0.05	11.99	76.89	9.42	12.56	13.40	100.00	99.44

SME Handbook	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	7091.4	100.00	0.11	0.11	4.22	0.36	0.05	11.96	100.00	100.00	100.00	100.00	100.00	100.00
Concentrate	67.5	0.95	**2.80**	**10.13**	**390.04**	**33.16**	**0.00**	**7.32**	**23.97**	**90.96**	**87.95**	**88.03**	**0.00**	**0.58**
Tails	7023.9	99.05	0.09	0.01	0.51	0.04	0.05	12.01	76.03	9.04	12.05	11.97	100.00	99.42

Concentrate Production - Unaccounted in Tails	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	7411.2	100.00	0.16	0.10	5.33	0.36	0.00	12.07	100.00	100.00	100.00	100.00	#DIV/0!	100.00
Concentrate	67.5	0.91	**2.80**	**10.13**	**390.04**	**33.16**	**0.00**	**7.32**	**16.40**	**89.93**	**66.67**	**84.69**	**#DIV/0!**	**0.55**
Tails	7343.7	99.09	0.13	0.01	1.79	0.06	0.00	12.11	83.60	10.07	33.33	15.31	#DIV/0!	99.45

ALTERNATIVE Unaccounted in the Conc	Weight		Assay, %						Distribution, %					
	g	%	Ni	Cu	Pd	Pt	Au	MgO	Ni	Cu	Pd	Pt	Au	MgO
Feed	7091.4	100.00	0.16	0.10	5.33	0.36	0.00	12.07	100.00	100.00	100.00	100.00	#DIV/0!	100.00
Concentrate	67.5	0.95	**7.45**	**9.77**	**506.35**	**32.96**	**-4.86**	**18.30**	**45.63**	**90.66**	**90.46**	**87.96**	**87.96**	**1.44**
Tails	7023.9	99.05	0.09	0.01	0.51	0.04	0.05	12.01	54.37	9.34	9.54	12.04	12.04	98.56

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GRINDING CIRCUIT EVALUATION FOR THE LAC DES ILES MINE

prepared for

NORTH AMERICAN PALLADIUM LTD.

Project 12126-002 – Final Report – Revision 2
June 29, 2008

NOTE:
This report refers to the samples as received.
The practice of this Company in issuing reports of this nature is to require the recipient not to publish the report or any part thereof without the written consent of SGS Minerals Services.

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Executive Summary

The existing SAG and ball mills from the Lac des Iles plant were used to configure a grinding circuit capable of treating 7,500 t/d to a final grind of 53 µm. The JKSimMet simulations indicated the following:

- A coarse F_{80} of 150 mm is required to supply coarse grinding media to the FAG mill.
- A pebble crusher is required to achieve the target throughput.
- The 30' x 14' AG mill, fitted with 3" grates, can be operated in a fully autogenous mode and achieve the target throughput of 7,500 t/d, although it is relatively tight.
- A single 20' x 34' mill can be used as a primary pebble mill to grind the ore down to 110 µm.
- A secondary 20' x 34' pebble mill is required to regrind the rougher/scavenger tails to 53 µm.
- In single-stage mode, only 5,350 t/d would be achieved by the 30' x 14' AG mill, to grind the ore to 110 µm.

It is imperative that the findings from the simulations be confirmed in a pilot plant. The balance between grinding and media survival must be confirmed in a pilot plant, over an extended period of time, for the pebble mill options. The pilot plant would also allow the confirmation of power requirements for the different grinding stages, FAG feed size requirement, the effect of pebble crushing, etc. Fully-autogenous circuits are highly ore-dependant and a design that is strictly based on computer simulation carries a high level of uncertainty.

Additional calculations were made using the old grinding mills available at the LDI site. The results showed that a throughput rate of 3,500 t/d and a final grind of about 90 µm can be achieved if all the mills available are used, and assuming that one of the ball mills can be successfully converted into a rod mill, which must be confirmed. The effect of the addition of a coarse classification and a coarse rougher flotation stage on the regrind circuit capacity was negligible. A total power requirement of ~3,170 kW was calculated for the 6 mills.

Simulations with three tower mills were also completed and a final grind of 53 µm with a power input of 3,970 kW would be required, in that particular case.

If this option is selected, SGS recommends operating two grinding lines in parallel, i.e. two rod mills and two ball mills, prior the regrind circuit. In order to do that, LDI must confirm with the mill supplier if one of the 9.0' x 12.0' mill can be modified and used as a rod mill.

A summary of the grinding circuit evaluated is presented in Table 1. For the pebble milling circuits, only the simulations with a pebble split of 50/50 between the pebble crusher and the pebble mills are shown. For the conventional circuits, the simulations representing the base case scenario (F_{80} of 22 mm, 3,500 t/d and two tower mills) is the only one presented.

Table 1: Simulations Summary

Primary Mill Circuit														Ball/Pebble Mill			Regrind Pebble Mill/Tower mill Circuit							Overall		
JKFile Name	Circuit Type	Feed Rate at 97% Availability		F_{80}	Pebble Crush.	Ball Charge	Grate Size	Pebble Media Split (%)			T_{80}	Gross Power Requirement		Gross Power Requirement		P_{80}	F_{80}	Feed Rate	Nb.	Gross Power Requirement			P_{80}	Total Gross Power	Gross Power Req.	Wio
File		t/h	t/d	mm	(Y/N)	% Vol	mm	to PC[1]	to PM[2]	to RM[3]	µm	kW	kWh/t	kW	kWh/t[4]	µm	µm	t/h		kW	kWh/t[5]	kWh/t[4]	µm	kW	kWh/t[4]	kWh/t
SIM-01	FAB	148	3,445	100	No	0	75	-	-	-	514	4,245	28.7	1,225	8.3	110	-	-		-	-	-	-	5,470	37.0	38.1
SIM-02	FAB	249	5,785	150	No	0	75	-	-	-	406	4,245	17.1	1,911	7.7	110	-	-		-	-	-	-	6,156	24.8	25.4
SIM-04	FAPC	322	7,501	150	Yes	0	75	50	50	0	447	4,245	13.2	2,818	8.7	110	-	-		-	-	-	-	7,063	21.9	22.4
SIM-07	FAPC	338	7,869	150	Yes	0	102	50	50	N/A	453	4,245	12.6	3,067	9.1	110	-	-		-	-	-	-	7,312	21.6	22.2
SIM-10	FAPC	338	7,869	150	Yes	0	102	50	28	22	453	4,245	12.6	2,844	8.4	110	160	128	1	2,351	18.3	7.0	53	9,439	27.9	19.7
SIM-12	SABC	322	7,501	100	Yes	5	75	50	30	20	665	5,105	15.8	2,839	8.8	110	160	122	1	2,195	17.9	6.8	53	10,139	31.5	22.3
SIM-14	FAC	230	5,354	150	Yes	0	102	50	N/A	50	110	4,245	18.5	-	-	-	160	87	1	1,973	22.6	8.6	53	6,218	27.0	19.1
SIM-15	Conv. - 2 lines	150	3,501	22	No	-	-	-	-	-	1,449	865	5.7	723	4.8	509	509	150	2	1,585	10.5	10.5	90	3,173	21.1	20.3
SIM-20*	Conv. - 2 lines	150	3,501	22	No	-	-	-	-	-	1,449	865	5.7	723	4.8	509	509	148	2	1,585	10.7	10.5	88	3,173	21.1	20.0

*Coarse flotation circuit added prior the tower mills circuit

[1] Fraction of the +1" pebbles reporting to the pebble crusher

[2] Fraction of the +1" pebbles reporting to the primary pebble mill

[3] Fraction of the +1" pebbles reporting to the regrind pebble mill

[4] kWh/t of primary mill feed

[5] kWh/t of regrind mill feed

Introduction

Mr. Erin Legault-Seguin of North American Palladium Ltd. requested that SGS Minerals Services (SGS) perform a grinding circuit evaluation for the Lac des Iles plant, located in Ontario. The objective of the program was to evaluate the amenability of the existing grinding equipment to process 7,500 t/d in fully a autogenous mode.

The report, which presented the preliminary grinding circuit designs, was initially issued on May 12, 2009. Later on, LDI requested to evaluate additional grinding options and this revised version of the report presents the results.

Mr. Greg Menard and Kurt Boyko of Nordmin Engineering Ltd. were also consulted during the simulation process.

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Discussion

1. Grinding Circuit Design Criteria

The client requested that SGS conduct grinding circuit evaluations for the Lac des Iles mine. This was done using the JKSimMet software, along with Bond's third theory of comminution. The objective of the study was to investigate the possibility of using the existing mills at the mine site to process 7,500 t/d in fully autogenous mode (FAG) for the primary grinding stage, and in pebble milling mode for the secondary grinding stage. Several grinding circuit configurations were initially proposed by LDI. After review of all the proposed circuits, four circuits were selected as listed below:

1) FAB (FAG milling + ball milling)

2) FABC (FAG milling + ball milling + pebble crushing)

3) FAPC (FAG milling + pebble milling with pebble crushing)

4) FAP (FAG milling + pebble milling)

Each circuit was simulated with the objective of treating 7,500 t/d (322 t/h at 97% availability) from a feed F_{80} of 100 mm down to 80% passing 110 µm, followed by a regrind circuit to regrind the ore to a P_{80} of 53 µm. For the first two years of production, it is planned to operate the concentrator in "batches" of 7,500 t/d for six days every two weeks. Then, starting year 3, the concentrator would be operated non-stop at 7,500 t/d.

Later, additional scenarios were requested by LDI to be added to the report and they are listed below:

5) FAC (Single-stage FAG milling + pebble crushing)

6) Conventional RM/BM at 3500 t/d

7) Conventional RM/BM at 3500 t/d, plus coarse rougher tails regrind

For the FAC circuit (5), the objective is to grind the ore down to 110 µm with the single-stage FAG mill closed with a pebble crusher, and to regrind the ore to a P_{80} of 53 µm with a pebble mill. For the two 3,500 t/d options (6 and 7), the objective is to grind the ore to 53 µm using a conventional rod mill/ ball mill circuit (plus the regrind circuit), with the old mills available at the LDI site.

A single 30' x 14' (EGL) SAG mill and two 20' x 34' ball mills are among the equipment available at the LDI site and were used for the grinding simulations. The characteristics of the mills, as they are understood by SGS, are presented in Table 2.

Table 2: LDI Mill Characteristics

Mill Dimensions	**SAG Mill**	**Ball Mill**	**Overall**
Nominal Diameter (feet)	30.0	20.0	-
Inside Liner Diameter (feet)	29.2	19.2	-
Inside Liner Diameter (metres)	8.90	5.86	-
Effective Grinding Length (feet)	14.0	34.0	-
Effective Grinding Length (metres)	4.27	10.4	-
Cone Angle (°)	15	15	-
Grate size (inch)	3	-	
Number of Equipment	1	2	-
Critical Speed (RPM)	14.2	17.5	-
% of Critical Speed (%)	78	78	-
Installed Power (kW)	6,341	12,682	19,023
Installed Power (HP)	8,500	17,000	25,500

The old mills characteristics are presented in Table 3. One 10.5' x 14.0 rod mill, as well as three 9.0' x 12.0' ball mills and two Vertimills are still available at the LDI site. It is understood that a third tower mill could be available as well. Note that the internal dimensions of the rod and the ball mills were not provided and were estimated. Also, 1,000 HP motors are fitted to each rod and ball mill, but the mills will only capable of delivering a small fraction of that based on those assumptions.

Table 3: Old LDI Mill Characteristics

Mill Dimensions	**Rod Mill**	**Ball Mill**	**Tower Mill**	**Overall**
Nominal Diameter (feet)	10.5	9.0	-	-
Inside Liner Diameter (feet)	*10.0*	*8.5*	-	-
Inside Liner Diameter (metres)	*3.05*	*2.59*	-	-
Nominal Length (feet)	14.0	12.0	-	-
Inside Liner Length (feet)	*13.5*	*11.5*	-	
Inside Liner Length (metres)	*4.1*	*3.5*	-	-
Cone Angle (°)	*15*	*15*	-	-
Number of Equipment	1	3	2	-
Critical Speed (RPM)	24.2	26.3	-	-
% of Critical Speed (%)	65	77	-	-
Installed Power (kW)	746	746	933	4,849
Installed Power (HP)	1,000	1,000	1,250	6,500

Italicised values were estimated

Assuming a media charge of 40% mills, the maximum power available is only 500 and 360 kW, for the rod and ball mills respectively.

The grindability test results performed on the Breccia sample, reported on March of 2002 under A.R. MacPherson Consultants Ltd. (ARMC) project 9876, were used for the grinding circuit design. The grindability test results are summarised in Table 4.

Table 4: Breccia Grindability Test Results

Sample Name	Relative Density	JK Parameters A x b	JK Parameters t_a	BWI kWh/t
Breccia	3.06	29.8	0.24	19.2

A feed F_{80} of 100 mm was initially used for the LDI FAG simulations. A coarser feed, with an F_{80} of about 150 mm was also simulated. The complete size distributions were computed based on experience, and are presented in Figure 1.

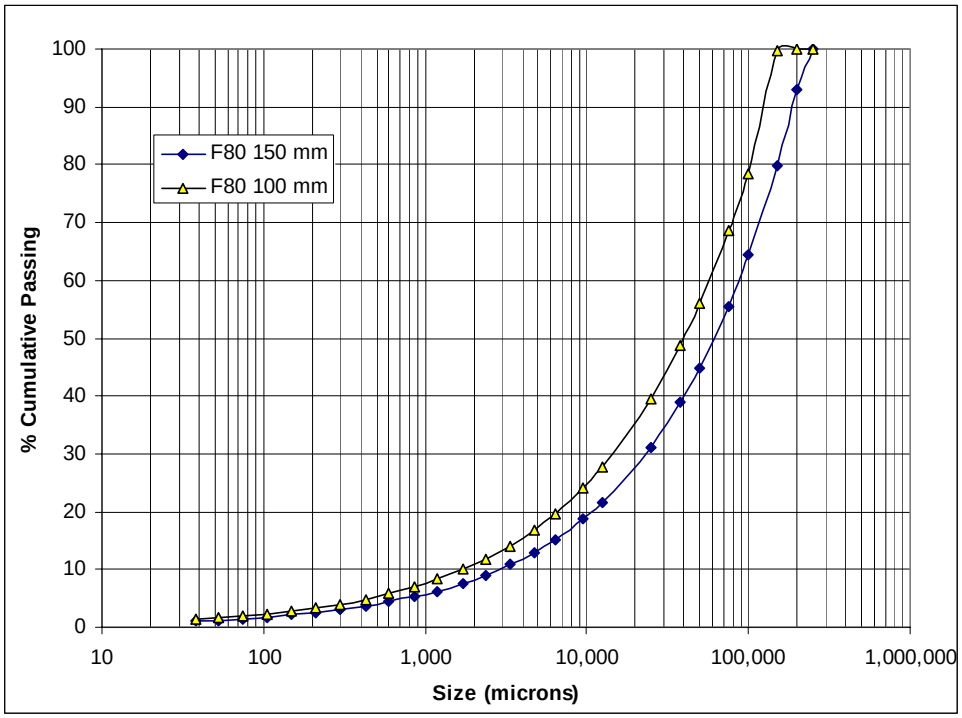

Figure 1: JKSimMet SAG Feed Size Particle Distributions

2. Grinding Circuit Simulations

The circuit simulations were performed using the JKSimMet software and Bond's third theory of comminution. The power requirement to the ball mill/pebble mill circuit was calculated using the Bond work index and the third theory of comminution, after correction for the fines in the SAG product with the phantom cyclone technique. The Bond efficiency factors were used, as appropriate. All the simulations were performed with a FAG mill charge level of 25%, which is the average in the JKSimMet model database, and is considered optimal for throughput. The default parameters of the JKSimMet model were used in the simulations. A classification screen of ¼" (6.4 mm) is currently installed at the LDI plant and was used in the grinding circuit simulations. The FAG mill grates were 3" (75 mm) and the FAG mill speed was set at 78% of critical speed. The results of the simulations are presented in Table 5 and Table 6.

Table 5: AG Mill Simulation Summary

AG Mill Circuit																													
JKFile Name	Circuit Type	Mill Size (Nom. dia x EGL)		Mill Size (inside liners)		Feed Rate at 97% Availability		Feed Size, mm		Pebble Crush.	Ball Charge	%Sol.	Mill Speed	Grate Size	Class. Slots Size	Recycle, t/h		Total +1" Pebble Media				Pebble Media Split (%)			Mill Charge	T_{80}	AG Gross Power Requirement		
File		Dia.	Lght	Meter		t/h	t/d	-12.7	F_{80}	(Y/N)	% Vol	w/w	% of Cr.	mm	mm	+1/4"	Cycl. U/F	%*	t/h	%	K_{80} mm	to PC[1]	to PM[2]	to RM[3]	% Vol	µm	kW	HP	kWh/t
SIM-01	FAB	30.0'	x 14.0'	8.9	x 4.27	148	3,445	28	100	No	0	67	78	75	6.4	89	-	60	-	-	-	-	-	-	25	514	4,245	5,690	28.7
SIM-02	FAB	30.0'	x 14.0'	8.9	x 4.27	249	5,785	22	150	No	0	67	78	75	6.4	83	-	33	-	-	-	-	-	-	25	406	4,245	5,690	17.1
SIM-03	FABC	30.0'	x 14.0'	8.9	x 4.27	320	7,450	22	150	Yes	0	67	78	75	6.4	62	-	28	28	9	51	100	N/A	N/A	25	458	4,245	5,690	13.3
SIM-04	FAPC	30.0'	x 14.0'	8.9	x 4.27	322	7,501	22	150	Yes	0	67	78	75	6.4	59	-	23	28	9	51	50	50	0	25	447	4,245	5,690	13.2
SIM-05	FAP	30.0'	x 14.0'	8.9	x 4.27	324	7,543	22	150	No	0	67	78	75	6.4	56	-	17	27	8	51	0	100	0	25	437	4,245	5,690	13.1
SIM-06	FABC	30.0'	x 14.0'	8.9	x 4.27	336	7,822	22	150	Yes	0	67	78	102	6.4	65	-	32	41	12	63	100	N/A	N/A	25	468	4,245	5,690	12.6
SIM-07	FAPC	30.0'	x 14.0'	8.9	x 4.27	338	7,869	22	150	Yes	0	67	78	102	6.4	61	-	24	40	12	63	50	50	N/A	25	453	4,245	5,690	12.6
SIM-08	FAP	30.0'	x 14.0'	8.9	x 4.27	340	7,915	22	150	No	0	67	78	102	6.4	57	-	17	39	11	63	0	100	N/A	25	438	4,245	5,690	12.5
SIM-09	FAPC	30.0'	x 14.0'	8.9	x 4.27	338	7,869	22	150	Yes	0	67	78	102	6.4	61	-	24	40	12	63	50	50	N/A	25	453	4,245	5,690	12.6
SIM-10	FAPC	30.0'	x 14.0'	8.9	x 4.27	338	7,869	22	150	Yes	0	67	78	102	6.4	61	-	24	40	12	63	50	28	22	25	453	4,245	5,690	12.6
SIM-11	FAP	30.0'	x 14.0'	8.9	x 4.27	340	7,915	22	150	No	0	67	78	102	6.4	57	-	17	39	11	63	0	54	46	25	438	4,245	5,690	12.5
SIM-12	SABC	30.0'	x 14.0'	8.9	x 4.27	322	7,501	28	100	Yes	5	67	78	75	6.4	75	-	29	35	11	50	50	30	20	25	665	5,105	6,843	15.8
SIM-13	SAB	30.0'	x 14.0'	8.9	x 4.27	322	7,501	28	100	No	5	67	78	75	6.4	70	-	22	34	11	50	0	58	42	25	639	5,105	6,843	15.8
SIM-14	FAC	30.0'	x 14.0'	8.9	x 4.27	230	5,354	22	150	Yes	0	67	78	102	6.4	42	282	148	33	14	47	50	N/A	50	25	110	4,245	5,690	18.5

*The total recycle, expressed in %, includes the fraction of the recycle reporting to the pebble crusher

[1] Fraction of the +1" pebbles reporting to the pebble crusher

[2] Fraction of the +1" pebbles reporting to the primary pebble mill

[3] Fraction of the +1" pebbles reporting to the regrind pebble mill

Table 6: Ball/Pebble Mill Simulation Summary

Ball/Pebble Mill Circuit										Regrind Pebble Mill Circuit									Overall		
JKFile Name	Circuit Type	F_{80}	Feed Rate	Gross Power Requirement			P_{80}		BWI	F_{80}	Feed Rate	Gross Power Requirement				P_{80}		BWI	Total Gross Power	Gross Power Req.	Wio
File		µm	t/h	kW	HP	kWh/t[4]	µm	RWI	kWh/t	µm	t/h	kW	HP	kWh/t[5]	kWh/t[4]	µm	RWI	kWh/t	kW	kWh/t[4]	kWh/t
SIM-01	FAB	514	148	1,225	1,642	8.3	110	19.2	19.2	-	-	-	-	-	-	-	-	-	5,470	37.0	38.1
SIM-02	FAB	406	249	1,911	2,562	7.7	110	19.2	19.2	-	-	-	-	-	-	-	-	-	6,156	24.8	25.4
SIM-03	FABC	458	320	2,589	3,470	8.1	110	19.2	19.2	-	-	-	-	-	-	-	-	-	6,834	21.4	21.9
SIM-04	FAPC	447	322	2,818	3,777	8.7	110	19.2	19.2	-	-	-	-	-	-	-	-	-	7,063	21.9	22.4
SIM-05	FAP	437	324	3,043	4,078	9.4	110	19.2	19.2	-	-	-	-	-	-	-	-	-	7,288	22.5	23.0
SIM-06	FABC	468	336	2,743	3,677	8.2	110	19.2	19.2	-	-	-	-	-	-	-	-	-	6,988	20.8	21.3
SIM-07	FAPC	453	338	3,067	4,111	9.1	110	19.2	19.2	-	-	-	-	-	-	-	-	-	7,312	21.6	22.2
SIM-08	FAP	438	340	3,375	4,524	9.9	110	19.2	19.2	-	-	-	-	-	-	-	-	-	7,620	22.4	23.0
SIM-09	FAPC	453	338	3,570	4,786	10.6	92	19.2	19.2	-	-	-	-	-	-	-	-	-	7,815	23.1	21.6
SIM-10	FAPC	453	338	2,844	3,812	8.4	110	19.2	19.2	160	128	2,351	3,151	18.3	7.0	53	19.2	19.2	9,439	27.9	19.7
SIM-11	FAP	438	340	2,920	3,915	8.6	110	19.2	19.2	160	129	2,683	3,596	20.8	7.9	53	19.2	19.2	9,848	29.0	20.4
SIM-12	SABC	665	322	2,839	3,805	8.8	110	19.2	19.2	160	122	2,195	2,942	17.9	6.8	53	19.2	19.2	10,139	31.5	22.3
SIM-13	SAB	639	322	3,151	4,224	9.8	110	19.2	19.2	160	122	2,456	3,293	20.1	7.6	53	19.2	19.2	10,713	33.2	23.5
SIM-14	FAC	-	-	-	-	-	-	19.2	-	160	87	1,973	2,645	22.6	8.6	53	19.2	19.2	6,218	27.0	19.1

[4] kWh/t of FAG mill feed

[5] kWh/t of regrind mill feed

2.1. FAB Circuit

The FAB circuit, as shown in Figure 2, consists of a fully autogenous mill (FAG) followed by a ball mill. The FAB circuit was simulated in SIM-01 and SIM-02. The first simulation (SIM-01) was performed with a feed F_{80} of 100 mm to the FAG mill, achieving only 3,450 t/d, which is much lower than the 7,500 t/d target. In SIM-02, a coarser feed with an F_{80} of 150 mm was simulated and an increase in throughput to 5,790 t/d was predicted.

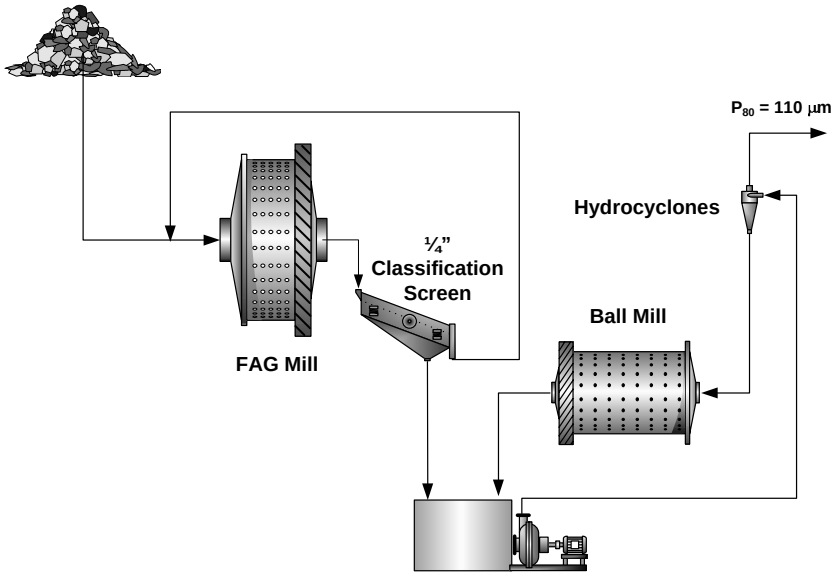

Figure 2: FAB Flowsheet

The FAB simulations showed that a coarse feed size will be required to supply grinding media for such a hard ore, and consequently, all the subsequent simulations were performed with an F_{80} of 150 mm.

2.2. FABC Circuit

The FABC circuit, which was simulated in SIM-03, is presented in Figure 3. In this configuration, the ¼" screen oversize is conveyed to a coarser screen, which was set at 1". The 1" screen undersize (-1"/+ ¼") is returned directly to the FAG mill, while the 1" screen oversize is redirected to the pebble crusher.

The addition of a pebble crusher to the FAB circuit (SIM-03) increased the throughput rate up to 7,450 t/d (SIM-03). The FAG power requirement was 4,245 kW or 13.3 kWh/t and the transfer size was about 460 µm. The ball mill power required to grind the ore down to 110 µm was about 2,600 kW (8.1 kWh/t). The overall power requirement was 21.4 kWh/t.

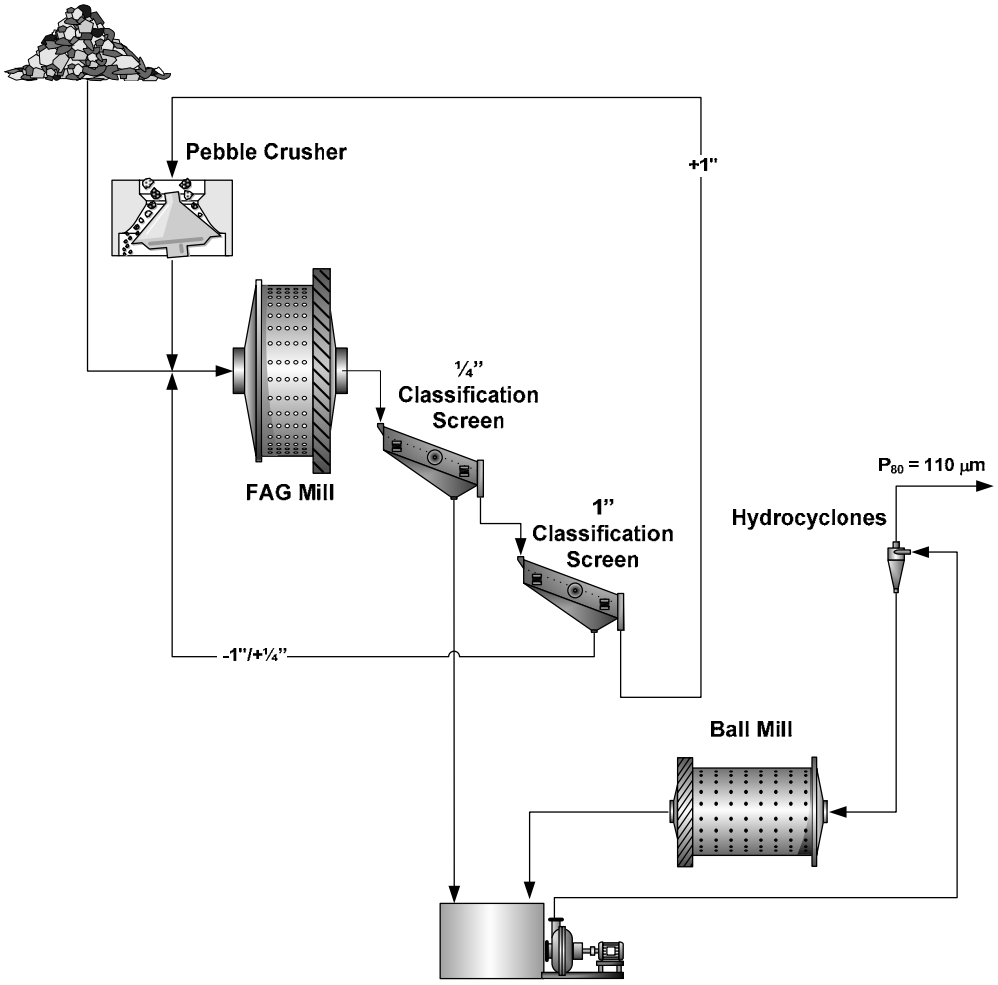

Figure 3: FABC Flowsheet

2.3. FAPC Circuit

The FAPC circuit is presented in Figure 4. For this circuit configuration, the pebbles (+1") are stored in a pebble bin. The required amount of pebbles for grinding media is split and fed to the pebble mill, while the pebble balance is redirected to the pebble crusher. Two scenarios were simulated for this circuit configuration, i.e. i) 50% of the pebbles are used as grinding media in the pebble mill, and; ii) 100% of the pebbles are required to the pebble mill. The second scenario corresponds to a "FAP" circuit, because the pebble crusher is not used.

Note that the theoretical power draw for a single pebble mill of 20' x 34' loaded with 40% of pebbles (assuming a pebble S.G. of 3.06 g/cm^3) and run at 78% of critical speed is about 3,570 kW.

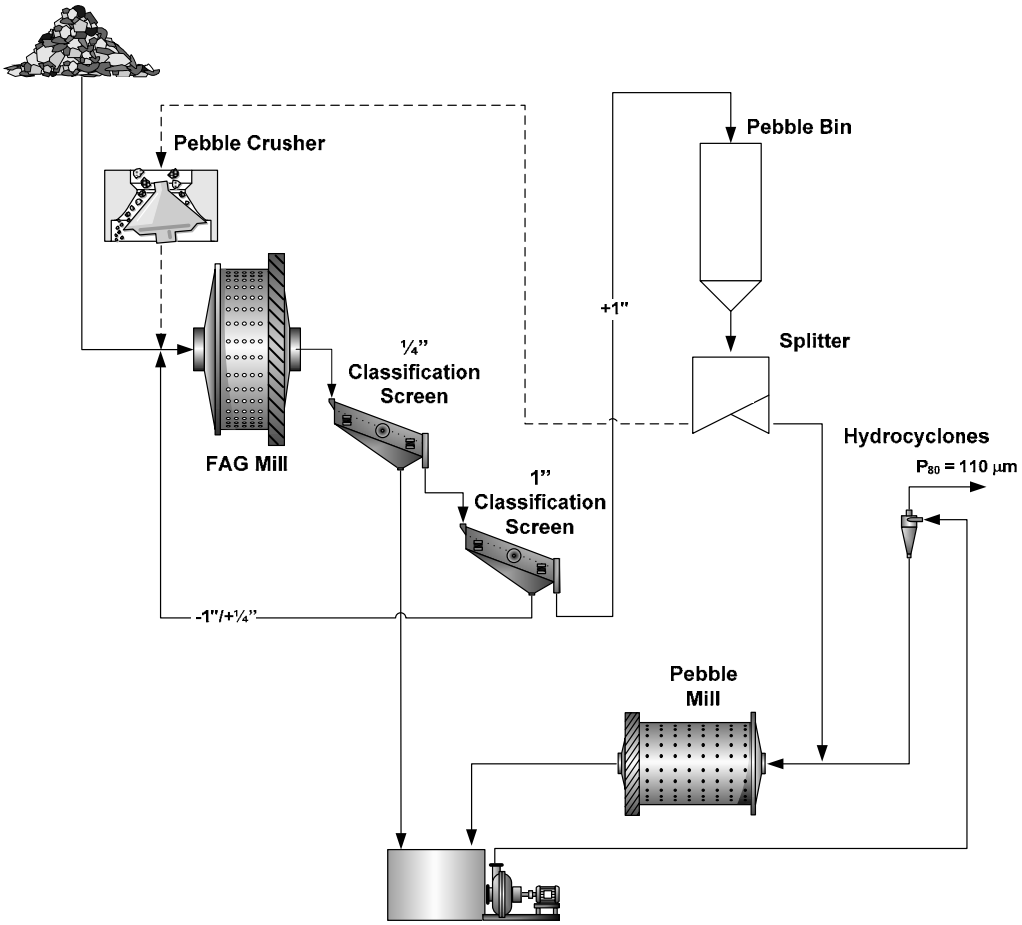

Figure 4: FAPC Flowsheet

SIM-04 simulated a pebble usage of 50% to the pebble mill. The FAG mill circuit achieved the target throughput of 7,500 t/d at a power requirement of 13.2 kWh/t. The pebble mill power requirement to grind the ore to 110 µm was 2,820 kW or 8.7 kWh/t, which is below the maximum calculated available pebble mill power of 3,570 kW. The pebble mill would have to be charged with ~31% of pebbles to achieve the

required power. The overall power requirement for that scenario was 21.9 kWh/t, which is slightly higher than the FABC (SIM-03) circuit. The slight power increase is due to the additional power required to grind the pebbles in the pebble mill to 110 µm.

In SIM-05, it was assumed that all the pebbles were used as grinding media (pebble usage of 100%) and the pebble crusher was not used. The FAG mill throughput (7,540 t/d) and the pebble mill power requirement (3,040 kW) slightly increased. The pebble mill power requirement corresponds to a pebble mill charge of about 34%. The overall power requirement was 22.5 kWh/t.

Because the simulated throughput was relatively close to the target, an additional series of simulations (SIM-06 to SIM-11) were performed using 4" grates to the FAG mill to assess sensitivity.

In SIM-06, all the pebbles were returned to the pebble crusher (FABC circuit). A throughput rate of 7,820 t/d was achieved and the FAG power requirement was slightly lower than for SIM-03 at 12.6 kWh/t. The ball mill and overall power requirements were 8.2 and 20.8 kWh/t, respectively.

For the 50% pebble usage scenario (SIM-07), the circuit throughput was slightly increased (7,870 t/d) and the pebble mill power requirement was 3,070 kW (~34% of pebble charge). The throughput rate could be increased up to 7,920 t/d for the 100% pebble usage (SIM-08), and the power requirement to the pebble mill to 3,380 kW (~38% of pebble charge). The overall power requirement for both simulations was 21.6 and 22.4 kWh/t, for SIM-07 and SIM-08, respectively.

2.4. Regrind Circuit

The regrind circuit, as part of the FAPC circuit, is presented in Figure 5. The primary pebble mill product is ground to 110 µm and is then classified to 53 µm, in order to separate the ground products into fine and coarse streams, which are floated separately. The flotation circuits are not detailed in Figure 5, for simplicity sake. The coarse rougher/scavenger tailing is cycloned to 53 µm, the overflow being considered as the final tails. The third cyclone underflow is then reground and classified at 53 µm in the second stage of cycloning. The grind target of the regrind mill is 80% passing 53 µm. The regrind calculations were based on the assumption that 38% of the feed reports to the third cyclone underflow, with a K_{80} of 160 µm.

LDI requested the investigation of an alternative scenario for the regrind circuit, where the third cyclone underflow would be sent directly to the primary pebble mill. For that particular case, a single pebble mill would be required. The flowsheet presented in Figure 5 represent the original scenario, with the additional regrind mill.

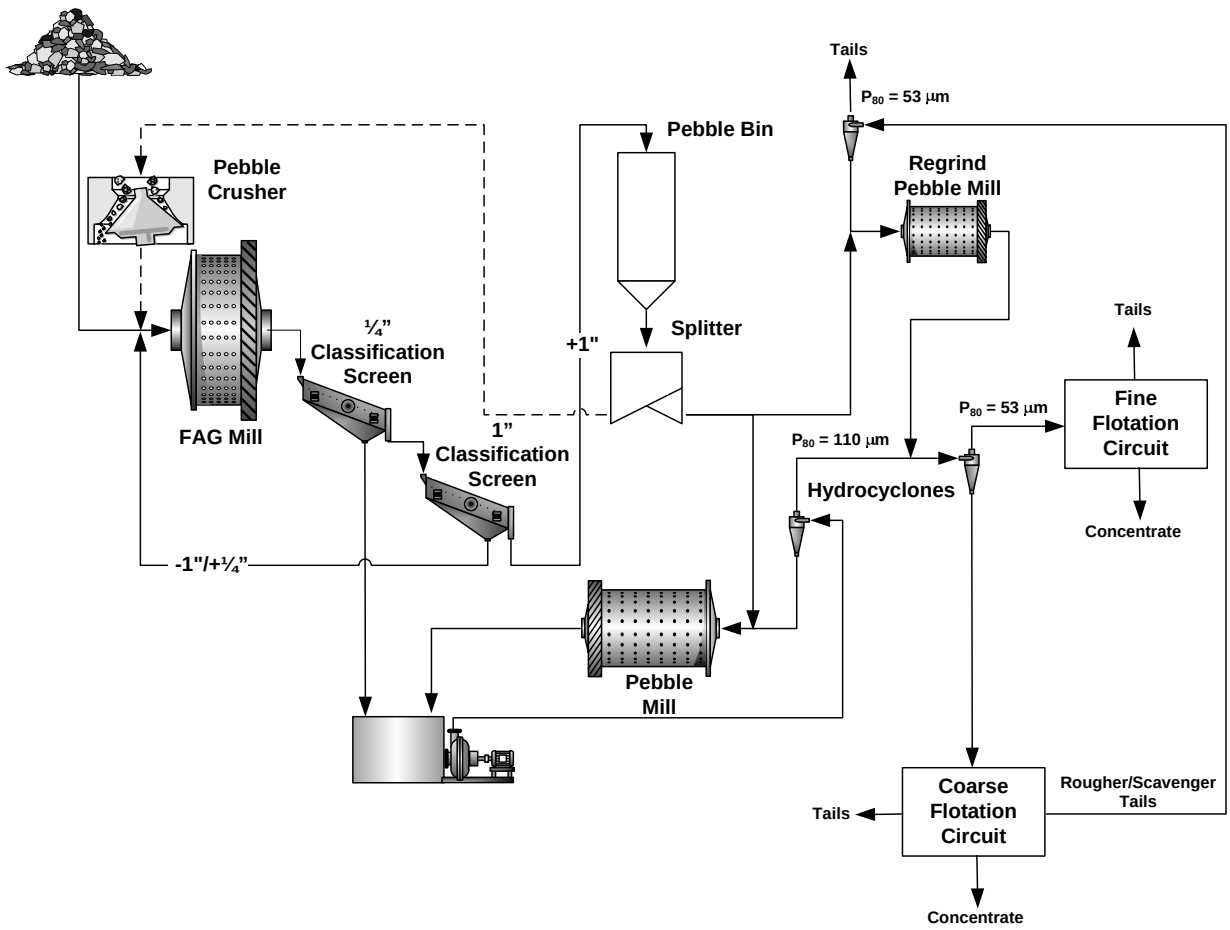

Figure 5: FAPC and Regrind Circuit Flowsheet

The alternative scenario was simulated in SIM-09, using the 4" grates and a 50% pebble usage, which corresponds to SIM-07. The results showed that when the maximum power to the pebble mill is used (3,570 kW), a final P_{80} of only 92 µm can be achieved, which is much coarser than the 53 micron target. As a result, a separate mill will have to be used as a regrind pebble mill. The second 20' x 34' ball mill can be used for this purpose.

The original scenario, as depicted in Figure 5, was simulated in SIM-10 and SIM-11, using an overall pebble usage of 50 and 100%, respectively. The grinding media was split between the primary and regrind pebble mills, proportionally to their power requirement. In SIM-10, power requirements of 2,840 and 2,350 kW were calculated for the primary and regrind pebble mills, respectively, which corresponded to mill loads of 32 and 26%, respectively. The overall power requirement was 27.9 kWh/t.

Even at a 100% pebble usage and a high throughput rate of 7,920 t/d (SIM-11), the power requirement of both the primary and regrind pebble mills is lower than the maximum available.

2.5. SAPC Circuit

All the previous simulations performed in FAG mode were performed assuming that the underground mine could supply a coarse feed size distribution (150 mm of F_{80}) to the grinding circuit. If the mine cannot supply a feed that is coarse enough to generate grinding media in the fully-autogenous mill, steel balls would have to be added to the mill. This scenario was simulated in SIM-12 and SIM-13, assuming that a feed F_{80} of 100 mm is achieved. For this set of simulations, a grate size of 75 mm (3") was simulated. The media usage to the pebble mills was assumed to be 50 and 100% for the SIM-12 and SIM-13, respectively.

The simulations showed that a ball charge of 5% to the SAG mill would be required to achieve 7,500 t/d. The SAG power requirement was 15.8 kWh/t, and the transfer size was coarser than from the FAG simulations, at about 650 µm. The overall power requirement, including the regrind mill, was 31.5 and 33.2 kWh/t, for the SIM-12 and SIM-13, respectively. The pebble mill power requirements from both the secondary and regrind mills were still lower than the maximum power available.

2.6. FAC Circuit

The FAC circuit is presented in Figure 6. As for the FAPC circuit configuration, the pebbles (+1") are stored in a pebble bin. The required amount of pebbles for grinding media is withdrawn from the bin and fed to the regrind pebble mill (which is not shown on the figure), while the pebble balance is sent to the pebble crusher. A single scenario was simulated for this circuit configuration, i.e. 50% of the pebbles are used as grinding media in the regrind pebble mill. A feed F_{80} of 150 mm and SAG grates of 4" were used in the FAC simulation.

The single-stage FAC mill circuit achieved a throughput of 5,350 t/d at a power requirement of 18.5 kWh/t (SIM-14). The total recycle rate was about 150%. The overall power requirement for that scenario was 27.0 kWh/t, including the regrind mill.

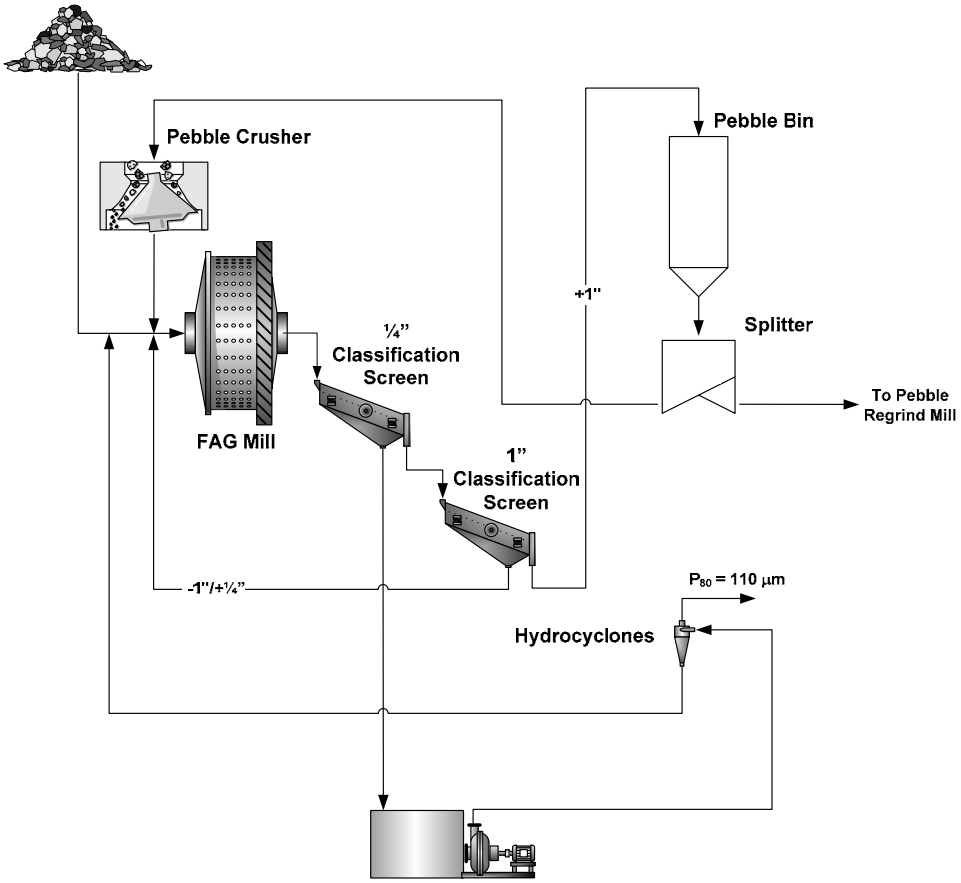

Figure 6: FAC Flowsheet

2.7. Conventional RM/BM Circuit Configuration

LDI requested to investigate the possibility of running the plant at 3,500 t/d using the old grinding equipment, consisting of 1 rod mill, 3 ball mills and up to 2 tower mills. The grinding circuit, as provided by LDI, would consist of 2 rod mill/ball mill grinding lines operating in parallel, followed by the regrind circuit. This circuit configuration would only work if one of the ball mills can be modified and used efficiently as a rod mill, which has not been confirmed yet (minimum generally recommended L:D ratio for rod mills is 1.4). No inefficiency factor was applied to the reconditioned 9.0' x 12.0' ball mill, when operated as a rod mill. An alternative circuit, which would include a single rod mill, followed by 3 ball mills operating in parallel and the regrind circuit was also simulated. The two circuit configurations are presented in Figure 7 and Figure 8, respectively.

The simulations were performed with a feed F_{80} of 22 mm to the rod mill(s), which is much coarser than the optimal size of 13 mm, and will, therefore, result in significant inefficiencies. The feed size was provided by Nordmin. The feed rate was split between the two rod mills, based on the ratio of their installed power. The calculated maximum power available for the rod mill (503 kW) and ball mills (361 kW) were used in the simulations. Each vertimill was assumed to draw 85% of the installed power. The

Bond third theory of comminution with appropriate correction factors were used for the calculation of the final grind. The Bond equations were also used to calculate the final grind produced by the tower mills, assuming that the tower mills are 20% more efficient than standard ball milling. This assumption should be checked with the mill supplier.

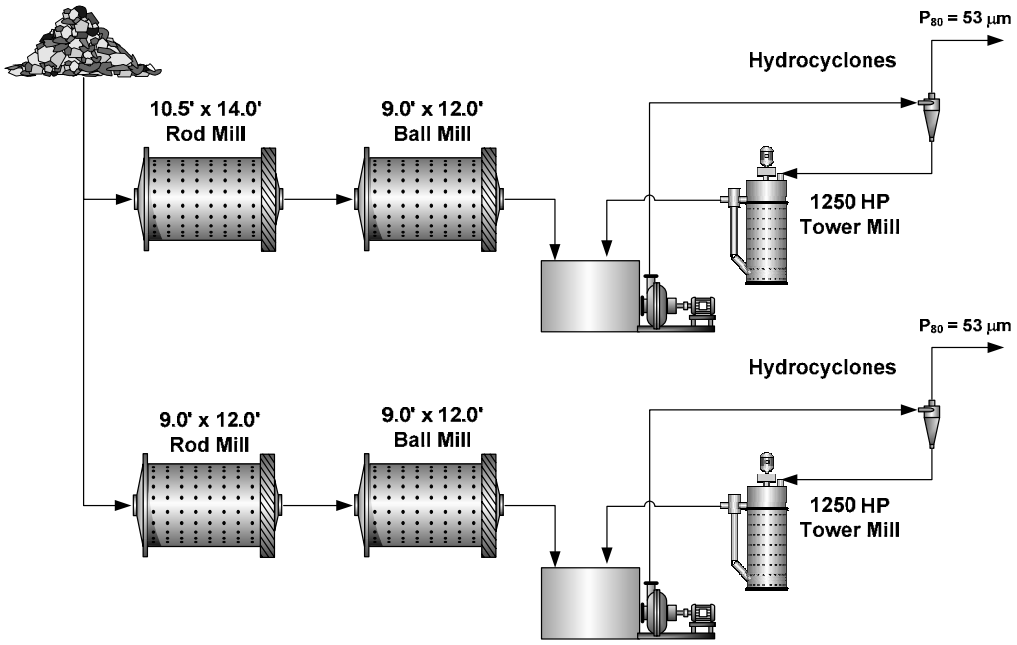

Figure 7: Conventional RM/BM Circuit – 2 Grinding lines

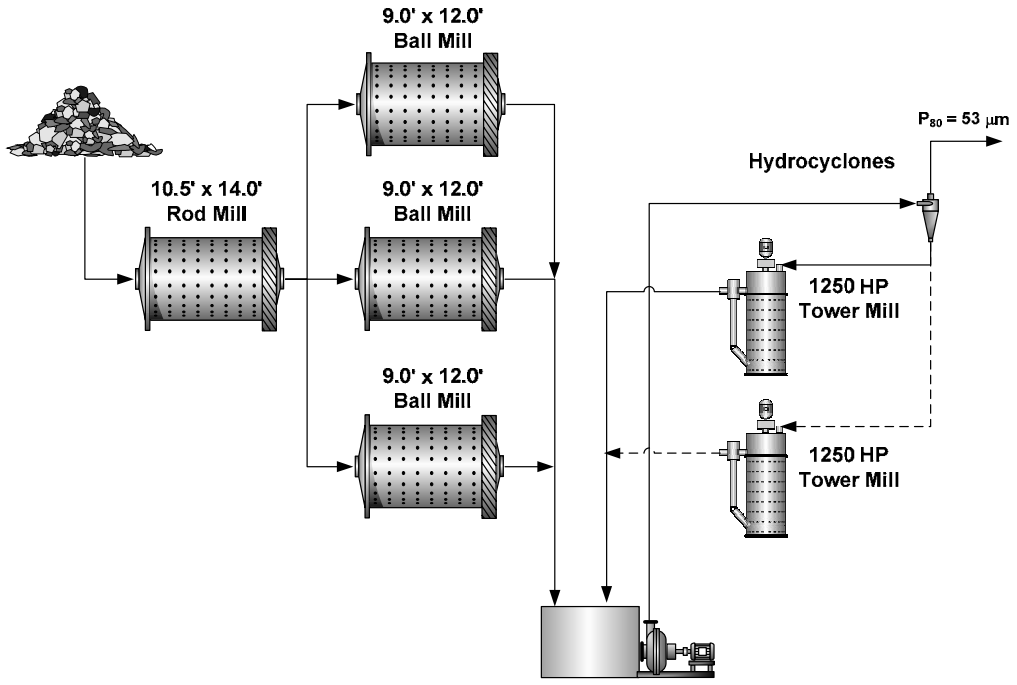

Figure 8: Conventional RM/BM Circuit - Alternative Configuration

The simulation results are presented in Table 7. The results from the parallel grinding lines are presented separately in the first two lines and the overall results are presented in the third line.

In SIM-15, two parallel grinding lines were simulated at a throughput rate of 3,500 t/d (150 t/h). The final grind achieved was 90 µm at a total power input of about 3,173 kW or 21.1 kWh/t.

In SIM-16, the final grind was fixed at 53 µm and the throughput rate to the grinding circuit was calculated. In order to achieve the target grind, a maximum of about 2,540 t/d (109 t/h) would be limited through the two parallel grinding lines. The specific power input would be 29.1 kWh/t.

A finer feed F_{80} of 12.7 mm was also simulated in SIM-17 and SIM-18. At 3,500 t/d (SIM-17), the final grind was 78 µm. The maximum throughput to achieve 53 µm was calculated in SIM-18 to be ~2,720 t/d (117 t/h). The effect of the finer feed is therefore significant.

A third tower mill was added to the circuit in SIM-19. The simulation was performed at 3,500 t/d with a F_{80} of 22 mm. A P_{80} of 53 µm was achieved and the total power input was about 3,965 kW (26.4 kWh/t). Note that the operating work index (Wio) of SIM-19 was 2 kWh/t lower than the Wio of SIM-16, even though the two simulations were performed at the same final grind (53 microns). This is due to the assumption that the tower mills are more efficient than the ball mills. In SIM-16, the throughput required to achieve 53 µm with 2 tower mills was lower than in SIM-19, which was performed with 3 tower mills. Consequently, the transfer size achieved in SIM-16 was finer than in SIM-19 (321 µm compared to 509 µm in SIM-19). The tower mill efficiency factor was therefore applied from 321 to 53 µm in SIM-16, rather than from 509 to 53 µm in SIM-19.

The alternative flowsheet was simulated, but the results are not presented in Table 7. The calculations indicated that the throughput rate of 150 t/h is too high for the 10.5’ x 14.0‘ rod mill. This alternative circuit configuration is therefore not recommended.

Table 7: Conventional RM/BM Circuit Simulations Results

		Primary Mill Circuit								Secondary Ball Mill Circuit								Tower Mill				Overall Circuit		
SIM	Circuit Type	Primary Mill	Nb. Mill	F_{80}	Feed Rate at 97% Availability		Gross Power Requirement			K_{80}	Sec. Mill	Nb. Mill	Gross Power Requirement			K_{80}	BWI	Nb. Mill	Gross Power Requirement			P_{80}	Total Gross Power		Wio
				µm	t/h	t/d	kW	HP	kWh/t	µm			kW	HP	kWh/t	µm	kWh/t		kW	HP	kWh/t	µm	kW	kWh/t	kWh/t
SIM-15	Conv. - 2 lines	10.5' x 14'	1	22,000	88	2,038	503	675	5.7	1,415	9' x 12'	1	361	484	4.1	567	19.2	1	793	1,063	9.1	108	1,657	18.9	20.1
SIM-15	Conv. - 2 lines	9' x 12'	1	22,000	63	1,463	361	484	5.7	1,495	9' x 12'	1	361	484	5.7	427	19.2	1	793	1,063	12.6	64	1,515	24.1	19.3
SIM-15	Conv. - 2 lines	Total	2	22,000	150	3,501	865	1,159	5.7	1,449	-	2	723	969	4.8	509	-	2	1,585	2,125	10.5	90	3,173	21.1	20.3
SIM-16	Conv. - 2 lines	10.5' x 14'	1	22,000	63	1,477	503	675	7.9	1,066	9' x 12'	1	361	484	5.7	363	19.2	1	793	1,063	12.5	63	1,657	26.1	20.7
SIM-16	Conv. - 2 lines	9' x 12'	1	22,000	46	1,060	361	484	7.9	1,090	9' x 12'	1	361	484	7.9	262	19.2	1	793	1,063	17.4	39	1,515	33.3	20.7
SIM-16	Conv. - 2 lines	Total	2	22,000	109	2,538	865	1,159	7.9	1,076	-	2	723	969	6.6	321	-	2	1,585	2,125	14.5	53	3,173	29.1	21.1
SIM-17	Conv. - 2 lines	10.5' x 14'	1	12,700	88	2,038	503	675	5.7	724	9' x 12'	1	361	484	4.1	385	19.2	1	793	1,063	9.1	92	1,657	18.9	18.8
SIM-17	Conv. - 2 lines	9' x 12'	1	12,700	63	1,463	361	484	5.7	746	9' x 12'	1	361	484	5.7	296	19.2	1	793	1,063	12.6	58	1,515	24.1	18.7
SIM-17	Conv. - 2 lines	Total	2	12,700	150	3,501	865	1,159	5.7	733	-	2	723	969	4.8	348	-	2	1,585	2,125	10.5	78	3,173	21.1	19.1
SIM-18	Conv. - 2 lines	10.5' x 14'	1	12,700	68	1,586	503	675	7.4	619	9' x 12'	1	361	484	5.3	285	19.2	1	793	1,063	11.6	62	1,657	24.3	19.5
SIM-18	Conv. - 2 lines	9' x 12'	1	12,700	49	1,138	361	484	7.4	631	9' x 12'	1	361	484	7.4	215	19.2	1	793	1,063	16.2	41	1,515	31.0	19.9
SIM-18	Conv. - 2 lines	Total	2	12,700	117	2,724	865	1,159	7.4	624	-	2	723	969	6.2	256	-	2	1,585	2,125	13.5	53	3,173	27.1	20.0
SIM-19	Conv. - 2 lines	10.5' x 14'	1	22,000	88	2,038	503	675	5.7	1,415	9' x 12'	1	361	484	4.1	567	19.2	-	-	-	-	-	-	-	-
SIM-19	Conv. - 2 lines	9' x 12'	1	22,000	63	1,463	361	484	5.7	1,495	9' x 12'	1	361	484	5.7	427	19.2	-	-	-	-	-	-	-	-
SIM-19	Conv. - 2 lines	Total	2	22,000	150	3,501	865	1,159	5.7	1,449	-	2	723	969	4.8	509	-	3	2,378	3,188	15.8	53	3,965	26.4	19.1

2.8. Conventional RM/BM Circuit Configuration – Coarse Rougher Flotation Circuit

For this grinding circuit scenario, an additional classification was added to the previous circuit configuration, as shown in Figure 9. The ball mill discharges would be classified to 150 µm, and the cyclone overflow would be fed to the rougher flotation circuit. The rougher tailings would then be redirected to the regrind circuit, along with the primary cyclone underflow. The mass pull to the coarse rougher flotation concentrate is expected to be very low, i.e. about 1.7%.

Note that the flotation circuit shown in Figure 9 was simplified.

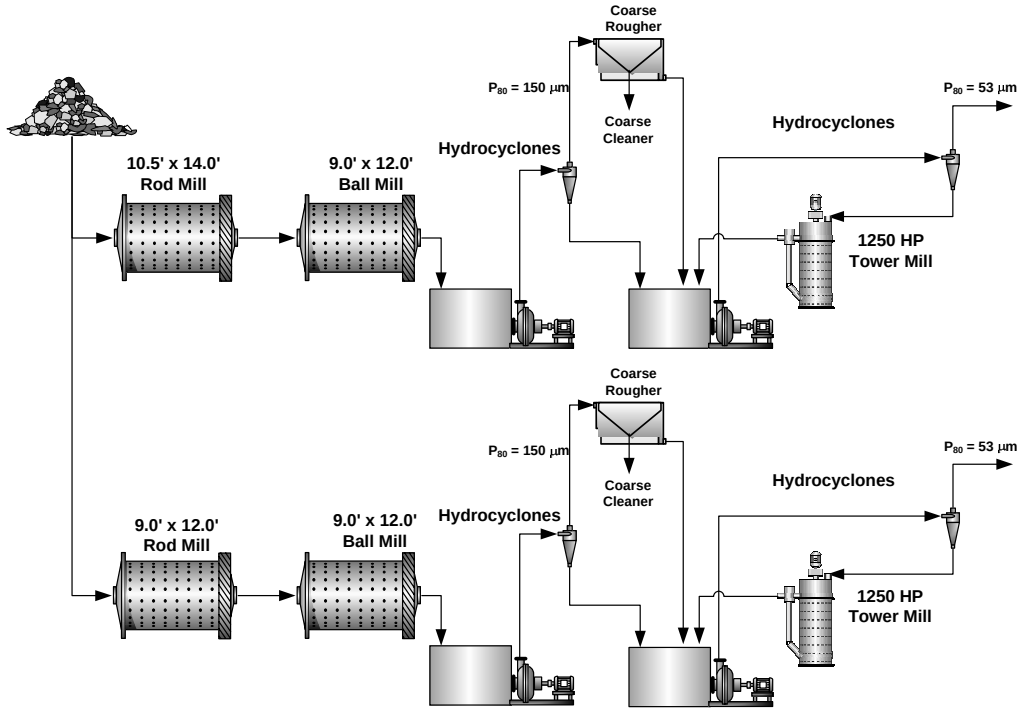

Figure 9: Conventional RM/BM Circuit – Coarse Rougher Flotation Circuit

The simulation results are presented in Table 8. The results from the primary rod mills and secondary ball mills were identical to those presented in Section 2.7, except for the simulations at fixed grind (SIM-21 and SIM-23). SIM-20, and SIM-22 were performed at 3,500 t/d with feed F_{80}'s of 22 and 12.7 mm, respectively. SIM-21 and SIM-23 were performed with a fixed final grind (53 µm of P_{80}) and the feed rate to the grinding circuit was allowed to vary. Final grinds of 88 and 76 µm were predicted in SIM-20 and SIM-22, while approximate throughput rates of 2,610 and 2770 t/d were obtained in SIM-21 and SIM-23. With the third tower (SIM-24), a final grind of 52 µm was achieved.

The alternative circuit configuration, i.e. 1 primary rod mill plus 3 secondary ball mills operating in parallel was not simulated for this circuit configuration.

Table 8: Conventional RM/BM Circuit Simulations Results - Coarse Rougher Flotation Circuit

		Primary Mill Circuit								Secondary Ball Mill Circuit								Tower Mill							Overall Circuit	
SIM	Circuit Type	Primary Mill	Nb. Mill	F_{80}	Feed Rate at 97% Availability		Gross Power Requirement			K_{80}	Sec. Mill	Nb. Mill	Gross Power Requirement			K_{80}	BWI	Feed Rate	Nb. Mill	Gross Power Requirement				P_{80}	Total Gross Power		Wio
				µm	t/h	t/d	kW	HP	kWh/t	µm			kW	HP	kWh/t	µm	kWh/t	t/h		kW	HP	kWh/t[1]	kWh/t[2]	µm	kW	kWh/t[2]	kWh/t
SIM-20	Conv. - 2 lines	10.5' x 14'	1	22,000	88	2,038	503	675	5.7	1,415	9' x 12'	1	361	484	4.1	567	19.2	86	1	793	1,063	9.2	9.1	106	1,657	18.9	19.9
SIM-20	Conv. - 2 lines	9' x 12'	1	22,000	63	1,463	361	484	5.7	1,495	9' x 12'	1	361	484	5.7	427	19.2	62	1	793	1,063	12.8	12.6	63	1,515	24.1	19.1
SIM-20	Conv. - 2 lines	Total	2	22,000	150	3,501	865	1,159	5.7	1,449	-	2	723	969	4.8	509	-	148	2	1,585	2,125	10.7	10.5	88	3,173	21.1	20.0
SIM-21	Conv. - 2 lines	10.5' x 14'	1	22,000	65	1,518	503	675	7.7	1,084	9' x 12'	1	361	484	5.5	375	19.2	64	1	793	1,063	12.4	12.2	62	1,657	25.4	20.1
SIM-21	Conv. - 2 lines	9' x 12'	1	22,000	47	1,089	361	484	7.7	1,110	9' x 12'	1	361	484	7.7	272	19.2	46	1	793	1,063	17.2	16.9	40	1,515	32.4	20.4
SIM-21	Conv. - 2 lines	Total	2	22,000	112	2,607	865	1,159	7.7	1,095	-	2	723	969	6.5	332	-	110	2	1,585	2,125	14.4	14.2	53	3,173	28.3	20.6
SIM-22	Conv. - 2 lines	10.5' x 14'	1	12,700	88	2,038	503	675	5.7	724	9' x 12'	1	361	484	4.1	386	19.2	86	1	793	1,063	9.2	9.1	90	1,657	18.9	18.6
SIM-22	Conv. - 2 lines	9' x 12'	1	12,700	63	1,463	361	484	5.7	746	9' x 12'	1	361	484	5.7	296	19.2	62	1	793	1,063	12.8	12.6	57	1,515	24.1	18.5
SIM-22	Conv. - 2 lines	Total	2	12,700	150	3,501	865	1,159	5.7	733	-	2	723	969	4.8	348	-	148	2	1,585	2,125	10.7	10.5	76	3,173	21.1	19.0
SIM-23	Conv. - 2 lines	10.5' x 14'	1	12,700	69	1,613	503	675	7.3	625	9' x 12'	1	361	484	5.2	291	19.2	68	1	793	1,063	11.6	11.4	62	1,657	23.9	19.3
SIM-23	Conv. - 2 lines	9' x 12'	1	12,700	50	1,158	361	484	7.3	637	9' x 12'	1	361	484	7.3	220	19.2	49	1	793	1,063	16.2	15.9	41	1,515	30.5	19.6
SIM-23	Conv. - 2 lines	Total	2	12,700	119	2,770	865	1,159	7.3	630	-	2	723	969	6.1	261	-	117	2	1,585	2,125	13.6	13.3	53	3,173	26.7	19.8
SIM-24	Conv. - 2 lines	10.5' x 14'	1	22,000	88	2,038	503	675	5.7	1,415	9' x 12'	1	361	484	4.1	567	19.2	-	-	-	-	-	-	-	-	-	-
SIM-24	Conv. - 2 lines	9' x 12'	1	22,000	63	1,463	361	484	5.7	1,495	9' x 12'	1	361	484	5.7	427	19.2	-	-	-	-	-	-	-	-	-	-
SIM-24	Conv. - 2 lines	Total	2	22,000	150	3,501	865	1,159	5.7	1,449	-	2	723	969	4.8	509	-	148	3	2,378	3,188	16.1	15.8	52	3,965	26.4	18.9

[1] kWh/t of tower mill feed

[2] kWh/t of primary mill feed

Conclusions and Recommendations

The JKSimMet simulations indicated that the 30' x 14' FAG mill can process 7,500 t/d in a fully autogenous mode, as long as coarse feed can be supplied to the FAG mill. A single 20' x 34' ball mill can be used as a pebble mill to grind the ore down to a P_{80} of 110 µm, but a second 20' x 34' ball mill will be required to regrind the coarse/rougher/scavenger stream down to 53 µm. The regrind mill can also be used as a pebble mill, in which case it may be preferable to classify the pebbles into coarse and fine, to be used in the primary and regrind pebble mills, respectively. A pebble crusher plant will be required to handle any excess of pebbles generated by the AG mill.

A pilot plant program is required to confirm power requirement and critical parameters, such as FAG mill pebble production. The pebble balance is unknown at this point, and some assumptions have to be made for the simulations. If the pebble mills consume more media than the FAG mill actually produce, steel balls would have to be added to one or both pebble mills. The pilot plant program could address additional issues such as the effect of feed size distribution, the effect of screen size aperture, and other operational concerns that LDI may have, but more importantly, it will confirm the power requirements that always carry a level of uncertainty for fully-autogenous circuits.

The old grinding mills available at the LDI site cannot also be used to grind the ore at a throughput rate of 3,500 t/d down to a P_{80} of 53 µm. At 3,500 t/d, the finest grind that can be achieved by the two parallel grinding lines would be about 90 µm. The calculated total power requirement was about 3,170 kW, or 21.1 kWh/t. Reducing the feed size from 22 to 12.7 mm would reduce the final grind to 78 µm.

It is understood that a third tower mill could possibly be used. The addition of a third tower mill would reduce the final grind to 53 µm. If the conventional circuit option is considered, LDI should check with the supplier of the 9.0' x 12.0 ball mills to determine if one of the mills can be modified and efficiently operated as a rod mill, despite the low L:D ratio.

References

1) Grinding System Review for the Lac des Iles Mine Based on Previous Reports and Data Sheets, A.R. MacPherson Consultants Ltd., ARM-0206, LR10044-039, July 24, 2002

APPENDIX E

MDE GEOTECHNICAL REPORTS

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To: Ron Moran

From: Kathy Kalenchuk

cc: W. F. Bawden

RE: Geotechnical mapping in the Offset Zone.

Date: March 12, 2010

In response to request from Ron Moran, Chief Engineer, Lac Des Isles Mine, Dr. K.S. Kalenchuk of Mine Design Engineering visited Lac des Iles (LDI) Mine March 7 to 13, 2011. The focus of this site visit was to collect geotechnical data in the Offset Zone and to train engineering staff in scanline mapping practices. Mapping was completed in the footwall, ore and hangingwall of the Offset ore body on levels 4900, 4850, 4815 and 4790. This report summarizes the main observations; all joint data has been submitted with this report (Offset_mapping_March2011_MDENG.xls).

Mapping Data

Figure 1 shows a stereonet pole plot of all joint data. Five joint sets are evident in the Offset Zone, three are well developed with joint set cluster having pole concentrations exceeding 5% (A, B and C) and two weaker sets having pole concentrations exceeding 3% (D and E). The orientations of these sets are summarized in Table 1. For comparison, Figure 2 illustrates the distribution of joint data for the Roby Zone. Roby Zone joint sets are summarized in Table 2; four sets are apparent in the joint data and an additional set was identified by Kalenchuk during a site visit in October, 2010. The joint set distribution in the Offset Zone varies slightly from the Roby Zone. Both zones have sup-vertical joint sets dipping southeast/northwest (A, A_R), east (B, B_R) and southwest/northeast (C, C_R). Inclined features occur in both zones, Roby has an inclined joint set dipping northwest (D_R), with few randomly dipping inclined joints. The Offset Zone features an inclined joint set dipping roughly north (D) in addition to a considerable number of randomly oriented inclined features. Steep features dipping north occur in the Offset Zone (E); these are not as well developed as in the Roby Zone.

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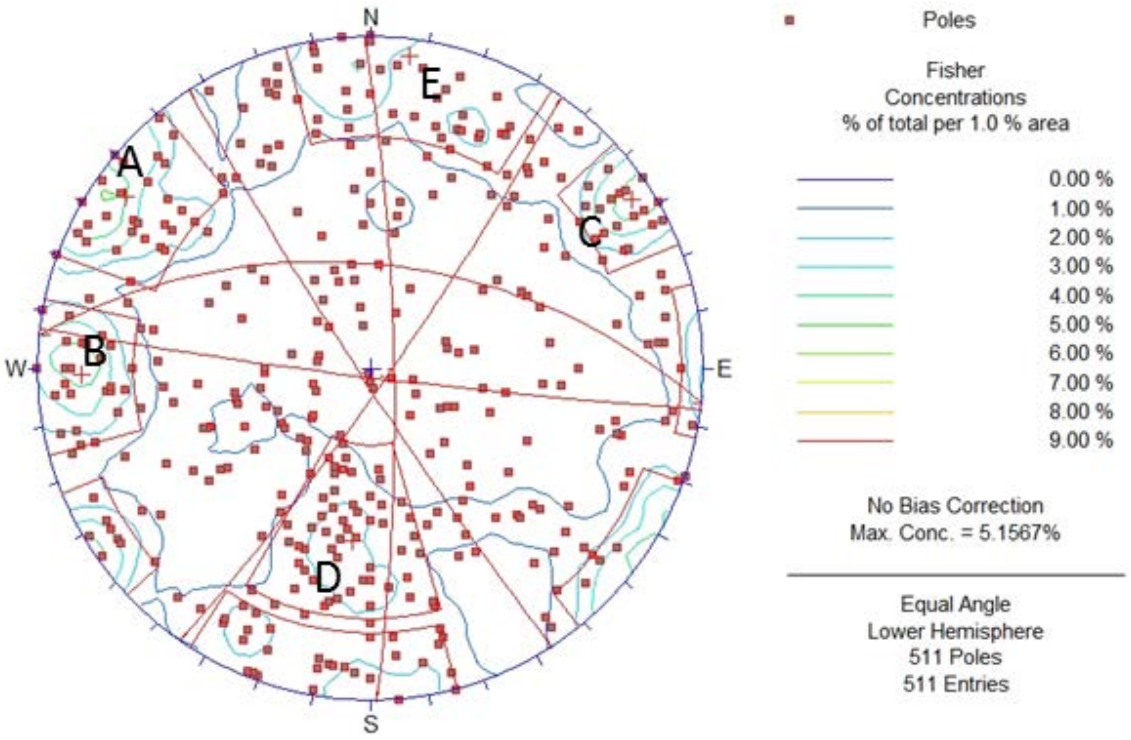

Figure 1: All joint data mapped in the Offset Zone with joint sets poles (labeled A through E) highlighted.

Table 1: Average dip and dip direction of joint sets identified in Offset Zone mapping data (DD = Dip Direction, SD = Standard Deviation, all values are in degrees).

Set ID	Dip	DD	SD
A	84	125	21
B	82	88	15
C	86	237	14
D	55	6	19
E	86	187	17

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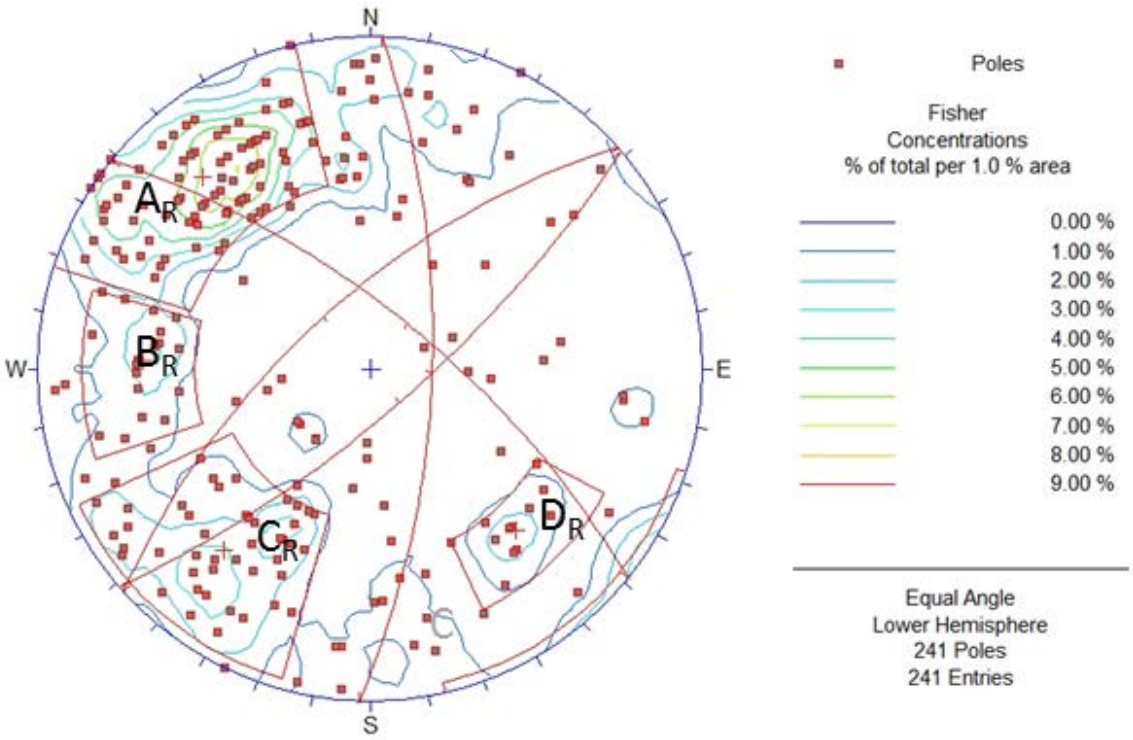

Figure 2: All joint data mapped in the Roby Zone with joint sets poles (labeled A_R through D_R) highlighted.

Table 2: Roby Zone joint sets (DD = Dip Direction, SD = Standard Deviation, all values are in degrees).

Set ID	Dip	DD	SD
A_R	74	140	19
B_R	71	94	17
C_R	73	43	10
D_R	67	304	12
E_R (Kalenchuk Observation)	30	110	-

The distribution of structural features varies spatially within the Offset Zone. Four domains have been identified and are referred to in this report as (1) footwall, (2) ore, (3) hangingwall, and (4) south. The approximate limits of these domains are illustrated in Figure 3. Figure 4 illustrates the joint structure distribution in each of these domains and Table 3 summarizes the orientation data. It should be noted rock type data is not available for the Roby Zone joint data so it is not possible to say if similar structural domains occur in the Roby as have been delineated for the Offset.

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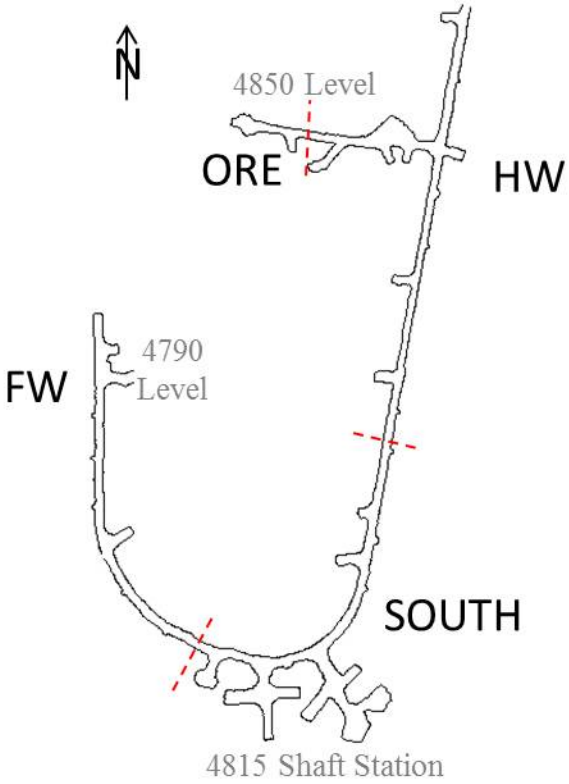

Figure 3: Illustration of approximate domain limits (denoted by dashed lines). The HW/South and Ore/HW boundaries are sharp and easily recognized. The FW/South boundary is gradual. The FW/Ore boundary was not observed.

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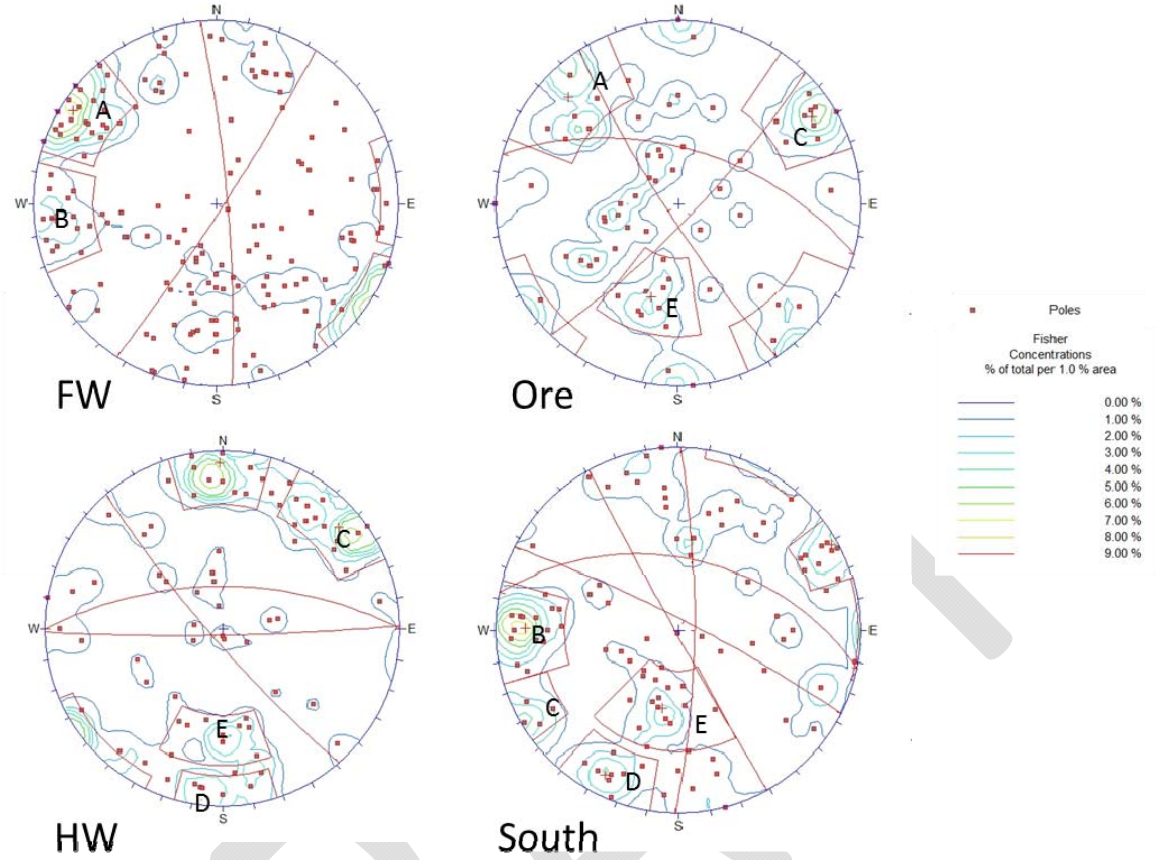

Figure 4: Pole plots for each of the four domains identified in the Offset zone.

Table 3: Offset Zone joint sets by domain (DD = Dip Direction, SD = Standard Deviation, all values are in degrees).

Set ID	A			B			C			D			E		
Domain	Dip	DD	SD	Dip	DD	SD	Dip	DD	SD	Dip	DD	SD	Dip	DD	SD
FW	85	122	11	83	85	12									
Ore	80	134	10				81	236	16				55	15	14
HW							82	229	12	86	178	15	62	0	13
South				80	91	10	88	241	10	82	27	9	47	12	17

Of interest in the Offset joint data is the common occurrence of inclined, highly persistent joints. Given that inclined features are least favorable to excavation stability the occurrence of these joints should be noted in new development. Where highly persistent inclined features are identified care must be taken to assess the potential for wedge formation and ensure that support design in adequate to maintain stability. There is a history of unanticipated inclined features causing stability issues in stopes and development at LDI; for example, failure of the 5035 sill pillar in Roby stope blocks 1,2 and 3 controlled by set E_R, and overbreak in the 4815 shaft station and the occurrence of

a large wedge in the adjacent 4815 washbay and storage area intersection controlled by set E.

1 Rockmass Quality

RQD (Rock Quality Designation) is a technique for estimating rockmass quality. It is commonly calculated as the percent of core runs made up of intact pieces exceeding 10 cm, however can be estimated from exposed rockwalls as illustrated in Figure 5. RQD has been estimated throughout the Offset Zone and is very good to excellent (Table 4).

The modified Rock Quality Index (Q') is a rockmass characterization parameter which takes into consideration RQD, number of joint sets and joint condition:

$$Q' = \frac{RQD}{J_n} \times \frac{J_r}{J_a}$$

Where: J_n = joint set number, J_r = joint roughness number and J_a = joint alteration number. Q' values are typically good in the Offset Zone. The South domain is fair (Q'=9), the ore (Q' = 28) and the hangingwall (Q'=20) are good and the footwall is good to very good (Q'=40).

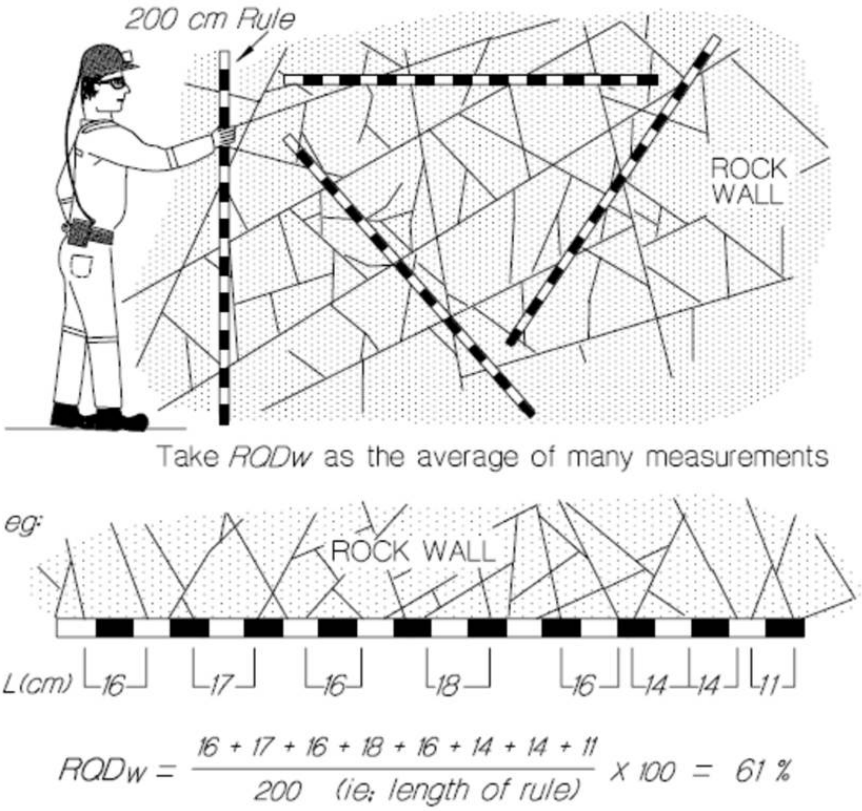

Figure 5: Estimation of RQD measurements from rockwall exposure.

Table 4: Offset Zone rockmass characterization parameters.

Domain	RQD	Jn	Jr	Ja	Q'
All	85-100	6 to 15	1.5 to 4	1	9 to 67 (avg.=29)
SOUTH	90-100	15	1.5	1	9 to 10 (avg. = 9.3)
FW	90-100	6	1.5 to 4	1	24 to 67 (avg.=40)
HW	90-100	12	2 to 3	1	17 to 23 (avg. = 20)
ORE	85-100	12	3 to 4	1	21 to 33 (avg. = 28)

The South domain of the Offset Zone will host critical mine infrastructure including the shaft and the implications of this fair quality rockmass on infrastructure design must be taken into consideration. Given the block nature of this domain and the occurrence of very persistent and inclined joint features, overbreak will be an issue during infrastructure development. Where empirical methods are utilized for support design the minimum standard for primary and secondary elements must be suitable to these least favorable ground conditions. Further, kinematic analysis should be completed to ensure that ground support will stabilize maximum potential wedges that may form by intersecting structural features.

2 Summary and Recommendations

The Offset Zone rockmass varies between four domains:

(1) Footwall: Good to very good quality rockmass with two joints sets plus random jointing. The rockmass can be blocky in areas where clusters of closely spaced sub-vertical joints occur.

(2) Ore: Good quality rockmass with three joints sets plus random. Joints are commonly poorly developed and discontinuous with limited trace length however high persistence features do occur and these are commonly inclined (unfavorably orientated).

(3) Hangingwall: This good quality rockmass is fairly massive. Three joint sets plus random have been identified, however joints are poorly developed and commonly discontinuous with very limited persistence.

(4) South: A fair quality rockmass with four joint sets plus random. Jointing in this domain is very well developed contributing to a very block rockmass. The occurrence of very persistent and inclined joint features in the South domain will contribute to challenges with overbreak during the development of critical mine infrastructure.

It is recommended that mine-wide support standards are at a minimum suitable to the least favorable ground conditions, which are observed in the South domain of the Offset Zone. Kinematic analysis should be completed to ensure that ground support will stabilize maximum potential wedges that may form by intersecting structural features.

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It is recommended that geotechnical mapping be continued with all new development in the Offset Zone. The establishment of a geomechanical database will allow for continued characterization of the Offset rockmass over life of mine. This will assist in early recognition of any changes to ground conditions and will be important to the calibration of models for predicting anticipated rockmass response mining.

_____

K.S. Kalenchuk, Ph.D.

_____

Review by W.F. Bawden, Ph.D., P.Eng.

DRAFT

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Ron Moran,
North American Palladium
Lac des Isles Mine
P.O. Box 10547 Station P
Thunder Bay, Ontario, P7B 6T9

November 15, 2010

Dear Ron,

RE: Borehole Camera Survey #2 5155 sill November 13, 2010

In response to your request, Mr. Peter Lausch and myself have visited Lac des Isles Mine November 14, 2010. This site visit was to survey boreholes in the #2 sill on 5155 following reports from drill crews of a significant parting approximately 2-3 m into the back and challenges with borehole cross-contamination during cable bolt grouting.

Seven boreholes were inspected by borehole camera spanning the sill back from west [footwall] to east [hangingwall] at the south end and the central portion of the sill near the access drift. Both the east and west sides of the sill had been drilled, cabled and grouted leaving only a central strip of four to six open boreholes per ring thereby limiting the extent of the inspection.

At the south end, using the borehole camera and a black and white monitor, a parting has been confirmed and was easily identified in all inspected boreholes. The parting is interpreted to be oriented near horizontal and approximately parallel to and about 2 meters [6.5 ft] above the back. To the eastern edge of the inspection area the parting is well defined with an approximate thickness of ~200 mm [8 inches] and appearing open with large whitish cobbles that are defined by distinct fractures perpendicular to parting plane. To the western edge, the feature appears to widen or splay. Here it is characterized as a wider zone (up to a few feet thick) of poor ground rather than the open parting with solid cobbles visible along the eastern edge. The upper and lower contacts of this wider parting are typically distinct but less obvious and the area between the upper and lower bounds has an agglomerate texture. At all locations inspected, there appears to be some borehole closure below and near the parting, above the parting the rockmass is in good condition and there is no borehole closure.

At the central inspection area [entrance to the sill], the same characteristic parting was identified. In addition, and in response the cablebolt crew's comments about

experiencing rapid drill string advance also at ~ 3m [9.8 ft] up into the back, the borehole camera was extended to its maximum depth and an additional parting was identified at a depth of approximately 4.6 m [15 feet].

The sill and the adjacent haulage drift were inspected and no outcrop of the feature was identified.

The combination of the cable crew's observations and the borehole observations suggests that there are several naturally occurring parallel partings in the back. The areal expanse of each parting could not be completely quantified but there is sufficient evidence that the sill area is covered by one or more. It is also obvious that the partings are not the result of stress but a natural occurrence. That being said, the fact that the parting(s) is/are open suggests that there has been some disturbance to the local rockmass.

K.S. Kalenchuk, Ph.D.

P. Lausch, M.Sc., P.Eng.

Photographs

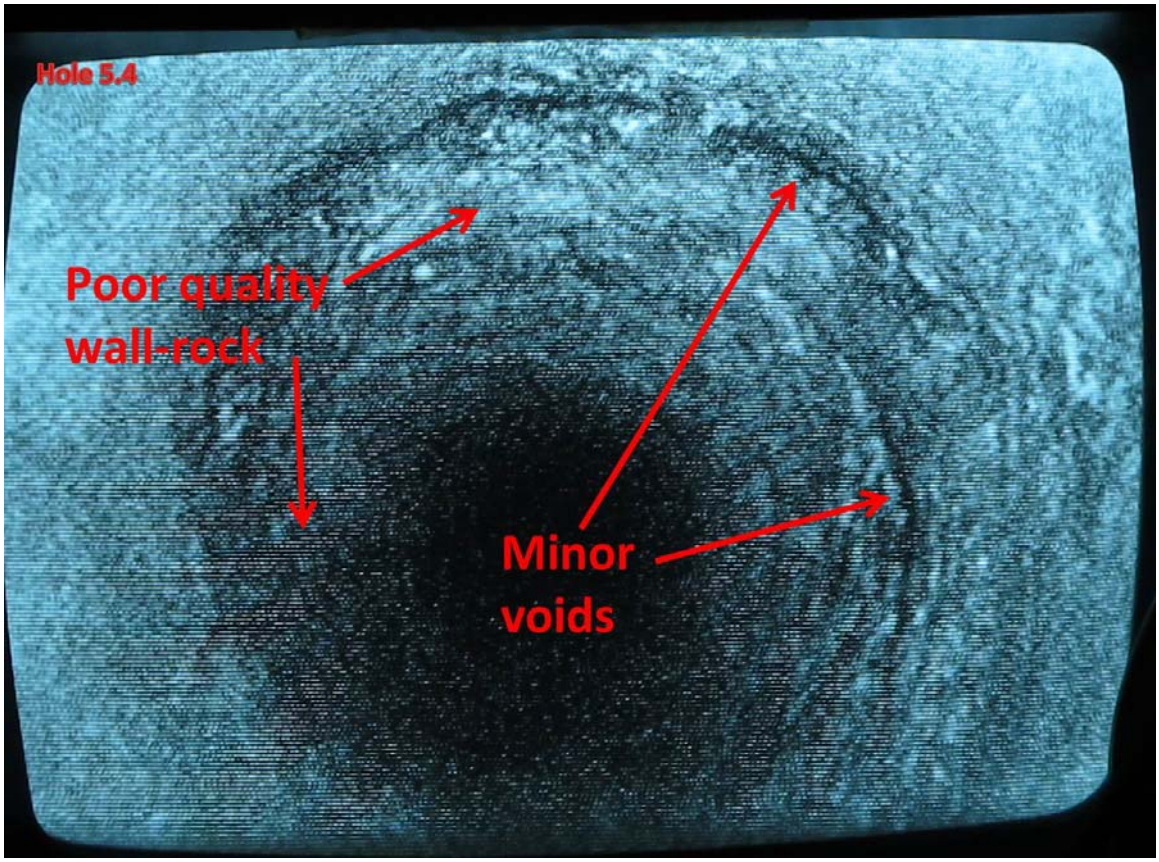

Figure 1: Hole 4 on ring 5.

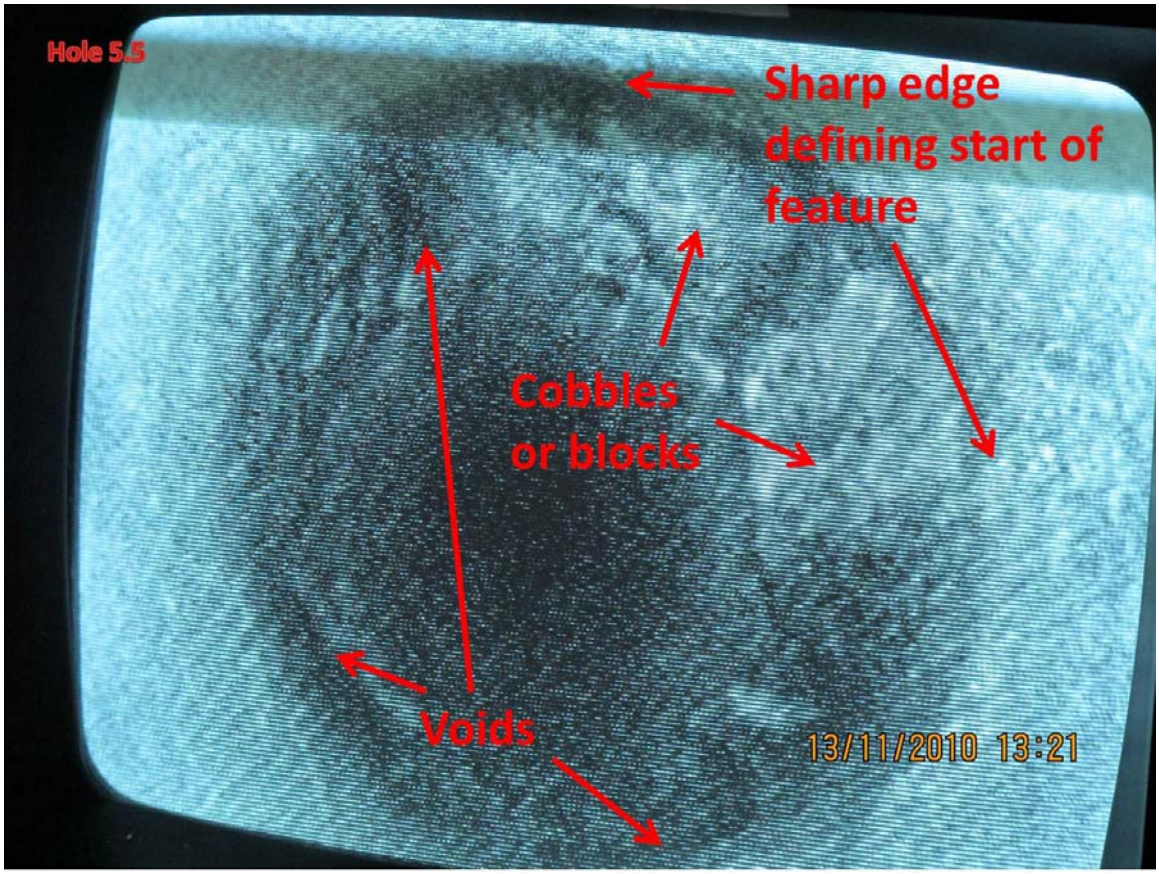

Figure 2: Hole 5 on ring 5.

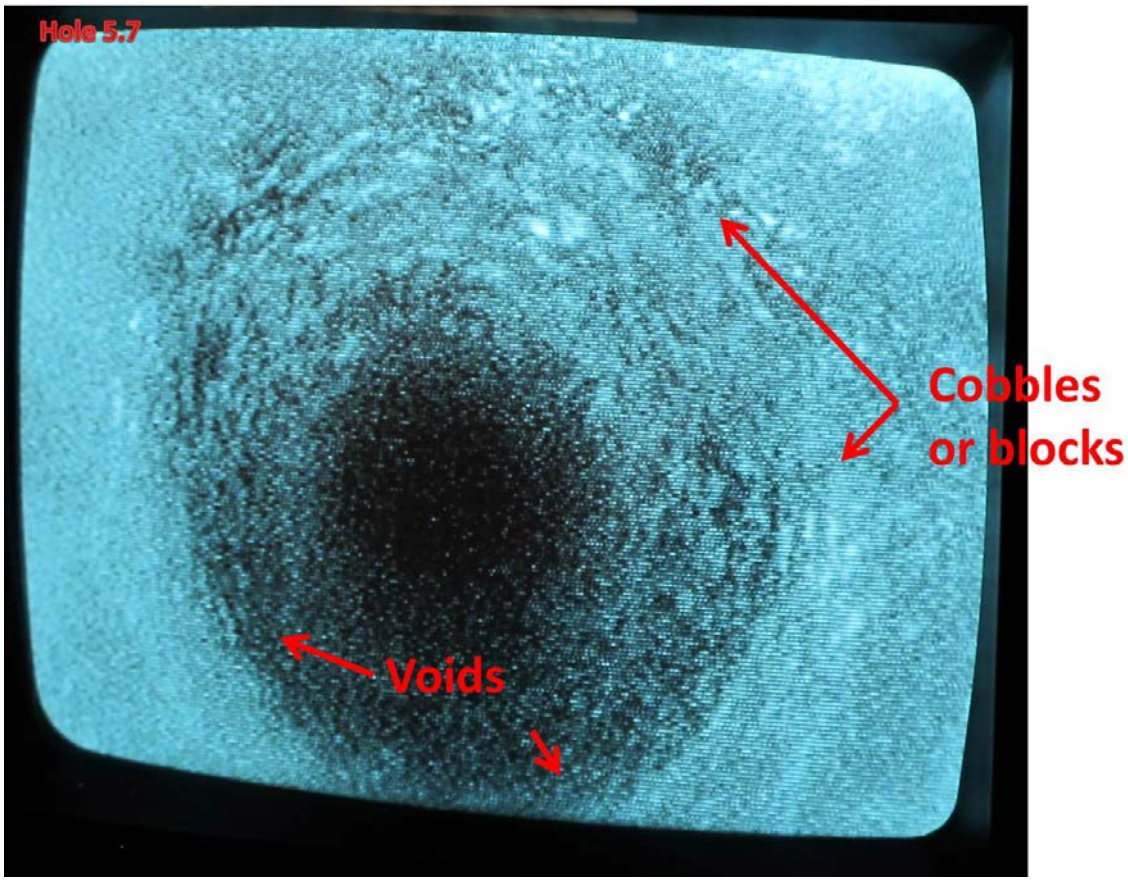

Figure 3: Hole 7 on ring 5.

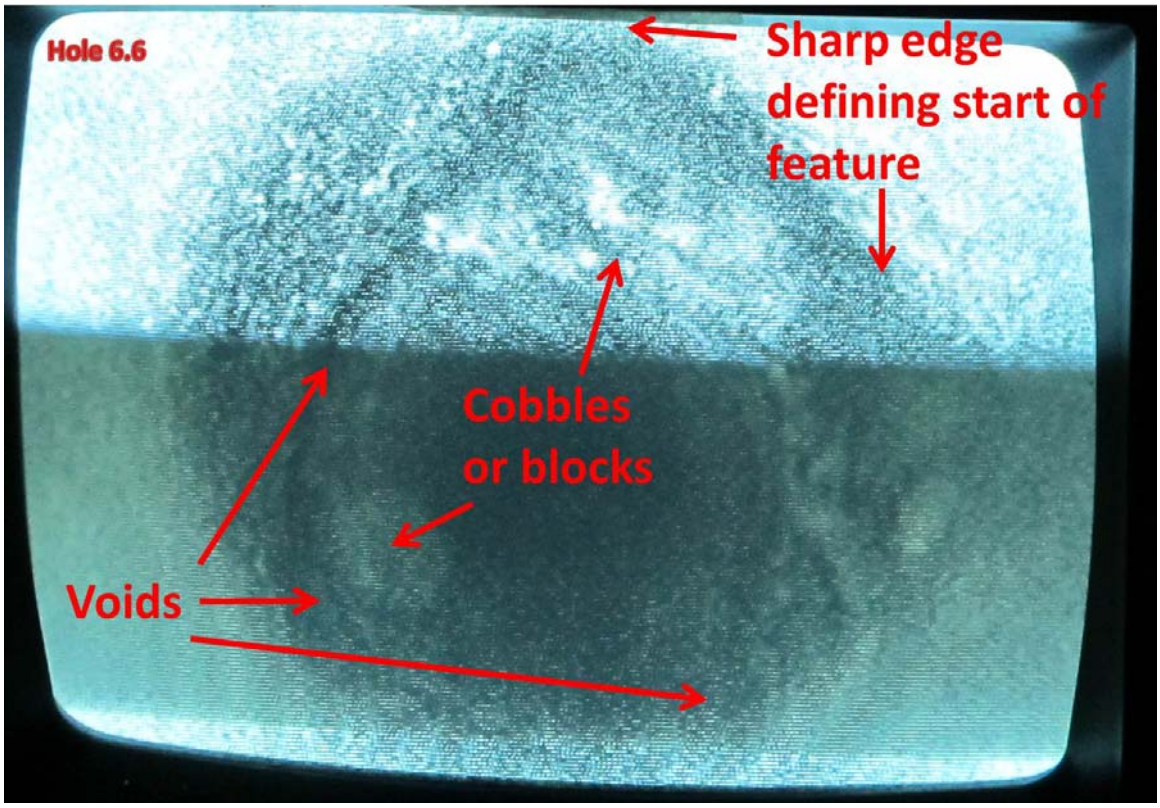

Figure 4: Hole 6 on ring 6.

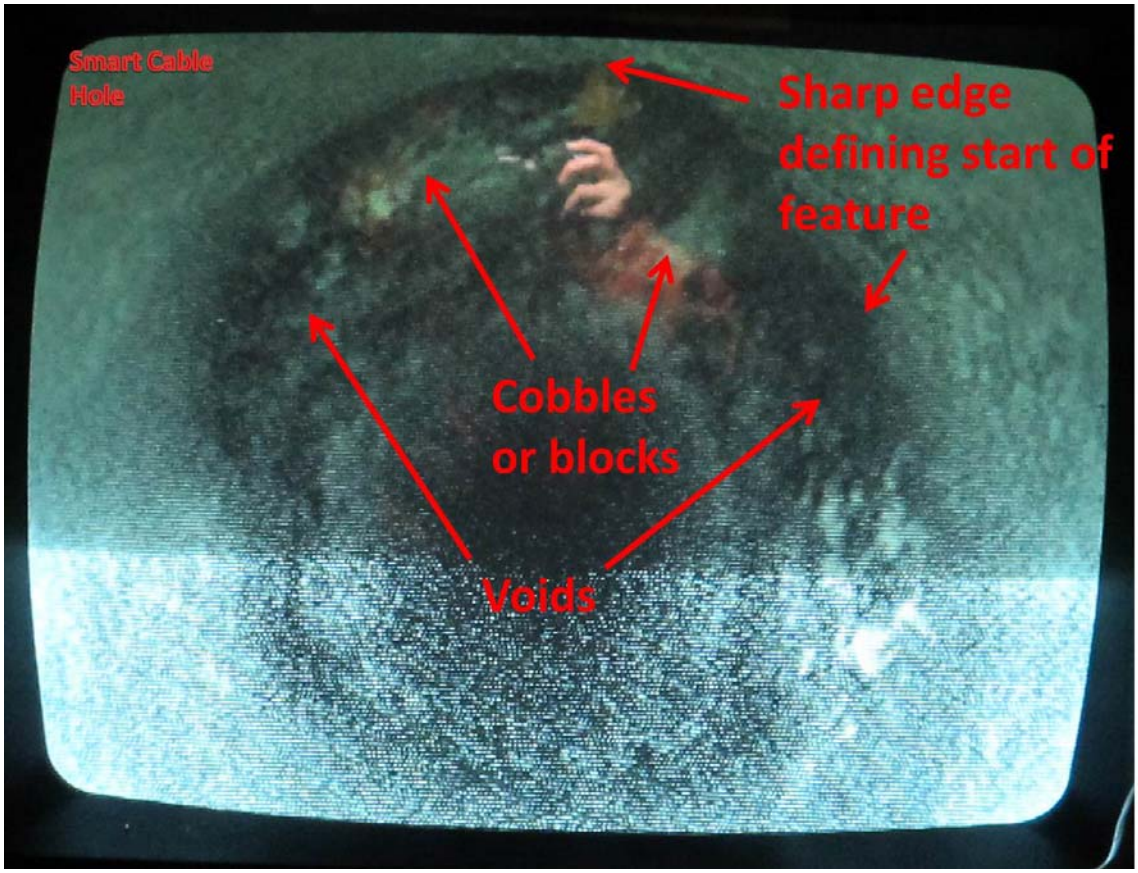

Figure 5: Smart cable hole east of stope access.

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To: Ron Moran

From: Kathy Kalenchuk

cc: W. F. Bawden

RE: 3-dimensional modelling to assess the anticipated mine induced seismicity and stress damage in the Offset Zone under the current plan for Super Shrinkage mining method.

Date: March 10, 2011

DRAFT

Executive Summary

In response to request from Ron Moran, Chief Engineer, Lac Des Iles Mine, Mine Design Engineering has developed, ran and analyzed three-dimensional boundary element models of the planned Super Shrinkage extraction of the Offset Zone. Results of this study confirm that the high extraction ratio of this mining method combined with the orebody depth will contribute to very high mine induced stresses. Mine induced seismicity and significant rockbursting is anticipated over the life of mine.

- Rib pillars will be seismic and rockmass yield is anticipated. The vertical nature of these pillars is favorable, however stopes will have to be kept full with broken muck, and later waste rock in order to provide the confinement necessary to avoid excessive rib pillar ravelling and associated damage propagation
- The 4740 sill pillar is expected to see some rockmass yield and will then be prone to unravelling failure similar to that experienced in the 5035 sill of the Roby Zone.
- The 4525 sill pillar will see very high mine induced stresses. This sill will be highly seismic and prone to rockbursting.
- Below approximately 4455 level the far field in situ stress exceeds 60 MPa. Development below this depth can expect to see minor stress induced damage.
- Following the current sequence for top-down, retreat to center the central pillar will see very high stresses. Rockburst risks will be extremely high and it will be difficult to maintain development in this area; very heavy dynamic support will be required. Further, significant borehole squeezing will occur. It is likely that some portion of the central pillar will be sterilized due to excessive seismicity and rockbursting.
- Development in areas adjacent to seismically active zones will be subjected to shakedown. Rockburst resistant support will be required in these areas and should be installed on development.

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- Highly seismic areas will require application of burst resistant support that should include fibrecrete with Swellex primary support with screen over the shotcrete. Deep support should consist of deep Super Swellex and/or debonded cable bolts. Secondary support depth will be dictated by the depth of seismic damage.
- Stopes will have to be kept full with broken muck, and later waste rock in order to minimize hangingwall ravelling. The application of HW cablebolts will also help to support the relaxed rockmass.

It is strongly recommended that changes be made to the mine sequence. Mining should either progress from the center out or retreat on strike to help control mine induced seismicity and maximize reserve extraction. An alternate mining method incorporating a cemented backfill such as paste fill is also recommended. This would permit extraction of most if not all pillars and would facilitate better control of mine induced seismicity.

Extensive instrumentation should be planned for the mine including a mine wide seismic system plus deformation monitors such as Multi-Point Borehole Extensometers or SMARTcables in key areas such as intersections, ore access drives and sill pillars. Instrument data will allow for better anticipation of rockmass response to planned mining and will be very important for model calibration. As mining progresses numerical models should be updated to accommodate any changes to mine plan and to calibrate input parameters as rockmass and instrument data become available.

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1 Introduction

The Offset Zone is a fault-displaced continuation of the Roby Zone. This steep west-dipping orebody extends from 600 m to about 1,300 m below surface over a strike length of 600 m with thickness up to 30 m. Planned mining will use a Super Shrinkage method with 60 m on strike panels separated by 10 m rib pillars. Mining is planned to start at both ends of the ore body and retreat to the centre, starting first with the upper block and the lower block being brought online later in mine life. This mining approach will result in a high extraction ratio, which combined with the orebody depth will contribute to high mine induced stresses.

Examine3D ™ software (RocScience Inc) has been used to simulate mining of the Offset Zone by the Super Shrinkage mining method. Constitutive parameters and boundary conditions for the in situ stress state have been assigned to the Offset models based on values defined in calibrated models of the Roby Zone. Analysis of stress concentrations throughout the Roby Zone mining area was completed for LDI by Kalenchuk (Report titled *"Assessment of wedge stability in drifts and intersections, 5035 sill pillar back analysis and 3-dimensional modelling to assess mine induced seismicity and stress damage"*, 2010). These three-dimensional elastic plastic models of the Roby Zone were calibrated such that the localized stress magnitudes of 0.4UCS and 0.6UCS correlated with approximate locations of reported seismicity and observed stress-damage, respectively. Figure 1 shows the location of the Offset Zone relative to the Roby Zone and Open Pit. The open pit and the Roby Zone have not been included in the numerical models as they are beyond the zone of influence for stresses in the Offset Zone vicinity.

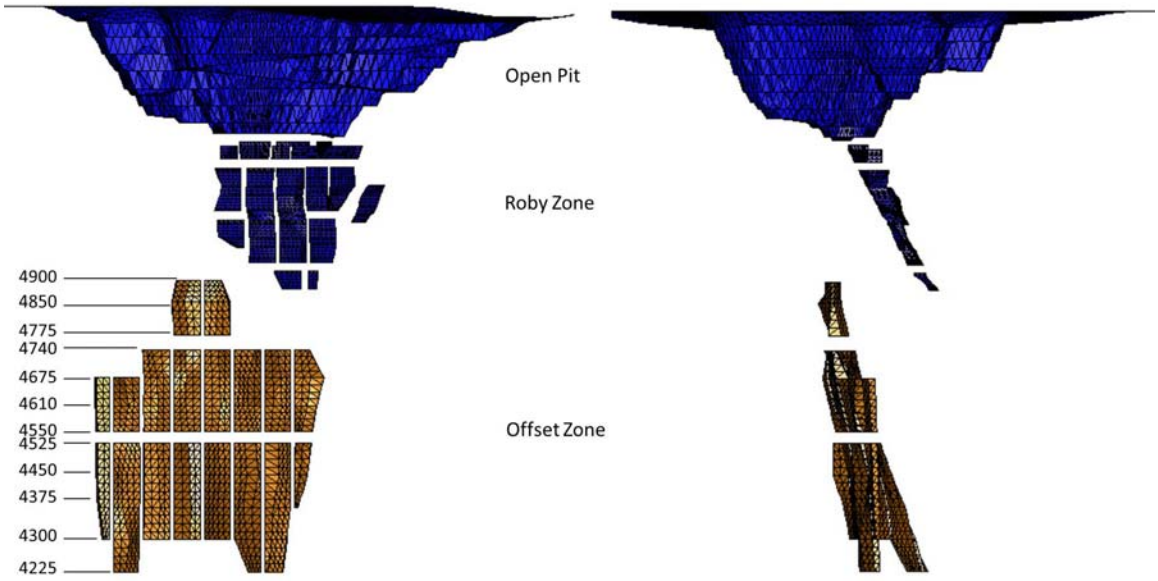

Figure 1: Location of the Offset Zone relative to the Roby Zone and the open pit;(left) looking west, (right) looking north.

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The current mine plan is to initiate at the top and progress down starting at the orebody edges and retreating to the center. Five simplified stages (Figure 2) have been assumed for this analysis to assess the changes in in situ stress over life of mine.

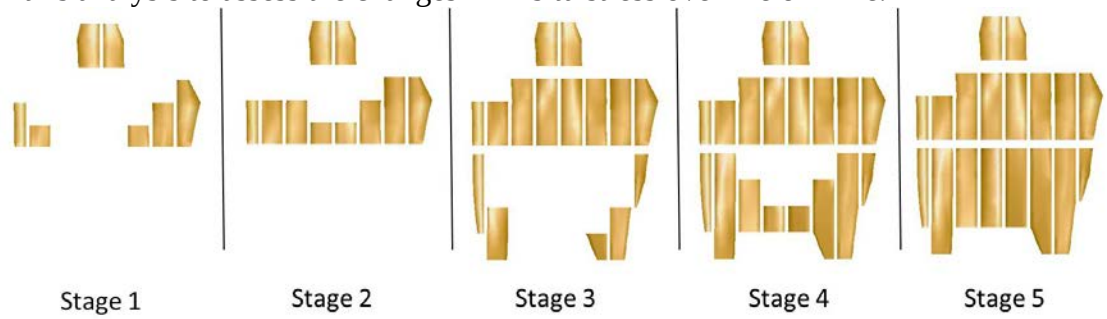

Figure 2: Mine Stages.

2 Model Development and Input Parameters

Assumed input parameters are summarized in Table 1. The unconfined compressive strength (UCS) is based on values summarized by various sources: Archibald, 2010 (66 to 212 MPa, average = 126 MPa), Knight Piésold, 2010 (50 to 200 MPa), Sitka Corp, 1999 (85 to 150 MPa), Itasca, 2004 (180 MPa) and various reports by Bawden (180 to 200 MPa). Hoek-Brown strength parameters are defined based on reasonable values for the LDI rockmass (GSI = 65, mi=27, D=0.8).

Table 1: Examine3D input parameters.

Parameter	**Value**
Young's Modulus	34 GPa
Poisson's Ratio	0.25
UCS	150 MPa
m	3.4
s	0.005
Density	2800 kg/m^3

Table 2 summarizes the assumed in situ stress state calibrated for models of the Roby Zone. In situ stresses are assumed to be continuous throughout the study region.

Table 2: Calibrated stress state.

	σ_v	σ_{h1}	σ_{h2}
Magnitude (MPa)	gravity	$2\sigma_v$	$1.5\sigma_v$
Dip (˚)	90	0	0
Dip Direction (˚)	0	90	0

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3 Results

Figure 3 and 4 illustrate the isosurfaces for in situ σ_1 (major principal stress) magnitudes of 60 MPa (0.4UCS) and 90 MPa (0.6UCS), respectively. The 60 MPa isosurface marks an approximate boundary where mine induced seismicity can be expected. The 90 MPa isosurface marks areas of high rockburst risk.

From Figure 3 it is apparent that mine induced seismicity can be anticipated in the rib pillars of the upper mining block in the 4525 sill and also in the 4740 sill as Block 2 is mined up over Stages 1 to 3. Between Stage 3 and the end of mine life significant seismicity is anticipated in the central pillar. Seismicity in the central pillar may cause shakedown in nearby development. Below approximately 4455 level the far field in situ σ_1 exceeds 60 MPa. Development below this depth can expect to see minor stress induced damage.

Figure 4 illustrates the σ_1 = 90MPa isosurface and demonstrates that rockmass yielding with bursting potential will occur in all rib pillars, starting as early as Stages 1 and 2. The vertical orientation of rib pillars is favorable to minimize failure. Effort will be required to keep stopes full with broken muck, and later waste rock, in order to provide the confinement necessary to avoid excessive rib pillar ravelling and associated damage propagation.

High drawpoint stresses will occur, initially in the lower block during Stage 3, and in both the lower and upper blocks though Stages 4 and 5. Given the high exposure of equipment and personnel to rock busting at drawpoints heavy, dynamic support will be required to maintain the integrity of stope access points.

As mining retreats from the edges to the center of the orebody high stresses will be pushed up through the central pillar ahead of the mining front. Rockburst risks will be very high and it will be difficult to maintain development in this area, very heavy dynamic support will be required. High stress in the central pillar will also cause significant borehole squeezing. Given the excessive seismicity and rockbursting anticipated for development in the central pillar it is likely that some portion of the ore in this area will be sterilized.

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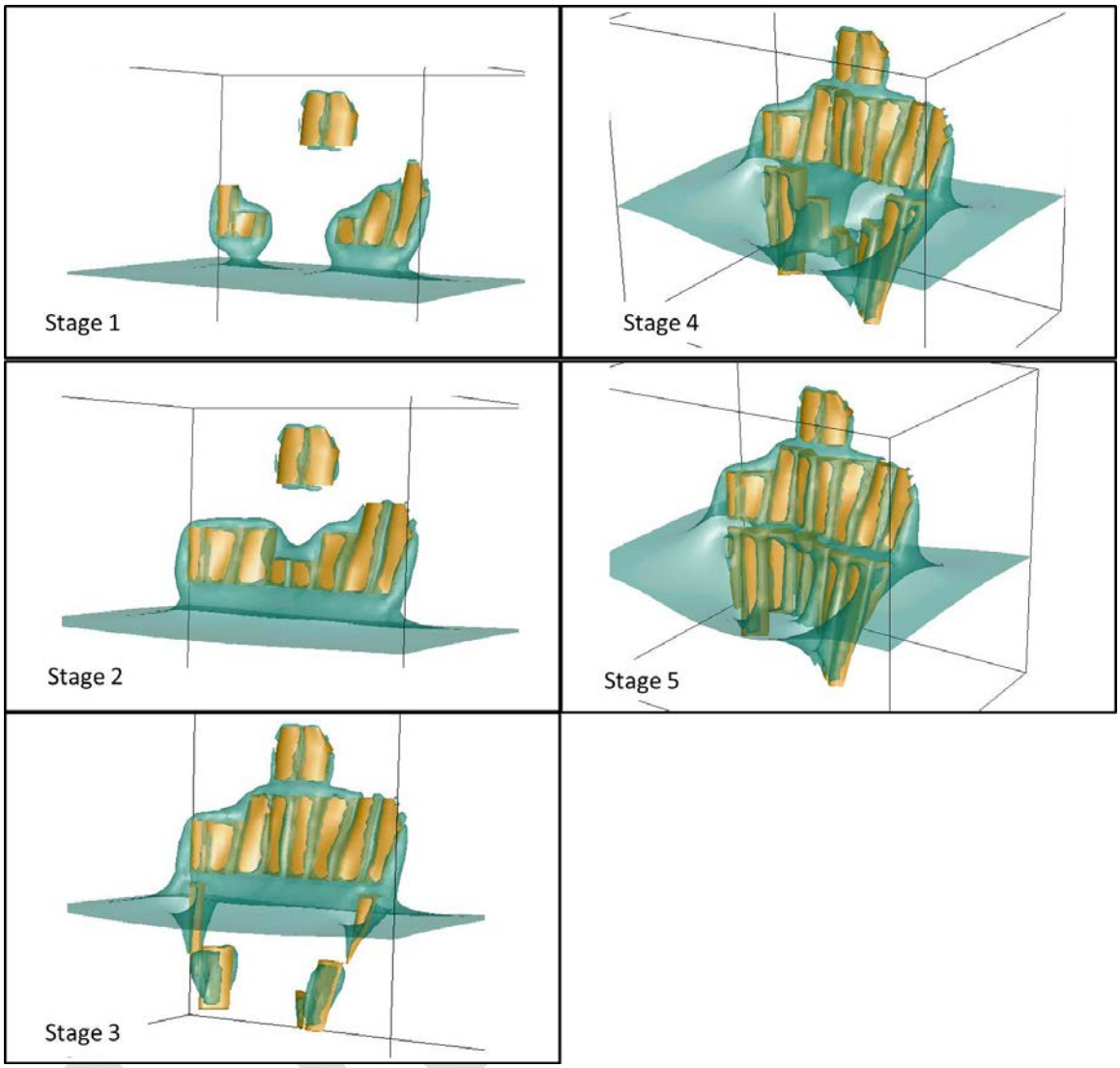

Figure 3: σ_1 = 60 MPa isosurface.

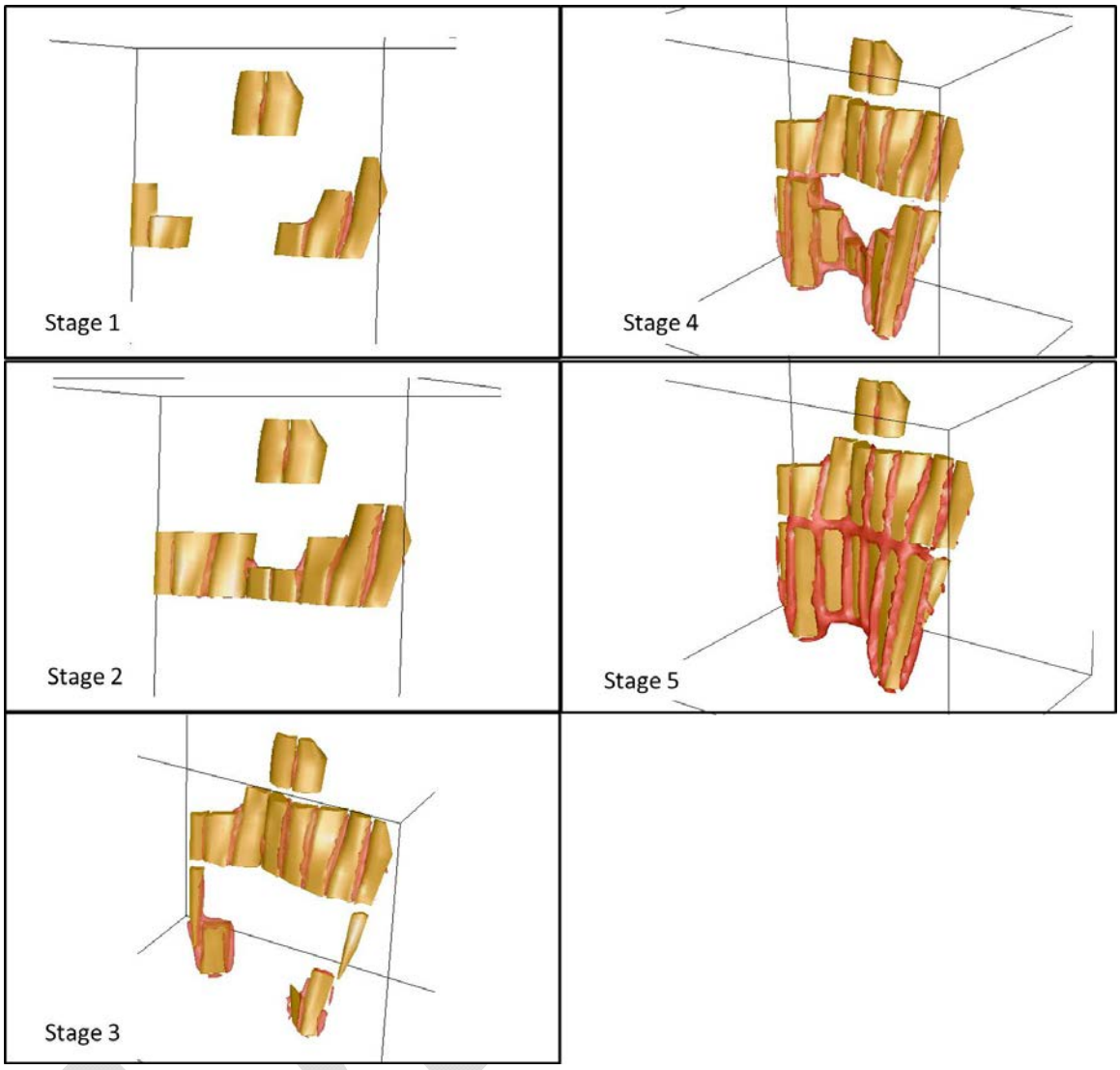

Figure 4: σ_1 = 90 MPa isosurface.

Figure 5 illustrates the expected distribution of σ_1 through the Offset Zone sill pillars. By Stage 3 mine induced seismicity will have initiated in both sills. Under a similar simulated stress state, the Roby Zone has experienced significant failures in the 5035 sill pillar. Without adequate sill support significant ravelling and instability can be anticipated in the Offset Zone. In stages 4 and 5 the 4525 sill will see very high stress magnitudes and will be highly seismic and prone to rockbursting. Highly seismic areas will require application of burst resistant support that should include fibrecrete with Swellex primary support with screen over the shotcrete. Deep support should consist of deep Super Swellex and/or debonded cable bolts. Secondary support depth will be dictated by the depth of seismic damage.

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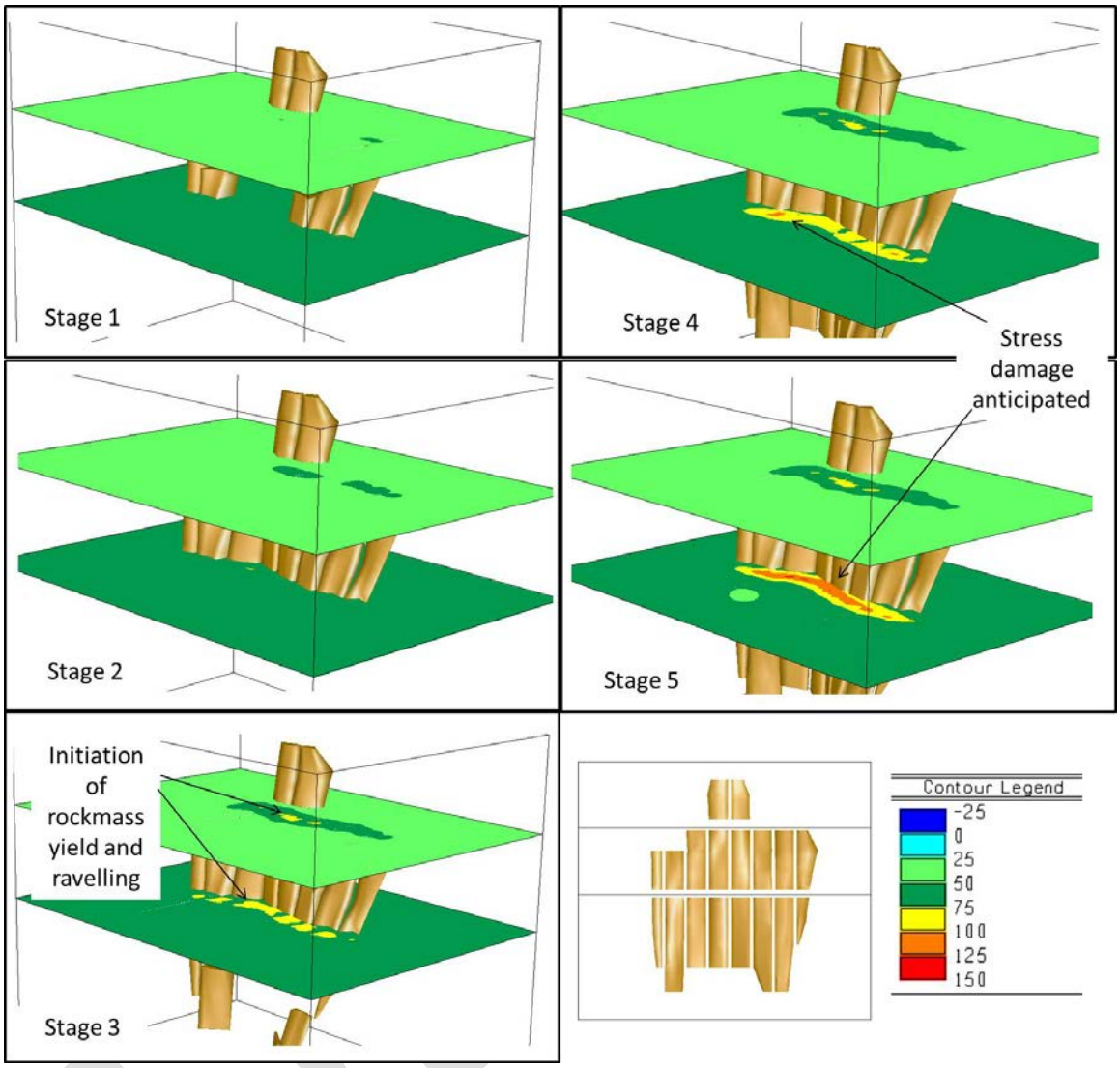

Figure 5: σ_1 magnitudes predicted in the Offset Zone sill pillars.

Figure 6 illustrates the relaxed stress state that will occur in Super Shrinkage stope hangingwalls. Significant hangingwall relaxation is not anticipated until late in the mining sequence. In the event that a rib pillar fails the hydraulic radius will increase and low stresses will extend deeper into the hangingwall, this circumstance may lead to excessive dilution and HW failure. Maintaining broken rock in the stopes will minimize HW ravelling. The application of HW cablebolts will also help to support the relaxed rockmass.

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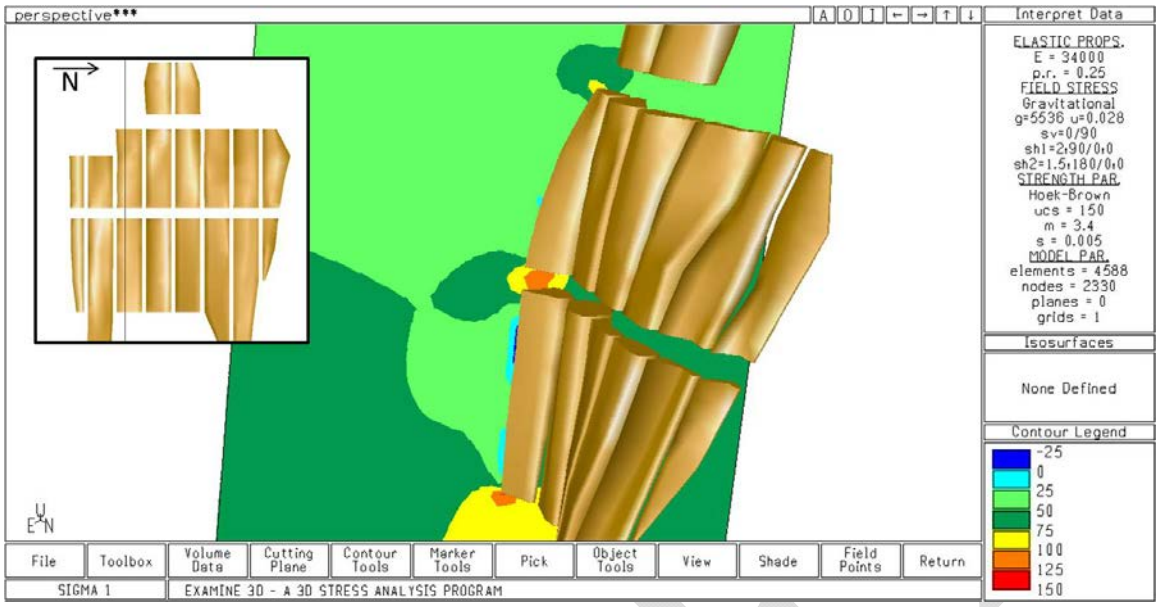

Figure 6: Cross-sections to illustrate the low σ_1 in the relaxed hangingwalls of Super Shrinkage stopes (inset shows section location).

4 Conclusions and Recommendations

The numerical modelling results presented in this report demonstrate that high mine induced stresses will be encountered over the Offset Zone mine life:

- Mine induced seismicity and significant rockbursting is anticipated over the life of mine following the current mine plan to use the Super Shrinkage method.
- Rib pillars will be seismic and rockmass yield is anticipated. The vertical nature of these pillars is favorable, however stopes will have to be kept full with broken muck, and later waste rock in order to provide the confinement necessary to avoid excessive rib pillar ravelling and associated damage propagation.
- The 4740 sill pillar is expected to see some rockmass yield and will be prone to unravelling failure similar to that experienced in the 5035 sill of the Roby Zone.
- The 4525 sill pillar will see very high mine induced stresses. This sill will be highly seismic and prone to rockbursting.
- Below approximately 4455 level the far field in situ σ_1 exceeds 60 MPa. Development below this depth can expect to see minor stress induced damage.
- Following the current sequence for top-down, retreat to center the central pillar will see very high stresses. Rockburst risks will be extremely high and it will be difficult to maintain development in this area, very heavy dynamic support will be required. Further, significant borehole squeezing will occur. It is likely that some portion of the central pillar will be sterilized due to excessive seismicity and bursting.

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- Development in areas adjacent to seismically active zones will be subjected to shakedown. Rockburst resistant support will be required in these areas and should be installed on development.
- Highly seismic areas will require application of burst resistant support that should include fibrecrete with Swellex primary support with screen over the shotcrete. Deep support should consist of deep Super Swellex and/or debonded cable bolts. Secondary support depth will be dictated by the depth of seismic damage.
- Stopes will have to be kept full with broken muck, and later waste rock in order to minimize hangingwall ravelling. The application of HW cablebolts will also help to support the relaxed rockmass.

Based on these findings it is recommended that the mine sequence be changed to mine either center out or retreat on strike to help control mine induced seismicity and maximize reserve extraction. An alternate mining method incorporating a cemented backfill such as paste fill is also recommended. This would permit extraction of most if not all pillars and would facilitate better control of mine induced seismicity.

Extensive instrumentation should be planned for the mine including a mine wide seismic system plus deformation monitors such as Multi-Point Borehole Extensometers or SMARTcables in key areas such as intersections, ore access drives and sill pillars. As mining progresses these numerical models should be updated to accommodate changes in input parameters as more data becomes available to characterize the Offset Zone rockmass and also to incorporate any changes to the planned mining. Instrumentation data will be very important for model calibration and can be used to better anticipate rockmass response to mine plans.

Mine Design Engineering would be happy to review the findings of this studyto ensure that the implications of these results are clear, and to clarify any questions or concerns with LDI at NAP headquarters in Toronto upon request.

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K.S. Kalenchuk, Ph.D.

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Review by W.F. Bawden, Ph.D., P.Eng.

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MEMO TO: Mr. G. Struble, COO, North American Palladium

FROM: W. F. Bawden

DATE: April 16, 2011

SUBJECT: Clarification on Offset Zone Mining

Further to Friday's meeting concerning recent modelling results [Kalenchuk report] from the Offset Zone I want to clarify a few issues that I am not certain everyone was clear about in the discussion [I personally had problems hearing the phone discussion]. These are listed below in point form.

- There was concern raised that predictions from the latest model results are significantly different than those in the scoping study. However, in my opinion there has been a material change in geotechnical conditions since the scoping study. When the scoping study was conducted there was no evidence of stress driven failure at the mine. Ground behaviour in the lower Roby zone since the fall of 2010 however has forced a complete re-evaluation of the impact of mine induced stress on future mining, particularly the Offset Zone.
- The above has led me to conclude that the mine method, and more **particularly the outside-in sequence**, modelled in the recent report by Kalenchuk will not be successful. Reasons for this are given in the conclusions to that study. I edited those conclusions to forcibly make this point as I believe the mine must consider this carefully before committing to a final mining method.
- Creation of the **large reducing pillar between blocks 2 and 3** [Fig. 4 – Stage4 in Kalenchuk report] would **create a major risk** that in my opinion the mine should avoid.
- **This does not mean that the Super Shrinkage method cannot be used.** If used however, sequencing must be center-out and, in my opinion, the panels will need to be filled with dry waste to confine the rib pillars so that they do not completely deteriorate and unravel under stress induced yield. Sill pillar support will have to also be given very careful attention.
- If the mine elects to use a cemented fill [CRF or paste] then the rib pillars can be eliminated.
- If a fill method is used this does not preclude the use of large, high tonnage stopes. Stopes can still be developed in the same manner as originally proposed [i.e. Alimak raise for slots and to cable the HW] and can still be opened to the strike length originally planned. Multiple drill levels can still be blasted to the draw points. The allowable stope height will depend on achievable fill strength as the exposed fill face must be stable when exposed by adjacent mining. Achieving

suitable fill strength for the tallest stopes being considered should not be a problem. A trade off study between binder cost and number of drawpoint levels would probably be required.

- Productivity issues with a filling method then revolve around achieving a center-out type sequence to drive the high stress onto the abutments rather than permanent pillars or secondary stopes while delivering the required tonnage to the mill.
- Note that 100m high Alimak stopes were routinely mined and paste filled at the Williams and David Bell mines and that a few 180m high Alimak stopes were also mined and paste filled. The paste in these stopes was exposed by adjacent mining and was stable. These stopes were much narrower [HW-FW] than those proposed for Offset, but binder content and hence fill strength was also modest.

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To: Ron Moran

From: Kathy Kalenchuk

cc: W. F. Bawden

RE: 3-Dimensional Numerical Assessment of the Impact that LDI Crown Pillar Mining will have on the local in situ state of stress.

Date: May 9, 2011

Executive Summary

In response to request from Ron Moran, Chief Engineer, Lac Des Iles Mine, Mine Design Engineering has developed, ran and analyzed three-dimensional boundary element models to simulate mining of the LDI crown pillar. Results indicate that mining of the crown pillar will cause negligible change in the mine induced stress state, and therefore it has been concluded that from a stress management point of view, mining of the crown pillar will not likely have any negatively influence the stability of Roby Zone underground workings. Nonetheless, consideration must be given to the fact that crown pillar mining will leave the 5155 sill as a critical pillar separating all lower Roby Zone mine workings from the overlying open pit. It is advisable that all stopes immediately below the 5155 sill be completely backfilled with muck pushed up to the brow prior to crown pillar mining. This will ensure that the stability of the 5155 sill pillar is maintained by restricting gravity driven raveling.

The recently installed micro-seismic system is tied into sensors on 5210, 5180, 5155 and 5120 levels. Monitoring seismicity in the upper Roby Zone will be very valuable to track any damage accumulation within the 5155 sill and surrounding area during crown pillar extraction. Further, MPBX instruments are installed in the sill of #2 stope block on 5155 level; these instruments will be valuable for measuring ground deformations that may occur in response to any changes in the in situ stress state. It is strongly recommended that the functionality of these instruments and the micro-seismic system be checked prior to initiation of crown pillar mining.

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1 Introduction

Underground mining at LDI has left an approximately 20 m thick crown pillar between the Roby Zone stopes and the overlying open pit. A plan has been suggested to mine the crown pillar from surface. In response to request from Ron Moran, Chief Engineer at the Lac Des Iles Mine (LDI), Mine Design Engineering (MDEng) has used the three-dimensional linear-elastic boundary element software Examine3D ™ (RocScience Inc.) to model mining of the LDI crown pillar. The models simulated in this study have assumed a simplified geometry extending the current bottom of the open pit to the 5210 sill. Figure 1 illustrates the base case model prior to crown pillar extraction and the subsequent model with the final mined out state. This study has allowed for predictions to be made regarding how crown pillar extraction will induce changes to the in situ stress state and influence stability of the underground Roby Zone mine workings. This report summarizes model development and analysis of the results. Based on these results recommendations are provided with respect to mining of the crown pillar.

Figure 1: Illustration of model geometries for (left) pre- and (right) post-crown pillar mining.

2 Model Development and Input Parameters

Constitutive parameters and boundary conditions have been assigned to the numerical models based on values defined in calibrated models of the Roby Zone. Analysis of stress concentrations throughout the Roby Zone mining area has been previously completed for LDI by MDEng (Report titled *"Assessment of wedge stability in drifts and intersections, 5035 sill pillar back analysis and 3-dimensional modelling to assess mine induced seismicity and stress damage", 2010*). In that study, Hoek-Brown strength parameters were defined based on reasonable values for the LDI rockmass (GSI = 65, m_i=27, D=0.8) and the unconfined compressive strength (UCS) was inferred based on information from various sources: Archibald, 2010 (66 to 212 MPa, average = 126 MPa), Knight Piésold, 2010 (50 to 200 MPa), Sitka Corp, 1999 (85 to 150 MPa), Itasca, 2004 (180 MPa) and various reports by Bawden (180 to 200 MPa). Model input parameters

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are summarized in Table 1. The calibrated Roby Zone models indicate that seismicity can be expected, where there is low confinement in the vicinity of mine workings and the predicted maximum principal stress is in the order of 60 MPa (0.4UCS). Stress induced damage is likely to occur when σ_1 is near, or in excess of, 90 MPa (0.6UCS).

Table 1: Examine3D input parameters.

Parameter	**Value**
Young's Modulus	34 GPa
Poisson's Ratio	0.25
UCS	150 MPa
m	3.4
s	0.005
Density	2800 kg/m^3

Table 2 summarizes the in situ stress state calibrated for models of the Roby Zone. In situ stresses are assumed to be continuous throughout the study region.

Table 2: Calibrated stress state.

	σ_v	σ_{h1}	σ_{h2}
Magnitude (MPa)	gravity	$2\sigma_v$	$1.5\sigma_v$
Dip (˚)	90	0	0
Dip Direction (˚)	0	90	0

2.1 Background

Figure 2 illustrates results from the calibrated models of the Roby Zone. These results are included here to demonstrate the simulated σ_1 magnitudes in the 5035 sill (between 50 and 75 MPa) which correlate well with reports of seismicity. Further, predicted areas where σ_1 magnitudes are in the order of 75 to 100 MPa corresponded well with the extent of sill pillar failure observed in November 2010. It should be noted that since the calibrated modelling study was completed the sill in #3 and #2 blocks continued to ravel and were eventually removed completely by planned blasting in order to avoid uncontrolled sill pillar failure. The calibrated models were also capable of predicting stress magnitudes exceeding 100 MPa which corresponded with stress damage and burst evidence on 4975 level near the rib pillar between #3 and #4 stopes.

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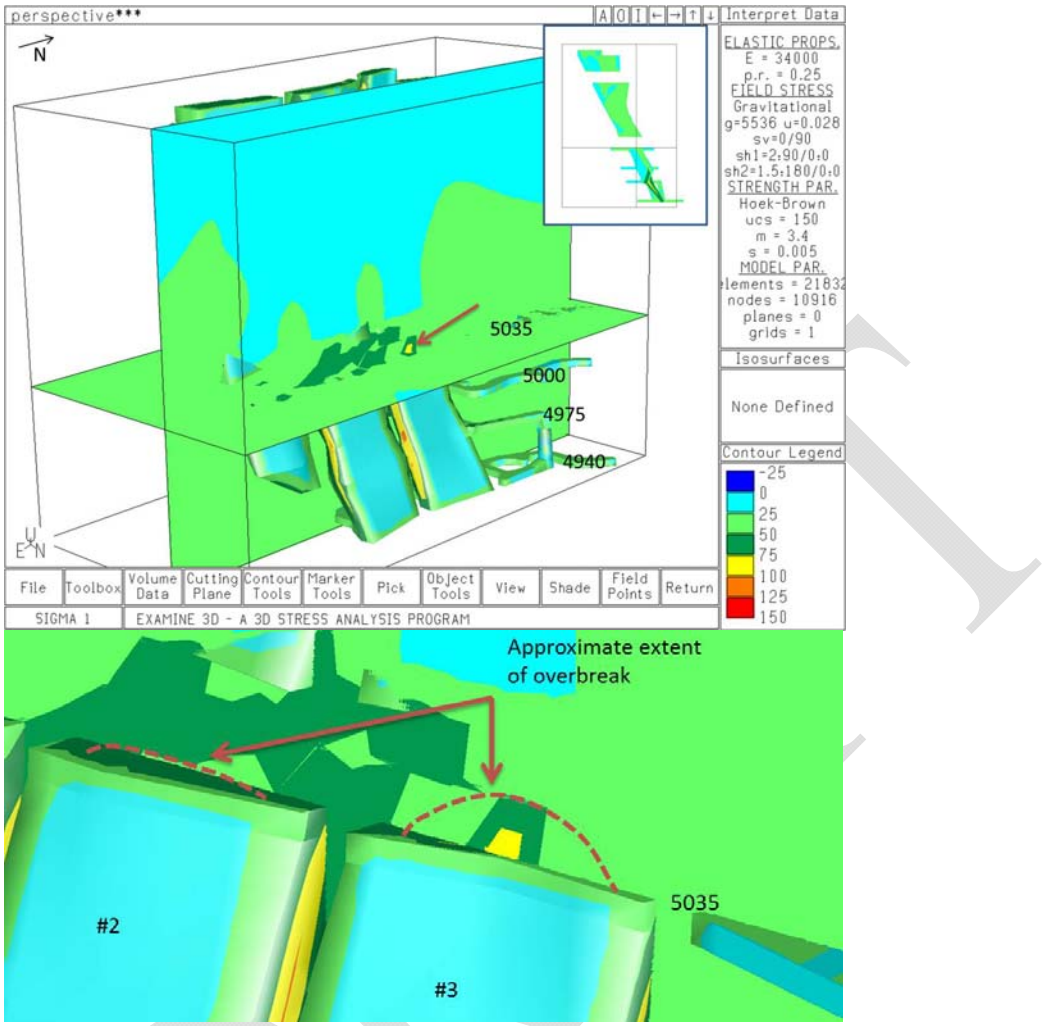

Figure 2: (top) Stress levels in 5035 sill exceeding 0.4UCS correlate well with seismicity reports and magnitudes between 75 and 100 MPa correspond to the extent of observed pillar failure in November 2010. Inset shows location of cutting planes (looking north). (bottom) Red dashed lines show the approximate extent of failure in the 5035 sill pillar during the November, 2010 inspection.

3 Results

Figure 3 illustrates the major principal stress magnitudes in the upper Roby Zone with all planned stopes excavated prior to crown pillar extraction. This mine induced stress distribution suggests that minimal seismicity likely occurred in the crown pillar and the 5155 sill pillar during mining of the adjacent stopes. Models indicate that removal of the crown pillar will cause very minor change in the mine induced stress state surrounding the Roby Zone (Figure 4). The only noticeable change in predicted stress state is a small increase in σ_1 through the 5155 sill. This small increase in compressive stress may

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contribute to small amounts of mine induced seismicity with localized yielding; however the overall sill pillar stability should not be negatively impacted.

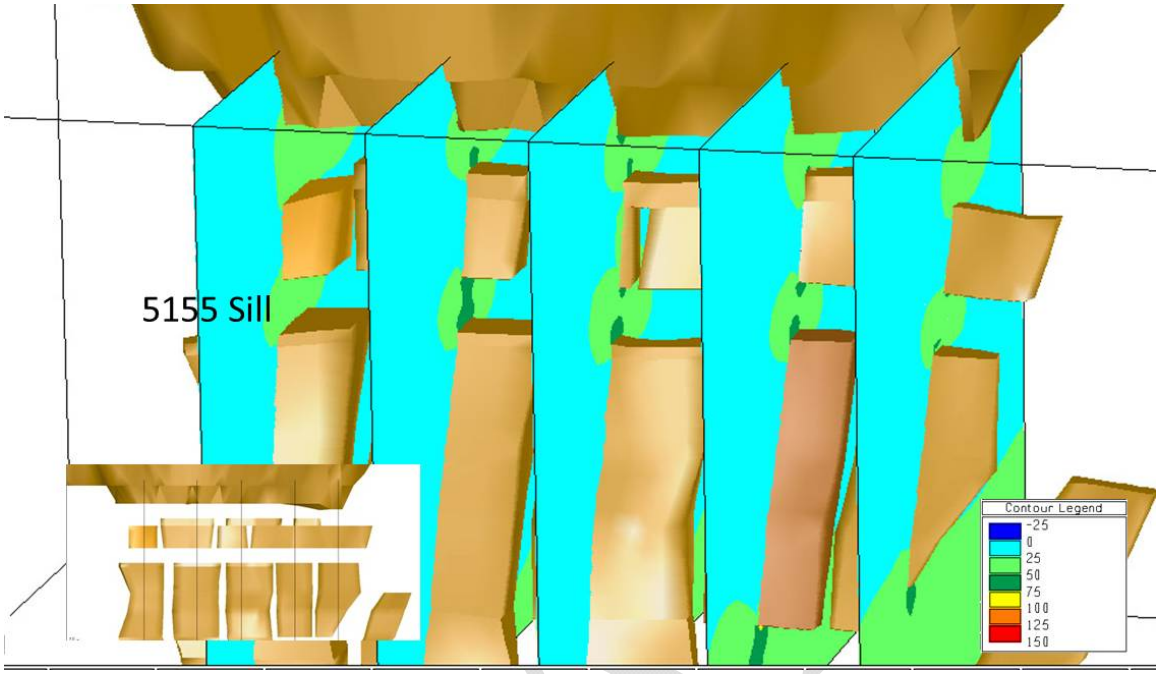

Figure 3: Vertical sections through each stope block looking northeast towards the hangingwall with contoured σ_1 (MPa) prior to simulated extraction of the crown pillar (inset shows section locations looking east).

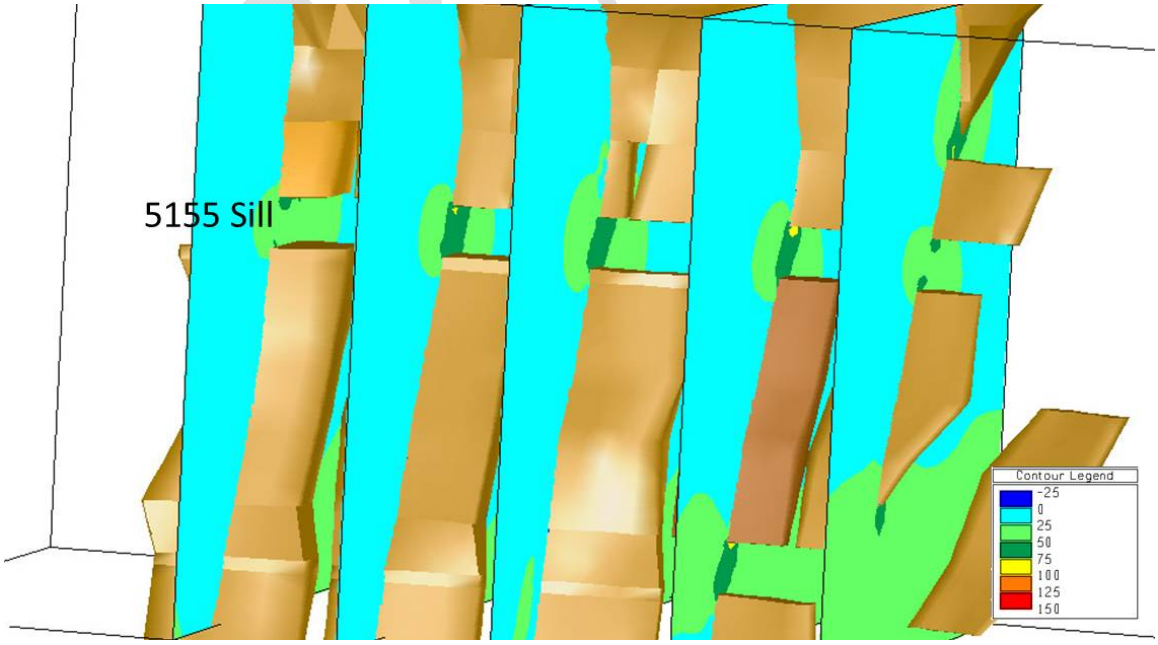

Figure 4: Vertical sections through each stope block looking northeast towards the hangingwall with contoured σ_1 (MPa) with the crown pillar removed.

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4 Conclusions and Recommendations

Results from numerical simulation of crown pillar removal indicate that negligible change in the mine induced stress state will occur. This suggests that, from a stress management point of view, mining of the crown pillar is not expected to negatively influence the stability of Roby Zone underground workings. However, it must be considered that mining of the crown pillar will leave the 5155 sill as a critical pillar separating all lower Roby Zone mine workings from the overlying open pit. It is advisable that all stopes immediately below the 5155 sill be completely backfilled with muck pushed up to the brow prior to crown pillar mining. This will ensure that the stability of the 5155 sill pillar is maintained by restricting gravity driven raveling.

The recently installed micro-seismic system is tied into sensors on 5210, 5180, 5155 and 5120. This will provide some coverage of the area of concern. Measured seismicity will be very valuable to track if damage is accumulating in the 5155 sill during crown pillar mining. Further, MPBX instruments are installed in the sill of #2 block on 5155 and these instruments will be valuable for measuring ground deformations that may occur with changes to the in situ stress state. It is strongly recommended that the functionality of these instruments and the micro-seismic system be checked prior to initiation of crown pillar mining.

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K.S. Kalenchuk, Ph.D.

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Review by W.F. Bawden, Ph.D., P.Eng.

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To: Ron Moran

From: Kathy Kalenchuk

cc: W. F. Bawden

RE: 2-Dimensional assessment of pit slope stability with mining of the crown pillar.

Date: December 9, 2011

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Executive Summary

In response to request from Ron Moran, Chief Engineer, Lac Des Iles Mine, Mine Design Engineering has developed, ran and analyzed two-dimensional finite element models to assess pit slope stability with mining of the LDI crown pillar. Pit slope stability has significant implications to the long term integrity of the Offset Zone ventilation shaft which is designed to outcrop in the pit slope. Numerical results indicate that mining of the crown pillar will cause negligible change in stability of the LDI open pit.

It has also been noted that the 5155 sill pillar is a critical pillar which will ultimately separate the Roby Zone mine workings from the overlying open pit. Mining of the crown pillar is not predicted to cause any major yield in that pillar. The LDI micro-seismic system provides some coverage of the pillar area and measured seismicity will be very valuable to track any new damage accumulation in the 5155 sill during crown pillar mining. MPBX instruments in the sill of #2 block on 5155 will be valuable for measuring ground deformations that may occur. It is strongly recommended that the functionality of these instruments and the micro-seismic system be checked prior to initiation of crown pillar mining. It is also recommended that all stopes below the 5155 sill be completely filled with rockfill prior to crown pillar extraction. Rockfill should be pushed tight to the stope backs to avoid the initiation and propagation of gravity driven sill pillar ravelling.

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1 Introduction

A 20 m thick crown pillar separates the Lac Des Iles (LDI) open pit from the underground workings of the LDI Roby Zone; mining of the crown pillar is planned for 2012. In response to request from Ron Moran, Chief Engineer at LDI, Mine Design Engineering (MDEng) has completed a two-dimensional finite element modelling study (using Phase2^{TM} RocScience Inc.) to assess the influence of crown pillar mining on the open pit slope stability. Pit slope stability has significant implications to the long term integrity of the Offset Zone ventilation shaft which is designed to outcrop in the pit slope.

2 Model Development and Input Parameters

Figure 1 illustrates a cross section through the open pit intersecting the planned ventilation shaft. This cross section is typical of the LDI open pit, and it has therefore been decided that additional cross sections are not required to further test pit slope stability.

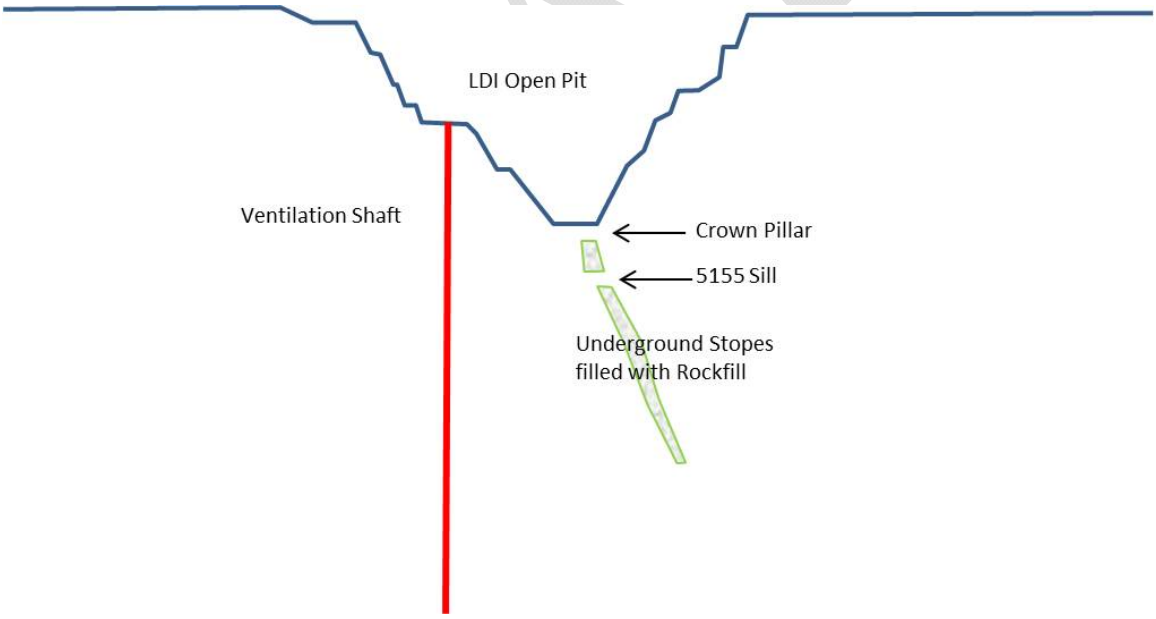

Figure 1: Schematic of an east-west cross section through the LDI open pit located at 32073 m North.

2.1 Rockmass Strength Parameters

Rockmass strength parameters have been assigned to the numerical models based on parameters used in a previously calibrated modelling study of the Roby Zone (Report titled *"Assessment of wedge stability in drifts and intersections, 5035 sill pillar back analysis and 3-dimensional modelling to assess mine induced seismicity and stress damage", 2010*). In that study the LDI rockmass is characterized by GSI = 65, m_i = 27, D = 0.8 and UCS = 150 MPa. For this study D is assumed to be 1 and GSI has been reduced to 60. This increase in disturbance factor and reduction of GSI reflect the fact that the rockmass is likely more disturbed and fractured in the open pit than it would be underground. From this characterization Mohr Coulomb strength parameters have been derived as friction = 29˚, cohesion = 7 MPa and tensile strength equal to 0.2 MPa. Post peak rockmass strength assumes a further reduction of GSI to 50, giving friction = 24˚ and cohesion = 5 MPa, residual tensile strength is assumed to be zero. The Young's Modulus is assumed to be 34 GPa and Poisson's ratio is assumed to be 0.25. Rockmass density is 2800 kg/m^3.

2.2 Rockfill Strength Parameters

Underground stopes in the Roby Zone are largely filled with unconsolidated rockfill. Rockfill is assumed to have zero cohesion and tensile strength. Density is assumed to be 1700 kg/m^3 with zero cohesion and friction equal to 36˚ based on "typical backfill properties" suggested by Hoek (2003).

2.3 Constitutive Models

Mohr Coulomb strength criterion is applied to both material types. The rockmass has strain softening behavior (lower residual strength). Rockfill is simulated as perfectly plastic.

2.4 Boundary Conditions

In situ stresses in the numerical models assume that vertical stresses are controlled by the weight of overlying rockmass and horizontal stresses are twice the vertical stresses. The model perimeters have roller boundaries applied allowing boundary parallel displacements with no boundary perpendicular displacements. The topographic surface is a free boundary.

2.5 Groundwater

Pit slopes are assumed to be dry.

2.6 Model Set-up

The following sequence is applied to numerical simulations:

1. Excavation of the open pit over a number of stages.
2. Excavation of underground stopes and application of rockfill.
3. Excavation of the crown pillar.

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3 Results

Figure 2 illustrates contour plots of the major (σ_1) and minor (σ_3) principal stresses prior to and post crown pillar mining. Crown pillar extraction causes σ_1 near the pit bottom to be pushed deeper into the rockmass below the upper stopes of the Roby Zone. This will induce a very minor increase in σ_1 through the 5155 sill pillar. The minor principal stress drops in vicinity of the pit bottom when the crown pillar is mined out; this due to loss of confinement when in situ stresses are forced deeper into the rockmass.

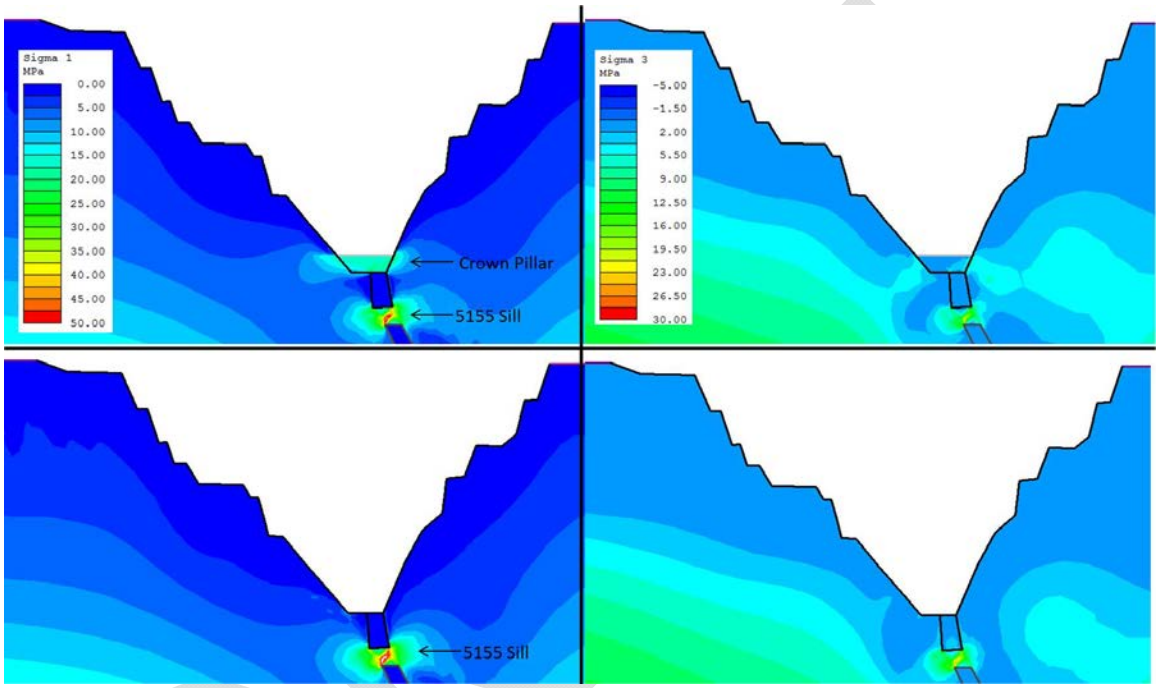

Figure 2: Contoured (left) major and (right) minor principal stresses for (top) prior to and (bottom) post crown pillar mining.

Figure 3 illustrates the predicted, accumulated rockmass tensile yield surrounding the LDI open pit and underground excavations. Tensile yield is evident along the topographic surface, pit walls and near open stopes. This surficial rockmass relaxation is typical in areas of low confinement; such as in pit slopes and near underground excavations. Figure 4 shows that shear yield is predicted below the open pit, this is caused by elevated σ_1 magnitudes near the pit bottom. Mining of the crown pillar does not result in any significant increase to the extent of predicted shear yield.

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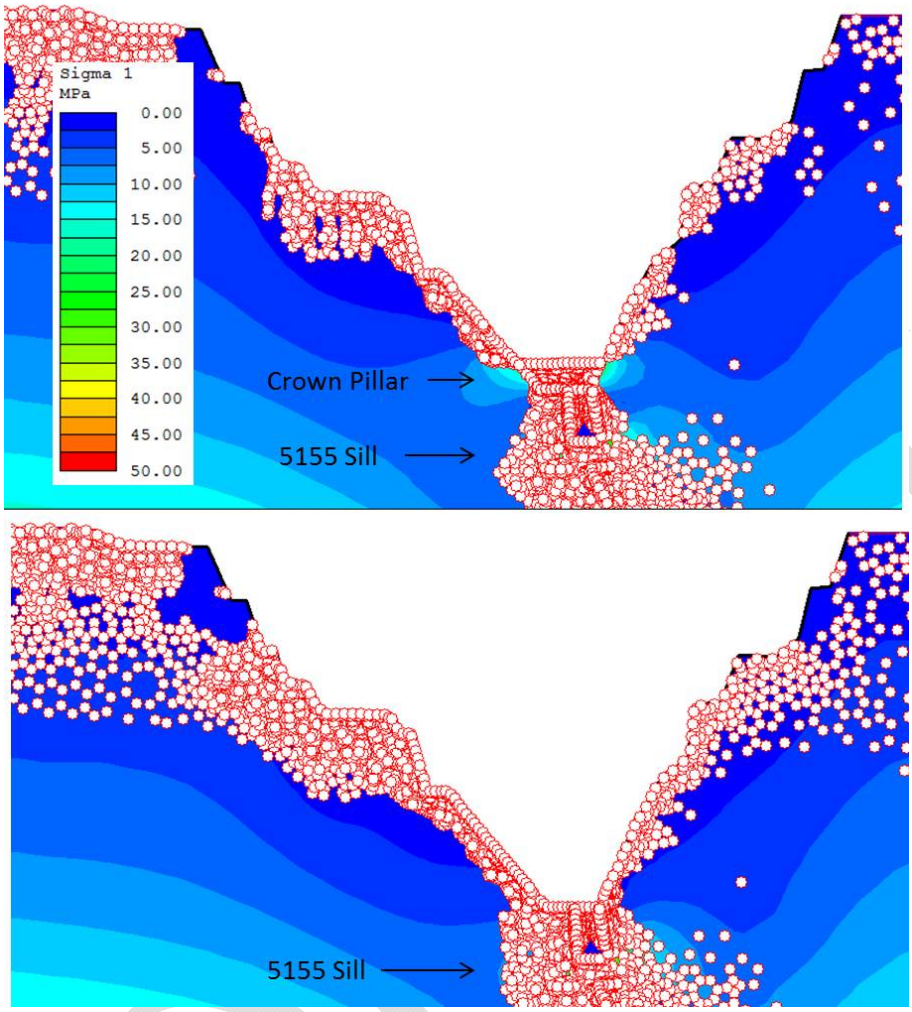

Figure 3: Tensile yield predicted (top) prior to and (bottom) post crown pillar mining.

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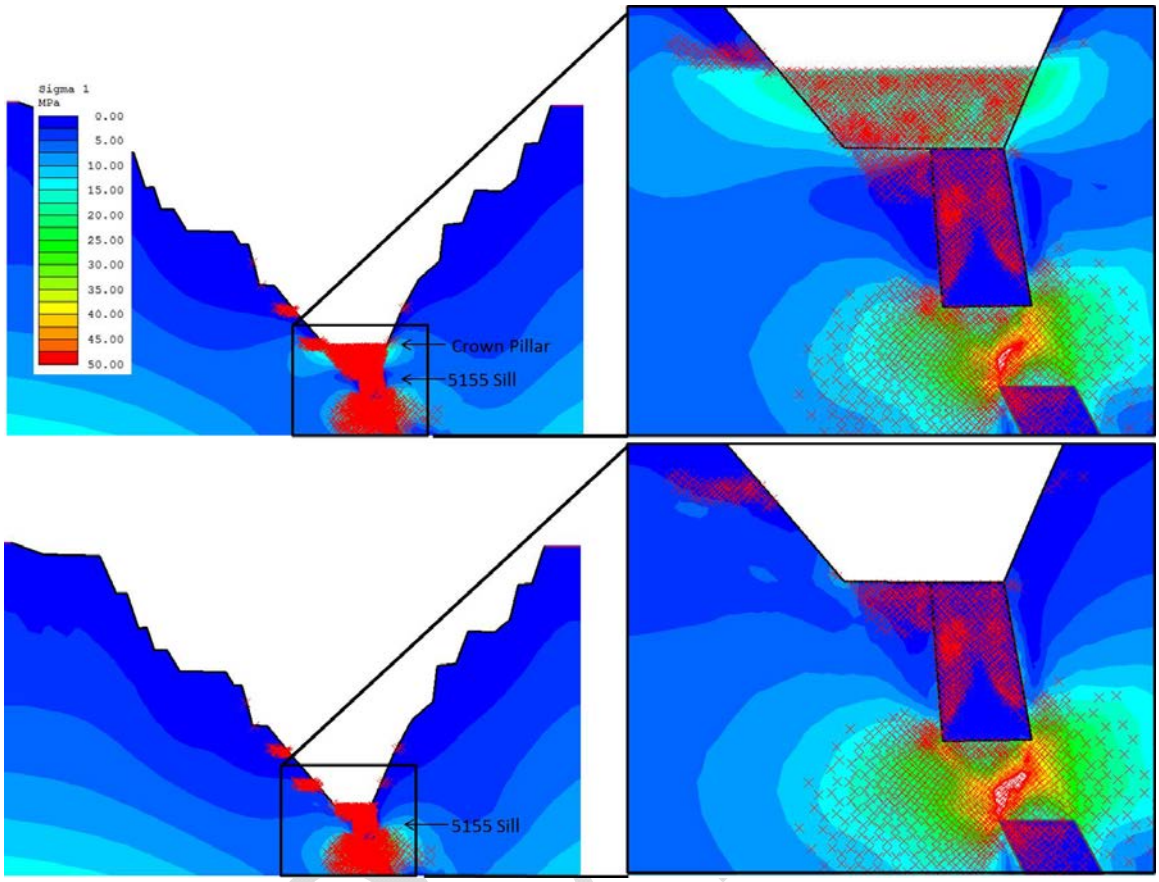

Figure 4: Shear yield predicted (top) prior to and (bottom) post crown pillar mining.

A Shear Strength Reduction (SSR) technique has been applied to numerical simulations. SSR iteratively adjusts material strength parameters by an SSR reduction factor until numerical non-convergence is achieved, which is indicative of instability. The critical SSR reduction factor can be taken as a factor of safety. The SSR reduction factor predicted for the west slope of the LDI open pit is 5.2 prior to crown pillar mining, and 5.0 after the crown pillar has been extracted. These results suggest that while there is an overall reduction in pit slope factor of safety, the factor of safety remains well above acceptable design limits (i.e SF = 1.5). Figure 5 shows contour plots of maximum shear strain when material strength parameters are reduced by the SSR reduction factor prior to and post crown pillar mining. The potential slip surfaces evident in this figure have low likelihood of developing given the high factor or safety.

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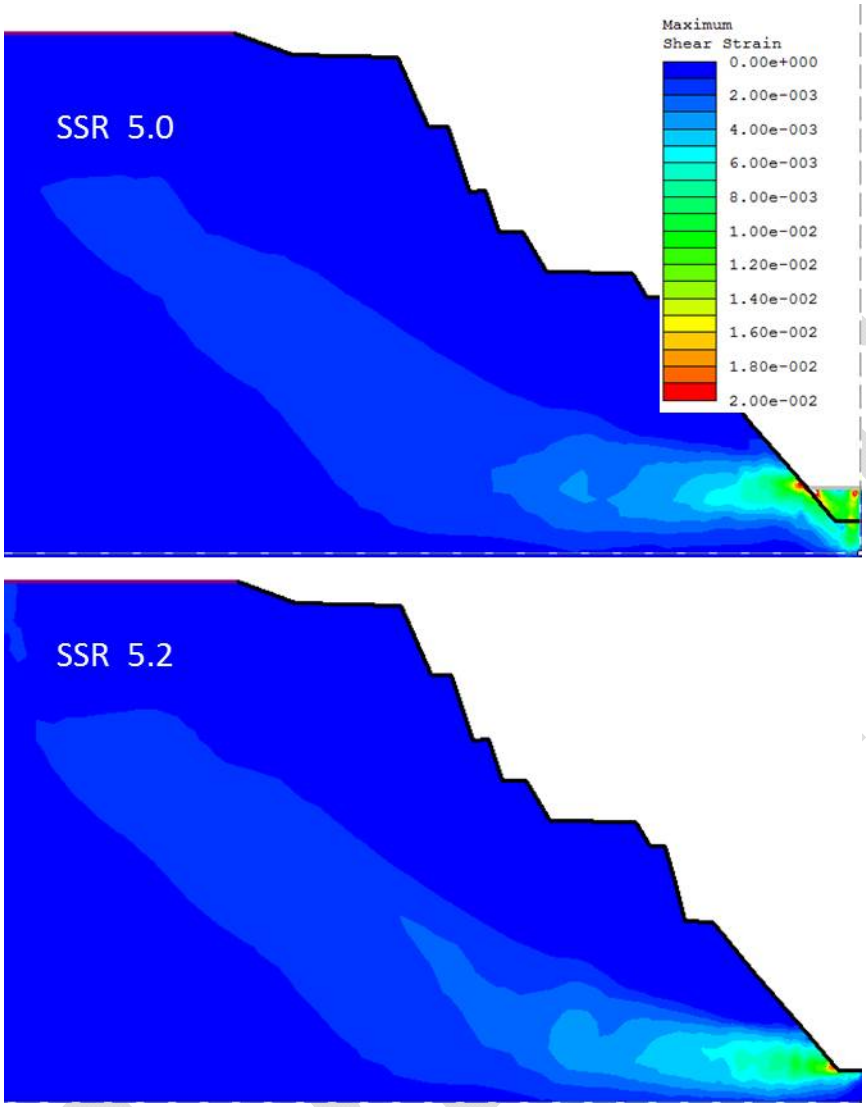

Figure 5: SSR plots showing contoured maximum shear strain (top) prior to and (bottom) post crown pillar mining.

4 Conclusions and Recommendations

Numerical modelling results suggest that excavation of the LDI crown pillar has negligible, negative impact on pit slope stability. Models suggest that the crown pillar rockmass has been yielded since initial pit excavation due to high principal stress magnitudes wrapping around the bottom of the pit. Underground mining of the Roby Zone had very little impact on crown pillar integrity. There is no significant, simulated change in the extent of pit slope yield, or the distribution of in situ stresses near the pit walls in response to crown pillar mining. Pit slope factor of safety is predicted to be well above acceptable design limits following the extraction of the crown pillar.

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It is noted that the 5155 sill pillar, a critical pillar which will ultimately separate the Roby mine workings from the overlying open pit, is likely to have already yielded. Mining of the crown pillar is not predicted to instigate further yield in that pillar. The LDI micro-seismic system is tied into sensors on 5210, 5180, 5155 and 5120. This will provide some coverage of the pillar area. Measured seismicity will be very valuable to track any new damage accumulation in the 5155 sill during crown pillar mining. Further, MPBX instruments located in the sill of #2 block on 5155 will be valuable for measuring ground deformations that may occur. It is strongly recommended that the functionality of these instruments and the micro-seismic system be checked prior to initiation of crown pillar mining. It is also recommended that all stopes below the 5155 sill be completely filled with rockfill prior to crown pillar extraction. Rockfill should be pushed tight to the stope backs to avoid the initiation and propagation of gravity driven sill pillar ravelling.

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5 References

Hoek, E., 2003. Analysis of the Stability of an anchor Block for a Suspension Bridge.

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