Preliminary Economic Assessment Technical Report Wellgreen Project, Yukon Territory, Canada

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This report entitled "Preliminary Economic Assessment Technical Report, Wellgreen Project, Yukon Territory, Canada", effective as of February 2, 2015 and dated March 18, 2015 was prepared and signed by the following authors:

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1 Executive Summary

1.1 Introduction and Summary Highlights

JDS Energy & Mining Inc. (JDS) was commissioned by Wellgreen Platinum Ltd. (Wellgreen Platinum) to conduct a preliminary economic assessment (this PEA or 2015 PEA) and technical report for the Wellgreen Platinum property (Wellgreen project or Property), an advanced platinum group metals (PGM), nickel, and copper project owned 100% by Wellgreen Platinum and located in southwest Yukon.

Two previous technical reports were prepared for the Wellgreen project pursuant to Canadian Securities Administrators' National Instrument 43-101 - *Standards for Disclosure for Mineral Projects* and Form 43-101F1 - *Technical Report* (collectively, NI 43 -101) and documenting a PEA and exploration work completed by Wellgreen Platinum on the project in 2012 and 2014. All technical reports were filed on SEDAR.

This technical report summarizes the results of the 2015 PEA study and was prepared following the guidelines of NI 43-101.

Highlights of the 2015 PEA:

- Average annual production of 208,880 ounces of platinum+palladium+gold (3E) (42% Pt, 51% Pd and 7% Au), along with 73 million pounds of nickel and 55 million pounds of copper over the first 16 years of operation at a production grade of 1.88 g/t platinum equivalent (Pt Eq.) or 0.50% nickel equivalent (Ni Eq.) (0.63 g/t 3E (46% Pt, 45% Pd and 8% Au), 0.27% Ni and 0.18% Cu), which equates to a net smelter return (NSR) of CAD$38.60 per tonne milled using the base case metal price assumptions set out below;
- Average strip ratio of 0.75 to 1 over the 25 year base case life of mine (LOM);
- LOM production to average 177,536 ounces of 3E (42% Pt, 51% Pd and 7% Au), 68 million pounds of nickel and 44 million pounds of copper per year over 25 years with the potential to add an additional 15 years using bulk underground mining or 31 years through additional open pit mining of Inferred Mineral Resources; and
- Total LOM production of 4.4 million ounces of 3E (42% Pt, 51% Pd and 7% Au), with 1.7 billion pounds of nickel and 1.1 billion pounds of copper in concentrate from approximately 34% of the current pit constrained Mineral Resource.

This preliminary economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Economic Highlights: (Unless otherwise noted, all dollar amounts in this PEA are in Canadian dollars (CAD$) and all figures with respect to the 2015 PEA reflect the Base Case. *Base Case metal price assumptions: US$1,450/oz Pt, US$800/oz Pd, US$1,250/oz Au, US$8.00/lb Ni, US$3.00/lb Cu, US$14.00/lb Co and US$0.90 = C$1.00*)

The 2015 PEA demonstrates potential robust economics that would position the Wellgreen project as one of the lowest cost PGM producers globally (see footnote 1 below), with all-in sustaining costs[1] of US$478 per ounce of 3E and US$5.96 per pound of Ni Eq. for base metals, on a co-product basis:

- Pre-tax net present value (NPV) of CAD$2.1 billion with a pre-tax internal rate of return (IRR) of 32.4%, and an after-tax NPV of CAD$1.2 billion with an after-tax IRR of 25.3% at a 7.5% discount rate;
- Average annual operating cash flow of CAD$338 million over the first 16 years and an average of CAD$301 million per year over the 25 year LOM;
- Initial capital expenditures of CAD$586 million (including contingencies in the amount of CAD$100 million) with expansion, sustaining and closure capital of CAD$964 million over the LOM;
- Payback of 2.6 years pre-tax and 3.1 years after taxes; and
- Total net smelter revenue of CAD$15.5 billion and operating cash-flow of CAD$7.5 billion over the LOM.

A detailed discussion of both the potential opportunities identified to enhance the base case economics and production levels for the project, as well as a detailed discussion of the various risks associated therewith can be found under Section 25 Interpretations and Conclusions.

1.2 Project Concept

The Wellgreen project is envisioned as a conventional open pit operation, with some selective higher grade underground mining. Milling would start at 25,000 tonnes per day (tpd) for the first five years of operation and then scale up to 50,000 tpd for an additional 20 years. Under the base case of the 2015 PEA, the mill would produce a bulk Ni-Cu-Co-PGM-Au concentrate through conventional sulphide flotation for shipping via existing deep sea ports south of the project in Alaska.

Mineralized mill feed material is planned to be mined mainly from a large open pit (383 Mt) with additional feed from an underground mine (9 Mt). The total planned mine life is approximately 25 years with 392 Mt of mineralized material mined and processed and 296 Mt of waste rock mined giving an overall strip ratio of 0.75 t of waste rock to 1 t of mill feed material.

Tailings, waste rock and mill feed stockpile facilities are planned to be placed near the open pit in purpose-built facilities.

Life of mine (LOM) concentrate production is estimated to be 9.7 Mt (dry) of a bulk Ni-Cu-Co-PGM-Au concentrate for shipment and refining through the port of Haines, Alaska.

Electrical power for the project is proposed to be generated on site with liquefied natural gas (LNG)-fueled generators.

[1] All-in sustaining costs are per payable ounce and use World Gold Council guidelines, which are non-GAAP measures that have no standardized meaning and may not be comparable to similar measures presented by other issuers.

1.3 Project Physical Description

The Wellgreen project is located approximately 317 km northwest of Whitehorse in southwestern Yukon, at an approximate latitude: 61°28'N and longitude: 139°32'W on NTS map sheet 115G/05 and 115G/06. The Wellgreen deposit is accessible by a 14 km road from the paved all-weather Alaska Highway to the north and east.

An all-weather airstrip is located approximately 15 km southeast of the Property at Burwash Landing. The airstrip is maintained by NAV CANADA and presently sees limited winter maintenance.

All-season, deep-sea ports are located in Haines, Alaska, approximately 400 km to the southeast, as well as Skagway, Alaska, which is currently utilized by Capstone Mining and Alexco Resources for the transport of mining concentrate material on bulk container ships to smelters. Both ports are year round ice free ports and are accessible by high-quality paved highways.

Work on the Wellgreen project can be conducted year-round. The regional climate is semi-arid, sub-arctic with relatively warm, dry summers and winters characterized by relatively dry, cold interior conditions, but tempered by west coast climate influences. The area lies in the rain shadow of the Saint Elias Mountains, with average annual total precipitation for the Burwash Landing station of 27.97 centimetre (cm) (11 inches) of which 19.2 cm (7.6 inches) typically falls as rain in summer and the remainder as snow in winter.

The Property is located in the Kluane Ranges, which are a continuous chain of foothills situated along the eastern flank of the Saint Elias Mountains. The topography across the Property is typical of the interior Yukon with slopes of 250 to 300 m, and the highest peaks exceed an elevation of 1,800 m. The main mineralized zone on the Property lies between an elevation of 1,250 m and 1,700 m on a moderate to steep south-facing slope.

The Property is comprised of 345 mineral claims in four groups totaling 5,933 hectares (ha). The claims were staked as early as 1952. Each claim is a Quartz Mining Claim with expiry dates that range from February 2015 to February 2032. The claims cover the known Wellgreen deposit as well as the Quill, Burwash and Arch properties. The Wellgreen deposit is located on 13 Quartz Mining Leases which all have an expiry date of December 5, 2020. The additional Wellgreen Platinum claims are located contiguous to the known deposit. The Wellgreen Platinum claims are 100% owned, directly or indirectly, by Wellgreen Platinum. Wellgreen Platinum's interest in the Property also consists of two surface leases covering 91.4 ha, which expire between 2022 and 2034.

The Property lies within the Kluane First Nation core area as defined by their treaty with Canada and the Yukon Government. An exploration co-operation agreement (ECA) was signed with Kluane First Nation August 1, 2012, and regular ECA meetings are held between the company and Kluane First Nation.

1.4 Project History, Exploration and Drilling

Prospectors W. Green, C. Aird and C. Hankins staked the first recorded mineral claims on the Property in 1952. Underground mining operations were initiated in 1971 with commercial production commencing in 1972 by Hudson Yukon Mining Co. Ltd. (Hudson Yukon Mining), a subsidiary of Hudson Bay Mining & Smelting Co. Ltd (HudBay). Production was suspended in 1973.

The Property was optioned to a joint venture between All-North Resources Ltd. (All-North) and Chevron Minerals in 1986 (Kluane JV) which acquired a 50% interest in the Property. That same year, Galactic Resources Ltd. purchased the Hudson Yukon Mining interest and net smelter returns royalty on the Property, and merged with All-North. In 1989, All-North purchased Chevron Minerals' 25% interest to acquire a 100% interest in the Property. Other joint ventures were formed on the Arch Property, which lies west of the Property.

In 1994, Northern Platinum Ltd. (Northern Platinum) acquired an 80% interest in the Property from All-North, with the remaining 20% purchased by Northern Platinum in 1999. Coronation Minerals Ltd. optioned the Property in 2005, but dropped the option in 2009. As a result, the Property was returned to Northern Platinum.

Prophecy Resource Corp. purchased Northern Platinum near the end of 2010. The Property and other nickel assets were spun out to Pacific Coast Nickel Corp, which then changed its name to Prophecy Platinum Corp. in June 2011. Prophecy Platinum Corp. changed its name to Wellgreen Platinum Ltd. in 2013.

The sample database supplied for the Property contains results from 776 surface and underground drill holes completed on the Property since 1952. Prior to 2006, drill core was selectively sampled in areas considered to have economic potential based on visual logging. Wellgreen Platinum assayed non-sampled intervals from the 1987-1988 drill programs in 2013 and re-assayed intervals that had been previously analyzed.

Wellgreen Platinum continues to conduct exploration and development activities at the Property, such as drilling surface exploration drill holes into identified targets that have the potential to increase the size of the resource and to enhance Wellgreen Platinum's understanding of the deposit.

1.5 Geology & Mineralization

The Wellgreen deposit occurs within, and along the lower margin of, an Upper Triassic ultramafic-mafic body, within the Quill Creek Complex. This assemblage of mafic-ultramafic rocks is 20 km long and closely intrudes along the contact between the Station Creek and Hasen Creek formations. The main mass of the Quill Creek Complex, the Wellgreen and Quill intrusions, is 4.7 km long and up to 1,000 m wide.

Mineralization on the Property occurs within the Quill Creek Complex, a layered intrusion which gradationally transitions from Dunite to Peridotite to Pyroxenite to Clinopyroxenite to Gabbro with a corresponding increasing sulphide content through this sequence toward contact with the Paleozoic sedimentary country rocks. Mineralization within the main Wellgreen deposit has been delineated into six zones of massive and disseminated mineralization known respectively as the Far East Zone, East Zone, Central Zone, West Zone, Far West Zone, and North Arm Zone.

The mineralization at the Wellgreen project is similar to gabbro-associated nickel deposits such as those found in Noril'sk in Russia; Raglan in, Northern Quebec; Stillwater in Montana; and Sudbury, Ontario, though it is unusual in comparison with the width of continuous disseminated mineralization and total platinum group metals (PGMs) content.

Exploration drilling has defined a mineralized zone over a 2.8 km East-West trend. The deposit averages 100 to 200 m in thickness at surface in the Far West Zone, expands to 500 m in thickness in the Central Zone and to nearly 1 km wide in the Far East Zone where the deposit remains open down dip and along trend.

The main sulphide minerals associated with potentially economic mineralization at the Wellgreen project include pentlandite (nickel), chalcopyrite (copper), and cobaltite (cobalt). The PGMs platinum, palladium, rhodium, iridium, ruthenium, and osmium, along with gold, are included in sperrylite, merenskyite, sudburyite, and other lesser known minerals that are often associated with magnetite, pyrrhotite, chalcopyrite, and pentlandite.

1.6 Metallurgical Testing and Mineral Processing

The recoveries of metals to concentrate and concentrate grade assumptions used in this PEA are based on a combination of metallurgical testing programs conducted between 1988 and 2014. Laboratory scale testing in 2013 and 2014 was performed by SGS Lakefield Research Limited (SGS) and XPS Consulting & Testwork Services (XPS), a Glencore company, under the supervision of the Company's independent metallurgical Qualified person and consultant, John Eggert, P. Eng., of Eggert Engineering Inc. (Eggert) with review and consultation by Dr. David Dreisinger. These test programs evaluated the effect of factors such as grind size, pH, conditioning, the use of various collectors, flotation reagents, dispersants and depressants on mineral recoveries and concentrate grades, magnetic separation and modifications to the mineral processing flowsheet.

In mid-2014, XPS completed a historical review of the 1988 to 2014 metallurgical test reports with the Company and John Eggert, P. Eng., the Qualified Person for metallurgical performance and mineral processing for this PEA. The fundamental conclusions from the review were:

- A bulk concentrate was the optimum approach for the updated PEA; and
- Magnetic separation of the bulk float tail followed by a regrind/flotation cycle would improve Ni and PGM recovery.

The historical review determined that there were three geo-metallurgical domains which required consolidation of data and testing:

- Gabbro/Massive Sulphides – Highest sulphur and grade with lowest serpentine content;
- Pyroxenite/Clinopyroxenite – Moderate sulphur and grade with moderate serpentine content; and
- Peridotite/Dunite – Lowest sulphur and grades and with moderate to high serpentine.

One of the key observations from the XPS review was that the optimization of sulphide flotation recovery varied based on the three metallurgical domains noted above. In general, the recovery of economic metals is highest from the Gabbro/Massive Sulphide domain, followed by the Clinopyroxenite/Pyroxenite domain and then by the Peridotite/Dunite domain. As a result of this observation, Wellgreen Platinum's geological team developed a system for classifying these rock types and conducted considerable re-logging of historic core so that the resource model included these specific geological domains.

A review of historical metallurgical testing programs also indicated that the majority of that testing was conducted on material that would be considered part of the Gabbro and

Pyroxenite/Clinopyroxenite domains. Very little testing had been conducted on the Peridotite domain and little flowsheet optimization work had been conducted.

Testing has shown that the material from each domain can be processed in the same circuit with variances related to grind size, conditioning time, pH and the use of magnetic separation with the majority of reagent selection applied across all the domains. However, given the different metallurgical performance of the different geological domains, the mine plan in this PEA was designed so that higher grade material, which is estimated to be comprised of 99% from the Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite domains, is processed in the mill during the first 16 years of operation and lower grade material, which is estimated to contain about 24% of material from the Peridotite/Dunite domain, is stockpiled and processed after mining is completed in Year 17.

Analysis of concentrate tails in past metallurgical testing programs indicated that a significant amount of the PGMs, particularly platinum, was not being captured in the sulphide flotation process because it was finer-grained and associated with the magnetic minerals magnetite and pyrrhotite. Testing was conducted to evaluate the benefit of adding a magnetic separation process to the flowsheet. Magnetic separation is a proven technology utilized in many operating Ni-PGM mines. The magnetic separation process was successful in capturing additional PGMs, nickel and copper through regrinding of a modest volume of magnetic material followed by conventional flotation, particularly in the Clinopyroxenite/Pyroxenite and Peridotite domains. This material can then be combined with the main sulphide concentrate to improve overall primary flotation recoveries or a separate PGM concentrate.

Preliminary testing of various leaching methods conducted in 2014 indicates that a PGM concentrate or tails from the magnetic flotation may be amenable to additional secondary processing, potentially adding to the recovery of PGMs. Additional metallurgical testing will further evaluate secondary processing options.

Recovery-concentrate grade curves for each metallurgical domain have been developed for platinum, palladium, gold, nickel, copper and cobalt using data from 183 batch tests and 12 locked cycle tests (LCTs) on 26 representative samples. The recovery-concentrate grade curves used linear regression to generate an equation to calculate recovery to concentrate by metal for each metallurgical domain based on nickel concentrate grade. Analysis of the test results indicated that recoveries were typically higher in LCTs than in batch tests, so adjustments were made to the linear regression equations to adjust batch test results upwards to reflect recoveries that are expected to be achieved in future LCTs and pilot plant testing.

Table 1.1 provides the anticipated recoveries to bulk concentrate by geological domain for a bulk concentrate grading 6% nickel. On this basis, the concentrates produced through conventional sulphide flotation are anticipated to grade 6-10% nickel with 4-8% copper and 11-14 g/t combined precious metals (platinum, palladium and gold). Table 1.4 provides the 2015 PEA mill feed by geo-metallurgical domain and Table 1.3 provides the resulting concentrate grades and metal recoveries.

Table 1.1: Estimated Metal Recoveries by Geologic Domain

Geological Domain	Recovery to Bulk Concentrate[1]					
	Ni	Cu	Co	Pt	Pd	Au
Gabbro/Massive Sulphide	83%	95%	68%	75%	81%	70%
Clinopyroxenite/Pyroxenite	75%	88%	64%	59%	73%	66%
Peridotite/Dunite	68%	66%	55%	58%	58%	59%

Source: Eggert, 2014
[1] Recoveries are normalized to a bulk concentrate grade containing 6% nickel

Table 1.2: 2015 PEA Base Case Mill Feed Tonnage by Geo-Metallurgical Domain

Geological Domain	PEA Base Case	
	First 16 years	Life of Mine
Gabbro	11%	8%
Clinopyroxenite/Pyroxenite	88%	83%
Peridotite	1%	10%
Total Mill Feed*	**100%**	**100%**

Source: Eggert, 2014
* Totals may not add due to rounding

Table 1.3: PEA Concentrate Grades and Metal Recoveries

Concentrate Grades	Nickel		Copper		PGMs+Au	
	6-9%		4-8%		12-17 g/t	
PEA Recoveries	Ni	Cu	Co	Pt	Pd	Au
Life of Mine	75%	89%	64%	61%	72%	60%
Years 1-16	76%	90%	65%	62%	73%	60%

Source: Eggert, 2014

The metallurgical test work conducted to date has identified multiple opportunities that should be explored through future test programs:

- There may remain additional potential to improve metal recoveries to bulk concentrates with additional optimization testing;
- The potential for using secondary processing methods for recovering additional PGMs from the magnetic concentrate flotation tails and the cleaner flotation tail;
- Determine if a separate PGM concentrate can be generated; and
- Historical results indicate that total PGM grades could increase by approximately 10-25% if exotic PGMs such as rhodium, iridium and osmium are included. These exotic PGMs were recovered in concentrates by HudBay in the 1970s and consistently show up in the metallurgical test work.

1.7 Mineral Resource Estimates

The updated mineral resource estimate incorporates data derived from new drilling and the re-assaying and re-logging of and historic core re-assaying conducted since 2011, which totaled nearly 40,000 m. This data was used along with other available historical data, some of which was re-logged, to develop a geologic model for the Wellgreen deposit that incorporates lithology and uses wire frames that constrain massive sulphide mineralization and unmineralized zones. Block grades were estimated using the Inverse Distance cubed (ID^3) method and search parameters derived from variography and zone geometry.

Mineral resources are classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves.

Table 1.4 presents the mineral resource estimate for the Wellgreen project at a base case cut-off grade of 0.57 g/t Pt Equivalent or 0.15% Ni Equivalent).

Table 1.4: Mineral Resource at a 0.57 g/t PtEq or 0.15% NiEq Cut-Off

Category	Tonnes 000s	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	3E g/t	Ni Eq. %	Pt Eq. g/t
Measured	92,293	0.260	0.155	0.015	0.252	0.246	0.052	0.550	0.449	1.713
Indicated	237,276	0.261	0.135	0.015	0.231	0.238	0.042	0.511	0.434	1.656
Total M&I	329,569	0.261	0.141	0.015	0.237	0.240	0.045	0.522	0.438	1.672
Inferred	846,389	0.237	0.139	0.015	0.234	0.226	0.047	0.507	0.412	1.571

Source: GeoSim, 2014

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. with an effective date of July 23, 2014.
2. Measured mineral resources are drilled on approximate 50 x 50 m drill spacing and confined to clinopyroxenite and peridotite/dunite domains. Indicated mineral resources are drilled on approximate 100 x 100 m drill spacing except for the massive sulphide and gabbro domains which used 50 x 50 m spacing.
3. Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries. Ni Eq% = Ni%+ Cu% x 3.00/8.35 + Co% x 13.00/8.35 + Pt [g/t]/31.103 x 1,500/8.35/22.046 + Pd [g/t]/31.103 x 750/8.35/22.046 + Au [g/t]/31.103 x 1,250/8.35/22.046. Pt Eq [g/t] = Ni Eq/100×2204.62×8.35 / 1,500×31.103
4. An optimized pit shell was generated using the following assumptions: metal prices in Note 3 above ; a 45° pit slope; assumed metallurgical recoveries of 70% for Ni, 90% for Cu, 64% for Co, 60% for Pt, 70% for Pd and 75% for Au; an exchange rate of CAN$1.00=USA$0.91; and mining costs of $2.00 per tonne, processing costs of $12.91 per tonne, and general & administrative charges of $1.10 per tonne (all expressed in Canadian dollars).
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
7. 3E = Pt + Pd +Au

In addition, Table 1.5 shows the higher grade portion of the resource within the constrained pit at a 1.9 g/t Pt Eq. or 0.50% Ni Eq. cut-off.

Table 1.5: Mineral Resource at a 1.9 g/t PtEq or 0.50% NiEq Cut-Off

Category	Tonnes 000s	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	3E g/t	Ni Eq. %	Pt Eq. g/t
Measured	21,854	0.326	0.301	0.019	0.454	0.366	0.103	0.923	0.653	2.492
Indicated	50,264	0.334	0.286	0.019	0.455	0.373	0.090	0.919	0.653	2.493
Total M&I	72,117	0.332	0.291	0.019	0.455	0.371	0.094	0.920	0.653	2.493
Inferred	173,684	0.309	0.301	0.018	0.456	0.352	0.098	0.906	0.631	2.410

Source: GeoSim, 2014

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. with an effective date of July 23, 2014.
2. Measured mineral resources are drilled on approximate 50 x 50 metre drill spacing and confined to clinopyroxenite and peridotite/dunite domains. Indicated mineral resources are drilled on approximate 100 x 100 metre drill spacing except for the massive sulphide and gabbro domains which used a 50 x 50 metre spacing.
3. Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries. NiEq% = Ni%+ Cu% x 3.00/8.35 + Co% x 13.00/8.35 + Pt [g/t]/31.103 x 1,500/8.35/22.046 + Pd [g/t]/31.103 x 750/8.35/22.046 + Au [g/t]/31.103 x 1,250/8.35/22.046. Pt Eq[g/t] = Ni Eq/100×2204.62×8.35 / 1,500×31.103
4. An optimized pit shell was generated using the following assumptions: metal prices in Note 3 above ; a 45 degree pit slope; assumed metallurgical recoveries of 70% for Ni, 90% for Cu, 64% for Co, 60% for Pt, 70% for Pd and 75% for Au; an exchange rate of CAN$1.00=USA$0.91; and mining costs of $2.00 per tonne, processing costs of $12.91 per tonne, and general & administrative charges of $1.10 per tonne (all expressed in Canadian dollars).
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
7. 3E = Pt + Pd +Au

1.7.1 Key Assumptions/Basis of Estimate

The sample database supplied for the Wellgreen project contains results from 776 surface and underground drill holes completed on the property since 1952 (Table 1.6). Four holes drilled in 2005 were not sampled and lay outside of the present resource limits.

Table 1.6: Drilling Summary

Year	Operator	Surface Drilling		Underground Drilling		Combined Drilling	
		Holes	Metres	Holes	Metres	Holes	Metres
1952	Yukon Mining	18	1,981.64			18	1,981.64
1953	Yukon Mining	27	2,499.67	27	692.57	54	3,192.24
1954	Yukon Mining	2	192.63	159	3,939.65	161	4,132.28
1955	Hudson Yukon Mining			154	9,019.37	154	9,019.37
1956	Hudson Yukon Mining			38	1,903.70	38	1,903.70
1969	Hudson Yukon Mining	13	1,314.30			13	1,314.30
1971	Hudson Yukon Mining			80	2,482.83	80	2,482.83
1972	Hudson Yukon Mining			23	990.26	23	990.26
1987	All North / Galactic Resources	46	5,027.19			46	5,027.19
1988	All North / Chevron	37	6,049.66	34	5,571.20	71	11,620.86
2001	Northern Platinum	6	530.04			6	530.04
2006	Coronation Minerals	11	2,016.87			11	2,016.87
2007	Coronation Minerals			3	576.99	3	576.99
2008	Coronation Minerals	13	4,654.62			13	4,654.62
2009	Northern Platinum	10	2,051.75			10	2,051.75
2010	Northern Platinum	7	2,254.77			7	2,254.77
2011	Wellgreen Platinum	6	1,925.12			6	1,925.12
2012	Wellgreen Platinum	22	5,566.20	29	5,416.91	51	10,983.11
2013	Wellgreen Platinum	27	2,792.93			16	2,792.93
Totals		**245**	**38,857.39**	**547**	**30,593.48**	**792**	**69,450.87**

Source: GeoSim Services Inc., 2014

Prior to 2006, drill core was selectively sampled in areas considered to have economic potential based on visual logging. In 2013, Wellgreen Platinum extensively re-logged historic core totaling 21,784 m from the Property to update the geologic model with new information. The Company assayed all available ultramafic intervals that had not been previously sampled. Where samples were available, Wellgreen re-assayed the historic intervals that had been previously analyzed, particularly from the 1987-1988 era drilling.

1.7.2 Geological Models

Lithologic wireframe models were created by Wellgreen Platinum geologic staff based on sectional geology interpretations. For the resource modeling, the dunite, peridotite, pyroxenite and clinopyroxenite were treated as a single domain for geostatistics with the gabbro/massive sulphide material confined to a separate domain. Historically, material that was not massive sulphide or gabbro was classified under the field term ‘Peridotite’. The sub-domains were created subsequent to grade estimation based largely on grade distribution and estimated ultramafic content, which include clinopyroxenite to pyroxenite to peridotite to dunite. The dunite material had 0.1% nickel deducted from the grade as an estimate of potential nickel silicate content which eliminated nearly all of this material from the resource estimate.

1.7.3 Mineral Reserve Estimates

Measured, Indicated and Inferred resources were used in the life-of-mine (LOM) plan and Inferred mineral resources represent approximately 50% of the material planned for processing. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study (PFS) or a feasibility study (FS) of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the Wellgreen project.

1.8 Mining

The Wellgreen deposit is amenable to large scale open pit mining with portions of high grade zones at depth having potential for extraction by underground mining methods.

1.8.1 Open Pit

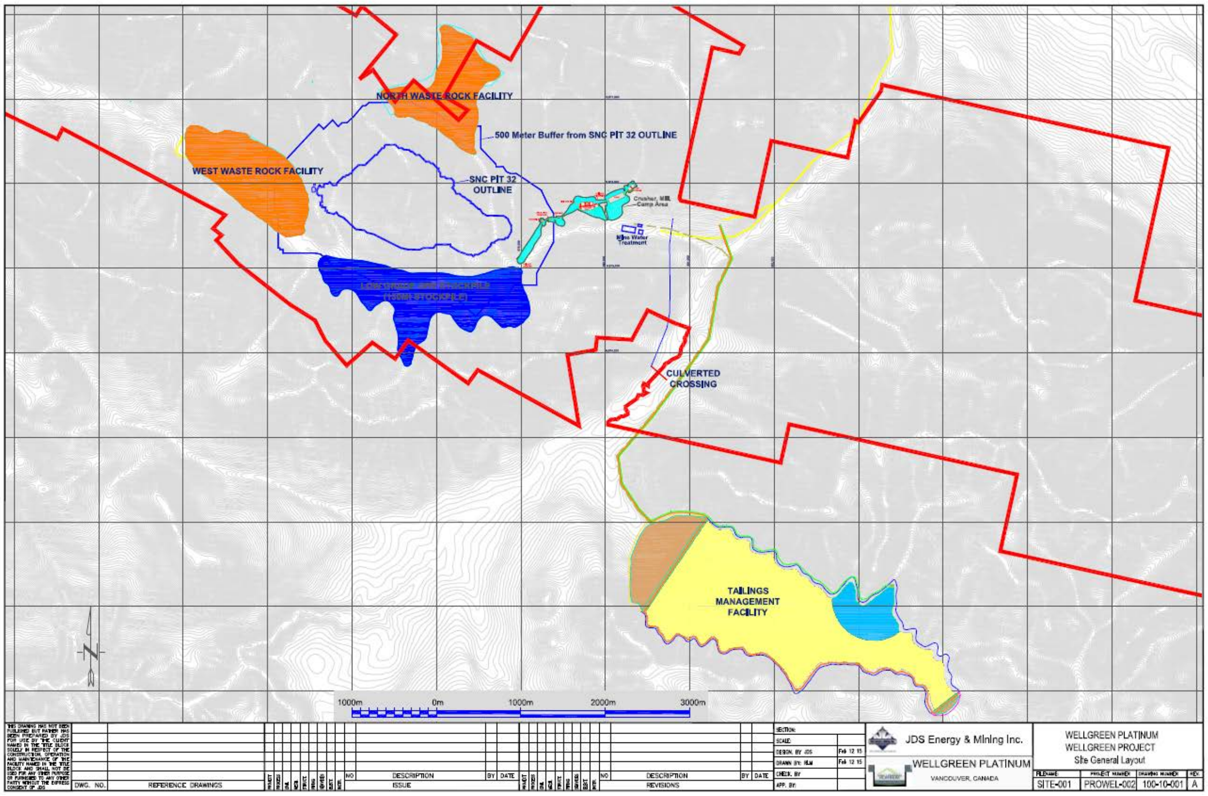
SNC-Lavalin Inc. (SNC) evaluated the open pit potential of the Property at a mill feed rate of 25,000 t/day increasing to 50,000 t/day in Year 6. The ultimate pit for the 2015 PEA base case is scheduled to be phased into four preliminary pushbacks. Mining cut-offs and stockpiling grades would be established for each pushback to target higher-grade mill feed.

Mill feed is planned to be hauled directly to the crusher and low grade material would be hauled to the long term stockpile and processed at the end of the mine life. Waste rock is planned to be hauled to the 1540 dump and the tailings management facility (TMF).

The pre-stripping period is scheduled to be one year in duration and provides the necessary construction materials for the tailings dam and other surface infrastructure facilities.

The general mine layout is shown in Figure 1.1.

Figure 1.1: Mine Site Layout



Source: JDS, 2015

See Figure 18.1 for full size rendition of the Mine Site Layout.

1.8.2 Pit Optimization

Pit optimization was completed with Whittle software. Optimized pit shells were generated with the Lerch-Grossman algorithm and variable revenue factor method. From this the optimized pit shell was selected.

1.8.3 Pit Optimization Parameters

A summary of the parameters are provided in Table 1.7.

Table 1.7: Pit Optimization Parameters

Item*	**Unit**		**Value**	
Exchange Rate	US$:C$		0.91	
Discount Rate	%		7.5	
Metal Prices				
Platinum	US$/troy oz		1,500	
Palladium	US$/troy oz		750	
Gold	US$/troy oz		1,250	
Nickel	US$/lb		8.35	
Copper	US$/lb		3	
Cobalt	US$/lb		13	
Metal Recoveries	**Unit**	**Gabbro/MS**	**Clinopyroxenite/ Pyroxenite**	**Peridotite**
Platinum	%	74.5	59.0	57.6
Palladium	%	80.5	73.0	58.4
Gold	%	69.8	65.8	58.8
Nickel	%	83.0	75.0	68.1
Copper	%	94.5	88.3	66.3
Cobalt	%	67.9	64.4	54.9
Mining Cost	$/tonne	2.20 + Db*0.005	Db = Difference in 10 m benches	
Processing Cost	$/tonne	13.11		
G&A	$/tonne	1.85		
Mining Recovery	%	99		
Mining Dilution	%	4		
Overall Pit Slope	degrees	40		
Mill throughput	t/day	25,000		
Shipping Cost	US$/t	123		
Bulk Con Ni%	%	6		
Smelting	$/t Con	175		
Payable	%	50-95		
Refining	$/unit	0.4 -15.0		
Deductions	g/t	0.5 - 5.0		

Source: SNC, 2015

*These parameters may vary from estimates used elsewhere in the report as they were preliminary in nature and further refined as the study progressed.

1.8.4 Ultimate Pit Design

Pit designs were completed with Hexagon MineSight 3-D software.

Fifty-one pit shells were generated with a variable revenue factor. Based on optimization results, pit shell 32 (inclusive of the 4 pit stages) was selected as the guide for the ultimate pit design for the 2015 PEA base case, the results of which are provided in Table 1.8. Dilution and mining recovery were based on analysis of similar operations and assumed to be 4% and 98%, respectively.

The ultimate design and pushbacks are preliminary and, therefore, do not include ramp access in the design.

Table 1.8: PEA Base Case Open Pit Results

Rock	Pt Eq g/t	Mt	Ni%	Cu%	Co%	Pt g/t	Pd g/t	Au g/t
Measured	>0.6	69.2	0.25	0.16	0.02	0.259	0.243	0.054
Indicated	>0.6	123.6	0.26	0.13	0.01	0.221	0.235	0.039
Inferred*	>0.6	198.9	0.25	0.12	0.01	0.215	0.235	0.037
Total Mineralized Material	>0.6	391.7	0.25	0.13	0.01	0.225	0.236	0.04
Waste		296.2						

SNC, 2015

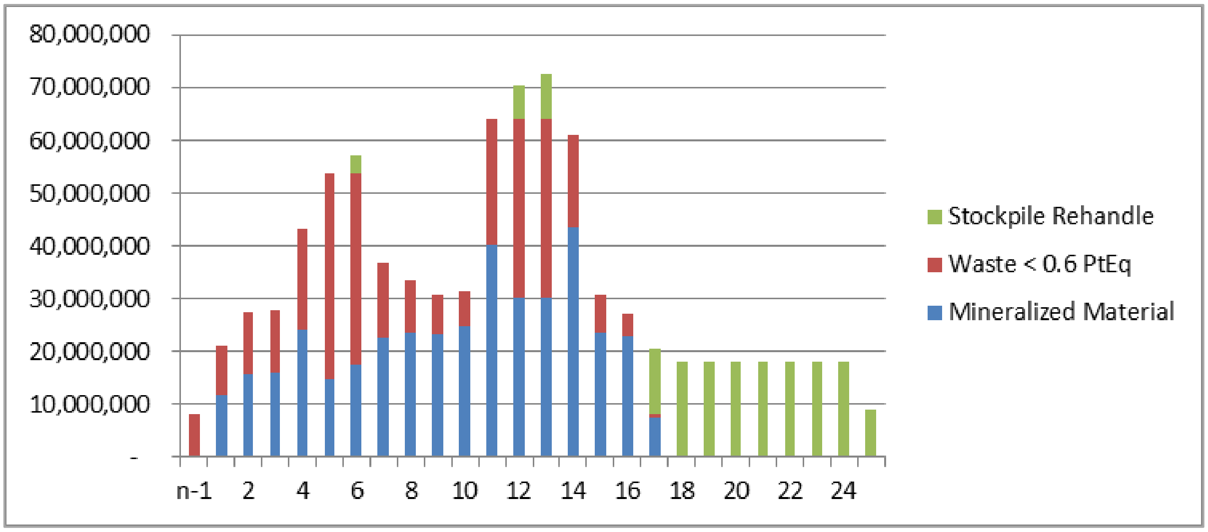
* Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them

1.8.5 Mining Schedule

The plant capacity is planned to commence with 25 kt/day for the first five years, then ramps up to 50 kt/day in year six and for the remainder of the LOM including processing of stockpiled mineralized material.

The pre-production period is scheduled to last for one year, mining 8.1Mt of material for construction of the tailings management facility (TMF). Mining operations for the base case are projected to last approximately 17 years followed by eight years of processing stockpiled material. The open pit mine production schedule is summarized in Figure 1.2.

Figure 1.2: Open Pit Mine Production Schedule



Source: SNC, 2015

In order to maintain a consistent open pit mobile fleet (and employee profile), contractor mining is planned on occasions due to significant stockpiling requirement and tailings storage facility expansion requirements. Contractors are planned to be utilized in years 4 through 6, and 11 through 14 when mining rates exceed 37.8 Mt/year. Contractor mining rates vary by year, but average 21.1 Mt/year over the seven years.

1.8.6 Underground Mining

The objective of the underground mine planning was to provide high grade mill feed early in the life of mine plan. The underground mining is planned to come from zones that would otherwise not be mined until late in the 2015 PEA base case mine plan or with the Stage 5 pit that is considered to be an opportunity in the 2015 PEA and is not part of the base case.

The underground mine design takes advantage of existing level development, ventilation and vertical development. The underground mine is scheduled to provide feed to the mill starting in year three of production with a relatively low capital requirement.

The current study reviewed the following four underground mining methods:

- Shrinkage mining: eliminated due to geotechnical concerns. These openings would affect open pit mining, which was scheduled to operate concurrently with the underground activities;
- Block caving: considered as an alternative to a Stage 5 open pit scenario;
- Open stoping with backfill: chosen for those blocks amenable to bulk mining; and
- Post pillar cut and fill: chosen for shallow dipping, high grade mineralization zones.

This study assumes that the lateral development and the post pillar cut and fill production mining would be completed by one contractor who would provide his own mobile equipment. This contractor would also be responsible for the remote mucking of the open stope. A second contractor is planned to be used for drilling and blasting the open stopes and installing the ground support cable bolting. The second contractor would be required to provide his own mobile equipment and grouting pumps.

1.9 Recovery Methods

The current project plan begins with a 25,000 t/d nominal mill utilizing conventional crushing. Crushing is planned to be in three stages with a primary gyratory crusher, a secondary cone crusher and a tertiary cone crusher in closed circuit with a screen. The circuit would produce a feed for two single stage ball mills operating in parallel.

Metal recovery is designed to be by bulk flotation followed by concentrate regrind and cleaning. In addition, a magnetic separation circuit on the rougher flotation tailings, followed by regrind and flotation cleaning would be used. A final bulk concentrate for sale planned to be produced. Regrind is proposed to be done by small ball mills or alternately stirred media mills. Concentrate for sale would be thickened, filtered and trucked off site. Tailings would be thickened and pumped to the tailings management facility.

In the sixth year, mill capacity is scheduled to be doubled to 50,000 t/d. The recovery process would remain the same. Increased capacity would be accomplished by twinning most of the circuit.

There are three tailings streams in the flowsheet; the magnetic tailings, the magnetic flotation tailings and the sulphide flotation tailings. There is potential for further processing of the latter two streams.

1.10 Project Infrastructure

Access to the project is planned via an upgraded existing 14 km access road off of the paved Alaska Highway. The general site layout is designed with two Phases. During Phase 1, the 25,000 t/d production phase, a 32 megawatt (MW) liquefied natural gas (LNG) fired power plant with three days fuel storage capacity would be constructed. An approximate pad area of 220,000 m^2 is planned for the power plant, LNG storage, camp, process plant, screening building, crushing building, stockpile, primary crusher and all associated conveyors.

Major building installations are planned to include a 7,500 m^2 process plant, a 450 m^2 maintenance shop warehouse, a 1,200 m^2 truck shop, a bulk explosives storage facility and two 85,000 litre (L) bulk fuel tanks. A 630,000 L combination fresh/firewater tank is planned to supply sufficient fire protection and fresh water to the plant. Potable water and waste water treatment systems would be included with the camp. The Phase 1 construction camp is planned to provide capacity for 580 people. A permanent operations camp with 250 person capacity is proposed to also be installed and remain in operation over the entire LOM.

Phase 2 is planned to include the following infrastructure components to increase production to 50,000 t/d:

- An additional 27 MW LNG fired power plant;
- Additional LNG storage farm with 4 – 60,000 gallon storage tanks;
- Additional LNG filling/dispensing system;

- New process building containing grinding mills and rougher flotation;
- Duplicate screening building;
- Secondary and tertiary crushing building extension;
- Fresh/Firewater tank extension; and
- Process water tank extension.

Tailings are designed to be placed in a conventional tailings management facility designed to store an ultimate capacity of approximately 402 Mt of tailings.

1.11 Environmental Studies

Baseline environmental studies have been commissioned to fulfill the requirements of an Executive Committee Screening of YESAB. The work being conducted will have added focus on a list of values identified through workshops with the relevant regulatory bodies and Kluane First Nation. Completion of the baseline studies is anticipated to take one field season for the purposes of the YESAB submission. Some data collection will be ongoing including but not limited to hydrology, hydrogeology and weather.

Environmental monitoring programs will be required through the life of the project and reclamation and closure period.

1.12 Production Schedule

An annual production schedule was developed for the project and is shown in detail in Section 16. Table 1.9 summarizes the production plan for the first 16 years of mining and the LOM.

Table 1.9: Production Summary

Item	Unit	Years 1-5	Years 6-16	Years 17-25 (Stockpiles)	LOM Value
Open Pit Mine Life	Years	5	11	0.3	16.3
Underground Mine Life	Years		6	0	6
Mineral Processing Life	Years	5	11	9	25
Total Mill Feed Material	M tonnes from open pit	42	194	155	392
	M tonnes from underground	6.9	2.6	0	9.5
Stockpiled Material	M tonnes	40	108	-149	0
Total Waste	M tonnes	91	196	1	296*
Total Material Mined	**M tonnes**	180	501	8	697*
Strip Ratio	waste: mineralized material	1.0	0.64		0.75
Processing Rate	t/d	25,000	50,000	50,000	45,000 avg.
	M tpa	9.1	18.3	18.3	16.4 avg.
Average Head Grades					
Nickel	%	0.32	0.27	0.21	0.26
Copper	%	0.31	0.15	0.08	0.14
Cobalt	%	0.02	0.01	0.01	0.01
Platinum	g/t	0.434	0.259	0.143	0.234
Palladium	g/t	0.346	0.271	0.173	0.241
Gold	g/t	0.087	0.045	0.025	0.042
Payable Metal					
Ni	M lbs	213.4	802.6	479.3	1,495.3
	Avg M lbs/yr	42.7	73.0	53.3	59.8
Cu	M lbs	246.4	531.8	199.7	977.9
	Avg M lbs/yr	49.2	48.3	22.2	39.1
Co	M lbs	2.2	14.0	12.2	28.4
	Avg M lbs/yr	0.4	1.3	1.4	1.1
Pt	k oz	328.5	817.1	328.1	1,473.8
	Avg oz/yr	65.7	74.3	36.5	59.0
Pd	k oz	301.0	1,023.5	483.8	1,808.3
	Avg oz/yr	60.2	93.0	53.8	72.3
Au	k oz	21.4	22.9	2.6	46.9
	Avg oz/yr	4.3	2.1	0.3	1.9
Concentrate Production					
Bulk Concentrate	k dmt	1,766	5,232	2,724	9,722
	Avg k dmt/yr	353	476	303	389

Source: JDS, 2015

* Includes 8M tonnes of waste pre-stripped in year -1

Totals may not add due to rounding

1.13 Marketing

This report does not include an independent concentrate marketing study. The marketing of bulk Ni-Cu concentrates is highly variable, depending on prevailing market conditions. The assumptions used in the study economics are shown in Table 1.10 and are based on information gathered from published feasibility studies, existing contracts and informal discussions with concentrate marketing specialists and are believed to be reasonable for the 2015 PEA. Additional work will need to be done to assess market terms for the Wellgreen concentrate in future studies.

In an environment of depressed metal demand, there is a possibility that the assumed smelter terms shown in Table 1.10 could be too optimistic; this could have a detrimental impact on the project's key performance indicators, including the economic viability of the Wellgreen project. In an environment where there is a deficit of nickel sulphide feed to smelters, assumed smelter terms could improve. These possibilities are discussed in greater detail in Section 19 of this report.

Table 1.10: Smelter Term Assumptions

Bulk Concentrate	Unit	Assumptions
Average LOM Concentrate Grades		
Nickel	%	8.0
Copper	%	5.2
Cobalt	%	0.4
Platinum	g/t	5.9
Palladium	g/t	7.2
Gold	g/t	1.0
Moisture Content	%	8
Smelter Parameters		
Payables (subject to a minimum deduction as per below)		
Nickel	%	90
Copper	%	88
Cobalt	%	50
Platinum	%	80
Palladium	%	80
Gold	%	80
Minimum Deductions		
Nickel	%	1
Copper	%	0.25
Cobalt	%	0.25
Platinum	g/t	1
Palladium	g/t	1
Gold	g/t	1
Treatment & Refining Charges		
Bulk concentrate treatment charge	US$/DMT	225
Nickel refining	US$/lb Ni	0.65
Copper refining	US$/lb Cu	0.4
Cobalt refining	US$/lb Co	3
Platinum refining	US$/oz Pt	15
Palladium refining	US$/oz Pd	15
Gold refining	US$/oz Au	15
Freight & Marketing Charges		
Truck Freight	US$/wmt conc	43.48
Ocean Freight	US$/wmt conc	60
Port charge	US$/wmt conc	13
Survey, Umpire	US$/wmt conc	3.2
US Customs	US$/wmt conc	1.85
Total Freight & Marketing	US$/wmt conc US$/dmt conc	121.53 132.1
Insurance	US $/$1K value	0.495

Source: JDS, 2015

The Base Case pricing used in the economic analysis was derived based on a combination of spot prices, three-year trailing average monthly prices, long-term consensus analyst forecasts, and a review of the price assumptions used by peer group companies in recent economic analyses. In addition to the Base Case scenario, the economic analysis also evaluated spot, peer study average and long term consensus forecast metal price scenarios. The metal prices are shown in Table 1.11.

Table 1.11: Metal Price and Foreign Exchange Rate Used in Economic Analysis Scenarios

Parameter	Units	PEA Base Case	Peer Study Prices[1]	Long Term Consensus Forecast[2]	Spot Feb. 2, 2015
Nickel	US$/lb	8.00	8.82	8.74	6.83
Copper	US$/lb	3.00	3.30	3.18	2.51
Cobalt	US$/lb	14.00	14.00	12.93	13.38
Platinum	US$/oz	1,450	1,661	1,450	1,223
Palladium	US$/oz	800	797	950	773
Gold	US$/oz	1,250	1,356	1,148	1,273
Exchange Rate[5]	C$/US$	0.900	0.900	0.877	0.800

[1] Mean price used by peers based on SEDAR filings over the past one year period
[2] Consensus analyst metal estimates for 2018 (2016 for cobalt) from Bloomberg, as at January 19, 2015
[3] FX based on 3-year average noon rates from the Bank of Canada on Jan. 19, 2015
Source: JDS, 2015

1.14 Capital Cost

Capital costs (CAPEX) were estimated from a combination of vendor quotes, first principles calculations, factored reference projects and experience. Table 1.12 shows the summary of the project's estimated CAPEX.

Table 1.12: Summary of Capital Cost Estimates

Capital Cost	Pre-Production (C$M)	Production (C$M)	LOM Total (C$M)
Mining Equipment	58.8	206.6	265.4
Pre-stripping	16.1	0.0	16.1
Site Development	36.8	0.0	36.8
Processing Plant	154.2	140.2	294.4
On-Site Infrastructure	89.7	53.4	143.2
Indirects	45.2	27.4	72.6
EPCM	30.2	16.3	46.4
Owner's Costs	9.6	0.1	9.7
Closure	0.0	60.0	60.0
Subtotal	**485.9**	**846.3**	**1,332.2**
Contingency	100.3	118.1	218.4
Total Capital Costs	**586.2**	**964.4**	**1,550.6**

Source: JDS, 2015

1.15 Operating Cost

Operating costs (OPEX) were estimated from a combination of vendor quotes, first principles calculations, factored reference projects and experience. Table 1.13 shows the summary of the project's estimated OPEX.

Table 1.13: Summary of Operating Costs

Operating Costs	C$/ milled	C$/ mined	Average C$M/Yr	LOM C$M
Open Pit Mining‡	3.65	2.10	58.7	1,466.3
Underground Miningº	1.29	0.74	14.6	516.2
Re-handle*	0.31	0.18	5.5	125.5
Processing	13.64	7.85	231.6	5,474.0
G&A	0.99	0.57	16.2	399.2
Total	**19.88**	**11.44**	**326.6**	**7,981.2**

Source: JDS, 2015
(‡) Open Pit Mining Costs are based on $2.13/t mined and a 0.8 strip ratio
(º) Underground Mining Costs are based on $54.49/t mined
(*) Re-handle cost is based on $0.75/tonne re-handled. Total material re-handled amounts to 167.3M tonnes over the life of mine.

1.16 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities to the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted that the tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations to represent an indicative value of the after-tax cash flows of the Wellgreen project.

Table 1.14 shows the summary of the economic results.

This preliminary economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

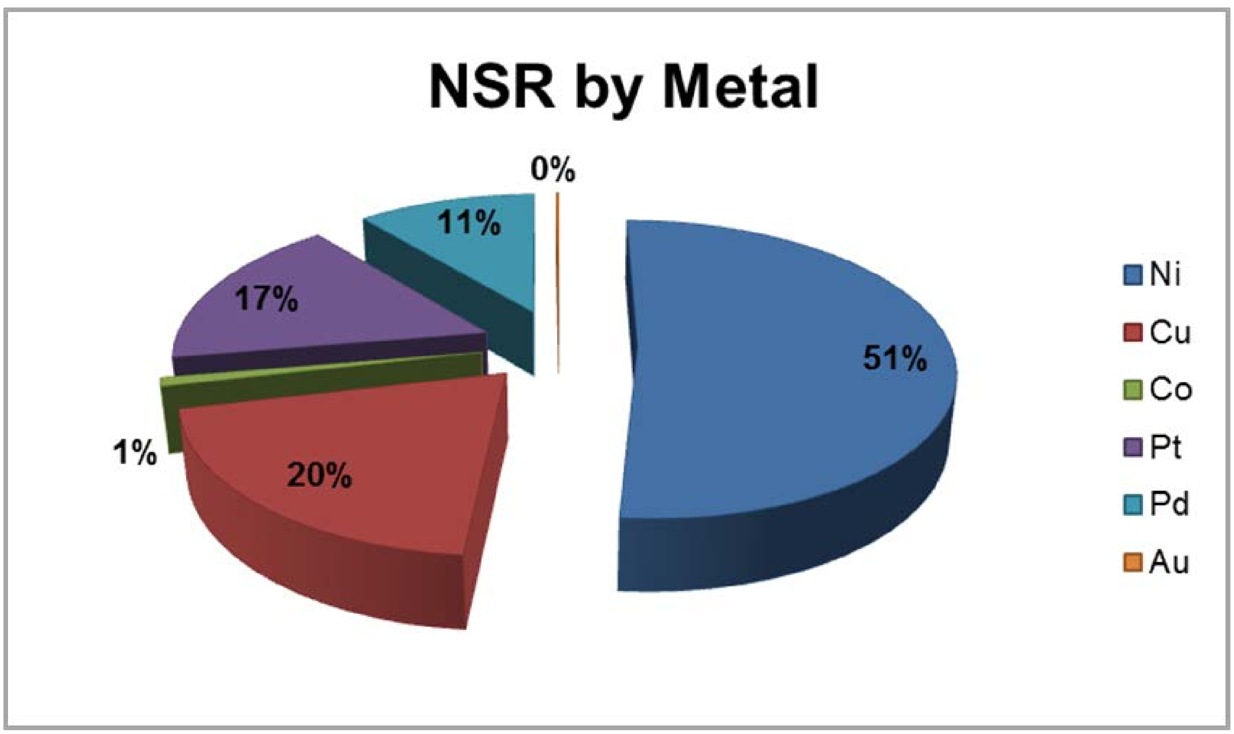
Table 1.14: Economic Results

Summary of Results	Unit	Base Case Scenario	Peer Base Case Prices	Long Term Consensus Forecast	Spot Prices as at Feb. 2, 2015
Nickel	US$/lb	8.00	8.82	8.74	6.83
Copper	US$/lb	3.00	3.30	3.18	2.51
Cobalt	US$/lb	14.00	14.00	12.93	13.38
Platinum	US$/oz	1,450	1,661	1,450	1,223
Palladium	US$/oz	800	797	950	773
Gold	US$/oz	1,250	1,356	1,148	1,273
Exchange Rate	C$/US$	0.900	0.900	0.877	0.800
Total LOM Pre-Tax Free Cash Flow	C$M	5,975.3	6,451.2	8,112.8	4,716.9
Average Annual Pre-Tax Free Cash Flow	C$M/Yr	239.0	258.0	324.5	188.7
LOM Income Taxes	C$M	2,265.4	2,447.5	3,085.1	1,786.0
Total LOM After-Tax Free Cash Flow	C$M	3,710.0	4,003.8	5,027.7	2,930.9
Average Annual After-Tax Free Cash Flow	C$M/Yr	148.4	160.2	201.1	117.2
Discount Rate	**%**	**7.5**	**7.5**	**7.5**	**7.5**
Pre-Tax NPV	**C$M**	**2,073.6**	**2,934.1**	**2,966.0**	**1,500.0**
Pre-Tax IRR	**%**	**32.4**	**41.6**	**41.5**	**25.8**
Pre-Tax Payback	**Years**	**2.6**	**2.0**	**2.0**	**4.4**
After-Tax NPV	**C$M**	**1,216.9**	**1,749.6**	**1,769.3**	**859.1**
After-Tax IRR	**%**	**25.3**	**32.1**	**32.1**	**20.4**
After-Tax Payback	**Years**	**3.1**	**2.3**	**2.4**	**6.2**

Source: JDS, 2015

The contribution by metal to the project economics are shown in Figure 1.3.

Figure 1.3: Life of Mine Net Revenues by Metal - Base Case Metal Prices



Source: JDS, 2015

1.17 Sensitivity Analysis

A sensitivity analysis was performed on the Base Case metal pricing scenarios to determine which factors most affect the project economics. The analysis revealed that the Wellgreen project is most sensitive to metal prices and foreign exchange rate, followed by head grade and operating costs. The project showed least sensitive to capital costs. Table 1.15 along with Figure 1.4 outline the results of the sensitivity test performed on the after-tax $NPV_{7.5\%}$ for the Base Case evaluated.

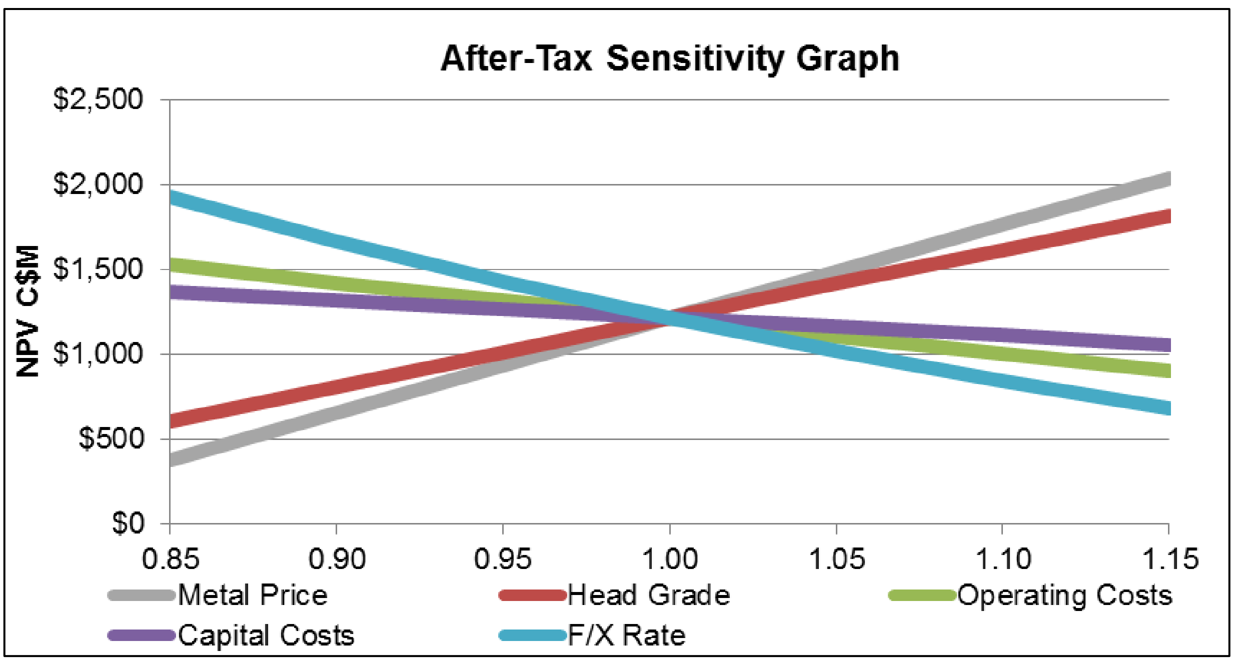
The Wellgreen project was also tested under various discount rates. The results of this sensitivity test are demonstrated in Table 1.16.

Table 1.15: Sensitivity Results for Base Case NPV

	After-Tax $NPV_{7.5\%}$ (C$M)						
Variable	**-15%**	**-10%**	**-5%**	**100%**	**+5%**	**+10%**	**+15%**
Metal Price	379	663	941	1,217	1,492	1,765	2,039
F/X Rate	1,928	1,665	1,430	1,217	1,024	848	686
Head Grade	606	811	1,014	1,217	1,419	1,620	1,821
OPEX	1,530	1,426	1,322	1,217	1,112	1,007	901
CAPEX	1,373	1,321	1,269	1,217	1,165	1,113	1,061

Source: JDS, 2015

Figure 1.4: Sensitivity Graph on Base Case Economic Results



Source: JDS, 2015

Table 1.16: Discount Rate Sensitivity Results on Base Case

Discount Rate	Pre-Tax NPV	After-Tax NPV
0%	5,975.3	3,710.0
5%	2,898.1	1,744.3
7.50%	2,073.6	1,216.9
10%	1,502.4	850.9
12%	1,167.6	636.0

Source: JDS, 2015

1.18 Interpretations and Conclusions

Industry standard mining and processing methods were used in this PEA. Sufficient information and data was available to the Qualified Persons (QPs) for a PEA-level study and the goal of producing a NI 43-101 compliant PEA study was achieved.

The preliminary economic results, based on the assumptions highlighted in this report, show a positive outcome.

It is important to note that this result is only preliminary and could change significantly as more information is gathered and market conditions change. This assessment includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The QPs of this report recommend that the Wellgreen project be advanced to a preliminary feasibility study level (PFS).

1.19 Risks, Opportunities and Recommendations

The most significant potential risks associated with the Wellgreen project are the ability to convert inferred resources to indicated and measured, geotechnical stability of pit walls and tailings facility, lower metal recoveries than those projected, the ability to produce a marketable concentrate, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal prices. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

External risks are, to a certain extent, beyond the control of the Wellgreen project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the Wellgreen project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates.

The most significant potential opportunities associated with the Wellgreen project are improved metallurgical recoveries by secondary processing and additional metallurgical & process testing, exotic PGM and silver credits, reduced waste mined with steeper pit walls, expansion of the mine life and production levels pit and block caving as an alternative to a phase 5 pit expansion or block caving, and possible connection to grid power.

JDS recommends that the project progress to a Pre-feasibility Study (PFS) level, with the necessary work conducted in two phases, and with Phase 2 contingent on the success of Phase 1.

The key areas for follow up work of Phase 1 of the pre-feasibility program in 2015 that JDS recommends Wellgreen Platinum pursue are listed below:

- Conduct initial drilling within the pit models designed to further upgrade Inferred Mineral Resources to Measured & Indicated Mineral Resources and test extensions of mineralization within the pit where it is unclassified, with the cost of such activities estimated to be $**[3.5 million]**;
- Implement additional metallurgical test programs in order to optimize recoveries from the main geo-metallurgical domains and conduct more detailed testing and assessment of potential secondary processing options, with the cost of such activities estimated to be $**[200,000]**;
- Commence evaluation of the cost and benefits of bringing the exotic PGMs such as rhodium, osmium, iridium and ruthenium into the mineral resource estimate, with the cost associated with such an evaluation estimated to be $**[100,000]**;
- Conduct additional geotechnical work to improve understanding of pit slopes and mine infrastructure, with the cost of such work estimated to be $**[200,000]**; and
- Conduct open pit trade-off studies, with the cost of such work estimated to be $**[100,000]**.

In aggregate, the total cost of Phase 1 of the PFS activities is estimated to be $4.1 million. If Phase 1 is successful, Wellgreen Platinum should consider pursuing Phase 2 of the PFS activities, which will be comprised of various activities such as drilling, sampling, assaying, geotechnical studies, metallurgical test work and engineering studies in order to further de-risk the Wellgreen project. It is estimated that the costs associated with completing Phase 2 may be in the range of $5 million to $10 million. However, a more definite estimate can by necessity only be made after Phase 1 is completed and a decision is taken by Wellgreen Platinum to pursue Phase 2.

2 Introduction

2.1 Basis of Technical Report

This Technical Report was compiled by JDS for Wellgreen Platinum. This technical report summarizes the results of the 2015 PEA study and was prepared following the guidelines of NI 43-101.

2.2 Scope of Work

This report summarizes the work carried out by the consultants and the scope of work for each company is listed below, and combined, makes up the total Project scope.

JDS scope of work included:

- Compile the technical report which includes the data and information provided by other consulting companies;
- Waste dump planning;
- Design required site infrastructure, identify proper sites, plant facilities and other ancillary facilities;
- Estimate process plant and infrastructure OPEX and CAPEX for the Project;
- Prepare a financial model and conduct an economic evaluation including sensitivity and Project risk analysis; and
- Interpret the results and make conclusions that lead to recommendations to improve value, reduce risks.

SNC scope of work included:

- Conduct pit optimization and mine planning and design;
- Select mining equipment;
- Establish potentially mineable resources; and
- Estimate mining OPEX and CAPEX.

SRK Consulting (U.S.) Inc. (SRK) scope of work included:

- PEA-level geotechnical assessment and estimate of appropriate overall pit slope angles.

Eggert scope of work included:

- Implement and supervise the metallurgical testing program;
- Develop a conceptual flowsheet, specifications and selection of process equipment;
- Establish recovery values based on metallurgical testing results; and
- Design processing to realize the predicted recoveries.

GeoSim scope of work included:

- Project setting, history and geology description; and
- Mineral resource estimate.

2.3 Qualifications, Responsibilities and Site Visits

The list of Qualified Persons is shown in Table 2.1.

Table 2.1: Qualified Persons

QP	Company	Report Section(s)	Site Visits
Michael Makarenko, P. Eng.	JDS	1 (except 1.4-1.9), 2, 3, 15, 18, 19-28	September 17-18, 2013
John Eggert, P.Eng.	Eggert	1.6, 1.9, 13, 17	Did not visit site
George Darling, P.Eng.	SNC	1.8, 16 (except 16.6)	Did not visit site
Mike Levy, P.E.	SRK	16.6	September 11-12, 2013
Ronald Simpson, P.Geo.	GeoSim	1.4-1.5, 1.7, 4-12, 14	September 17, 2013

Source: JDS, 2015

The Property is in an exploration stage and site visits by John Eggert, P. Eng. and George Darling, P. Eng. were not necessary to complete this PEA. They relied on information and knowledge from Wellgreen Platinum and JDS.

2.4 Units, Currency and Rounding

Unless otherwise specified or noted, the units used in this PEA are metric. Every effort has been made to clearly display the appropriate units being used throughout this PEA. Currency is in Canadian dollars (C$ or $).

This PEA includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Qualified Persons do not consider them to be material.

2.5 Sources of Information

The sources of information include data and reports supplied by Wellgreen Platinum personnel as well as documents cited throughout the report and referenced in Section 28. In particular, background Property information was directly taken from the 2014 Mineral Resource Estimate.

All tables and figures are sourced from JDS, unless otherwise indicated.

3 Reliance on Other Experts

The Qualified Person's opinions contained herein are based on information provided by Wellgreen Platinum and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

The following non-Qualified Person specialists were relied upon for specific advice:

- Wentworth Taylor, an Independent CA, for taxation information; and
- Loralee Johnstone – JDS Environment & Permitting Manager for environmental, permitting and First Nation information.

The tailings management facility sub-section 18.22 was provided by Knight Piésold (KP). Michael Makarenko, P. Eng., reviewed this sub-section and assumed responsibility for its content.

Mineral processing was written by John Eggert P.Eng. who assumed responsibility for the content. Mr. Eggert was assisted by Dr. David Dreisinger, P.Eng.

Underground mine planning was written by George Darling, P.Eng. who assumed responsibility for the content. Mr. Darling was assisted by Mr. Chris Paige who commented on underground mine planning.

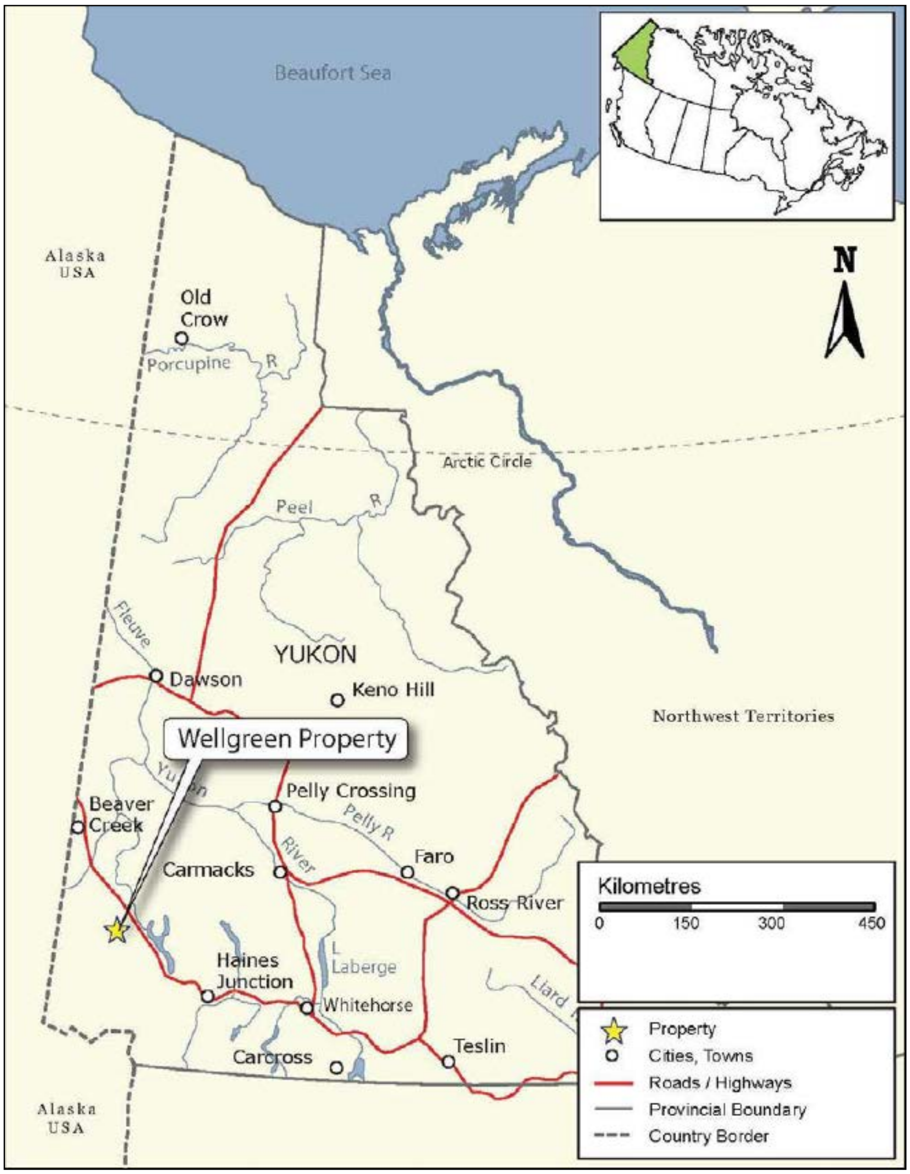
The Qualified Person's used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

4 Property Description and Location

4.1 Location

The Property is located approximately 317 km northwest of Whitehorse in southwestern Yukon, at an approximate latitude: 61°28'N, longitude: 139°32'W on NTS map sheet 115G/05 and 115G/06 (Figure 4.1). The Wellgreen project is accessible by a 14 km road from the paved all-weather Alaska Highway to the northeast. The Property lies within the Kluane First Nation core area as defined by their treaty with Canada and the Yukon Government.

Figure 4.1: Wellgreen Platinum Location Map



Source: GeoSim, Wellgreen Platinum, 2015

4.2 Tenure History

Prospectors W. Green, C. Aird and C. Hankins staked the first recorded mineral claims on the Property in 1952. Underground mining operations were initiated in 1972 by Hudson Yukon Mining, a subsidiary of HudBay and ceased in 1973. The Property has changed ownership several times over the last sixty years as outlined in Chapter 6. Wellgreen Platinum has had ownership of the Property since 2011.

4.3 Mineral Tenure

The description below and the list of claims provided in Table 4.1 have been derived from records and information supplied by Wellgreen Platinum and sourced from the Yukon Mining Recorder. A map of the Wellgreen project claims is shown in Figure 4.2.

The Property is comprised of 345 mineral claims in four groups totaling 5,933 ha. The claims were staked as early as 1952. Each claim is a Quartz Mining Claim with expiry dates that range from December 2015 to February 2032. The claims cover the known Wellgreen deposit as well as the Quill, Burwash and Arch properties. The Wellgreen deposit is located on 13 Quartz Mining Leases which all have an expiry date of December 5, 2020. The additional Wellgreen project claims are located contiguous to the known deposit. The Wellgreen project claims are 100% owned, directly or indirectly, by Wellgreen Platinum.

In the Yukon, all work undertaken on the surface for hard rock mineral claims and leases is regulated under the Quartz Mining Act (QMA) through the Quartz Mining Land Use Regulation and is managed by the Mining Recorder's Office.

A mineral claim is a parcel of land located or granted for hard rock mining. A claim also includes any ditches or water rights used for mining the claim, and all other things belonging to or used in the working of the claim for mining purposes. The holder of a mineral claim is entitled to all minerals found in veins or lodes, together with the right to enter on and use and occupy the surface of the claim for the efficient and miner-like operation of the mines and minerals contained in the claim. Continued tenure to the mineral rights is dependent upon work performed on the claim or a group of claims. Renewal of a quartz claim requires C$100 of work be done per claim per year. Where work is not performed, the claimant may make a payment in lieu of work.

A Quartz Mining Lease is the most secure form of mineral title in the Yukon. A lease is applied for when a company is contemplating production and would like to bring their claims to lease. This relieves the company of the annual work requirement – however there are annual rental fees of C$200 per lease. Quartz Mining Leases are issued for 21 years and can be renewed for an additional 21 year term, provided that during the original term of the lease, all conditions of the lease and provisions of the legislation have been adhered to.

Wellgreen Platinum's interest in the Property also consists of two surface leases issued by the Government of Canada and administered by the Government of Yukon: Lease 115G05-001 and 115G11-003, as described below and in Table 4.2.

Lease 115G05-001 covers a 69.7 ha parcel of land located near the headwaters of Nickel Creek proximal to the known Wellgreen deposit (Figure 4.3). Various operators have conducted historic exploration activities on this parcel of land since the 1950s, and exploration activities were carried out by Northern Platinum Ltd. (Northern Platinum) and Coronation Minerals Ltd. (Coronation

Minerals) since the late 1990s. Northern Platinum held a lease on this same area from the early 1990s until October 31, 2011. Prior to expiration, the 21-year lease was assigned to Prophecy Platinum Corp. (now Wellgreen Platinum), who then applied for renewal of the lease. This lease was renewed on June 1, 2013 and expires on May 31, 2034.

Lease 115G11-003 covers a 21.7 ha parcel of land located adjacent to kilometre 1728 on the Alaska Highway (Figure 4.3). This 10-year lease was granted on November 1, 2012 and expires on October 31, 2022. Northern Platinum held a similar but larger (62.7 ha) lease parcel from November 1, 2001 until October 31, 2011. This lease included the historic Hudson Yukon Mining mill site used in the 1970s as part of the Wellgreen project underground mining operation. Since the late 1990s, Northern Platinum used the old mill site for its core shack and as access to the Property. Pursuant to the requirements of the previous surface lease, which included the old mill site, Northern Platinum finalized a Reclamation Plan for the Mill Site, which was approved by the Government of Yukon in early 2010. Final accepted closure of the Reclamation Plan remains outstanding and is in discussion with the Government of Yukon.

Table 4.1: Mineral Claims

Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255471078	YA94968	BARNY 1	1	0905144 B.C. Ltd	21.77	11/02/2016
255436862	YA96005	BARNY 10	10	0905144 B.C. Ltd	21.33	11/02/2016
255480289	YA96006	BARNY 11	11	0905144 B.C. Ltd	21.45	11/02/2016
255374427	YA96007	BARNY 12	12	0905144 B.C. Ltd	20.97	11/02/2016
255395375	YA96008	BARNY 13	13	0905144 B.C. Ltd	18.56	11/02/2016
255275812	YA96009	BARNY 14	14	0905144 B.C. Ltd	17.43	11/02/2016
255386642	YA96867	BARNY 19	19	0905144 B.C. Ltd	21.40	11/02/2016
255368165	YA94969	BARNY 2	2	0905144 B.C. Ltd	20.91	11/02/2016
255372140	YA96868	BARNY 20	20	0905144 B.C. Ltd	21.55	11/02/2016
255439972	YA96869	BARNY 21	21	0905144 B.C. Ltd	21.28	11/02/2016
255439973	YA96870	BARNY 22	22	0905144 B.C. Ltd	21.46	11/02/2016
255281896	YA96871	BARNY 23	23	0905144 B.C. Ltd	22.38	11/02/2016
255364888	YA96872	BARNY 24	24	0905144 B.C. Ltd	22.20	11/02/2016
255482398	YA96873	BARNY 25	25	0905144 B.C. Ltd	10.01	11/02/2016
255303134	YA96874	BARNY 26	26	0905144 B.C. Ltd	17.26	11/02/2016
255237338	YA96875	BARNY 27	27	0905144 B.C. Ltd	17.67	11/02/2016
255244829	YA96876	BARNY 28	28	0905144 B.C. Ltd	17.86	11/02/2016
255374482	YA96877	BARNY 29	29	0905144 B.C. Ltd	17.61	11/02/2016
255368162	YA94970	BARNY 3	3	0905144 B.C. Ltd	21.30	11/02/2016
255238220	YA96878	BARNY 30	30	0905144 B.C. Ltd	8.90	11/02/2016
255343901	YA96879	BARNY 31	31	0905144 B.C. Ltd	13.52	11/02/2016
255343902	YA96880	BARNY 32	32	0905144 B.C. Ltd	20.44	11/02/2016
255286354	YA97896	BARNY 33	33	0905144 B.C. Ltd	5.83	11/02/2016
255401444	YA97897	BARNY 34	34	0905144 B.C. Ltd	12.61	11/02/2016
255307009	YA97898	BARNY 35	35	0905144 B.C. Ltd	17.53	11/02/2016
255466384	YA97899	BARNY 36	36	0905144 B.C. Ltd	15.97	11/02/2016
255445219	YA97900	BARNY 37	37	0905144 B.C. Ltd	17.73	11/02/2016
255341634	YA97901	BARNY 38	38	0905144 B.C. Ltd	11.22	11/02/2016
255319213	YA97902	BARNY 39	39	0905144 B.C. Ltd	11.49	11/02/2016

Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255376993	YA94971	BARNY 4	4	0905144 B.C. Ltd	20.27	11/02/2016
255298951	YA97904	BARNY 41	41	0905144 B.C. Ltd	19.04	11/02/2016
255488160	YA97905	BARNY 42	42	0905144 B.C. Ltd	14.77	11/02/2016
255286355	YA97906	BARNY 43	43	0905144 B.C. Ltd	13.13	11/02/2016
255307002	YA97908	BARNY 45	45	0905144 B.C. Ltd	14.80	11/02/2016
255466382	YA97910	BARNY 47	47	0905144 B.C. Ltd	15.04	11/02/2016
255219141	YA97911	BARNY 48	48	0905144 B.C. Ltd	9.37	11/02/2016
255214334	YA97912	BARNY 49	49	0905144 B.C. Ltd	12.96	11/02/2016
255267745	YA94972	BARNY 5	5	0905144 B.C. Ltd	21.28	11/02/2016
255321701	YB08307	BARNY 50	50	0905144 B.C. Ltd	5.32	11/02/2016
255297032	YA94973	BARNY 6	6	0905144 B.C. Ltd	20.66	11/02/2016
255345079	YA96002	BARNY 7	7	0905144 B.C. Ltd	21.86	11/02/2016
255259002	YA96003	BARNY 8	8	0905144 B.C. Ltd	14.28	11/02/2016
255265611	YA96004	BARNY 9	9	0905144 B.C. Ltd	21.82	11/02/2016
255417668	63029	BETTY 1	1	0905144 B.C. Ltd	10.38	05/12/2020
255417669	63030	BETTY 2	2	0905144 B.C. Ltd	11.58	05/12/2020
255202620	63031	BETTY 3	3	0905144 B.C. Ltd	11.83	05/12/2020
255353542	63032	BETTY 4	4	0905144 B.C. Ltd	10.93	05/12/2020
255273340	63033	BETTY 5	5	0905144 B.C. Ltd	18.41	05/12/2020
255305051	63034	BETTY 6	6	0905144 B.C. Ltd	17.59	05/12/2020
255374194	63035	BETTY 7	7	0905144 B.C. Ltd	19.50	05/12/2020
255239243	63036	BETTY 8	8	0905144 B.C. Ltd	21.20	05/12/2020
255448781	YC26564	BUR 1	1	Wellgreen Platinum Ltd.	20.90	23/02/2028
255341170	YC26573	BUR 10	10	Wellgreen Platinum Ltd.	20.90	23/02/2028
255470107	YC26574	BUR 11	11	Wellgreen Platinum Ltd.	20.91	23/02/2028
255365682	YC26575	BUR 12	12	Wellgreen Platinum Ltd.	20.90	23/02/2028
255287494	YC26576	BUR 13	13	Wellgreen Platinum Ltd.	20.90	23/02/2028
255208677	YC26577	BUR 14	14	Wellgreen Platinum Ltd.	20.90	23/02/2028
255204216	YC26578	BUR 15	15	Wellgreen Platinum Ltd.	20.86	23/02/2028
255311044	YC26579	BUR 16	16	Wellgreen Platinum Ltd.	20.90	23/02/2028
255311043	YC26580	BUR 17	17	Wellgreen Platinum Ltd.	20.88	23/02/2028
255449662	YC26581	BUR 18	18	Wellgreen Platinum Ltd.	20.88	23/02/2028
255390297	YC26582	BUR 19	19	Wellgreen Platinum Ltd.	20.86	23/02/2028
255444256	YC26565	BUR 2	2	Wellgreen Platinum Ltd.	20.92	23/02/2028
255297900	YC26583	BUR 20	20	Wellgreen Platinum Ltd.	20.90	23/02/2028
255235072	YC26584	BUR 21	21	Wellgreen Platinum Ltd.	20.86	23/02/2028
255330008	YC26585	BUR 22	22	Wellgreen Platinum Ltd.	20.90	23/02/2028
255333327	YC26586	BUR 23	23	Wellgreen Platinum Ltd.	20.86	23/02/2028
255361429	YC26587	BUR 24	24	Wellgreen Platinum Ltd.	20.90	23/02/2028
255425063	YC26588	BUR 25	25	Wellgreen Platinum Ltd.	20.86	23/02/2028
255420340	YC26589	BUR 26	26	Wellgreen Platinum Ltd.	20.90	23/02/2028
255420339	YC26590	BUR 27	27	Wellgreen Platinum Ltd.	20.90	23/02/2028
255432346	YC26591	BUR 28	28	Wellgreen Platinum Ltd.	20.90	23/02/2028
255212022	YC26592	BUR 29	29	Wellgreen Platinum Ltd.	20.90	23/02/2028
255407168	YC26566	BUR 3	3	Wellgreen Platinum Ltd.	20.90	23/02/2028
255239094	YC26593	BUR 30	30	Wellgreen Platinum Ltd.	20.90	23/02/2028

Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255261006	YC26594	BUR 31	31	Wellgreen Platinum Ltd.	20.90	23/02/2028
255314320	YC26595	BUR 32	32	Wellgreen Platinum Ltd.	20.90	23/02/2028
255252928	YC26596	BUR 33	33	Wellgreen Platinum Ltd.	20.90	23/02/2028
255392466	YC26597	BUR 34	34	Wellgreen Platinum Ltd.	20.90	23/02/2028
255391892	YC26598	BUR 35	35	Wellgreen Platinum Ltd.	20.90	23/02/2028
255305851	YC26599	BUR 36	36	Wellgreen Platinum Ltd.	20.84	23/02/2028
255420346	YC26600	BUR 37	37	Wellgreen Platinum Ltd.	20.90	23/02/2028
255432347	YC26601	BUR 38	38	Wellgreen Platinum Ltd.	20.90	23/02/2028
255212023	YC26602	BUR 39	39	Wellgreen Platinum Ltd.	20.90	23/02/2028
255408233	YC26567	BUR 4	4	Wellgreen Platinum Ltd.	20.90	23/02/2028
255239093	YC26603	BUR 40	40	Wellgreen Platinum Ltd.	20.90	23/02/2028
255261007	YC26604	BUR 41	41	Wellgreen Platinum Ltd.	20.90	23/02/2028
255314319	YC26605	BUR 42	42	Wellgreen Platinum Ltd.	20.90	23/02/2028
255252927	YC26606	BUR 43	43	Wellgreen Platinum Ltd.	20.90	23/02/2028
255392465	YC26607	BUR 44	44	Wellgreen Platinum Ltd.	20.90	23/02/2028
255391891	YC26608	BUR 45	45	Wellgreen Platinum Ltd.	20.93	23/02/2028
255305852	YC26609	BUR 46	46	Wellgreen Platinum Ltd.	20.90	23/02/2028
255305853	YC26610	BUR 47	47	Wellgreen Platinum Ltd.	20.90	23/02/2028
255199557	YC26611	BUR 48	48	Wellgreen Platinum Ltd.	20.90	23/02/2028
255213972	YC26612	BUR 49	49	Wellgreen Platinum Ltd.	20.90	23/02/2028
255194695	YC26568	BUR 5	5	Wellgreen Platinum Ltd.	20.90	23/02/2028
255213398	YC26613	BUR 50	50	Wellgreen Platinum Ltd.	20.90	23/02/2028
255361855	YC26614	BUR 51	51	Wellgreen Platinum Ltd.	20.90	23/02/2028
255263047	YC26615	BUR 52	52	Wellgreen Platinum Ltd.	20.90	23/02/2028
255372816	YC26616	BUR 53	53	Wellgreen Platinum Ltd.	20.90	23/02/2028
255343156	YC26617	BUR 54	54	Wellgreen Platinum Ltd.	20.90	23/02/2028
255191470	YC26618	BUR 55	55	Wellgreen Platinum Ltd.	20.90	23/02/2028
255265699	YC26619	BUR 56	56	Wellgreen Platinum Ltd.	20.90	23/02/2028
255265700	YC26620	BUR 57	57	Wellgreen Platinum Ltd.	20.90	23/02/2028
255424058	YC26621	BUR 58	58	Wellgreen Platinum Ltd.	20.90	23/02/2028
255186696	YC26569	BUR 6	6	Wellgreen Platinum Ltd.	20.90	23/02/2028
255186695	YC26570	BUR 7	7	Wellgreen Platinum Ltd.	20.89	23/02/2028
255188306	YC26571	BUR 8	8	Wellgreen Platinum Ltd.	20.90	23/02/2028
255313686	YC26572	BUR 9	9	Wellgreen Platinum Ltd.	20.88	23/02/2028
255415544	YB36423	BURWASH 1	1	Wellgreen Platinum Ltd.	20.90	23/02/2032
255278679	YC18485	BURWASH 10	10	Wellgreen Platinum Ltd.	17.35	23/02/2028
255433321	YC18486	BURWASH 11	11	Wellgreen Platinum Ltd.	3.55	23/02/2028
255447087	YC18487	BURWASH 12	12	Wellgreen Platinum Ltd.	20.90	23/02/2028
255256822	YC18488	BURWASH 13	13	Wellgreen Platinum Ltd.	20.90	23/02/2028
255380089	YC18489	BURWASH 14	14	Wellgreen Platinum Ltd.	20.90	23/02/2028
255380085	YC18490	BURWASH 15	15	Wellgreen Platinum Ltd.	20.90	23/02/2028
255231673	YC18491	BURWASH 16	16	Wellgreen Platinum Ltd.	20.89	23/02/2028
255310690	YC18492	BURWASH 17	17	Wellgreen Platinum Ltd.	20.90	23/02/2028
255419833	YC18493	BURWASH 18	18	Wellgreen Platinum Ltd.	20.90	23/02/2028
255215793	YC18494	BURWASH 19	19	Wellgreen Platinum Ltd.	20.90	23/02/2028
255224793	YB36424	BURWASH 2	2	Wellgreen Platinum Ltd.	20.90	23/02/2032

Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255301450	YC18495	BURWASH 20	20	Wellgreen Platinum Ltd.	20.90	23/02/2028
255189337	YC18496	BURWASH 21	21	Wellgreen Platinum Ltd.	20.90	23/02/2028
255412582	YC18497	BURWASH 22	22	Wellgreen Platinum Ltd.	20.90	23/02/2028
255469116	YC18498	BURWASH 23	23	Wellgreen Platinum Ltd.	20.92	23/02/2028
255298647	YC18499	BURWASH 24	24	Wellgreen Platinum Ltd.	20.90	23/02/2028
255380086	YC18500	BURWASH 25	25	Wellgreen Platinum Ltd.	20.92	23/02/2028
255231672	YC18501	BURWASH 26	26	Wellgreen Platinum Ltd.	20.88	23/02/2028
255310689	YC18502	BURWASH 27	27	Wellgreen Platinum Ltd.	20.90	23/02/2028
255419832	YC18503	BURWASH 28	28	Wellgreen Platinum Ltd.	20.90	23/02/2028
255215792	YC18504	BURWASH 29	29	Wellgreen Platinum Ltd.	20.90	23/02/2028
255225554	YB36425	BURWASH 3	3	Wellgreen Platinum Ltd.	20.90	23/02/2032
255301451	YC18505	BURWASH 30	30	Wellgreen Platinum Ltd.	20.90	23/02/2028
255189336	YC18506	BURWASH 31	31	Wellgreen Platinum Ltd.	20.90	23/02/2028
255412581	YC18507	BURWASH 32	32	Wellgreen Platinum Ltd.	20.90	23/02/2028
255469117	YC18508	BURWASH 33	33	Wellgreen Platinum Ltd.	20.90	23/02/2028
255268606	YB36426	BURWASH 4	4	Wellgreen Platinum Ltd.	20.90	23/02/2032
255465192	YB36427	BURWASH 5	5	Wellgreen Platinum Ltd.	20.90	23/02/2032
255220670	YB36428	BURWASH 6	6	Wellgreen Platinum Ltd.	20.90	23/02/2032
255296805	YB36429	BURWASH 7	7	Wellgreen Platinum Ltd.	20.90	23/02/2032
255296804	YB36430	BURWASH 8	8	Wellgreen Platinum Ltd.	20.90	23/02/2032
255356452	YB36431	BURWASH 9	9	Wellgreen Platinum Ltd.	20.90	23/02/2032
255483424	60775	DISCOVERY 1	1	0905144 B.C. Ltd	10.49	05/12/2020
255371918	60776	DISCOVERY 2	2	0905144 B.C. Ltd	10.50	05/12/2020
255398440	60777	DISCOVERY 3	3	0905144 B.C. Ltd	16.08	05/12/2020
255308986	60778	DISCOVERY 4	4	0905144 B.C. Ltd	16.82	05/12/2020
255483720	60779	DISCOVERY 5	5	0905144 B.C. Ltd	13.35	05/12/2020
255483723	60780	DISCOVERY 6	6	0905144 B.C. Ltd	16.69	05/12/2020
255387541	60781	DISCOVERY 7	7	0905144 B.C. Ltd	13.66	05/12/2020
255242566	60782	DISCOVERY 8	8	0905144 B.C. Ltd	11.57	05/12/2020
255465231	63001	IRISH 1	1	0905144 B.C. Ltd	19.66	05/12/2020
255304897	63002	IRISH 2	2	0905144 B.C. Ltd	15.14	05/12/2020
255269815	63003	IRISH 3	3	0905144 B.C. Ltd	11.06	05/12/2020
255206646	63006	IRISH 6	6	0905144 B.C. Ltd	16.41	05/12/2020
255440541	64828	JEEP 234	234	0905144 B.C. Ltd	4.22	05/12/2020
255227576	64830	JEEP 236	236	0905144 B.C. Ltd	5.61	05/12/2020
255455244	64122	JEEP 238	238	0905144 B.C. Ltd	6.75	05/12/2020
255402797	64832	JEEP 240	240	0905144 B.C. Ltd	6.21	05/12/2020
255306668	64834	JEEP 242	242	0905144 B.C. Ltd	8.00	05/12/2020
255267816	64836	JEEP 244	244	0905144 B.C. Ltd	12.24	05/12/2020
255488272	66569	JEEP 265	265	0905144 B.C. Ltd	9.98	05/12/2020
255433251	66571	JEEP 267	267	0905144 B.C. Ltd	19.70	05/12/2020
255196868	66572	JEEP 268	268	0905144 B.C. Ltd	18.46	05/12/2020
255344858	64742	JEEP 96	96	0905144 B.C. Ltd	11.93	05/12/2020
255420333	YD127061	KAT 1	1	0905144 B.C. Ltd	17.60	05/12/2015
255395583	YD127070	KAT 10	10	0905144 B.C. Ltd	3.06	05/12/2016
255220014	YD127071	KAT 11	11	0905144 B.C. Ltd	5.63	05/12/2016

Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255229506	YD127072	KAT 12	12	0905144 B.C. Ltd	19.87	05/12/2016
255202477	YD127073	KAT 13	13	0905144 B.C. Ltd	2.73	05/12/2016
255307546	YD127074	KAT 14	14	0905144 B.C. Ltd	20.57	05/12/2016
255243017	YD127075	KAT 15	15	0905144 B.C. Ltd	5.94	05/12/2016
255228993	YD127076	KAT 16	16	0905144 B.C. Ltd	20.90	05/12/2016
255261062	YD127077	KAT 17	17	0905144 B.C. Ltd	6.52	05/12/2016
255274249	YD127078	KAT 18	18	0905144 B.C. Ltd	20.90	05/12/2016
255375030	YD127079	KAT 19	19	0905144 B.C. Ltd	11.07	05/12/2016
255306298	YD127062	KAT 2	2	0905144 B.C. Ltd	20.90	05/12/2015
255375031	YD127080	KAT 20	20	0905144 B.C. Ltd	20.90	05/12/2016
255335925	YD127081	KAT 21	21	0905144 B.C. Ltd	15.54	05/12/2016
255319961	YD127082	KAT 22	22	0905144 B.C. Ltd	20.90	05/12/2016
255226927	YD127083	KAT 23	23	0905144 B.C. Ltd	10.86	05/12/2016
255228115	YD127084	KAT 24	24	0905144 B.C. Ltd	20.90	05/12/2016
255463251	YD127085	KAT 25	25	0905144 B.C. Ltd	13.90	05/12/2016
255475900	YD127086	KAT 26	26	0905144 B.C. Ltd	20.90	05/12/2016
255483347	YD127087	KAT 27	27	0905144 B.C. Ltd	7.65	05/12/2016
255324089	YD127088	KAT 28	28	0905144 B.C. Ltd	15.69	05/12/2016
255421725	YD127089	KAT 29	29	0905144 B.C. Ltd	7.86	05/12/2016
255464975	YD127063	KAT 3	3	0905144 B.C. Ltd	18.08	05/12/2015
255421724	YD127090	KAT 30	30	0905144 B.C. Ltd	2.44	05/12/2016
255391234	YD127091	KAT 31	31	0905144 B.C. Ltd	2.10	05/12/2016
255351085	YD127092	KAT 32	32	0905144 B.C. Ltd	0.92	05/12/2016
255446367	YD127093	KAT 33	33	0905144 B.C. Ltd	1.14	05/12/2016
255250742	YD127094	KAT 34	34	0905144 B.C. Ltd	2.84	05/12/2016
255342915	YD127095	KAT 35	35	0905144 B.C. Ltd	5.49	05/12/2017
255486397	YD127096	KAT 36	36	0905144 B.C. Ltd	3.26	05/12/2017
255353924	YD127097	KAT 37	37	0905144 B.C. Ltd	16.92	05/12/2017
255442257	YD127098	KAT 38	38	0905144 B.C. Ltd	20.02	05/12/2017
255216253	YD127099	KAT 39	39	0905144 B.C. Ltd	16.97	05/12/2017
255371098	YD127064	KAT 4	4	0905144 B.C. Ltd	14.39	05/12/2015
255421726	YD127100	KAT 40	40	0905144 B.C. Ltd	20.02	05/12/2017
255391233	YD127101	KAT 41	41	0905144 B.C. Ltd	16.02	05/12/2017
255351084	YD127102	KAT 42	42	0905144 B.C. Ltd	20.02	05/12/2017
255459530	YE70953	KAT 43	43	0905144 B.C. Ltd	14.24	05/12/2017
255254398	YE70954	KAT 44	44	0905144 B.C. Ltd	20.02	05/12/2017
255335825	YE70955	KAT 45	45	0905144 B.C. Ltd	10.36	05/12/2017
255209640	YE70956	KAT 46	46	0905144 B.C. Ltd	20.02	05/12/2017
255243515	YE70957	KAT 47	47	0905144 B.C. Ltd	17.69	05/12/2017
255383568	YE70958	KAT 48	48	0905144 B.C. Ltd	13.71	05/12/2017
255408243	YE70959	KAT 49	49	0905144 B.C. Ltd	20.90	05/12/2017
255222385	YD127065	KAT 5	5	0905144 B.C. Ltd	16.65	05/12/2015
255408240	YE70960	KAT 50	50	0905144 B.C. Ltd	19.89	05/12/2017
255239361	YE70961	KAT 51	51	0905144 B.C. Ltd	20.90	05/12/2017
255214708	YE70962	KAT 52	52	0905144 B.C. Ltd	13.92	05/12/2017
255370850	YE70963	KAT 53	53	0905144 B.C. Ltd	20.90	05/12/2017

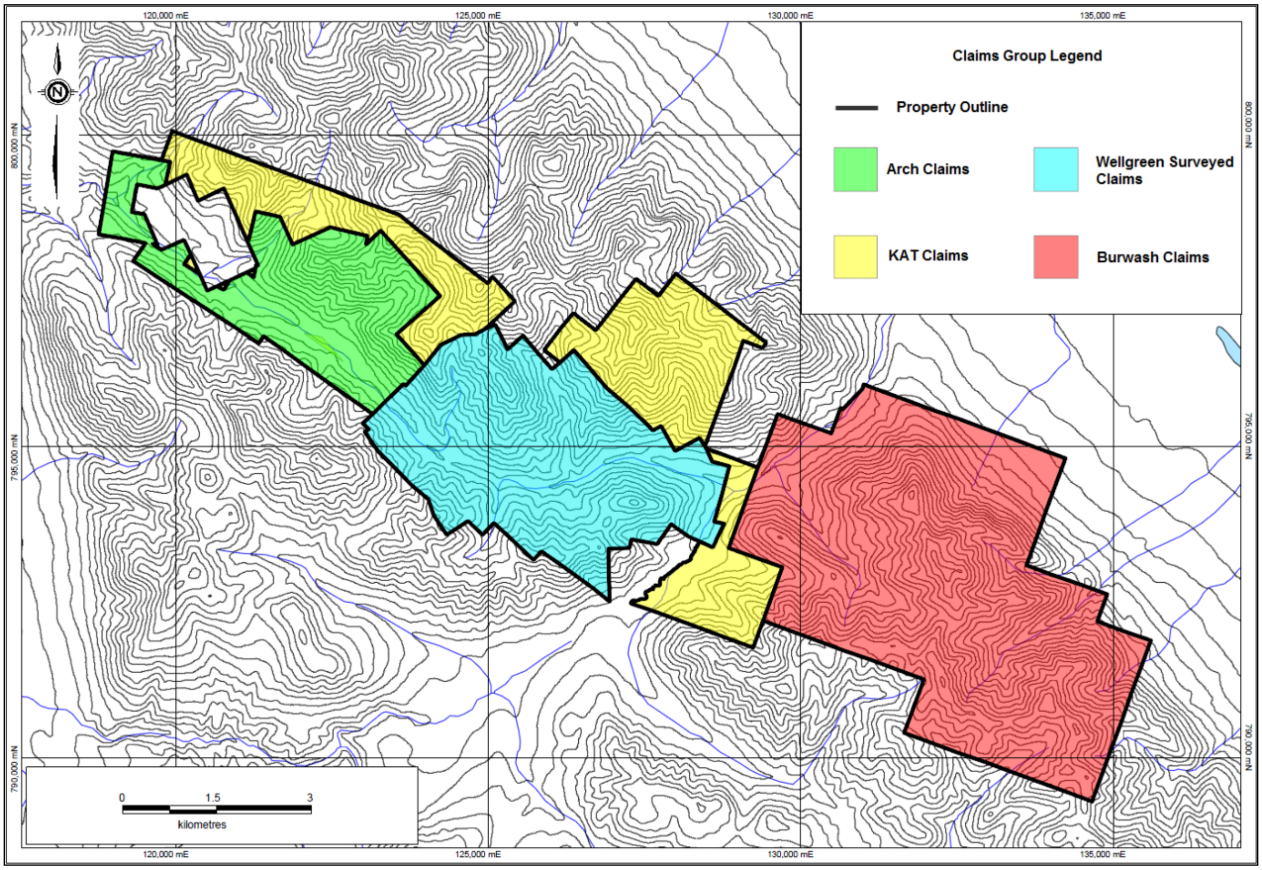
Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255285825	YE70964	KAT 54	54	0905144 B.C. Ltd	12.49	05/12/2017
255485235	YE70965	KAT 55	55	0905144 B.C. Ltd	20.90	05/12/2017
255233304	YE70966	KAT 56	56	0905144 B.C. Ltd	20.90	05/12/2017
255416376	YE70967	KAT 57	57	0905144 B.C. Ltd	20.90	05/12/2017
255472178	YE70968	KAT 58	58	0905144 B.C. Ltd	20.90	05/12/2017
255208652	YE70969	KAT 59	59	0905144 B.C. Ltd	20.90	05/12/2017
255256264	YD127066	KAT 6	6	0905144 B.C. Ltd	10.11	05/12/2015
255208651	YE70970	KAT 60	60	0905144 B.C. Ltd	20.90	05/12/2017
255299226	YE70971	KAT 61	61	0905144 B.C. Ltd	20.90	05/12/2017
255385373	YE70972	KAT 62	62	0905144 B.C. Ltd	20.90	05/12/2017
255302490	YE70973	KAT 63	63	0905144 B.C. Ltd	20.90	05/12/2017
255401861	YE70974	KAT 64	64	0905144 B.C. Ltd	20.90	05/12/2017
255430256	YE70975	KAT 65	65	0905144 B.C. Ltd	20.90	05/12/2017
255479008	YE70976	KAT 66	66	0905144 B.C. Ltd	20.90	05/12/2017
255450671	YE70977	KAT 67	67	0905144 B.C. Ltd	20.90	05/12/2017
255379738	YE70978	KAT 68	68	0905144 B.C. Ltd	20.90	05/12/2017
255208987	YE70979	KAT 69	69	0905144 B.C. Ltd	16.97	05/12/2017
255321350	YD127067	KAT 7	7	0905144 B.C. Ltd	16.45	05/12/2016
255208988	YE70980	KAT 70	70	0905144 B.C. Ltd	19.65	05/12/2017
255186557	YE70981	KAT 71	71	0905144 B.C. Ltd	8.54	05/12/2017
255411115	YE70982	KAT 72	72	0905144 B.C. Ltd	19.65	05/12/2017
255300597	YE70983	KAT 73	73	0905144 B.C. Ltd	14.09	05/12/2017
255212296	YE70984	KAT 74	74	0905144 B.C. Ltd	18.21	05/12/2017
255414584	YE70985	KAT 75	75	0905144 B.C. Ltd	2.86	05/12/2017
255349638	YE70986	KAT 76	76	0905144 B.C. Ltd	7.56	05/12/2017
255284606	YE70987	KAT 77	77	0905144 B.C. Ltd	4.35	05/12/2017
255380687	YE70988	KAT 78	78	0905144 B.C. Ltd	8.00	05/12/2017
255374635	YE70989	KAT 79	79	0905144 B.C. Ltd	9.84	05/12/2017
255222585	YD127068	KAT 8	8	0905144 B.C. Ltd	6.60	05/12/2016
255374634	YE70990	KAT 80	80	0905144 B.C. Ltd	8.44	05/12/2017
255484112	YE70991	KAT 81	81	0905144 B.C. Ltd	10.92	05/12/2017
255360105	YE70992	KAT 82	82	0905144 B.C. Ltd	5.71	05/12/2017
255338965	YE70993	KAT 83	83	0905144 B.C. Ltd	11.70	05/12/2016
255465014	YE70994	KAT 84	84	0905144 B.C. Ltd	19.60	05/12/2016
255269253	YE70995	KAT 85	85	0905144 B.C. Ltd	8.78	05/12/2016
255394142	YE70996	KAT 86	86	0905144 B.C. Ltd	19.49	05/12/2016
255395582	YD127069	KAT 9	9	0905144 B.C. Ltd	16.10	05/12/2016
255246379	63021	MAC 1	1	0905144 B.C. Ltd	12.62	05/12/2020
255339148	63022	MAC 2	2	0905144 B.C. Ltd	12.47	05/12/2020
255488812	63023	MAC 3	3	0905144 B.C. Ltd	14.20	05/12/2020
255292889	63024	MAC 4	4	0905144 B.C. Ltd	11.19	05/12/2020
255358734	63025	MAC 5	5	0905144 B.C. Ltd	9.82	05/12/2020
255188418	63026	MAC 6	6	0905144 B.C. Ltd	8.44	05/12/2020
255485515	63027	MAC 7	7	0905144 B.C. Ltd	7.64	05/12/2020
255451126	63028	MAC 8	8	0905144 B.C. Ltd	13.84	05/12/2020
255248317	YA96015	MUS 12	12	0905144 B.C. Ltd	20.99	11/02/2016

Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255215036	YA96017	MUS 14	14	0905144 B.C. Ltd	20.37	11/02/2016
255479174	YA96019	MUS 16	16	0905144 B.C. Ltd	16.12	11/02/2016
255294268	YA94966	MUS 5	5	0905144 B.C. Ltd	20.87	11/02/2016
255348463	YA94967	MUS 6	6	0905144 B.C. Ltd	20.74	11/02/2016
255276532	70829	QUILL	0	0905144 B.C. Ltd	11.14	05/12/2020
255432273	60767	QUILL 1	1	0905144 B.C. Ltd	16.78	05/12/2020
255293495	60768	QUILL 2	2	0905144 B.C. Ltd	17.13	05/12/2020
255237754	60769	QUILL 3	3	0905144 B.C. Ltd	20.89	05/12/2020
255237753	60770	QUILL 4	4	0905144 B.C. Ltd	20.55	05/12/2020
255345310	60771	QUILL 5	5	0905144 B.C. Ltd	20.78	05/12/2020
255317542	60772	QUILL 6	6	0905144 B.C. Ltd	20.78	05/12/2020
255414585	60773	QUILL 7	7	0905144 B.C. Ltd	14.01	05/12/2020
255306630	60774	QUILL 8	8	0905144 B.C. Ltd	16.52	05/12/2020
255237331	60791	RAM 1	1	0905144 B.C. Ltd	15.76	05/12/2020
255194628	60792	RAM 2	2	0905144 B.C. Ltd	20.88	05/12/2020
255473495	60793	RAM 3	3	0905144 B.C. Ltd	20.07	05/12/2020
255321702	60794	RAM 4	4	0905144 B.C. Ltd	19.86	05/12/2020
255461652	60795	RAM 5	5	0905144 B.C. Ltd	7.89	05/12/2020
255295666	60796	RAM 6	6	0905144 B.C. Ltd	22.07	05/12/2020
255484170	60797	RAM 7	7	0905144 B.C. Ltd	16.18	05/12/2020
255268746	60798	RAM 8	8	0905144 B.C. Ltd	13.55	05/12/2020
255290877	63037	RED 1	1	0905144 B.C. Ltd	15.34	05/12/2020
255422779	63038	RED 2	2	0905144 B.C. Ltd	13.53	05/12/2020
255371645	63039	RED 3	3	0905144 B.C. Ltd	16.09	05/12/2020
255371646	63040	RED 4	4	0905144 B.C. Ltd	20.69	05/12/2020
255230014	63041	RED 5	5	0905144 B.C. Ltd	20.87	05/12/2020
255373427	63042	RED 6	6	0905144 B.C. Ltd	15.65	05/12/2020
255296763	63043	RED 7	7	0905144 B.C. Ltd	15.46	05/12/2020
255428355	63044	RED 8	8	0905144 B.C. Ltd	19.10	05/12/2020
255307559	71432	ROSS 1	1	0905144 B.C. Ltd	16.47	05/12/2020
255232983	64076	ROSS 15	15	0905144 B.C. Ltd	20.74	05/12/2020
255438455	64077	ROSS 16	16	0905144 B.C. Ltd	20.74	05/12/2020
255246320	71433	ROSS 2	2	0905144 B.C. Ltd	19.75	05/12/2020
255476056	64066	ROSS 25	25	0905144 B.C. Ltd	15.94	05/12/2020
255369169	71434	ROSS 3	3	0905144 B.C. Ltd	13.18	05/12/2020
255299744	71435	ROSS 4	4	0905144 B.C. Ltd	11.97	05/12/2020
255208678	64086	ROSS 85	85	0905144 B.C. Ltd	20.88	05/12/2020
255334385	64087	ROSS 86	86	0905144 B.C. Ltd	21.11	05/12/2020
255308911	64084	ROSS 94	94	0905144 B.C. Ltd	22.04	05/12/2020
255343676	64085	ROSS 95	95	0905144 B.C. Ltd	23.86	05/12/2020
255375577	64587	ROSS 96	96	0905144 B.C. Ltd	23.98	05/12/2020
255465279	YC40144	RUB 1	1	Wellgreen Platinum Ltd.	20.90	23/02/2025
255209790	YC40153	RUB 10	10	Wellgreen Platinum Ltd.	20.90	23/02/2025
255311005	YC40154	RUB 11	11	Wellgreen Platinum Ltd.	20.90	23/02/2025
255191381	YC40155	RUB 12	12	Wellgreen Platinum Ltd.	20.90	23/02/2025
255282567	YC40156	RUB 13	13	Wellgreen Platinum Ltd.	20.90	23/02/2025

Quartz Claim #	Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
255479512	YC40157	RUB 14	14	Wellgreen Platinum Ltd.	20.90	23/02/2025
255391201	YC40158	RUB 15	15	Wellgreen Platinum Ltd.	20.90	23/02/2025
255292963	YC40159	RUB 16	16	Wellgreen Platinum Ltd.	20.90	23/02/2025
255292962	YC40160	RUB 17	17	Wellgreen Platinum Ltd.	20.90	23/02/2025
255323582	YC40161	RUB 18	18	Wellgreen Platinum Ltd.	20.90	23/02/2025
255468455	YC40162	RUB 19	19	Wellgreen Platinum Ltd.	20.90	23/02/2025
255272964	YC40145	RUB 2	2	Wellgreen Platinum Ltd.	20.90	23/02/2025
255403324	YC40163	RUB 20	20	Wellgreen Platinum Ltd.	20.90	23/02/2025
255263623	YC40164	RUB 21	21	Wellgreen Platinum Ltd.	20.77	23/02/2025
255400446	YC40165	RUB 22	22	Wellgreen Platinum Ltd.	20.90	23/02/2025
255262529	YC40166	RUB 23	23	Wellgreen Platinum Ltd.	14.03	23/02/2025
255443181	YC40167	RUB 24	24	Wellgreen Platinum Ltd.	20.90	23/02/2025
255329627	YC40168	RUB 25	25	Wellgreen Platinum Ltd.	20.90	23/02/2025
255472223	YC40169	RUB 26	26	Wellgreen Platinum Ltd.	20.90	23/02/2025
255472226	YC40170	RUB 27	27	Wellgreen Platinum Ltd.	20.90	23/02/2025
255360592	YC40171	RUB 28	28	Wellgreen Platinum Ltd.	20.90	23/02/2025
255351307	YC40172	RUB 29	29	Wellgreen Platinum Ltd.	20.90	23/02/2025
255223558	YC40146	RUB 3	3	Wellgreen Platinum Ltd.	20.90	23/02/2025
255412399	YC40147	RUB 4	4	Wellgreen Platinum Ltd.	20.90	23/02/2025
255365449	YC40148	RUB 5	5	Wellgreen Platinum Ltd.	20.90	23/02/2025
255454703	YC40149	RUB 6	6	Wellgreen Platinum Ltd.	20.90	23/02/2025
255454702	YC40150	RUB 7	7	Wellgreen Platinum Ltd.	20.90	23/02/2025
255418583	YC40151	RUB 8	8	Wellgreen Platinum Ltd.	20.90	23/02/2025
255262760	YC40152	RUB 9	9	Wellgreen Platinum Ltd.	20.90	23/02/2025
255402284	63013	SAM 1	1	0905144 B.C. Ltd	6.04	05/12/2020
255373683	63014	SAM 2	2	0905144 B.C. Ltd	9.72	05/12/2020
255346916	63015	SAM 3	3	0905144 B.C. Ltd	15.78	05/12/2020
255206451	63016	SAM 4	4	0905144 B.C. Ltd	10.64	05/12/2020
255344282	63017	SAM 5	5	0905144 B.C. Ltd	12.55	05/12/2020
255384593	63018	SAM 6	6	0905144 B.C. Ltd	16.92	05/12/2020
255325051	63019	SAM 7	7	0905144 B.C. Ltd	14.27	05/12/2020
255325052	63020	SAM 8	8	0905144 B.C. Ltd	10.32	05/12/2020
255429399	60783	WAGONER 1	1	0905144 B.C. Ltd	18.46	05/12/2020
255345822	60784	WAGONER 2	2	0905144 B.C. Ltd	18.46	05/12/2020
255221053	60785	WAGONER 3	3	0905144 B.C. Ltd	13.58	05/12/2020
255427401	60786	WAGONER 4	4	0905144 B.C. Ltd	14.37	05/12/2020
255304421	60787	WAGONER 5	5	0905144 B.C. Ltd	16.00	05/12/2020
255456791	60788	WAGONER 6	6	0905144 B.C. Ltd	16.00	05/12/2020
255320890	60789	WAGONER 7	7	0905144 B.C. Ltd	13.88	05/12/2020
255320891	60790	WAGONER 8	8	0905144 B.C. Ltd	15.14	05/12/2020

Source: GeoSim, Wellgreen Platinum, 2015

Figure 4.2: Mineral Tenure



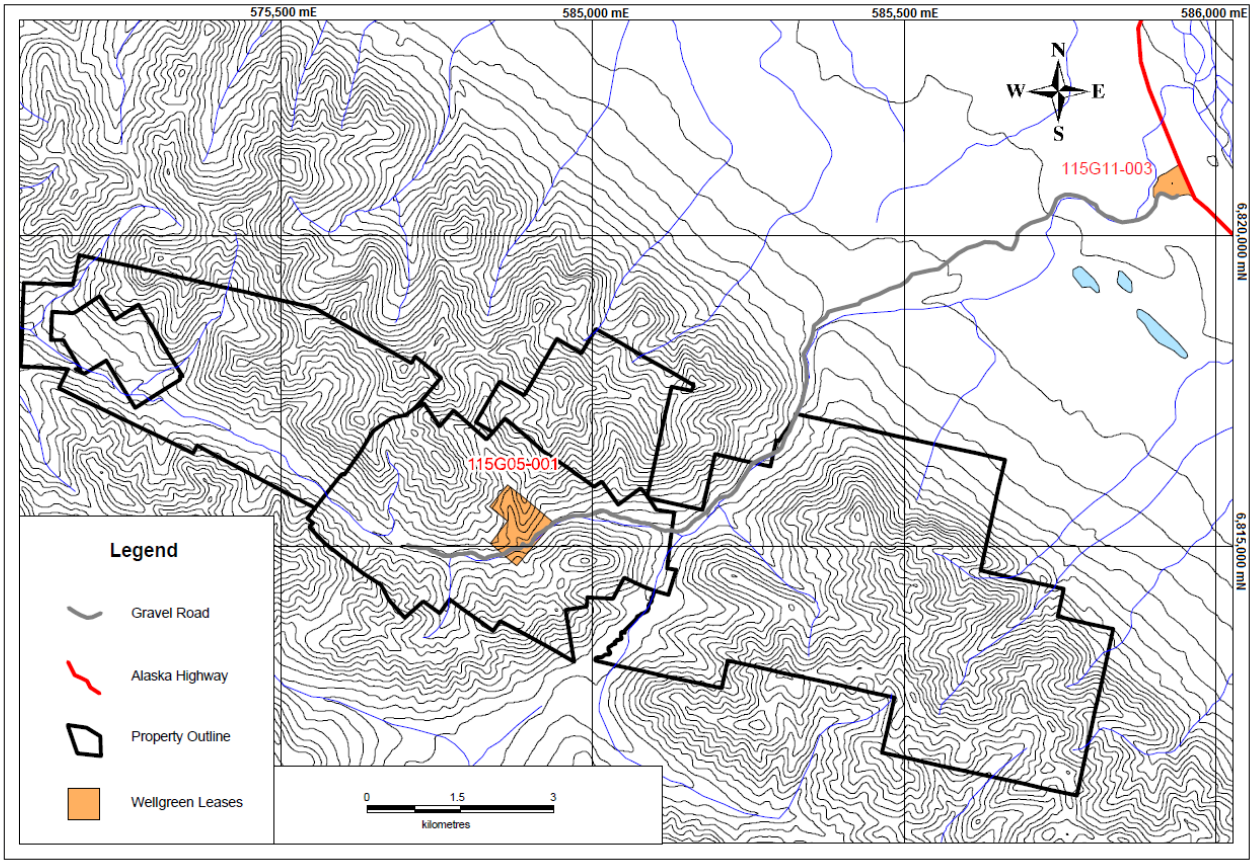
Source: GeoSim, Wellgreen Platinum, 2015

Table 4.2: Surface Leases

Land Disposition#	Pid	Application	Disposition	Tenure Purpose	Area (ha)	Disposition Date	Expiry Date
2753634	100015069		115G05-001	Industrial	69.7	24/08/1971	30/05/2034
2753541	100023288	2363L	115G11-003	Commercial	21.7	20/01/1971	31/10/2022

Source: GeoSim, Wellgreen Platinum, 2015

Figure 4.3: Surface Leases



Source: GeoSim, Wellgreen Platinum, 2015

4.4 Property Ownership and History

Wellgreen Platinum has owned a consolidated 100% interest in the Property since June 2011. Details of how Wellgreen Platinum acquired its 100% ownership of the Property are summarized below.

An underlying agreement dated April 27, 1999 between Kaieteur Resource Corporation (Kaieteur) (formerly International All-North Resources Ltd. (All-North)), Northern Platinum, and J. Patrick Sheridan related to Northern Platinum's interest in the Arch Joint Venture. Under this agreement, Northern Platinum agreed to purchase from Kaieteur all of its All-North interest in the Property, and its interest in the Arch Joint Venture on an "as is" basis for a sum of Cdn$62,500 to be paid in cash and shares. The agreement acknowledged that Northern Platinum had already earned a 20% interest in the project and, under this agreement, Northern Platinum acquired the remaining 80% interest. Kaieteur warranted it was the beneficial owner of the All-North Property interest but did not provide the same warranties for the Arch Joint Venture because certain historical documentation for underlying agreements was incomplete – hence the "as is" stipulation. On September 22, 2010, Northern Platinum (who at that time owned a 100% interest in the Property, subject to a 50% back-in right held by Belleterre Quebec) was acquired by Prophecy Resource Corp. As a result, Prophecy Resource Corp. became the owner of a 100% interest in the Property (subject to the 50% back-in right held by Belleterre Quebec). Subsequently on September 24, 2010, Prophecy Resource Corp. acquired the 50% back-in right held by Belleterre Quebec, resulting in Prophecy Resource Corp. acquiring a 100% interest in the Property, free of any back-in rights.

In June 2011, Prophecy Resource Corp. spun out all of its North American platinum and nickel assets, including its entire 100% interest in the Property, to 0905144 B.C. Ltd., a wholly-owned subsidiary of Pacific Coast Nickel Corp. (Wellgreen Platinum's predecessor company). As a result of the spin-out transaction, Pacific Coast Nickel Corp. acquired 100% ownership of the Property.

Immediately upon completion of this spin-out transaction, Pacific Coast Nickel Corp. changed its name to Prophecy Platinum Corp., and in December 2013, Prophecy Platinum Corp. changed its name to Wellgreen Platinum Ltd.

4.5 Permits

In the Yukon, the Quartz Mining Land Use Regulation and the Placer Mining Land Use Regulation consist of a classification system based on varying levels of specific activities. These threshold levels categorize exploration activities into four classes of operation. Classes 1 through 4 represent activities with increasing potential to cause adverse environmental impacts.

Wellgreen Platinum currently holds two Class 3 Operating Plan permits through the Yukon Government Mining Land Use Division (see Figure 4.4).

Permit LQ00323b covers the claims on which the current mineral resource has been delineated as well as the upper camp of the Property located on surface Lease 115G05-001. This permit expires July 20, 2021.

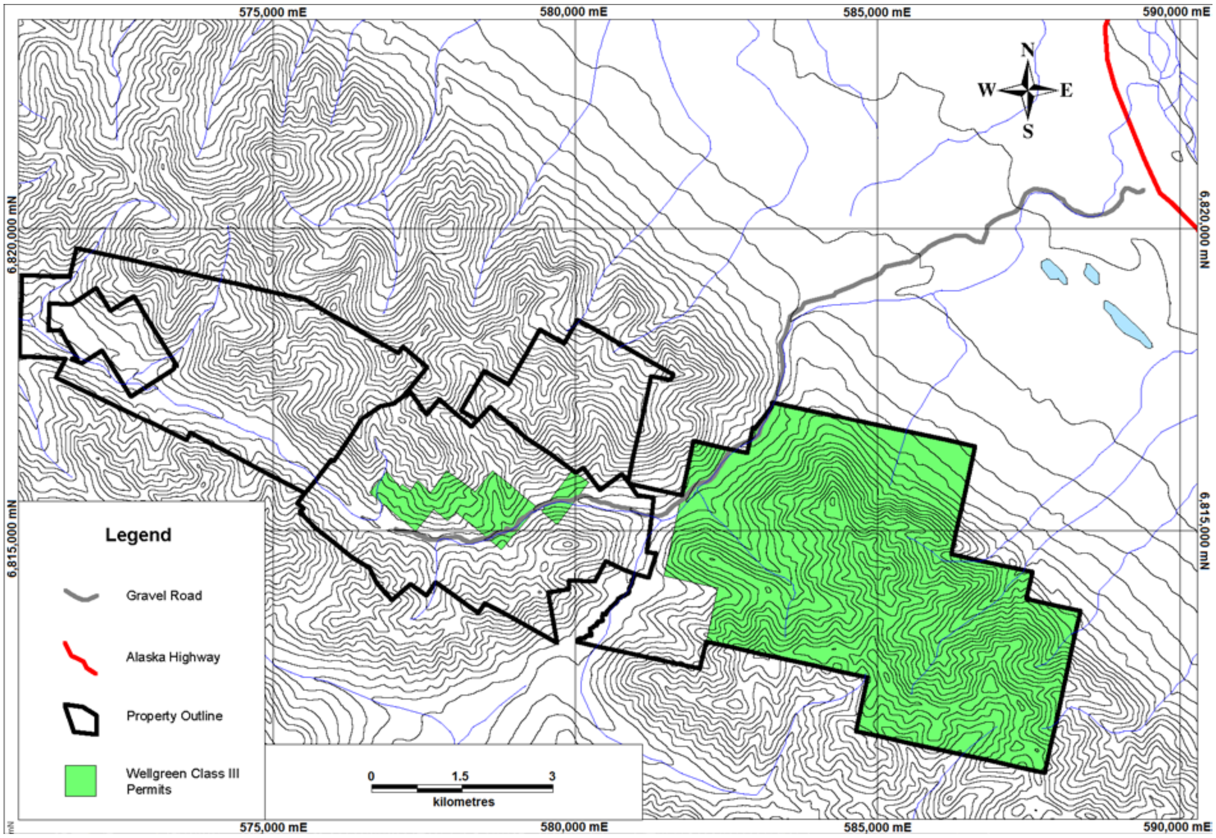
Permit LQ00259a covers the majority of the Burwash Property claims. This permit expires May 14, 2017.

Class 3 Programs require:

- Submission of a detailed Operating Plan to the Mining Lands Office;
- Assessment through Yukon Environmental and Socio-economic Assessment Board;
- That the Operating Plan be approved before any other exploration activities can proceed; and
- The Operating Plan may entail multi-year exploration programs to allow greater flexibility for the operator.

Class 3 Program terms and conditions are presented in Table 4.3.

Figure 4.4: Operating Plan Permits



Source: GeoSim, Wellgreen Platinum, 2015

Table 4.3: Class 3 Operating Permit Terms

Element	Terms and Conditions
Establishing new access roads per program	[NIL]
Off-Road use of vehicles in summer	[NIL]
Corridor width	1 m wide x 4000 m over the length of the project
Lines	Vegetative mat will not be disturbed
Establishment of trails per program	Spurs from main road to access drills sites
# of clearings per claim, including existing clearings	Up to 10 clearings per claim
Surface area of each clearing	Up to 25 m^2
Total volume of trenching	Up to 1,800 m^3
# of person days per camp	Approximately 1,200 person days
# of persons in a camp at any one time	12 persons
Fuel Storage in a stationary container	Diesel: 400 L stored in 200 L drums
	Gasoline: 200 L stored in 20 L jerry cans
Upgrading of access roads per	Existing 4x4 road will have to have winter sloughing bladed off annually
Used of vehicles on existing roads or trails	Annually from June to October

Source: Wellgreen Platinum, Yukon Government - Energy, Mines and Resources, 2015

In addition, exploration at the Quill claims is currently taking place under a Class 1 “threshold”, i.e. in the Yukon a written Class I permit is not issued.

4.6 Environmental Liabilities

Wellgreen Platinum has cleaned up surface debris at the old mill site and removed contaminated soils, pursuant to the Reclamation Plan referred to in Section 4.3 and in accordance with the terms of the old surface lease. These activities were initiated in 2009 and were completed in 2013 under the direction of Access Consulting Group of Whitehorse. The majority of the contaminated soils on the existing Lease 115G11-003 have now been removed and disposed of in Tervita’s Northern Rockies Landfill in Fort Nelson B.C. One small patch of hydrocarbon contamination remains underneath a site maintenance building. It was left during the initial clean up as it is being utilized. Once the structure is demolished, delineation and remediation will take place.

Some additional reclamation activities remain outstanding associated with the historic HudBay Mill Site and 1970s tailings impoundments which are not on Wellgreen Platinum controlled lands. The Government of Yukon, the Federal Government of Canada and HudBay, with technical support from Wellgreen Platinum, are in discussions concerning the final reclamation and restoration of these historic sites. The outstanding amount with respect to these additional reclamation activities is estimated to be approximately C$1.5 million.

4.7 First Nations

Surface Rights Legislation for Yukon First Nations is provided under the Umbrella Final Agreement between the Government of Canada, Government of Yukon, and Yukon First Nations. This legislation provides a mechanism to resolve disputes over access rights (Mining Yukon 2011 and Minister of Public Works and Government Services Canada 2003).

The Kluane First Nation has a settled land claim, which provides them with access, rights and obligations to land and resources, and the right to govern their own affairs. The Kluane First Nation signed final and self-government agreements with the Yukon and Canadian governments on October 18, 2003. The effective date of these agreements was February 2, 2004 (Yukon ECO 2011a).

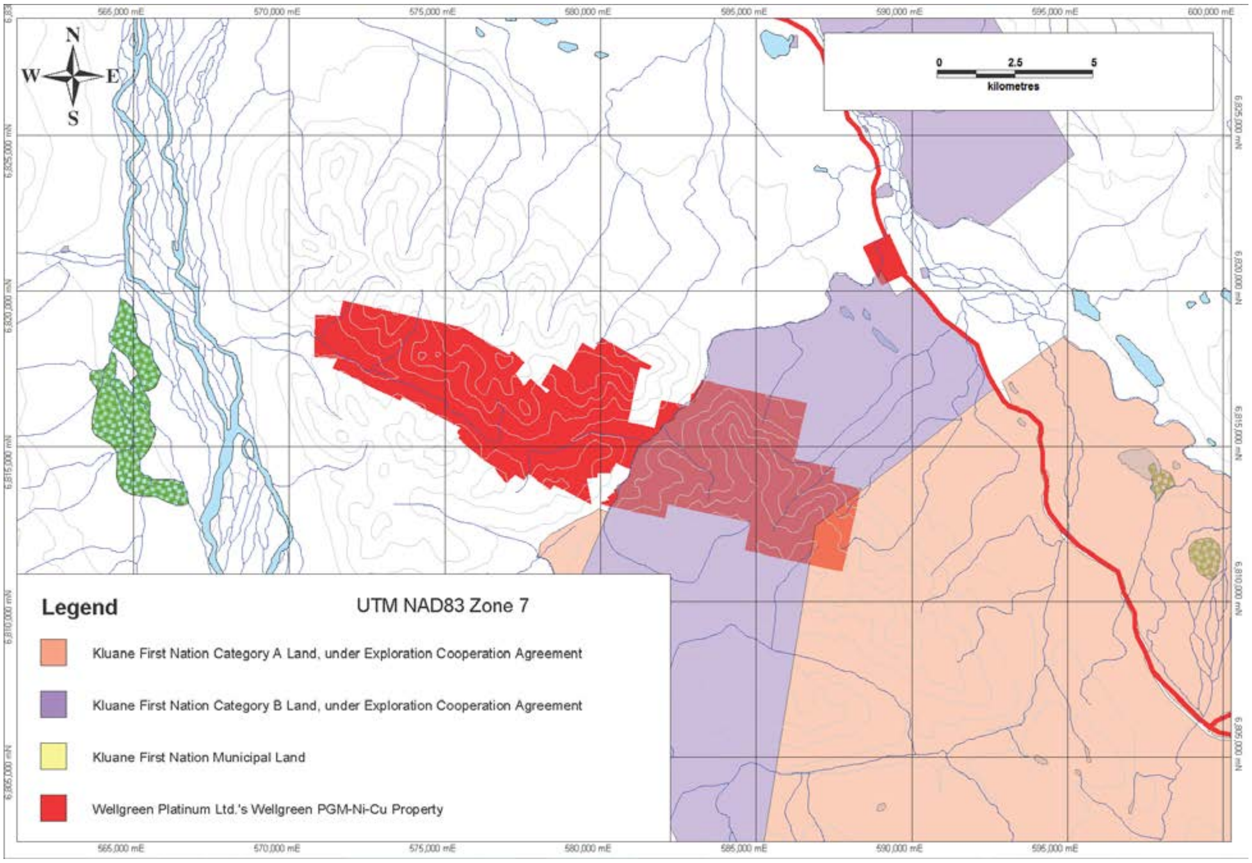
The Property is located in the "core area" of the Kluane First Nation as defined by the Umbrella Final Agreement. The Property partially overlaps on Category B land (R-49 B) and Category A (R-01A) land owned by the Kluane First Nation (Figure 4.5) (Minister of Public Works and Government Services Canada 2003). As of the signing of the Kluane First Nation Final Agreement, the Kluane First Nation holds both the surface rights and the subsurface/mineral rights on Category A land, while on Category B land, the Kluane First Nation owns the surface rights to this land, but not that which is below the surface. However, land belonging to persons holding a right, title, interest, license, and permit on the land prior to the time the area was claimed as Settlement Land are not subject to this legislation (Minister of Public Works and Government Services Canada 2003).

Surface Rights Legislation for Yukon First Nations is provided under the Umbrella Final Agreement between the Government of Canada, Government of Yukon, and Yukon First Nations. This legislation provides a mechanism to resolve disputes over access rights (Mining Yukon 2011 and Minister of Public Works and Government Services Canada 2003).

The White River First Nation finalized negotiations toward final and self-government agreements with the Canadian and Yukon governments in 2002, when a Memorandum of Understanding (MOU) was signed signifying the completion of the negotiation process. However, the White River First Nation decided not to ratify the negotiated agreements and there have been no negotiations since. As such, the White River First Nation does not have a settled land claim. Under the terms of the Umbrella Final Agreement, the White River First Nation was allocated Category A and Category B land in their "core area", which have been "interim protected" from third-party interests, pending the settlement or abandonment of a land claim agreement (Yukon ECO 2011b). The "core area" for White River First Nation lies well to the west and north of the Property and is separated from the Kluane First Nation "core area" by an area of overlapping traditional use. On December 18, 2014, the White River First Nations and Government of the Yukon Territory jointly announced that the two parties have initiated preliminary negotiations with the goal of reaching a reconciliation agreement. The intent of the reconciliation agreement discussions is to provide the parties with a process to constructively resolve issues relating to land use and other matters.

Wellgreen Platinum signed an exploration co-operation agreement (ECA) with the Kluane First Nation August 1, 2012, pursuant to which regular ECA meetings are held between Wellgreen Platinum and the Kluane First Nation. The agreement also provides that Wellgreen Platinum will continue to engage the White River First Nation with respect to discussions related to community presentations as well as training and employment opportunities.

Figure 4.5: Kluane First Nations Land Status



Source: GeoSim, Wellgreen, 2015

Other than as set out in this Section 4, to the extent known, there are no other environmental liabilities to which the Property is subject and no other significant factors that may affect access, title or the right or ability to perform work on the Property.

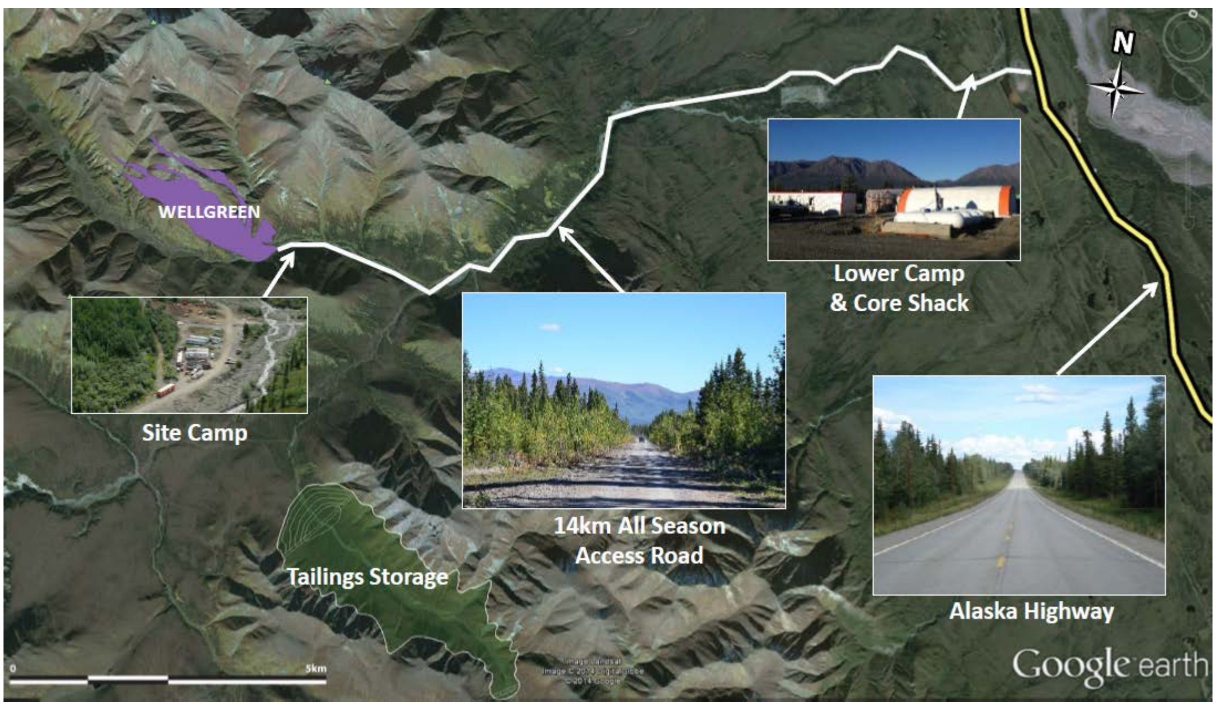
5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Property is located approximately 317 km northwest of Whitehorse, Yukon and can be reached via the paved all-weather Alaska Highway which is maintained by the Government of Yukon (approximately kilometre 1,726). From the highway to the Wellgreen deposit, travel is by gravel road (mine access road) that runs southwest beside Quill Creek for a distance of 14 km (Figure 5.1).

An all-weather airstrip is also located approximately 15 km southeast of the Property at Burwash Landing. It is maintained by NAV CANADA and presently sees limited winter maintenance.

All-season, deep-sea ports are located in Haines, Alaska, 410 km to the southeast, as well as Skagway, Alaska, which is currently utilized by Capstone Mining and Alexco Resources for the transport of mining concentrate material on bulk container ships to smelters. Both ports are year round ice free ports and are accessible by high-quality paved highways.

Figure 5.1: Project Access



Source: Wellgreen Platinum, 2015

5.1 Climate

The regional climate is semi-arid, sub-arctic with relatively warm, dry summers and winters characterized by relatively dry, cold interior conditions, but tempered by west coast climate influences. Weather records have been historically recorded at the Burwash Landing weather station (806.8 masl). The area lies in the rain shadow of the Saint Elias Mountains, with average annual total precipitation for the Burwash Landing station of 27.97 cm (11 inches) of which 19.2 cm (7.6 inches) typically falls as rain in summer and the remainder as snow in winter.

A meteorological station was installed near the Upper Camp approximately 600 m southeast of the adit portal on October 27, 2012 by EBA, a Tetra Tech Company from Whitehorse. It consists of a standard 10 m tower with instrumentation to measure wind speed and direction, air temperature, relative humidity, barometric pressure, incident solar radiation, and water-equivalent precipitation. An evaporation pan was installed in June 2013 at the same location to enable evaporation rates to be recorded over the summer months. Data is collected and stored on a regular basis by EBA.

Data collection recorded over the first year of installation returned the following:

- Maximum air temperature was 24.6°C on June 27, 2013;
- Minimum air temperature was -37.4°C on January 28, 2013;
- Greatest monthly precipitation was 25.2 cm in July 2013; and
- Least monthly precipitation was 0.38 cm in March 2013.

The Project operates all year round.

5.2 Local Resources Infrastructure

The villages of Burwash Landing and Destruction Bay are located 15 km and 30 km, respectively, southeast from the Property. In addition to the airstrip at Burwash Landing, these towns have lodging, food and fuel with potential for future subdivision development to provide housing for mining personnel.

5.2.1 Power

Generators installed for the exploration programs currently supply power on the Property. Haines Junction is the current limit of the high capacity grid and hydroelectric system of Yukon Energy Corporation (YEC), which is approximately 140 km from the Property along the Alaska Highway. Currently, it is believed that there is 20 megawatts of surplus capacity on the YEC grid.

Wellgreen Platinum has signed memoranda of understanding with liquefied natural gas suppliers in Alaska and Western Canada to supply the energy needs for the initial project. In addition, Wellgreen Platinum has signed a memorandum of understanding with General Electric Canada to provide products and services for the Property, which includes complete power generation and the transmission network for the Property.

5.2.2 Water

A water supply, adequate for drilling operations, can be pumped from local creeks. Potable and non-potable water was supplied for the camp from the surface waters of Nickel Creek. The surface waters of Arid Creek were tested by Maxxam Analytics and subjected to their “Drinking Water

Analysis" package once a month during the 2013 field season and all tests confirmed that the water was potable.

Wellgreen Platinum has installed a UV filtration system that the surface water must filter through prior to being dispensed for drinking as per the Yukon Public Health and Safety Act regulations. All local creeks freeze solid during the winter months, therefore in order to maintain a year round camp or mining operation, drilling of water wells will be required.

It has been assumed that sufficient water supplies from pit dewatering will be available for the mill processing needs of the project.

5.2.3 Mining Personnel

Yukon has no government debt, no territorial sales tax and a highly competitive taxation regime, all of which encourage investment in the mining sector. Skilled labour and equipment are available in the city of Whitehorse (population 24,500) and the community of Haines Junction (area population of approximately 800 people). Limited services are also available in the two closest communities, Burwash Landing and Destruction Bay.

5.3 Physiography

The Property is located in the Kluane Ranges, which are a continuous chain of foothills situated along the eastern flank of the Saint Elias Mountains. The topography across the Property is typical of the interior Yukon with slopes of 250 to 300 m, and the highest peaks exceed an elevation of 1,800 m.

The main mineralized zone on the Property lies between an elevation of 1,250 and 1,700 m on a moderate to steep south-facing slope. Water drainage on the property is mainly east and then north into the Quill Creek drainage.

Vegetation consists of typical alpine vegetation on the hillsides, along with a mixture of pine, spruce and poplar trees located in the lower elevations and creed beds.

6 History

6.1 Prior Ownership and Ownership Changes

W. Green, C. Aird, & C Hankins were the prospectors who discovered the surface showing near Arid Creek in 1952. The property was optioned to Yukon Mining Company, a subsidiary of HudBay that same year, which was then transferred to another subsidiary called Hudson Yukon Mining in 1955.

The Property was optioned to a joint venture between All North Resources Ltd. (All-North) and Chevron Minerals in 1986 (Kluane JV) which acquired a 50% interest in the Property. That same year, Galactic Resources Ltd. purchased the Hudson Yukon Mining interest and net smelter returns royalty on the property, and merged with All-North. In 1989, All North purchased Chevron Minerals' 25% interest to acquire 100% interest in the Property. Other joint ventures were formed on the Arch Property, which lies west of the Property.

In 1994, Northern Platinum acquired an 80% interest in the Property from All-North, with the remaining 20% purchased in 1999. Coronation Minerals optioned the Property in 2005, but dropped the option in 2009. The Property was then returned to Northern Platinum.

Prophecy Resource Corp. purchased Northern Platinum near the end of 2010. The Property and other nickel assets were spun out to its subsidiary Pacific Coast Nickel Corp, which then changed its name to Prophecy Platinum Corp. in 2011. Prophecy Platinum Corp. changed its name to Wellgreen Platinum Ltd. in 2013.

6.2 Previous Exploration and Development

During the tenure of HudBay, a total of 25,017 m of drilling was completed in 60 surface and 481 underground drill holes. Additionally, HudBay undertook 4,267 m of underground development including internal shafts. Ground geophysics and a soil geochemical survey were also conducted.

Between 1987 and 1988 during the Kluane JV, 16,648 m of drilling was completed in 83 surface and 34 underground holes with some rehabilitation of the underground workings and slashing of new drill stations. Additional exploration included geological mapping and sampling, VLF and magnetic surveys, and surface trenching.

From 1996 to 2005, Northern Platinum drilled 4,471 m of surface diamond (10 holes) and reverse circulation (57) holes.

Coronation drilled 7,248 m in 24 surface and three underground holes from 2006 to 2008. This program resulted in the discovery of the deep mineralization in the East Zone. An aeromagnetic survey of 854 line kilometres was also carried out.

In 2009 and 2010, Northern Platinum drilled 4,190 m in 16 core holes prior to its acquisition by Prophecy Resources Corp. Prophecy Resources Corp. drilled one more 117 m hole.

In 2011, Prophecy Platinum Corp. (now Wellgreen Platinum Ltd.) drilled 1,925 m in six core holes. This drill program resulted in an updated resource and PEA.

In 2012, Prophecy Platinum Corp. (now Wellgreen Platinum Ltd.) drilled 10,983 metres in 51 core holes.

In 2013, Prophecy Platinum Corp. (now Wellgreen Platinum Ltd.) drilled 27 drill holes which totalled 2,793 m of new drilling, along with assaying another 8,462 metres of core from approximately 21,784 m of re-logged historical drill core from 108 holes.

Additional information regarding a brief description of the exploration programs, to the extent known, is discussed in Section 10.

6.3 Historic Mineral Resource and Reserve Estimates

A Qualified Person has not completed sufficient work to classify any historical estimates as a current mineral resources or mineral reserves, therefore, Wellgreen Platinum is not treating the historical estimates as mineral resources or mineral reserves.

6.4 Historic Production

Hudson Yukon Mining commenced commercial production in 1972. Mined mineralized material was trucked down from the mine to the mill site near the current lower camp, beside the Alaska Highway. Production ceased in 1973 due to falling metal prices, and discontinuous massive sulphide horizons. A total of 171,652 tonnes grading 2.23% Ni, 1.39% Cu, 1.3 g/t Pt, 0.92 g/t Pd, 0.17 g/t Au, 0.40 g/t Rh, 0.42 g/t Ru, 0.25 g/t Ir, 0.20 g/t Os, and 0.20 g/t Re were milled to produce 33,853 tons of concentrate, which was shipped to Sumitomo in Japan.

7 Geological Setting and Mineralization

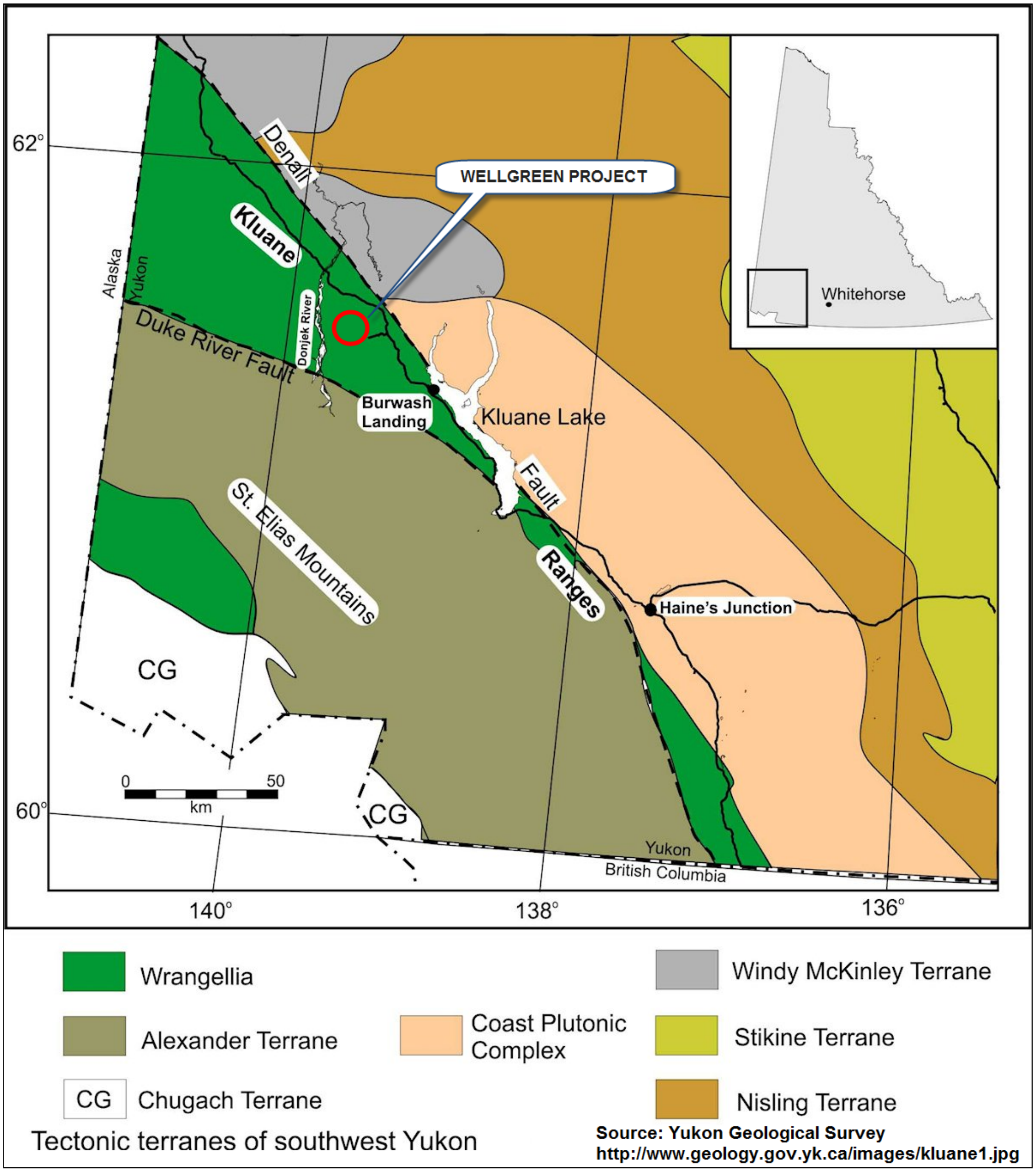
7.1 Regional Geology

The Property is located within the Insular Superterrane, which is dominantly composed of two older terranes (Wrangellia and Alexander) that were amalgamated at approximately 320 million years (Ma) (Figure 7.1). These terranes are composed of island arc and ocean floor volcanic rocks with thick assemblages of overlying oceanic sedimentary rocks that range in age from 220 to 400 Ma. Wrangellia exhibits a package of platform-type limestones that are several kilometres thick conformably overlying a 230 Ma old package of volcanic rocks (the Nikolai Group) that is present on the Property.

The Property is contained within the Kluane Ultramafic Belt, which is situated within the Wrangellia Terrane. This terrane is complex and variable, extends from Vancouver Island to central Alaska, and is most commonly characterized by the widespread exposure of Triassic flood basalts and complementary intrusive rocks (Figure 7.2). The ultramafic intrusives of the Wrangellia Terrane represent one of the largest tracts of nickel-copper-PGE mineralization in North America, second in size to the Proterozoic Circum-Superior Belt in Northern Quebec which rims the Archean Superior province (Hulbert, 1997).

The exposed base of Wrangellia is comprised of Pennsylvanian to Permian arc volcanic rocks and Permian sedimentary rocks of the Skolai Group and includes the Hasen Creek Formation and the Station Creek Formation. The Skolai Group is unconformably overlain by the Middle and Late Triassic Nikolai Group generally consisting of basalt flows with minor intercalated limestone. Mafic and ultramafic intrusions are common throughout the area and are generally located near the contact between the Station Creek and Hasen Creek formations. The intrusions commonly exhibit magmatic sulphide associated nickel-copper-PGE and gold mineralization. These sills, which represent individual members of the Kluane Ultramafic Belt, are thought by some to be part of a sub-volcanic system that fed the Nikolai Formation flood basalts (Israel 2004). However, there is some field evidence which suggests that the Nikolai Formation basalts may have been fed instead by the 232.2 ± 1 Ma Maple Creek Gabbro (Mortensen & Hulbert, 1992). This gabbro occurs as a series of dikes and plugs that are observed to cross-cut the sills of the Kluane Ultramafic Belt and in one case are exposed as feeders to the Nikolai Group basalt (Hulbert, 1997). The Kluane Belt is bound on the northeast by the Shakwak Fault, which is a major terrain boundary. The fault's latest movement is described as dextral (right-lateral).

Figure 7.1: Regional Geologic Setting



Source: Yukon Geological Survey, 2015

7.2 Local Geology

Israel and Zeyl (2004) provides the most recent regional geological mapping for the Property as illustrated in Figure 7.2. Hulbert (1997) also provides a description and discussion of detailed geology and interpretation covering the Wellgreen deposit area from maps completed by Archer, Cathro and Associates, who have compiled and reinterpreted exploration results for the Kluane JV programs carried out on behalf of All-North. The descriptions and classifications of the geological framework for the Property from these sources are not consistent.

The oldest rocks on the Property are represented by the Pennsylvanian and/or Permian Station Creek Formation. The Station Creek Formation underlies significant portions of the Property. The formation consists of light to medium green volcanic breccia, tuffs and tuffaceous sandstones and also contains a component of basalt. The Station Creek Formation is conformably overlain by the Permian Hasen Creek Formation, which consists of a range of metasediments; greywacke, thin-bedded siltstone turbidites, chert/quartzite, argillite, and limestones as well as volcaniclastics and tuffs. These rocks are folded into a series of parallel, sometimes overturned, synclines and anticlines.

The Hasen Creek Formation rocks are unconformably overlain by locally amygdaloidal flood basalt, volcanic breccias and metasediments of the Upper Triassic Nikolai Group. The Nikolai Group rocks are also folded into a series of southeast-northwest trending anticlines and synclines.

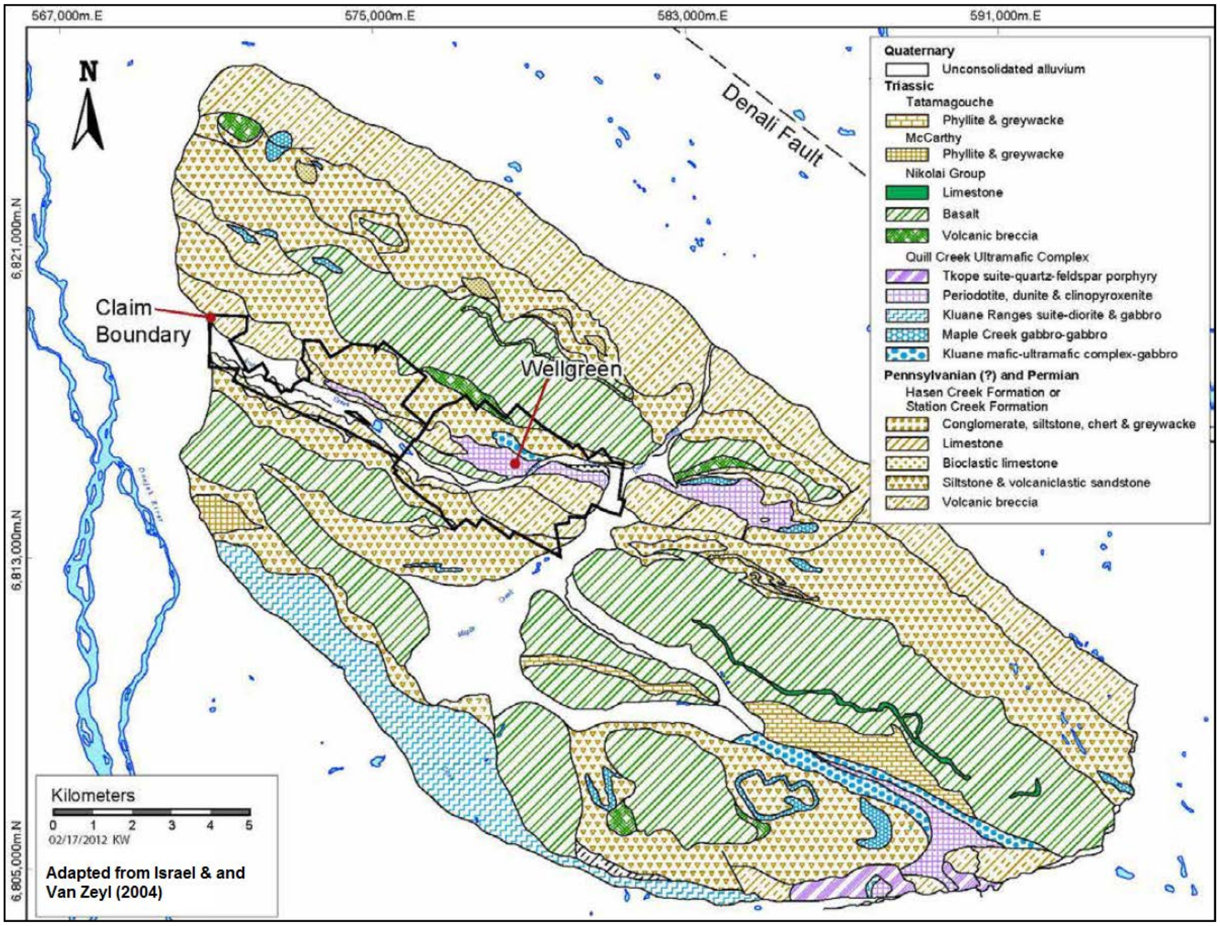
In the Wellgreen deposit area, Nikolai Group mafic volcanics occur in the area immediately south of the Quill Creek Complex. The volcanics have been interpreted to be in fault contact with the upper part of the Quill Creek Complex and Station Creek Formation rocks (Israel and Zeyl 2004).

There is an abundant series of relatively small intrusions into Paleozoic metasediments and the Quill Creek Complex. They are mapped as andesitic to gabbroic dikes and plugs that are part of the Maple Creek Gabbro, and are likely correlated with the Nikolai Formation. Hulbert (1997) describes these same rocks as felsic dikes, which may have been gabbro dikes that experienced post-emplacement alteration. Many of these small intrusions are associated with the northeast-southwest oriented faults that cut the stratigraphic sequence and the Quill Creek Complex, while others are parallel to the structural grain of the Station Creek and Hasen Creek Formations.

The youngest rocks on the Property are represented by the Cretaceous intermediate and mafic intrusive belonging to the Kluane Ranges suite.

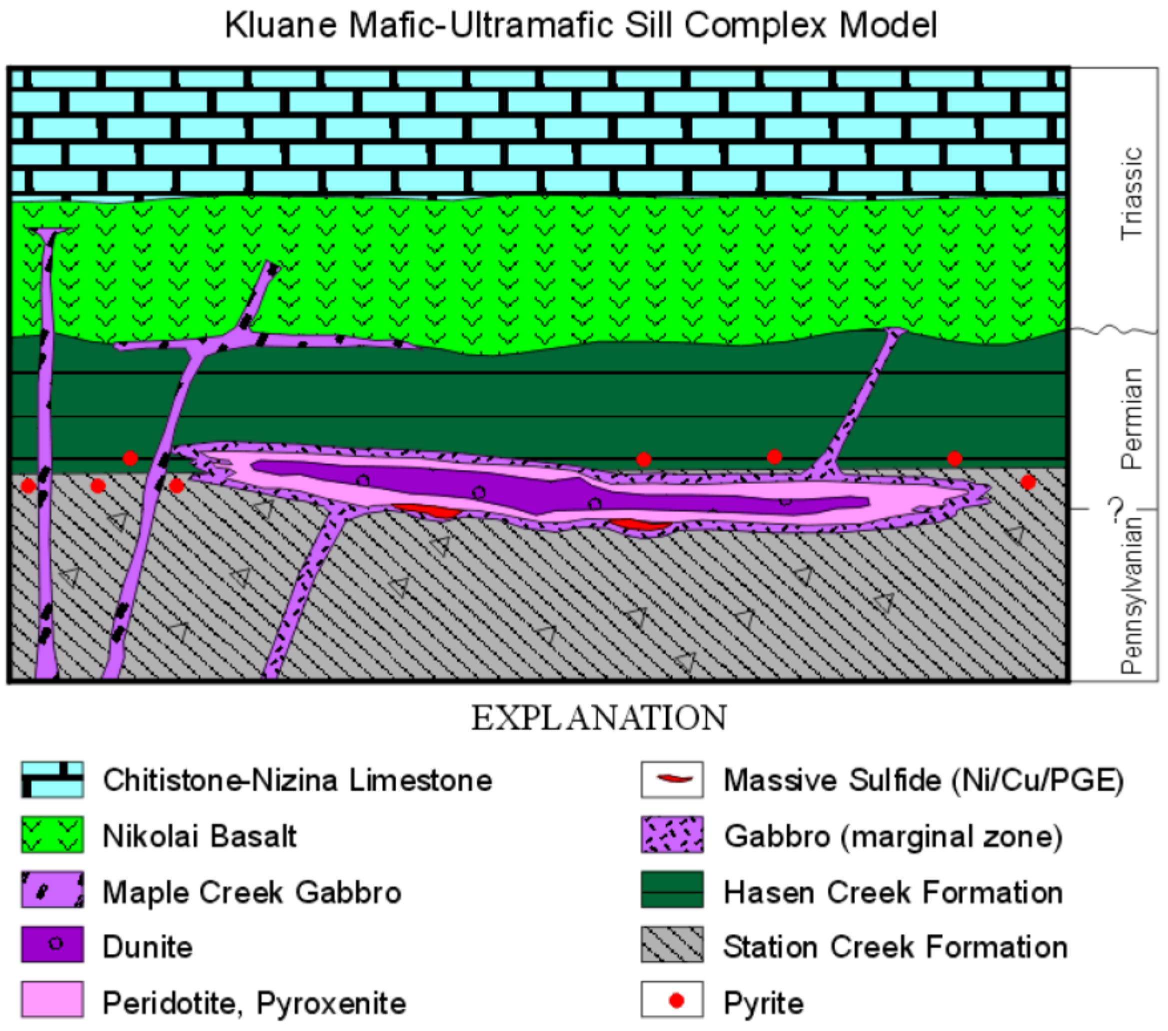
Longitudinal faults and/or shears are common in the ultramafic rocks. Some of these faults occur along lithological contacts. The most prominent of these is coincident with Maple Creek. Hulbert (1997) describes two western faults as west-dipping reverse faults. Two faults present in the western portion of the Wellgreen project intrusion offset the mafic-ultramafic rocks and dip steeply to the southeast.

Figure 7.2: Geology of the Quill Creek Area



Source: Israel & Van Zeyl, 2004

Figure 7.3: Kluane Mafic-Ultramafic Sill Complex Model



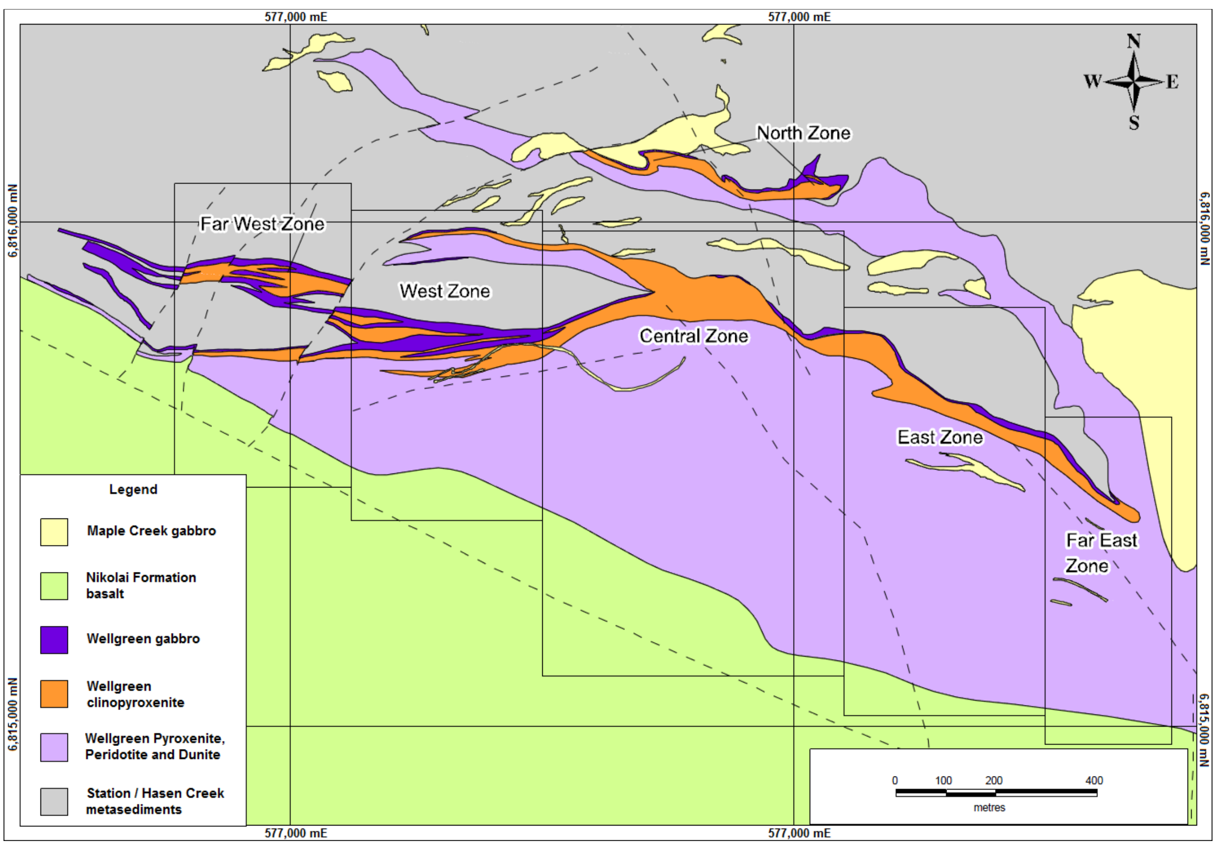
Schematic section of typical Kluane mafic-ultramafic intrusive complex model (pre-deformation) showing the outer marginal gabbroic envelope, and the zonal development of progressively more ultramafic rocks (i.e. Pyroxenite, peridotite and dunite, respectively) towards the core of the intrusion. Note the massive sulphide concentrations at the base of the complex and the intrusions preferential emplacement at or near the pyritic Station Creek-Hasen Creek Formations contact zone. (L. Hulbert 1995, GSC OF3057)

Source: Hulbert, 1995

7.3 Property Geology

The Wellgreen deposit occurs within, and along the lower margin of, an Upper Triassic ultramafic-mafic body, within the Quill Creek Complex. This assemblage of mafic-ultramafic rocks is 20 km long and closely intrudes along the contact between the Station Creek and Hasen Creek formations. The main mass of the Quill Creek Complex, the Wellgreen project and Quill intrusions, is 4.7 km long and up to 1 km wide. A smaller mass of similar intrusives is located along strike to the northwest and southeast, known as the Arch and Burwash intrusions, respectively. The Quill Creek Complex consists of a main intrusion and an associated group of upright to locally overturned, steeply south dipping sills. Based on drill information the northernmost sill, called the North Arm, and the main Wellgreen project sill appear to be contiguous at depth. The Quill Creek Complex layered intrusion which gradationally transitions from Dunite to Peridotite to Pyroxenite to Clinopyroxenite to Gabbro with a corresponding increasing sulphide and mineralization content through this sequence toward contact with the Paleozoic sedimentary country rocks. The intrusions are variably serpentinized and locally deformed. Locally, the sills have a lower gabbroic margin adjacent to a chilled contact with Paleozoic rocks. Recent observations indicate that many of these marginal gabbros may actually be endo-skarn units that appear to be the direct result of digestion and hybridization of limestone present in the Hasen Creek country rocks by the Wellgreen project parent magma(s). Mafic-rich exo-skarns also occur in the floor rocks adjacent to the marginal facies gabbro, particularly where the metasediment host includes limestone or calcareous rocks. The intrusives are zoned upwards/southward away from the lower gabbroic zone through zones of Clinopyroxenite, Pyroxenite, Peridotite, and Dunite. This zonation may be directly related to the degree of interaction with the reactive wall-rocks and appears to reflect the relative sulphide content of the rocks with highest sulphide content at the lower margins grading up to the least sulphide content in the upper parts of the tabular intrusion mostly as Dunite.

Figure 7.4: Property Geology



Source: GeoSim, Wellgreen Platinum, 2015

7.4 Mineralization

Mineralization on the Property occurs within the Quill Creek Complex. This variably serpentinized, ultramafic-gabbroic body intrudes Pennsylvanian-Permian sedimentary and volcanic rocks. Historic exploration and development programs defined two main zones of gabbro-hosted massive and disseminated sulphide mineralization known as the East Zone and West Zone. These zones have since been subdivided into the contiguous Far East, East, West, and Far West Zones with the connecting Central Zone. The historic North Arm Zone has only limited drilling to date**.**

7.4.1 Far East Zone

The Far East Zone represents the easternmost part of the Wellgreen project intrusion. The Zone lies between 578250E and Arid Creek, at approximately 578750E (coordinate system North American Datum 1983, Zone 7). The large plug of Maple Creek Gabbro represents the eastern boundary of the zone (Figure 7.3). In both the current East and Far East Zones, historic exploration efforts focused on defining massive sulphide horizons and lenses near the contact between the Wellgreen project Intrusion and Hasen Creek metasediments and as such this contact is very well defined. This sedimentary contact was historically interpreted to be the steeply dipping southern footwall to mineralization based on the data available at the time, but more recent work in the East Zone (see below) showed that the sedimentary contact was a wedge of metasediments in a much larger ultramafic body. This change in understanding in the nature of the sedimentary contact was demonstrated in the Far East Zone by drill holes 154, 160, and 165. Further drilling determined that the main Wellgreen project Intrusion is likely contiguous with the southern contact of the North Arm.

The typical steeply-dipping lithological sequence of Dunite-Peridotite-Pyroxenite-Clinopyroxenite-Gabbro with massive sulphide is very well defined in the Far East Zone. The core of the Far East Zone shows a broad sub-horizontal sulphide-rich pyroxenite, clinopyroxenite, and gabbro/skarn horizon with a second clinopyroxenite and gabbro enriched zone at the lower contact with the metasediments.

In the easternmost portion of the Far East Zone, all lithologies exhibit a similar sub-horizontal dip to the symmetrical sequence further west: with Dunite transitioning to Peridotite then Pyroxenite, Clinopyroxenite, and Gabbro with skarn units and massive sulphide immediately prior to the basal contact with Station and Hasen Creek metasediments. This lower sequence is interpreted to be contiguous with the basal sequence observed 350 metres farther to the west. The basal contact is interpreted to be contiguous with the northern contact of the North Arm. In additional, the foot-wedge pinches out to the east such that, in the upper portion of the intrusion, the various contact-proximal lithologies are absent.

7.4.2 East Zone

The East Zone lies between 577900E and 578250E, and was historically explored for massive sulphide at the Wellgreen project-footwedge contact. As mentioned above, this Zone was the first in which the change in the footwall contact's orientation was observed in drill core. The Peridotite-Pyroxenite-Clinopyroxenite-Gabbro sequence is observed to wrap around the base of the wedge in the East Zone. Historic drill holes ended in mineralized ultramafic material such that it is currently unknown how thick the mafic-ultramafic package is beneath the foot-wedge which remains open at depth.

The historic East Zone (current East and Far East Zones combined) was mined by Hudson Yukon Mining in 1972 and 1973, and approximately 171,652 t of mineralized material was extracted.

7.4.3 Central Zone

The Central Zone lies between 577500E and 577900E. The eastern portion of the Zone is similar to the East Zone whereby well mineralized Peridotite gradationally transitions to Pyroxenite to Clinopyroxenite and Gabbro units are observed near the contact with dominantly Station Creek metasediments. The western portion of the Central Zone exhibits a sub-horizontal, symmetrical, mineralized unit similar to that intersected at depth in the Far East Zone. Additional drilling will be required to test whether the higher grade sub-horizontal mineralization intersected in the Central zone connects with that in the East and Far East zones. This represents a high priority exploration target.

7.4.4 West Zone

The West Zone lies between 577120E and 577500E. Similar to the western portion of the Central Zone, well mineralized Pyroxenite overlies a comparatively thick package of Clinopyroxenite and Gabbro with significant semi-massive and massive sulphide zones. The small wedge of sedimentary rocks that separates the Middle Arm from the main Wellgreen project Intrusion is still present, and was intersected by two drill holes in 2001. The West Zone remains open at depth and additional drilling will be required to test whether the higher grade mineralization connects with the sub-horizontal higher grade zone in the core of the Central Zone.

7.4.5 Far West Zone

The Far West Zone lies between 576720E and 577120E, and the northern part of the Zone is interpreted to be a branching sill from the main Wellgreen project Intrusion. This sill is generally zoned outwards, with well mineralized Pyroxenite in the centre grading to Clinopyroxenite and Gabbro towards the contact with the metasedimentary country rocks. Grades in the Far West Zone are significantly elevated starting at surface with high sulphide content. This Zone has not been tested at depth to explore for connectivity with the West and Central Zones.

7.4.6 North Arm Zone

The North Arm Zone is located in the east-central portion of a narrow 1,200 metre long sill, positioned approximately 150 metres stratigraphically below the main Wellgreen project Intrusion. It was discovered by Hudson Yukon Mining in the 1950s and explored in 1987 with three drill holes by All-North. All of these drill holes intersected mineralization. The geology of this zone is similar to both the East and West Zones. Mineralization consists of massive sulphide lenses, disseminated sulphide in Gabbro and Clinopyroxenite, and fracture fillings in footwall Hasen Creek metasediments. The North Arm Zone was tested in 1988 and 2005 by limited drilling and was determined to have a sub-vertical dip. The information collected to date suggests that the North Arm Zone is relatively narrow in comparison with the main Wellgreen project body at surface, but it does represent a prospective area of nickel-copper mineralization that warrants further work and may be contiguous with the main Wellgreen project Intrusion at depth.

7.4.7 BSB Zone

The BSB Zone material, a clay-rich very high-grade style of mineralization, was discovered in 2004 by prospector David Javorsky.

The current understanding of the BSB material is that it represents highly weathered massive sulphide horizon whereby platinum group elements (PGEs) were concentrated by supergene enrichment processes. The showings occur above the ice limit of the last glacial maximum (above 1600 metres above sea level (MASL)) and were thus exposed for millennia and never eroded by ice. This long-lived exposure allowed for the in-situ development of a highly oxidized and weathered zone, where PGEs, which were originally present in either solid-solution or as discrete phases in massive sulphide, were dissolved and re-precipitated.

Clays and panned concentrates were studied using X-ray diffraction (XRD) and electron microprobe. The mineralogy of the un-panned samples is consistent with strongly weathered and oxidized massive sulphide (limonite, goethite etc.) while one panned concentrate contained sperrylite and native gold and another contained a palladium-sulphur-selenium-antimony mineral and electrum.

Though high grade, these zones are not believed to contain large tonnages of oxide material at this time.

7.4.8 Minerals

Table 7.1 to Table 7.3 after Cabri et al. (1993) list the opaque minerals and PGM-bearing minerals found in the deposit. The elevated presence of rhodium, iridium, osmium, rhenium, and ruthenium within the mineral suite provide an opportunity for additional potential economic contributions from these metals. Rhodium is present at Wellgreen project in highly anomalous concentrations as compared to the concentrations found in Noril'sk ores in Russia and other significant ultramafic systems globally (Hulbert 1997).

Table 7.1: Opaque Minerals Observed in the Wellgreen Project Deposit

Major Minerals*	
Pyrrhotite	$Fe_{1-X}S$
Pentlandite	$(Fe, Ni)_9S_8$
Chalcopyrite	$CuFeS_2$
Magnetite	Fe_3O_4
Ilmenite	$FeTiO_3$
Less Common to Rare Minerals *	
Violarite	$FeNi_2S_4$
Sphalerite	(Zn,Fe)S
Chromite	$FeCr_2O_4$
Cobaltite**	CoAsS/NiAsS
Aresenopyrite	FeAsS
Ulimannite	NiSbS
Siegenite argentopentlandite	$(Ni, Ag)(Fe, Ni)_8S_8$
Gold/electrum	(Au/Ag)
Melonite	$NiTe_2$
Bismuth tellurides	Bi-Te (?)
Galena	PbS
Altaite	PbTe
Kickline	NiAs
Covellite	CuS
Breuithauptite	NiSb
Barite	$BaSO_4$
Titanite hessite	$CaTiSiO_2Ag_2Te$
Matildite	$AgBiS_2$
Undefined	Cu-Fe-Ba-S**

Source: Cabri et al., 1993

Notes: *Ideal Formula. **Unidentified mineral of the cobalt-gersdorffite series.

Table 7.2: Primary PGM-Bearing Minerals

Mineral	Formula
Sperrylite	$PtAs_2$
Sudburyite	PdSb
Testibiopalladite	PdSbTe
Merenskyite	$PdTe_2$
Moncheite	$PtTe_2$
Michernerite	PdBiTe
Stibiojaiadinite	Pd_5Sb_2
Mertielte II	Pd_8Sb_3
Geversite	$PtSb_2$
Hollingworthite	RhAsS
Froodite	$PdBi_2$
Unidentified	$(Pd,Ni)_2(Te,Sb)_3$
Unidentified	$(Pd,Ni)_3(Te,Sb)_4$
Unidentified	Pd(Bi,Te)
Unidentified	$Pd_3Ni(Sb,Te,Bi)_5$
Laurite	RuS_2
Kotuiskite	$PdTe_2$
Pt-Fe alloy(s)	Pt_3Fe or PtFe(?)
Unidentified	Re>Ir>Os>Ru alloy
Unidentified	Pd-Hg
Iridium	Ir
Unidentified	Re sulphide (?)

Source: Cabri et al., 1993

Table 7.3: Additional PGM-Bearing Minerals

Mineral	Formula	Metal Content
Melonite	$(Ni,Pd,Pt)Te_2$	Up to 15.1%Pd; up to 9.37% Pt
Unidentified	$(Ni,Pd)_2(Te,Sb)_3$	Up to 22.8% Pd
Unidentified	$(Ni,Pd)_3(Te,Sb)_4$	Up to 15.9% Pd
Breuithauptite	(Ni,Pd)Sb	Up to 18.9% Pd
Hextestibio-panickelite	$(Ni,Pd)_2SbTe$	Up to 15.9% Pd
Ullmannite	(Ni,Pd)SbS	Up to 0.09% Pd
Cobaltite	(Co,Rh)AsS	Up to 2.7% Rh, in zones
Pentaldite	(Pt,Rh,Ru)*	Up to 34 Pd, 12 Rh, 13 Ru (ppm)
Chalcopyrite	(Ru,Rh,Pd)*	Up to 10 Ru, 10 Rh, 9 Pd (ppm)
Pyrrhotite	(Pd)*	Up to 5.6 Pd (ppm)

Source: Cabri et al., 1993
Note: *Trace levels as determined by proton microprobe.

8 Deposit Types

The Wellgreen deposit is hosted in the Quill Creek Complex, one of a number of mafic-ultramafic sills that are enriched in nickel-copper-PGE mineralization that outcrop within the Kluane Ultramafic Belt of the Wrangellia Terrane in southwestern Yukon. The sills which form the Kluane mafic-ultramafic complex are thought to be part of a sub-volcanic system that fed the Nikolai Formation flood basalts and have been compared to the Noril'sk in Russia.

Similar deposits also occur elsewhere in Canada (Franklin sills; Bedard et al., 2011; Cape Smith Belt; Giovenazzo et al., 1989), in China (Yangluiping Instrusions; Xie-Yan Song et al. 2003, Jinchuan; Tonnelier, 2010), and southern Africa (Uitkomst intrusion; Maier et al., 2013, floor of eastern Bushveld Complex; Maier et al., 2001).

Many sill-hosted Ni-Cu-PGE deposits are generally considered to be part of a large, interconnected magmatic system that fed voluminous flood basalts and resulted from the impingement of a mantle plume upon the base of the crust. At Noril'sk, the main sulphide bodies formed from segregated sulphide at the base of magmatic conduits through which multiple pulses of magma travelled, and this mechanism is believed to have been also applied to the Wellgreen deposit. The Quill Creek complex intruded a Pennsylvanian-Permian island arc, whereas many of the other deposits are Precambrian and all intruded into cratons. Greene et al. (2010) offer compelling evidence that the mafic-ultramafic intrusions and flood basalts of Wrangellia were formed in an oceanic plateau, which itself was formed by a mantle plume (Richards, 1991), and the terrane was subsequently accreted to the margin of North America in the Jurassic. These circumstances make Wellgreen unique among other sill-hosted Ni-Cu-PGM deposits.

9 Exploration

Historic exploration carried out by previous operators is summarized in Section 6. Exploration relevant to the mineral resource update is presented below.

9.1 Exploration Potential

The property extends over an 18 km mineralized trend with multiple exploration targets.

9.2 Grids and Surveys

In 2013, Wellgreen Platinum conducted a collar monument and surveying program. This effort was undertaken to modernize the Property's drill database by changing the coordinate system for all data from local mine grid to Universal Transverse Mercator, North American Datum 1983, zone 7 in order to prepare for this PEA. Many holes on the Property were never surveyed or designated with monuments, and those that were surveyed used the mine grid coordinate system. A differential global position system (DGPS) was used to survey 58 holes. Most collar positions were changed by a few metres; however some collars were more than 30 m away from their supposed locations.

For road and trail surveys, the Trimble unit was carried on the operator's back whilst they were driving an all-terrain vehicle (ATV). The instrument took a measurement every few seconds. For drill collar surveys, the Trimble was activated directly over the collar and its position was measured every few seconds for one minute. The average of the measurements was then corrected using the base station located in Juneau, Alaska.

9.3 Geological Mapping

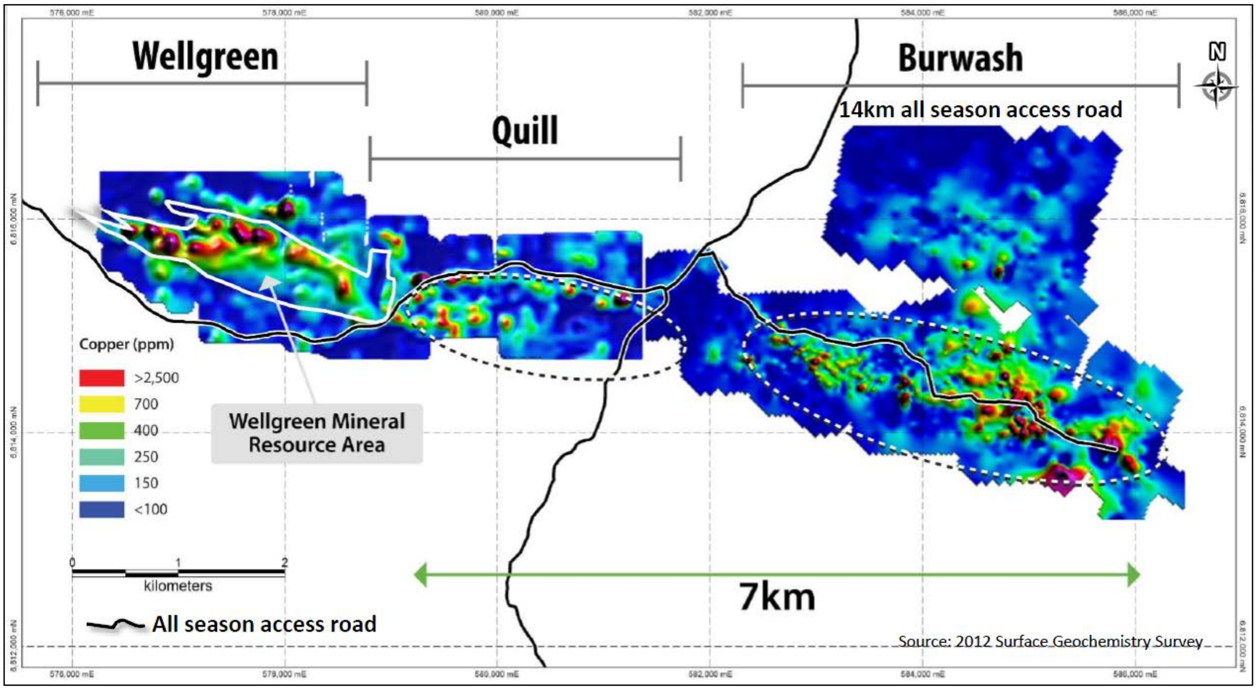
In 2013, a three day mapping program was undertaken on the eastern portion of the Property, east of Arid Creek and northeast of the upper camp. Parts of this area were exposed by undocumented bulldozer trenching. This mapping effort led to a better understanding of the contacts between the Wellgreen intrusion, the Maple Creek Gabbro, and the Hasen Creek sediments.

9.4 Geochemical Sampling

In 2012, a soil sampling survey was undertaken over the Wellgreen project/Quill, Burwash and Arch properties**.** Results for Cu are presented in Figure 9.1.

Soil samples were taken on a 25 m nominal spacing across the Property, and soil augers and mattocks were used to try to get to the B or C horizons. The samples were placed in Kraft sample bags and shipped to the ALS Global preparation facility in Whitehorse, YT. Sample pulps were then sent to ALS Global's lab in Vancouver, BC for assay.

Figure 9.1: Cu Soil Geochemistry - 2012

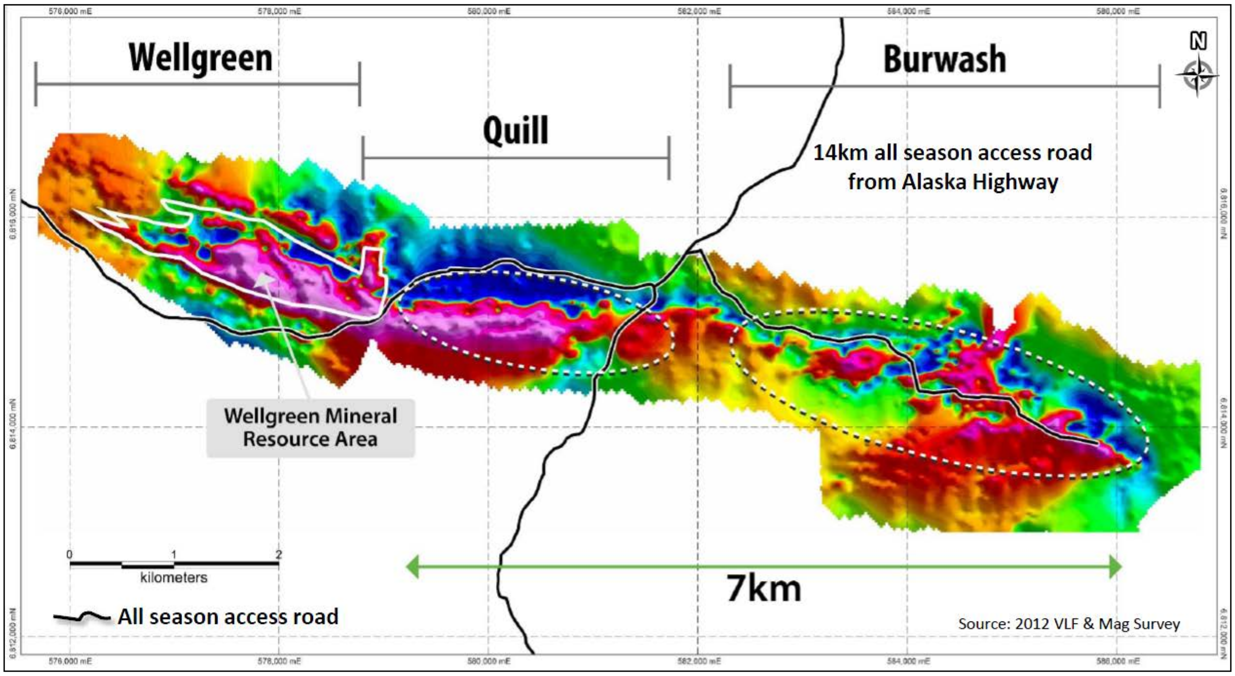


Source: Wellgreen Platinum, 2015

9.5 Geophysics

In 2012, a Mag-VLF survey was conducted over the Wellgreen project/Quill, Burwash, and parts of the Arch property. The survey over the Wellgreen project/Quill consisted of 57 lines for a total of 62.74 line kilometres (Figure 9.2).

Figure 9.2: Magnetic-VLF Survey Extent



Source: Wellgreen Platinum, 2015

10 Drilling

10.1 Historic Drilling

Considerable surface and underground drilling was completed in the 1950s by Hudson Yukon Mining, an operating subsidiary of HudBay. Additional drilling was completed under the auspices of the Kluane JV (All-North, Chevron and Galactic Resources) in the 1980s by Archer, Cathro & Associates Ltd. Drill logs, assay summaries and assay certificates for many of these historic drill holes are available and have been compiled into a database along with more recent drill data. This historic work has not been completely documented, however much of the data has been located and digitized.

10.1.1 Northern Platinum Drilling

Northern Platinum conducted numerous drill campaigns on the Property between 1996 and 2010, three of which had never been documented. The drilling conducted by Northern Platinum in 2009 and 2010 was designed to extend and expand the potential resource of the Wellgreen deposit by targeting mineralization up dip of the East Zone and east along strike. Drilling was completed by E. Caron Diamond Drilling Ltd. of Whitehorse. All holes drilled in 2009 and 2010 were HQ diameter and all drilling was run in five foot intervals (1.52 m). Ten holes were drilled in the East Zone in 2009, totalling 2051.75 m. In 2010, prior to its acquisition by Prophecy Resources Corp., Northern Platinum drilled six holes in the East Zone. After the acquisition, one more hole was drilled, bringing the 2010 total to 2,254.77 m.

10.1.2 1996 Drill Program

In 1996 Northern Platinum conducted a previously undocumented reverse circulation (RC) program that focused on the historic East and West Zones. Drilling was completed by Northern Platinum staff on an Ingersoll Rand ECM-350 3.5" diameter RC drill. A total of fifty-seven holes totaling 3,873.7 m were drilled and drilling was run on five foot intervals (1.52 m). Data from this program was not used in the resource estimate due to lack of confidence in collar locations and was not entered into the resource drill hole database.

10.1.3 2001 Drill Program

Another previously undocumented drill program was conducted in 2001. This program targeted mineralization along the historic footwall contact and is the only program to have drill-tested the Middle Arm, a splay off of the main the Property Intrusion in the West Zone. Drilling was conducted by E. Caron Diamond Drilling Ltd. of Whitehorse. A total of six drill holes were completed on the Property and one hole on the adjacent Arch property, for a total of 591.92 m. All drilling was run at HQ diameter at5 ft intervals (1.52 m).

10.1.4 2005 Drill Program

A small, undocumented program was conducted in 2005. This program focused on the North Arm, specifically on a showing with very high PGE concentrations named the BSB zone. Drilling was completed by Northern Platinum staff on an Ingersoll Rand electrochemical machining (ECM)-350

3.5” diameter RC drill. A total of four holes were completed totaling 67.05 m. All drilling was run at 5 ft intervals (1.52 m).

10.1.5 2006-2008 Coronation Minerals Drilling

The holes drilled on the Property by Coronation Minerals in 2006 were for the purpose of validating the historical drilling done by the Kluane JV in 1987 and 1988. The program was designed by WGM with a total of 24 holes proposed. Coronation Minerals engaged E. Caron Diamond Drilling Ltd. of Whitehorse, Yukon as the drill contractor. All of the surface drilling was HQ, and holes were reduced to NQ as the depth increased and ground conditions became unfavourable. The underground drilling was all BTW core size. The drilling began in late July 2006 and a total of 11 holes were completed for 2,016.87 m. Ten of the holes drilled in 2006 were drilled in order to “twin” historical holes drilled by the Kluane JV.

In 2007, three underground holes were completed totalling 576.99 m. Two of the holes were designed to “twin” historical holes.

In 2008, 13 additional surface diamond drill holes were drilled by Coronation Minerals.

10.2 Wellgreen Platinum Drilling

10.2.1 2011 Drill Program

The drilling conducted by Wellgreen Platinum in 2011 was designed initially to delineate the potential resource of the Wellgreen deposit by targeting the area between the East and West Zones to prove that the zones are not separate, but rather one continuous zone. The focus of the program evolved to test the hanging wall disseminated sulphides located in the ultramafic unit.

Drilling was completed by E. Caron Diamond Drilling Ltd. of Whitehorse. A total of nine drill holes were completed during the 2011 drill program from June to October, however three collar locations were never recorded and are considered lost. All holes were drilled HQ and all drilling was run in 5-ft intervals (1.52 m). Including the lost holes, a total of 2269.17 m was drilled in 2011.

Drill hole collar information is shown in Table 10.1 and illustrated in Figure 10.1. Significant intercepts based on a 0.15% nickel equivalent (NiEq) cut-off grade are presented in Table 10.2.

Table 10.1: Wellgreen Platinum 2011 Drill Collars

Hole-ID	UTM East	UTM North	Elev (masl)	Length (m)	Azimuth (°)	Dip (°)
WS11-184	578685.05	6815205.87	1258.99	507.49	0.00	-45.00
WS11-185	578330.32	6815188.05	1377.77	59.13	0.00	-55.00
WS11-188	577672.32	6815572.03	1635.18	491.03	0.00	-70.00
WS11-190	577875.57	6815531.60	1549.15	373.08	0.00	-70.00
WS11-191	577472.52	6815514.96	1556.38	89.92	0.00	-70.00
WS11-192	577774.13	6815578.23	1600.58	404.47	0.00	-70.00

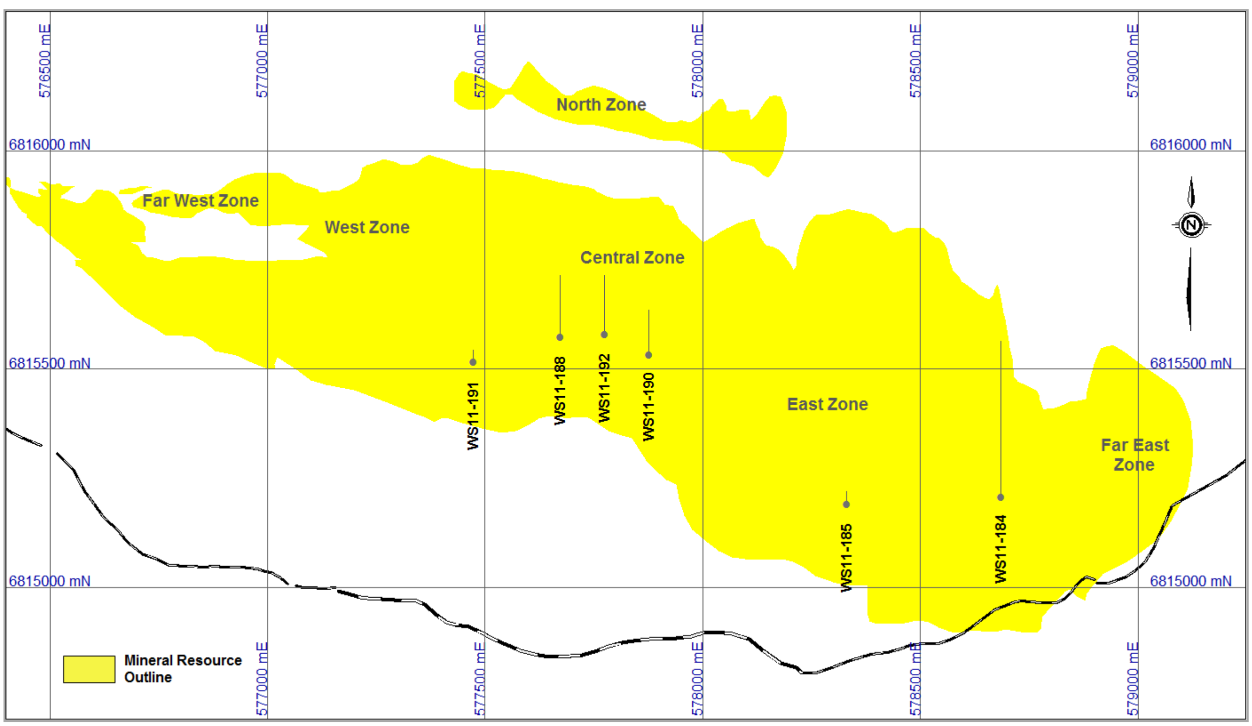
Source: Wellgreen Platinum, 2015

Table 10.2: Significant Intercepts 2011 Drilling

Hole-ID	From (m)	To (m)	Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)	PtEq (g/t)
WS11-184	8.23	111.07	102.84	0.212	0.018	0.013	0.070	0.104	0.010	0.273	1.043
WS11-184	137.16	480.67	343.51	0.330	0.168	0.016	0.248	0.288	0.037	0.526	2.008
WS11-185	8.99	59.13	50.14	0.207	0.022	0.014	0.059	0.097	0.006	0.266	1.014
WS11-188	6.40	471.40	465.00	0.285	0.186	0.016	0.335	0.321	0.050	0.517	1.972
WS11-190	4.27	294.07	289.80	0.259	0.065	0.015	0.129	0.200	0.020	0.370	1.411
WS11-190	309.59	364.57	54.98	0.230	0.260	0.013	0.352	0.302	0.069	0.490	1.872
WS11-191	7.07	85.04	77.97	0.214	0.021	0.012	0.085	0.142	0.017	0.285	1.089
WS11-192	9.45	394.35	384.90	0.299	0.146	0.016	0.281	0.303	0.038	0.498	1.901

Source: Wellgreen Platinum, 2015
Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries.
Minimum width 10 m; Maximum internal dilution 6 m

Figure 10.1: 2011 Drill Plan



Source: Wellgreen Platinum, 2015

10.2.2 2012 Drill Program

The surface drilling conducted by Wellgreen Platinum in 2012 was designed to infill the potential resource of the Wellgreen deposit in the East and West Zones. The underground program focused on upgrading the resource category of the high-grade hanging-wall gabbro in the East Zone.

Surface drilling was completed by Foraco International SA of Toronto, ON, while underground drilling was completed by DMAC Drilling of Aldergrove, BC. A total of 22 drill holes from surface and an additional 29 drill holes from underground were completed during the 2012 drill program from February to November, totalling 10,983.11 m. All holes were drilled HQ, locally down-sizing to NQ in poor ground conditions, and all drilling was run in 5 ft intervals (1.52 m).

Drill hole collar information is shown in Table 10.3 and illustrated in Figure 10.2 and Figure 10.3. Significant intercepts based on a 0.15% NiEq cut-off grade are presented in Table 10.4.

Table 10.3: Wellgreen Platinum 2012 Drill Collars

Hole-ID	UTM East	UTM North	Elev (masl)	Length (m)	Azimuth (°)	Dip (°)
WS12-193	578286.94	6815402.94	1444.19	462.50	30.00	-85.00
WS12-194	578286.94	6815402.94	1444.19	234.00	30.00	-65.00
WS12-195	578286.94	6815402.94	1444.19	201.20	30.00	-45.00
WS12-196	578286.94	6815402.94	1444.19	223.50	30.00	-55.00
WS12-197	578286.94	6815402.94	1444.19	196.50	0.00	-47.00
WS12-198	576690.53	6815849.37	1481.18	178.00	0.00	-47.00
WS12-199	578328.48	6815373.20	1426.59	200.50	0.00	-55.00
WS12-200	578328.48	6815373.20	1426.59	208.00	0.00	-65.00
WS12-201	576641.17	6815825.15	1487.28	151.00	0.00	-50.00
WS12-202	578378.65	6815356.76	1403.58	260.50	330.00	-85.00
WS12-203	578378.65	6815356.76	1403.58	325.00	330.00	-65.00
WS12-204	578378.65	6815356.76	1403.58	489.00	330.00	-45.00
WS12-205	578378.65	6815356.76	1403.58	455.00	0.00	-55.00
WS12-206	576594.58	6815827.65	1494.45	161.50	0.00	-63.00
WS12-207	576945.39	6815769.41	1479.94	267.00	0.00	-45.00
WS12-208	576991.86	6815890.49	1544.50	142.50	0.00	-72.00
WS12-209	577041.68	6815892.44	1552.25	107.00	0.00	-45.00
WS12-210	578074.55	6815527.22	1496.33	214.50	0.00	-51.00
WS12-211	577344.86	6815754.47	1569.38	75.00	0.00	-54.00
WS12-212	578077.47	6815423.84	1449.60	174.00	0.00	-45.00
WS12-213	577348.92	6815610.72	1533.85	346.50	0.00	-54.00
WS12-214	577624.29	6815574.38	1631.70	493.50	0.00	-50.00
WU12-520	578482.66	6815532.39	1298.90	156.67	200.00	33.00
WU12-521	578482.66	6815532.39	1298.90	302.36	200.00	-27.00
WU12-522	578482.66	6815532.39	1298.90	21.95	200.00	-3.00
WU12-523	578482.66	6815532.39	1298.90	271.27	200.00	-6.90
WU12-524	578482.66	6815532.39	1298.90	200.86	170.00	-9.80
WU12-525	578482.66	6815532.39	1298.90	150.27	170.00	30.00
WU12-526	578482.66	6815532.39	1298.90	101.19	147.00	36.00
WU12-527	578482.66	6815532.39	1298.90	242.32	200.00	-17.00
WU12-528	578482.66	6815532.39	1298.90	290.17	147.00	-9.00

Hole-ID	UTM East	UTM North	Elev (masl)	Length (m)	Azimuth (°)	Dip (°)
WU12-529	578482.66	6815532.39	1298.90	264.57	147.00	-30.00
WU12-530	578216.77	6815527.99	1303.19	189.28	145.00	-2.00
WU12-531	578216.77	6815527.99	1303.19	215.19	145.00	-15.00
WU12-532	578216.77	6815527.99	1303.19	193.85	145.00	25.00
WU12-533	578216.77	6815527.99	1303.19	129.24	180.00	-16.00
WU12-534	578216.77	6815527.99	1303.19	117.04	180.00	21.00
WU12-535	578216.77	6815527.99	1303.19	94.18	180.00	54.00
WU12-536	578216.77	6815527.99	1303.19	131.06	210.00	33.00
WU12-537	578216.77	6815527.99	1303.19	128.93	210.00	-3.00
WU12-538	578216.77	6815527.99	1303.19	213.06	210.00	-33.00
WU12-539	578216.77	6815527.99	1303.19	242.01	145.00	-30.00
WU12-540	578216.77	6815527.99	1303.19	304.50	145.00	-55.00
WU12-541	578154.21	6815545.54	1302.74	268.22	167.00	-60.00
WU12-542	578154.21	6815545.54	1302.74	205.44	167.00	-30.00
WU12-543	578154.21	6815545.54	1302.74	158.50	167.00	0.00
WU12-544	578154.21	6815545.54	1302.74	154.53	185.00	-10.00
WU12-545	578154.21	6815545.54	1302.74	206.65	225.00	-25.00
WU12-546	578154.21	6815545.54	1302.74	156.67	225.00	-2.00
WU12-547	578150.94	6815542.48	1302.74	75.59	225.00	25.00
WU12-548	578150.94	6815542.48	1302.74	231.34	185.00	-30.00

Source: Wellgreen Platinum, 2015

Table 10.4: Significant Intercepts 2012 Drilling

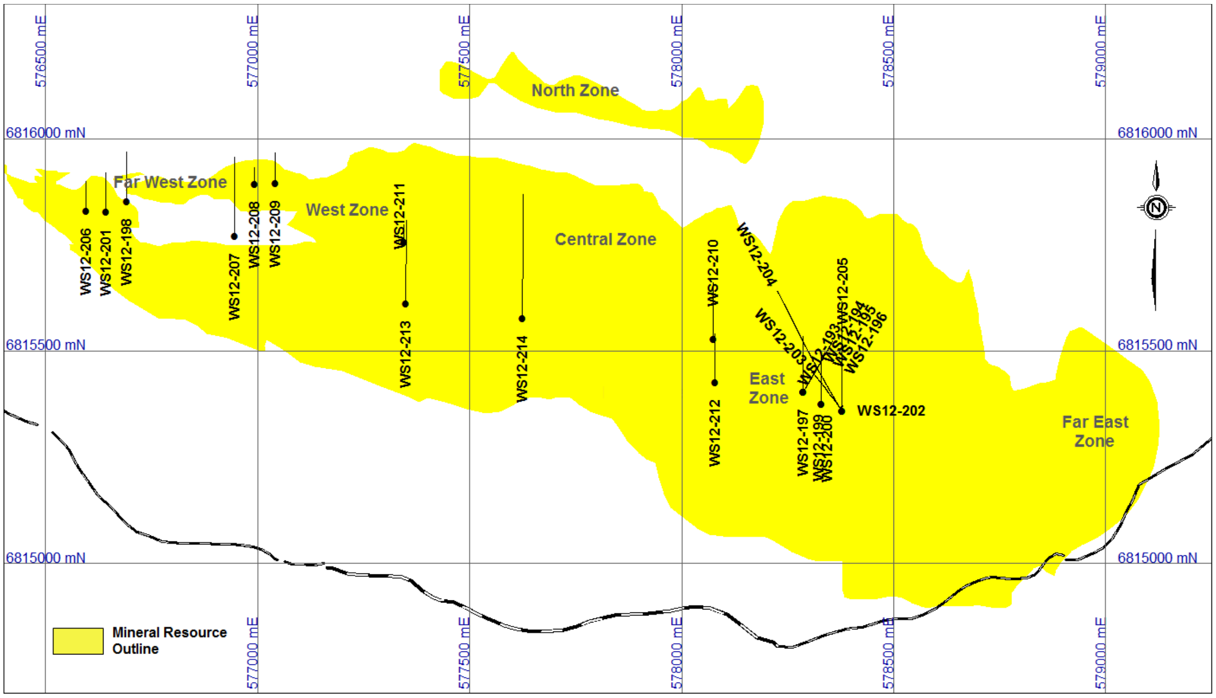
Hole	From (m)	To (m)	Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)	PtEq (g/t)
WS12-193	3.05	56.00	52.95	0.240	0.033	0.013	0.093	0.145	0.011	0.318	1.213
WS12-193	104.77	462.50	357.73	0.288	0.108	0.016	0.191	0.250	0.027	0.439	1.677
WS12-194	0.00	177.54	177.54	0.244	0.098	0.014	0.178	0.209	0.039	0.384	1.467
WS12-194	199.00	217.00	18.00	0.370	0.815	0.025	0.444	0.295	0.087	0.876	3.343
WS12-195	0.00	118.71	118.71	0.258	0.065	0.014	0.122	0.182	0.020	0.363	1.385
WS12-195	132.50	151.32	18.82	0.259	0.238	0.016	0.312	0.246	0.055	0.495	1.888
WS12-195	161.24	190.01	28.77	0.719	0.552	0.036	0.551	0.435	0.088	1.195	4.559
WS12-196	0.00	135.72	135.72	0.260	0.086	0.014	0.138	0.197	0.026	0.381	1.454
WS12-196	147.81	162.33	14.52	0.252	0.239	0.018	0.370	0.258	0.108	0.521	1.987
WS12-196	177.98	195.00	17.02	0.415	0.699	0.029	0.806	0.434	0.193	1.021	3.898
WS12-197	0.00	157.00	157.00	0.264	0.070	0.013	0.137	0.194	0.058	0.384	1.465
WS12-197	163.26	184.40	21.14	0.380	0.638	0.024	0.788	0.556	0.141	0.956	3.647
WS12-198	79.00	91.00	12.00	0.109	0.176	0.011	0.027	0.012	0.023	0.204	0.778
WS12-199	0.00	62.29	62.29	0.257	0.085	0.014	0.160	0.217	0.024	0.386	1.472
WS12-199	74.27	180.87	106.60	0.315	0.397	0.020	0.381	0.341	0.118	0.658	2.514
WS12-200	0.00	84.52	84.52	0.253	0.096	0.014	0.143	0.213	0.054	0.386	1.474
WS12-200	110.34	195.55	85.21	0.280	0.460	0.020	0.527	0.331	0.127	0.686	2.617
WS12-201	42.80	71.32	28.52	0.262	0.204	0.017	0.352	0.171	0.030	0.482	1.840
WS12-202	0.00	106.54	106.54	0.268	0.078	0.015	0.150	0.213	0.021	0.391	1.493
WS12-202	141.35	260.50	119.15	0.265	0.086	0.015	0.150	0.201	0.021	0.390	1.489
WS12-203	0.00	230.59	230.59	0.269	0.098	0.016	0.180	0.226	0.037	0.413	1.578
WS12-203	237.37	325.00	87.63	0.297	0.186	0.017	0.246	0.251	0.065	0.502	1.918
WS12-204	0.00	122.36	122.36	0.266	0.077	0.016	0.142	0.201	0.020	0.386	1.472
WS12-204	129.39	207.00	77.61	0.262	0.262	0.016	0.341	0.274	0.060	0.519	1.982
WS12-204	256.66	274.90	18.24	0.125	0.215	0.011	0.235	0.147	0.076	0.317	1.210
WS12-204	281.50	312.00	30.50	0.105	0.112	0.012	0.178	0.092	0.083	0.242	0.922
WS12-204	330.00	346.10	16.10	0.078	0.144	0.012	0.104	0.040	0.035	0.188	0.718
WS12-204	393.42	489.00	95.58	0.265	0.108	0.017	0.329	0.259	0.022	0.456	1.741
WS12-205	0.00	185.00	185.00	0.260	0.121	0.016	0.214	0.214	0.040	0.421	1.607
WS12-205	197.00	241.10	44.10	0.354	0.877	0.026	0.559	0.307	0.203	0.941	3.591
WS12-205	261.30	299.00	37.70	0.125	0.219	0.012	0.197	0.096	0.053	0.298	1.138
WS12-205	363.10	455.00	91.90	0.344	0.162	0.016	0.327	0.378	0.034	0.570	2.175
WS12-206	25.82	39.30	13.48	0.217	0.046	0.014	0.106	0.154	0.014	0.306	1.167
WS12-207	200.73	229.00	28.27	0.121	0.216	0.014	0.084	0.040	0.050	0.258	0.984
WS12-208	0.00	128.50	128.50	0.364	0.660	0.029	0.717	0.364	0.208	0.926	3.536
WS12-209	0.00	69.50	69.50	0.473	0.443	0.030	0.520	0.322	0.097	0.879	3.355
WS12-210	0.00	101.40	101.40	0.259	0.057	0.015	0.114	0.172	0.014	0.358	1.368
WS12-210	123.15	143.50	20.35	0.286	0.104	0.015	0.189	0.274	0.035	0.440	1.679
WS12-210	151.50	187.00	35.50	0.206	0.274	0.015	0.190	0.147	0.057	0.409	1.562
WS12-211	1.50	69.00	67.50	0.378	0.548	0.023	0.624	0.433	0.088	0.849	3.242
WS12-212	0.00	174.00	174.00	0.249	0.046	0.015	0.101	0.160	0.012	0.339	1.293
WS12-213	0.00	60.34	60.34	0.285	0.164	0.016	0.162	0.242	0.021	0.447	1.707
WS12-213	67.79	259.30	191.51	0.245	0.189	0.015	0.383	0.299	0.066	0.491	1.873
WS12-214	0.00	379.50	379.50	0.272	0.209	0.017	0.278	0.259	0.063	0.494	1.886

Source: Wellgreen Platinum, 2015

Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries.

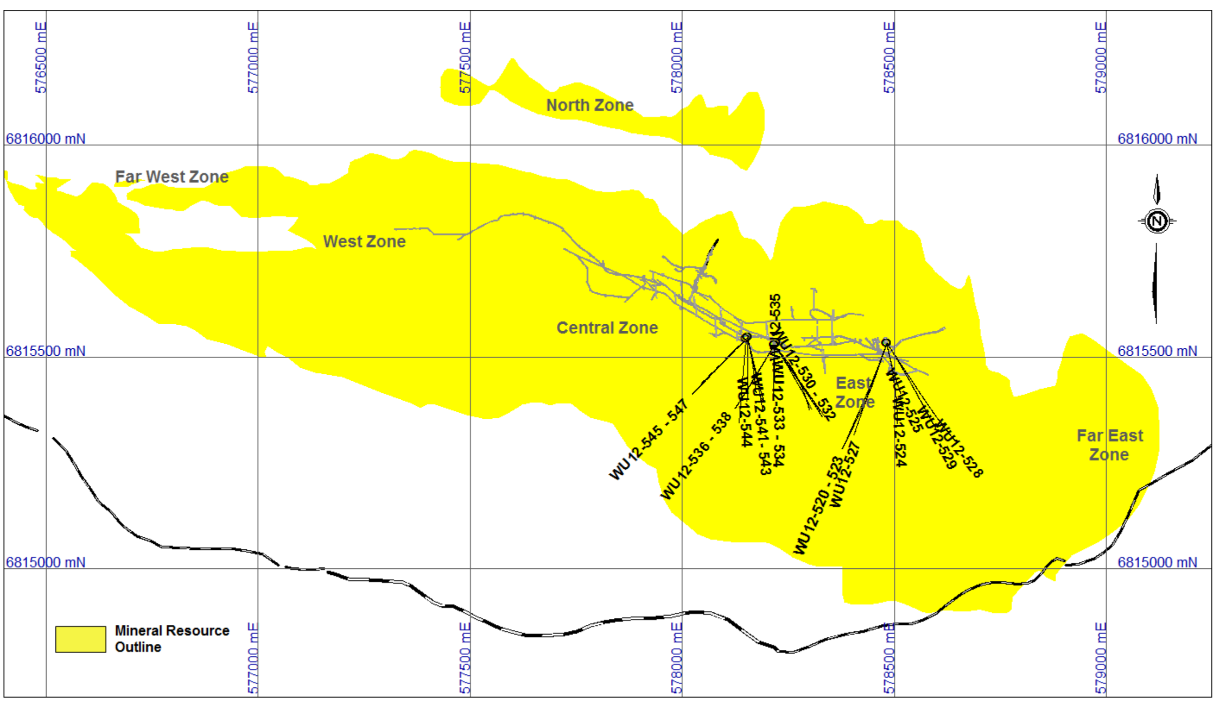
Minimum width is 10 m; maximum internal dilution is 6 m.

Figure 10.2: 2012 Surface Drilling



Source: Wellgreen Platinum, 2015

Figure 10.3: 2012 Underground Drilling



Source: Wellgreen Platinum, 2015

10.2.3 2013 Drill Program

The drilling conducted by Wellgreen Platinum in 2013 was designed to extend, expand, and upgrade the resource of the Wellgreen deposit. The program initially focused on defining and expanding the Far East Zone and a second program drilled in-fill holes in the resource with dual purpose geologic definition and ground water monitoring wells in the Wellgreen project and in areas of potential future mine infrastructure.

The first drill program was completed by Boart Longyear of South Jordan, Utah, USA. A total of nine drill holes were completed during the 2013 drill program from July to October, totalling 2,027 m. Eight of the nine holes were drilled with 5.5" RC, one of which was continued in HQ and later downsized to NQ, and one hole was drilled HQ. All drilling was run in 3 m intervals.

The second program was completed by Midnight Sun Drilling of Whitehorse. A total of 18 vertical holes were completed during the program from October to November, totaling 765.93 m. All holes were drilled with 4.5" RC and were run in 5 ft intervals (1.52 m).

Drill hole collar information is shown in Table 10.5 and illustrated in Figure 10.4. Significant intercepts based on a 0.15% NiEq cut-off grade are presented in Table 10.6.

Table 10.5: Wellgreen Platinum 2013 Drill Collars

Hole-ID	UTM East	UTM North	Elev (masl)	Length (m)	Azimuth (°)	Dip (°)
MW13-01	577001.87	6815858.76	1527.43	79.25	0.00	-90.00
MW13-02A	576141.92	6815645.82	1298.76	33.53	0.00	-90.00
MW13-02B	576133.87	6815653.00	1298.43	48.77	0.00	-90.00
MW13-03A	571062.44	6818429.77	1055.65	28.35	0.00	-90.00
MW13-03B	571072.71	6818420.25	1054.48	46.79	0.00	-90.00
MW13-04A	577731.90	6814791.54	1291.66	22.25	0.00	-90.00
MW13-04B	577732.24	6814799.08	1291.28	46.63	0.00	-90.00
MW13-05A	578587.77	6815617.52	1299.16	7.32	0.00	-90.00
MW13-06A	580589.54	6815443.36	1138.39	7.32	0.00	-90.00
MW13-06B	580593.06	6815437.95	1134.20	39.62	0.00	-90.00
MW13-07A	582993.43	6816606.88	1010.01	16.20	0.00	-90.00
MW13-07B	582991.17	6816603.73	1009.97	34.29	0.00	-90.00
MW13-08A	583907.75	6810188.50	1438.54	34.70	0.00	-90.00
MW13-08B	583903.06	6810192.89	1440.79	52.73	0.00	-90.00
MW13-09A	580295.61	6813122.73	1162.90	15.20	0.00	-90.00
MW13-09B	580289.23	6813111.33	1162.63	39.62	0.00	-90.00
WS13-215	578347.45	6815182.35	1369.79	831.00	358.00	-55.13
WS13-216	576818.93	6815833.00	1459.09	103.00	2.00	-52.02
WS13-217	578439.45	6815248.90	1357.06	353.00	0.00	-61.25
WS13-218	576864.78	6815886.04	1485.80	75.00	2.00	-50.54
WS13-219	576927.25	6815860.03	1511.67	64.00	1.00	-50.00
WS13-220	577022.75	6815836.86	1518.91	150.00	1.00	-50.86
WS13-221	577425.36	6815699.64	1590.93	175.00	1.00	-66.42
WS13-222	577609.08	6815732.81	1704.24	172.00	0.00	-71.29
WS13-223	578438.35	6815255.87	1358.85	104.00	1.00	-60.00
WS13-224	577001.87	6815858.76	1527.43	121.92	0.00	-90.00
WS13-225	578592.86	6815620.49	1299.71	91.44	0.00	-90.00

Source: Wellgreen Platinum, 2015

Table 10.6: Significant Intercepts 2013 Drilling

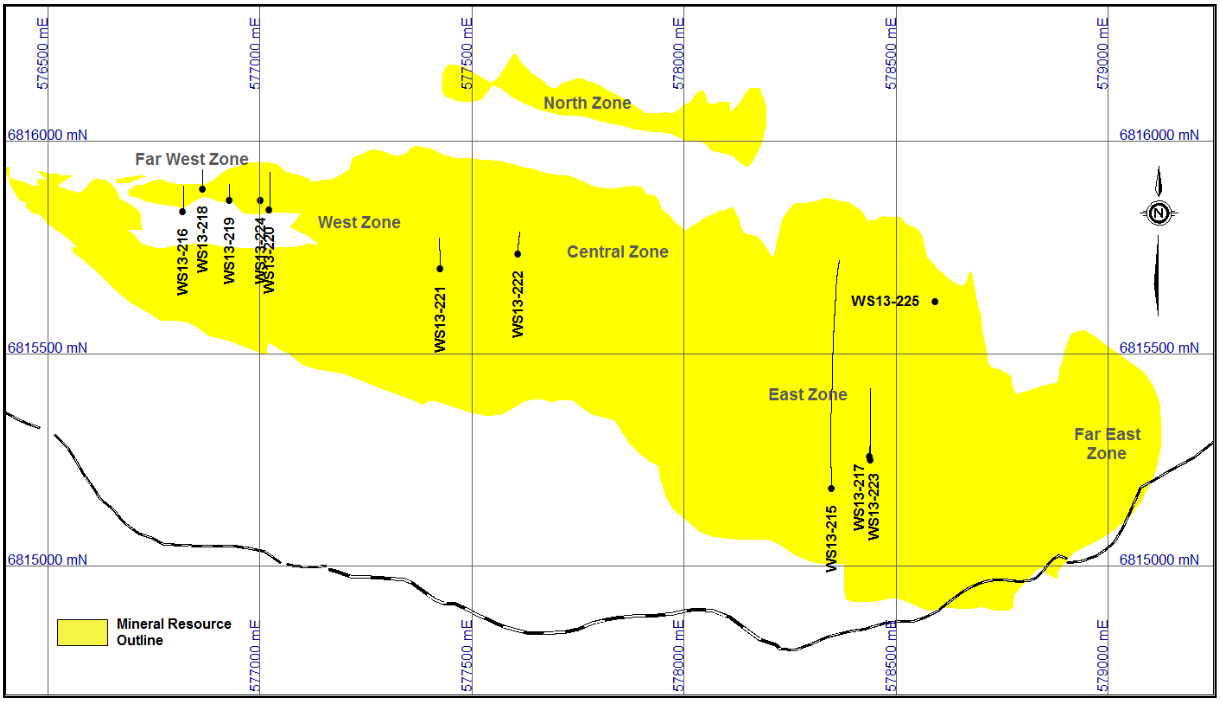
Hole	From (m)	To (m)	Width (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	NiEq %	PtEq g/t
WS13-215	0.00	762.00	762.00	0.290	0.153	0.016	0.243	0.232	0.051	0.476	1.817
WS13-215	771.00	783.00	12.00	0.127	0.252	0.009	0.077	0.033	0.028	0.262	1.000
WS13-216	43.00	79.00	36.00	0.144	0.243	0.014	0.218	0.096	0.070	0.337	1.288
WS13-217	0.00	353.00	353.00	0.285	0.089	0.016	0.182	0.236	0.039	0.429	1.636
WS13-218	0.00	22.00	22.00	0.244	0.625	0.020	0.565	0.280	0.214	0.731	2.789
WS13-219	0.00	64.00	64.00	0.289	0.661	0.022	0.814	0.407	0.282	0.889	3.394
WS13-220	0.00	150.00	150.00	0.242	0.452	0.020	0.566	0.308	0.189	0.665	2.540
WS13-221	0.00	142.00	142.00	0.242	0.194	0.015	0.299	0.254	0.093	0.466	1.780
WS13-222	0.00	172.00	172.00	0.326	0.187	0.017	0.256	0.257	0.041	0.528	2.017
WS13-223	4.20	104.00	99.80	0.269	0.053	0.015	0.149	0.183	0.021	0.378	1.444

Hole	From (m)	To (m)	Width (m)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	NiEq %	PtEq g/t
WS13-224	0.00	76.20	76.20	0.177	0.145	0.013	0.361	0.185	0.064	0.382	1.458
WS13-225	1.52	91.44	89.92	0.186	0.021	0.013	0.066	0.098	0.008	0.246	0.940
WU12-520	11.89	148.11	136.22	0.254	0.150	0.015	0.185	0.226	0.042	0.418	1.597
WU12-521	29.57	302.36	272.79	0.227	0.105	0.014	0.204	0.186	0.036	0.372	1.421
WU12-523	22.46	117.96	95.50	0.267	0.213	0.016	0.244	0.258	0.049	0.477	1.819
WU12-523	128.32	271.27	142.95	0.239	0.082	0.014	0.216	0.198	0.033	0.380	1.450
WU12-524	31.09	131.06	99.97	0.238	0.188	0.016	0.262	0.231	0.043	0.439	1.675
WU12-524	140.21	200.86	60.65	0.264	0.083	0.015	0.257	0.253	0.034	0.426	1.625
WU12-525	13.72	150.27	136.55	0.253	0.133	0.016	0.201	0.201	0.042	0.414	1.579
WU12-526	39.32	56.08	16.76	0.168	0.100	0.013	0.059	0.053	0.010	0.248	0.948
WU12-526	66.07	101.19	35.12	0.231	0.129	0.014	0.234	0.161	0.066	0.396	1.510
WU12-527	28.33	119.41	91.08	0.223	0.175	0.016	0.297	0.269	0.057	0.436	1.663
WU12-527	126.71	242.32	115.61	0.285	0.110	0.015	0.212	0.253	0.033	0.443	1.691
WU12-528	72.85	249.68	176.83	0.278	0.185	0.018	0.304	0.240	0.042	0.492	1.880
WU12-529	87.78	201.78	114.00	0.143	0.150	0.013	0.247	0.140	0.077	0.318	1.213
WU12-529	209.70	264.57	54.87	0.278	0.106	0.016	0.220	0.226	0.030	0.434	1.656
WU12-530	0.00	16.51	16.51	0.300	0.579	0.018	0.599	0.412	0.095	0.767	2.927
WU12-530	23.12	189.28	166.16	0.310	0.127	0.016	0.183	0.239	0.032	0.468	1.785
WU12-531	0.00	17.98	17.98	0.279	0.664	0.018	0.587	0.386	0.100	0.771	2.943
WU12-531	25.60	215.19	189.59	0.265	0.130	0.015	0.234	0.230	0.046	0.436	1.665
WU12-532	0.00	193.85	193.85	0.247	0.102	0.014	0.185	0.208	0.038	0.390	1.487
WU12-533	0.00	10.36	10.36	0.239	0.980	0.018	0.651	0.406	0.121	0.870	3.319
WU12-533	19.51	129.24	109.73	0.312	0.120	0.015	0.191	0.252	0.030	0.469	1.789
WU12-534	0.00	117.04	117.04	0.279	0.135	0.016	0.198	0.223	0.036	0.440	1.680
WU12-535	0.00	10.87	10.87	0.218	0.459	0.015	0.595	0.382	0.253	0.668	2.549
WU12-535	18.03	94.18	76.15	0.289	0.131	0.015	0.223	0.242	0.042	0.460	1.754
WU12-536	15.51	131.06	115.55	0.270	0.076	0.015	0.139	0.187	0.025	0.387	1.477
WU12-537	0.00	128.93	128.93	0.279	0.137	0.015	0.214	0.264	0.043	0.452	1.726
WU12-538	0.00	17.98	17.98	0.161	0.478	0.013	0.710	0.388	0.111	0.614	2.344
WU12-538	25.76	213.06	187.30	0.268	0.102	0.015	0.199	0.220	0.039	0.417	1.592
WU12-539	0.00	21.03	21.03	0.440	0.774	0.027	0.803	0.720	0.121	1.091	4.166
WU12-539	27.13	242.01	214.88	0.279	0.145	0.015	0.231	0.255	0.036	0.457	1.746
WU12-540	4.57	21.03	16.46	0.430	0.766	0.024	0.852	0.543	0.195	1.079	4.117
WU12-540	36.27	59.13	22.86	0.435	0.718	0.018	1.260	1.024	0.240	1.238	4.726
WU12-540	80.47	304.50	224.03	0.284	0.132	0.014	0.218	0.240	0.047	0.451	1.722
WU12-541	0.00	44.04	44.04	0.236	0.474	0.015	0.582	0.328	0.120	0.652	2.490
WU12-541	54.99	268.22	213.23	0.351	0.144	0.017	0.236	0.348	0.034	0.543	2.073
WU12-542	17.27	205.44	188.17	0.273	0.108	0.016	0.205	0.246	0.036	0.430	1.642
WU12-543	13.94	158.11	144.17	0.269	0.090	0.015	0.149	0.218	0.022	0.398	1.519
WU12-544	11.73	154.53	142.80	0.304	0.103	0.016	0.193	0.274	0.027	0.459	1.751
WU12-545	22.76	203.61	180.85	0.280	0.095	0.016	0.172	0.250	0.024	0.422	1.609
WU12-546	19.93	156.67	136.74	0.271	0.083	0.015	0.144	0.207	0.022	0.393	1.502
WU12-547	0.00	75.59	75.59	0.249	0.118	0.014	0.190	0.241	0.042	0.404	1.540
WU12-548	16.76	231.34	214.58	0.262	0.090	0.015	0.187	0.222	0.028	0.402	1.533

Source: Wellgreen Platinum, 2015

Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries.

Figure 10.4: 2013 Drilling



Source: Wellgreen Platinum, 2015

10.2.4 2013 Re-Sampling of Historic Drill Core

Wellgreen Platinum sampled and assayed previously non-sampled core intervals and re-assayed all available sampled intervals from the 1987-1988 programs in 2013. A total of 3,087 samples were analyzed from 108 holes (8,462 metres). The locations of these drill holes are shown in Figure 10.5. Significant intercepts based on a 0.15% NiEq cut-off grade are presented in Table 10.7.

Table 10.7: Significant Intercepts From Re-sampled 1987-1988 Core

Hole	From (m)	To (m)	Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)	PtEq (g/t)
WS87-061	42.50	84.30	41.80	0.849	0.263	0.628	0.025	0.846	0.415	0.213	3.242
WS87-062	73.25	119.40	46.15	0.853	0.270	0.626	0.021	0.814	0.475	0.232	3.257
WS87-064	2.13	56.00	53.87	0.792	0.337	0.456	0.021	0.651	0.430	0.148	3.023
WS87-065	2.44	23.47	21.03	0.794	0.338	0.369	0.021	0.726	0.472	0.182	3.031
WS87-065	30.78	104.10	73.32	0.905	0.360	0.625	0.025	0.757	0.482	0.090	3.455
WS87-066	2.44	78.24	75.80	0.500	0.318	0.131	0.015	0.235	0.329	0.033	1.908
WS87-066	89.76	103.18	13.42	1.065	0.534	0.683	0.034	0.617	0.413	0.079	4.065
WS87-067	7.64	151.50	143.86	0.481	0.297	0.152	0.014	0.235	0.291	0.038	1.837
WS87-068	3.05	49.93	46.88	0.530	0.314	0.175	0.013	0.282	0.390	0.038	2.024

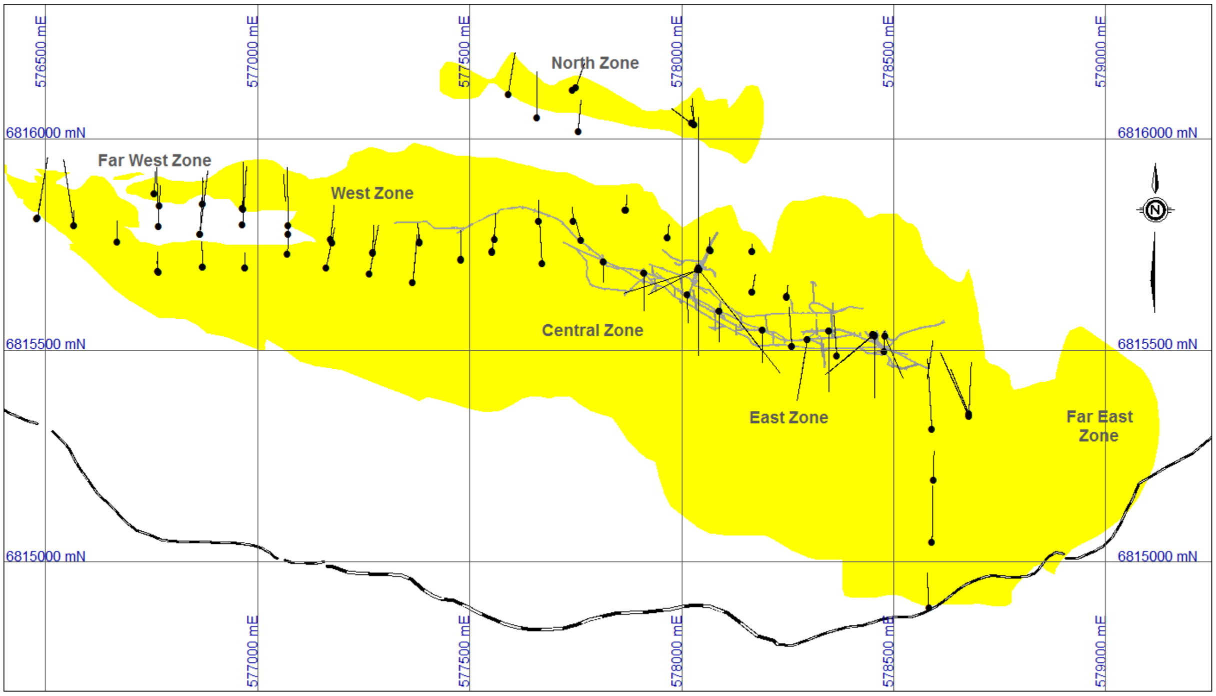
Hole	From (m)	To (m)	Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)	PtEq (g/t)
WS87-069	3.05	33.22	30.17	0.420	0.277	0.109	0.014	0.173	0.257	0.020	1.604
WS87-070	6.10	56.00	49.90	0.324	0.219	0.078	0.012	0.121	0.175	0.016	1.235
WS87-071	18.29	102.00	83.71	0.359	0.228	0.104	0.012	0.155	0.212	0.032	1.371
WS87-072	4.88	38.85	33.97	0.396	0.264	0.116	0.012	0.144	0.223	0.022	1.511
WS87-073	9.10	28.90	19.80	0.374	0.237	0.068	0.011	0.217	0.253	0.023	1.427
WS87-074	10.51	47.55	37.04	0.389	0.160	0.259	0.013	0.276	0.147	0.107	1.483
WS87-074	61.87	83.80	21.93	0.599	0.166	0.491	0.015	0.634	0.250	0.160	2.287
WS87-075	13.22	49.15	35.93	0.563	0.346	0.167	0.016	0.341	0.300	0.016	2.149
WS87-076	4.88	39.95	35.07	0.799	0.511	0.167	0.020	0.476	0.509	0.026	3.048
WS87-077	3.05	115.15	112.10	0.460	0.193	0.278	0.016	0.363	0.210	0.094	1.756
WS87-078	3.81	84.43	80.62	0.504	0.308	0.109	0.014	0.301	0.384	0.024	1.922
WS87-079	1.83	19.87	18.04	0.621	0.380	0.120	0.015	0.394	0.499	0.030	2.370
WS87-080	3.05	36.00	32.95	0.755	0.443	0.185	0.017	0.490	0.640	0.037	2.884
WS87-081	3.05	95.40	92.35	0.475	0.200	0.277	0.017	0.400	0.226	0.070	1.812
WS87-082	1.22	26.43	25.21	0.274	0.183	0.074	0.011	0.100	0.149	0.013	1.047
WS87-083	6.40	42.06	35.66	0.338	0.216	0.094	0.012	0.144	0.203	0.019	1.289
WS87-084	10.73	59.30	48.57	0.296	0.181	0.086	0.010	0.155	0.183	0.018	1.128
WS87-085	9.14	46.23	37.09	0.394	0.182	0.127	0.014	0.359	0.261	0.075	1.505
WS87-085	55.23	67.75	12.52	0.326	0.039	0.034	0.002	0.749	0.515	0.038	1.244
WS87-086	3.05	69.70	66.65	0.569	0.370	0.176	0.016	0.254	0.291	0.027	2.170
WS87-087	3.66	31.90	28.24	0.787	0.207	0.671	0.016	0.964	0.275	0.118	3.003
WS87-087	39.70	162.72	123.02	0.755	0.230	0.588	0.019	0.714	0.342	0.245	2.882
WS87-088	3.05	20.32	17.27	0.469	0.304	0.140	0.016	0.174	0.295	0.028	1.789
WS87-088	34.13	150.00	115.87	0.553	0.326	0.213	0.016	0.260	0.349	0.053	2.109
WS87-090	4.32	52.32	48.00	0.406	0.275	0.087	0.015	0.156	0.240	0.021	1.550
WS87-090	64.32	118.61	54.29	0.413	0.262	0.120	0.015	0.171	0.245	0.036	1.576
WS87-090	158.00	169.84	11.84	0.660	0.258	0.397	0.020	0.590	0.393	0.102	2.519
WS87-091	3.05	75.40	72.35	0.367	0.246	0.074	0.012	0.157	0.231	0.017	1.400
WS87-092	11.15	95.15	84.00	0.701	0.279	0.437	0.000	0.657	0.407	0.184	2.677
WS87-093	9.45	70.10	60.65	0.373	0.226	0.107	0.012	0.184	0.268	0.032	1.425
WS87-094	20.42	148.15	127.73	0.419	0.252	0.137	0.014	0.205	0.232	0.050	1.599
WS87-095	3.00	22.55	19.55	0.389	0.225	0.110	0.014	0.272	0.184	0.039	1.486
WS87-096	151.44	173.61	22.17	0.369	0.160	0.328	0.015	0.170	0.074	0.065	1.408
WS87-097	8.45	75.90	67.45	0.442	0.253	0.151	0.013	0.235	0.362	0.026	1.689
WS87-097	108.34	128.00	19.66	0.303	0.203	0.082	0.010	0.099	0.199	0.015	1.158
WS87-098	71.93	161.24	89.31	0.499	0.246	0.273	0.008	0.306	0.302	0.104	1.905
WS87-099	12.00	28.84	16.84	0.194	0.098	0.147	0.011	0.062	0.029	0.029	0.742
WS87-100	3.35	82.54	79.19	0.360	0.222	0.119	0.014	0.175	0.176	0.023	1.373
WS87-102	1.83	210.48	208.65	0.275	0.203	0.025	0.013	0.084	0.130	0.014	1.050
WS87-103	3.66	110.20	106.54	0.445	0.266	0.116	0.015	0.262	0.267	0.047	1.700
WS87-104	151.79	175.00	23.21	0.385	0.095	0.319	0.011	0.365	0.177	0.181	1.469
WS87-104	182.95	215.49	32.54	0.322	0.140	0.292	0.012	0.139	0.060	0.067	1.231
WS87-105	3.66	45.25	41.59	0.371	0.216	0.093	0.012	0.245	0.221	0.039	1.415
WS88-106	3.05	52.25	49.20	0.400	0.227	0.212	0.000	0.308	0.123	0.000	1.526
WS88-107	90.95	116.00	25.05	0.382	0.179	0.369	0.015	0.105	0.042	0.066	1.458

Hole	From (m)	To (m)	Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)	PtEq (g/t)
WS88-108	16.56	85.28	68.72	0.260	0.204	0.017	0.013	0.055	0.090	0.015	0.993
WS88-108	90.85	108.81	17.96	0.190	0.143	0.020	0.010	0.046	0.081	0.011	0.725
WS88-109	34.28	55.35	21.07	0.660	0.374	0.130	0.016	0.633	0.295	0.046	2.520
WS88-110	4.70	35.46	30.76	0.264	0.209	0.019	0.014	0.050	0.090	0.006	1.007
WS88-110	43.63	165.87	122.24	0.397	0.249	0.109	0.015	0.202	0.198	0.034	1.517
WS88-111	74.26	124.80	50.54	0.416	0.257	0.089	0.014	0.254	0.208	0.051	1.588
WS88-112	12.00	32.18	20.18	0.250	0.190	0.019	0.013	0.058	0.115	0.011	0.954
WS88-112	39.90	71.74	31.84	0.410	0.260	0.072	0.016	0.235	0.249	0.027	1.566
WS88-112	79.82	154.80	74.98	0.635	0.250	0.345	0.018	0.585	0.331	0.168	2.425
WS88-113	39.12	61.17	22.05	0.422	0.180	0.342	0.015	0.246	0.102	0.084	1.610
WS88-114	4.96	69.03	64.07	0.301	0.214	0.057	0.013	0.097	0.130	0.017	1.149
WS88-114	76.25	355.42	279.17	0.420	0.263	0.105	0.014	0.216	0.254	0.031	1.601
WS88-117	203.14	235.00	31.86	0.217	0.109	0.148	0.014	0.037	0.017	0.100	0.829
WS88-119	59.75	83.57	23.82	0.300	0.218	0.038	0.014	0.098	0.138	0.012	1.146
WS88-120	8.00	27.46	19.46	0.402	0.257	0.067	0.015	0.258	0.186	0.027	1.534
WS88-120	50.75	123.30	72.55	0.470	0.252	0.188	0.020	0.323	0.160	0.060	1.793
WS88-120	132.35	270.66	138.31	0.494	0.303	0.114	0.017	0.292	0.304	0.036	1.885
WS88-122	61.25	140.61	79.36	0.209	0.161	0.014	0.013	0.048	0.065	0.012	0.799
WS88-123	110.64	131.92	21.28	0.315	0.193	0.110	0.010	0.126	0.191	0.042	1.204
WS88-124	79.91	118.14	38.23	0.367	0.259	0.052	0.015	0.132	0.188	0.034	1.403
WS88-124	144.01	155.55	11.54	0.458	0.195	0.188	0.010	0.551	0.194	0.049	1.749
WS88-125	120.85	133.25	12.40	0.240	0.164	0.039	0.009	0.085	0.124	0.043	0.915
WS88-127	3.35	38.80	35.45	0.354	0.217	0.104	0.011	0.167	0.263	0.026	1.353
WS88-128	17.00	58.52	41.52	0.419	0.279	0.092	0.014	0.179	0.258	0.018	1.598
WS88-129	22.02	50.49	28.47	0.304	0.223	0.031	0.014	0.104	0.138	0.013	1.160
WS88-130	11.00	61.67	50.67	0.308	0.191	0.086	0.012	0.172	0.152	0.013	1.177
WS88-131	24.00	39.47	15.47	0.287	0.226	0.026	0.013	0.063	0.109	0.003	1.094
WS88-131	117.35	142.60	25.25	0.516	0.302	0.154	0.012	0.270	0.463	0.035	1.969
WS88-132	7.92	75.82	67.90	0.350	0.221	0.082	0.013	0.195	0.167	0.030	1.337
WS88-133	9.14	28.85	19.71	0.507	0.295	0.091	0.015	0.363	0.345	0.073	1.934
WS88-133	38.96	98.20	59.24	0.394	0.237	0.087	0.014	0.254	0.216	0.041	1.504
WS88-134	4.88	44.78	39.90	0.360	0.220	0.096	0.013	0.220	0.167	0.025	1.376
WS88-135	11.30	47.18	35.88	0.299	0.189	0.056	0.013	0.176	0.148	0.020	1.141
WS88-137	2.97	75.00	72.03	0.402	0.271	0.109	0.000	0.222	0.258	0.000	1.534
WS88-137	82.91	135.00	52.09	0.430	0.205	0.202	0.000	0.444	0.280	0.000	1.642
WS88-137	146.00	172.90	26.90	0.726	0.387	0.336	0.000	0.653	0.358	0.000	2.771
WS88-138	59.27	141.80	82.53	0.631	0.328	0.348	0.019	0.378	0.272	0.067	2.409
WS88-139	4.27	199.38	195.11	0.471	0.259	0.156	0.016	0.318	0.278	0.054	1.797
WS88-139	213.66	375.60	161.94	0.887	0.372	0.534	0.022	0.733	0.448	0.174	3.385
WS88-140	24.38	63.40	39.02	0.313	0.184	0.138	0.000	0.216	0.174	0.000	1.193
WS88-141	0.00	97.00	97.00	0.340	0.238	0.071	0.013	0.116	0.168	0.025	1.299
WS88-141	108.20	145.00	36.80	0.318	0.158	0.125	0.009	0.202	0.197	0.102	1.212
WS88-142	16.15	214.42	198.27	0.435	0.295	0.107	0.015	0.166	0.224	0.028	1.659
WU88-483	83.50	130.60	47.10	0.438	0.193	0.236	0.016	0.341	0.190	0.096	1.671
WU88-484	119.90	163.54	43.64	0.612	0.287	0.367	0.021	0.409	0.234	0.098	2.334

Hole	From (m)	To (m)	Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)	PtEq (g/t)
WU88-485	21.45	38.95	17.50	0.796	0.455	0.522	0.025	0.301	0.208	0.040	3.038
WU88-485	45.30	184.56	139.26	0.799	0.290	0.586	0.018	0.704	0.376	0.165	3.049
WU88-486	76.00	135.60	59.60	0.997	0.215	0.721	0.019	1.219	0.620	0.428	3.806
WU88-486	156.25	168.10	11.85	0.489	0.230	0.397	0.015	0.246	0.165	0.027	1.865
WU88-487	114.10	133.10	19.00	0.475	0.214	0.355	0.016	0.269	0.104	0.111	1.811
WU88-487	142.76	157.50	14.74	0.776	0.245	0.691	0.018	0.665	0.313	0.182	2.962
WU88-487	192.90	207.30	14.40	0.335	0.198	0.164	0.011	0.148	0.137	0.024	1.279
WU88-488	15.85	140.78	124.93	0.509	0.229	0.284	0.016	0.388	0.244	0.090	1.944
WU88-489	1.57	22.25	20.68	0.468	0.130	0.330	0.011	0.489	0.302	0.157	1.786
WU88-490	15.95	39.30	23.35	0.734	0.278	0.520	0.015	0.575	0.513	0.124	2.802
WU88-490	57.20	109.90	52.70	0.670	0.383	0.234	0.018	0.337	0.508	0.089	2.556
WU88-491	19.41	43.47	24.06	0.544	0.226	0.361	0.013	0.463	0.270	0.054	2.075
WU88-491	52.70	146.90	94.20	0.442	0.275	0.130	0.016	0.210	0.248	0.043	1.688
WU88-492	74.30	109.25	34.95	0.429	0.079	0.078	0.006	0.861	0.414	0.156	1.639
WU88-493	10.30	49.00	38.70	0.558	0.244	0.317	0.016	0.428	0.246	0.137	2.128
WU88-493	68.70	79.55	10.85	0.619	0.352	0.219	0.018	0.335	0.450	0.062	2.362
WU88-494	0.00	35.35	35.35	0.793	0.332	0.451	0.021	0.629	0.507	0.163	3.029
WU88-495	1.10	46.93	45.83	1.142	0.301	0.875	0.024	1.206	0.814	0.312	4.360
WU88-495	67.01	92.70	25.69	0.548	0.293	0.185	0.015	0.371	0.423	0.053	2.090
WU88-495	94.60	105.16	10.56	0.164	0.028	0.014	0.002	0.216	0.350	0.114	0.624
WU88-496	2.23	38.16	35.93	2.009	0.787	1.572	0.047	1.413	1.169	0.273	7.668
WU88-497	6.24	17.10	10.86	0.360	0.060	0.027	0.003	0.362	0.378	0.650	1.372
WU88-498	48.51	181.66	133.15	1.185	0.646	0.573	0.028	0.686	0.634	0.119	4.523
WU88-500	8.40	90.11	81.71	0.840	0.398	0.481	0.021	0.582	0.394	0.151	3.206
WU88-500	98.76	114.00	15.24	0.676	0.310	0.309	0.023	0.521	0.366	0.160	2.580
WU88-501	33.50	105.50	72.00	0.781	0.287	0.527	0.023	0.603	0.344	0.300	2.979
WU88-501	114.50	126.10	11.60	0.573	0.204	0.380	0.014	0.519	0.297	0.165	2.186
WU88-501	137.40	154.84	17.44	0.473	0.279	0.126	0.014	0.289	0.314	0.047	1.806
WU88-502	8.10	170.69	162.59	0.412	0.216	0.170	0.016	0.289	0.177	0.050	1.572
WU88-503	25.95	91.44	65.49	0.445	0.262	0.148	0.016	0.242	0.260	0.039	1.698
WU88-504	1.60	13.11	11.51	0.464	0.274	0.139	0.013	0.280	0.313	0.030	1.772
WU88-505	0.00	49.68	49.68	0.518	0.336	0.127	0.015	0.236	0.345	0.028	1.979
WU88-507	8.80	54.50	45.70	0.545	0.304	0.247	0.011	0.357	0.291	0.016	2.079
WU88-508	134.50	146.29	11.79	1.308	0.481	0.875	0.036	1.103	0.742	0.325	4.994
WU88-508	167.30	245.06	77.76	0.458	0.290	0.120	0.016	0.222	0.266	0.034	1.750
WU88-509	173.23	197.15	23.92	0.610	0.276	0.171	0.013	0.575	0.636	0.082	2.327
WU88-509	207.50	217.93	10.43	0.585	0.330	0.196	0.022	0.364	0.338	0.049	2.232
WU88-510	166.03	221.59	55.56	0.673	0.262	0.388	0.018	0.600	0.403	0.159	2.571
WU88-511	190.20	248.72	58.52	0.717	0.169	0.531	0.017	0.815	0.435	0.275	2.738
WU88-514	186.42	336.19	149.77	0.459	0.287	0.132	0.016	0.206	0.289	0.038	1.752
WU88-515	153.15	164.90	11.75	0.163	0.069	0.087	0.013	0.109	0.055	0.031	0.621
WU88-515	182.40	366.80	184.40	0.426	0.253	0.135	0.014	0.218	0.269	0.044	1.625
WU88-515	377.90	401.73	23.83	0.436	0.285	0.108	0.015	0.177	0.278	0.029	1.663
WU88-516	451.71	467.46	15.75	0.267	0.141	0.035	0.008	0.061	0.091	0.336	1.020

Source: Wellgreen Platinum, 2015

Figure 10.5: Re-sampled 1987-1988 Drill Holes



Source: Wellgreen Platinum, 2015

10.3 Recovery

Core recovery is generally good to excellent and is not considered to be a factor affecting resource estimation.

10.4 Collar Surveys

Prior to the 2013 field season, drill collars were spotted with a compass and chain off the local mine grid, with the final completed collars surveyed with a hand held GPS, compass and chain or a total station GPS, or not at all. In 2013 all collars were spotted using a hand-held GPS and surveyed with a DGPS.

10.5 Downhole Surveys

Down-hole surveys were performed differently in different years depending on the operator at the time. HudBay, Archer-Cathro, and Northern Platinum (from 1996-2005) used acid dip tests to determine hole deviation, either at regular intervals or, in the case of Northern Platinum, at the end of each hole. Coronation Minerals used acid dip tests in 2006 and 2007, and used a Reflex Single Shot magnetic tool in 2008. Northern Platinum (from 2009-2010) and Prophecy Resources Corp. (2011) reported use of a ReflexIt© tool, and survey readings were collected approximately 9 m off the bottom of the hole and at approximately 152 m intervals up the hole, however, no azimuth data was recorded.

In 2012, Wellgreen Platinum completed down-hole surveys using the Reflex Maxibor II© tool. Survey readings were collected every 3 m up the hole. Some measurements or surveys were subject to tool malfunction and deemed unreliable.

In 2013, Wellgreen Platinum completed down-hole surveys using the Icefield Tools Gyro Shot® tool. Survey readings were collected approximately 9 m off the bottom of the hole and at every 18 m up the hole.

Geotechnical/groundwater holes drilled in the Wellgreen deposit were spotted with a hand-held GPS and were surveyed with differential GPS (DGPS). Down-hole surveys were not conducted due to the shallow lengths and vertical dips of the holes.

10.6 Sample Length/True Thickness

The mineralized zone is irregular and not tabular in shape. True thickness cannot be determined and was not used as a factor in the resource model.

10.6.1 2014 Drill Program

During October and November of 2014, Wellgreen Platinum completed 2,916.49 m of drilling in 8 holes. Most holes were started with an RC rig and finished with a core rig. Holes were sampled but have not yet been analyzed. Drill hole collar information is shown in Table 10.8.

Table 10.8 Wellgreen Platinum 2014 Drill Collars

Hole-ID	UTM East	UTM North	Elev (masl)	Length (m)	Azimuth (°)	Dip (°)
WS14-226	577369	6815459	1516	773	0	-58.99
WS14-227	577474	6815429	1545	590	0	-56.06
WS14-228	577550	6815555	1596	413.74	0	-67.07
WS14-229	577650	6815475	1590	590	0	-69.52
WS14-230	577285	6815501	1483	63.85	0	-55
WS14-231	578679	6815345	1273	430.1	0	-74.2
MW14-10A	578779	6812257	1145	6.3	0	-90
MW14-10B	578782	6812253	1145	49.5	0	-90

Source: Geosim, 2015

11 Sample Preparation, Analyses and Security

11.1 Sampling Methods

11.1.1 Historic Drill Programs 1952-1988

Sampling details for historic programs have not been verified by GeoSim. No documented quality assurance/quality control (QA/QC) programs were available for review. However, based on assay results it appears that Hudson Yukon Mining only sampled intervals considered to be well mineralized.

Drill programs in 1987-1988 were supervised by Archer Cathro & Associates Ltd. Assessment reports filed from these years do not document sampling or analytical details, however only "mineralized" intervals were sampled. In 1987 mineralized portions of 53 older underground core holes were re-assayed for Cu, Ni, Co, Au, Pt, and Pd.

Wellgreen Platinum sampled and assayed previously non-sampled core intervals and re-assayed all available sampled intervals from the 1987-1988 programs in 2013. A total of 3,087 samples were analyzed from 108 holes (8,462 m).

11.1.2 Northern Platinum Programs 1996-2005

There is no documentation on sampling details for the older Northern Platinum programs, however based on handwritten assays in paper drill logs samples were taken every 5 ft (1.52 m) and were assayed for Cu, Ni, and Co, and sometimes for Pt, Pd, and Au.

11.1.3 Coronation Minerals Programs 2006-2008

The drill core was logged and sampled by the company's geologist and assistants under the direct supervision of Mr. Rory Calhoun, P.Geo., at the designated facilities of the Coronation Minerals base camp on site. The geologist would record lithology, mineralization, structures, sample number, etc., and the assistants would record the geotechnical data (rock quality designation (RQD)) and recovery.

Sample length was variable based on lithology and mineralization observed by the geologist and the core was marked accordingly. Most sampled intervals were 1.52 m or 5 ft in length. The assistant transported the core into the saw shack and cut it in half using a core saw. After cutting, the core was returned to the core tray and the geologist would sample it. Half of the split core would be placed in a plastic sample bag with the sample tag. The sample number was also written on the outside of each bag for easy identification. No sample tags were left in the core trays.

All of the data from logging the core was recorded in hand written logs and then transferred to Microsoft Excel™ spreadsheets, for later import into a geological software package.

11.1.4 Northern Platinum 2009-2010 Programs

All samples, including field-inserted Standards and Blanks, were sent to Loring Laboratories in Calgary, AB for assaying. Similar to the Coronation Minerals programs, Northern Platinum sampled core based on lithology and observed mineralization, and where no contacts were present used a nominal 5 ft (1.52 m) sample interval.

11.1.5 Wellgreen Platinum Programs 2011-2013

The sampling methodology adopted by Wellgreen Platinum was as follows:

The drill core is delivered to the core shack by the drill contractor, and the core boxes are sorted and placed in groups of three. The group of boxes is photographed, and run markers and other marker blocks are checked for accuracy.

The geologist or technician collects RQD and recovery data, and the geologist logs the core. Prior to 2013 all recovery, RQD, and geology data was hand-written onto paper forms which were then entered into spreadsheets. From 2013 onwards, all of this data is captured digitally in an Access database.

Ideally there is only one geologist logging each individual hole for consistency. The minimum sample unit is 2 ft; maximum sample length is 3 m, and samples do not cross lithological contacts. In 2013, the sample interval was written on a lab-provided tag which was then stapled into the box. The tag displays the sample number and interval. Previously, the sample was marked on the box with the footage and sample number in permanent marker.

Processed boxes of core are taken to the core cutting facility for cutting by a technician. The saw uses fresh water which drains into sump below the floor before decanting to the creek. The core is cut and the technician places the samples in clean plastic bags with a sample tag. The sample number is written on the outside of the sample bag. Starting in 2012, half of the core was taken for possible future metallurgical sample while a quarter was left in the box and another quarter sent to the lab for assay.

11.1.6 Wellgreen Platinum Soil Geochemical Sampling 2012

Soil samples were taken on a 25 m nominal spacing across the Property, and soil augers and mattocks were used to try to get to the B or C horizons. The samples were placed in Kraft sample bags and shipped to the ALS Global preparation facility in Whitehorse, YT. Sample pulps were then sent to ALS Global's lab in Vancouver, BC for assay.

The following QA/QC controls were inserted into the sample batches before shipment:

Blanks CDN0BL-10 (Granitic Material): 3 g of material was inserted every 25th sample and every 100th sample contained 30g of material. All samples were analyzed by the ME-ICP process while only the larger 30 g standards contained enough material to pass through the Pt-Pd-Au fire assay and ICP-AES finish. These occurred on sample tag numbers ending in 11, 36, 61, and 86.

GSC Standard (Till-1): 3g of material was inserted every 25th sample and every 100th sample contained 30 g of material. All samples were be analyzed by Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP-AES) while only the larger 30 g standards contained enough material to pass through the Pt-Pd-Au fire assay and finish. These occurred on sample tag numbers ending in 5, 30, 55, 80 and 100.

Duplicates: Duplicates were collected from within 2 m of the original sample location every 25th sample. These occurred on sample tag numbers ending in 2, 27, 52, and 77.

Field Standard: Field standards were collected from two suitable locations from the central and eastern portions of the Property Grid. Material was dried, sieved to fines, hand-mixed, and selected using the 'Method of Dips'. 100 g of field standard was inserted every 25th sample. These occurred on sample tag numbers ending in 10, 35, 60 and 85. The field standard collection process was photographed.

11.2 Density Determinations

A total of 6,705 specific gravity measurements were made using the water immersion method on core samples from the 1987 and 2013 drill programs. Specific gravity measurements during the 2012 field season were done at ALS using a pycnometer.

11.3 Metallurgical Sampling

Select intervals from drilling in a number of programs beginning in the 1980s and 2000s have been selected for use in metallurgical test work which is on-going (See Section 13).

11.4 Sample Preparation and Analysis

11.4.1 Historic Programs 1952-1988

Hudson Yukon Mining assayed all core at their internal lab in Flin Flon, Manitoba, and Archer-Cathro assayed all core at Bondar-Clegg & Company Ltd. in North Vancouver. No sample preparation details are available from the Hudson Yukon Mining documentation; however, the Archer-Cathro core was analyzed for Pt and Pd by fire assay, and Cu and Ni by atomic absorption (AAS). In addition, some samples were analyzed for the other PMEs and as such underwent neutron activation.

While no documentation exists for how samples were prepared from the historic and the more recent programs (conducted from 1996-2005), it was assumed that sample preparation methods at the various laboratories are generally consistent with current industry best practices since reputable firms were utilized.

11.4.2 Northern Platinum 1996-2010 Programs

Most samples, including field-inserted Standards and Blanks, were sent to Loring Laboratories in Calgary, AB for assaying. In 2009 samples were also analyzed at ALS Global in North Vancouver, BC. Loring Laboratories has ISO 9001:2000 certification and ALS Global has ISO/IEC 17025:2005 and ISO 9001:2000 certification.

A 30 element package, including copper, nickel, and cobalt reported in parts per million was analyzed by aqua regia "partial digestion" followed by ICP analyses. Gold, platinum, palladium and rhodium were analyzed by four acid digestion followed by a 30 g fire assay with an atomic absorption (AA) finish.

11.4.3 Wellgreen Platinum Programs 2011-2013

All samples collected in 2011 and 2012, including field-inserted Standards and Blanks, were sent to ALS Global in Vancouver, BC, for assaying. All samples in 2013 were sent to ACME Laboratories in

Vancouver, BC, for analysis. Both labs have ISO/IEC 17025:2005 and ISO 9001:2000 certification, and are independent of Wellgreen Platinum. The samples were assayed for copper, nickel, cobalt, gold, platinum, and palladium.

The following is a brief description of the sample preparation:

- Samples are sorted into numerical order and then dried;
- Once dried, the material was crushed using a jaw crusher; and
- The sample is then split to get a 250 g sample for pulverizing.

The total 250 g of split sample is pulverized to 85% passing 75 micrometres (μm).

Gold, platinum, palladium were assayed by fire assay fusion of 30 g with an ICP finish. The resulting values were reported in parts per million.

Copper, nickel, and cobalt were assayed by four-acid "near total" digestion AAS. If any of the assays returned values above the detection limits, the sample was re-assayed using a similar method (ICP-AES or AAS).

11.5 Quality Assurance and Quality Control

QA/QC on Hudson, Kluane and Northern Platinum drilling programs is not documented but was believed to conform to industry standards at the time. This would have consisted solely of internal laboratory standards, blanks and duplicates.

In drilling and re-assaying programs carried out between 2006 and 2010 (by Coronation Minerals and Northern Platinum) blanks, Standard Reference Material (SRM), and duplicates were inserted into the sample stream approximately every 20th sample.

11.5.1 Standards

Eight standard reference materials (SRMs) have been used since 2006 to monitor laboratory performance. Six of these are site specific SRMs collected from the Property and were prepared by CANMET Mining and Mineral Sciences Laboratory in Ottawa as part of the Canadian Certified Reference Material Project (CCRMP). Two of the standards were purchased from Ore Research and Exploration Pty. Ltd. (OREAS) and were sourced from the West Musgrave region of Western Australia. All SRMs had certified values for Pt and Pd and most were certified for Au, Cu and Ni. Only two SRMs had certified values for Co. Where certified values were not present, provisional values were supplied. The SRMs and reference values are shown in Table 11.1.

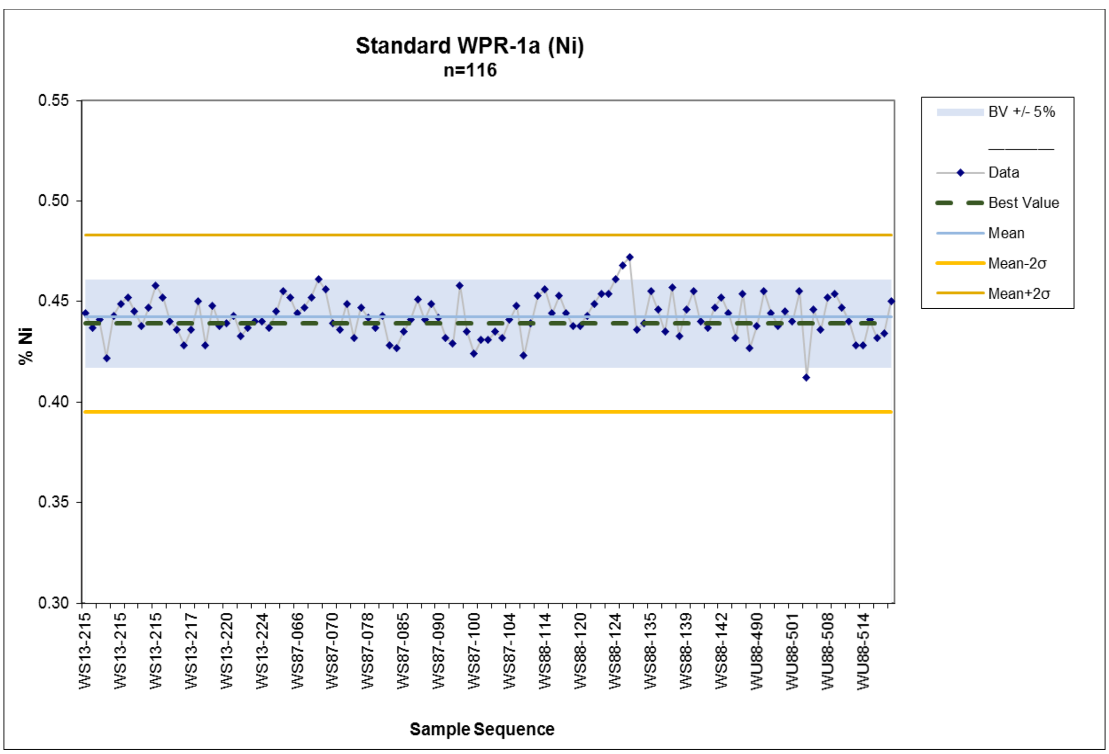
Table 11.1: Standard Reference Materials

SRM Code	Source	Programs	Au ppm	Pt ppm	Pd ppm	Cu %	Co %	Ni %
OREAS 13P	WA 2004	2006,2008	0.047	0.047	0.070	0.250	0.009	0.226
OREAS 14P	WA 2003	2006	0.051	0.099	0.150	0.997	0.075	2.090
WMG-1	Site 1994	2006-10	0.110	0.731	0.382	0.590	0.020	0.270
WPR-1	Site 1994	2006-12	0.042	0.285	0.235	0.164	0.018	0.290
WGB-1	Site 1997	2006-13	0.003	0.006	0.014	0.011	0.003	0.008
WMS-1a	Site 2007	2008-12 (88 re)	0.300	1.910	1.450	1.396	0.145	3.020
WMG-1a	Site 2011	2012 (87-88 re)	0.062	0.899	0.484	0.712	0.019	0.248
WPR-1a	Site 2012	2013 (88 re)	0.050	0.452	0.614	0.299	0.021	0.439
				= Provisional (not certified value)				

= Provisional (not certified value)
Source: GeoSim, 2015

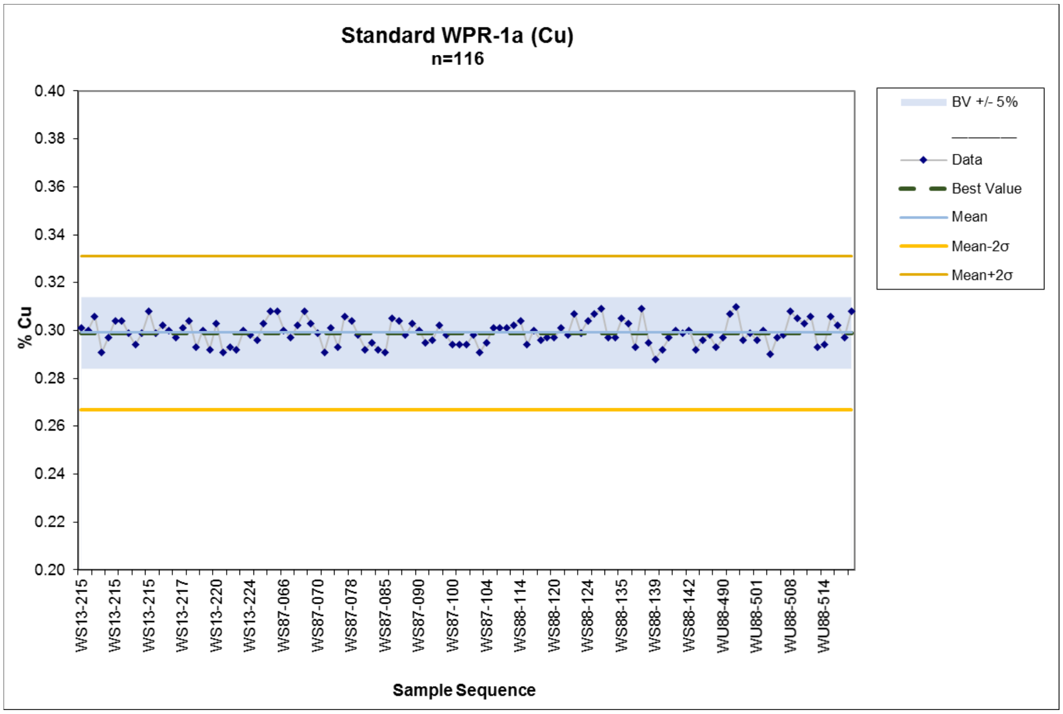
Standards performed within acceptable limits. Gold showed the most variability but this is not considered unusual at this low level of concentration. Examples of the control charts are presented in Figure 11.1 to Figure 11.6.

Figure 11.1: Standard Control Chart WPR-1a for Ni



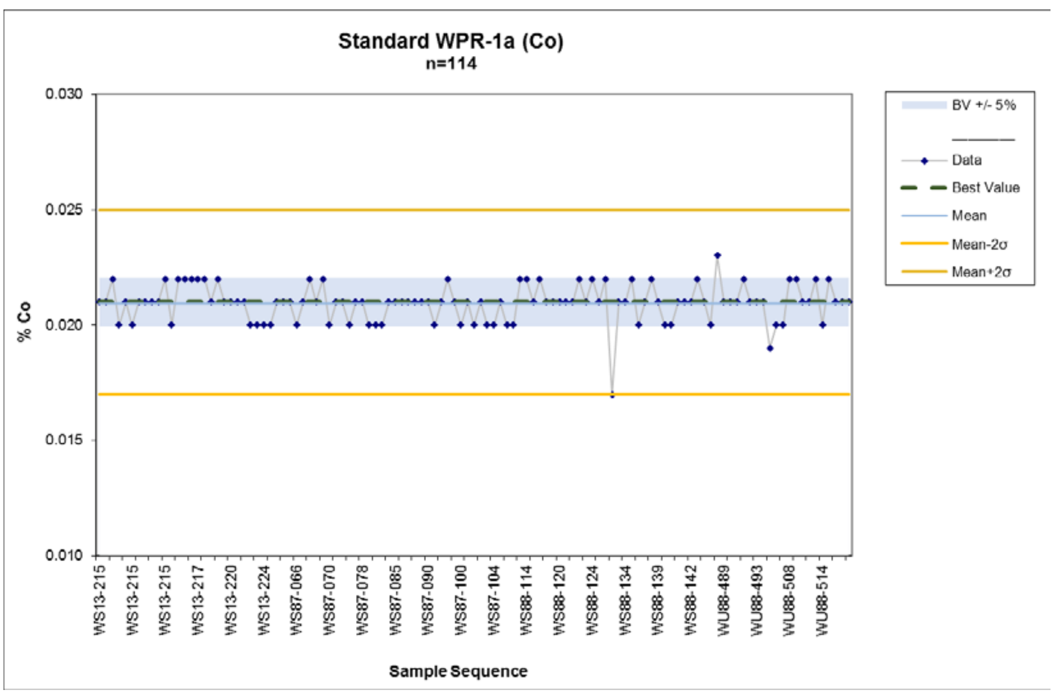
Source: GeoSim, 2015

Figure 11.2: Standard Control Chart WPR-1a for Cu



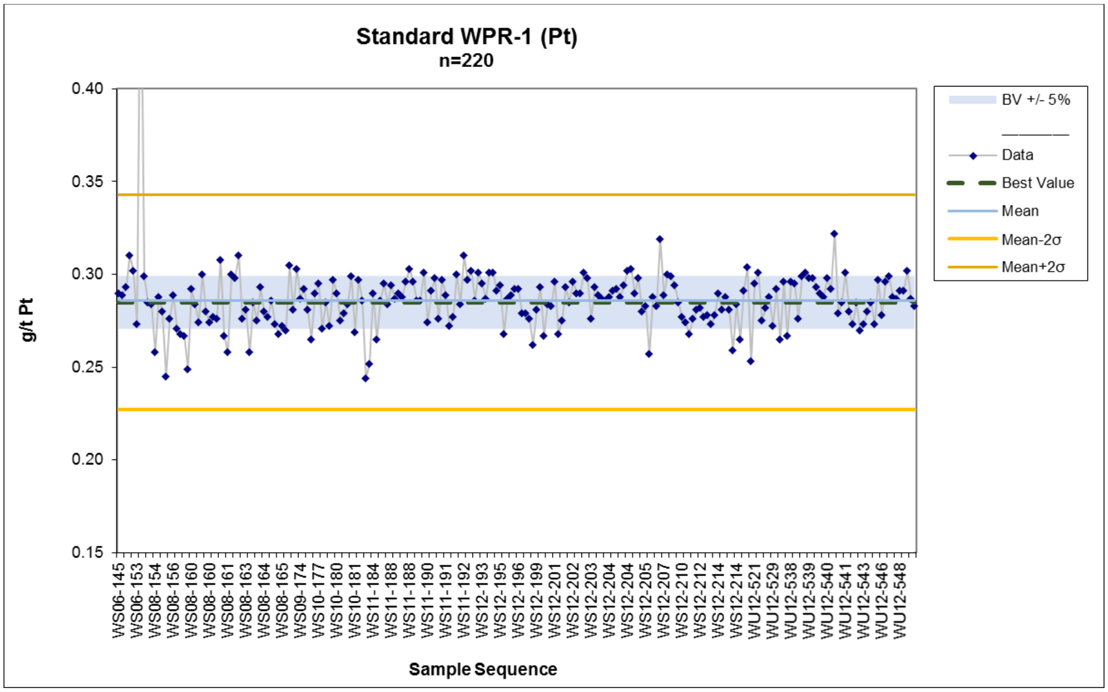
Source: GeoSim, 2015

Figure 11.3: Standard Control Chart WPR-1a for Co



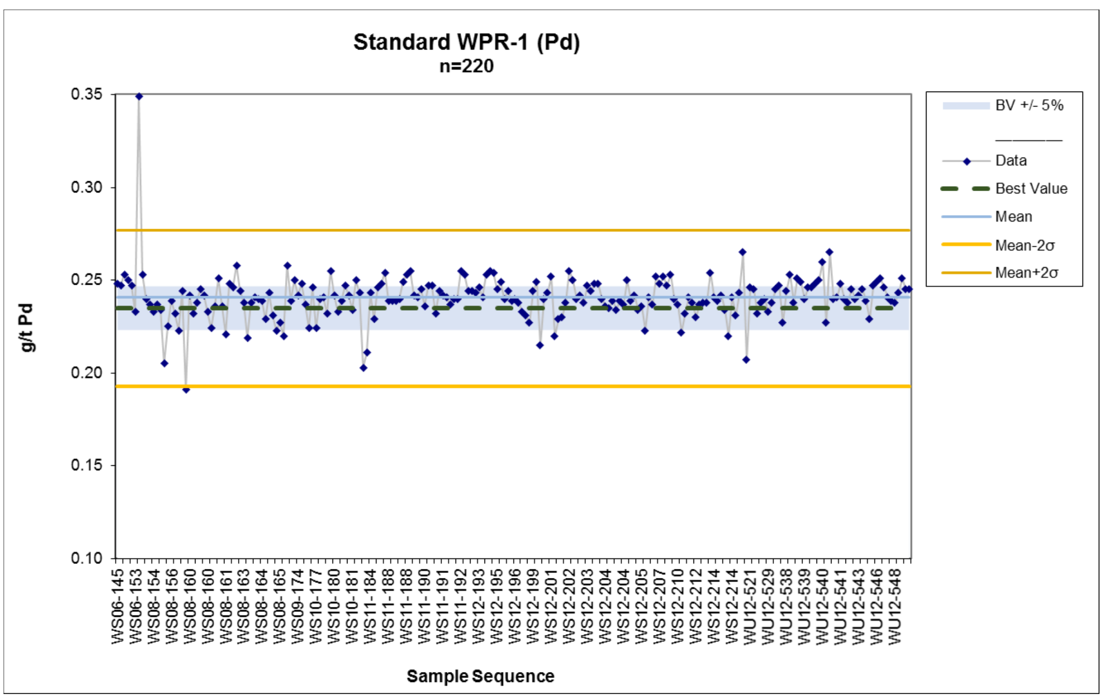
Source: GeoSim, 2015

Figure 11.4: Standard Control Chart WPR-1 for Pt



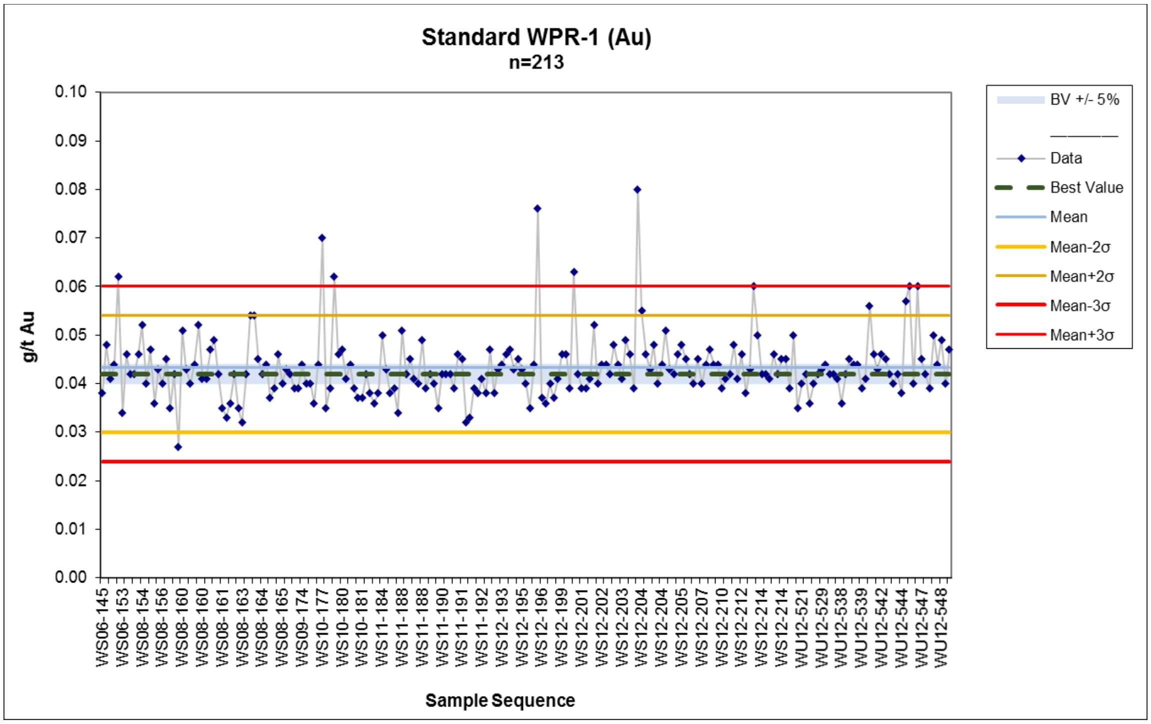
Source: GeoSim, 2015

Figure 11.5: Standard Control Chart WPR-1 for Pd



Source: GeoSim, 2015

Figure 11.6: Standard Control Chart WPR-1 for Au



Source: GeoSim, 2015

Standard WGB-1 is described as 'Gabbro Rock PGE Reference Material' but due to very low levels of base and precious metals it would be more suitable as a blank. It is recommended that the use of this SRM be discontinued.

11.5.2 Blanks

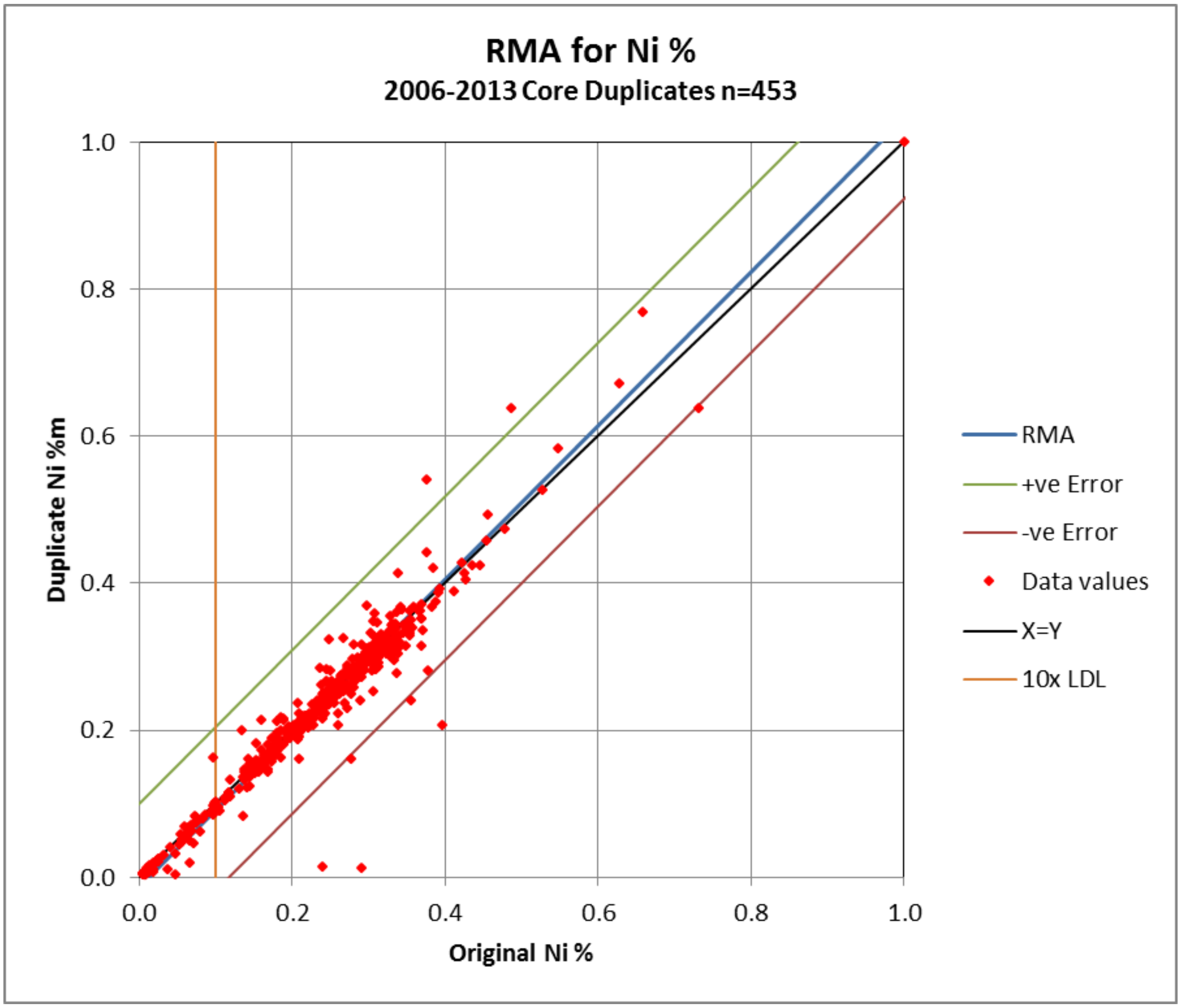
Blank samples were used to check for contamination during sample preparation. The material was obtained from two sources: granodiorite from a nearby road quarry, and garden marble from hardware stores in Whitehorse, Yukon. A blank sample was normally inserted into the sample stream after the SRM or immediately following a massive sulphide interval. A total of 731 blanks were inserted in the sampling process and analyzed between 2006 and 2013. Blank failures were checked to ensure that they did not appear immediately after higher grade samples. No significant contamination was indicated.

11.5.3 Duplicates

A quarter core duplicate sample was taken approximately every 20th sample up to August, 2012 for a total of 625. Since that time, 81 coarse rejects have been used as duplicate checks. Pulp duplicates were also available from the 1987-88 re-sampling program and the 2013 program. A total of 130 pulp duplicates for Ni and Cu returned above detection values.

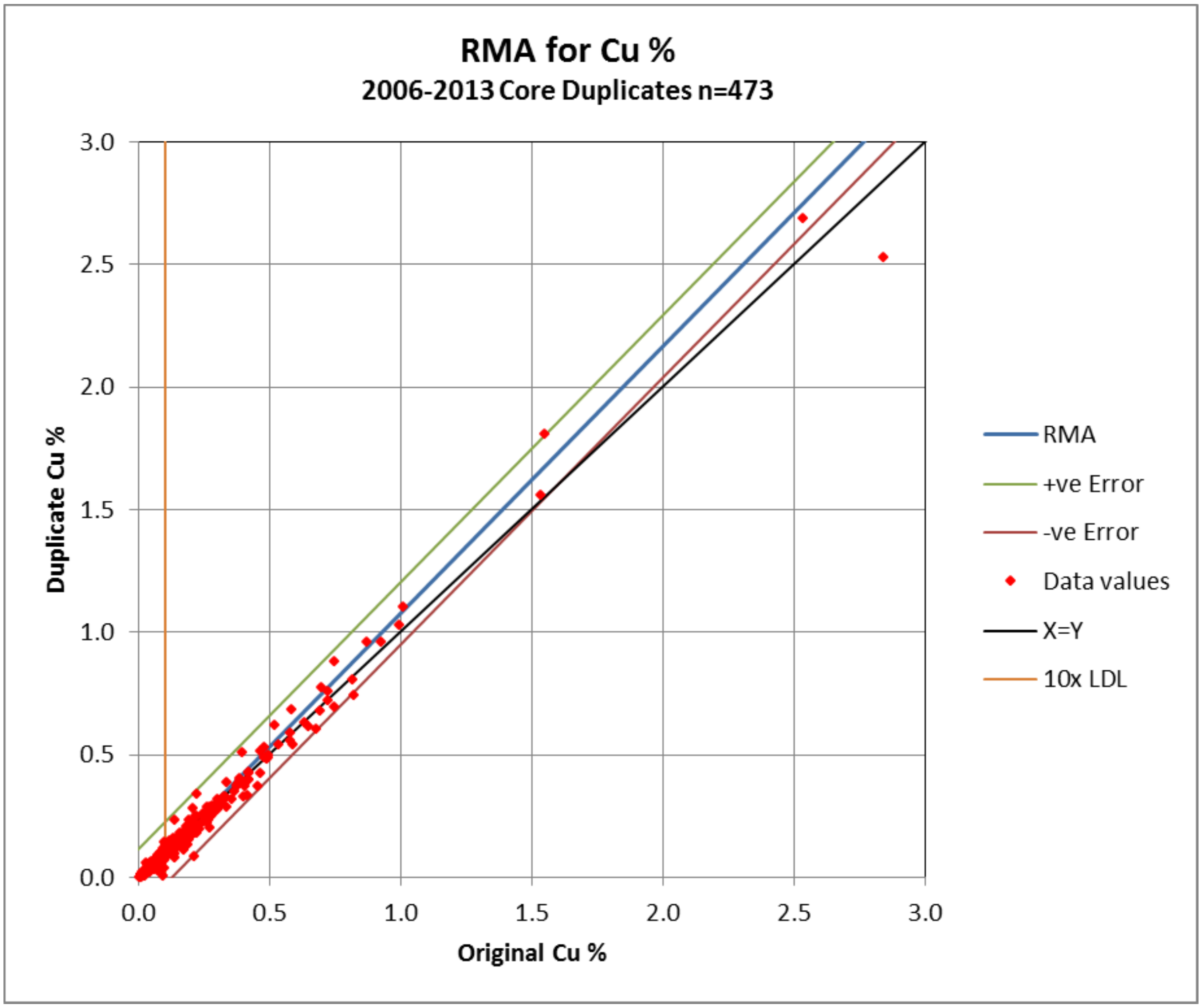
Scatter plots for the quarter core duplicates with reduced major axis (RMA) are shown in Figure 11.7 to Figure 11.12. Statistics are shown in Table 11.2. The slopes of the RMA lines show no significant bias with less than 1% for Ni, Cu, Co, and Pd and less than 2% for Pt and Au.

Figure 11.7: RMA Plot Quarter Core for Ni



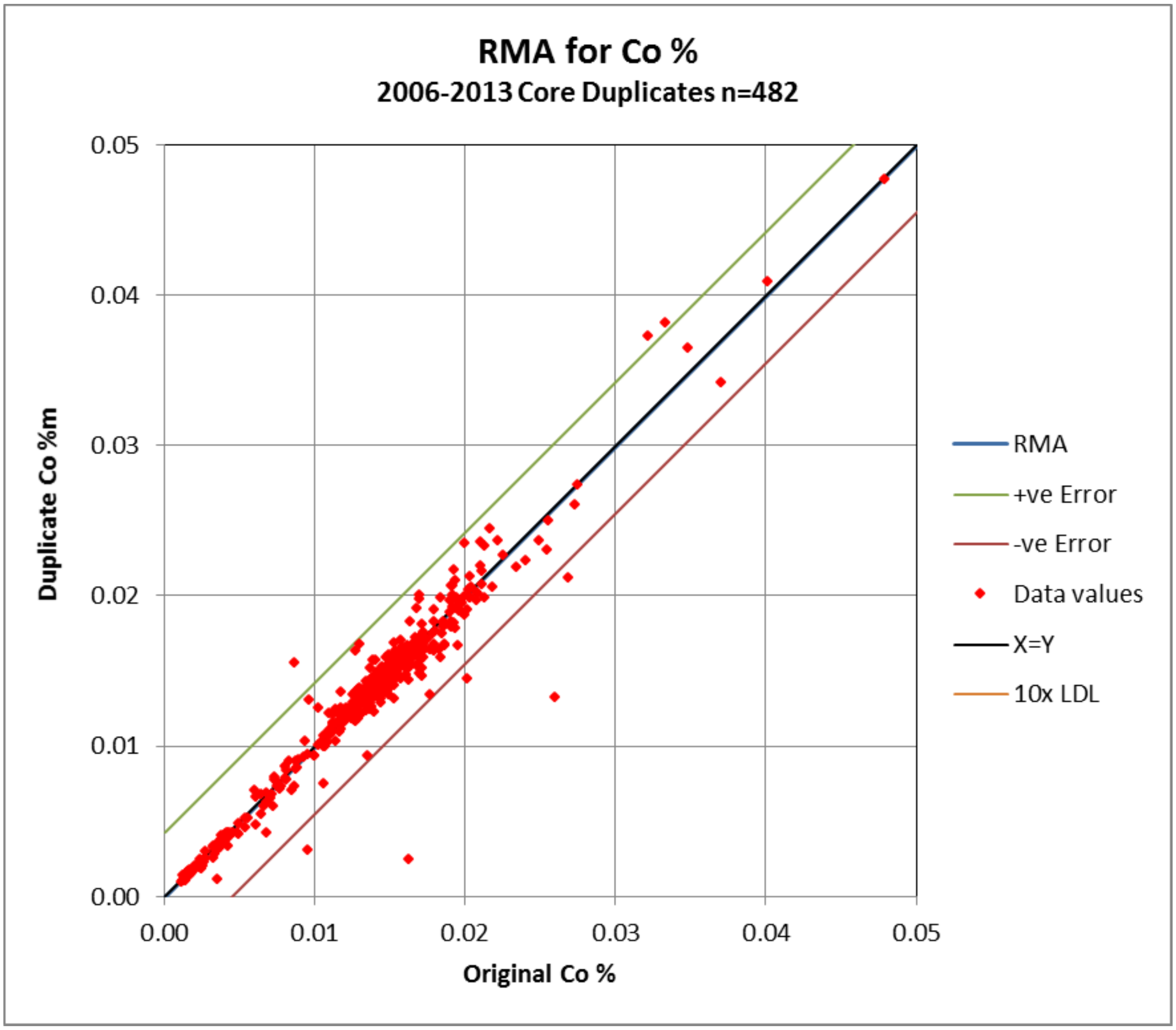
Source: GeoSim, 2015

Figure 11.8: RMA Plot Quarter Core for Cu



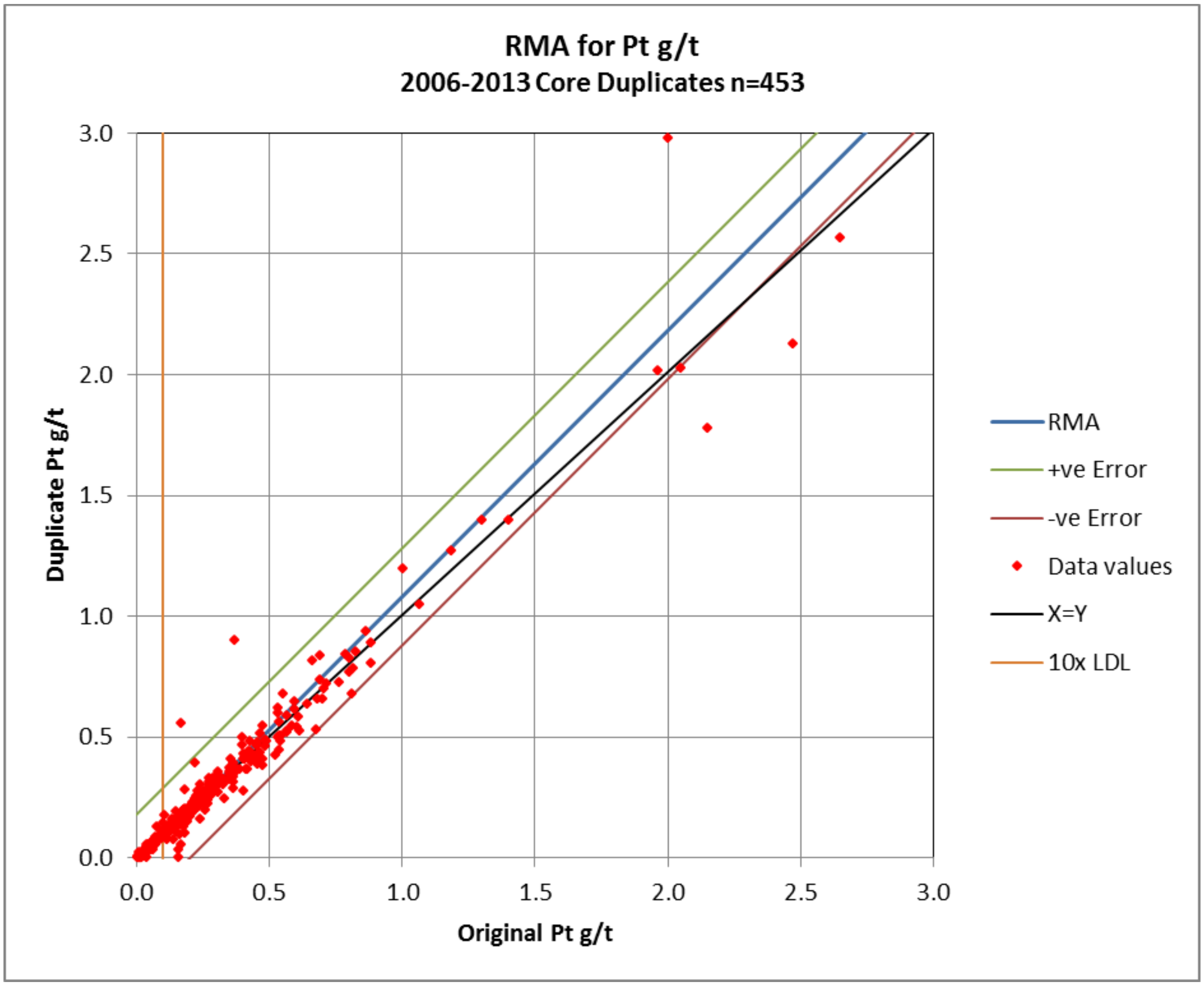
Source: GeoSim, 2015

Figure 11.9: RMA Plot Quarter Core for Co



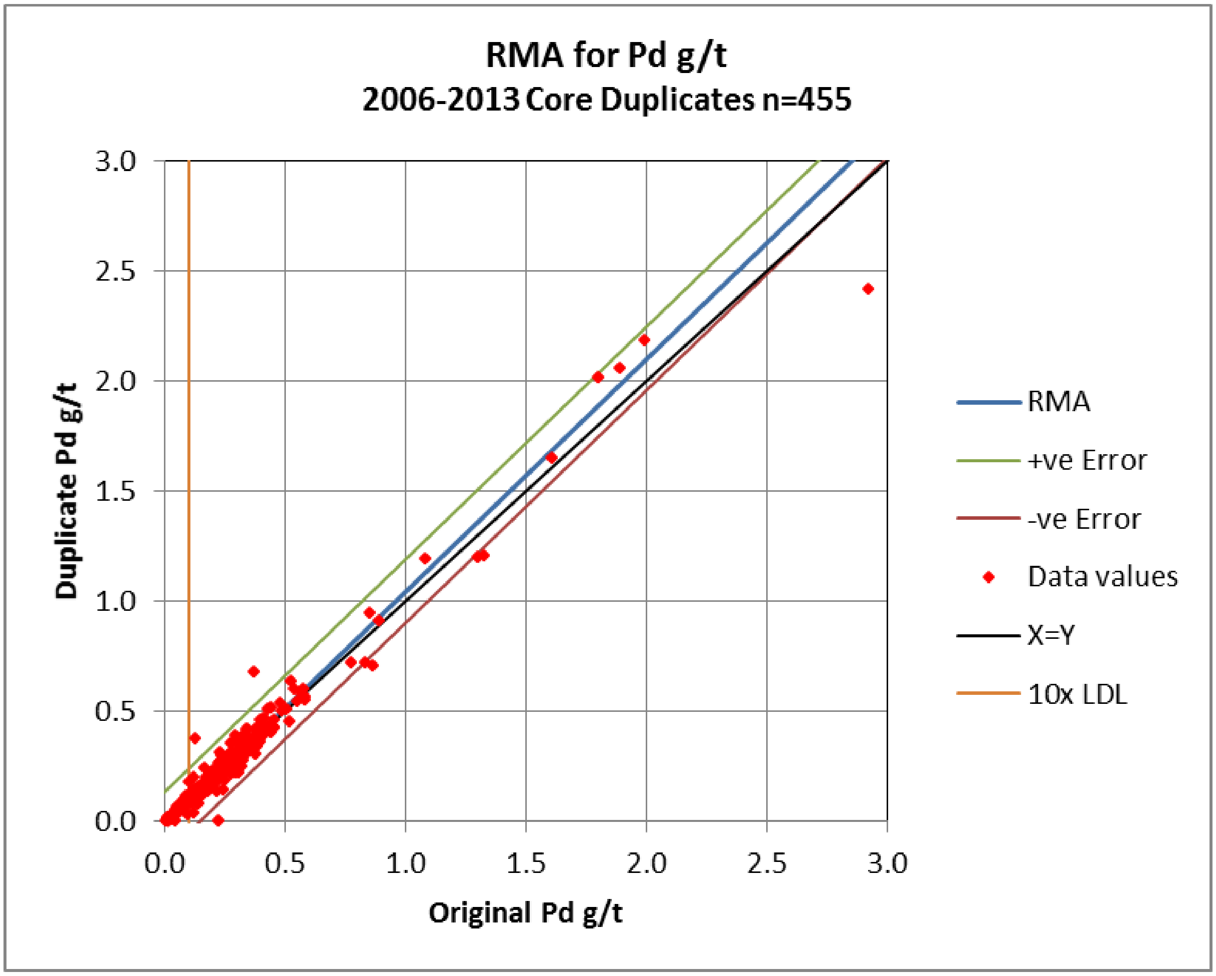
Source: GeoSim, 2015

Figure 11.10: RMA Plot Quarter Core for Pt



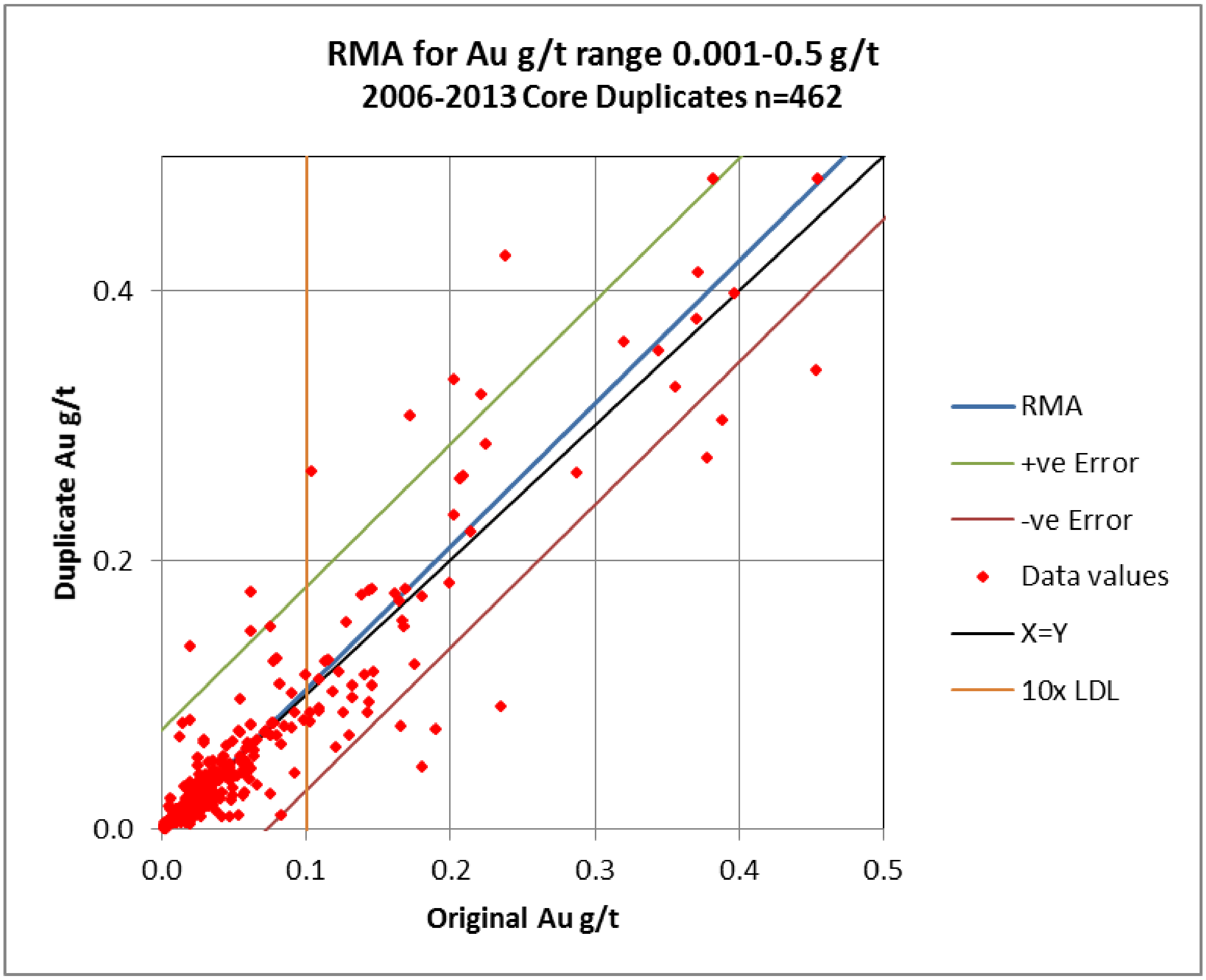
Source: GeoSim, 2015

Figure 11.11: RMA Plot Quarter Core for Pd



Source: GeoSim, 2015

Figure 11.12: RMA Plot Quarter Core for Au



Source: GeoSim, 2015

Table 11.2: Quarter Core Duplicate Performance from RMA plots

Element	Count	Mean Orig	Mean check	Bias	CV_{AVR} %	Int	Slope	S_{RMA}	95% CI	R^2
Ni	453	0.261	0.260	0.22%	14.22	-0.012	1.046	0.057	0.111	0.981
Cu	473	0.171	0.172	-0.81%	13.83	-0.014	1.090	0.065	0.127	0.976
Co	482	0.015	0.015	0.79%	8.96	0.000	0.998	0.002	0.004	0.984
Pt	482	0.277	0.282	-1.65%	18.60	-0.022	1.096	0.102	0.200	0.982
Pd	455	0.247	0.249	-0.81%	15.47	-0.012	1.056	0.074	0.145	0.970
Au *	455	0.047	0.048	-1.72%	26.89	-0.002	1.061	0.038	0.075	0.865

Source: GeoSim, 2015
* Au range from 0.001 – 0.50 g/t

The coefficient of variation CV_{AVR}(%) is a common standard by which to assess the performance of duplicates in geochemical datasets with n>500 (Stanley and Lawie, 2007).

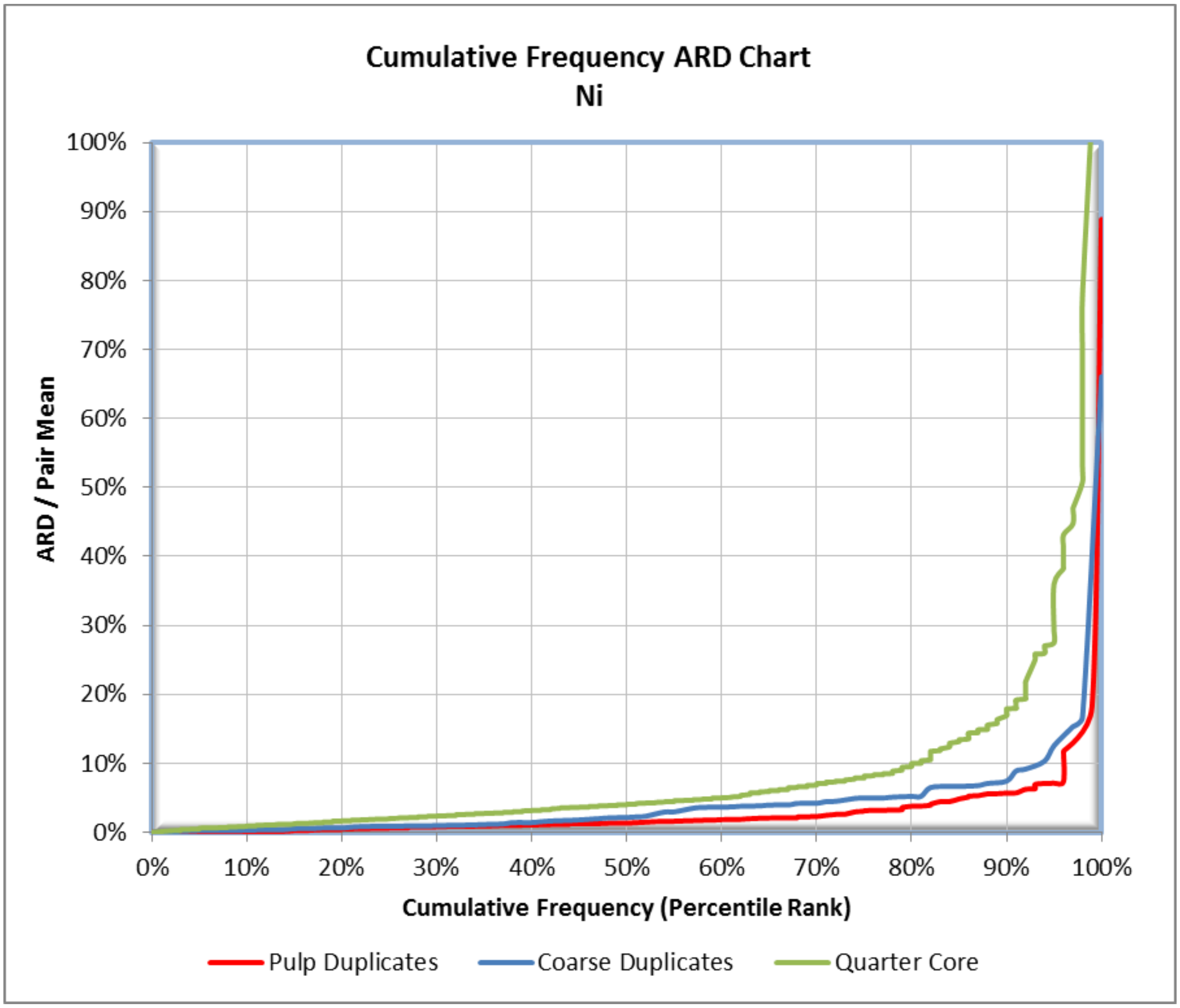
The calculation for CV_{AVR} (%) is:

$$CV_{AVR}(\%) = 100 \times \sqrt{\frac{2}{N}\sum_{i=1}^{N}\left(\frac{(a_i - b_i)^2}{(a_i + b_i)^2}\right)}$$

Only the quarter core data fits the large population criteria. For field duplicates the acceptable CV_{AVR} limit is 30%. The values for Ni, Cu, and Co were less than half of this level. Pt and Pd showed acceptable performance at 18.6 and 15.5% respectively. Au approached the limit at 27.07% indicating more variability attributed to the large number of assays near the detection limit.

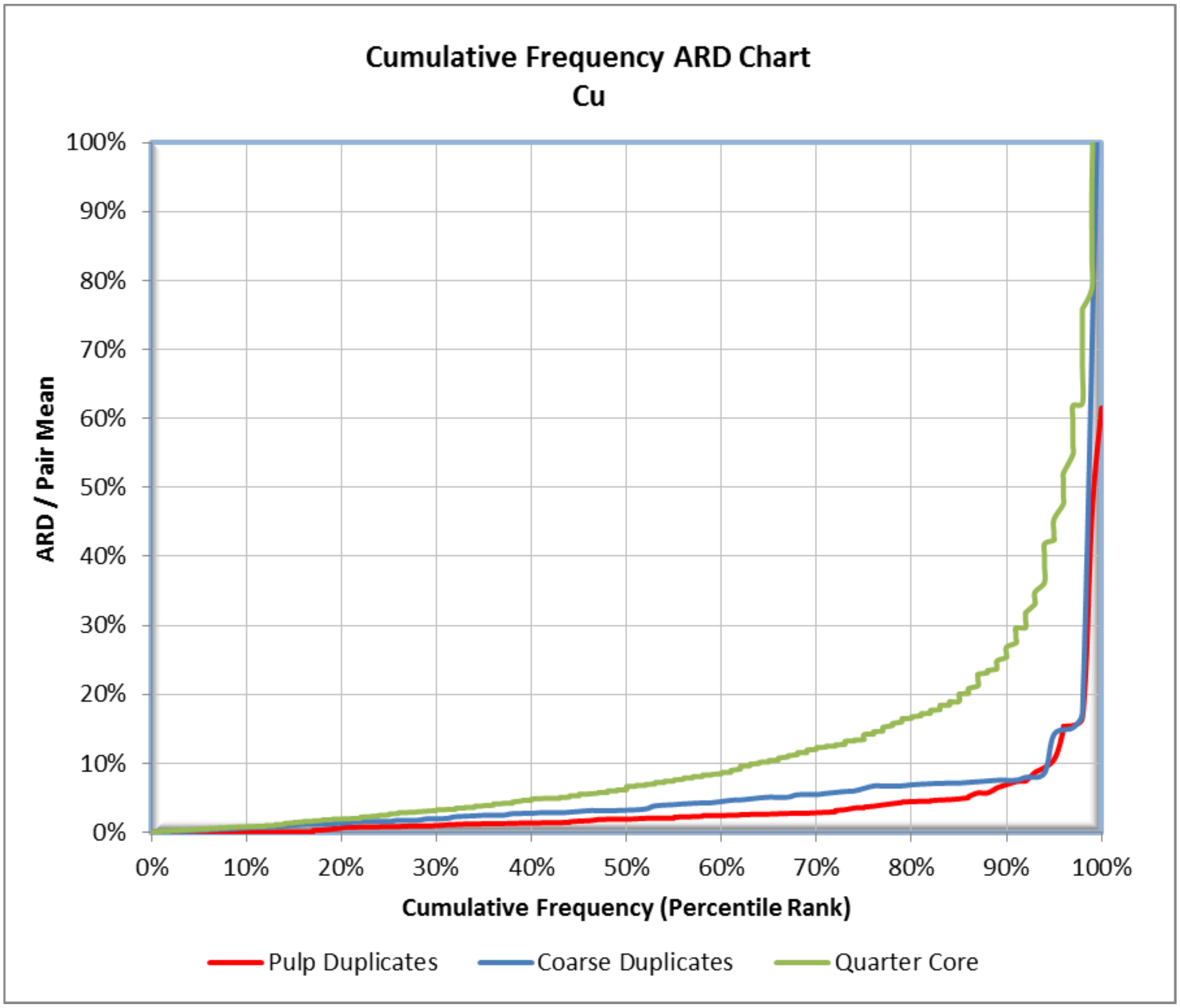
Absolute relative difference (ARD) charts were also generated to compare the duplicate results for the various elements. Generally recommended thresholds are less than 10% ARD at the 90% cumulative frequency limit for pulps, less than 20% for coarse rejects and less than 30% for core or field duplicates. Ni, Cu, and Co are all within these thresholds as displayed in Figure 11.13 to Figure 11.15. Pd at 15% ARD for pulps at the 20% cumulative frequency threshold is marginally high while Pt shows more variability with an ARD around 25% at this level (Figure 11.16 and Figure 11.17). Results for Au show the highest variability due to the large number of assays close to detection limit (Figure 11.18). The coarse reject results are often close to the pulp results and are likely due to significantly fewer samples in the populations plotted and variability at higher grade levels.

Figure 11.13: ARD Chart for Ni



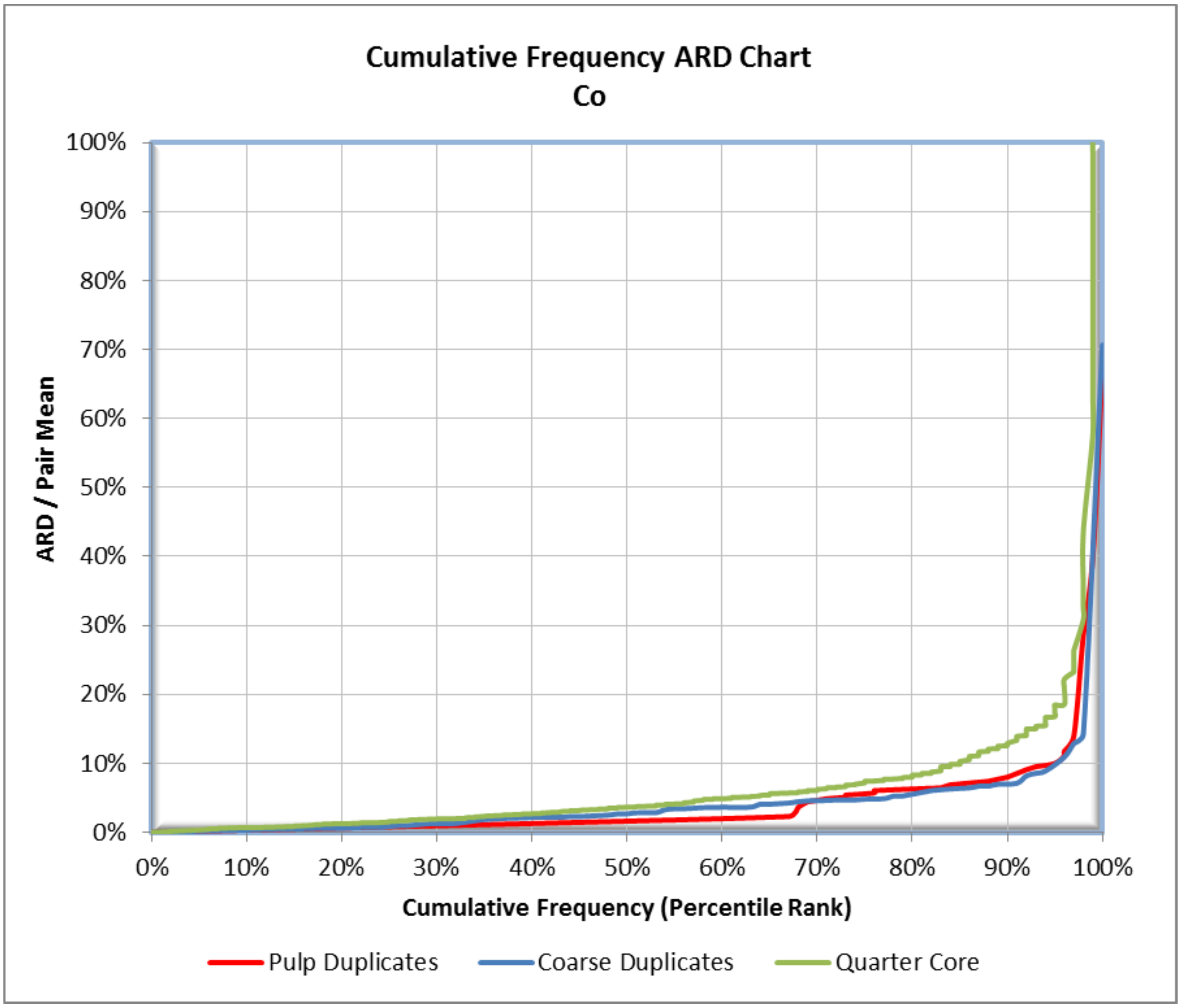
Source: GeoSim, 2015

Figure 11.14: ARD Chart for Cu



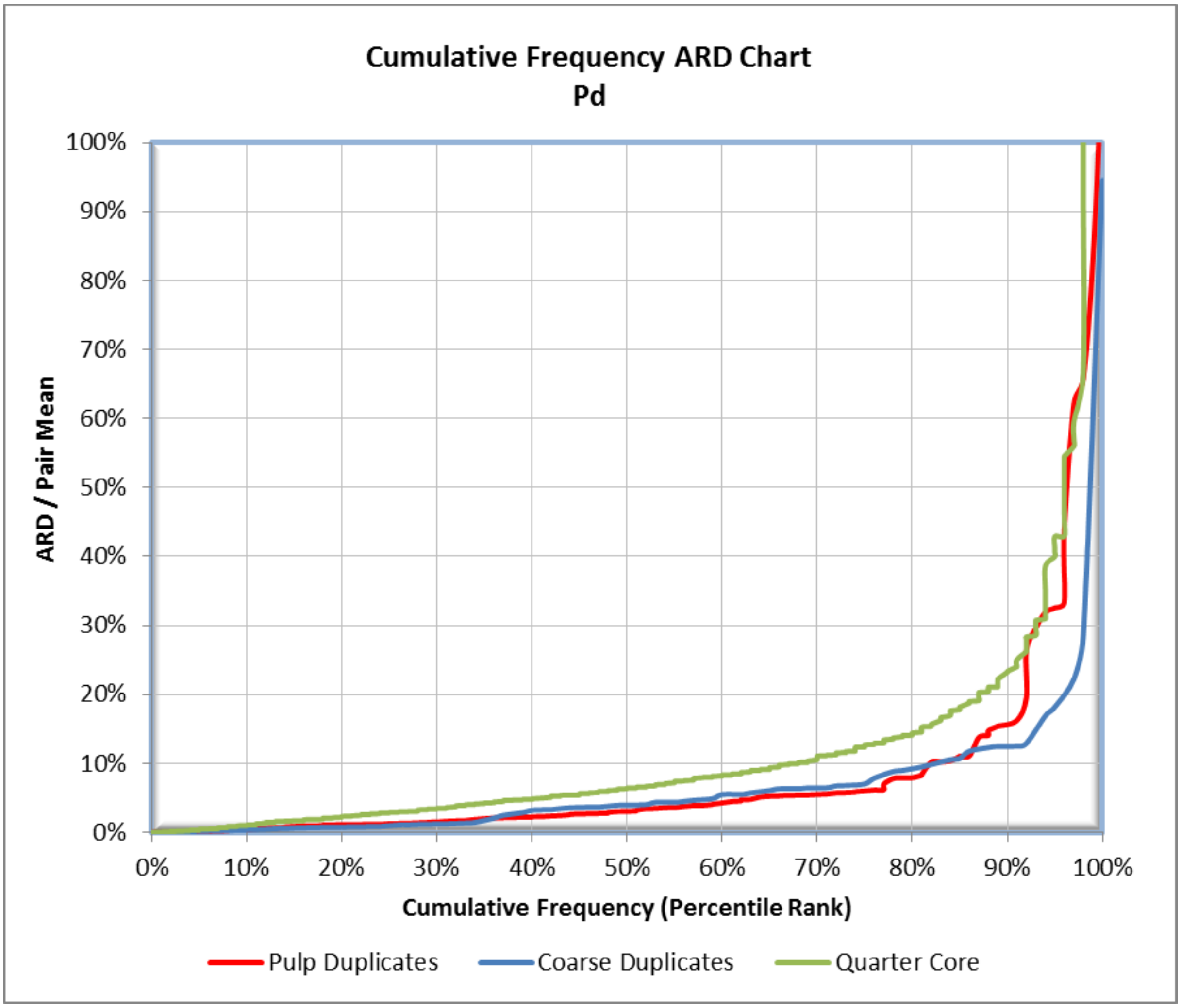
Source: GeoSim, 2015

Figure 11.15: ARD Chart for Co



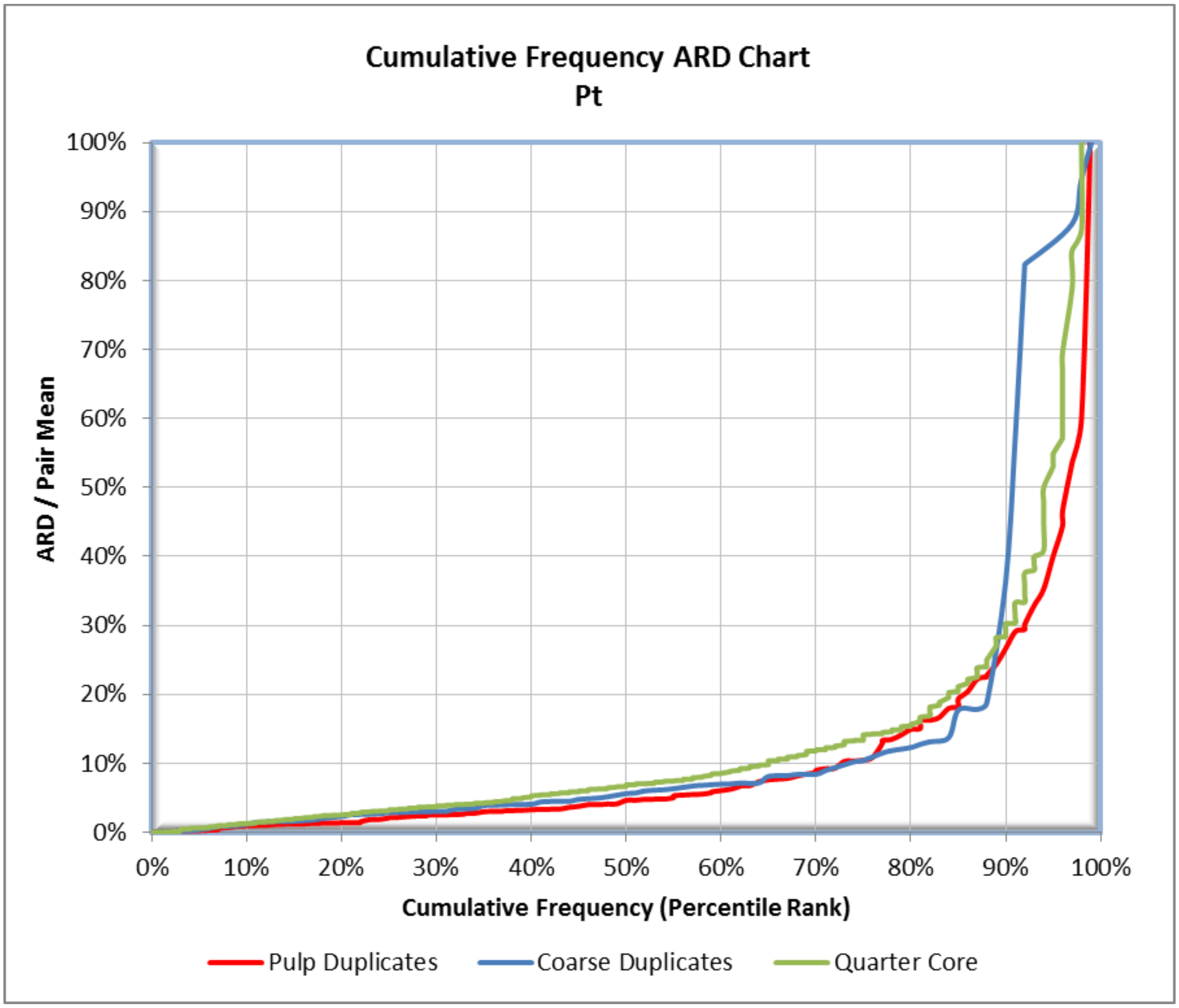
Source: GeoSim, 2015

Figure 11.16: ARD Chart for Pd



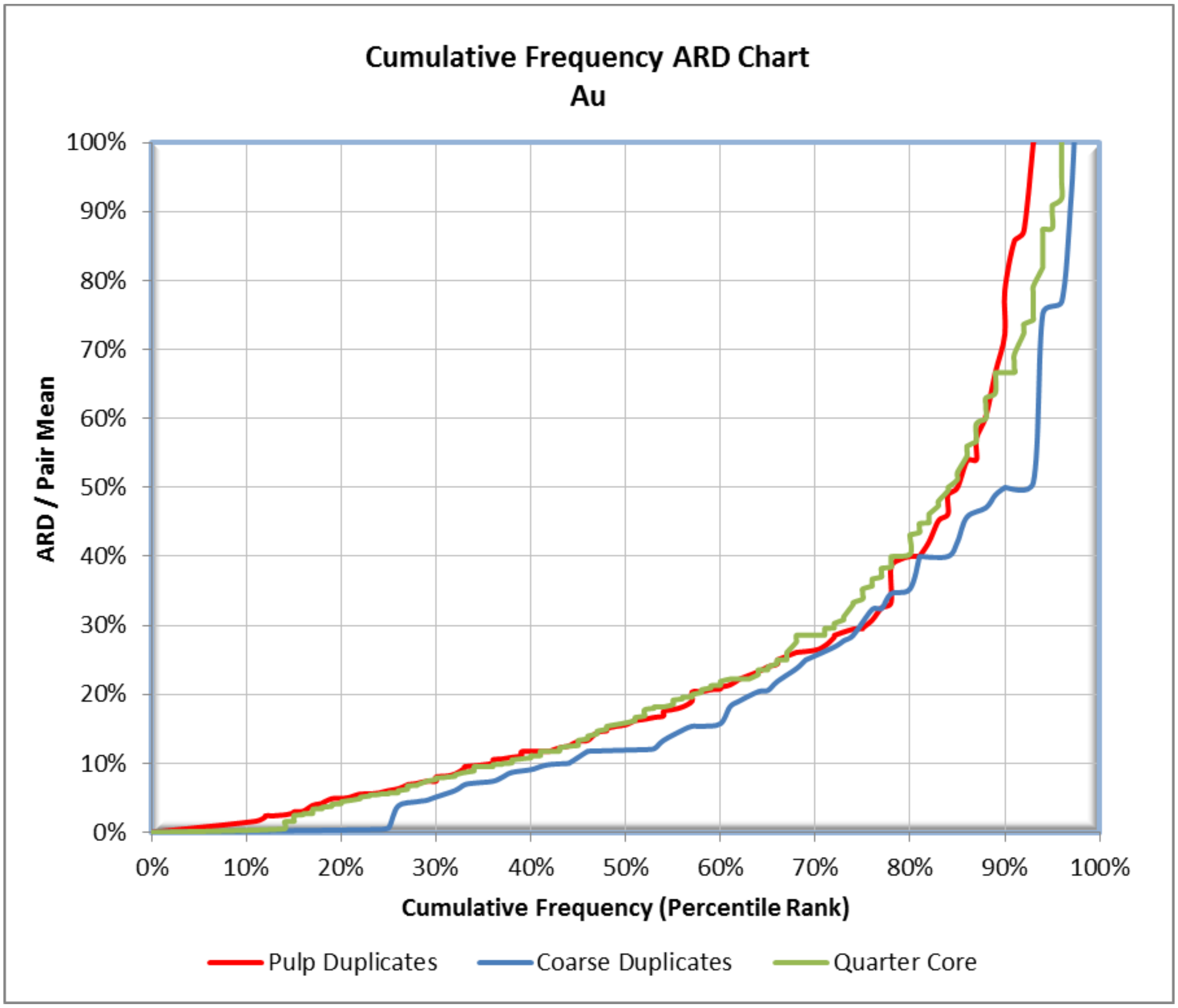
Source: GeoSim, 2015

Figure 11.17: ARD Chart for Pt



Source: GeoSim, 2015

Figure 11.18: ARD Chart for Au



Source: GeoSim, 2015

11.6 Databases

A centralized MS-Access database is maintained in the Wellgreen Platinum corporate office.

11.7 Sample Security

The security measures for drill programs carried out prior to 2009 are undocumented but are believed to have conformed to industry best practices at the time. In 2009-2010, after the sample bags were sealed, company personnel would transport them to the Northern Platinum geological office. The samples were stored there and only the geologist and camp manager had access. When enough samples had accumulated, company personnel would pack them in plastic containers, label them, and take the containers to the shipper (Air North) in Whitehorse. Since 2011, the rice bags full of samples were temporarily stored in the core shack located in the lower camp and shipped approximately once per week to Whitehorse.

11.8 Opinion on Adequacy

GeoSim is of the opinion that the adequacy of sample preparation, security and analytical procedures are sufficiently reliable to support the mineral resource estimation and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices at the time of collection.

12 Data Verification

12.1 Site Visit Validation

Ronald G. Simpson of GeoSim visited the site on September 17, 2013. The purpose of the visit was to review the drilling, sampling, and QA/QC procedures. The geology and mineralization encountered in the drill holes completed to date were also reviewed. During the site visit Mr. Simpson verified:

- Collar locations are reasonably accurate by comparing several drill hole database collar locations with hand-held GPS readings;
- Drill hole collars are clearly marked with sturdy wooden fence posts, and the drill hole identity, orientation, and depth are inscribed onto a metal tag or a concrete slab (Figure 12.1);
- Down-holes surveys for surface holes are routinely taken at 15 to 25 m intervals using a Reflex single-shot unit;
- Drill logs compare well with observed core intervals;
- Core recoveries were generally high through the mineralized zones; and
- Specific gravity is determined using a water immersion method where the weight of the sample in air and in water is measured with an electronic scale.

Mr. Simpson did not collect independent samples as the property had a record of metal production. Sulphide mineralization observed in drill core was consistent with reported base metal grades.

Figure 12.1: Drill Hole Collar Markers

Source: GeoSim, 2015

12.2 Database Verification

Drill data are typically verified prior to mineral resource estimation by comparing data in the Property database to data in original sources. For most of the data, the original sources are electronic data files; therefore, the majority of the comparisons were performed using software tools.

Un-sampled intervals were identified and entered into the database and assay fields flagged with '-1' to identify them as missing.

GeoSim examined the sample database for location accuracy, down hole survey errors, typographical errors, interval errors and missing sample intervals. Several issues were identified and corrected prior to mineral resource estimation.

12.3 Data Adequacy

Based on the site visit observations, GeoSim concludes that drilling, logging, and sampling of drill core during the exploration programs carried out by Wellgreen Platinum and previous operators have been conducted in a manner appropriate to the style of mineralization present on the property.

The process of data verification performed by GeoSim indicates that the data collected by Wellgreen Platinum and previous operators from the Property adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits, and adequately support the geological interpretations for the purpose of this PEA. GeoSim is of the opinion that the analytical and database quality are adequate for the purposes of this PEA.

QA/QC with respect to the results received to date for the 2006 through 2013 exploration programs and re-assaying of core from the 1987/88 programs is acceptable, and protocols have been reasonably well documented.

Legacy data collected prior to 2006, with the exception of re-assayed core from 1987-88, is not considered to be sufficiently reliable on its own to support a measured or indicated mineral resource classification.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

The recoveries of metals to concentrate and concentrate-grade assumptions used in this PEA are based on a combination of metallurgical testing programs conducted between 1988 and 2014. SGS and XPS conducted the laboratory scale testing programs in 2013 and 2014. These programs were supervised by John Eggert, P. Eng., of Eggert, and reviewed by Dr. David Dreisinger. John Eggert is Wellgreen Platinum's independent Qualified Person as defined by NI 43-101 regarding metallurgical performance and mineral processing.

The test programs evaluated the effects of the following:

- Grind size, pH, and conditioning time;
- The use of various collectors, flotation reagents, dispersants and depressants on mineral recoveries and concentrate grades;
- Magnetic separation; and
- Modifications to the mineral processing flowsheet.

In mid-2014, XPS reviewed the historical metallurgical test reports (1988–2014) with Wellgreen Platinum and John Eggert, and concluded the following:

- A bulk concentrate is the optimal approach for the updated PEA; SGS results from 2013/2014 showed that following an initial copper flotation step, sequential flotation resulted in poor nickel and PGM recoveries; and
- A magnetic separation of the bulk float tail followed by a regrind/flotation cycle improves nickel and PGM recoveries.

The historical review also concluded that there were six geological domains that form three metallurgical domains. The three geo-metallurgical domains are as follows:

- Gabbro/Massive Sulphide, highest sulphur content and grade with lowest serpentine content;
- Clinopyroxenite/Pyroxenite, moderate sulphur content and grade with moderate serpentine content; and
- Peridotite/Dunite, lowest sulphur content and grade with moderate to high serpentine content.

One of the key observations from the XPS review was that the optimization of sulphide flotation recovery varied with the metallurgical domains. In general, the recovery of economic metals was highest within the Gabbro/Massive Sulphide metallurgical domain, followed by the Clinopyroxenite/Pyroxenite metallurgical domain, and the recovery of economic metals was lowest within the Peridotite/Dunite metallurgi domain. As a result of this observation, Wellgreen Platinum's geology group developed a classification system for these rock types, and conducted considerable re-logging of historic core so that the resource model could include this information.

The historical metallurgical testing programs also indicated that most of the tests were conducted on material that would be considered part of the Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite domains, and minimal testing and flowsheet optimization work had been done on the Peridotite/Dunite domain. Therefore, XPS was hired to conduct a test program that focused on the Peridotite/Dunite samples to better understand recoveries using a bulk concentrate approach and a magnetic separation step. Recommendations included improving the domain classification system in the field and improving the flowsheet optimization for material containing a higher proportion of peridotite.

Testing has shown that material taken from each of the three metallurgical domains can be processed in the same circuit with variances related to grind size, conditioning time, and pH, and the use of magnetic separation; with the majority of reagent selection applied across all the domains. However, given the unique metallurgical performance of each geological domain, the mine plan was designed to process the higher grade material (consisting of about 99% from the Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite geological domains) during the first 16 years of operation, and then to stockpile the lower grade material (consisting of about 24% of material from the Peridotite/Dunite domain) until it is processed after mining is completed in Year 17. Peridotite material represents approximately 25% of the material in the Stage 5 Opportunity pit.

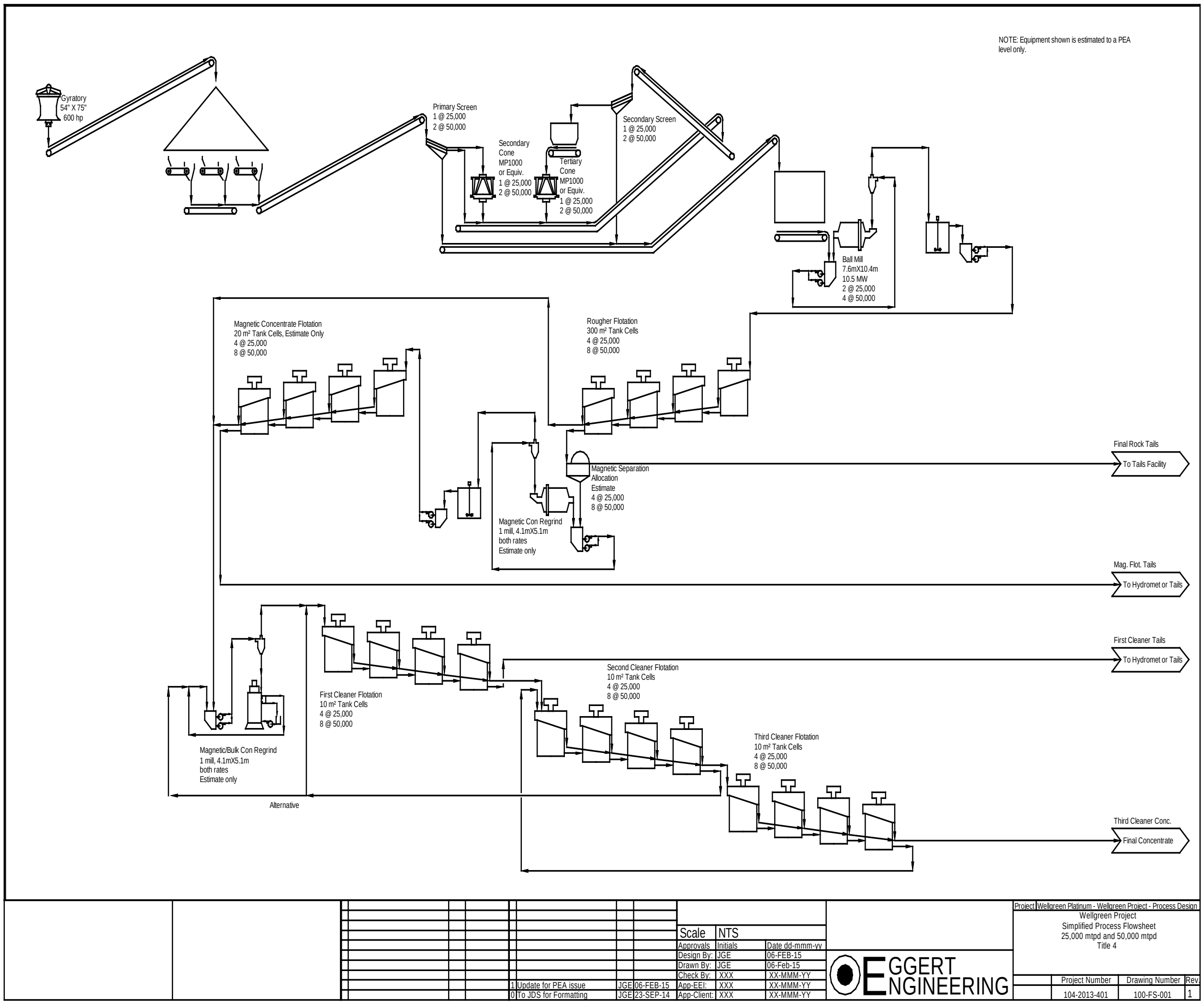
An analysis of the concentrate tails from previous metallurgical testing programs indicated that a significant amount of PGMs, particularly platinum, was lost in the sulphide flotation process because, as a group, the PGMs are finer-grained and associated with the magnetic minerals, magnetite and pyrrhotite. As a result, tests were conducted to evaluate the benefit of adding a magnetic separation process to the flowsheet; this is a proven technology used in many operating nickel-PGM mines. The magnetic separation process proved successful and captured additional PGMs, and nickel and copper by regrinding a small volume of magnetic material followed by conventional flotation. This was particularly evident in the Clinopyroxenite/Pyroxenite and Peridotite/Dunite domains. This material could then be combined with the main sulphide concentrate to improve overall primary flotation recoveries or used to generate a separate PGM concentrate.

Preliminary testing of various leaching methods indicated that a PGM concentrate or tails from the magnetic flotation and cleaner tails might be amenable to additional secondary processing, and potentially increase overall PGM recovery by 20% to 30%. Additional metallurgical testing will be required to further evaluate these secondary processing options.

Current testing and historic mining at the Property has shown the presence of significant, exotic PGMs, such as rhodium, iridium, osmium and ruthenium. Testing has also shown that these exotic PGMs could increase the total PGM content by 10% to 25% over platinum and palladium. Additional work is required to determine whether these exotic PGMs should be included in the current mineral resource estimate or future economic assessments.

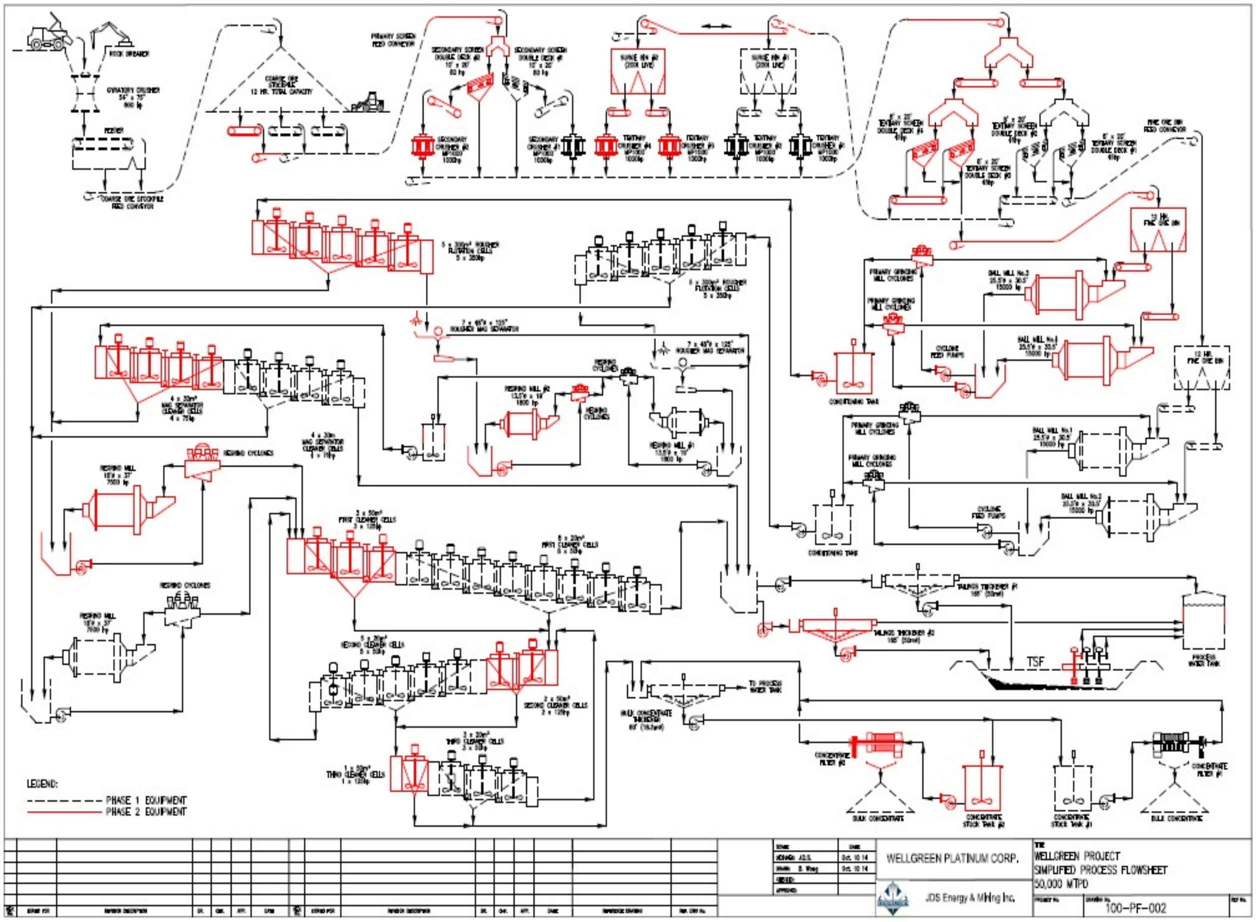
Detailed flowsheets for 25,000 t/d and 50,000 t/d throughput scenarios are shown in Figure 13.1 and Figure 13.2. These figures summarize the best understanding to date of the processes required to achieve optimum concentrate grade and recovery to concentrate.

Figure 13.1: Wellgreen Project Flowsheet Throughput (25,000 t/d)



Source: Eggert, 2015

Figure 13.2: Wellgreen Project Flowsheet Throughput (50,000 t/d)

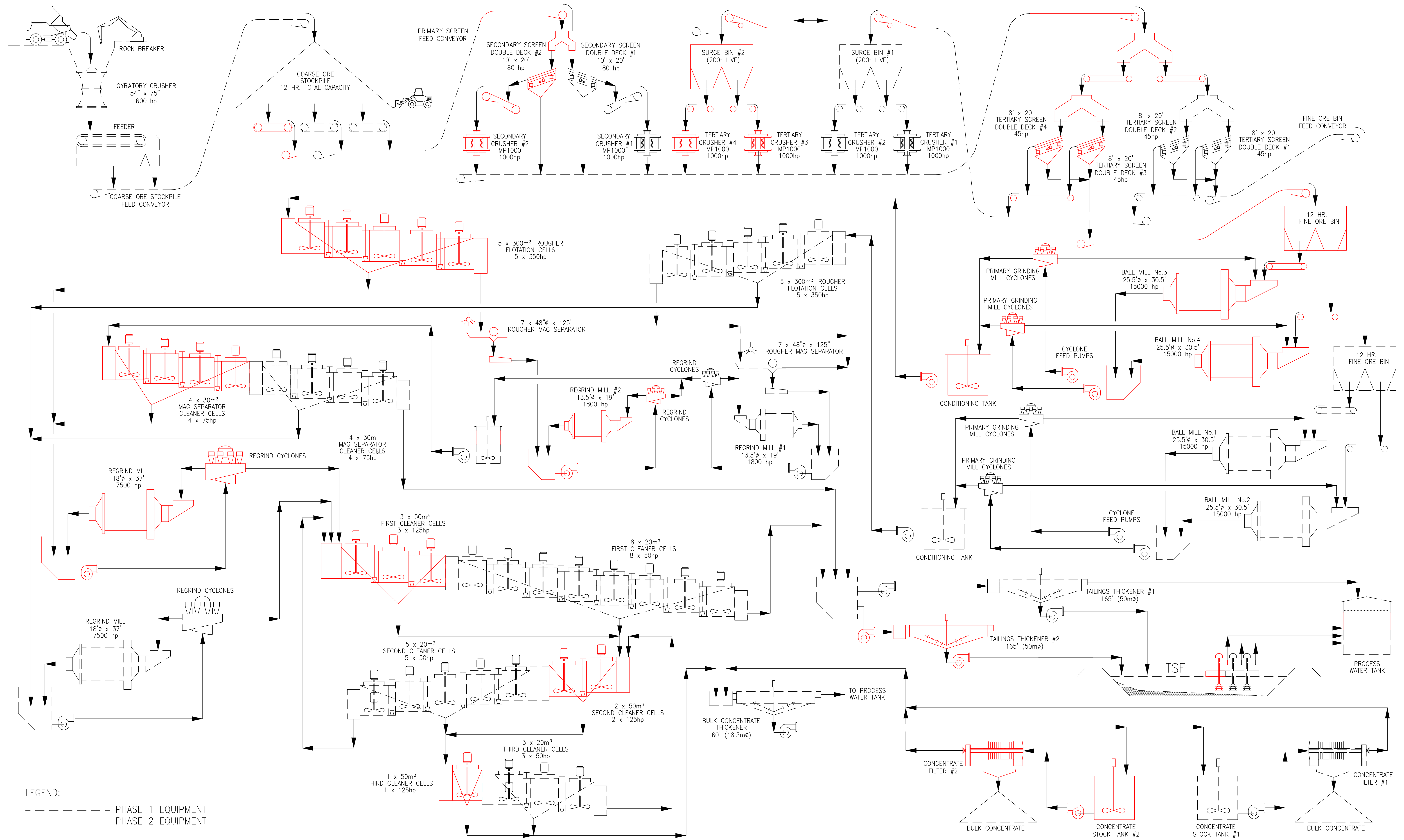


Source: JDS, 2015

Wellgreen Project Flowsheet Throughput (50,000 t/d) Preview Only – See next page for larger size rendition.

Recovery-grade curves for each geo-metallurgical domain were developed for platinum, palladium, gold, nickel, copper and cobalt using data from 183 batch tests and 12 locked cycle tests (LCTs) on 26 representative samples. The recovery-grade curves used linear regression to generate an equation to calculate recovery to bulk concentrate by metal for each metallurgical domain based on a normalized nickel grade. Analysis of the test results indicated that recoveries were typically higher in LCTs than in batch tests, so adjustments were made to the linear regression equations to adjust batch test results upwards to reflect recoveries that are expected in future LCTs and pilot-plant testing.

Table 13.1 shows the recovery to bulk concentrate by geological domain for a bulk concentrate grading 6% Ni. Based on this, the concentrates produced through conventional sulphide flotation are anticipated to grade 6-10% Ni, with 4-8% Cu and 11-14 g/t combined platinum, palladium and gold. Table 13.2 provides the 2015 PEA mill feed by geo-metallurgical domain and Table 13.3 provides the resulting concentrate grades and metal recoveries for the 2015 PEA.



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	REFERENCE DRAWING	REF. DWG No.

SCALE:		DATE:
DESIGNED:	J.D.S.	Oct. 10 14
DRAWN:	B. Wong	Oct. 10 14
CHECKED:		
APPROVED:		

WELLGREEN PLATINUM CORP.

JDS Energy & Mining Inc.

TITLE
WELLGREEN PROJECT
SIMPLIFIED PROCESS FLOWSHEET
50,000 MTPD

PROJECT No. | DRAWING No. 100-PF-002 | REV No.

Table 13.1: Estimated Metal Recoveries by Geological Domain

Geo-Metallurgical Domain	Recovery to Bulk Concentrate [1]					
	Ni	Cu	Co	Pt	Pd	Au
Gabbro/Massive Sulphide	83%	95%	68%	75%	81%	70%
Clinopyroxenite/Pyroxenite	75%	88%	64%	59%	73%	66%
Peridotite/Dunite	68%	66%	55%	58%	58%	59%

Source: Eggert, 2014
[1] Recoveries are normalized to a bulk concentrate grade containing 6% Ni.

Table 13.2: 2015 PEA Base Case Mill Feed Tonnage by Geo-Metallurgical Domain

Geological Domain	PEA Base Case Mill Feed Split	
	First 16 years	Life of Mine
Gabbro	11%	8%
Clinopyroxenite/Pyroxenite	88%	83%
Peridotite	1%	10%
Total Mill Feed*	**100%**	**100%**

Source: SNC, 2015
* Totals may not add due to rounding

Table 13.3: 2015 PEA Recoveries

Concentrate Grades	Nickel		Copper		PGMs+Au	
	6-9%		4-8%		12-17 g/t	
PEA Recoveries	Ni	Cu	Co	Pt	Pd	Au
Life of Mine	75%	89%	64%	61%	72%	60%
Years 1-16	76%	90%	65%	62%	73%	60%

Source: Eggert, 2015

13.2 Historical Metallurgical Testing

Considerable metallurgical test work has been done on the Property to understand the recovery characteristics of the main rock types using bulk flotation, in addition to limited testing of sequential flotation. Metallurgical test programs that focused primarily on bulk flotation were completed in 1988 by SGS, in 2011 by G&T Metallurgical Services Ltd. (G&T), in 2012 by SGS and in 2014 by XPS. In 2014, SGS conducted a limited program that reviewed the potential benefits of sequential flotation. These studies were considered in detail for this PEA to estimate the metallurgical performance and mineral processing designs for the main rock types.

It is noted that the 2014 XPS and 2011 G&T programs focused primarily on testing samples classified as peridotite, as did the 2014 SGS test program. The extensive 1988 and 2012 SGS test programs focused primarily on samples containing predominantly pyroxenite, clinopyroxenite and gabbro, which are expected to make up 99% of the material used in the first 16 years of the mine life. Stockpiled material to be processed after the first 16 years of mine life is expected to contain 76% of the pyroxenite, clinopyroxenite and gabbro material. With its higher serpentine content and typically lower grades, peridotite tends to have lower overall recoveries compared to the other rock types, and minimal test work has been completed to date for this rock type.

Historical metallurgical reports from Inco Technical Services (Inco Tech) and CANMET were not available. However, the information from these reports was discussed in the 1989 Watts, Griffis and McOuat Limited (WGM) PFS Study report that was reviewed as part of this PEA. In addition, many of the historical tests were described in the Wellgreen Project Preliminary Economic Assessment prepared by Tetra Tech Wardrop, with an effective date of August 1, 2012 (2012 PEA).

13.2.1 SGS Lakefield – 1988

As discussed in the 2012 PEA, drill core rejects from the 1987 drilling program were tested in 1988 and 1989 at SGS, Inco Tech and CANMET to investigate the metallurgical behaviour and obtain data on the mineralization. This test work was summarized in the WGM 1989 PFS report. Additional test work was done at CANMET in the 1990s and summarized by Cabri et al. (1993).

Analysis of preliminary metallurgical tests conducted in early 1988 indicated that a bulk concentrate grading approximately 5% Cu and 4% Ni would recover up to 95% of the copper, 85% of the nickel, 80% of the platinum and 80% of the palladium.

These results were produced from a feed that contained 0.87% Cu, 0.65% Ni, 1.03 g/t Pt and 0.75 g/t Pd. A sequential flotation program that produced separate copper and nickel concentrates was also investigated.

SGS continued to test optimization of the flowsheet and reagent scheme with a bulk concentrate approach, and it issued a second report in November 1988. The lower grade materials included in these samples were similar to the material that could be anticipated at an open pit operation. The introduction of a high-speed conditioning step before the cleaning step of the bulk concentrate resulted in an increase in the concentrate grades and a slight increase in copper recovery. The results from those flotation tests are shown in Table 13.4 and Table 13.5. Magnetic separation was tested on the bulk flotation concentrate, but not the bulk flotation tails, to determine if a PGM concentrate could be developed. Gravity separation was also introduced before the flotation circuit to determine if there was an improvement in the overall PGM recoveries. (In contrast, the 2014 Mineral Resource Estimate includes a magnetic separation step on the bulk flotation tails, but not the bulk flotation concentrate, to increase PGM recovery and generate a separate PGM concentrate.)

Table 13.4: SGS Lakefield Flotation Test Comparison

High Speed Conditioning	Test No.	Product	Weight (%)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Cu (%)	Ni (%)	Pt (%)	Pd (%)
No	54	Bulk Cleaner Concentrate	8.26	10.260	5.690	9.470	7.010	94.1	76.8	68.3	70.6
		Bulk Combined Tail	91.74	0.058	0.160	0.396	0.263	5.9	23.2	31.7	29.4
		Head (Calculated)	100.00	0.900	0.610	1.150	0.820	100.0	100.0	100.0	100.0
Yes	80	Bulk Cleaner Concentrate	8.57	10.800	5.490	6.050	5.330	96.1	81.1	62.0	71.8
		Bulk Combined Tail	91.43	0.042	0.120	0.360	0.200	3.9	18.9	38.0	28.2
		Head (Calculated)	100.00	0.960	0.580	0.840	0.640	100.0	100.0	100.0	100.0
Yes	79	Bulk Cleaner Concentrate	10.22	8.760	4.610	6.230	4.700	95.5	82.1	68.0	71.4
		Bulk Combined Tail	89.78	0.047	0.115	0.330	0.210	4.5	17.9	32.0	28.6
		Head (Calculated)	100.00	0.940	0.570	0.940	0.670	100.0	100.0	100.0	100.0

Source: SGS, 1988

Table 13.5: SGS Lakefield Flotation Test Results for Lower Grade Mineralized Material

Test No.	Product	Weight (%)	Cu (%)	Ni (%)	Pt (g/t)	Pd (g/t)	Cu (%)	Ni (%)	Pt (%)	Pd (%)
54	Bulk 3rd Cleaner Concentrate	4.38	12.100	6.990	8.720	7.120	93.7	74.1	51.0	63.2
	Bulk 1st Cleaner Concentrate	6.27	8.610	5.230	7.200	5.410	95.4	79.3	60.3	68.7
	Bulk Rougher Concentrate	11.63	4.700	3.000	4.310	3.110	96.6	84.2	67.0	73.2
	Bulk Rougher Tail	88.37	0.022	0.074	0.280	0.150	3.4	15.8	33.0	26.8
	Head (Calculated)	100.00	0.570	0.410	0.750	0.490	100.0	100.0	100.0	100.0

Source: SGS, 1988

As noted in the 2012 PEA, an elemental analysis of a typical copper/nickel cleaner concentrate produced in the SGS laboratory was completed and the results are shown in Table 13.6. The sample indicates low levels of gold, magnesium oxide and PGMs in the concentrate.

Table 13.6: SGS Lakefield Cleaner Concentrate Analysis

Element	Measurement	Content
Copper	%	11.5
Nickel	%	5.4
Cobalt	%	n/a
Gold	oz/t	0.091
Silver	oz/t	1.04
Platinum	oz/t	0.2
Palladium	oz/t	0.18
Rhodium	oz/t	0.005
Iron	%	36.6
Sulphur	%	29
Lead	%	0.02
Zinc	%	0.59
Arsenic	%	0.43
Antimony	%	0.004
Silica	%	8.54
Alumina	%	1.11
Lime	%	1.17
Magnesium oxide	%	3.13

Source: SGS, 1988

13.2.2 Inco Tech and CANMET – 1988

The PFS issued by WGM in April 1989 referred to metallurgical test programs conducted by SGS Lakefield in 1988 (Section 13.2.1) and CANMET. The Inco Tech report, “Laboratory Batch Flotation Tests on Wellgreen Composite No. 2,” has not been sourced.

Additional CANMET work was completed and documented in the February 1991 report ”Process Mineralogy of Samples from the Wellgreen Cu-Ni-Pt-Pd Deposit, Yukon.”

13.2.3 G&T– May 2011

In May 2011, G&T received approximately 609 kg of coarsely crushed mineralized material from the Property. From this material, a composite called *Peridotite Composite 1* was constructed. The chemical composition is shown in Table 13.8.

Table 13.7: Chemical Composition of Peridotite Composite 1

Element	Copper	Nickel	Iron	Sulphur	Platinum	Palladium	Carbon
Symbol	**Cu**	**Ni**	**Fe**	**S**	**Pt**	**Pd**	**C**
Units	**%**	**%**	**%**	**%**	**g/t**	**g/t**	**%**
Peridotite Composite 1	0.29	0.26	10.3	1.80	0.28	0.25	0.17

Source: G&T, 2011

Six batch flotation tests were completed using this peridotite material: three rougher flotation tests and three cleaner flotation tests. The conclusions from this test work can be found in G&T's report, "Metallurgical Assessment of the Wellgreen Deposit, Yukon Territory, Canada – KM2833; May 5, 2011." The testing program did not include any magnetic separation of the bulk flotation tail. Results from the G&T 2011 report included the following:

- Chalcopyrite and pentlandite were present in almost equal quantities: approximately 0.8% each. Pyrrhotite was the predominant sulphide mineral present: 3.3%. The fragmentation characteristics of the composite, measured at 93µm K80, determined that 34% of the chalcopyrite and 35% of the pentlandite were liberated. With such low liberation levels, a fine, primary grind size would be anticipated;
- Flotation test data determined that the peridotite material could be processed at a primary grind size of 65 to 93µm K80. The finer, primary grind size of 65µm K80 produced slightly superior recoveries compared to the rougher concentrates. Regrinding had a limited impact on flotation performance. With standard flotation conditions, results for this sample were similar to those observed for other nickel deposits at comparable head grades;
- Due to the poor liberation of the sample, investigation of finer primary grinds could potentially result in improved sulphide recoveries. In conjunction with this testing, it was recommended that hardness characteristics be investigated to determine the associated power requirements, and, therefore, economic viability of grinding;
- The addition of Calgon had a positive influence on flotation performance; it substantially improved the copper, nickel, platinum and palladium recoveries. In the final test, 66% of the copper and 64% of the nickel were recovered into a concentrate grading 7.1% Cu and 6.4% Ni;
- Platinum and palladium recoveries were 25% and 53%, respectively. Precious metals (i.e., gold, platinum and palladium) comprised 11.2 g/t in the bulk concentrate, which would likely increase its smelter value;
- The peridotite concentrate contained almost 11% magnesium oxide. At that level, smelting penalties or issues with the sale of the concentrate could be anticipated. Mineralogical analysis of the concentrate was recommended to determine the forms of magnesium in the concentrate and the reasons for recovery and contamination of the bulk concentrate. Using this information, methods for rejecting the identified minerals could be investigated;
- Testing samples from varying geological origins and feed grades was recommended to determine the variation in metallurgical performance across the deposit. Samples that represented the one of the significant geological rock types or an expected plant feed grade could be given priority; and
- Locked cycle testing should be included to measure metallurgical performance on a continuous basis. At this point, quantitative mineralogy should be considered to understand the loss of copper-nickel sulphides in the tailings stream. This analysis could confirm or reject the idea that performance is related to poor fragmentation characteristics.

13.2.4 SGS Vancouver – 2012

In October 2011, SGS Vancouver began a test program that included two composites: a master composite and a high-nickel composite. The master composite was based on two shipments of peridotite, pyroxenite, clinopyroxenite and gabbro submitted to SGS by the Company. The master composite consisted of 80% peridotite+pyroxenite+clinopyroxenite, 15% gabbro, and 5% massive sulphide. The master composite was first riffled to prepare material for Bond work index and Abrasion index testing. All remaining master composite material was crushed to minus 10 mesh and split into 2-kg test charges. The head assay for the master composite is shown in Table 13.8.

Table 13.8: 2012 SGS Master Composite Head Assays

Sample	Cu	Ni	Ni(s)	Co	Fe	S	C(t)	MgO	Pt	Pd	Au	Rh
	%	%	%	%	%	%	%	%	g/t	g/t	g/t	g/t
Master Composite	0.33	0.42	0.37	0.018	11.9	2.53	0.06	22.8	0.41	0.45	0.04	0.04

Source: SGS, 2012

Later in the test program, 120 kg of material was shipped to SGS by the Company and a high-nickel composite was prepared. The high-nickel composite consisted of 70% pyroxenite+clinopyroxenite, 13% gabbro, and 17% massive sulphide. The high-nickel composite material was crushed to minus 10 mesh and split into 2-kg test charges. The head assay for the high-nickel composite is shown in Table 13.9.

Table 13.9: 2012 High Nickel Composite Head Assays

Sample	Cu	Ni	Ni(s)	Co	Fe	S	C(t)	MgO	Pt	Pd	Au	Rh
	%	%	%	%	%	%	%	%	g/t	g/t	g/t	g/t
High Ni Composite	0.52	0.83	0.69	0.044	18.1	6.45	0.04	19.8	0.57	0.61	0.10	0.10

Source: SGS, 2012

The 2012 SGS program included sample preparation, mineralogy and flotation testing. The flotation testwork investigated reagent and flowsheet options for the recovery of a bulk copper-nickel-PGM concentrate and a copper concentrate. Scoping copper-nickel separation tests were also conducted on the bulk copper-nickel concentrate. Batch rougher kinetics, batch cleaner and locked cycle flotation testing were conducted on each of the two composites.

A detailed feed mineralogy was completed on the master composite using Quantitative Evaluation Of Minerals By Scanning Electron Microscopy (QEMSCAN™); this identified any mineral liberations and associations that could be used to develop grade recovery relationships for the sample. Based on the mineral liberation information, primary grind and regrind targets were estimated to achieve the final target concentrate grade. Overall, chalcopyrite, pentlandite, and pyrrhotite liberations were sufficient to produce good metallurgical performance in a bulk rougher circuit. Regrinding of rougher concentrates was recommended to improve the liberation of chalcopyrite and pentlandite and to optimize the performance of the cleaner circuit. In the case of the master composite, the maximum nickel recovery during the rougher flotation process was expected to be approximately 85% of the total nickel.

Standard Bond grindability tests and Abrasion index tests were also conducted. The Bond work index (BWI) from those initial tests was determined to be 19.7 kWh/t for the Wellgreen project master composite. This is considered to be a hard material in the context of the SGS BWI database. The abrasion index was in the soft range of abrasiveness with a Bond abrasion index of 0.088.

A preliminary flotation test was conducted on the master composite. The effects of grind, collector, talc pre-float and carboxymethyl cellulose (CMC) on rougher kinetics were tested. Due to the lack of any positive effect of talc pre-float and CMC addition, the base case (70 g/t sodium isopropyl xanthate (SIPX)) was shown to be the preferred case. Reagent schemes included gangue depression that targeted talc, chlorites, and serpentine as potential diluents in the bulk concentrate. A 90-µm primary grind size was identified as optimum. Additional cleaner flotation tests were conducted based on the rougher flotation conditions of the base case. Open-circuit cleaner testing was conducted to test the effects of the regrind and dispersants/depressants on circuit recovery and bulk copper-nickel concentrate grade. The preliminary cleaner flotation test results showed that an 18% Cu+Ni concentrate grade could be expected at the average copper and nickel recoveries of 79% and 50%, respectively. Under the same test conditions, a combined grade of 14 g/t was achieved for platinum, palladium and gold at recoveries of 22%, 53% and 53%, respectively. It is noted that these samples exhibited a wide range of magnesium oxide (MgO) content in the composites, ranging from 0.6% to 26%.

SGS conducted a small magnetic separation of the bulk concentrate tail followed by a re-grind/flotation process (LCT 2 Tail F). This small test recovered 1.1% of the platinum, 1.0% of the palladium and 1.8% of the gold. Magnetic separation was conducted in LCT 3, where 0.21% of the mass flow was concentrated with a recovery of 3.2% Pt, 2.5% Pd and 2.5% Au.

Following the preliminary testing, test work was done to optimize the flowsheet through a more detailed program. A proposed split flowsheet recommended taking advantage of the parallel cleaner lines. The viability of the split flowsheet was confirmed by means of locked cycle testing through re-circulation of middling streams. The average locked cycle test results over the last three cycles showed that the master composite produced copper, nickel and final bulk concentrates projections as shown in Table 13.10.

Based on the results and observations of this test program, SGS recommended the following tests to increase the confidence in the metallurgical predictions and to further develop this material:

- Test various reagent and optimization combinations to improve nickel and PGM grades and recoveries.
- Extend variability testing to include the following:
    - Production composites, lithology composites, special location and grade variance. Point samples should be used to confirm the developed flowsheet from a geo-metallurgical perspective;
    - Design comminution testing for proper mill sizing and production forecasting. Selection of specific samples for SAG mill design (including, but not limited to, JK DWT, SMC, SPI, CWI) and ball mill sizing (BWI);
- Conduct a variability program using test samples from various geological origins and feed grades to collect additional information; and
- Conduct additional grinding tests on more variability composites.

The results for the high-nickel composite indicated the production of a combined concentrate with 14.5% Cu+Ni grade at average copper and nickel recoveries of 88% and 73%, respectively. Under the same test conditions, the combined platinum, palladium and gold grade of 9 g/t would be expected at recoveries of 38%, 73% and 62%, respectively.

Table 13.10: Copper and Nickel Metallurgical Predictions for Master Composite

LCT-3	Weight	Assays, (Cu, Ni, S, Fe, MgO %) (Pt, Pd, Au g/t)								% Distribution					
Product	**%**	**Cu**	**Ni**	**S**	**Pt**	**Pd**	**Au**	**Fe**	**MgO**	**Cu**	**Ni**	**S**	**Pt**	**Pd**	**Au**
Bulk Clnr 2 Conc.	2.78	11.0	9.28	27.2	3.39	10.3	0.80	31.6	4.13	83.5	57.6	25.9	22.7	62.9	50.3
Ni 3rd Clnr Conc.	2.36	0.59	1.78	12.80	3.55	1.45	0.11	22.4	17.9	3.8	9.4	10.3	20.2	7.5	6.1
Ni 1st Clnr Tail	15.70	0.10	0.34	4.41	0.66	0.41	0.03	--	--	4.3	12.1	23.8	24.9	14.1	12.1
Ni Scav Tail	69.9	0.04	0.11	1.06	0.14	0.06	0.02	--	--	7.6	16.6	25.4	24.1	9.5	25.4
Magnetic Clnr Conc.	0.21	0.85	1.22	8.93	6.19	5.41	0.52	24.4	18.4	0.5	0.6	0.7	3.2	2.5	2.5
Magnetic Rghr Tail	9.00	0.01	0.19	4.50	0.23	0.18	0.02	--	--	0.2	3.7	13.8	4.9	3.5	3.6
Combined Concentrates	**5.36**	**6.01**	**5.66**	**20.1**	**3.57**	**6.22**	**0.48**	**27.3**	**10.8**	**87.8**	**67.6**	**36.9**	**46.0**	**72.9**	**58.9**
Head (Calculated)		0.37	0.45	2.92	0.42	0.46	0.04								

Source: SGS, 2012

13.2.5 SGS Lakefield – 2013/2014

The 2013/2014 SGS Lakefield program, completed and reported in the Flowsheet Development and Variability Testing on Samples from the Wellgreen Deposit report from SGS Canada Inc, Lakefield (Legault and Imeson) dated January 12, 2015. The program was designed to test a sequential concentrate approach, not a bulk concentrate approach. Various flowsheets were investigated; most were focused on initial copper flotation followed by nickel flotation that included magnetic separation at various stages. The entire program comprised 49 tests: F-1 to F-49.

The initial batch tests, F-1 to F-4, were performed on four samples of upper, heavily altered peridotite to assess rougher flotation time. Tests F-5 and F-6 attempted to improve results with de-sliming. In all cases, AERO 4037 was used as the initial reagent, and SIBX was used in the third to fifth roughers. All of the tests demonstrated an increase in the third rougher flotation nickel assay, indicating that the AERO 4037 was not collecting nickel.

Tests F-9, F-10 and F-14 evaluated whether the lower grade, upper alteration zone flotation performance could be improved by de-sliming. Test F-15 attempted non-sulphide gangue pre-flotation with sulphide added to the primary grind; test F 16 used sodium silicate. Test F-17 used sequential flotation; test F-19 used a different collector and no frother. Test F-21 used CMC, and test F-22 used even more CMC than test F-21.

Test F-25 UHS used PAX, magnetic separation before flotation and a bulk flotation. This produced an acceptable concentrate at 8% Cu and 8% Ni, with 21% Cu recovery and 11% Ni. The recovery results of test F-25 confirmed that early activation of the nickel is critical, and that a strong collector is essential to recover both copper and nickel. It also indicated that the near-surface altered materials can be processed to produce a marketable concentrate. Test F-26 duplicated test F-25 on the low sulphur component of the upper, highly altered zone. The results for this test did not produce an acceptable concentrate using a sequential flotation approach.

Only test F-25 produced a concentrate with significant copper and nickel grades. This test involved a cleaning step that was not used in most of the other tests. The results showed that the heavily altered, lower grade materials located near surface do not generally respond well to sequential flotation. Recoveries were very low and there was minimal upgrading. The material had a very low copper head grade, with less than 0.1% Cu.

Tests F-38 to F-42 were performed on a blend of samples initially identified as gabbro. However, upon further review, it was determined that these samples were a mixture of gabbro, clinopyroxenite and pyroxenite. There was no attempt at optimization during these tests; instead, they were performed to determine variability throughout the mineralized zones. The use of three different grind times and different test conditions reduced the utility of these tests.

The remaining tests were performed on two different blends of three parts lower peridotite to five parts pyroxenite, clinopyroxenite and gabbro; this was referred to as the lower ultramafic composite (LUC). Tests F-7 and F-8 were batch tests used to establish rougher flotation times for the LUC blend. Test F-8 incorporated a magnetic separation step ahead of flotation. Tests F-11, F-12 and F-13 assessed how the LUC responded to a sequential flotation with a copper concentrate product and then a copper-nickel product. Test F-11 had no pH modification in the copper rougher and triethylenetetramine (TETA) was added. Test F-12 had 550 ppm lime added to the rougher for pH modification and TETA was added. Test F-13 had no pH modification in the copper rougher and no TETA was added. Test F-18 added a magnetic separation step after the nickel flotation and both lime and TETA were added. F-20 had no magnetic separation step, but both lime and TETA were added.

The test results were conclusive: a sequential flowsheet is sub-optimal for this mix of rock types. These results were also confirmed by the 2014 XPS tests conducted on peridotite material using a bulk flotation process that substantially increased recovery of metals to concentrate. A major advantage of the SGS 2013 program was that it recognized that, although the sequential flotation of blended peridotite, pyroxenite/clinopyroxenite and gabbro material was not optimal, a substantial portion of the PGM metals that were lost to tailings in previous test work could be recovered in a magnetic concentrate. This concentrate is produced after the bulk flotation of sulphides. Additional work is required to fully assess the opportunities of using magnetic separation.

13.2.6 XPS – 2014

In mid-2014, XPS and Wellgreen Platinum completed an historical review of the reports from 1988 to 2014 and concluded the following:

- A bulk concentrate is the optimal approach for the 2014 Mineral Resource Estimate and SGS results from 2013/2014 showed that following an initial copper flotation step, sequential flotation resulted in poor nickel and PGM recoveries;
- A magnetic separation of the bulk float tail followed by a regrind/flotation cycle improves nickel and PGM recoveries; and
- To better understand the differences in optimal recoveries, testwork should focus on the three main metallurgical domains: Gabbro/Massive Sulphide, Pyroxenite/Clinopyroxenite, and Peridotite/Dunite.

Given the limited metallurgical testing completed historically on the Peridotite/Dunite domain, XPS and Wellgreen Platinum agreed that testing should focus on that domain. The intent of the 2014 program was to establish concentrate grade and recovery estimates suitable for block modelling and to provide estimates for the preliminary design of the process plant. The plant will initially process the higher grade gabbro and clinopyroxenite/pyroxenite materials that are available in the first 16 years of mine operations, and include grinding capabilities to process the peridotite material later in the mine plan. Due to its lower recovery rate, the 2014 Mineral Resource Estimate production plan defers processing the Peridotite/Dunite material as long as possible.

Evaluation of the peridotite material comprised 13 tests. The first four evaluated the optimal grind size for testing. QEMSCAM™ analysis of the feed was used to estimate the optimal grind. The flotation bench tests indicated that the coarsest grind (75 µm) optimized performance. Table 13.12 summarizes the data for the grind evaluations, as provided by XPS.

Table 13.11: Summary of Grind Evaluations

Ro Conc	Mass Rec %	% Cu	% Ni	ppm PGM	Cu Rec %	Ni Rec %	PGM rec %
75 µm	6.1	2.21	3.4	4.38	69.3	54.1	35.1
50 µm - 1	7.8	1.72	2.47	3.35	74.8	55.6	34.4
50 µm - 2	7.7	1.75	2.43	3.34	74.9	53.7	34
35 µm	13	1.07	1.56	2.29	75	57.8	39.2
Mag Conc	**Mass Rec %**	**% Cu**	**% Ni**	**ppm PGM**	**Cu Rec %**	**Ni Rec %**	**PGM rec %**
75 µm	19.8	0.15	0.32	1.52	15.3	16.6	39.9
50 µm - 1	14.5	0.15	0.25	1.71	12.2	10.5	32.7
50 µm - 2	13.4	0.16	0.26	1.73	11.9	10	30.6
35 µm	9.4	0.16	0.25	1.64	8.1	6.7	20.4
Final Tail	**Mass Rec %**	**% Cu**	**% Ni**	**ppm PGM**	**Cu Rec %**	**Ni Rec %**	**PGM rec %**
75 µm	74.2	0.04	0.15	0.26	15.3	29.2	25
50 µm - 1	77.8	0.03	0.15	0.32	13.1	33.9	32.9
50 µm - 2	78.9	0.03	0.16	0.34	13.2	36.3	35.5
35 µm	77.7	0.04	0.16	0.39	16.8	35.5	40.5

Source: XPS, 2014

XPS concluded that increasing grind size of the peridotite material would result in the following:

- Flotation concentrate
  - Mass recovery decreased;
  - Grade increased;
  - Metals recovery decreased;
- Magnetic concentrate
  - Mass recovery increased;
  - Metals recovery increased;
- Final tailings
  - Similar mass retained;
  - Similar copper and nickel grade retained, but PGM grade decreased; and
  - PGM losses decreased.

Overall, the optimal grind size for this peridotite material was determined to be P80 of 75 µm because the magnetic separator worked more efficiently at coarser grind sizes where it produced the highest PGM recovery. Copper and nickel rougher recoveries were 69.3% and 54.1%, respectively. The excellent selectivity of the concentrate grades indicates that the recoveries will probably improve with extended flotation time.

After determining the optimal grind size, seven bench tests were performed. These tests evaluated various approaches to maximize the recoveries. The most successful method used the reagent Calgon (a dispersant).

Two cleaner tests were performed. Test results were consistent with the expected recoveries determined by the following statistical analysis (according to the XPS report on the cleaner testing):

- Bulk final concentrate (bulk third cleaner + magnetic cleaner) recovered 62.7% Cu, 58.2% Ni, 49.8% Au, 36% Pd, and 21.6% Pt with concentrate grades of 5.4% Cu and 8.8% Ni;
- Bulk third cleaner concentrate contained 0.27 ppm Rh, 0.43 ppm Ru, 0.26 ppm Ir, and 0.11 ppm Os;
- First cleaner tailings contained more than 16% of palladium and platinum. This stream should be sent to magnetic separation to extract magnetite;
- Magnetic tailing contains 21% Pd and 26% Pt. The recoveries are expected to exceed 30% once the first cleaner tailings are processed through magnetic separation. This stream is a potential candidate for secondary processing, such as hydrometallurgy or direct leaching; and
- Pyrrhotite concentrate did not achieve any significant sulphide upgrades, but it did contain elevated levels of platinum. This could be another candidate to send for leaching.

13.3 Mineral Types

During 2013 and 2014, Wellgreen Platinum consolidated the exploration drilling and logging information into nine standard designations; previously, there were 73 designations. Wellgreen Platinum and Eggert reviewed this information and categorized the batch tests and locked cycle tests from 1988 to 2014 into the following three distinct metallurgical domains:

- Gabbro/Massive Sulphide, highest sulphur and grade with lowest serpentine content;
- Clinopyroxenite/Pyroxenite, moderate sulphur and grade with moderate serpentine content; and
- Peridotite/Dunite, lowest sulphur and grade with moderate to high serpentine content.

Most of the test work was completed on the higher grade, gabbro/massive sulphide and clinopyroxenite/pyroxenite rock types, and the least amount of work was completed on the peridotite and dunite rock types. For planning purposes, the dunite material was included in the Peridotite domain. As a conservative measure; the nickel grade for dunite was reduced by 0.1%; this effectively excluded nearly all of the dunite material from the resource model.

13.3.1 Work Index

During the 2013 test program, SGS Lakefield performed a series of per-domain work index tests. The domains are summarized in Table 13.12 and Table 13.13. To determine these domains, the strip logs were checked to confirm the actual core that was issued for the metallurgical test program with the predicted domain from the model.

Table 13.12: Metallurgical Sample Summary

Sample Lithology		Area	Drill Hole	From (m)	To (m)
Dunite/Peridotite	Low Sulphur	East	WS12-210	2.1	45
Peridotite		East	WS12-210	54	87
Peridotite		Central	WS12-214	24	78.0
Peridotite/Pyroxenite		Far East	WS12-203	249.9	286
Pyroxenite, Clinopyroxenite, Gabbro Mix					
	Pyroxenite/Clinopyroxenite	Far West	WS12-208	13.5	45
	Clinopyroxenite/Gabbro	West	WS12-213	219.8	258
	Pyroxenite	East	WS12-210	159	187
	Clinopyroxenite/Gabbro	Central	WS12-214	319.4	349.6
	Pyroxenite/Clinopyroxenite	Far East	WS12-204	171	206.9

Source: Eggert, 2014

Table 13.13: Bond Work Indices of Metallurgical Samples

	Bond Work Indices				
Sample	Ball		Rod		Sample Lithology
203 Lower	20.6	kWh/t	14.9	kWh/t	Peridotite/Peridotite/Pyroxenite
204 Gabbro	21.3	kWh/t	19.4	kWh/t	Pyroxenite/Clinopyroxenite
208 Lower	19.1	kWh/t	15.9	kWh/t	Pyroxenite/Clinopyroxenite
213 Gabbro	19.1	kWh/t	17.7	kWh/t	Gabbro/Clinopyroxenite
214 Gabbro	16.0	kWh/t	14.5	kWh/t	Gabbro/Clinopyroxenite
210 Gabbro	20.2	kWh/t	19.1	kWh/t	Pyroxenite
214 Upper Hi S	18.8	kWh/t	14.9	kWh/t	Peridotite
214 Upper Lo S	15.1	kWh/t	11.1	kWh/t	Peridotite
210 Upper Hi S	14.4	kWh/t	9.4	kWh/t	Peridotite
210 Upper Lo S	17.1	kWh/t	11.2	kWh/t	Dunite/Peridotite

Source: SGS, Eggert, 2014

The average bond ball index for the Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite domain samples was 19.1 kWh/t; this includes 16 kWh/t for the 214 gabbro/clinopyroxenite sample. These are similar to historic values quoted for the property and they reflect the nature of previous programs, some of which tested only the higher grade, lower tonnage gabbros. The average bond ball index for the peridotite was 17.2 kWh/t. The Clinopyroxenite/Pyroxenite and Gabbro/Massive Sulphide domains comprise approximately 90% of the mill feed in the Life of Mine plan and are approximately 99% of the mill feed during the first 16 years of operation.

Ball mill sizing assumed an average Bond Work Index of 19.0 kWh/t. In the early stages of milling, tonnages will need to be slightly lower if the mill feed is taken strictly from the higher grade Gabbro/Massive Sulphide domain. In addition, the higher grades result in a higher proportion of material directed to concentrates; therefore, the tonnage will need to be decreased to ensure that downstream processes are not overwhelmed. As such, the higher work index does not change the required process design. In the later stages of the project, as lower grade peridotite begins to enter production, the work index will also decrease. Therefore, the project will be able to increase the throughput without affecting the final grind.

13.3.2 Concentrate Grades and Recovery by Domain

A systematic approach was used to estimate concentrate grade and recovery for the three domains. The first step was to eliminate tests that did not achieve potential processing options. Many tests from 1988 to 2014 were conducted to assess grind size, reagent schedules, conditioning time, the effect of depressants, etc. By design, many of these tests produce less than optimal results. The remaining results were analyzed to estimate grade-recovery relationships that could be projected with confidence to estimate the recoveries and concentrate quality during future pilot-plant testing and additional LCT. Where possible, LCT results were used to estimate the concentrate grade-recovery relationship. To date, there have been no LCTs on the Peridotite/Dunite domain; however, this will change during the next stage of testing.

The batch/LCT tests that could reasonably reflect future metallurgical performance, to a PEA level, were evaluated for all of the primary metals to ensured results would not be too optimistic, to a PEA level (in other words, enhancing the performance of one metal might negatively affect the performance of other metals).

13.3.3 Summary of Test Programs

Table 13.14 to Table 13.16 summarize the metallurgical tests used to analyze each domain.

Table 13.14: Metallurgical Tests Used for Gabbro/Massive Sulphide Domain

ID No	Test No	Sample	Type of Test	Mag Setup	Flotation
261	51	Gabbro	LCT	No Mag	Bulk
268	53	Gabbro	LCT	No Mag	Bulk
123	26	Gabbro	Batch	Mag	Bulk
241	47	Gabbro	Batch	No Mag	Bulk
1176	F-34	Clinopyroxenite	Batch	Mag	Sequential

Source: Eggert, 2014

Table 13.15: Metallurgical Tests Used for Clinopyroxenite/Pyroxenite Domain

ID No	Test No	Sample	Type of Test	Mag Setup	Flotation
842a*	LCT -1	Clinopyroxenite	LCT	No Mag	Sequential
477	F-4	Clinopyroxenite	Batch	No Mag	Bulk
853	LCT-3	Clinopyroxenite	LCT	Mag	Bulk
860	LCT-4	Clinopyroxenite	LCT	Mag	Bulk
1176	F-34	Clinopyroxenite	Batch	Mag	Sequential
491	F6	Clinopyroxenite	Batch	No Mag	Bulk
801	HNI-F1	Clinopyroxenite	Batch	No Mag	Bulk
624	F23	Clinopyroxenite	Batch	No Mag	Bulk
251	49	Clinopyroxenite	Batch	No Mag	Bulk
1298a	F44	Clinopyroxenite	Batch	Mag	Sequential
1166	F33	Clinopyroxenite	Batch	Mag	Sequential

Source: Eggert, 2014
* Used in copper and nickel analysis only; not used in platinum and palladium analysis.

Table 13.16: Metallurgical Tests Used for Peridotite/Dunite Domain

ID No	Test No	Sample	Type of Test	Mag Setup	Flotation
1390b	13	Peridotite	Batch	Mag	Bulk
1156	F-32	Peridotite	Batch	Mag	Bulk

Source: Eggert, 2014

13.3.4 Gabbro/Massive Sulphide Metallurgical Domain

The Gabbro/Massive Sulphide metallurgical domain tends to be higher grade; therefore, the recoveries, for any particular targeted concentrate grade, are expected to be higher than the other two metallurgical domains. The locked cycle test results for the Gabbro/Massive Sulphide domain, where conditions have been considered optimal and not experimental to date, are summarized in Table 13.17.

Table 13.17: Gabbro/Massive Sulphide Summary

Test Sort ID	Mass to Conc.	Concentrate Ni		Concentrate Cu		Concentrate Pt		Concentrate Pd	
		Grade (%)	Recovery (%)	Grade (%)	Recovery (%)	Grade (g/t)	Recovery (%)	Grade (g/t)	Recovery (%)
261	15.02	3.59	80.0	5.38	93.1	6.10	75.5	4.00	76.0
268	14.91	3.37	80.1	4.67	93.1	6.00	78.6	3.60	77.0
123	25.7	2.23	87.5	3.34	94.7	3.14	82.5	2.23	90.6
241	17.07	7.85	79.3	6.43	91.3	5.41	65.3	5.71	70.2
1176	49.98	0.69	93.1	0.72	95.4	0.72	93.5	0.50	96.2

Source: Eggert, 2014

A linear regression of the nickel concentrate grade and nickel recovery produces the following equation:

Ni Recovery = -1.769 * Ni in Concentrate Grade + 90.7 + 3.5, with an R^2 of 0.60

Optimization of the grind, re-grind and reagent addition, as well as the magnetic separation process, is expected to increase the recovery by 3.5%; this is based on the results of two optimized tests that shifted the linear regression upward compared to batch test results. This optimized result will be confirmed by future locked cycle and pilot-plant testing.

Similarly, concentrate copper grade versus recovery of copper to concentrate produces the following equation:

Cu Recovery = -0.575 * Ni in Concentrate Grade + 95.6 + 2.4, with an R^2 of 0.88

Platinum recovery produces the following equation:

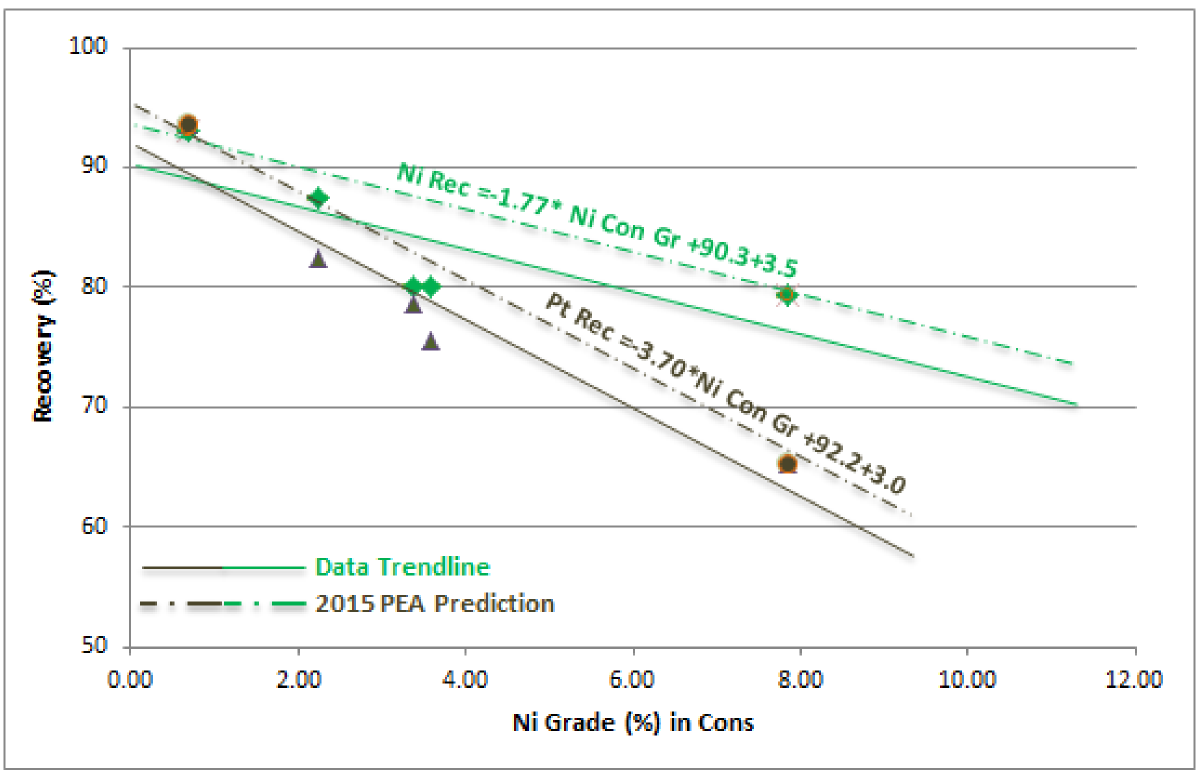
Pt Recovery = -3.70 * Ni in Concentrate Grade +92.2 + 3.0, with an R^2 of 0.67

Palladium recovery produces the following equation:

Pd Recovery = -3.582 * Ni in Concentrate Grade + 94.7 + 3.8, with an R^2 of 0.94

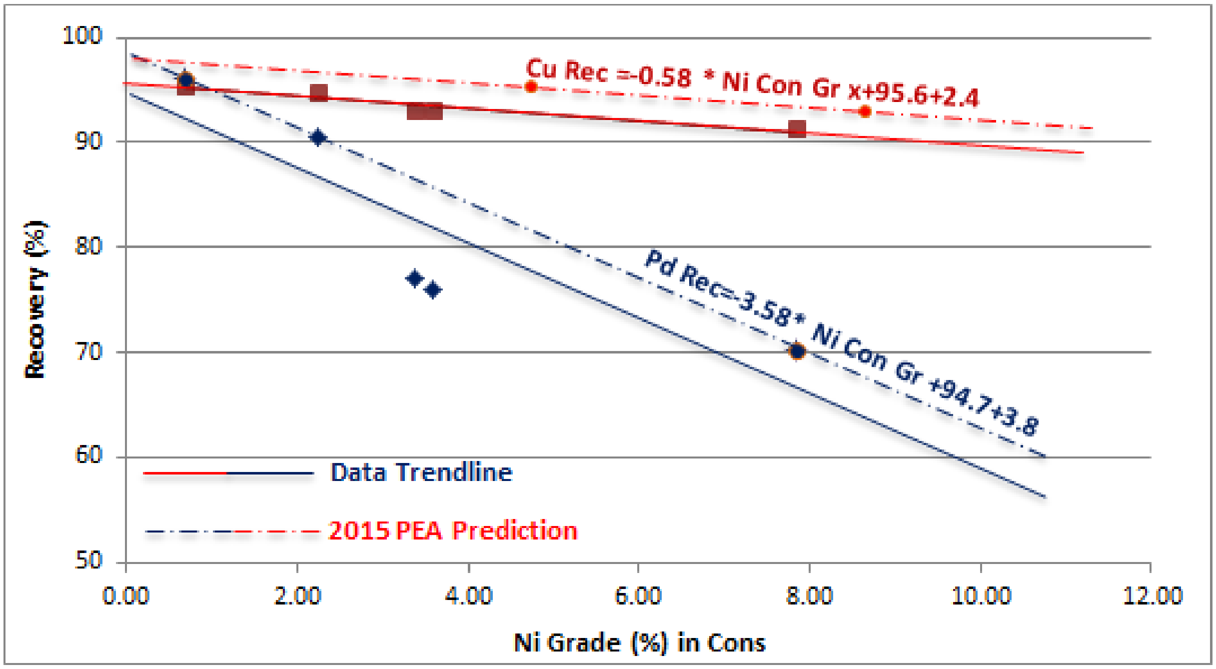
The metallurgical test results and linear regression equations are shown in Figure 13.3 and Figure 13.4. The concentrate grade compared to the expected recovery can be used to estimate the recovery of other metals based on the domain with an expected grade for any other metal.

Figure 13.3: Gabbro/Massive Sulphide Nickel & PtRecovery vs. Ni Grade in Concentrates



Source: Eggert, 2014

Figure 13.4: Gabbro/Massive Sulphide Cu and Pd Recovery versus Ni Grade in Concentrates



Source: Eggert, 2014

13.3.5 Clinopyroxenite/Pyroxenite Domain

Clinopyroxenite/Pyroxenite is the dominant domain processed in the LOM plan. In terms of grade, recovery and concentrate quality, this domain is between the Gabbro/Massive Sulphide and Peridotite/Dunite domains. Eleven bulk flotation locked cycle tests were identified as exclusively Clinopyroxenite/Pyroxenite domain material; these are shown in Table 13.18.

Table 13.18: Clinopyroxenite/Pyroxenite Summary

Test Sort ID	Mass to Conc.%	Concentrate Ni		Concentrate Cu		Concentrate Pt		Concentrate Pd	
		Grade (%)	Recovery (%)	Grade (%)	Recovery (%)	Grade (g/t)	Recovery (%)	Grade (g/t)	Recovery (%)
842a	3.26	9.17	62.70	8.99	86.20				
477	20.40	1.82	82.40	1.63	93.10	1.58	78.70	1.93	89.20
853	5.35	5.66	67.60	6.01	87.80	3.57	46.10	6.20	72.90
860	6.83	8.07	68.00	6.78	86.60	2.56	32.40	5.00	66.90
1176	49.98	0.69	93.10	0.72	95.38	0.72	93.53	0.50	96.15
491	22.10	1.68	81.30	1.49	90.60	1.36	73.40	1.60	85.00
801	26.19	2.77	86.80	1.92	94.00	1.50	68.10	1.77	84.00
624	4.71	6.50	64.00	5.61	79.20	4.00	46.80	6.80	66.00
251	8.07	4.12	70.40	11.80	90.70	11.80	69.30	6.52	66.60
1298a	1.73	7.87	35.78	15.76	67.36	4.49	16.43	5.27	29.05
1166	44.55	0.75	92.58	0.82	95.66	6.60	85.72	0.64	89.58

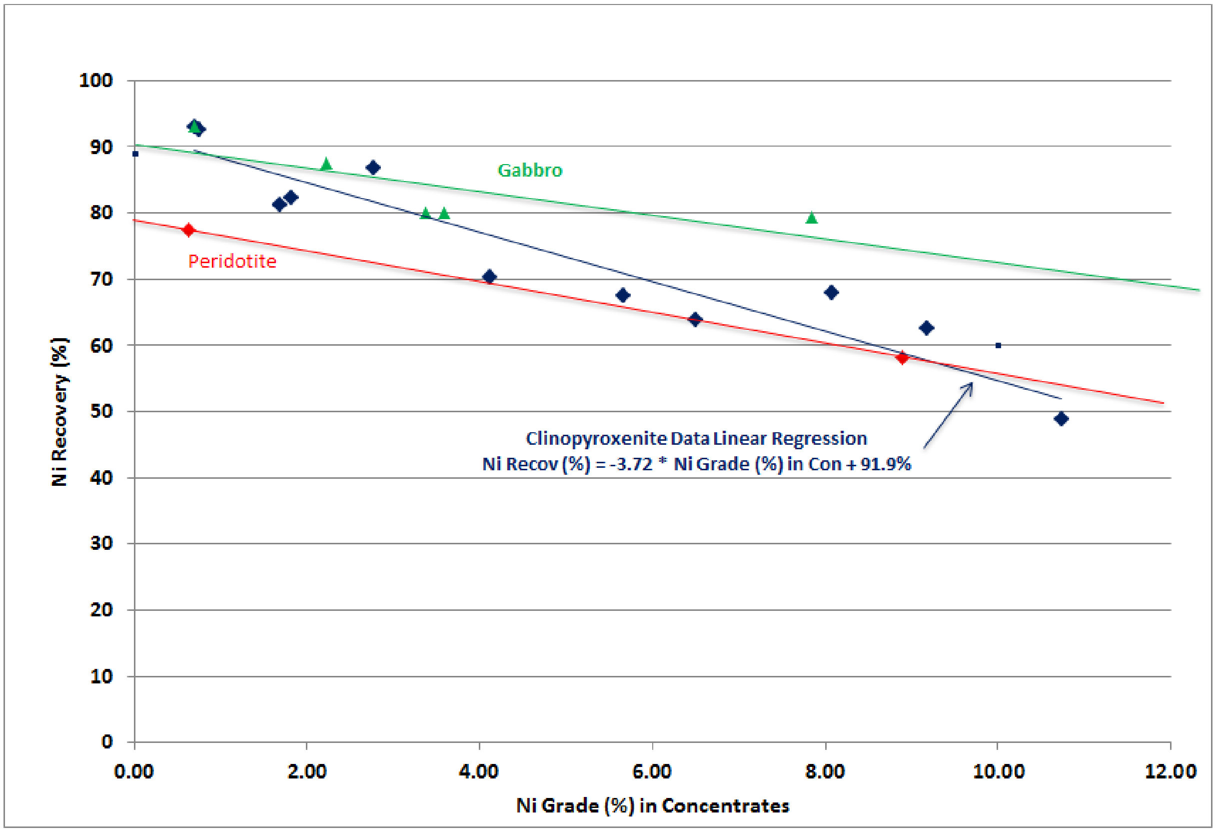
Source: Eggert, 2014

The initial linear regression of the nickel concentrate grade and nickel recovery for the Clinopyroxenite/Pyroxenite domain produced the following equation:

Ni Recovery = -3.72 * Ni in Concentrate Grade + 91.9, with an R^2 of 0.60

As shown in Figure 13.5, the Clinopyroxenite/Pyroxenite domain is projected to achieve a lower recovery than the Peridotite/Dunite domain. In addition, at lower concentrate levels, the Clinopyroxenite/Pyroxenite domain was projected to have a higher nickel recovery than the Gabbro/Massive Sulphide domain.

Figure 13.5: Clinopyroxenite/Pyroxenite Ni Recovery versus Ni Grade in Concentrates - Initial Data

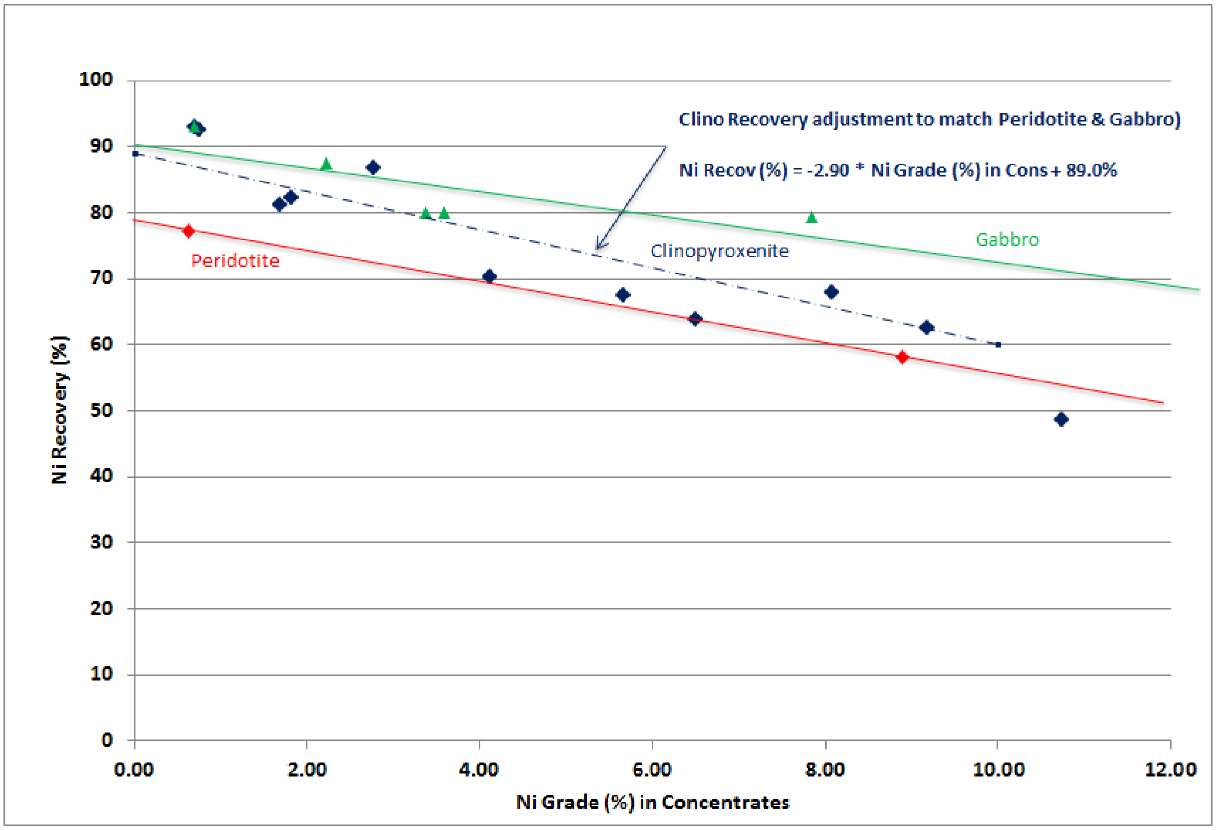


Source: Eggert, 2014

Because the Clinopyroxenite/Pyroxenite domain is expected to perform better than the Peridotite/Dunite domain, but not as well as the Gabbro/Massive Sulphide domain, the linear regression equation was adjusted:

Ni Recovery = -2.90 * Ni Concentrate Grade + 89.0

Figure 13.6: Clinopyroxenite/Pyroxenite Ni Recovery versus Ni Grade in Concentrates - Initial Adjustment

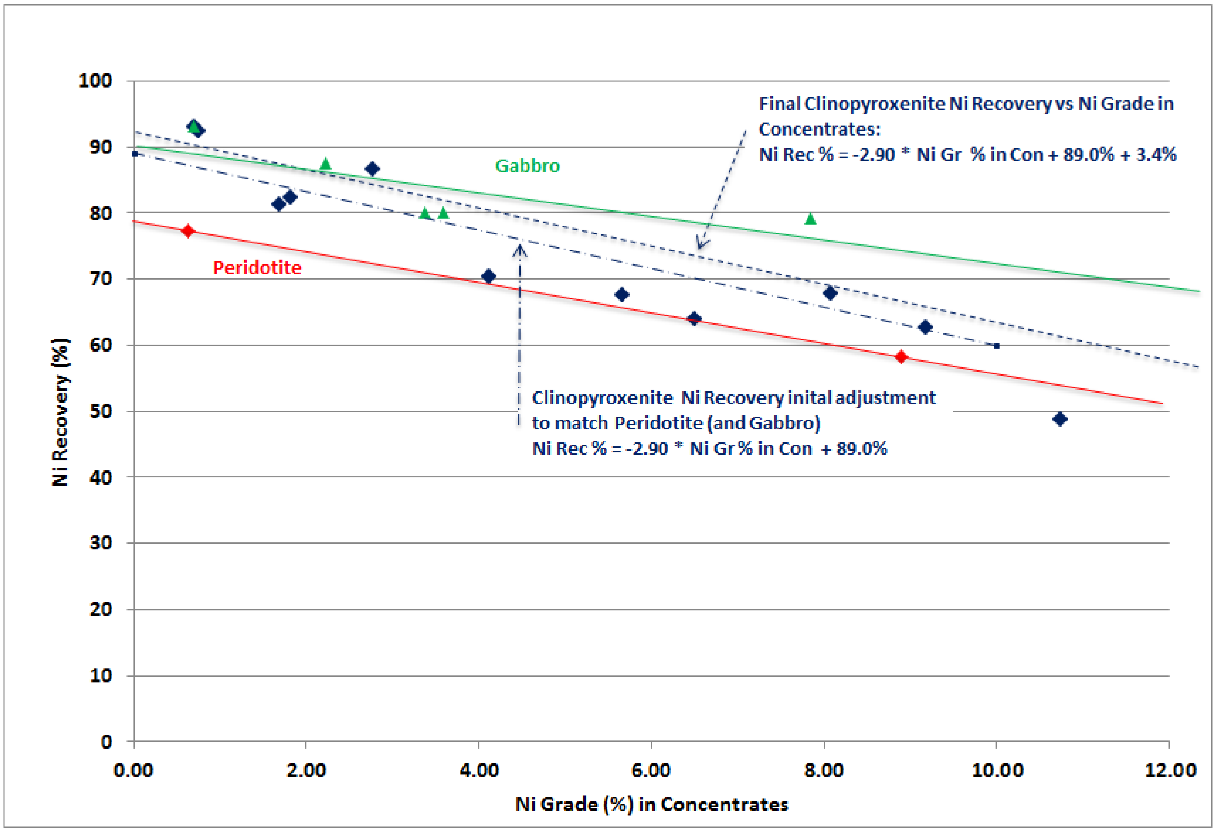


Source: Eggert, 2014

The regression formula was then adjusted upwards by 50% of the average results using tests 860a and 1176; this resulted in a 3.4% incremental increase in recovery. Therefore, the final regression equation (shown in Figure 13.7) is:

Ni Recovery = -2.90 * Ni in Concentrate Grade + 89.0 + 3.4

Figure 13.7: Clinopyroxenite/Pyroxenite Ni Recovery versus Ni Grade in Concentrates - Final Adjustment



Source: Eggert, 2014

The copper recovery equation is based on the linear regression with a fit of R^2 = 0.75 and an incremental recovery increase of 1.8% based on the results of tests 801 and 860a:

Cu Recovery = -1.66 * Ni in Concentrate Grade + 96.3 + 1.8

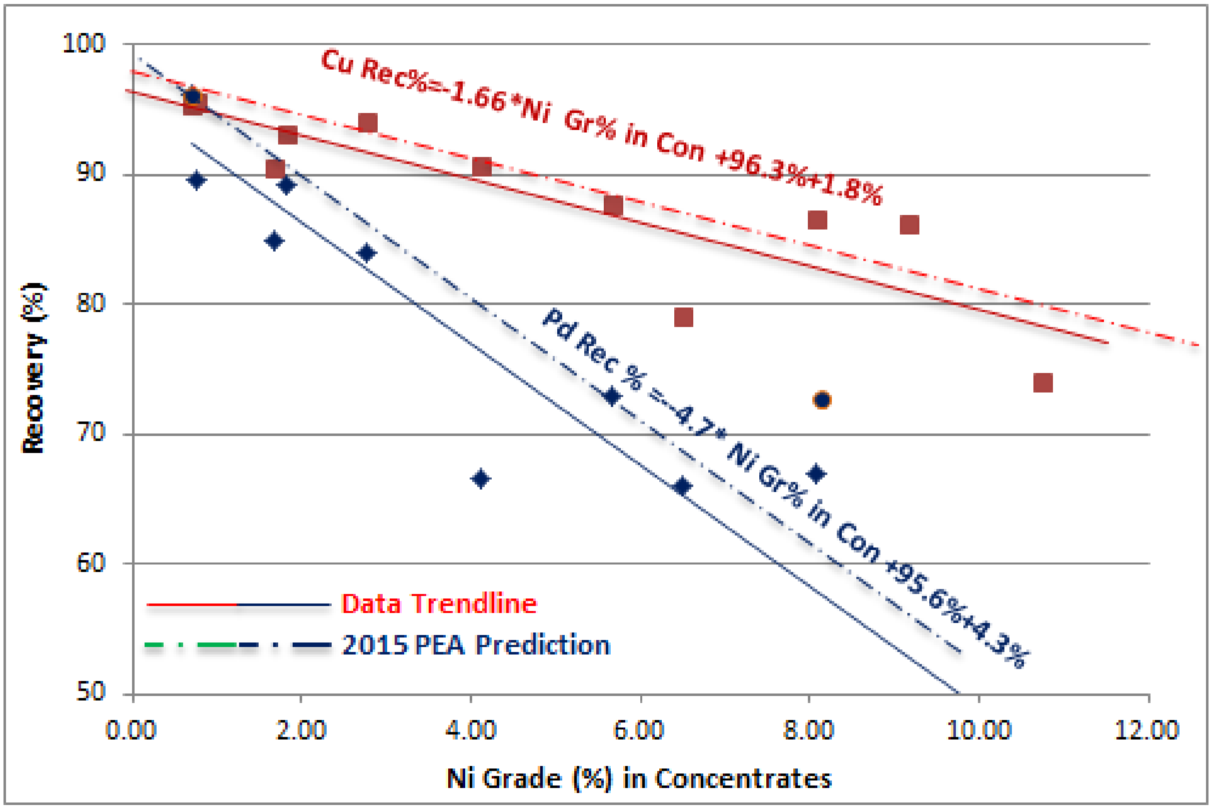
Palladium performance follows a linear regression of recovery versus the nickel grade in concentrate with an R^2 of 0.897:

Pd Recovery = -4.68 * Ni in Concentrate Grade + 95.6 + 4.4

This includes the 50% incremental increase in recovery based on tests 1166 and 860a.

The performance of palladium and copper are shown in Figure 13.8.

Figure 13.8: Clinopyroxenite/Pyroxenite Cu & Pd Recovery versus Ni Grade in Concentrates



Source: Eggert, 2014

Platinum recoveries indicated an excellent R^2 value of 0.951. The initial linear regression was very close to the Peridotite/Dunite domain:

Clinopyroxenite/Pyroxenite **Pt Recovery = -6.68 * Ni in Concentrate Grade + 90.4**

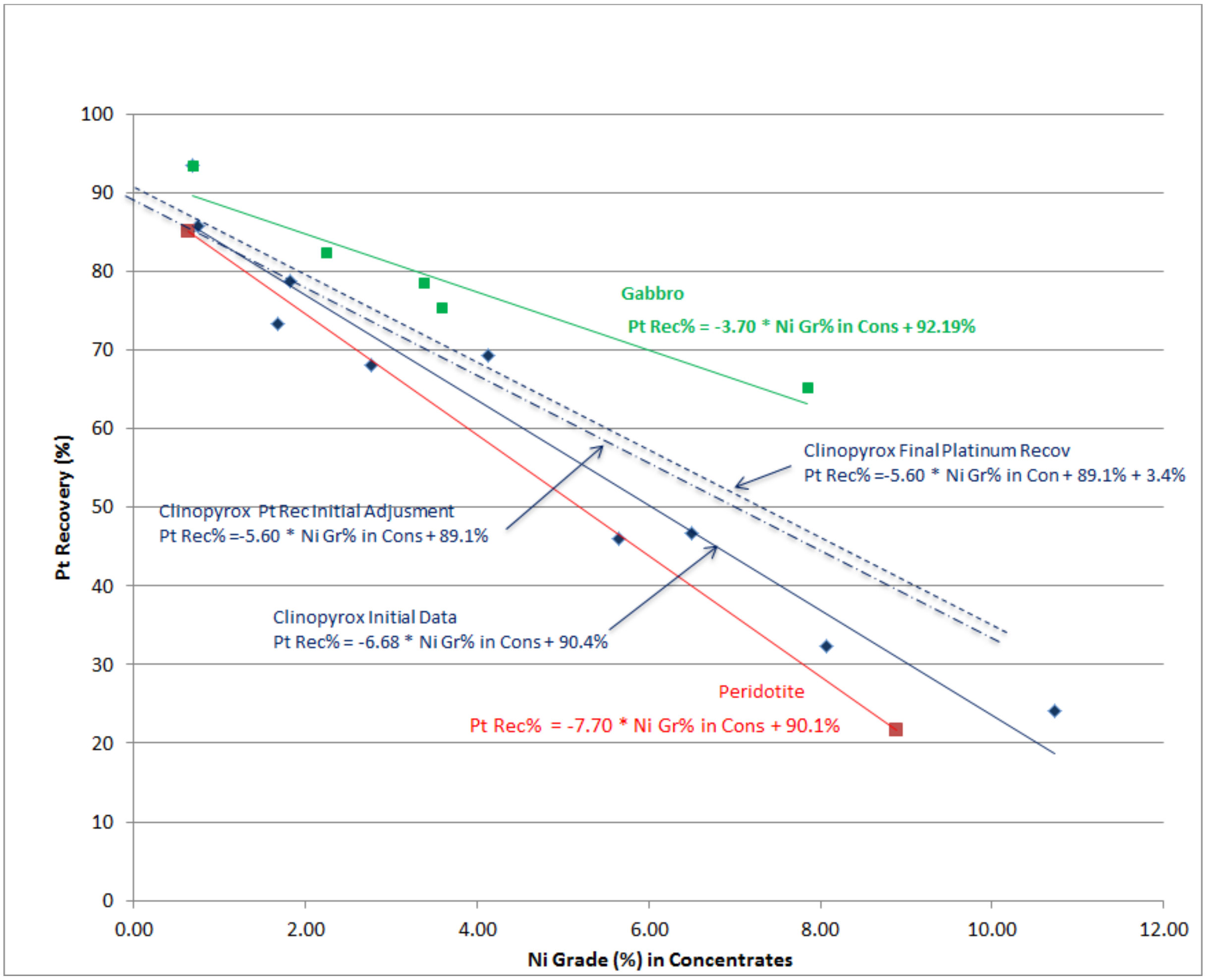
XPS Peridotite **Pt Recovery = -7.70 * Ni in Concentrate Grade + 90.1**

The success of the XPS peridotite test program indicates that future LCT and pilot-plant test programs could enhance the performance of the Clinopyroxenite/Pyroxenite domain. In addition, based on the results from tests 1176 and 251, these programs have the potential to increase performance; the average supports an increase of 3.5% recovery in the following regression:

Pt Recovery = -5.6 * Ni in Concentrate Grade + 89.1 + 3.5

The metallurgical test results and linear regressions for platinum are shown in Figure 13.9.

Figure 13.9: Clinopyroxenite/Pyroxenite Ni Concentrate Grade versus Pt Recovery to Concentrate



Source: Eggert, 2014

13.3.6 Peridotite/Dunite Domain

The Peridotite/Dunite domain has the lowest grade and lowest sulphur content; of the three domains, it exhibits the lowest metallurgical performance with respect to concentrate nickel grade versus recoveries. In this domain, PGM recoveries also tend to decrease faster than base metals as concentrate base metal grades increase.

The 2014 Mineral Resource Estimate analysis focused on two peridotite tests: Test F32 by SGS in 2013 (Sort ID 1176) and XPS test 13 (Sort ID 1390b); both were batch tests (Table 13.19).

Table 13.19: Peridotite/Dunite Domain Test Summary

Test	Mass to	Concentrate Ni		Concentrate Cu		Concentrate Pt		Concentrate Pd	
Sort ID	Conc. (%)	Grade (%)	Recov (%)	Grade (%)	Recov (%)	Grade (g/t)	Recov (%)	Grade (g/t)	Recov (%)
1390b	2.20	8.89	58.20	5.45	62.70	3.52	21.62	4.74	36.01
1156	39.08	0.64	77.35	0.29	65.96	0.54	85.13	0.64	89.09

Source: Eggert, 2014

The test results and linear regression plots for these two batch tests are shown in Figure 13.5. Because only two points were considered, the R^2 value is 1 in all cases. As noted elsewhere in this report, material from the Peridotite/Dunite domain will be stockpiled as much as possible to the end of the mine life and then processed as low grade mill feed. In the first 16 years of mine operations, the Peridotite/Dunite domain accounts for approximately 1% of the mill feed. However, in the latter years of the mine life, when the stockpiles are processed, it is expected to account for approximately 24% of the material processed in the mill. Therefore, XPS completed an initial review of the historical metallurgical testing at the Property and then completed a batch test with initial optimization of grind size, regrinding, magnetic separation, conditioning time and reagent selection specific to the Peridotite/Dunite domain.

With respect to dunite, this PEA has taken a conservative approach and decreased the nickel grade by 0.1%; this has removed nearly all dunite from the resource model. The remaining amount of dunite is combined with the Peridotite domain.

The projected improvement in copper recovery (1.3%), created by additional LCT and pilot-plant test programs, is based on 50% of the historical batch test to locked cycle test optimization [(2.22 + 2.74)/ 2 * 50% = 1.3] achieved in the past, and calculated for a 6% and 8% Ni in concentrate grade. (Table 13.20):

Table 13.20: Anticipated Incremental Increase in Copper Recovery

Test Program	6% Ni Concentrate	8% Ni Concentrate
LCT Copper Recovery	86.6 %	83.3 %
Batch Copper Recovery	84.4 %	80.6 %
LCT Increase from Batch to LCT	2.2 %	2.7 %

Source: Eggert, 2014

The anticipated increase in palladium recovery (1.9% in future LCT and pilot-plant test programs) is based on palladium recoveries in the Clinopyroxenite/Pyroxenite domain (Table 13.21).

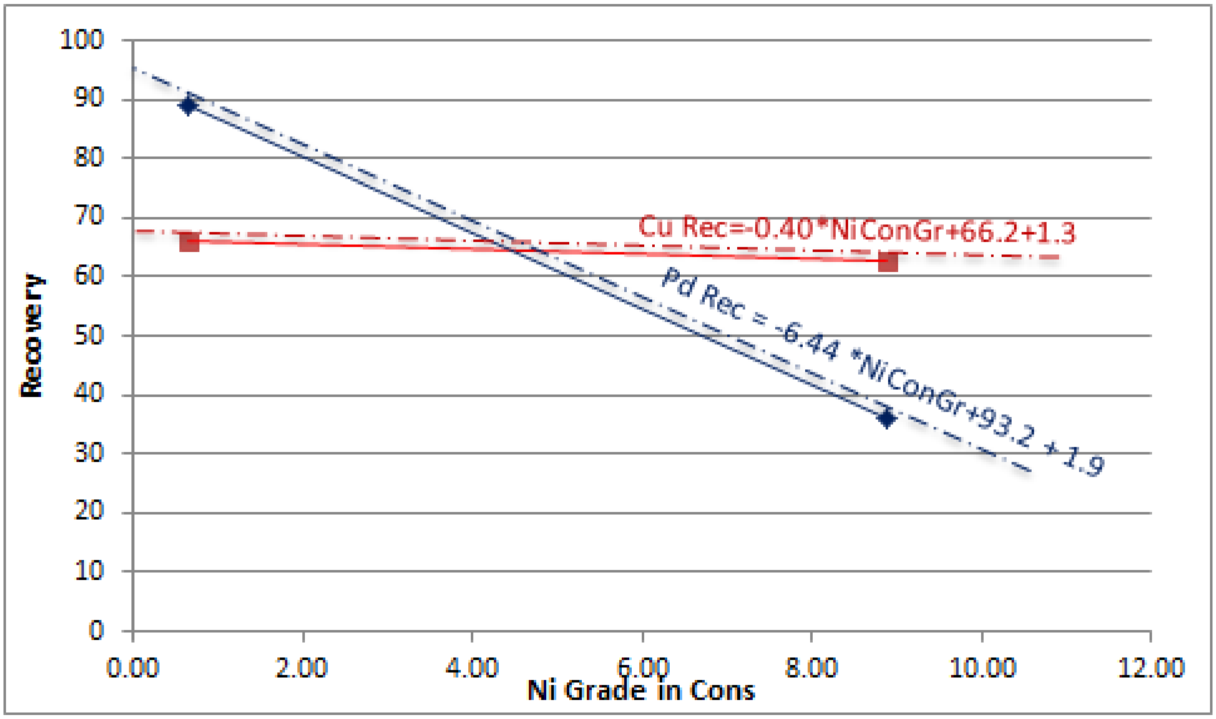
Table 13.21: Anticipated Incremental Increase in Palladium Recovery

Test Program	6% Ni Concentrate	8% Ni Concentrate
LCT	68.5 %	59.7 %
Batch	64.2 %	56.5 %
LCT Increase	4.40 %	3.19 %

Source: Eggert, 2014

The average increase for the two concentrates was 3.8%; this was then factored by 50% for palladium recoveries in the Peridotite/Dunite domain.

Figure 13.10: Peridotite/Dunite Cu and Pd Recovery versus Ni Grade in Concentrates



Source: Eggert, 2014

To predict the increase in nickel recoveries created by additional LCT and pilot-plant test programs, one-half of the incremental increase that occurred in the Clinopyroxenite/Pyroxenite domain was used. This is summarized in Table 13.22; the average is expressed as 3.25 = (5.90 + 7.08) /2 * 50%.

Table 13.22: Anticipated Incremental Increase in Nickel Recovery

Test Program	6% Ni Concentrate	8% Ni Concentrate
Clino LCT Ni Recovery	71.6 %	65.8 %
Clino Batch Ni Recovery	65.7 %	58.7 %
Ni Recovery Increase from Batch to LCT	5.9 %	7.1 %

Source: Eggert, 2014

The 6.9% incremental increase in platinum recovery is based on the Clinopyroxenite/Pyroxenite domain, as shown in Table 13.23. The improved platinum recoveries from batch testing to the predicted LCT test program are an incremental increase of 13.8%; this was then factored by 50% for the Peridotite/Dunite domain (Table 13.23).

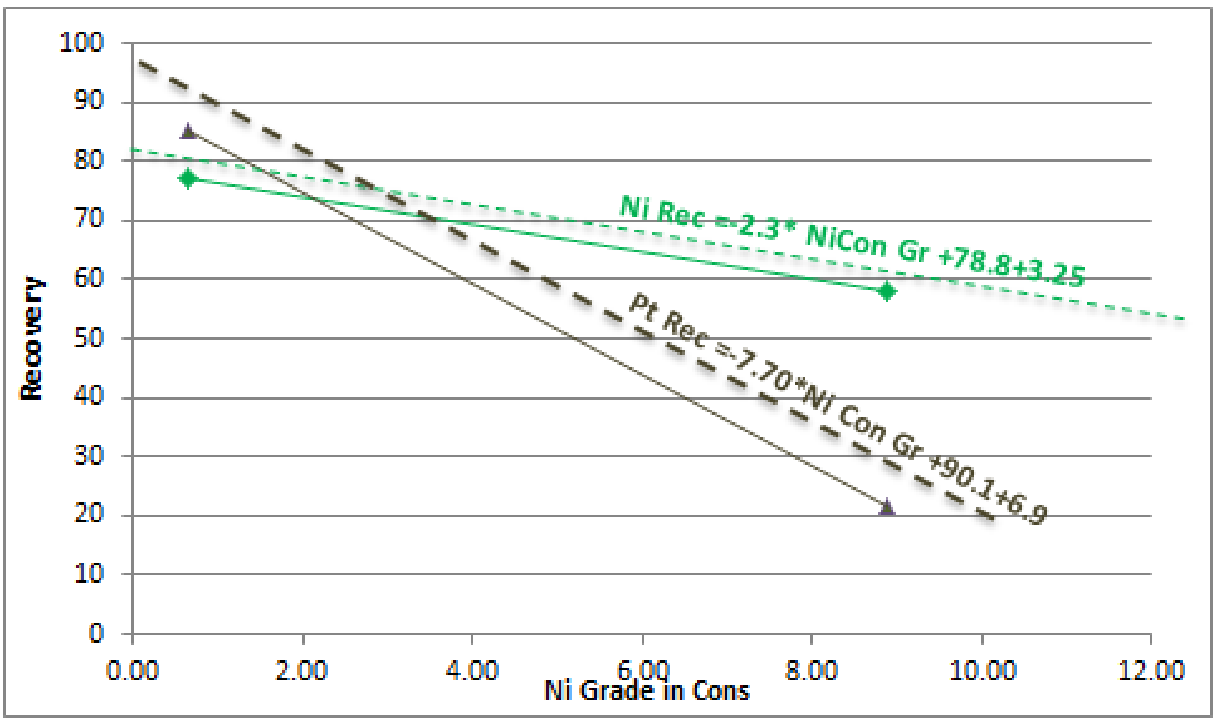
Table 13.23: Anticipated Incremental Increase in Platinum Recovery

Test Program	Pt Rec in 6% Ni Concentrate
LCT	55.5 %
Batch	41.7 %
LCT Increase	13.8 %

Source: Eggert, 2014

Figure 13.11 shows the platinum and nickel recoveries and the expected optimization after the extensive LCT and pilot-plant tests.

Figure 13.11: Peridotite/Dunite Ni and Pt Recovery versus Ni Grade in Concentrates



Source: Eggert, 2014

The XPS results associated with the Peridotite/Dunite domain are summarized as follows:

Primary Grind

- Coarser grind produced better upgrading of sulphides;
- Recovery by flotation was optimized at a grind of 50 µm (A grind finer than 50 µm does not improve recovery;
- A fine grind decreased the recovery of magnetite using a magnetic separator, and significantly reduced the recovery of precious metals; and
- A 75-µm grind was determined to be the best of the three options tested.

Rougher Screening Tests

- Using dispersant or depressant, regardless of the type, enhanced nickel recoveries;
- CMC and brine solution versus gangue resulted in better selectivity for nickel;
- Generally, copper kinetics and recovery were not affected by test conditions; this included the use of Calgon;
- Tests indicated maximum rougher recoveries were 69% Cu and 62% Ni; and
- Rougher concentrate recovered up to 35% Au, 55% Pd, and 40% Pt.

Mineralogy and Entitlement

- Feed contained approximately 5% sulphides, 11.5% magnetite, 70% serpentine, and a balance of other gangue minerals;
- 84% of the nickel existed in pentlandite; however, approximately 15% of the pentlandite forms sub-micron inclusions within the serpentine grains;
- Overall, nickel entitlement of this sample was 69% for flotation; and
- Micrographs showed significant complexes of sulphides/magnetite/serpentine in concentrates and tailings. Regrinding is required to liberate and upgrade concentrate.

Cleaner Testing

- Bulk final concentrate (bulk third cleaner + magnetic cleaner) recovered 62.7% Cu, 58.2% Ni, 49.8% Au, 36.0% Pd, and 21.6% Pt at a combined concentrate grade of 5.4% Cu and 8.8% Ni;
- Bulk third cleaner concentrate also contained 0.27 ppm Rh, 0.43 ppm Ru, 0.26 ppm Ir, and 0.11 ppm Os;
- First cleaner tailings contained over 16% of combined Pd and Pt. This stream should be sent to magnetic separation to extract magnetite;
- Magnetic tailings contained 21% Pd and 26% Pt. The recoveries are expected to be more than 30% once the first cleaner tailings are processed through magnetic separation. This stream is a potential candidate for secondary processing, such as hydrometallurgy or direct leaching; and
- Pyrrhotite concentrate did not achieve any significant upgrading of sulphides, but did contain elevated levels of platinum. This could be another candidate to send for leaching.

13.3.7 Gold Recoveries

Clinopyroxenite/Pyroxenite Domain

The metallurgical gold data associated with the Clinopyroxenite/Pyroxenite domain is shown in Table 13.24. (When no gold analysis was done, tests were deleted).

Table 13.24: Clinopyroxenite/Pyroxenite Test Summary

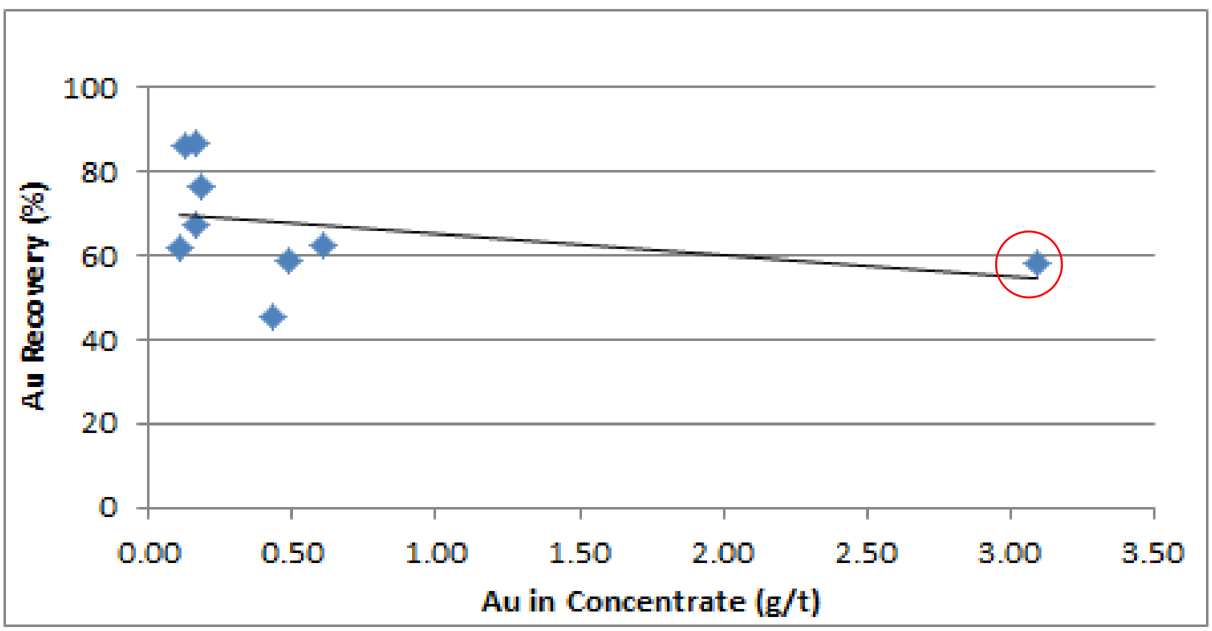
Sort ID	Test No	Mass Pull (%)	Cu Head (%)	Ni Head (%)	Au Conc. Grade (g/t)	Au Recovery (%)	Ni Grade (%)
842a	LCT-1	3.26	0.34	0.48	n/a	n/a	9.17
477	F4	20.40	0.36	0.45	0.16	67.50	1.82
853	LCT-3	5.35	0.37	0.45	0.48	58.90	5.66
860	LCT-4	7.42	0.55	0.83	0.64	62.30	8.15
1176	F-34	49.98	0.38	0.37	0.13	86.28	0.69
491	F6	22.10	0.36	0.46	0.11	61.80	1.68
801	HNI-F1	26.19	0.53	0.83	0.18	76.50	2.77
624	F23	4.71	0.33	0.48	0.43	45.40	6.50
1298a	F-44	2.60	0.41	0.38	3.09	57.96	10.74
1166	F-33	44.55	0.38	0.36	0.16	86.60	0.75
251	49	8.07	1.05	0.47	n/a	n/a	4.12

Source: Eggert, 2014

* n/a indicates that gold analysis was not performed.

As shown in Figure 13.12, sample 1298a was interpreted to be an outlier.

Figure 13.12: Gold Recoveries versus Gold Grade in Concentrates with Test 1298a Outlier

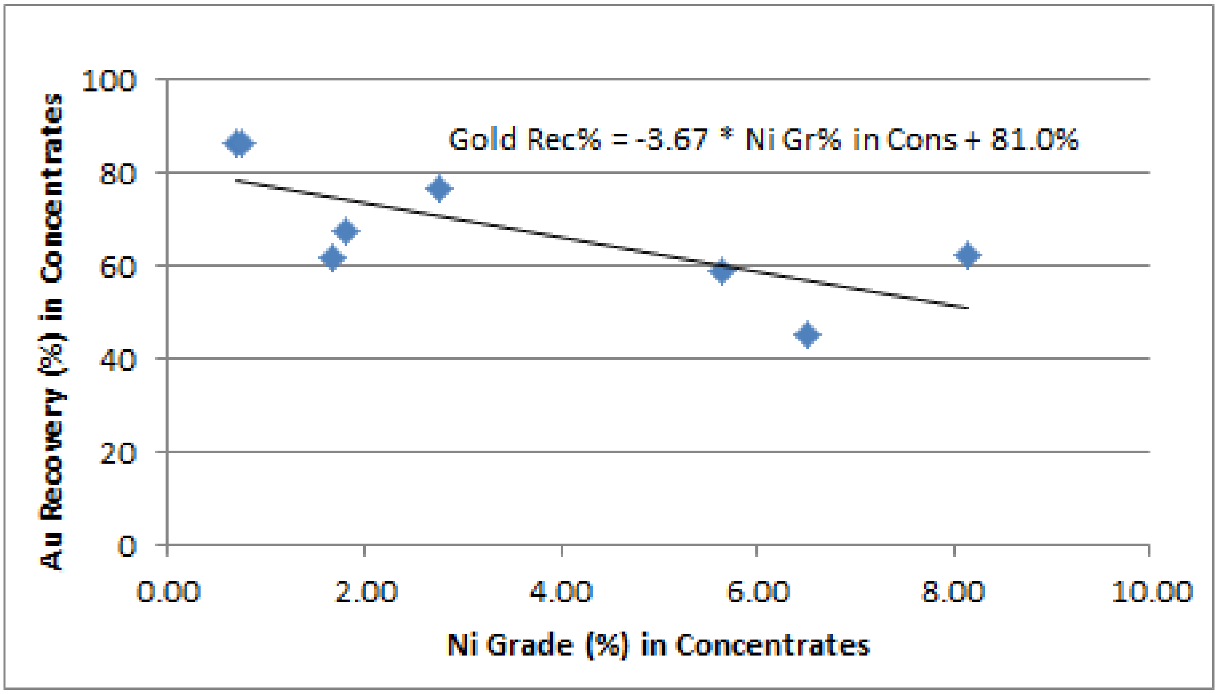


Source: Eggert, 2014

Figure 13.13 shows the gold metallurgical performance and linear regression formulas with Test 1298a removed: this resulted in the following linear regression:

Au Recovery = -3.68 * Ni in Concentrate Grade + 81.0

Figure 13.13: Gold Recoveries versus Gold Grade in Concentrates without Test 1298a Outlier



Source: Eggert, 2014

Gabbro/Massive Sulphide Domain

The metallurgical gold data associated with the Gabbro/Massive Sulphide domain is shown in Table 13.25. A single data point was used to generate the metallurgical performance predictions.

Table 13.25: Gabbro/Massive Sulphide Test Summary

Test No	Mass Pull (%)	Cu Head (%)	Ni Head (%)	Au Con Grade (g/t)	Au Recovery (%)	Ni Con Grade (%)
51	15.02	0.87	0.65	n/a	n/a	3.59
53	14.91	0.75	0.68	n/a	n/a	3.37
26	25.70	0.91	0.65	n/a	n/a	2.23
47	17.07	1.20	1.69	n/a	n/a	7.85
F-34	49.98	0.38	0.37	0.13	86.28	0.69

Source: Eggert, 2014
* n/a indicates that gold analysis was not performed.

A conservative approach was used to determine where gold recovery would be the same for both the Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite domains. The Clinopyroxenite/Pyroxenite domain is expected to perform at a lower level than the Gabbro/Massive Sulphide domain.

The Gabbro/Massive Sulphide linear regression formula is:

Au Recovery = -3.68 * Ni in Concentrate Grade + 81.0 + 7.8

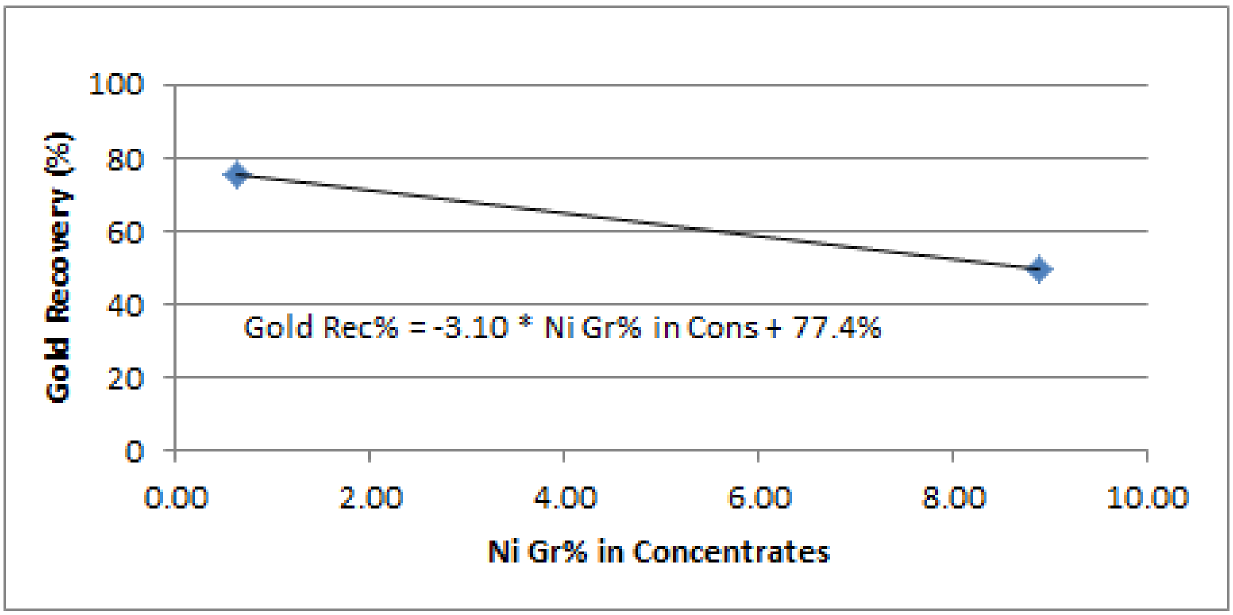
Based on the regression formula, Test F-34 is expected to have a recovery of 86.3%, at a 0.69% Ni bulk concentrate grade versus the actual recovery of 86.3%.

At a 6% Ni bulk concentrate grade, the regression formula predicts a recovery of 66.7%. The concentrate grade is also based on the Clinopyroxenite/Pyroxenite domain and equates to 0.65 g/t when the bulk concentrate is grading 6% Ni.

Peridotite/Dunite Domain

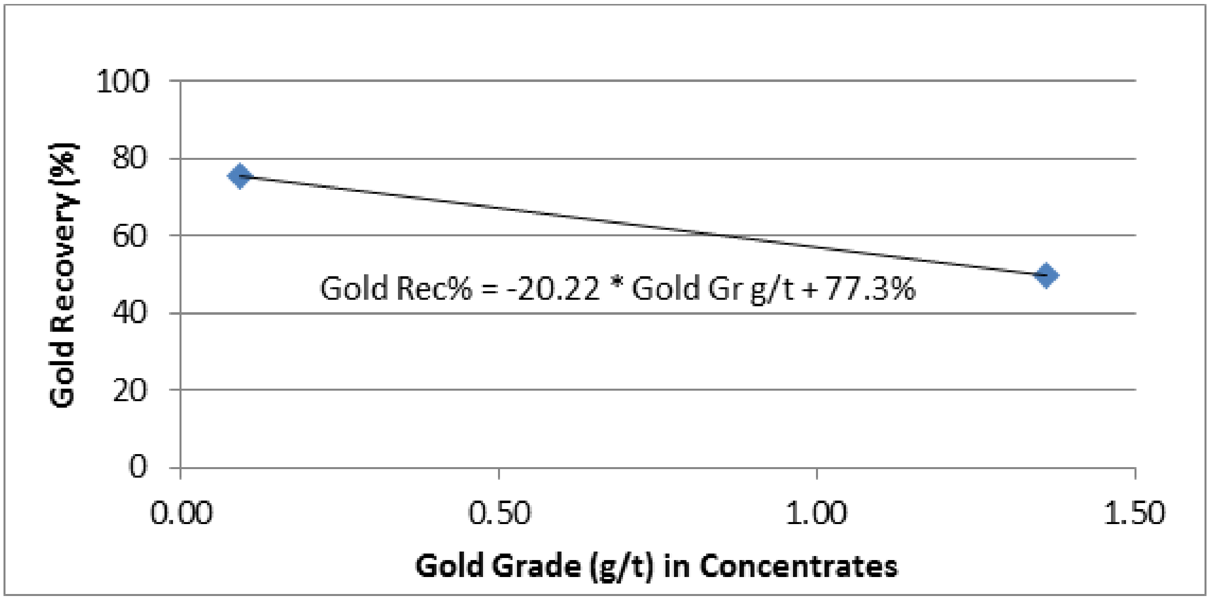
The Peridotite/Dunite domain would exhibit a gold recovery of 58.8% with a grade of 0.92 g/t Au in the bulk concentrate. Given the lack of metallurgical testing on this domain, the study did not include an incremental increase for gold metallurgical performance created by additional LCT and pilot-plant test programs.

Figure 13.14: Gold Recoveries versus Nickel Grade in Concentrates for Peridotite/Dunite Domain



Source: Eggert, 2014

Figure 13.15: Peridotite/Dunite Domain Gold Recoveries versus Gold Grade in Concentrates



Source: Eggert, 2014

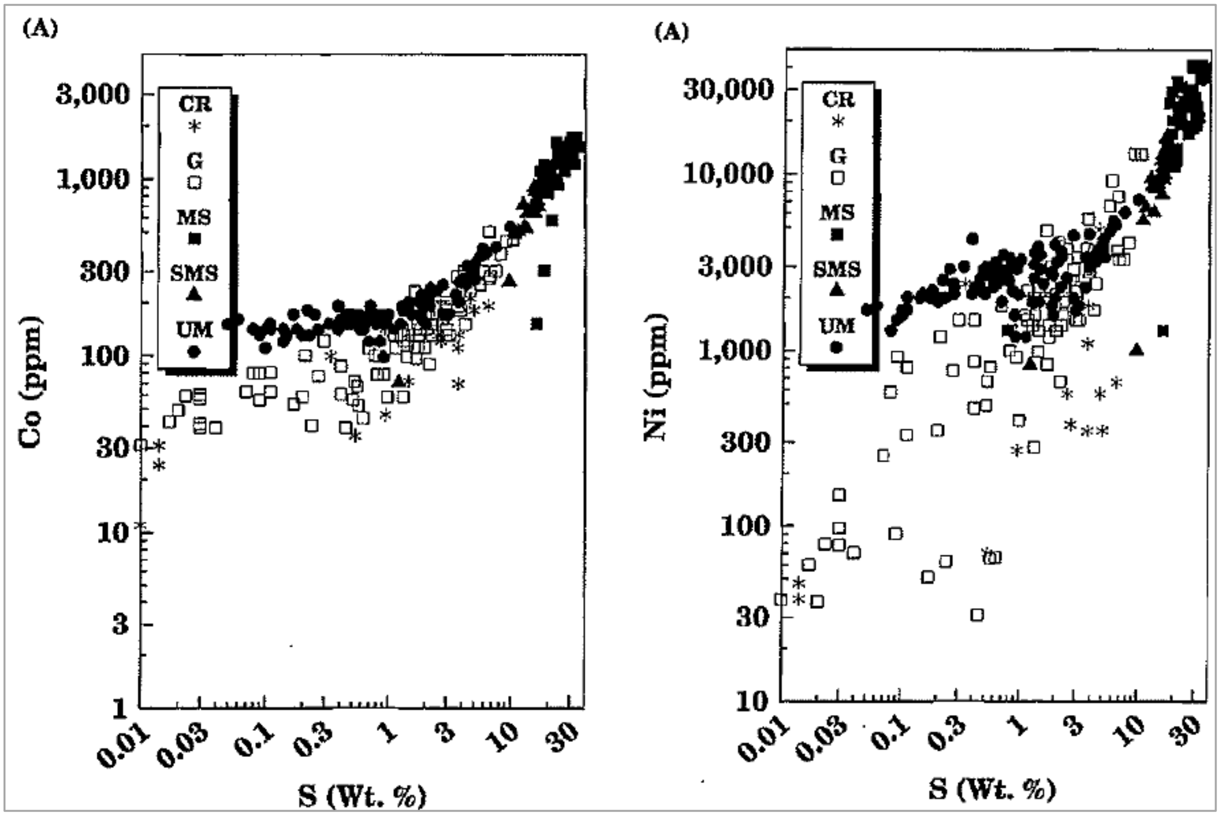
13.3.8 Cobalt Recoveries

A single data point with a cobalt assay was used to calculate the metallurgical performance for the Gabbro/Massive Sulphide, Clinopyroxenite/Pyroxenite and Peridotite/Dunite domains: peridotite with a 48.4% recovery and a 0.52% Co grade, in a concentrate grading 8.89% Ni. Therefore, the 2012 PEA results will be used for the Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite domains, and the single data point will be used for the Peridotite/Dunite domain as follows:

- 64.4% recovery in Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite domains with a 0.60% Co grade in the bulk concentrate; and
- 48.4% recovery in Peridotite/Dunite domain with a 0.52% cobalt grade in the bulk concentrate.

The metallurgical performance then relies on the correlation between cobalt and nickel, as shown in Figure 13.16, to determine a potential incremental increase related to future metallurgical test programs.

Figure 13.16: Scatter Plots of Cobalt and Nickel versus Sulphur



Source: L.J. Hulbert, 1995

The incremental increase in the cobalt metallurgical performance for the three domains is projected to be similar to nickel because they are both sulphide metals with good correlation. A conservative approach was taken in the Clinopyroxenite/Pyroxenite domain: the incremental value was set to zero to ensure it was consistent with the Gabbro/Massive Sulphide and Peridotite/Dunite domains (Table 13.26).

Table 13.26: Cobalt Recoveries with Incremental Increases by Domain

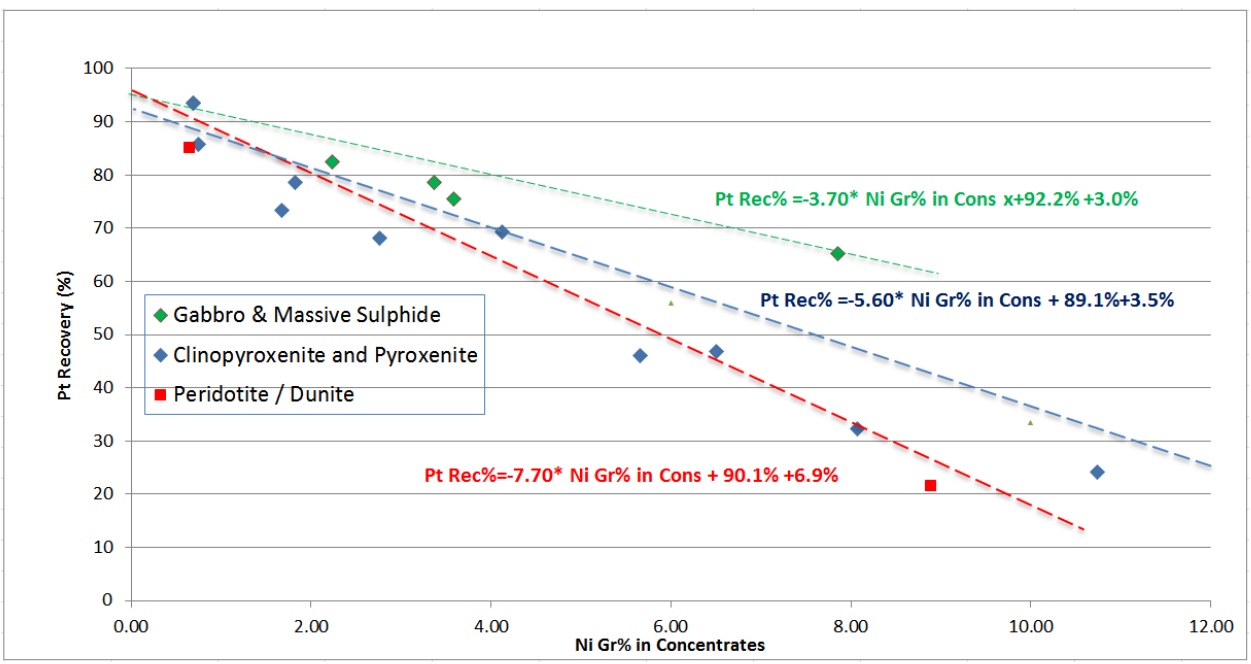
	Co Recovery (%)	Co Conc. Gr (%)	2014 Resource Grade
Gabbro/Massive Sulphide			
Base	64.4	0.6	
Ni Incremental	3.2		
Subtotal Gabbro/Massive Sulphide	67.6	0.6	0.019%
Clinopyroxenite/Pyroxenite			
Base	64.4	0.6	
Ni Incremental	0		
Subtotal Clinopyroxenite/Pyroxenite	64.4	0.6	0.016%
Peridotite/Dunite			
Base	48.4	0.52	
Ni Incremental	6.7		
Subtotal Peridotite/Dunite	55.1	0.52	0.014%

Source: Eggert, 2014

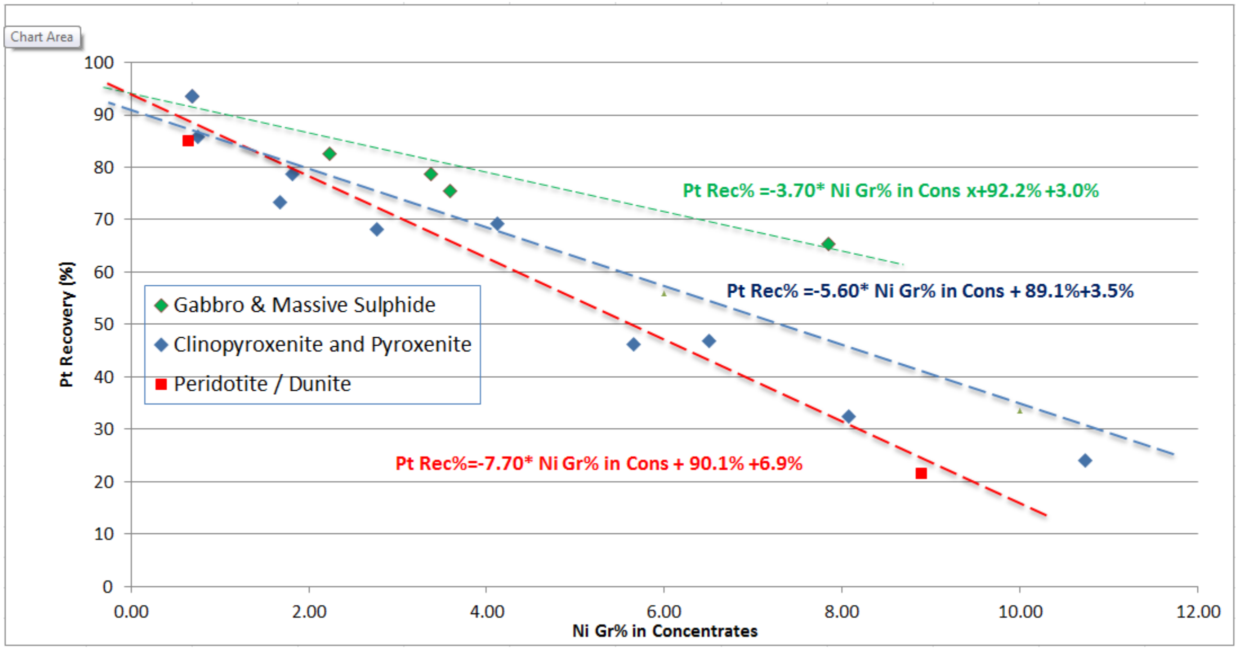
13.3.9 Summary of Recoveries by Domain

Figure 13.17 to Figure 13.20 summarize recoveries of the primary metals by domain and indicate the linear regression formulas used in this PEA.

Figure 13.17: Platinum Recoveries by Metallurgical Domain

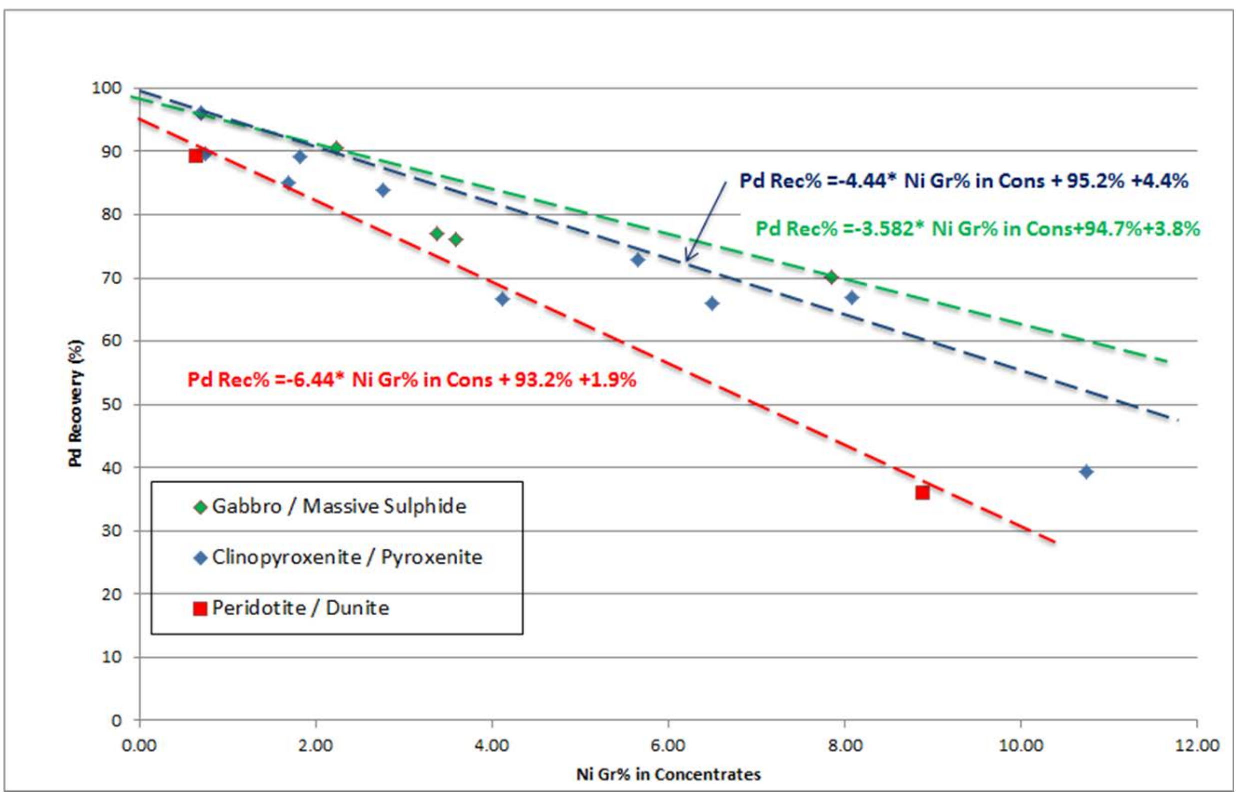


Source: Eggert, 2014



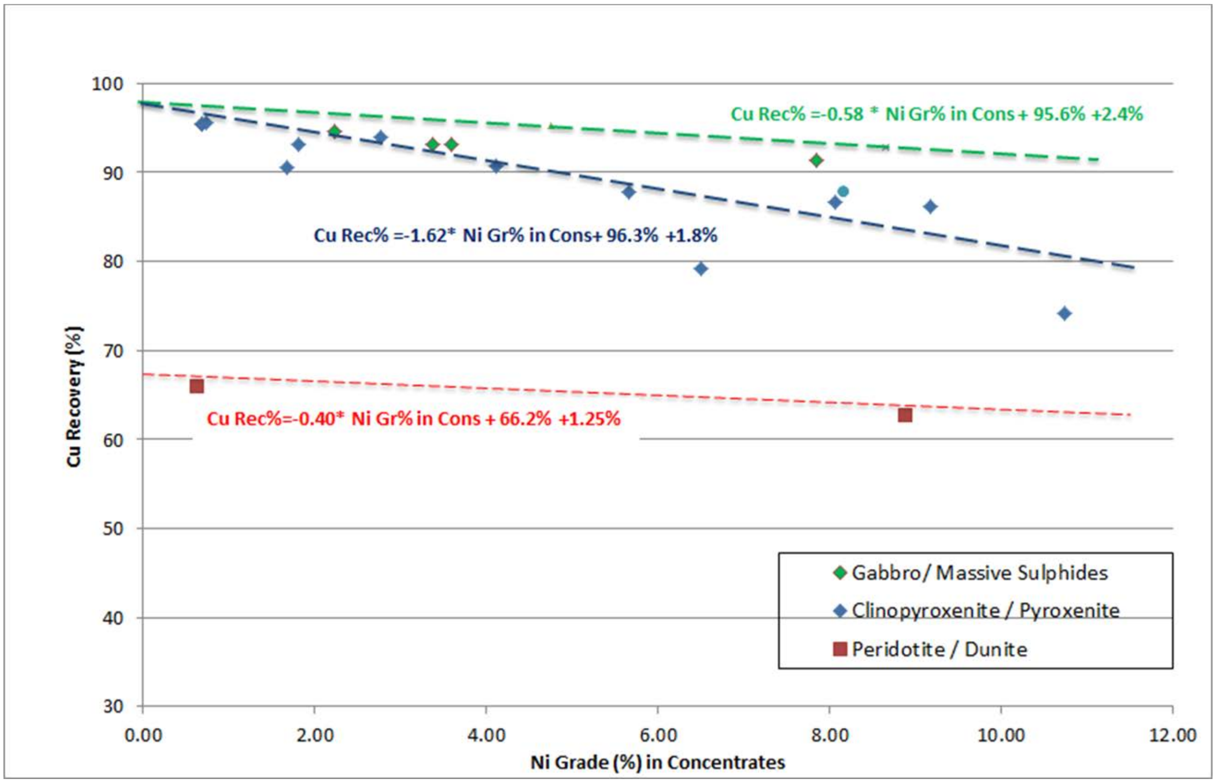
Source: Eggert, 2014

Figure 13.18: Palladium Recoveries by Metallurgical Domain



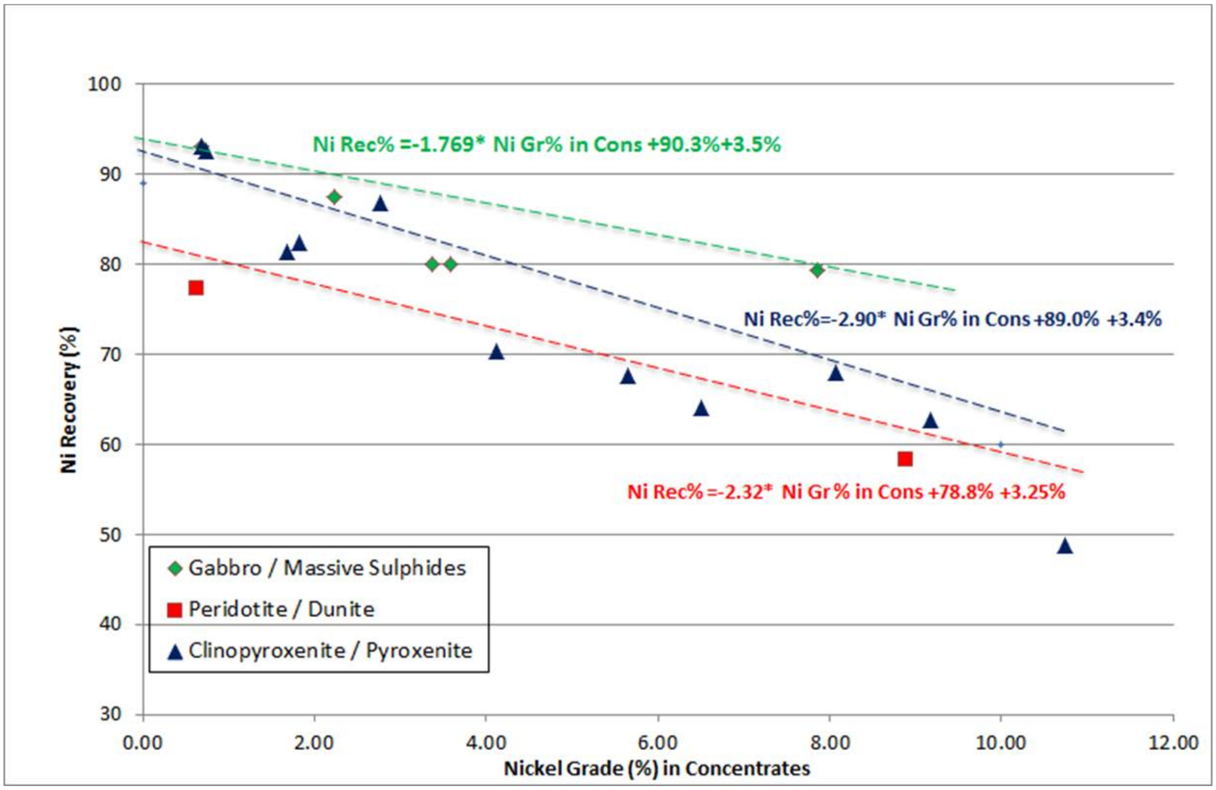
Source: Eggert, 2014

Figure 13.19: Copper Recoveries by Domain



Source: Eggert, 2014

Figure 13.20: Nickel Recoveries by Domain



Source: Eggert, 2014

13.4 Silver Grade Estimation

Silver is not considered in the PEA economic model since it is not part of the mineral resource estimate.

Historically, drill core samples from the Wellgreen project were only selectively sampled for silver, therefore, the mineral resource estimate in the 2015 PEA does not include silver. The historical data indicates a wide range of silver grades, from less than 2 g/t in the master composite sample (SGS testing in 2012) to 17.3 g/t in Composite 2 from G&T testing in 2012; however, data points were sporadic, not specifically tied to current geologic domains, and often near-to detection limits; this made results less reliable.

The following is a summary of silver assays in the Wellgreen Platinum database compiled by the Ron Simpson, P.Geo., of GeoSim Services Inc.:

- Pre-1987: No silver data in database;
- 1987/88 re-sampled core: All silver truncated to zero decimal places;
- 2006-2007: No silver data entered in database;
- 2008: below detection set to 0.25 g/t;

- Avg 0.9 g/t with <0.5 set to 0.25 (if below detection set to 0 then average decreases to 0.8 g/t)
- 2009: No silver data entered in database;
- 2010: below detection set to 0.25 g/t;
- Average 0.7 g/t with <0.5 set to 0.25 (if below detection set to 0 then average decreases to 0.5 g/t)
- 2011 data: Below detection limit <0.5 are inconsistent - some remain at 0.5 instead of 0 or 50% of the value;
- 40% of samples have silver data averaging 1 g/t;
- 2012: below detection set to 0.25 g/t;
- 39% of samples have silver data averaging 1.2 g/t; and
- 2013: silver data truncated to 0 decimal places.

To estimate the silver content by geologic domain, the assay data from samples selected for metallurgical testing beginning in late 2012 were analyzed. In total, 156 samples from six drill holes (i.e., WS12-203, WS12-204, WS12-208, WS12-210, WS12-213 and WS12-214) across the Wellgreen project were analyzed for silver content. As a conservative measure, the silver content was assumed to be zero where results were below the detection limit for silver (0.5 g/t); this resulted in a global average of 1.10 g/t silver. The average silver grades were then tabulated by geologic domain (Table 13.27).

Table 13.27: Average Silver Grade by Geological Domain

Geological Domain	# samples	Silver Grade (g/t)
Gabbro/Massive Sulphide	48	2.256
Clinopyroxenite/Pyroxenite	20	0.675
Peridotite/Dunite	88	0.595
Total/Average	**156**	**1.100**

Source: Eggert, Wellgreen Platinum, 2015

13.4.1 Silver Recoveries

There is very limited data available regarding silver recoveries to concentrate; historical information provided only silver grades in concentrate without head grades. One data point from LCT-80 conducted by SGS in 1988 indicates that silver recovery was 81.7% based on a concentrate grade of 30.7 g/t and a calculated head grade of 3.22 g/t. Therefore, it is estimated that silver recoveries, to a PEA level, would be approximately 75-80% after detailed locked cycle tests were optimized.

13.4.2 Silver Concentrate Grades

A review of metallurgical test work indicated two data points for silver grades in concentrate produced from the Wellgreen project's drill core samples. The first data point is from metallurgical testing conducted by SGS in 1988. Lakefield conducted testing on lower grade samples that reflected anticipated material from the open pit operation (Table 13.28).

Table 13.28: SGS Flotation Test Results for Lower Grade Mineralized Material

Test No.	Product	Weight	Grade				Distribution			
			Cu	Ni	Pt	Pd	Cu	Ni	Pt	Pd
		(%)	(%)	(%)	(g/t)	(g/t)	(%)	(%)	(%)	(%)
54	Bulk 3rd Cleaner Concentrate	4.38	12.100	6.990	8.720	7.120	93.7	74.1	51.0	63.2
	Bulk 1st Cleaner Concentrate	6.27	8.610	5.230	7.200	5.410	95.4	79.3	60.3	68.7
	Bulk Rougher Concentrate	11.63	4.700	3.000	4.310	3.110	96.6	84.2	67.0	73.2
	Bulk Rougher Tail	88.37	0.022	0.074	0.280	0.150	3.4	15.8	33.0	26.8
	Head (Calculated)	100.00	0.570	0.410	0.750	0.490	100.0	100.0	100.0	100.0

Source: SGS Lakefield, 1988

The elemental analysis of the typical cleaner concentrate product from the Lakefield laboratory in 1988 is shown in Table 13.29. The concentrate contained 1.04 oz/t or 32.3 g/t of silver in a concentrate grading 5.4% Ni and 11.5% Cu.

Table 13.29: SGS Cleaner Concentrate Analysis

Element	Measurement	Content
Copper	%	11.5
Nickel	%	5.4
Cobalt	%	n/a
Gold	oz/t	0.091
Silver	oz/t	1.04
Platinum	oz/t	0.2
Palladium	oz/t	0.18
Rhodium	oz/t	0.005
Iron	%	36.6
Sulphur	%	29
Lead	%	0.02
Zinc	%	0.59
Arsenic	%	0.43
Antimony	%	0.004
Silica	%	8.54
Alumina	%	1.11
Lime	%	1.17
Magnesium oxide	%	3.13

Source: SGS Lakefield, 1988

Another data point for silver is from metallurgical testing by G&T Metallurgical Services documented in a May 5, 2011 report entitled “Metallurgical Assessment of the Wellgreen Deposit, Yukon Territory, Canada – KM2833”. G&T’s work was completed on a composite sample classified as peridotite; it contained metal grades (Table 13.30) that are comparable to the current Mineral Resource estimate for the Property.

Table 13.30: Chemical Composition of Peridotite Composite 1

Element	Copper	Nickel	Iron	Sulphur	Platinum	Palladium	Carbon
Symbol	**Cu**	**Ni**	**Fe**	**S**	**Pt**	**Pd**	**C**
Units	**%**	**%**	**%**	**%**	**g/t**	**g/t**	**%**
Peridotite Composite 1	0.29	0.26	10.3	1.80	0.28	0.25	0.17

Source: G&T, 2011

Six batch flotation tests were conducted by G&T on the composite sample. The concentrate from the final cleaner test was analyzed to determine the chemical constituents of the concentrate, and the results are shown in Table 13.31. The contained silver was 11.6 g/t, which is likely less than what could be achieved using locked cycle tests or in production. It is noted that this testing was on material classified as peridotite.

Table 13.31: Concentrate Quality

Element	Assay	Units	Test 6 Concentrate
Antimony	Sb	%	0.011
Arsenic	As	ppm	71
Bismuth	Bi	ppm	<20
Cadmium	Cd	ppm	18
Carbon	C	%	0.02
Cobalt	Co	ppm	4,034
Copper	Cu	%	7.10
Fluorine	F	ppm	82
Gold	Au	g/t	3.09
Iron	Fe	%	27.6
Lead	Pb	ppm	156
Mercury	Hg	ppm	<1
Molybdenum	Mo	%	0.005
Nickel	Ni	%*	6.40
Palladium	Pd	g/t	5.40
Platinum	Pt	g/t	2.72
Selenium	Se	ppm	73
Sulphur	S	%	21.3
Silver	Ag	g/t	11.6
Zinc	Zn	ppm	890
Aluminum Oxide	Al_2O_3	%	0.98
Calcium Oxide	CaO	%	1.06
Magnesium Oxide	MgO	%	10.9
Manganese Oxide	MnO	%	0.089
Phosphorus Pentoxide	P_2O_3	%	0.048
Silica	SiO_2	%	15.0

Source: G&T, 2011

Finally, ICP analysis of locked cycle testing conducted by SGS in February 2014 indicated a silver content of 15 g/t in the final concentrate (Table 13.32).

Table 13.32: SGS Locked Cycle Test Results

Lower Ultramafic Complex (LUC)		Au	Pt	Pd	Ru	Rh	Ir	Ag	Co	Mg0
	Weight	g/t	g/t	g/t	g/t	g/t	g/t	g/t	%	%
LCT1 Cu 3rd Cl Conc D-F	0.90	3.73	3.34	5.34	0.21	0.18	0.29	48	0.185	3.28
LCT1 Mag 2nd Cl Conc D-F	2.90	0.41	3.17	1.99	0.34	0.23	0.43	4	0.179	3.96
LCT1 Ni 3rd Cl Conc D-F	1.10	1.37	4.78	4.21	0.37	0.24	0.46	17	0.502	3.3
LCT1 Mag Ro Tail D-F	23.40	0.05	0.48	0.27	0.05	0.03	0.05			
LCT1 Ni Ro Tail D-F	69.60	0.02	0.18	0.08	< 0.05	< 0.02	< 0.04			
Final Concentrate	4.90	1.17	3.56	3.10				15		

Source: SGS, 2014

Therefore, the best silver grade data by geologic domain is from the 2012 drill core samples used for metallurgical testing. Based on these samples, it is predicted that continued optimization during the locked cycle tests for specific metallurgical domains followed by batch test programs could produce the following estimated average silver grades:

- 2.256 g/t for gabbro;
- 0.675 g/t for clinopyroxenite/pyroxenite; and
- 0.595 g/t for peridotite.

Although there is minimal data for silver recoveries to concentrate, it is predicted that recovery would be approximately 80% for Gabbro/Massive Sulphide, 75% for Clinopyroxenite/Pyroxenite and 65% for Peridotite/Dunite. It is anticipated that this will result in a blended recovery of approximately 77% in the 2015 PEA LOMP.

The combination of silver grades and assumed recoveries is expected to generate silver in concentrate grades that range between 15 g/t and 24 g/t; this is within the range of concentrates demonstrated in past metallurgical testing.

No economic value was attributed to the silver in this PEA. Additional testing will be conducted in the next round of studies to better quantify the grade and economic contribution of silver to the project.

13.5 Exotic Metals

Exotic metals are not considered in the 2015 PEA economic modelling since they are not part of the mineral resource estimate.

13.5.1 Review of Historic Exotic Metals Assays

HudBay reported exotic metals (i.e., rhodium, ruthenium, iridium and osmium) in the mine concentrate production sent to the Sumitomo smelter. The head grades from HudBay's historic production at the Property showed a 48% increase in total PGM content (g/t) versus platinum and palladium only. Additional work by Dr. Larry Hulbert and others has also shown that the Wellgreen deposit has significant enrichment of exotic PGMs. However, historically, only select drill core samples were assayed for exotic PGMs because the cost of assaying for the additional elements significantly increases total assay costs. Despite the lack of systematic sampling for exotic PGMs throughout the deposit, significant sampling has been done to estimate grades for exotic PGMs at the Property (Table 13.33).

Table 13.33: Summary of Test Results with Exotic PGMs

	Ni (%)	Cu (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Ru (g/t)	Ir (g/t)	Os (g/t)	4E (g/t)	Increase to Pt+Pd
HudBay Mining	2.23	1.39	1.3	0.92	0.40	0.42	0.25	0.20	1.27	48%
1988 Lakefield Composite	0.65	0.91	1.05	0.60	0.13	0.07	0.07	0.14	0.41	25%
Dr. Hulbert Study	0.83	0.89	0.81	0.67	0.132	0.222	0.107	0.158	0.618	42%
Dr. Hulbert Study 0.2% to 0.6% Ni	0.31	0.51	0.45	0.38	0.034	0.043	0.026	0.041	0.143	17%
Hole 188 samples	0.36	0.53	0.17	0.25	0.021	0.045	0.017	0.021	0.106	25%
Drill database with full 6E assays	0.49	0.59	0.81	0.54	0.043	0.062	0.069	0.041	0.215	16%
Drill database 0.2% to 0.6% Ni	0.30	0.22	0.34	0.32	0.011	0.038	0.031	0.022	0.102	16%
Exotic PGM composition for 0.2% to 0.6% Ni					26%	31%	21%	22%		

Source: Eggert, Wellgreen Platinum, 2015

A strong correlation exists between the grade of nickel and the content of exotic PGMs. As shown in Table 13.33, the exotic PGMs can increase the PGM content by more than 40% in higher grade material (i.e., nickel greater than 0.89%, based on data from HudBay and Dr. Hulbert), and between 16% and 25% in material that has an average nickel grade of between 0.28% to 0.36%. The 2015 PEA base case shows nickel grades in the first 16 years of production ranging from 0.25% to 0.36%, and, based on these grades, the data suggests that exotic PGMs could increase Pt+Pd grades by an estimated 15% or more.

On average, rhodium accounts for 26% of the exotic PGMs, ruthenium 31%, iridium 21% and osmium 22%. At current metal prices, approximately 60% of the value of exotic PGMs would be from rhodium, which is comparable in value to platinum.

13.5.2 Exotic PGM Grades by Geologic Domain

The Property drill database was analyzed to determine the exotic PGM-content by rock type; the results are shown in Table 13.34. As expected, exotic PGMs have the largest impact on total PGM grades in Gabbro/Massive Sulphide domain, where they increase the Pt+Pd grade by an average of 54%. Pt+Pd grades are increased by 15% in the Clinopyroxenite/Pyroxenite domain and 17% in the Peridotite/Dunite domain.

Table 13.34: SGS Locked Cycle Test Results (SGS, February 2014)

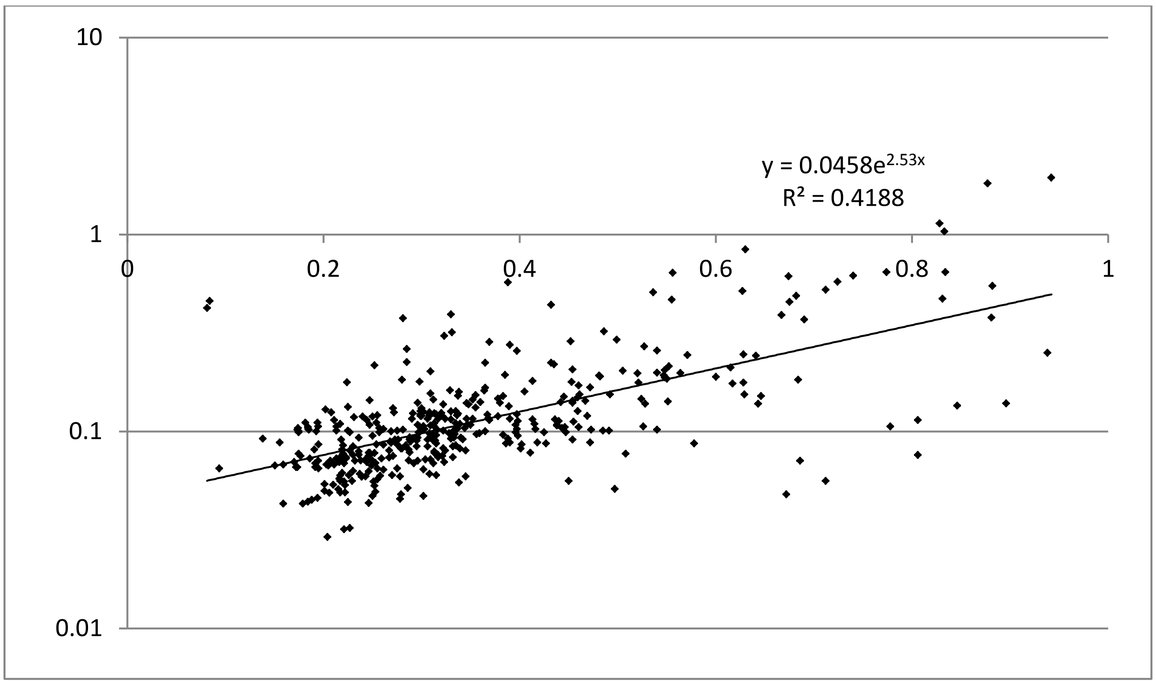
Domain	Ni (%)	Cu (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Ru (g/t)	Ir (g/t)	Os (g/t)	4E (g/t)	% increase
Gabbro	0.3092	0.3673	0.3876	0.2884	0.0185	0.1182	0.1431	0.0821	0.3619	54%
Clino/Pyroxenite	0.2396	0.3254	0.4534	0.3163	0.0093	0.0474	0.0342	0.0266	0.1176	15%
Peridotite	0.2862	0.1341	0.2469	0.2674	0.0105	0.0365	0.0220	0.0187	0.0875	17%

Source: SGS, 2014

13.5.3 Predicting Exotic PGM Levels Based on Other Metal Grades

The Property drill database was analyzed to determine the correlation between exotic PGMs and other metals (i.e., Ni, Cu, Pt and Pd). The strongest correlation was found between nickel and exotic PGMs (Figure 13.21), followed by palladium and exotic PGMs.

Figure 13.21: Exotic PGM Grades (4E in g/t) versus Nickel Grades (%), for Nickel <1%



Source: Eggert, Wellgreen Platinum, 2015

The drill database assay data for the exotic PGMs was then analyzed based on ranges of nickel grades. As shown in Table 13.35 and Figure 13.22, all of the exotic PGMs demonstrate relatively smooth exponential increases in grade as the nickel grade increases. This supports the correlation between exotic PGM grades and nickel grades. Plotting the combined exotic PGM (4E) grade against the nickel grade, and adding a trend line, results in the following predictive formula:

$4E = 0.0381e^{(3.0782 \times \text{Ni grade})}$.

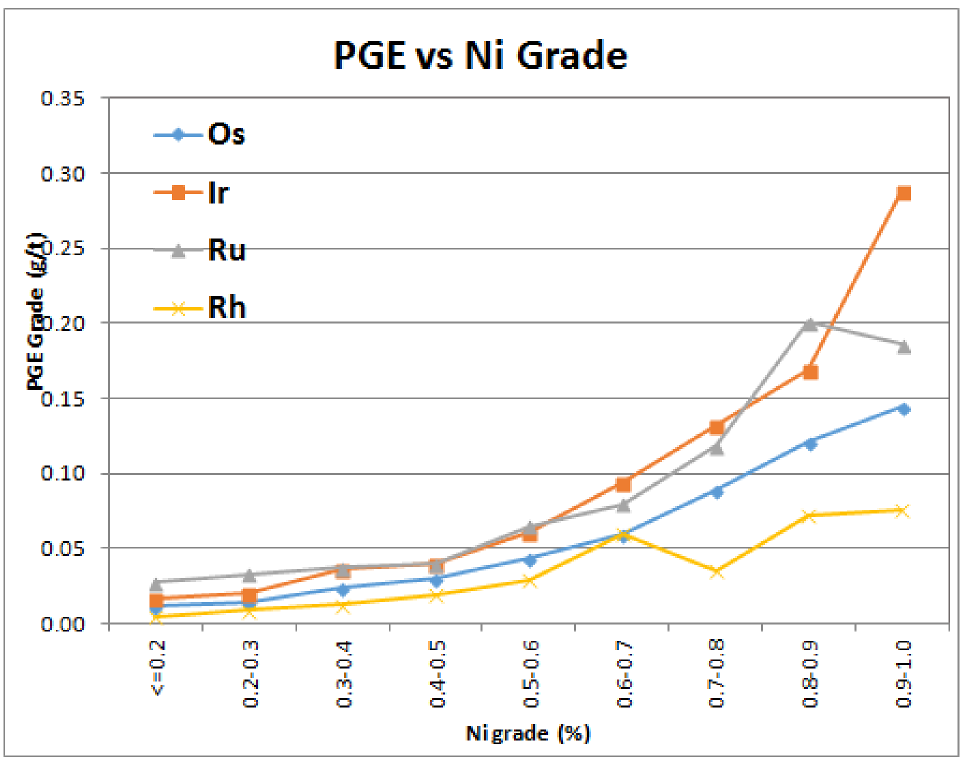
This formula predicts total exotic PGMs of 0.098 g/t when the nickel grade is 0.30%. Applied to the 2015 PEA model, it suggests an increase in the grade of total PGMs by approximately 15%, compared to Pt+Pd alone.

Table 13.35: Drill Database Exotic PGM Grades by Nickel Grade Ranges

Ni grade range (%)	Ni (%)	Cu (%)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Ru (g/t)	Ir (g/t)	Os (g/t)	4E (g/t)	% increase to PGMs
<=0.2	0.095	0.146	0.189	0.117	0.004	0.027	0.016	0.011	0.059	19%
0.2-0.3	0.255	0.185	0.288	0.250	0.009	0.033	0.020	0.013	0.075	14%
0.3-0.4	0.336	0.222	0.345	0.345	0.012	0.037	0.036	0.024	0.110	16%
0.4-0.5	0.442	0.382	0.537	0.547	0.019	0.040	0.040	0.030	0.128	12%
0.5-0.6	0.545	0.643	0.800	0.704	0.029	0.065	0.061	0.044	0.199	13%
0.6-0.7	0.653	0.822	1.048	0.871	0.060	0.080	0.094	0.060	0.293	15%
0.7-0.8	0.737	1.084	1.397	1.016	0.035	0.118	0.132	0.089	0.375	16%
0.8-0.9	0.852	0.977	1.029	0.911	0.072	0.201	0.169	0.121	0.563	29%
0.9-1.0	0.946	1.295	1.216	0.888	0.075	0.186	0.289	0.145	0.695	33%
>1.0	2.129	1.485	1.608	1.321	0.127	0.277	0.401	0.213	1.019	35%

Source: Eggert, Wellgreen Platinum, 2015

Figure 13.22: Drill Database Exotic PGM Grades by Nickel Grade Range



Source: Eggert, Wellgreen Platinum, 2015

13.5.4 Exotic PGM Recoveries and Concentrate Grades

There has been limited testing of exotic PGMs recoveries to concentrate. In early 2014, XPS conducted Test 13 on peridotite; it indicated that the bulk third cleaner concentrate contained 0.27 ppm rhodium, 0.43 ppm ruthenium, 0.26 ppm iridium and 0.11 ppm osmium (total = 1.07 g/t). Results are shown in Table 13.36. Head grades were not given for the exotic PGMs, but the predicted 4E content was 0.105 g/t based on the 0.33% nickel grade. Recoveries would need to be approximately 22% to achieve 1.07 g/t in the total cleaner concentrate; this is very close to the platinum recovery in this peridotite sample of 21.6%.

Table 13.36: Summary of Test 13 Results (XPS 2014)

Test 13	Mass (%)	Grade						Distribution					
		Cu (%)	Ni (%)	S (%)	Au (g/t)	Pd (g/t)	Pt (g/t)	Cu (%)	Ni (%)	S (%)	Au (%)	Pd (%)	Pt (%)
Bulk Clnr Con	2.0	5.77	9.40	23.1	1.380	4.680	3.380	60.2	55.9	32.6	45.5	31.9	18.6
Mag Clnr Con	0.2	2.12	3.41	12.1	1.167	5.346	4.909	2.5	2.3	1.9	4.3	4.1	3.0
Total Clnr Con	2.2	5.40	8.80	22.0	1.359	4.747	3.534	62.7	58.2	34.6	49.8	36.0	21.6
Po Con	1.7	0.22	0.27	8.0	0.075	0.672	1.394	1.9	1.3	9.4	2.0	3.8	6.4
1st Clnr Tail	6.1	0.22	0.44	2.7	0.073	0.775	0.988	7.0	8.0	11.5	7.4	16.1	16.6
2nd Clnr Tail	0.7	0.67	1.60	5.0	0.154	1.867	1.859	2.3	3.1	2.3	1.7	4.1	3.3
3rd Clnr Tail	0.2	1.46	3.41	9.2	0.332	3.896	3.407	1.6	2.1	1.4	1.2	2.8	2.0
Mag Clnr Tail	12.5	0.13	0.12	2.0	0.043	0.491	0.753	8.5	4.5	17.5	8.9	21.0	26.0
Final Ro Tail	76.6	0.04	0.10	0.4	0.023	0.062	0.114	16.0	22.8	23.3	29.1	16.2	24.1

Source: XPS, 2014

Recovery data from LCT-1 by SGS in 2012 indicated exotic PGM recoveries of 23.1% for Rh, 17.3% for Ru, 20.5% for Ir and 22.4% for Os (average of 20.8%). These recoveries are similar to the combined platinum recovery of 24.6% (Table 13.37).

Table 13.37: LCT-1 Results from 2012 Testing by SGS

LCT-1	Weight	Assay Grades													% Distribution												
Product	%	Cu	Ni	S	Pt	Pd	Au	Rh	Ru	Ir	Os	Co	Fe	MgO	Cu	Ni	S	Pt	Pd	Au	Rh	Ru	Ir	Os	Co	Fe	MgO
		%	%	%	g/t	g/t	g/t	ppb	ppb	ppb	ppb	%	%	%													
Cu Conc	1.00	23.2	0.88	28.3	2.16	4.83	1.44	177	192	150	140	0.045	28.5	2.83	68.2	1.8	9.5	4.9	11.0	31.2	4.5	2.7	3.4	4.1	1.5	2.3	0.1
Cu Rougher Tail (Ni Conc.)	1.78	2.55	14.4	26.7	3.34	10.9	0.32	323	455	329	270	0.870	32.5	4.55	13.4	53.9	16.1	13.5	44.6	12.4	14.8	11.6	13.4	14.2	52.9	4.7	0.4
Ni 3rd Clnr Conc	0.48	3.24	7.03	27.2	5.72	5.90	0.43	311	430	338	290	0.360	41.8	5.04	4.6	7.0	4.4	6.2	6.5	4.5	3.8	2.9	3.7	4.1	5.9	1.6	0.1
Ni 1st Clnr Tail	15.0	0.13	0.48	7.48	1.02	0.61	0.05	90	112	106	72	0.023	18.5	20.4	5.8	15.0	38.0	34.7	21.0	16.3	34.8	24.1	36.4	31.9	11.8	22.5	13.5
Ni Scav Tail	81.7	0.03	0.13	1.15	0.22	0.09	0.02	<20	<50	23	19	<0.01	10.4	23.8	8.0	22.2	31.9	40.7	16.9	35.6	42.0	58.6	43.0	45.7	27.9	68.9	85.9
Total Ni Conc.	2.26	2.69	12.9	26.8	3.84	9.84	0.34	320	450	331	274	0.762	34.5	4.66	18.0	60.9	20.5	19.7	51.1	16.9	18.6	14.6	17.1	18.3	58.8	6.3	0.5
Head (calc.)		0.34	0.48	2.95	0.44	0.44	0.05	39	70	44	33.9	0.029	12.3	22.7													

Source: SGS, 2012

13.6 Concentrate

Bulk concentrate quality is determined by the following process:

1. Based on the production schedule and the linear regression analysis associated with the three geological domains, calculate the amount of nickel and copper recovered;
2. Based on a nickel in concentrate grade of 6%, calculate the amount of bulk concentrate; and
3. Using the historical linear regression analysis, calculate the projected copper in concentrate grade where the nickel grade is 6% (Table 13.38);

Table 13.38: Estimated Concentrate Grades at a 6% Normalized Ni Grade

Geological Domain	Ni %	Cu %
Gabbro/Massive Sulphide	6.00	7.89
Clinopyroxenite/Pyroxenite	6.00	7.79
Peridotite/Dunite	6.00	6.50

Source: Eggert, 2014

4. Calculate the amount of bulk concentrate required to contain the recovered copper metal at the projected copper in concentrate grade;
5. Average the two bulk concentrate calculations to determine the final mass of bulk concentrate as well as the final nickel and copper concentrate grades.

The bulk concentrate mass and quality estimates are shown in Table 13.39.

Table 13.39: Bulk Concentrate Mass and Quality

	Units	Years 1 - 10	Years 11 - 16	Years 17 - 24
Ni Recovered	t	32,998	37,600	27,178
Conc. normalized to Ni (6%)	dmt	549,964	626,669	452,962
Cu Recovered	t	22,635	27,947	11,443
Predicted Cu grade in 6% Ni Con	%	7.77	7.79	7.45
Conc. normalized to Cu	dmt	291,137	358,528	153,068
Final PEA Bulk Con	dmt	420,551	492,598	303,015
Ni Grade	%	7.75	7.69	8.88
Cu Grade	%	5.51	5.59	3.86
Co Grade	%	0.36	0.37	0.45
Bulk Con total base metal grade	%	13.62	13.65	13.20
Bulk Con PGM + Au grade	g/t	14.416	15.640	12.737

Source: Eggert, 2014

13.7 Comments

The wide variation in work indices between domains indicates that it might be possible, when blending mineralized material from different domains, to achieve the higher recoveries tested for the lower grade domains. The lower grade domains are easier to grind and will, therefore, be finer than the higher grade domains. Testing the lower grade domains has shown that the finer the grind, the better the recovery; although this hypothesis needs to be confirmed with additional work.

Most of the historic testing was completed on the higher grade material from the Gabbro/Massive Sulphide and Clinopyroxenite/Pyroxenite domains. For this reason, there is increased confidence in these results; these higher grade materials are anticipated to represent 99% of the processed material in the first 16 years of the mine plan. The Peridotite/Dunite domain material is anticipated to be processed later in the mine plan when it will become approximately 24% of the processed material or 10% over the LOM base case. The tests performed by SGS and XPS in 2014 focused on peridotite material, and additional testing is required to further refine recovery of this lower grade domain. The recent peridotite results were incorporated into the plant design parameters.

14 Mineral Resource Estimate

14.1 Summary

This mineral resource estimate is an update to those previously prepared for the Property (Carter et al, 2012). The mineral resource estimate was prepared using GEOVIA Surpac© v6.5 software by Ronald G. Simpson P. Geo, a Qualified Person of GeoSim. Table 14.1 presents the mineral resource estimate for the Property at a base case cut-off grade of 0.57 g/t Pt Equivalent or 0.15% Ni Equivalent.

Table 14.1: Mineral Resource at a 0.57 g/t PtEq or 0.15% NiEq cut-off

Category	Tonnes 000s	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	3E g/t	Ni Eq. %	Pt Eq. g/t
Measured	92,293	0.260	0.155	0.015	0.252	0.246	0.052	0.550	0.449	1.713
Indicated	237,276	0.261	0.135	0.015	0.231	0.238	0.042	0.511	0.434	1.656
Total M&I	329,569	0.261	0.141	0.015	0.237	0.240	0.045	0.522	0.438	1.672
Inferred	846,389	0.237	0.139	0.015	0.234	0.226	0.047	0.507	0.412	1.571

Source: GeoSim, 2014

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. with an effective date of July 23, 2014.
2. Measured mineral resources are drilled on approximate 50 x 50 metre drill spacing and confined to clinopyroxenite and peridotite/dunite domains. Indicated mineral resources are drilled on approximate 100 x 100 metre drill spacing except for the massive sulphide and gabbro domains which used 50 x 50 metre spacing.
3. Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries. Ni Eq% = Ni%+ Cu% x 3.00/8.35 + Co% x 13.00/8.35 + Pt [g/t]/31.103 x 1,500/8.35/22.046 + Pd [g/t]/31.103 x 750/8.35/22.046 + Au [g/t]/31.103 x 1,250/8.35/22.046. Pt Eq[g/t] = Ni Eq/100×2204.62×8.35 / 1,500×31.103
4. An optimized pit shell was generated using the following assumptions: metal prices in Note 3 above ; a 45 degree pit slope; assumed metallurgical recoveries of 70% for Ni, 90% for Cu, 64% for Co, 60% for Pt, 70% for Pd and 75% for Au; an exchange rate of CAN$1.00=USA$0.91; and mining costs of $2.00 per tonne, processing costs of $12.91 per tonne, and general & administrative charges of $1.10 per tonne (all expressed in Canadian dollars).
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
7. 3E = Pt + Pd + Au

In addition, Table 14.2 below shows the higher grade portion of the resource within the constrained pit at a 1.9 g/t Pt Eq. or 0.50% Ni Eq. cut-off.

Table 14.2: Mineral Resource at a 1.9 g/t PtEq or 0.50% NiEq cut-off

Category	Tonnes (000s)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	3E (g/t)	Ni Eq. (%)	Pt Eq. (g/t)
Measured	21,854	0.326	0.301	0.019	0.454	0.366	0.103	0.923	0.653	2.492
Indicated	50,264	0.334	0.286	0.019	0.455	0.373	0.090	0.919	0.653	2.493
Total M&I	72,117	0.332	0.291	0.019	0.455	0.371	0.094	0.920	0.653	2.493
Inferred	173,684	0.309	0.301	0.018	0.456	0.352	0.098	0.906	0.631	2.410

Source: GeoSim, 2014

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. with an effective date of July 23, 2014.
2. Measured mineral resources are drilled on approximate 50 x 50 metre drill spacing and confined to clinopyroxenite and peridotite/dunite domains. Indicated mineral resources are drilled on approximate 100 x 100 metre drill spacing except for the massive sulphide and gabbro domains which used 50 x 50 m spacing.
3. Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries. Ni Eq% = Ni%+ Cu% x 3.00/8.35 + Co% x 13.00/8.35 + Pt [g/t]/31.103 x 1,500/8.35/22.046 + Pd [g/t]/31.103 x 750/8.35/22.046 + Au [g/t]/31.103 x 1,250/8.35/22.046. Pt Eq[g/t] = Ni Eq/100×2204.62×8.35 / 1,500×31.103
4. An optimized pit shell was generated using the following assumptions: metal prices in Note 3 above ; a 45 degree pit slope; assumed metallurgical recoveries of 70% for Ni, 90% for Cu, 64% for Co, 60% for Pt, 70% for Pd and 75% for Au; an exchange rate of CAN$1.00=US$0.91; and mining costs of $2.00 per tonne, processing costs of $12.91 per tonne, and general & administrative charges of $1.10 per tonne (all expressed in Canadian dollars).
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
7. 3E = Pt + Pd + Au

14.2 Key Assumptions/Basis of Estimate

The sample database supplied for the Wellgreen project contains results from 776 surface and underground drill holes completed on the property since 1952 (Table 14.3). Four holes drilled in 2005 were not sampled and lay outside of the present resource limits. Data from the the 1996 RC program was not available at the time of the resource estimate and problems with the accuracy of collar locations were subsequently identified. Sixteen holes completed in 2013 for water monitoring are not included as the only ones located in the resource area were twins of previous holes and not assayed.

Table 14.3: Drilling Summary

Year	Operator	Surface Drilling		Underground Drilling		Combined Drilling	
		Holes	Metres	Holes	Metres	Holes	Metres
1952	Yukon Mining	18	1,981.64			18	1,981.64
1953	Yukon Mining	27	2,499.67	27	692.57	54	3,192.24
1954	Yukon Mining	2	192.63	159	3,939.65	161	4,132.28
1955	Hudson Yukon Mining			154	9,019.37	154	9,019.37
1956	Hudson Yukon Mining			38	1,903.70	38	1,903.70
1969	Hudson Yukon Mining	13	1,314.30			13	1,314.30
1971	Hudson Yukon Mining			80	2,482.83	80	2,482.83
1972	Hudson Yukon Mining			23	990.26	23	990.26
1987	All North / Galactic Resources	46	5,027.19			46	5,027.19
1988	All North / Chevron	37	6,049.66	34	5,571.20	71	11,620.86
2001	Northern Platinum	6	530.04			6	530.04
2006	Coronation Minerals	11	2,016.87			11	2,016.87
2007	Coronation Minerals			3	576.99	3	576.99
2008	Coronation Minerals	13	4,654.62			13	4,654.62
2009	Northern Platinum	10	2,051.75			10	2,051.75
2010	Northern Platinum	7	2,254.77			7	2,254.77
2011	Wellgreen Platinum	6	1,925.12			6	1,925.12
2012	Wellgreen Platinum	22	5,566.20	29	5,416.91	51	10,983.11
2013	Wellgreen Platinum	11	2,240.36			11	2,240.36
Totals		**229**	**38,304.82**	**547**	**30,593.48**	**776**	**68,898.30**

Source: GeoSim, Wellgreen Platinum, 2014

Prior to 2006, drill core was selectively sampled in areas considered to have economic potential based on visual logging. Wellgreen Platinum assayed non-sampled intervals from the 1987-1988 drill programs in 2013 and re-assayed intervals that had been previously analyzed.

14.3 Geological Models

Lithologic wireframe models were created by Wellgreen Platinum geologic staff based on sectional geology interpretations (Figure 14.1). Model blocks that were within the respective wireframes were assigned integer codes as presented in Table 14.4.

For the resource modeling, the dunite, peridotite, pyroxenite and clinopyroxenite were treated as a single domain for geostatistics with the gabbro/massive sulphide material confined to a separate domain. Historically, material that was not massive sulphide or gabbro was classified under the field term ‘Peridotite’. The sub-domains were created subsequent to grade estimation based largely on grade distribution and estimated ultramafic content which include clinopyroxenite to pyroxenite to peridotite to dunite. The dunite material had 0.1% nickel deducted from the grade as an estimate of potential nickel silicate content which eliminated nearly all of this material from the resource estimate.

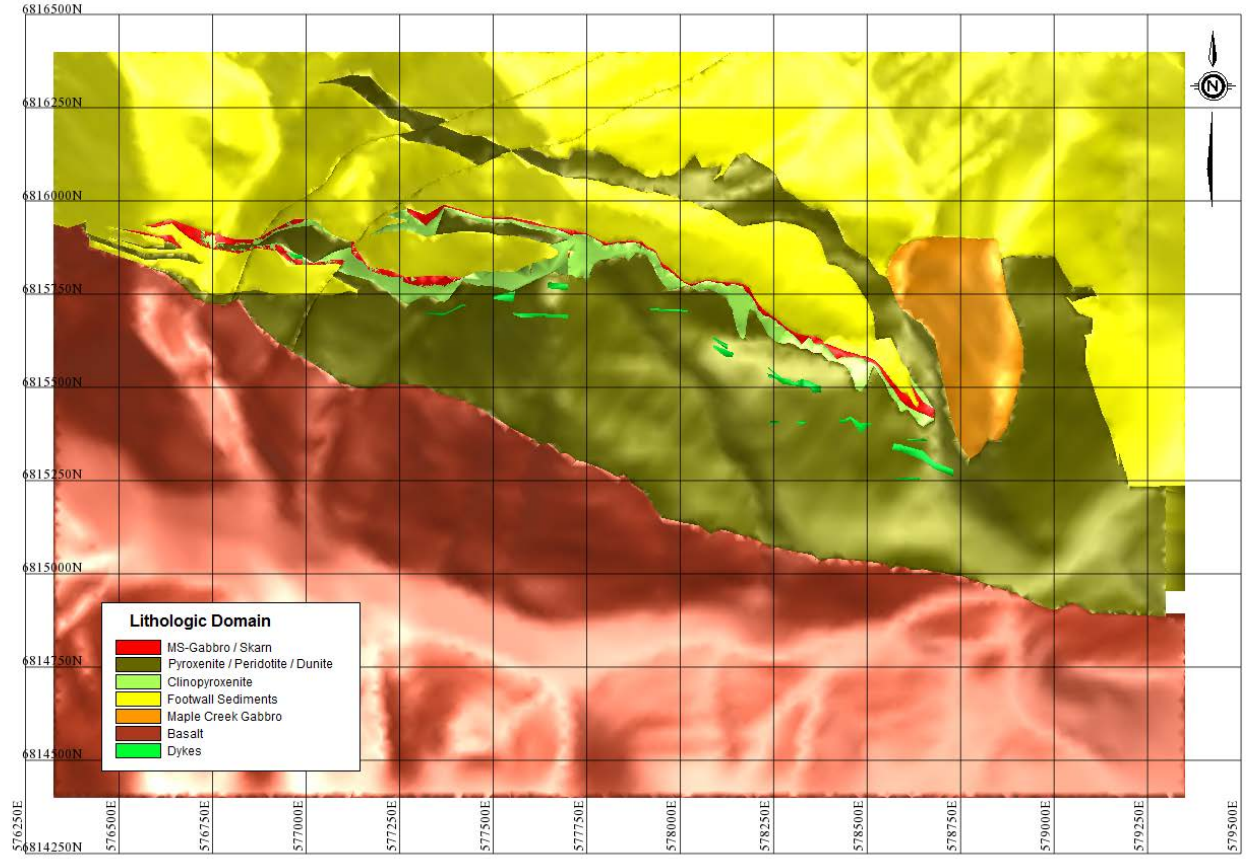
Grade estimation was confined to the Peridotite complex (including dunite, peridotite, pyroxenite, and clinopyroxenite) and the MS-Gabbro domains. The extent of the MS-Gabbro (MS-Gb) domain along the Peridotite and footwall sediment contacts is illustrated in Figure 14.1 and Figure 14.2.

Table 14.4: Lithologic Domain Coding

Lithologic Domain	Model Code
Overburden	99
MS-Gabbro / Skarn	101
Far West Gabbro	110
Clinopyroxenite	150
Pyroxenite	201
Far West Peridotite	202
Peridotite	205
Dunite	251
Footwall Sediments	301
Mixed Gabbro/Sediments	302
Maple Creek Gabbro	401
Basalt	501
Dykes	701
Xenoliths	801
Undefined	601

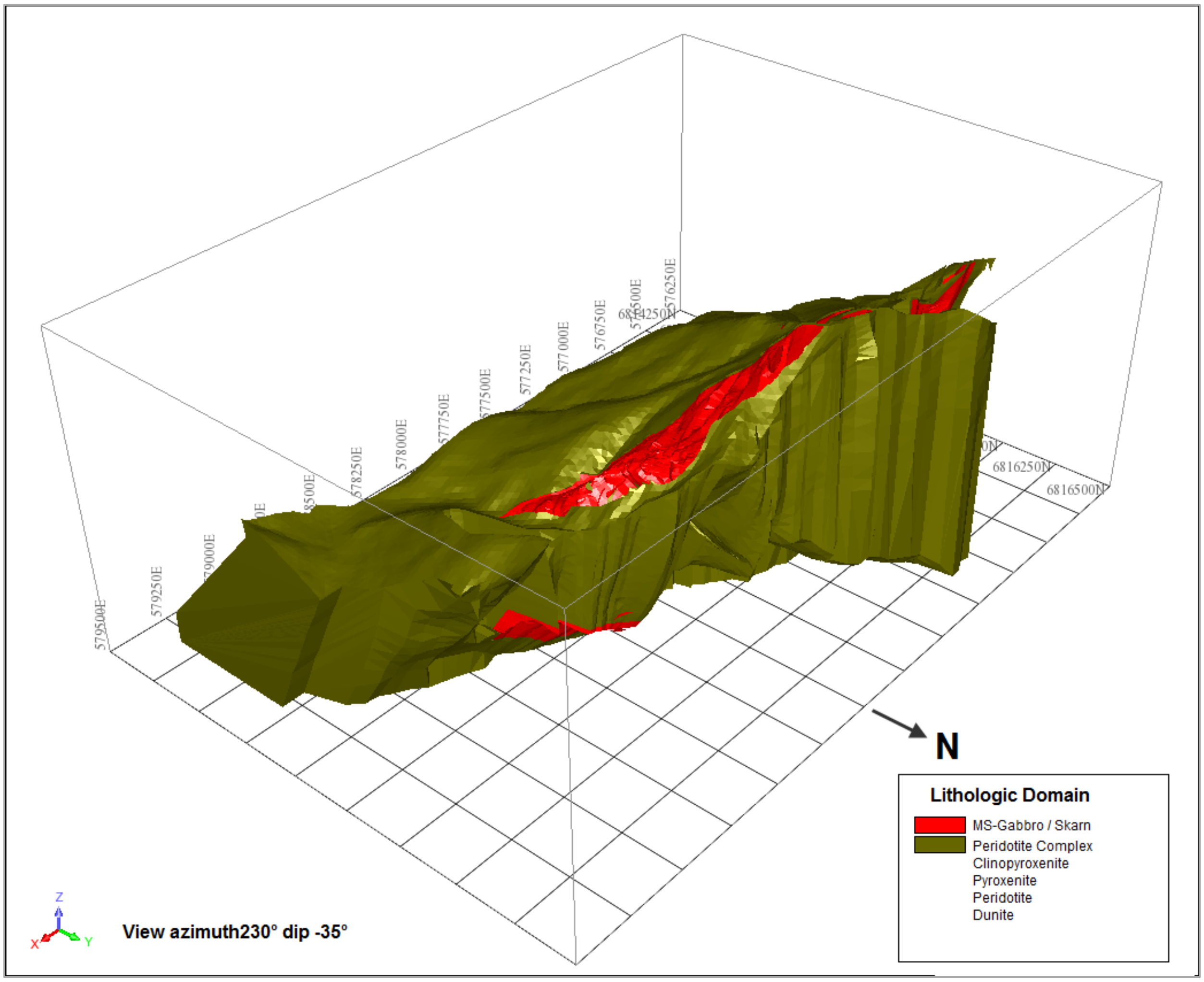
Source: Wellgreen Platinum, 2015

Figure 14.1: Plan View of Lithologic Domains



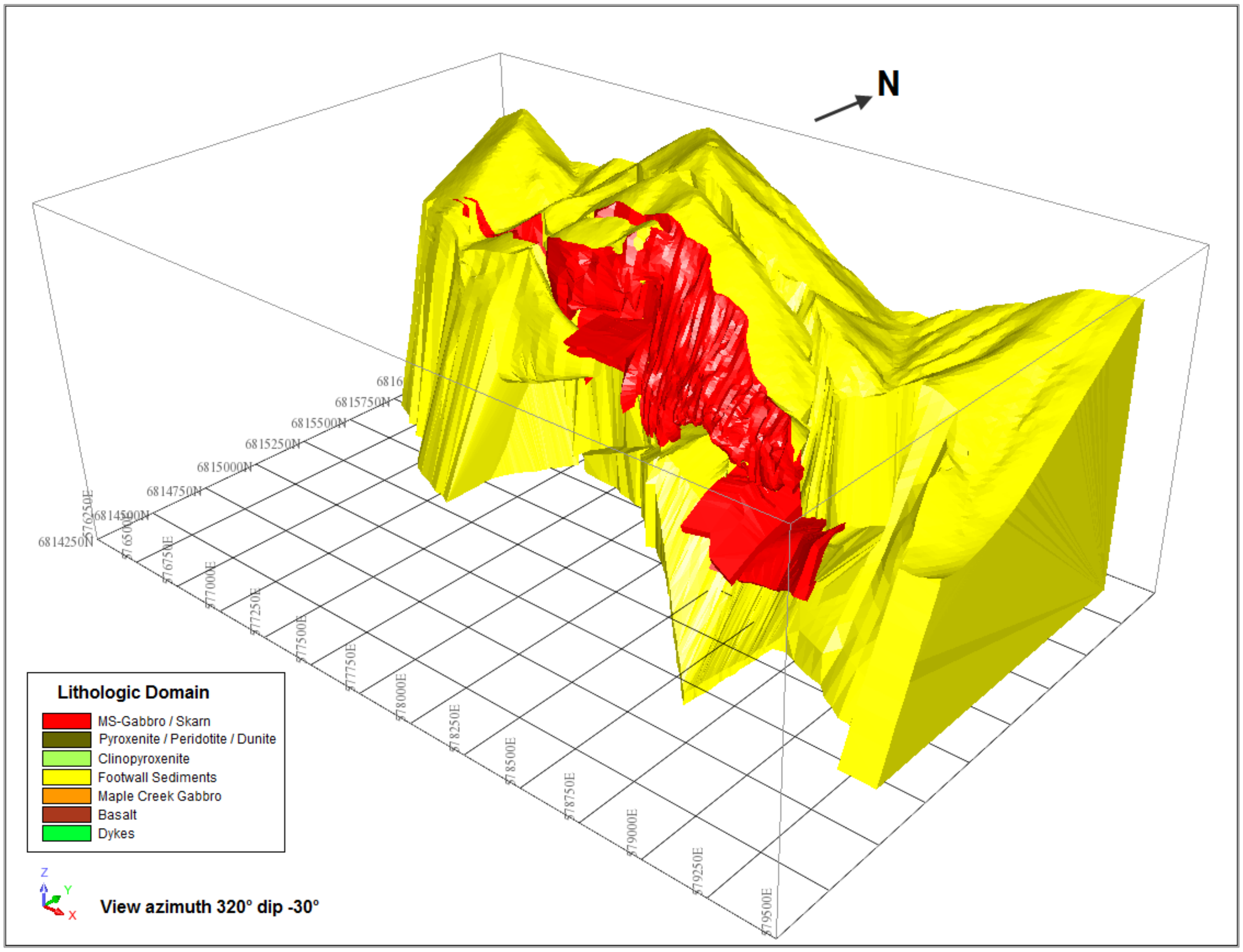
Source: GeoSim, Wellgreen Platinum, 2014

Figure 14.2: Perspective View showing MS-Gb along Peridotite Complex Contacts



Source: GeoSim, Wellgreen Platinum, 2014

Figure 14.3: Perspective View showing MS-Gb along Footwall Sediment Contacts



Source: GeoSim, Wellgreen Platinum, 2014

14.4 Exploratory Data Analysis

Nominal sample lengths varied from 1.2 to 3.05 m (4 to 10 ft) for the various drill programs. It was decided to composite all data to 3 m intervals prior to statistical analysis. Only 2.5% of the sampled intervals exceeded 3.05 m in length.

Composite statistics were generated within the MS-Gabbro and the combined Dunite/ Peridotite/Pyroxenite/Clinopyroxenite domains. The average grades using the pre-1987 legacy data were considerably higher than the post-1987 data due to selective sampling of higher grade intervals. In the MS-Gabbro domain, all average grades are significantly higher in the selective sampling data due to the presence of massive sulphide bodies which were tightly constrained. The statistics for the uncapped composites are presented in Table 14.5 to Table 14.8. Cumulative frequency distributions for Ni and Cu by domain are illustrated in Figure 14.5 to Figure 14.7.

Table 14.5: Composite Statistics Pre-1987 Data – Peridotite Complex Domain

	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
n	647	455	161	81	97	15
Min	0.010	0.045	0.002	0.103	0.069	0.137
Max	1.755	2.019	0.170	2.057	1.303	6.857
Median	0.276	0.158	0.015	0.358	0.343	0.343
Mean	**0.310**	**0.248**	**0.021**	**0.471**	**0.450**	**1.333**
Variance	0.044	0.052	0.000	0.109	0.074	5.144
Std Dev	0.209	0.229	0.021	0.330	0.271	2.268
CV	0.67	0.92	0.99	0.70	0.60	1.70

Source: GeoSim, 2014

Table 14.6: Composite Statistics 1987-2013 Data – Peridotite Complex Domain

	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
n	8358	8354	8212	8336	8357	8199
Min	0.001	0.000	0.001	0.001	0.001	0.001
Max	2.566	3.375	0.104	4.780	2.637	1.500
Median	0.255	0.100	0.015	0.181	0.219	0.025
Mean	**0.252**	**0.138**	**0.015**	**0.232**	**0.231**	**0.046**
Variance	0.010	0.023	0.000	0.041	0.019	0.005
Std Dev	0.099	0.150	0.004	0.203	0.137	0.070
CV	0.39	1.09	0.29	0.88	0.59	1.52

Source: GeoSim, 2014

Table 14.7: Composite Statistics pre-1987 Data- MS-Gabbro Domain

	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
n	839	829	443	497	493	170
Min	0.010	0.016	0.006	0.137	0.103	0.062
Max	5.732	4.440	0.670	9.600	10.971	5.143
Median	0.520	0.780	0.065	1.078	0.756	0.410
Mean	**1.018**	**0.912**	**0.079**	**1.374**	**1.096**	**0.620**
Variance	1.353	0.434	0.005	1.177	1.397	0.343
Std Dev	1.163	0.659	0.067	1.085	1.182	0.586
CV	1.14	0.72	0.85	0.79	1.08	0.95

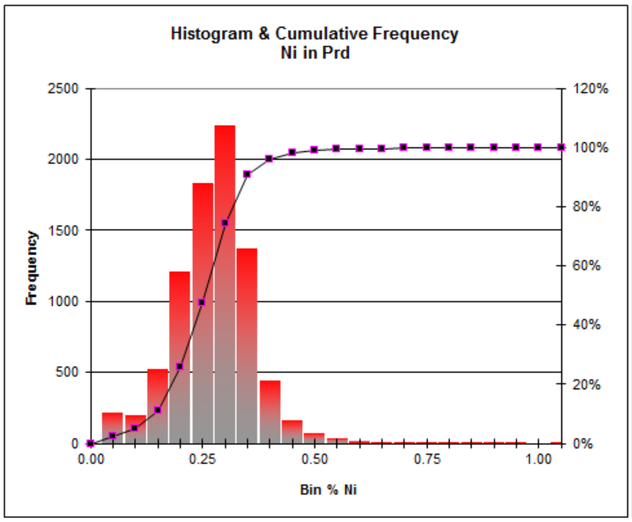
Source: GeoSim, 2014

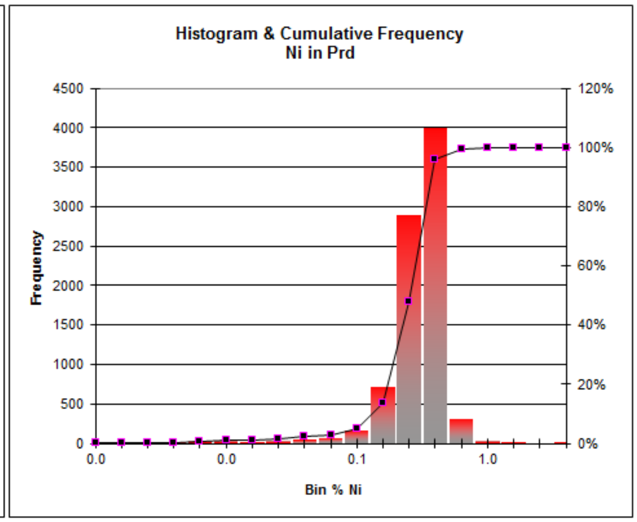
Table 14.8: Composite Statistics 1987-2013 Data - MS-Gabbro Domain

	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
n	1516	1516	1497	1499	1515	1461
Min	0.001	0.001	0.000	0.002	0.001	0.001
Max	5.147	4.195	0.275	4.155	3.578	3.748
Median	0.187	0.280	0.013	0.345	0.239	0.050
Mean	**0.272**	**0.381**	**0.017**	**0.454**	**0.300**	**0.098**
Variance	0.140	0.159	0.000	0.226	0.103	0.027
Std Dev	0.374	0.398	0.020	0.475	0.321	0.166
CV	1.37	1.05	1.16	1.05	1.07	1.69

Source: GeoSim, 2014

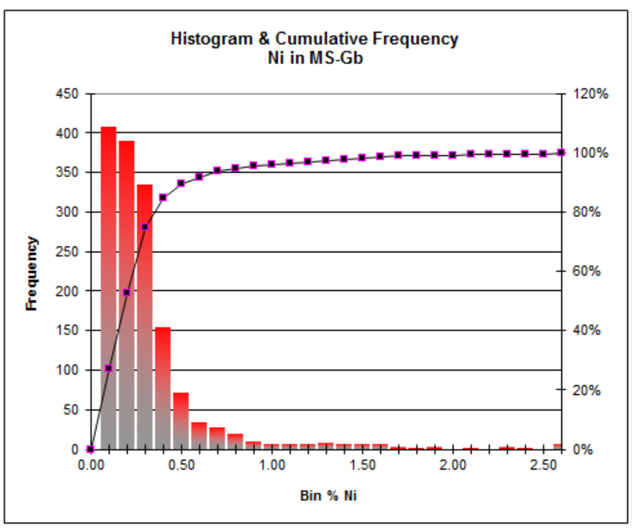
Figure 14.4: Frequency Distribution of Ni in Peridotite Complex (Prd)

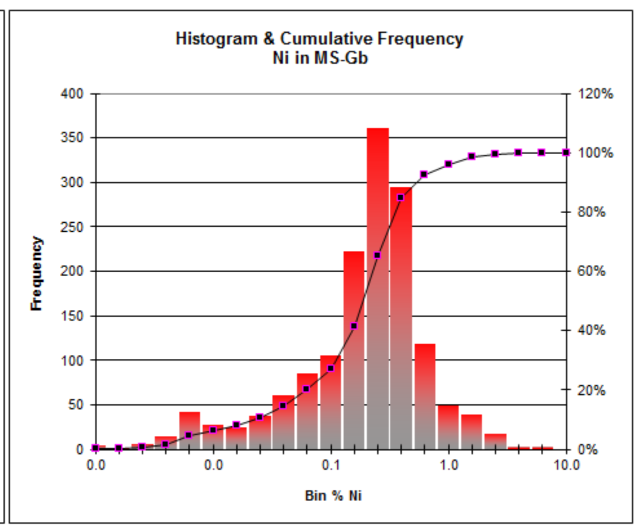




Source: GeoSim, 2014

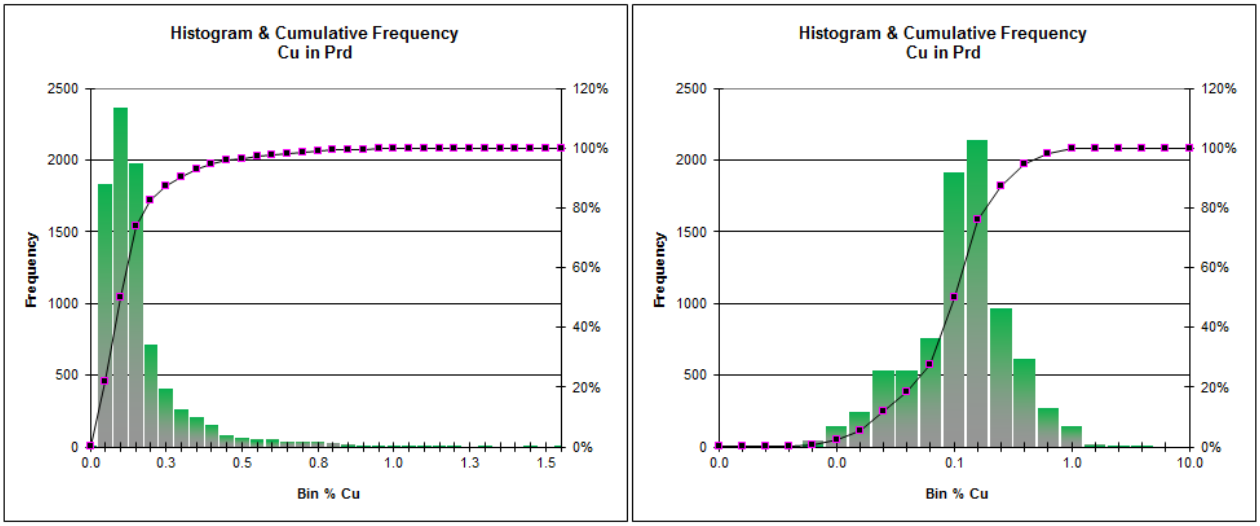
Figure 14.5: Frequency Distribution of Ni in MS-Gabbro





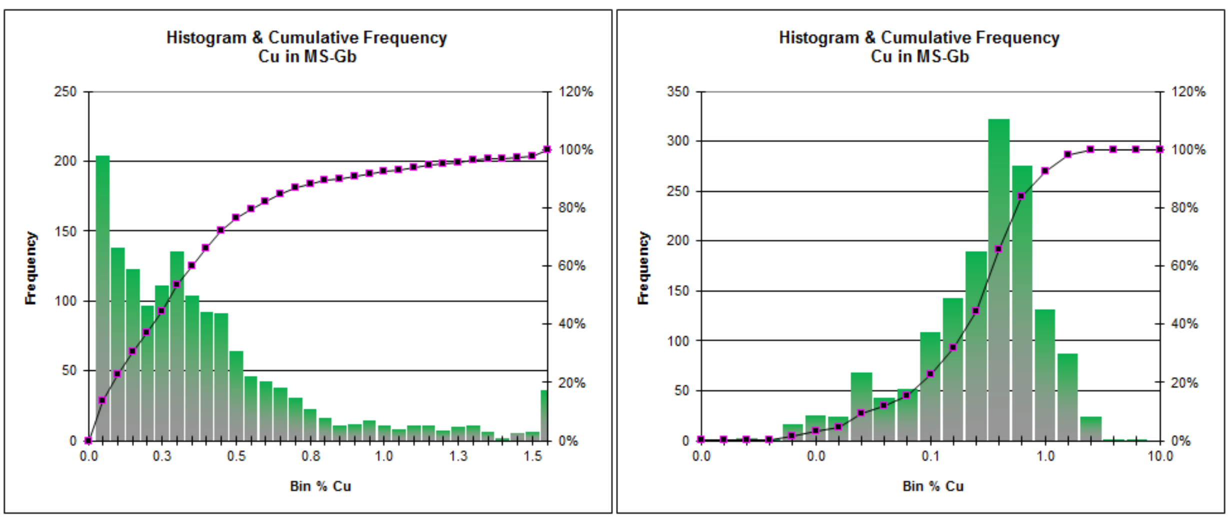
Source: GeoSim, 2014

Figure 14.6: Frequency Distribution of Cu in Peridotite Complex



Source: GeoSim 2014

Figure 14.7: Frequency Distribution of Cu in MS-Gabbro



Source: GeoSim, 2014

14.5 Density Assignment

The project database contains a total of 6,705 density measurements made on core samples from the 1987 through 2013 drill programs.

Model blocks were assigned the mean density value for the corresponding lithology as shown in Table 14.9

Table 14.9: Density Assignments

Lithologic Domain	Model Code	Density	No. Measurements
Overburden	99	2.10	0
MS-Gabbro / Skarn	101	3.06	622

Lithologic Domain	Model Code	Density	No. Measurements
Far West Gabbro	110	2.89	130
Clinopyroxenite	150	2.95	903
Pyroxenite	201	2.82	3243
Far West Peridotite	202	2.98	44
Low Grade Peridotite	205	2.75	385
Dunite	251	2.72	21
Footwall Sediments	301	2.76	1092
Mixed Gabbro/Sediments	302	2.76	0
Maple Creek Gabbro	401	2.80	0
Basalt	501	2.77	63
Dykes	701	3.03	81
Xenoliths	801	2.76	0
Undefined	601	2.75	0

Source: GeoSim, 2014

14.6 Grade Capping/Outlier Restrictions

Grade distribution in the composited data was examined to determine if grade capping or special treatment of high outliers was warranted. Cumulative log probability plots were examined for outlier populations separately in the Peridotite/Clinopyroxenite and MS-Gabbro domains. Only recent data from the post 1987 drilling was used in this study to eliminate the bias inherent in selective sampling from legacy data.

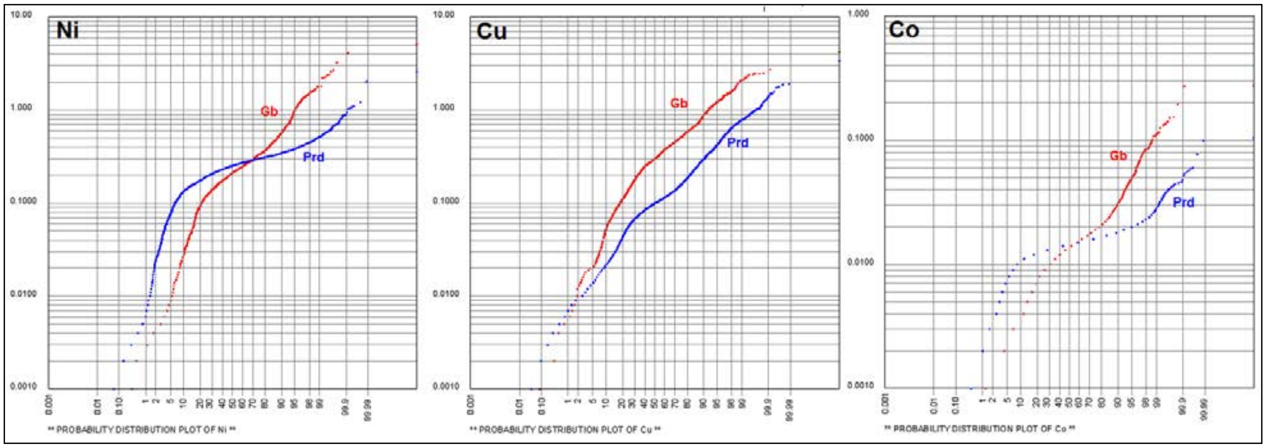
It was concluded that outliers above selected thresholds should be given a limited range of influence. The levels selected are shown in Table 14.10. Cumulative log probability plot (CPP) charts are illustrated in Figure 14.8 and Figure 14.9. There were very few outliers overall as indicated by the relative percent of composites above the threshold levels.

Table 14.10: Outlier Restrictions

Domain	MS-Gb		Prd/Clpx	
	Cap Grade	% of Composites above Cap	Cap Grade	% of Composites above cap
Ni %	2.0	0.91%	1.0	0.11%
Cu %	2.2	0.82%	1.5	0.07%
Co %	0.2	0.17%	0.045	0.13%
Pt g/t	2.1	1.33%	2.0	0.06%
Pd g/t	1.5	0.99%	1.2	0.04%
Au g/t	0.7	1.01%	0.55	0.21%

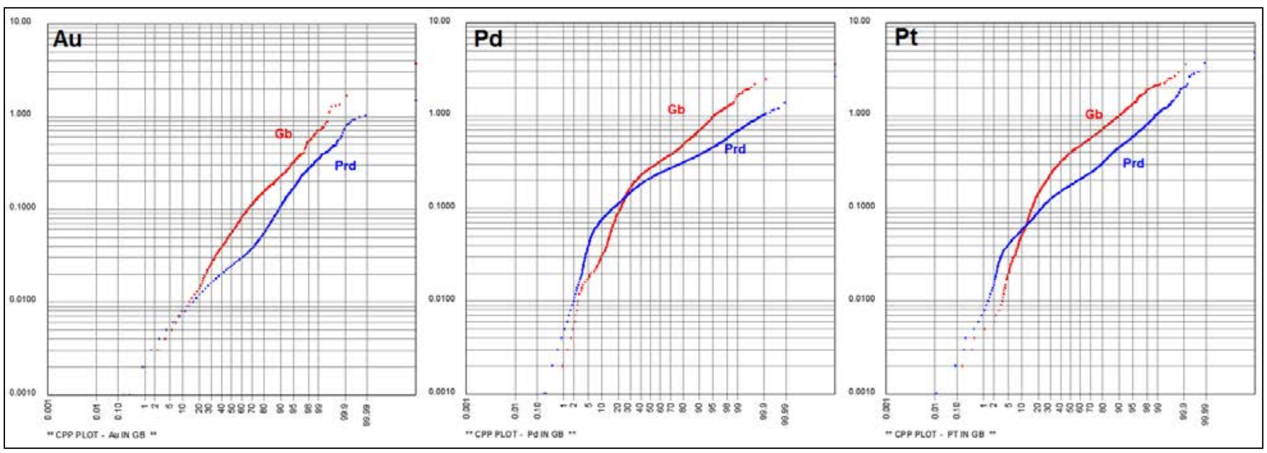
Source: GeoSim, 2014

Figure 14.8: Cumulative Log Probability Plots for Ni, Cu and Co



Source: GeoSim, 2014

Figure 14.9: Cumulative Log Probability Plots for Au, Pt and Pd



Source: GeoSim, 2014

14.7 Compositing

Downhole composites for Ni, Cu, Co, Pt, Pd, Au, and sulphur were created within the individual domains using the 'best fit' method. This procedure produces samples of variable length, but equal length within a contiguous drill hole zone, ensuring the composite length is as close as possible to the nominated composite length. In this case, the nominated length was set at 3 m.

Diluted composites from pre 1987 drilling were generated within the MS-Gabbro domain by assigning values for non-sampled intervals a 0 grade for Ni and Cu. Other elements were evaluated on a hole-by-hole basis to decide whether it was necessary to dilute missing or non-sampled data. If a hole contained some analytical data for other elements then non-sampled intervals were set to a 0 grade, otherwise they were ignored. All gold values were removed from the pre-1987 data as they were highly selective.

14.8 Variography

14.8.1 MS-Gabbro Domain

The MS-Gabbro domain tends to be a narrower zone along the footwall contact with the sediments containing pods of massive sulphides with high grades or as subhorizontal horizons within the cores of the thickest parts of the ultramafic package. As the orientation of the contact is not consistent it was not possible to model reliable directional variograms in all areas and it was decided to use the zone geometry to develop search ellipsoid orientations and anisotropy. Directional variograms in the plane of the most consistent portion of the zone (901) showed maximum ranges of approximately 100 m with no preferred orientation either along strike or down dip (Table 14.11).

Table 14.11: Variogram Models - MS-Gabbro Domain

Item	Axis	Azim	Plunge	co	c1	a1	c2	a2
Ni	major	196	-68	0.386	0.261	20	0.1	100
	semi-major	106	0	0.386	0.261	20	0.1	100
	minor	196	22	0.386	0.261	5	0.1	25
Cu	major	196	-68	0.13	0.383	16	0.189	100
	semi-major	106	0	0.13	0.383	16	0.189	100
	minor	196	22	0.13	0.383	10	0.189	25
Co	major	196	-68	0.12	0.241	16	0.16	90
	semi-major	106	0	0.12	0.241	16	0.16	90
	minor	196	22	0.12	0.241	10	0.16	22
Pt	major	196	-68	0.15	0.25	12	0.285	90
	semi-major	106	0	0.15	0.25	12	0.285	90
	minor	196	22	0.15	0.25	10	0.285	22
Pd	major	196	-68	0.15	0.32	15	0.234	90
	semi-major	106	0	0.15	0.32	15	0.234	90
	minor	196	22	0.15	0.32	10	0.234	22
Au	major	196	-68	0.282	0.332	15.5	0.121	100
	semi-major	106	0	0.282	0.332	15.5	0.121	100
	minor	196	22	0.282	0.332	12	0.121	25
S	major	196	-68	0.18	0.41	9.2	0.12	100
	semi-major	106	0	0.18	0.41	9.2	0.12	100
	minor	196	22	0.18	0.41	5	0.12	25

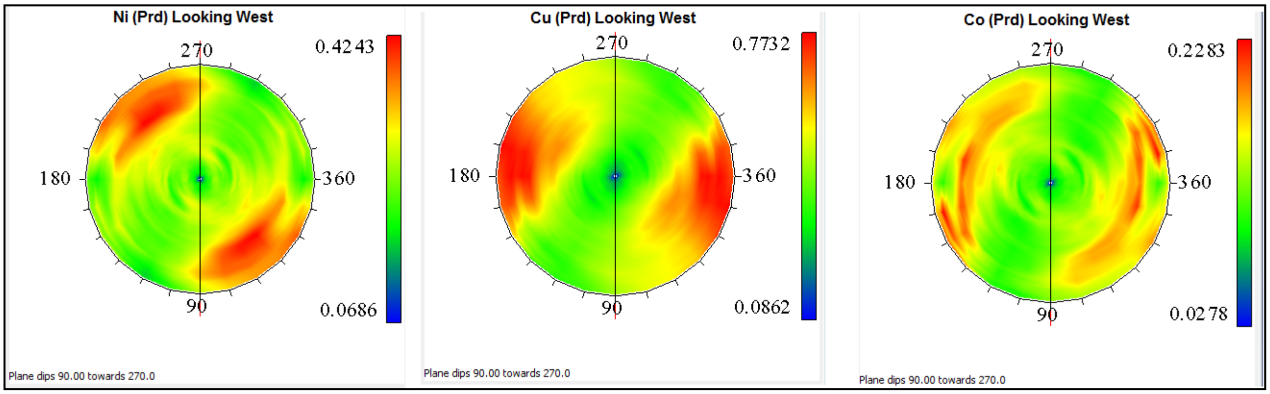
Source: GeoSim, 2014

14.8.2 Peridotite/Pyroxenite/Clinopyroxenite Domain

Directional pairwise relative variograms were modeled for the elements in the combined dunite/peridotite/pyroxenite/clinopyroxenite domains. Results revealed a moderate anisotropy with the major axis dipping to the south as shown in the variogram maps in Figure 14.10. Nested spherical structures were modeled for all elements (Table 14.12). Most elements had maximum ranges around 250 metres. The maximum range for sulphur exceeded 500 m.

The Far West peridotite domain did not have sufficient data for modeling variograms and search ellipsoids for grade interpolation were based on the zone geometry.

Figure 14.10: Variogram Maps - Peridotite/Pyroxenite/Clinopyroxenite Domain



Source: GeoSim, 2014

Table 14.12: Variogram Models – Peridotite/Pyroxenite/Clinopyroxenite Domain

Item	Axis	Azim	Plunge	co	c1	a1	c2	a2	c3	a3
Ni	major	100	0	0.05	0.068	26.8	0.04	250		
	semi-major	190	-61	0.05	0.068	25	0.04	208		
	minor	10	-29	0.05	0.068	25	0.04	167		
Cu	major	100	0	0.066	0.149	26.3	0.138	255		
	semi-major	190	-61	0.066	0.149	20	0.138	213		
	minor	10	-29	0.066	0.149	20	0.138	170		
Co	major	100	0	0.013	0.022	12	0.027	50	0.014	200
	semi-major	190	-61	0.013	0.022	8	0.034	30	0.014	167
	minor	10	-29	0.013	0.022	8	0.034	30	0.014	133
Pt	major	108	0	0.081	0.08	31	0.156	260		
	semi-major	198	-56	0.081	0.08	25	0.156	215		
	minor	18	-34	0.081	0.08	25	0.156	170		
Pd	major	116	0	0.081	0.085	30	0.109	250		
	semi-major	206	-51	0.081	0.085	20	0.109	200		
	minor	26	-39	0.081	0.085	20	0.109	167		
Au	major	116	0	0.12	0.123	25.8	0.071	97	0.125	260
	semi-major	206	-51	0.12	0.123	20	0.071	90	0.125	215
	minor	26	-39	0.12	0.123	20	0.071	85	0.125	175
S	major	116	0	0.15	0.108	18.4	0.045	141	0.284	520
	semi-major	206	-51	0.15	0.108	18.4	0.045	141	0.284	520
	minor	26	-39	0.15	0.108	18.4	0.045	120	0.284	350

Source: GeoSim, 2014

14.9 Estimation/Interpolation Methods

14.9.1 MS-Gabbro Domains

Twelve separate search domains were identified within the MS-Gabbro limits based primarily on the zone geometry. Soft boundaries were used where grades were contiguous (domains 901-905). Hard or semi-hard boundaries were used for isolated zones 906-912.

Grades were estimated in three passes using the Inverse Distance Cubed method (ID^3). For the 12 search domains within the MS-Gabbro, the first pass included uncapped legacy data with non-sampled intervals assigned a 0 value (diluted composites) and uncapped 1987-2013 composites. Gold data from pre-1987 holes was excluded from all grade estimation.

The second pass used only 1987-2013 capped composites. Blocks estimated in both the 1st and 2nd passes were compared and the final grade was the greater of the two estimates. It was assumed that if the first pass was lower in grade it was due to diluting the missing intervals to zero grade. The goal was to simulate the erratic nature of the massive sulphide pods along the footwall contact without overly smearing the high grades. Approximately 17% of all estimated blocks were included in the first pass.

The final pass used only capped 1987-2013 composites and the maximum search varied from 250 to 300 m in order to estimate most blocks within the various search domains.

Search parameters for the MS-Gabbro domains are shown in Table 14.13. The locations of the search domains are illustrated in Figure 14.11.

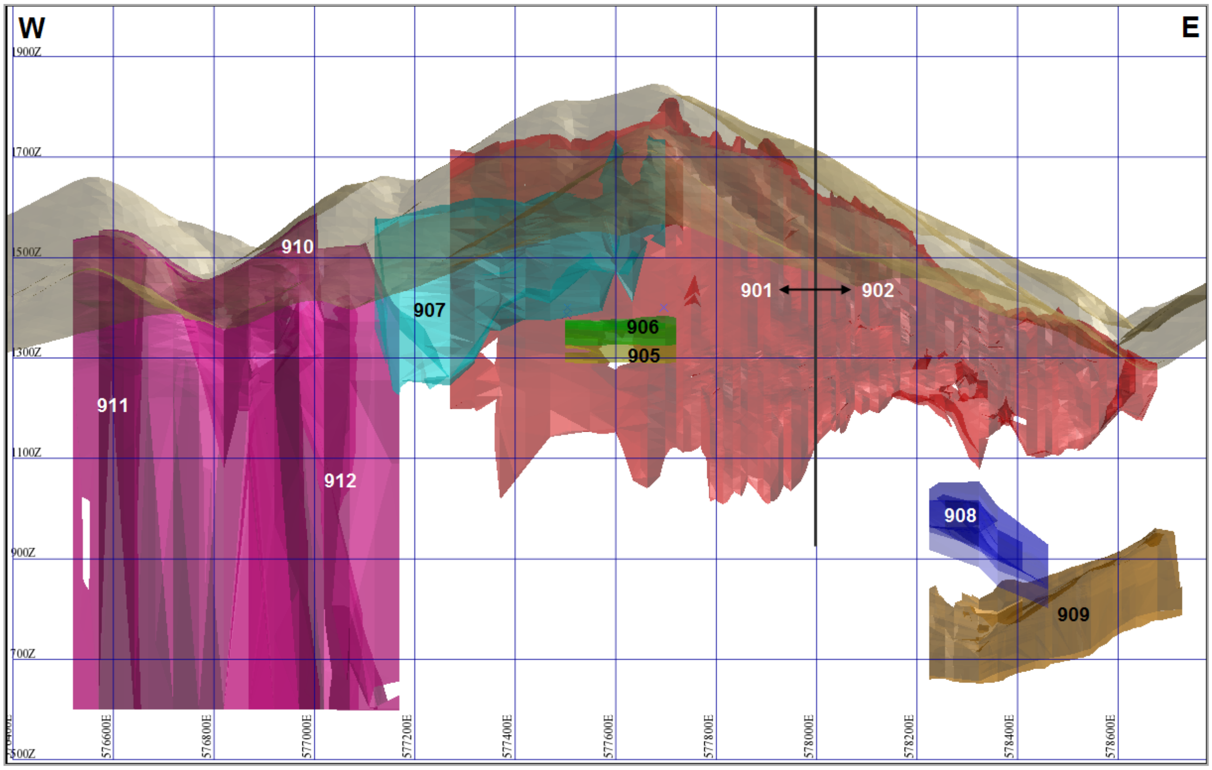
Table 14.13: Search Parameters for MS-Gabbro Domains

MS-Gabbro Domain	Pass	Data	Search Distances			Composites			Search Ellipsoid ZXY LRL		
			Major Axis	Semi-major Axis	Minor Axis	Min No.	Max No.	Max / Hole	Bearing	Plunge	Dip
901	1	All *	25	25	5	2	12	2	196	-68	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	250	250	62.5	3	16	3			
902	1	All *	25	25	5	2	12	2	190	-50	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	250	250	62.5	3	16	3			
903	1	All *	25	25	5	2	12	2	195	-90	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	250	250	62.5	3	16	3			
904	1	All *	25	25	5	2	12	2	350	-30	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	250	250	62.5	3	16	3			
905	1	All *	25	25	5	2	12	2	117	-18	0
	2	1987-2013	100	100	25	2	16	2			
	3	1987-2013	250	250	62.5	2	16	3			
906	1	All *	25	25	5	2	12	2	180	-9	0
	2	1987-2013	100	100	25	2	16	2			
	3	1987-2013	250	250	62.5	2	16	3			
907	1	All *	25	25	5	2	12	2	4	-90	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	250	250	62.5	3	16	3			
908	1	All *	25	25	5	2	12	2	122	-32	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	250	250	62.5	3	16	3			
909	1	All *	25	25	5	2	12	2	228	-37	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	250	250	62.5	3	16	3			
910	1	All *	25	25	5	2	12	2	350	-90	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	300	300	75	3	16	3			
911	1	All *	25	25	5	2	12	2	182	-77	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	300	300	75	3	16	3			
912	1	All *	25	25	5	2	12	2	350	-90	0
	2	1987-2013	100	100	25	3	16	2			
	3	1987-2013	300	300	75	3	16	3			

Source: GeoSim, 2014

* Included pre-1987 holes uncapped with missing intervals assigned a 0 grade

Figure 14.11: MS-Gabbro Search Domains – View Looking North



Source: GeoSim, 2014

14.9.2 Peridotite/Pyroxenite/Clinopyroxenite Domains

Five separate search domains were identified within the Dunite/Peridotite/Pyroxenite/Clinopyroxenite limits based on variograms models and zone geometry. Pre-1987 composites were not used for estimating grades as there were few sampled intervals and those that were analyzed were often missing Co, Pt, Pd, or Au values.

Grades were estimated in three passes using the Inverse Distance Cubed method (ID^3). For the five search domains, the first pass used uncapped 1987-2013 composites in order to restrict outlier values to a maximum range of 25 m along the major search axes. The second pass used capped composites and a maximum anisotropic range of 100 m with a two hole minimum. The final pass again used capped composites and the maximum search was set at 300 m for the larger domains and 200 m for domains 204 and 205.

After grades were estimated Ni values of blocks falling in the Dunite sub-domain were reduced by 0.1% under the assumption that this level of Ni was in silicate form and not recoverable.

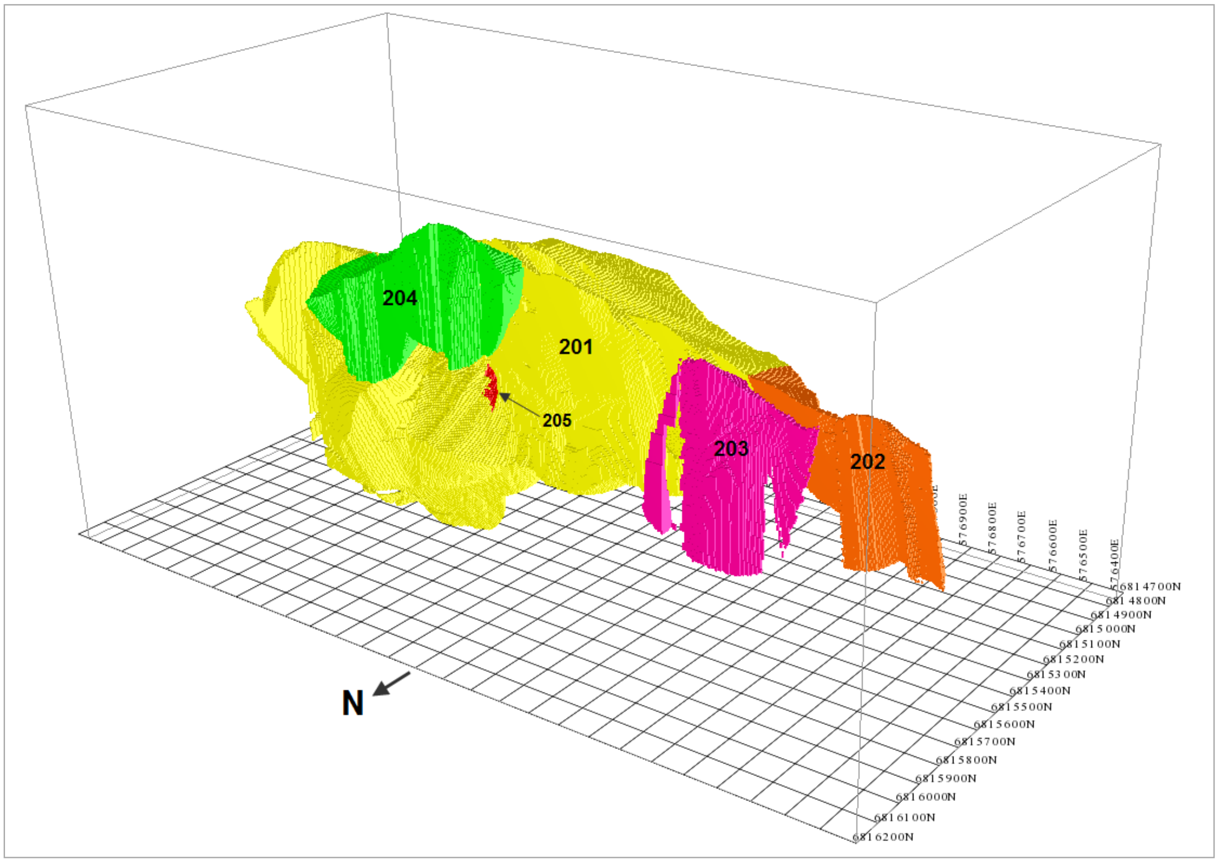
Search parameters for the Dunite/Peridotite/Clinopyroxenite domains are shown in Table 14.14. The locations of the search domains are illustrated in Figure 14.12.

Table 14.14: Search Parameters for Peridotite/Pyroxenite/Clinopyroxenite Domains

Peridotite Domain Code	Pass	Composite Data	Search Distances			Composites			Search Ellipsoid ZXY LRL		
			Major Axis	Semi-major Axis	Minor Axis	Min No.	Max No.	Max / Hole	Bearing	Plunge	Dip
201 Ni-Cu-Co	1	Uncapped	25	25	5	2	12	2	100	0	-61
	2	Capped	100	83	67	4	16	3			
	3	Capped	300	250	200	4	16	4			
201 Pt-Pd-Au	1	Uncapped	25	25	5	2	12	2	116	0	-51
	2	Capped	100	83	67	4	16	3			
	3	Capped	300	250	200	4	16	4			
202	1	Uncapped	25	21	17	2	12	2	218	-70	0
	2	Capped	100	83	67	4	16	3			
	3	Capped	300	250	200	4	16	4			
203	1	Uncapped	25	21	17	2	12	2	0	-90	0
	2	Capped	100	83	67	4	16	3			
	3	Capped	300	250	200	4	16	4			
204	1	Uncapped	25	21	17	2	12	2	10	-90	0
	2	Capped	100	83	67	4	16	3			
	3	Capped	200	167	133	4	16	4			
205	1	Uncapped	25	25	6	2	12	2	270	0	-52
	2	Capped	100	100	25	4	16	3			
	3	Capped	200	200	50	4	16	4			

Source: GeoSim, 2014

Figure 14.12: Peridotite/Pyroxenite/Clinopyroxenite Search Domains



Source: GeoSim, 2014

14.9.3 Sulphur Estimation

Sulphur content was estimated in a single pass using the Inverse Distance Squared method (ID^2). Search parameters are presented in Table 14.15 and Table 14.16.

Table 14.15: Search Parameters for Sulphur Content in MS-Gabbro Domains

MS-Gabbro Domain	Search Distances			Composites			Search Ellipsoid ZXY LRL		
	Major Axis	Semi-major Axis	Minor Axis	Min No.	Max No.	Max / Hole	Bearing	Plunge	Dip
901	300	300	75	4	16	4	196	-68	0
902	300	300	75	4	16	4	190	-50	0
903	300	300	75	4	16	4	195	-90	0
904	300	300	75	4	16	4	350	-30	0
905	300	300	75	2	16	2	117	-18	0
906	300	300	75	2	16	2	180	-9	0
907	300	300	75	2	16	2	4	-90	0
908	300	300	75	2	16	2	122	-32	0
909	300	300	75	2	16	2	228	-37	0
910	300	300	75	2	16	2	350	-90	0
911	300	300	75	2	16	2	182	-77	0
912	300	300	75	2	16	2	350	-90	0

Source: GeoSim, 2014

Table 14.16: Search Parameters for Sulphur Content in Peridotite/Pyroxenite/Clinopyroxenite Domains

Peridotite Domain	Search Distances			Composites			Search Ellipsoid ZXY LRL		
	Major Axis	Semi-major Axis	Minor Axis	Min No.	Max No.	Max / Hole	Bearing	Plunge	Dip
201/205	350	350	233	4	16	4	105	0	-67
202	350	350	233	2	16	2	218	-70	0
203	350	350	233	2	16	2	0	-90	0
204	350	350	233	2	16	2	105	0	-67

Source: GeoSim, 2014

14.10 Block Model Validation

14.10.1 Visual Inspection

Model verification was initially carried out by visual comparison of blocks and composite grades in plan and section views. The estimated block grades showed reasonable correlation with adjacent composite grades.

14.10.2 Global Bias Check

A comparison of global mean values between composites and block estimates within Peridotite domains shows a reasonably close relationship with composites and block model values (Table 14.17). Comparison of global block vs. composite data within the MS-Gabbro is not statistically meaningful due to the erratic and highly variable nature of the mineralization combined with selective sampling of historic drilling.

Table 14.17: Global Mean Grade Comparison in Peridotite/Pyroxenite/Clinopyroxenite

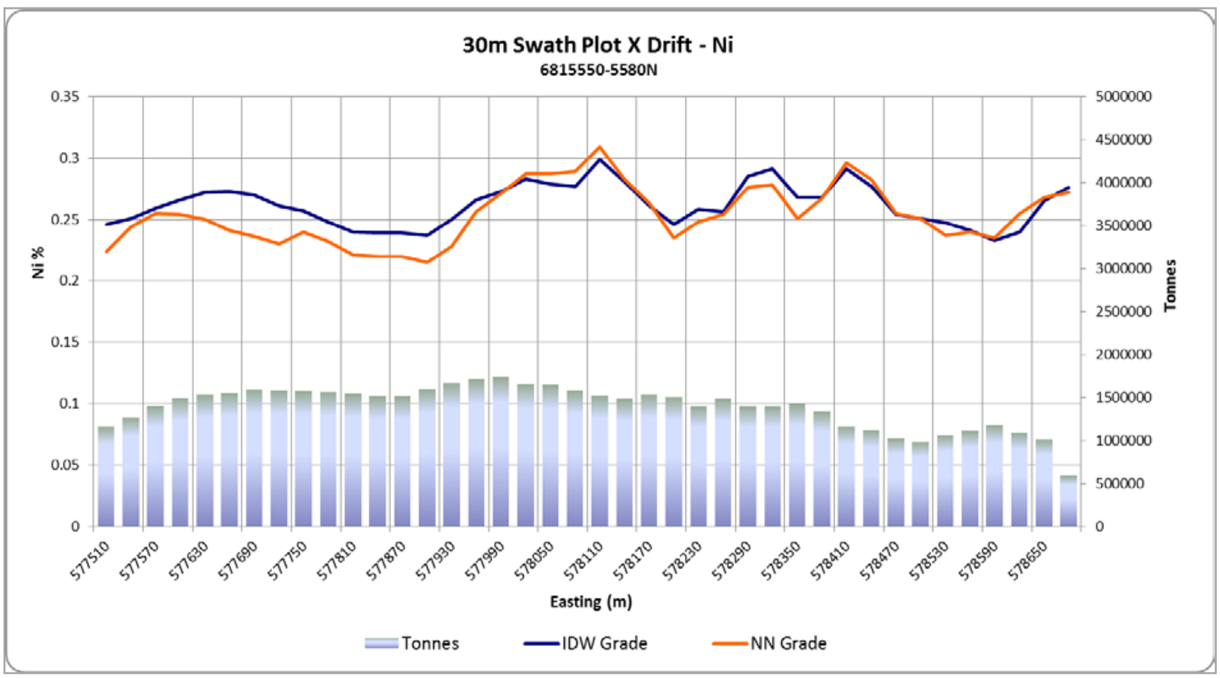
Data	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
Composites	0.252	0.138	0.015	0.232	0.231	0.046
Capped Composites	0.252	0.138	0.015	0.231	0.231	0.046
ID^3 Measured/Indicated	0.253	0.125	0.015	0.218	0.227	0.041
ID^3 Inferred	0.229	0.107	0.014	0.192	0.196	0.039

Source: GeoSim, 2014

14.10.3 Check for Local Bias

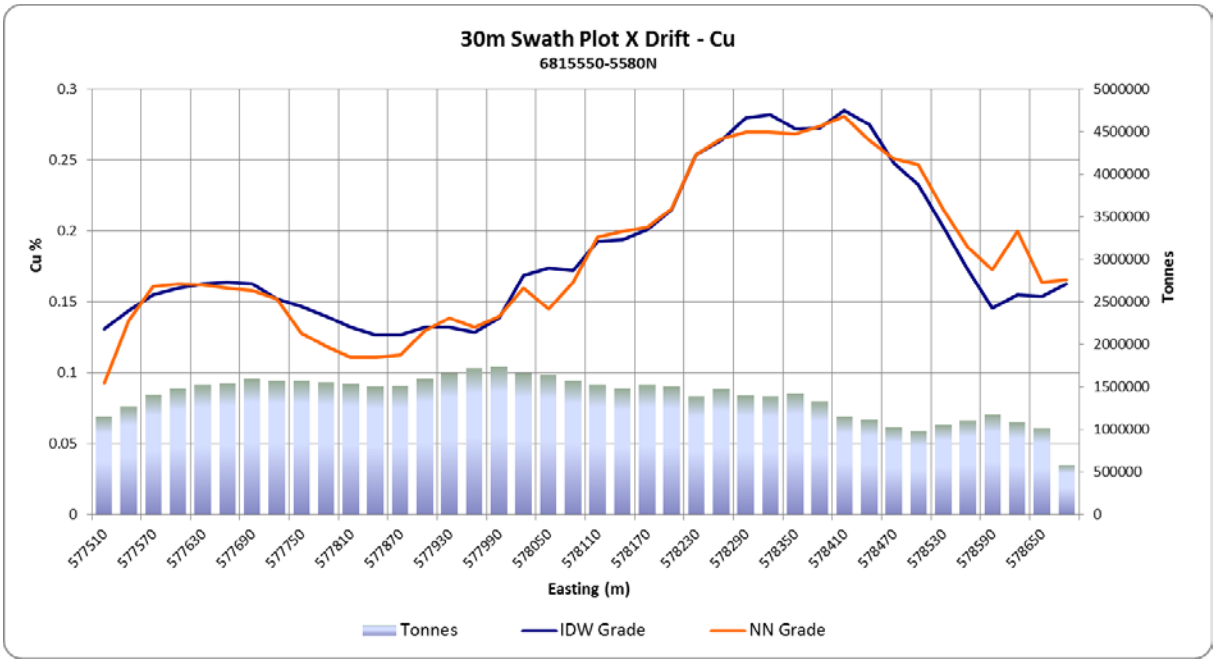
Swath plots were generated to assess the model for local bias by comparing ID^3 and nearest neighbour estimates on panels through the deposit. Results show a reasonable comparison between the methods, particularly in the main portions of the deposit indicated by the bar charts (Figure 14.13 to Figure 14.18).

Figure 14.13: Swath Plot X Drift - Ni



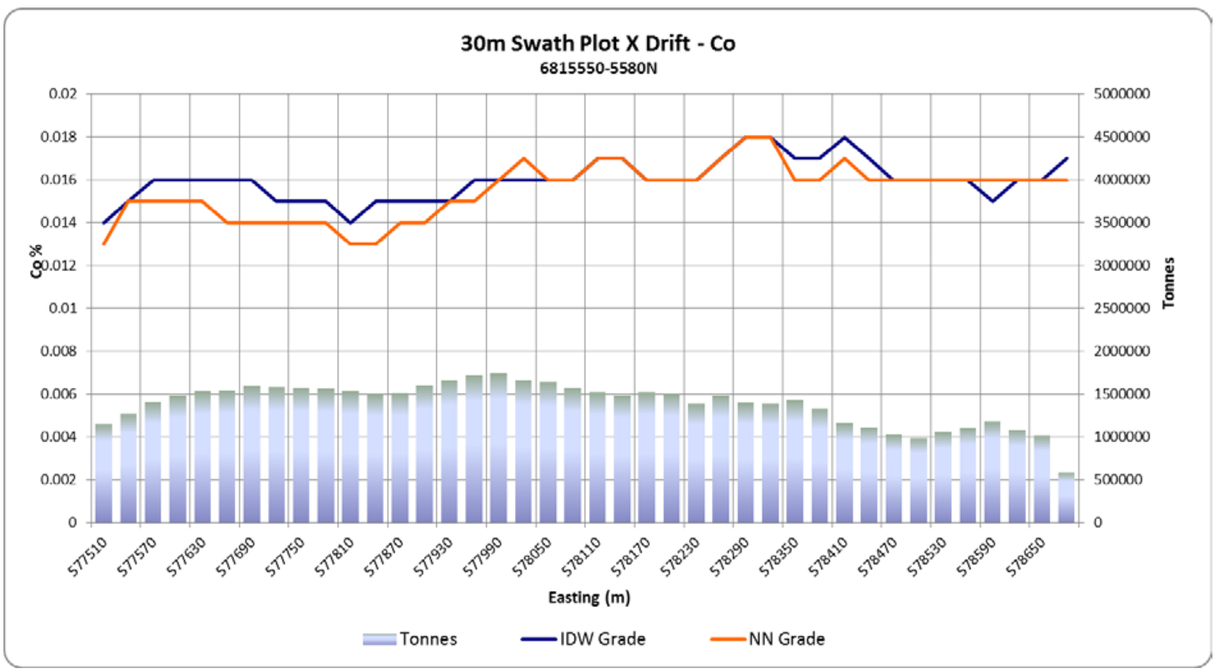
Source: GeoSim, 2014

Figure 14.14: Swath Plot X Drift - Cu



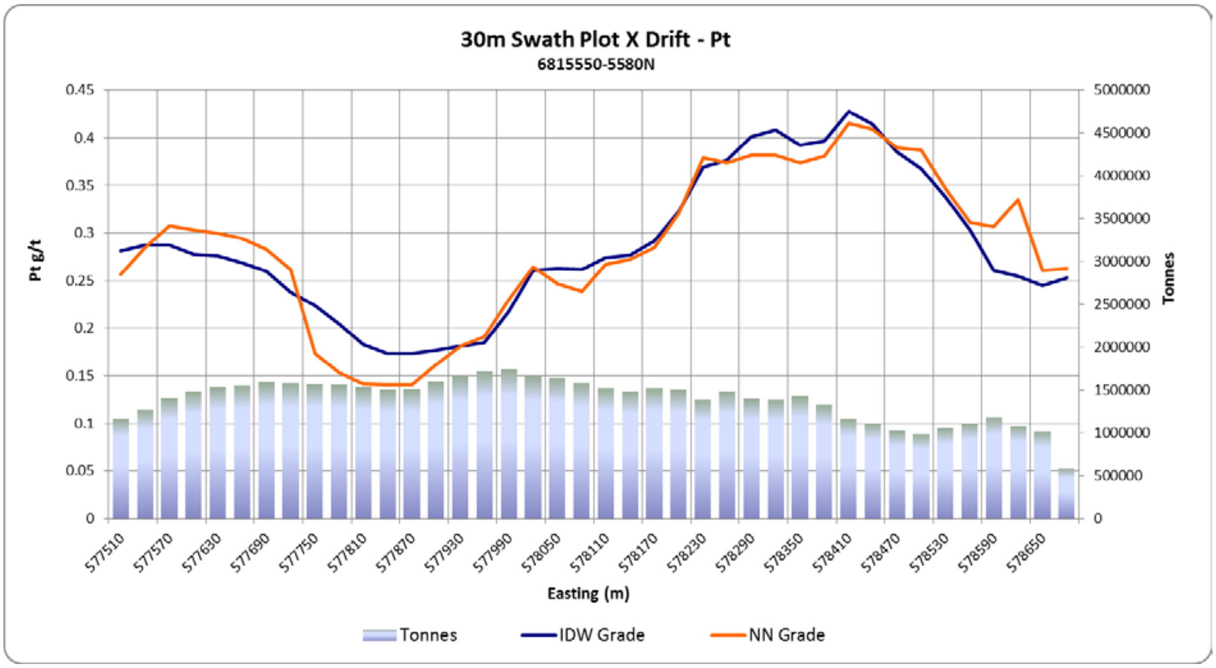
Source: GeoSim, 2014

Figure 14.15: Swath Plot X Drift - Co



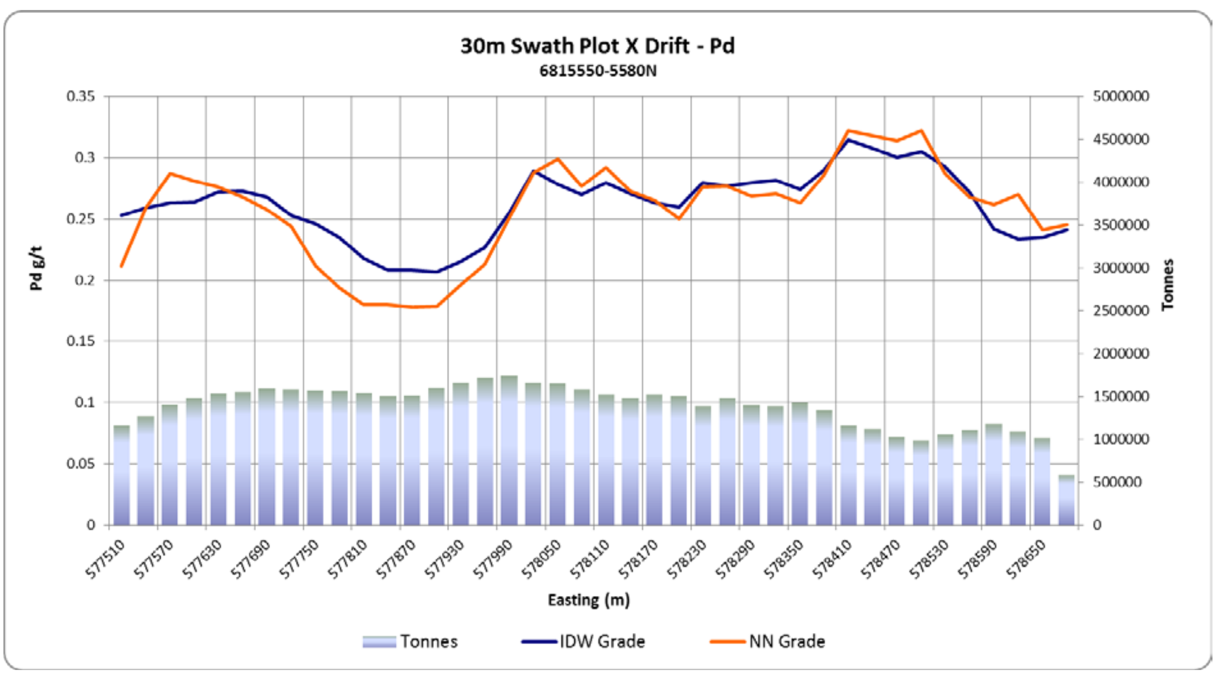
Source: GeoSim, 2014

Figure 14.16: Swath Plot X Drift - Pt



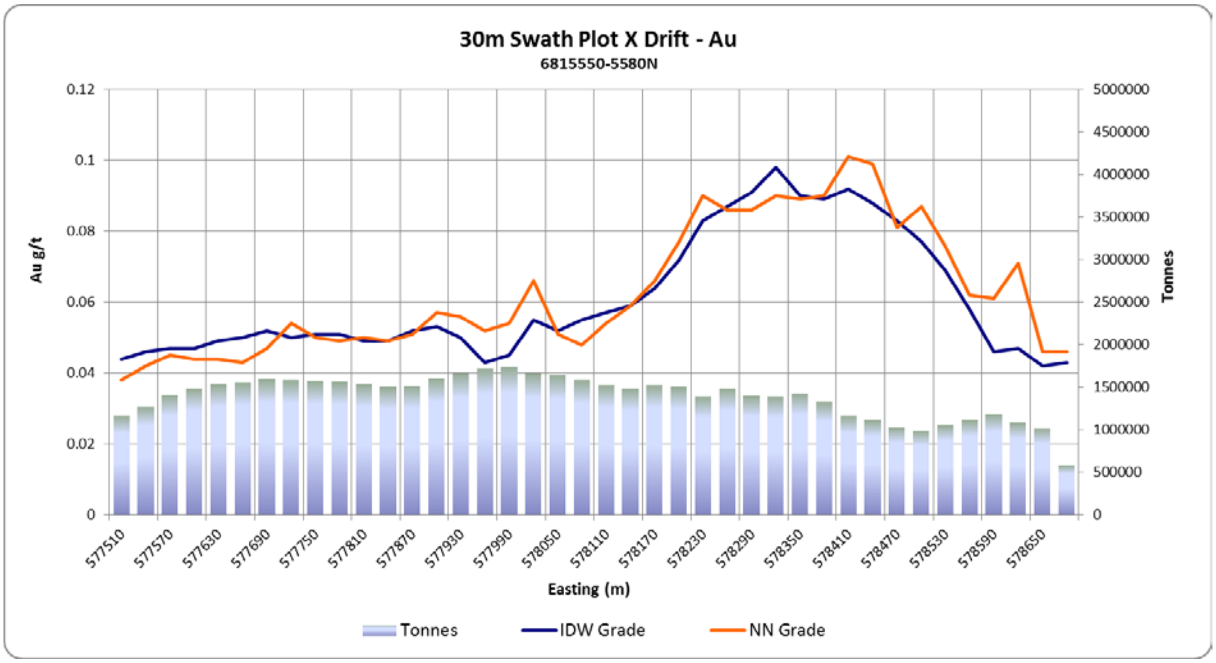
Source: GeoSim, 2014

Figure 14.17: Swath Plot X Drift - Pd



Source: GeoSim, 2014

Figure 14.18: Swath Plot X Drift - Au



Source: GeoSim, 2014

14.11 Classification of Mineral Resources

Resource classifications used in this study conform to the CIM Definition Standards for Mineral Resources and Mineral Reserves.

In order to be classified as a measured mineral resource a block had to meet the following conditions:

- Restricted to the main southern Dunite/Peridotite/Pyroxenite/Clinopyroxenite domains (excluding domain 204);
- Estimated using only 1987-2013 data (mostly re-sampled 1987-88 intervals);
- Not extrapolated beyond drilling limits; and
- Within a 50 m drill hole spacing based on 1987-2013 drilling.
-

Some isolated blocks and clusters were downgraded to indicated mineral resource based on visual examination.

Blocks not assigned to the measured mineral resource category were classified as indicated mineral resource if they met the following conditions:

- Estimated in the second pass using only post 1987 data and a minimum of two drill holes;
- Within an approximate 50 m x 50 m drill spacing based on 1987-2013 drilling within MS-Gabbro domains;

- With a 100 m x 100 m drill spacing based on 1987-2013 drilling within Dunite/Peridotite/Clinopyroxenite domains; and
- Not extrapolated more than 50 metres beyond drilling limits.

Blocks not classified as measured or indicated mineral resource were assigned to the inferred mineral resource category provided that they were extrapolated no further than 200 m. An exception was made for a few blocks that were constrained by the MS-Gabbro wireframes that were included in the inferred category to eliminate interior gaps in the model.

14.12 Metal Equivalency Grades

For the resource estimate, nickel equivalent (NiEq) values were calculated using metal price assumptions of US$ $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au.

NiEq [%] = (Ni +Cu *0.359+Co *1.557+Au *0.218+Pt *0.262+Pd *0.131)

PtEq [g/t] = NiEq / 100 * 2204.62 * 8.35 / 1,500 x 31.103

14.13 Reasonable Prospects of Economic Extraction

To assess reasonable prospects for eventual economic extraction a floating cone optimized pit, was prepared using the general economic and technical assumptions listed in Table 14.18 and metal prices stated in Section 14.12.

Table 14.18: Pit Optimization Parameters

	Parameter
Pit Slope	45°
Mining Cost	C$2.00/tonne
Processing Cost	C$12.91/tonne
G&A Cost	C$1.10/tonne
Nickel Recovery	70%
Copper Recovery	90%
Cobalt Recovery	64%
Platinum Recovery	60%
Palladium Recovery	70%
Gold Recovery	90%
Exchange Rate USD:CAD	0.91

Source: JDS, 2015

Blocks falling outside of the optimized pit shell were not considered to be part of the mineral resource.

14.14 Mineral Resource Estimate

Mineral Resources are classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves.

Table 14.19 presents the mineral resource estimate for the Wellgreen project at a base case cut-off grade of 0.57 g/t Pt Equivalent or 0.15% Ni Equivalent.

Table 14.19: Mineral Resource at a 0.57 g/t PtEq or 0.15% NiEq cut-off

Category	Tonnes 000s	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	3E g/t	Ni Eq. %	Pt Eq. %
Measured	92,293	0.260	0.155	0.015	0.252	0.246	0.052	0.550	0.449	1.713
Indicated	237,276	0.261	0.135	0.015	0.231	0.238	0.042	0.511	0.434	1.656
Total M&I	329,569	0.261	0.141	0.015	0.237	0.240	0.045	0.522	0.438	1.672
Inferred	846,389	0.237	0.139	0.015	0.234	0.226	0.047	0.507	0.412	1.571

Source: GeoSim, 2014

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. with an effective date of July 23, 2014.
2. Measured mineral resources are drilled on approximate 50 x 50 metre drill spacing and confined to clinopyroxenite and peridotite/dunite domains. Indicated mineral resources are drilled on approximate 100 x 100 m drill spacing except for the massive sulphide and gabbro domains which used 50 x 50 m spacing.
3. Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries. Ni Eq% = Ni%+ Cu% x 3.00/8.35 + Co% x 13.00/8.35 + Pt [g/t]/31.103 x 1,500/8.35/22.046 + Pd [g/t]/31.103 x 750/8.35/22.046 + Au [g/t]/31.103 x 1,250/8.35/22.046. Pt Eq[g/t] = Ni Eq/100×2204.62×8.35 / 1,500×31.103
4. An optimized pit shell was generated using the following assumptions: metal prices in Note 3 above ; a 45 degree pit slope; assumed metallurgical recoveries of 70% for Ni, 90% for Cu, 64% for Co, 60% for Pt, 70% for Pd and 75% for Au; an exchange rate of C$1.00=US$$0.91; and mining costs of $2.00 per tonne, processing costs of $12.91 per tonne, and general & administrative charges of $1.10 per tonne (all expressed in Canadian dollars).
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
7. 3E = Pt + Pd + Au

In addition, Table 14.20 below shows the higher grade portion of the resource within the constrained pit at a 1.9 g/t Pt Eq. or 0.50% Ni Eq. cut-off.

Table 14.20: Mineral Resource at a 1.9 g/t PtEq or 0.50 NiEq Cut-off

Category	Tonnes 000s	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	3E g/t	Ni Eq. %	Pt Eq. %
Measured	21,854	0.326	0.301	0.019	0.454	0.366	0.103	0.923	0.653	2.492
Indicated	50,264	0.334	0.286	0.019	0.455	0.373	0.090	0.919	0.653	2.493
Total M&I	72,117	0.332	0.291	0.019	0.455	0.371	0.094	0.920	0.653	2.493
Inferred	173,684	0.309	0.301	0.018	0.456	0.352	0.098	0.906	0.631	2.410

Source: GeoSim, 2014

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. with an effective date of July 23, 2014.
2. Measured mineral resources are drilled on approximate 50 x 50 m drill spacing and confined to clinopyroxenite and peridotite/dunite domains. Indicated mineral resources are drilled on approximate 100 x 100 m drill spacing except for the massive sulphide and gabbro domains which used 50 x 50 m spacing.
3. Nickel equivalent (Ni Eq. %) and platinum equivalent (Pt Eq. g/t) calculations reflect total gross metal content using US$ of $8.35/lb Ni, $3.00/lb Cu, $13.00/lb Co, $1,500/oz Pt, $750/oz Pd and $1,250/oz Au and have not been adjusted to reflect metallurgical recoveries. Ni Eq% = Ni%+ Cu% x 3.00/8.35 + Co% x 13.00/8.35 + Pt [g/t]/31.103 x 1,500/8.35/22.046 + Pd [g/t]/31.103 x 750/8.35/22.046 + Au [g/t]/31.103 x 1,250/8.35/22.046. Pt Eq[g/t] = Ni Eq/100×2204.62×8.35 / 1,500×31.103
4. An optimized pit shell was generated using the following assumptions: metal prices in Note 3 above ; a 45 degree pit slope;

assumed metallurgical recoveries of 70% for Ni, 90% for Cu, 64% for Co, 60% for Pt, 70% for Pd and 75% for Au; an exchange rate of CAN$1.00=USA$0.91; and mining costs of $2.00 per tonne, processing costs of $12.91 per tonne, and general & administrative charges of $1.10 per tonne (all expressed in Canadian dollars).

5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
7. 3E = Pt + Pd + Au

Table 14.21 to Table 14.24 show the sensitivities of the resource to cut-off grade.

Table 14.21: Sensitivity to Cut-off Grade – Measured Resource Category

% NiEq Cut-off	Tonnes 000's	Grades								
		Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	Grade 3E g/t	NiEq %	PtEq g/t
0.10	93,332	0.257	0.154	0.015	0.250	0.244	0.051	0.546	0.445	1.700
0.15	**92,293**	**0.260**	**0.155**	**0.015**	**0.252**	**0.246**	**0.052**	**0.550**	**0.449**	**1.713**
0.20	90,815	0.262	0.156	0.015	0.254	0.248	0.052	0.555	0.453	1.730
0.25	88,625	0.266	0.158	0.016	0.257	0.251	0.053	0.561	0.459	1.751
0.30	83,231	0.272	0.164	0.016	0.266	0.258	0.054	0.578	0.470	1.796
0.35	71,784	0.282	0.176	0.016	0.284	0.274	0.058	0.617	0.493	1.883
0.40	55,642	0.295	0.196	0.017	0.315	0.296	0.065	0.676	0.527	2.012
0.45	36,455	0.311	0.237	0.018	0.371	0.329	0.080	0.779	0.581	2.217
0.50	21,854	0.326	0.301	0.019	0.454	0.366	0.103	0.923	0.653	2.492

Source: GeoSim, 2014

Table 14.22: Sensitivity to Cut-off Grade - Indicated Resource Category

% NiEq Cut-off	Tonnes 000's	Grades								
		Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	Grade 3E g/t	NiEq %	PtEq g/t
0.10	249,006	0.252	0.130	0.015	0.223	0.231	0.041	0.495	0.419	1.601
0.15	**237,276**	**0.261**	**0.135**	**0.015**	**0.231**	**0.238**	**0.042**	**0.511**	**0.434**	**1.656**
0.20	229,001	0.267	0.138	0.015	0.236	0.243	0.043	0.523	0.443	1.692
0.25	223,554	0.270	0.140	0.015	0.240	0.247	0.044	0.530	0.449	1.713
0.30	207,082	0.276	0.147	0.015	0.251	0.257	0.046	0.553	0.462	1.764
0.35	173,273	0.286	0.164	0.016	0.275	0.276	0.051	0.602	0.489	1.865
0.40	132,328	0.298	0.186	0.017	0.309	0.299	0.057	0.666	0.523	1.998
0.45	83,313	0.315	0.228	0.018	0.372	0.335	0.071	0.778	0.581	2.219
0.50	50,264	0.334	0.286	0.019	0.455	0.373	0.090	0.919	0.653	2.493

Source: GeoSim, 2014

Table 14.23: Sensitivity to Cut-off Grade - Measured and Indicated Resource Categories

% NiEq Cut-off	Tonnes 000's	Grades								
		Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	Grade 3E g/t	NiEq %	PtEq g/t
0.10	342,338	0.253	0.136	0.015	0.230	0.235	0.044	0.509	0.427	1.630
0.15	**329,569**	**0.261**	**0.141**	**0.015**	**0.237**	**0.240**	**0.045**	**0.522**	**0.438**	**1.672**
0.20	319,816	0.266	0.143	0.015	0.241	0.245	0.046	0.532	0.446	1.702
0.25	312,179	0.269	0.145	0.015	0.245	0.248	0.046	0.539	0.452	1.725
0.30	290,314	0.275	0.152	0.016	0.255	0.257	0.048	0.560	0.464	1.771
0.35	245,057	0.285	0.167	0.016	0.278	0.276	0.053	0.607	0.490	1.870

0.40	187,970	0.297	0.189	0.017	0.311	0.298	0.059	0.668	0.525	2.004
0.45	119,768	0.314	0.231	0.018	0.372	0.333	0.073	0.778	0.581	2.218
0.50	72,117	0.332	0.291	0.019	0.455	0.371	0.094	0.920	0.653	2.493

Source: GeoSim, 2014

Table 14.24: Sensitivity to Cut-off Grade - Inferred Resource Category

% NiEq Cut-off	Tonnes 000's	Grades								
		Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t	Grade 3E g/t	NiEq %	PtEq g/t
0.10	946,412	0.220	0.127	0.015	0.216	0.211	0.043	0.470	0.381	1.456
0.15	**846,389**	**0.237**	**0.139**	**0.015**	**0.234**	**0.226**	**0.047**	**0.507**	**0.412**	**1.571**
0.20	774,501	0.250	0.149	0.015	0.249	0.236	0.050	0.534	0.434	1.656
0.25	747,897	0.254	0.153	0.015	0.255	0.240	0.051	0.546	0.441	1.685
0.30	697,852	0.258	0.160	0.015	0.265	0.248	0.053	0.566	0.453	1.728
0.35	564,699	0.267	0.183	0.016	0.294	0.267	0.061	0.622	0.483	1.842
0.40	415,192	0.281	0.209	0.016	0.331	0.292	0.069	0.692	0.522	1.992
0.45	265,603	0.297	0.251	0.017	0.393	0.325	0.082	0.801	0.577	2.202
0.50	173,684	0.309	0.301	0.018	0.456	0.352	0.098	0.906	0.631	2.410

Source: GeoSim, 2014

14.15 Factors That May Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the mineral resource estimate include:

- Commodity price assumptions;
- Pit slope angles;
- Metal recovery assumptions; and
- Mining and Process cost assumptions.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the Yukon Territory, Canada in terms of environmental, permitting, taxation, socio-economic, marketing and political factors. GeoSim is not aware of any legal or title issues that would materially affect the mineral resource estimate.

15 Mineral Reserve Estimate

Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the Wellgreen project.

16 Mining Methods

SNC-Lavalin conducted underground and open pit mine optimization studies on the Wellgreen deposit. The optimum mining method was determined to be a combination of conventional truck shovel open pit mining and two underground mining methods. This section will discuss the open pit mining and the underground mining studies.

16.1 Open Pit Mining

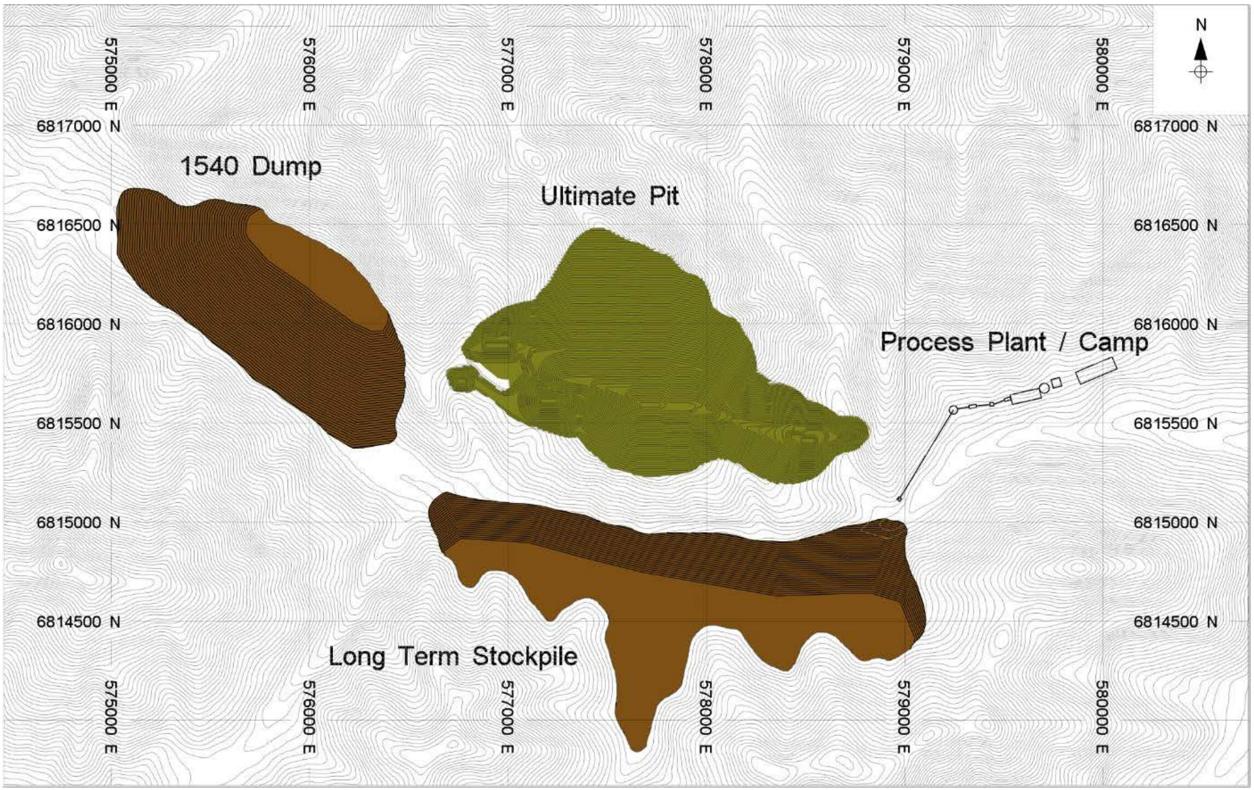
SNC-Lavalin evaluated the open pit potential of the Property at a mill feed rate of 25,000 t/day increasing to 50,000 t/day in year six. The ultimate pit for the 2015 PEA base case has been phased into four preliminary pushbacks. Mining cut offs and stockpiling grades have been established for each pushback to target higher-grade mill feed.

Mill feed will be hauled directly to the crusher and low grade material will be hauled to the long term stockpile and processed at the end of the mine life. Waste rock will be hauled to the 1540 dump and the tailings dam facility.

The pre-stripping period is one year in duration and provides the necessary construction materials for the tailings dam.

The general mine layout is shown in Figure 16.1.

Figure 16.1: Mine General Layout



Source: SNC, 2015

16.2 Pit Optimization

The pit optimization process included inferred mineral resources that are considered too speculative geologically to be considered reserves and there is no certainty that the preliminary economic assessment will be realized.

Pit optimization was completed with GEOVIA Whittle software. Optimized pit shells were generated with the Lerch-Grossman algorithm and variable revenue factor method. From this the optimized pit shell was selected.

16.2.1 Pit Optimization Parameters

Wellgreen Platinum and SNC-Lavalin reviewed mine optimization parameters necessary to determine the optimized economic open pit profile. Mine operating costs were developed for input by SNC-Lavalin based on other recent projects and on estimates provided by Wellgreen Platinum. Process recoveries and estimated process costs were developed from information provided by Wellgreen Platinum. These initial parameters were applied to determine the optimized pit shells. The metal prices utilized to optimize the pit shells are equal to the metal prices utilized to generate the resources. The remaining parameters differ slightly from that utilized in the resources. These values are also different than those utilized in the final cash flows.

A summary of these parameters is provided in Table 16.1 and Table 16.2.

Table 16.1: Pit Optimization Metal Prices

Item	Unit	Value
Exchange Rate	USD:CAD	0.91
Discount Rate	%	7.5
Metal Prices		
Platinum	US$/troy oz	1,500
Palladium	US$/troy oz	750
Gold	US$/troy oz	1,250
Nickel	US$/lb	8.35
Copper	US$/lb	3
Cobalt	US$/lb	13

Source: JDS, 2015

Table 16.2: Pit Optimization Recoveries & Other Parameters

Metal Recoveries	Unit	Gabbro/MS	Clinopyroxenite/ Pyroxenite	Peridotite
Platinum	%	74.5	59	57.6
Palladium	%	80.5	73	58.4
Gold	%	69.8	65.8	58.8
Nickel	%	83.0	75.0	68.1
Copper	%	94.5	88.3	66.3
Cobalt	%	67.9	64.4	54.9
Mining Cost	$/tonne	.20 + Db*0.005		Db = Difference in 10m benches
Processing Cost	$/tonne	13.11		
G&A	$/tonne	1.85		
Mining Recovery	%	99		
Mining Dilution	%	4		
Overall Pit Slope	degrees	40		
Mill throughput	t/day	25,000		
Shipping Cost	US$/t	123		
Bulk Con Ni%	%	6		
Smelting	$/tCon	175		
Payable	%	50-95		
Refining	$/unit	0.4 -15.0		
Deductions	g/t	0.5 - 5.0		

Source: SNC 2015

16.2.2 Geological Block Model

The 3-D mineral inventory model was produced by GeoSim Services Inc. The QP for the mineral resources is Ron Simpson, P. Geo. SNC-Lavalin has not audited or verified the block model and has relied on the work of Mr. Simpson.

The provided resource model was a 5 m x 5 m x 5 m (XYZ) ASCII format block model. This model was then re-blocked within Whittle to 20 m x 20 m x 10 m for optimization. Each block in the model is coded as a specific rock type without a variable percent model. The block model variables can be seen in Table 16.3.

Table 16.3: Block Model Variables

Variable
Ni %
Cu %
Co %
Pt g/t
Pd g/t
Au g/t
Class
Litho
SG

Source SNC, 2015

16.2.3 Overall Open Pit Slope Angle

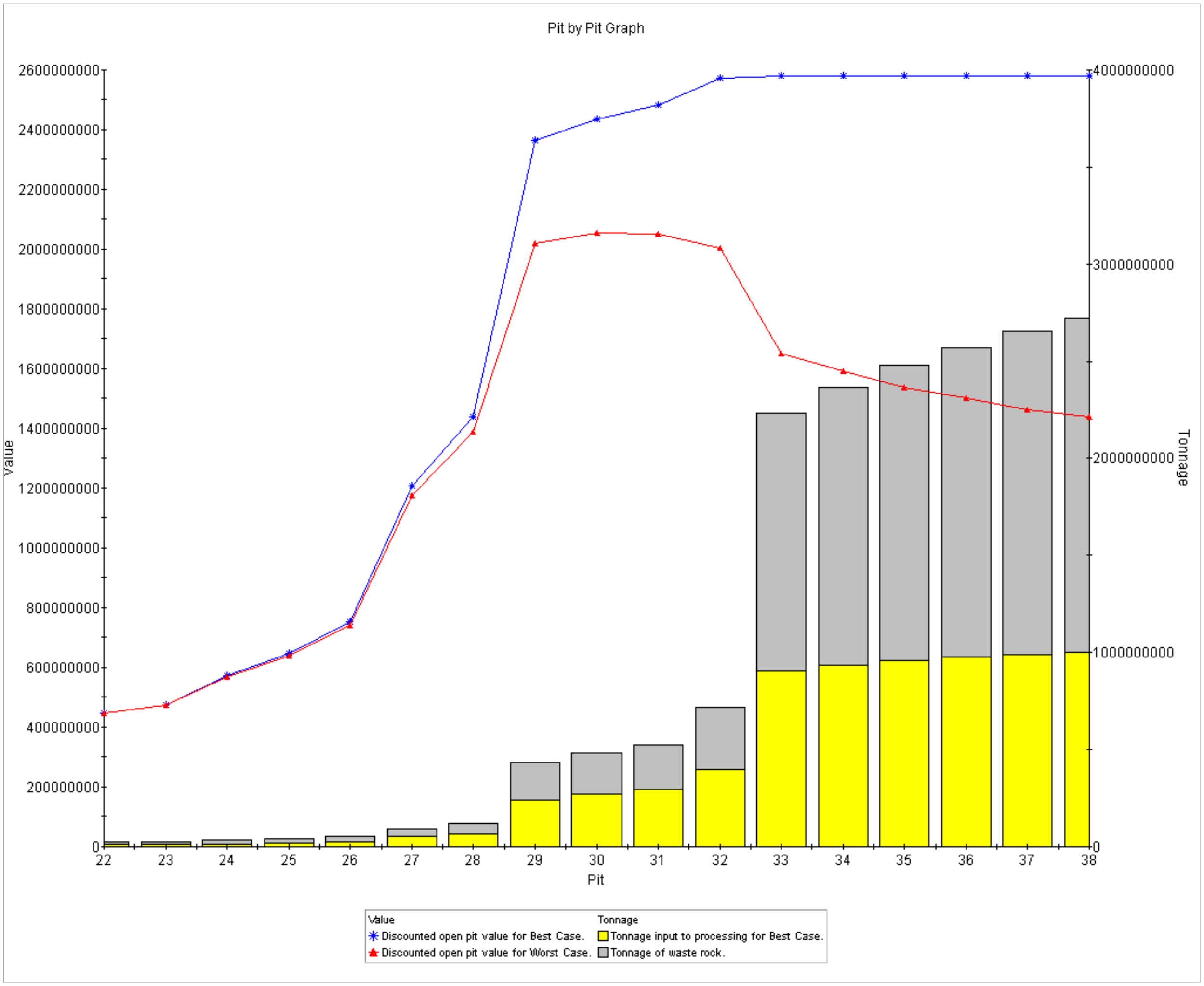
SRK Consulting (US) Inc. (SRK) provided SNC-Lavalin with the initial open pit geotechnical requirements utilized in the optimization. The overall pit slope angles were limited to 40 degrees for walls greater than 500 m, and pit slope angles of 45 degrees were selected for walls less than 500 m. This geotechnical criterion was only used for optimization and differs slightly from the final slope angles used in the detailed design.

16.2.4 Pit Optimization Results

Fifty-one pit shells were generated with a variable revenue factor. Based on the value curves, pit 32 was selected as the optimized pit shell to bring forward into design and scheduling. This shell provides the framework for design of the ultimate pit design.

A summary of the pit optimization results is shown in Figure 16.2.

Figure 16.2: Pit Optimization Results



Source: SNC, 2015

The blue line represents the optimal cash flow resulting from consecutive mining of incremental pit shells. The red line represents a cash flow resulting from mining top down, bench by bench and is considered to be the most conservative approach with respect to Net Present Value (NPV).

The optimized pit shells were reviewed, whereupon it was determined that a four stage approach would be developed for the economic analysis. The four stages (pit shell 25, pit shell 28, pit shell 29 and the ultimate pit shell 32) utilized in the 2015 PEA base case cash flow as well as the fifth stage (pit shell 33), which is considered to be an opportunity and is not part of the 2015 PEA economic analysis, are indicated in Table 16.4.

Table 16.4: Pit Optimization Results

Pit	Revenue Factor	Rock MTonnes	Mill Feed MTonnes	Strip Ratio	Ni	Cu	Co	Pt	Pd	Au
					%	%	%	g/t	g/t	g/t
17	0.41	15.6	5.2	2.0	0.34	0.58	0.02	0.74	0.39	0.17
18	0.42	16.9	5.6	2.0	0.34	0.57	0.02	0.73	0.39	0.16
19	0.43	17.7	5.9	2.0	0.34	0.56	0.02	0.72	0.38	0.16
20	0.44	18.3	6.2	2.0	0.34	0.54	0.02	0.71	0.38	0.16
21	0.46	18.9	6.5	1.9	0.34	0.53	0.02	0.69	0.38	0.15
22	0.47	24.6	7.8	2.2	0.33	0.51	0.02	0.68	0.37	0.15
23	0.48	26.5	8.7	2.1	0.32	0.49	0.02	0.65	0.37	0.14
24	0.49	36.7	11.4	2.2	0.31	0.45	0.02	0.61	0.36	0.13
25	**0.51**	**43.6**	**14.1**	**2.1**	**0.3**	**0.41**	**0.02**	**0.57**	**0.35**	**0.12**
26	0.52	52.9	17.6	2.0	0.3	0.38	0.02	0.52	0.35	0.11
27	0.54	91.8	33.8	1.7	0.3	0.29	0.02	0.42	0.32	0.08
28	**0.56**	**120.9**	**47.8**	**1.5**	**0.3**	**0.26**	**0.02**	**0.38**	**0.31**	**0.08**
29	**0.58**	**435.8**	**151.5**	**1.9**	**0.29**	**0.21**	**0.02**	**0.33**	**0.31**	**0.06**
30	0.59	480.9	187.1	1.6	0.29	0.2	0.02	0.31	0.3	0.06
31	0.62	525.7	222.3	1.4	0.28	0.18	0.02	0.3	0.29	0.05
32	**0.64**	**715.1**	**309.6**	**1.3**	**0.28**	**0.17**	**0.02**	**0.28**	**0.28**	**0.05**
33	0.66	2,233.8	736.9	2.0	0.27	0.16	0.02	0.27	0.27	0.05
34	0.69	2,365.6	794.1	2.0	0.27	0.16	0.02	0.27	0.26	0.05
35	0.72	2,481.2	845.7	1.9	0.27	0.16	0.02	0.27	0.26	0.05
36	0.75	2,568.4	894.1	1.9	0.26	0.16	0.02	0.26	0.25	0.05
37	0.78	2,656.0	942.2	1.8	0.26	0.15	0.02	0.25	0.25	0.05

Source: SNC, 2015
Note: The values shown per pit are total values, and not cumulative. For example, Pit 33 is larger than Pit 32 and contains all previous pits

16.3 Ultimate Pit Design

Pit designs were completed with Hexagon MineSight 3-D software. Based on optimization results, pit shell 32 (inclusive of the 4 pit stages) was selected as the guide for the ultimate pit design for the 2015 PEA base case, the results of which are provided in Table 16.5. Dilution and mining recovery were based on analysis of similar operations and assumed to be 4% and 98%, respectively.

The ultimate design (Stage 4, Figure 16.3) and pushbacks are preliminary and, therefore, do not include ramp access in the design. During pre-feasibility, trade off studies for extraction and location of the ramp will be completed. Due to the orientation of the deposit relative to the topography the amount of waste development required to access the bottom of the pit is considerably less than a comparable open pit mine in flat topography.

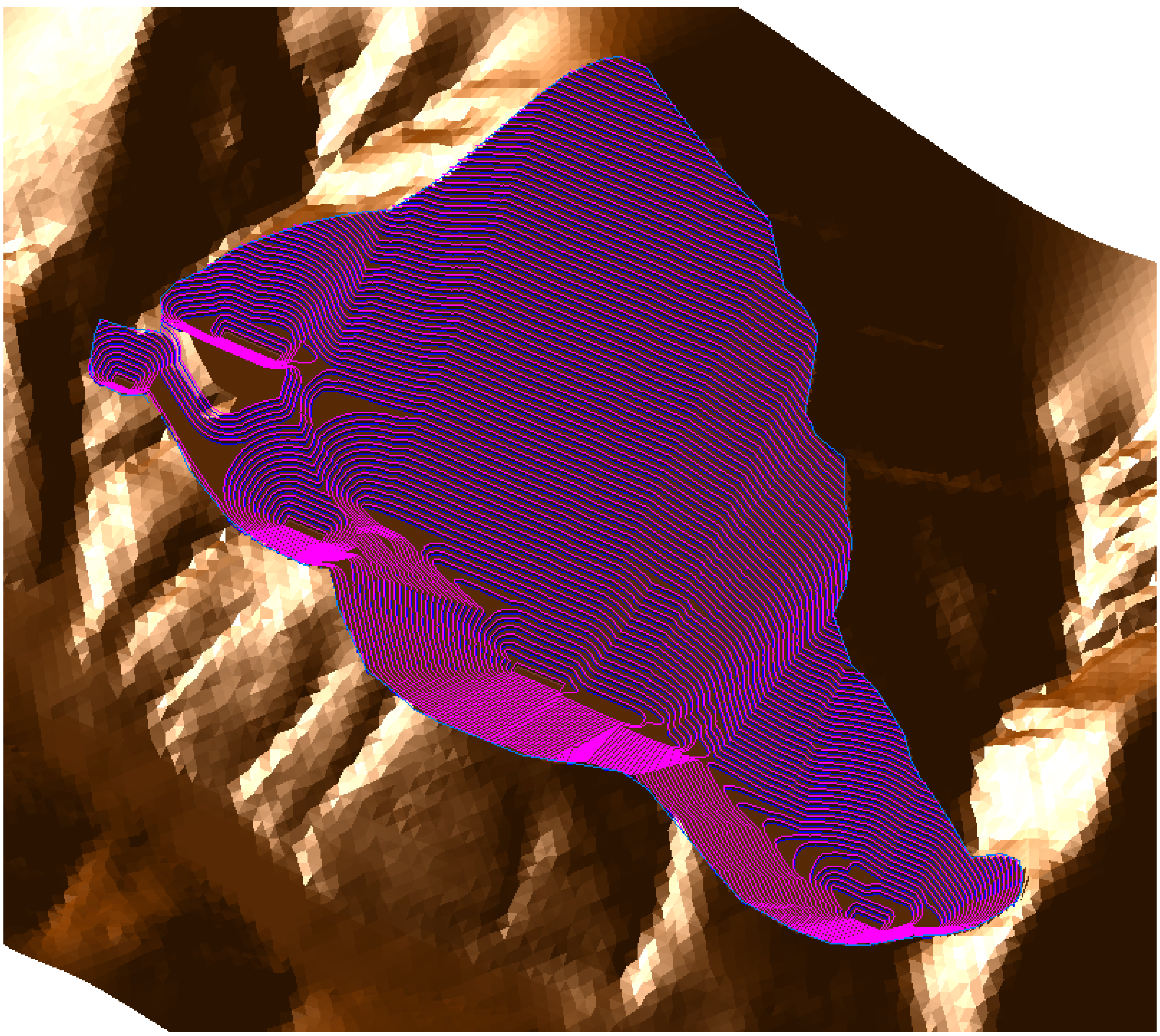
Table 16.5: PEA Base Case Pit Results

Rock	Pt Eq g/t	MTonnes	Ni%	Cu%	Co%	Pt g/t	Pd g/t	Au g/t
Measured	>0.6	69.2	0.25%	0.16%	0.02%	0.259	0.243	0.054
Indicated	>0.6	123.6	0.26%	0.13%	0.01%	0.221	0.235	0.039
Inferred*	>0.6	198.9	0.25%	0.12%	0.01%	0.215	0.235	0.037
Total Mineralized Material	>0.6	391.7	0.25%	0.13%	0.01%	0.225	0.236	0.04
Waste		296.2						

Source: SNC, 2015

* Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Figure 16.3: Ultimate Stage 4 Pit Design (Pit 32)



Source: SNC, 2015

In order to improve overall economics and production levels, elevated mill feed cut offs were selected by Wellgreen Platinum. A low grade stockpile consisting of mineralized material from 0.60 g/t Pt Eq to 1.0 g/t Pt Eq was established as well as a high grade stockpile from 1.0g/t Pt Eq to various grades depending on the stage of the pit. The two stockpiles, by stage, are summarized in Table 16.6:

Table 16.6: Pit Phase Cut-offs

Stage	Low Grade Stockpile	High Grade Stockpile	Mill Feed
1	0.6 g/t Pt Equiv – 1.0 g/t Pt Eq	1.0 g/t Pt Eq – 1.5 g/t Pt Eq	>1.5 g/t Pt Eq
2	0.6 g/t Pt Equiv – 1.0 g/t Pt Eq	1.0 g/t Pt Eq – 1.7 g/t Pt Eq	>1.7 g/t Pt Eq
3	0.6 g/t Pt Equiv – 1.0 g/t Pt Eq	1.0 g/t Pt Eq – 1.3 g/t Pt Eq	>1.3 g/t Pt Eq
4	0.6 g/t Pt Equiv – 1.0 g/t Pt Eq	1.0 g/t Pt Eq – 1.5 g/t Pt Eq	>1.5 g/t Pt Eq

Source: SNC 2015

16.3.1 Metal Equivalent Calculation

Metal equivalent was used for reporting and bin sizes only. Cash flows and economics consider recoveries separately for all rock types. The following equations were used to calculate Nickel and Platinum equivalent grades of metal content without considering recovery.

$$\begin{aligned}\boldsymbol{NiEq}\% = {} & \boldsymbol{Ni}\% + \boldsymbol{Cu}\% \times \boldsymbol{Cu}\$/\boldsymbol{lb} \,/\, \boldsymbol{Ni}\$/\boldsymbol{lb} + \boldsymbol{Co}\% \times \boldsymbol{Co}\$/\boldsymbol{lb} \,/\, \boldsymbol{Ni}\$/\boldsymbol{lb} + \boldsymbol{Pt}\,[\boldsymbol{g/t}]/\mathbf{31.103} \\ & \times \boldsymbol{Pt}\,\$/\boldsymbol{oz}\,/\boldsymbol{Ni}\$/\boldsymbol{lb}\,/\mathbf{22.046} + \boldsymbol{Pd}\,[\boldsymbol{g/t}]/\mathbf{31.103} \times \boldsymbol{Pd}\,\$/\boldsymbol{oz} \,/\, \boldsymbol{Ni}\,\$/\boldsymbol{lb}\,/\mathbf{22.046} \\ & + \boldsymbol{Au}\,[\boldsymbol{g/t}]/\mathbf{31.103} \times \boldsymbol{Au}\,\$ \,/\, \boldsymbol{Ni}\,\$ \,/\, \mathbf{22.046}.\end{aligned}$$

$$\boldsymbol{Pt\ Eq}\,[\boldsymbol{g/t}] = \boldsymbol{Ni\ Eq}/\mathbf{100} \times \mathbf{2204.62} \times \boldsymbol{Ni}\$/\boldsymbol{lb} \,/\, \boldsymbol{Pt}\$/\boldsymbol{oz} \times \mathbf{31.103}$$

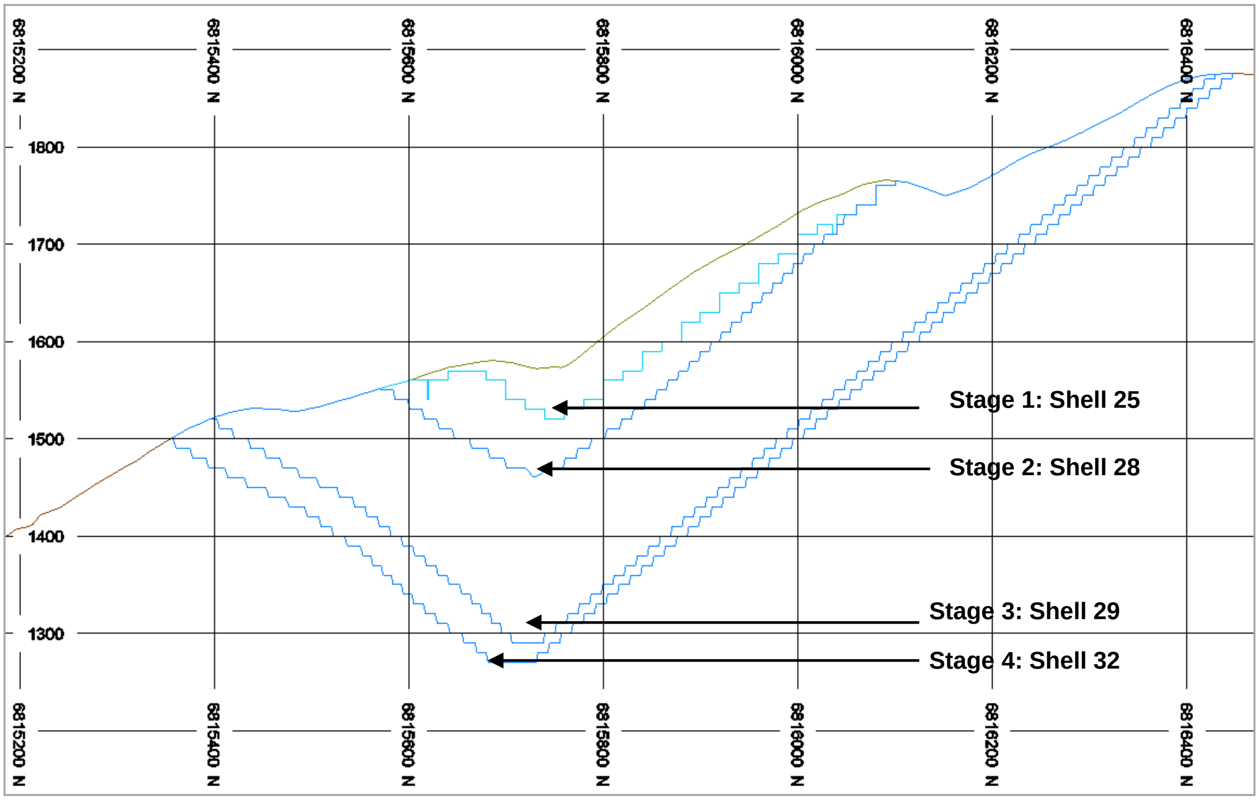
16.4 Open Pit Mine Development Sequence

As noted above, mining activities will be divided into four phases; each phase, or pushback, represents certain periods of mine life and satisfies all the mine design criteria. The mining pushback sequence achieves the following objectives:

- Targets the highest value material during the initial years of mine operations;
- Provides waste rock necessary for mine infrastructure construction, including the tailings storage facility; and
- Balances the overall trucking requirements.

The phases are shown in Figure 16.4.

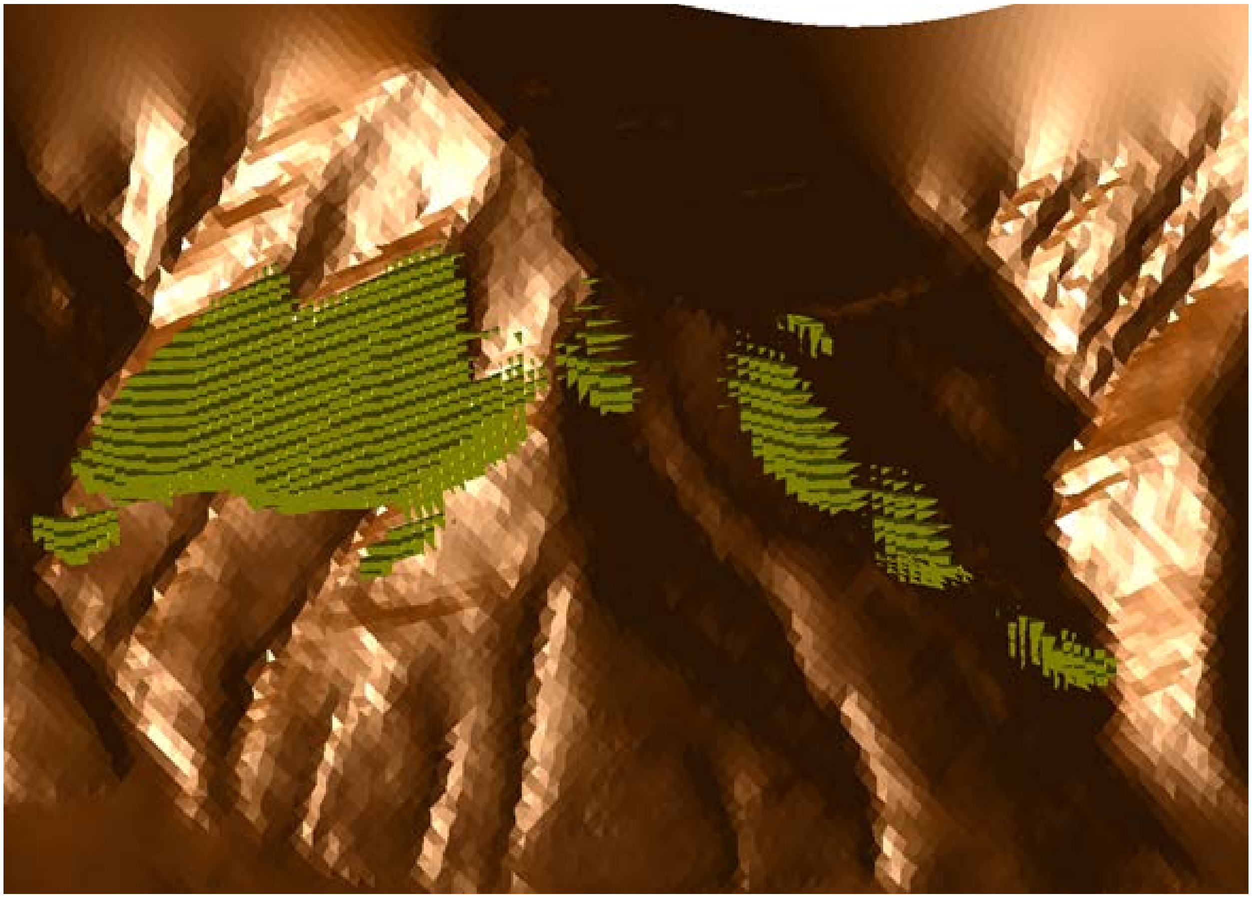
Figure 16.4: Four Stage Pit Cross Section



Source: SNC 2015

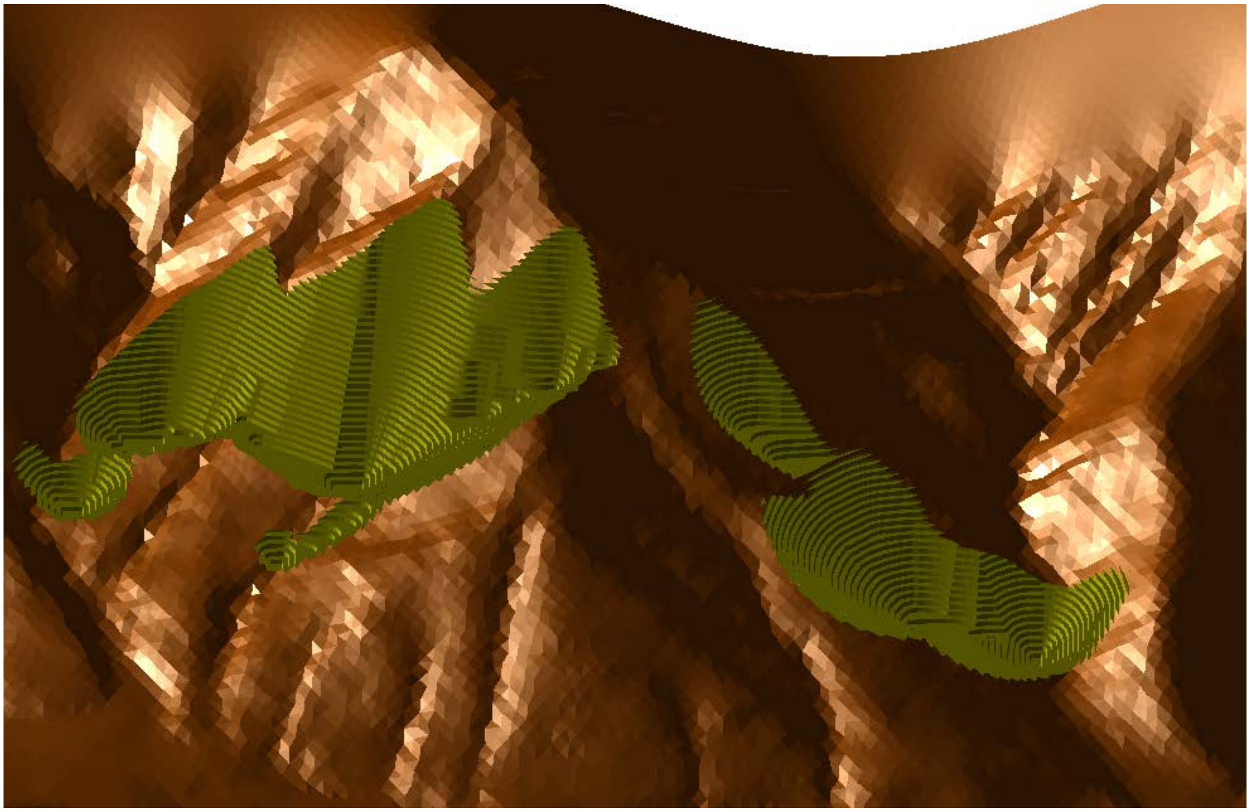
3D views of the final pit stages can be seen in Figures 16.5 through 16.8.

Figure 16.5: Phase 1 Isometric View



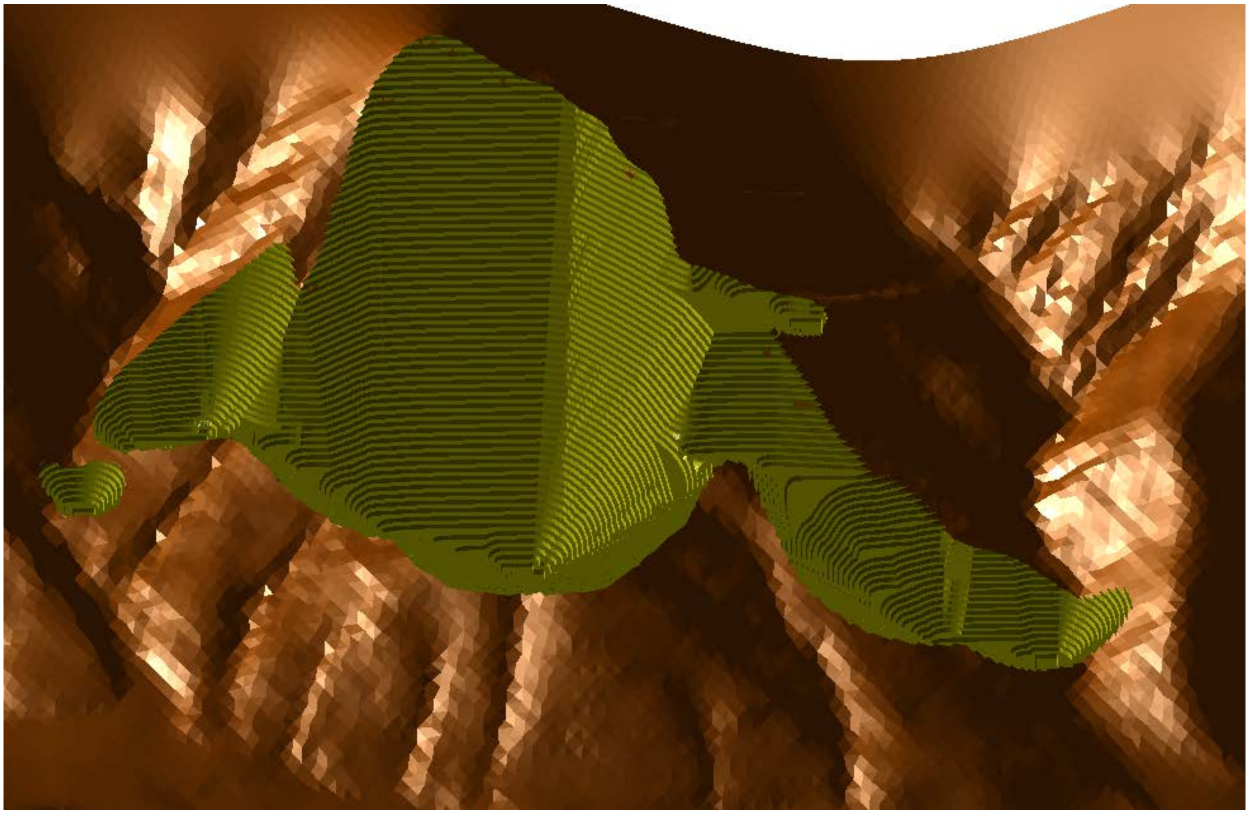
Source: SNC, 2015

Figure 16.6: Phase 2 Isometric View



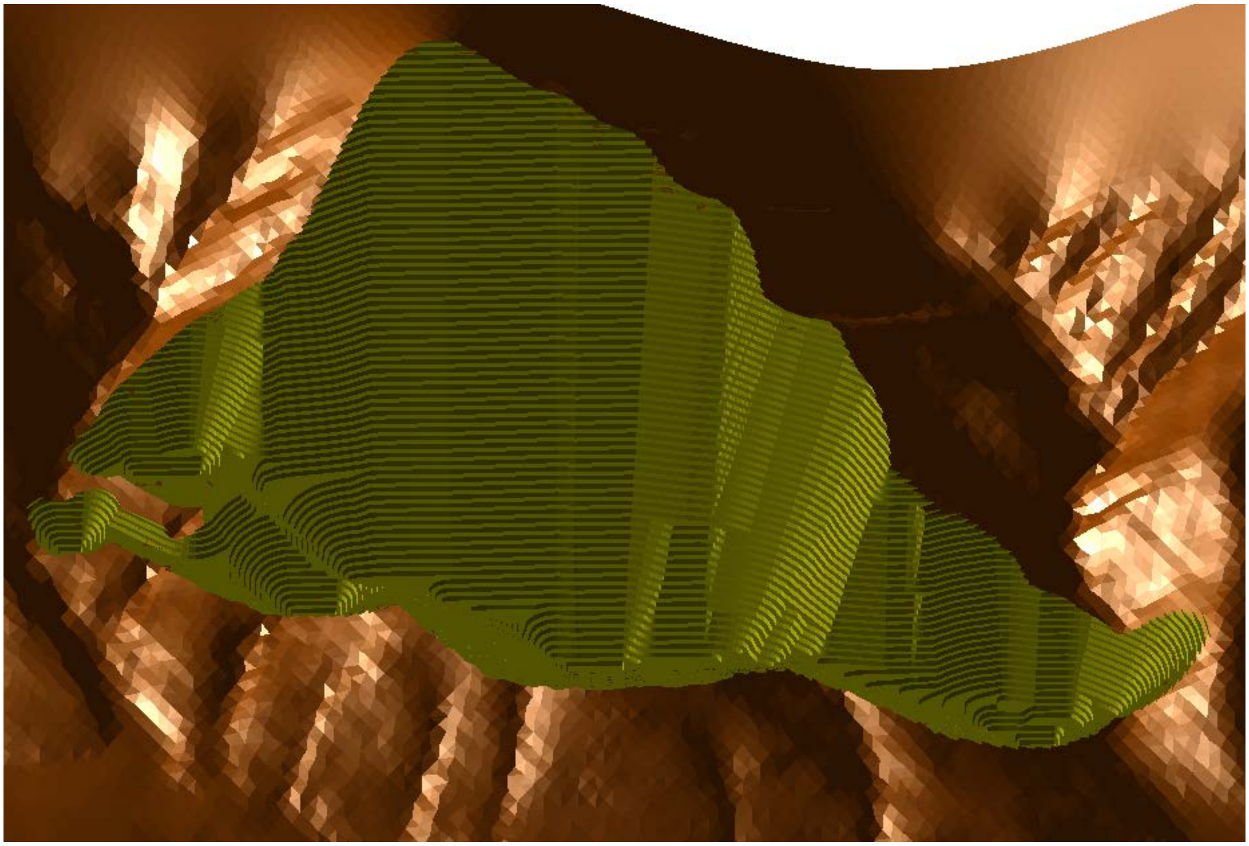
Source: SNC, 2015

Figure 16.7: Phase 3 Isometric View



Source: SNC, 2015

Figure 16.8: Phase 4 Isometric View



Source: SNC 2015

16.5 Waste Rock Storage and Long Term Stockpile

Waste rock storage facilities and long term stockpiles were designed by JDS and provided to SNC-Lavalin.

Waste material will be hauled to the 1540 West dump and the tailings dam facility. Waste rock dumps are designed at 22 ° face angle over an average dump height of 200m. An alternate dump was designed at the 1720 m elevation with capacity of 59 Mm^3 however it was not considered due to the longer haul cycle requirement. Total capacity for waste storage is 189 Mm^3.

The long term stockpile is designed at 26° with a capacity of 72Mm^3.

The layout of the dumps and stockpile can be seen in Figure 16.1.

16.6 Open Pit Geotechnical Criteria

SRK (2014) conducted a preliminary geotechnical assessment for the Wellgreen project to provide estimates of suitable pit slope angles for PEA-level mine planning. The preliminary assessment was based on information available at the time, including resource drilling data and core photographs, rock quality designation (RQD) data, Whittle pit shells, geologic models and relevant background reports. Specific geotechnical drilling and testing were not conducted as part of the assessment.

16.6.1 Rock Mass Characteristics

Estimates of rock mass characteristics were developed based on available resource core photographs and data obtained during the resource logging program. No geotechnical studies have been conducted to date for the Wellgreen project.

Based on the current geologic model and ultimate pit shell, the southern pit wall will have a maximum height of approximately 320 m and will be comprised mostly of the mafic-ultramafic intrusive package. It appears that a significant portion of the upper mafic intrusive complex (dunite, peridotite and clinopyroxenite) is highly altered and serpentinized which is expected to be of relatively low geomechanical quality, exhibiting high fracture frequency and low fracture strength. Based on core photographs, SRK estimates rock mass rating (RMR) values between 45 and 55 for the mafic-ultramafic intrusive unit, according to the Bieniawski (1989) system. Other potential challenges for the south wall include its close proximity to Nickel Creek and potential for elevated pore water pressures, as well as the nature of the fault contact between the Nikolai Formation basalt and mafic-ultramafic intrusive complex.

The final north pit wall will be high with a maximum slope height of approximately 660 meters and comprised mostly of the Hansen Creek metasedimentary rocks. Based on review of available information, the metasediment rock mass is of generally good geomechanical quality with relatively low fracture frequency and higher intact rock strength. SRK estimates RMR values between approximately 65 and 80 for the metasedimentary units based on review of core photographs. It does not appear from the core photo review that the metasedimentary units are anisotropic in strength or have a dominant fabric along relict bedding or foliation planes.

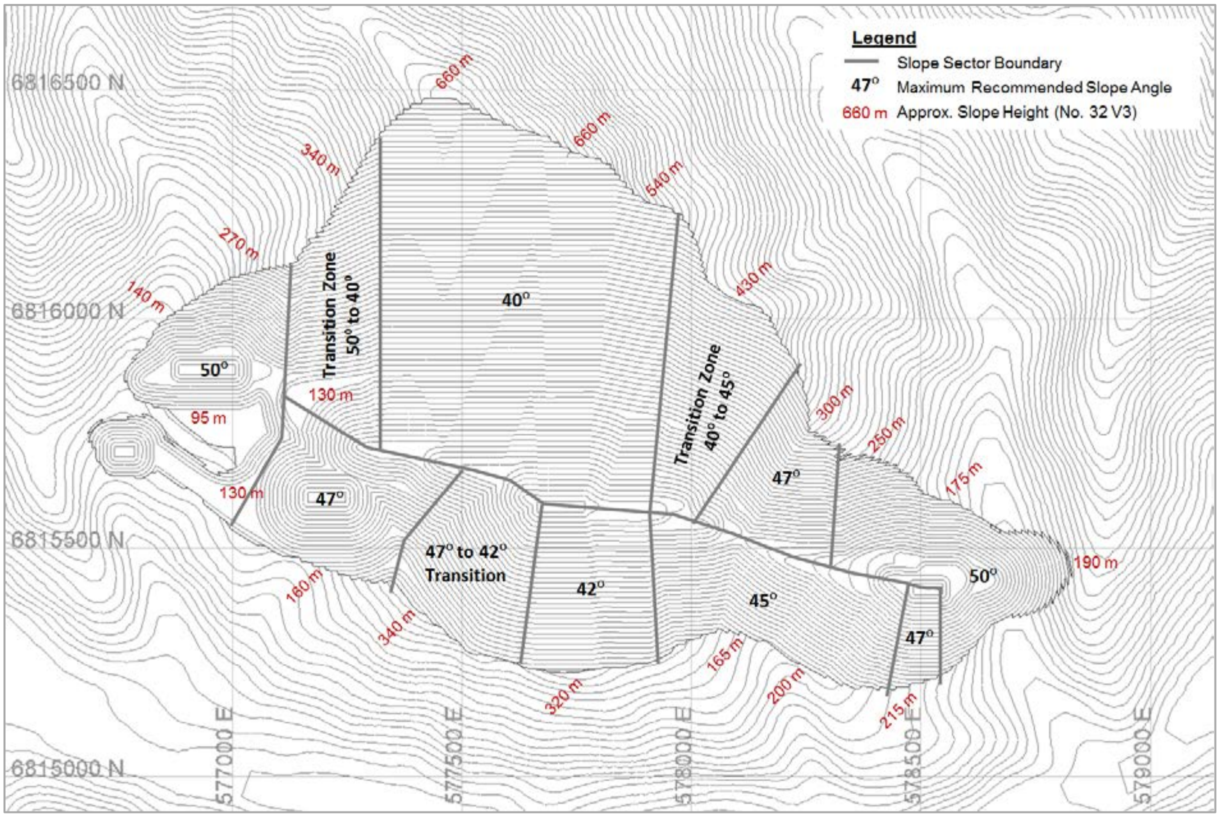
In addition to the mafic-ultramafic intrusives and metasediments, andesite and gabbro dykes also exist within the pit area. The dykes are expected to be of generally good rock quality but, given their low percentage of the overall rock mass, they cannot be relied upon for strength at this stage of investigation.

Hydrogeological conditions are not well known for the site; however, SRK understands that the exploration decline located within the central portion of the pit is flooded below the portal elevation (approximate elevation 1280 m) suggesting pit slopes will be at least partially saturated. The orientation and extents of major structures and jointing are also unknown at this time and may have significant impacts on achievable slope angles at later stages of project development.

16.6.2 Pit Slope Design Parameters

To allow steeper slope angles in areas with lower slope heights and minimize stripping to the extent possible, the pit was divided into individual slope design sectors based on slope height and dominant geology. Estimates of suitable overall slope angles were then developed for each of the individual sectors. The overall slope recommendations ranged between 40 and 50 °. The individual slope sectors and their respective recommended maximum slope angles are shown on Figure 16.9.

Figure 16.9: Recommended Slope Angles for PEA Mine Planning (Pit Shell No. 32 V3)



Source: SRK, 2015

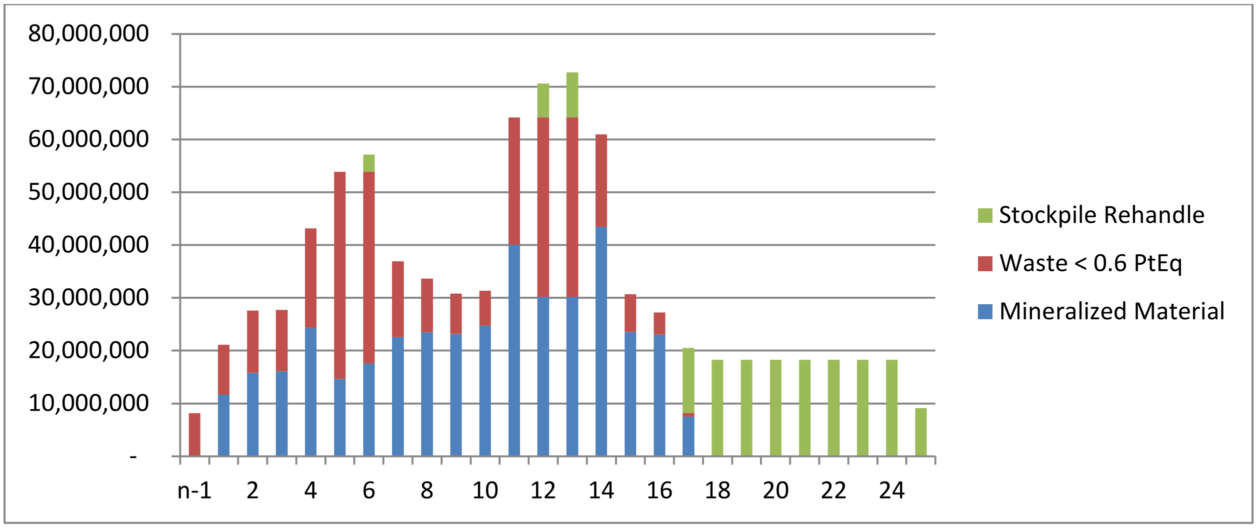
SRK believes the estimates of suitable pit slope angles are reasonable for the anticipated rock mass conditions based on available information and compared to other operating open pit mines. As with all PEA level assessments, recommendations for pit slope angles could change once actual geotechnical drilling and testing have been completed and dominant structural conditions are known.

16.7 Open Pit Production Schedule

The plant capacity commences with 25kt/day for the first five years, then ramps up to 50kt/day in year 6 and for the remainder of the Life of Mine Plan including processing of stockpiled mineralized material.

The pre-production period lasts for one year, mining 8.1Mt of material for construction of the tailings storage facility. Mining operations last approximately 17 years followed by eight years of processing stockpiled material. The mine production schedule is summarized in Figure 16.10

Figure 16.10: Mine Production Schedule



Source: JDS, 2015

In this study it is assumed that multiple pit phases can safely be mined concurrently to reduce the peak stripping requirement between pit stages. Future studies will consider this detail in pit phase designs.

In order to maintain a consistent open pit mobile fleet (and employee profile), contractor mining is required on occasions due to significant stockpiling requirement and tailings storage facility expansion requirements. Therefore, contractors are utilized in years 4 through 6, and 11 through 14 when mining rates exceed 37.8Mt/year. Contractor mining rates vary by year, but average 21.1Mt/year over the seven years. Contracting consists of drilling 400,000 m of 12-1/4" production blast holes in years 5, 6 and 11-16 with one electric drill and providing up to 20 trucks from years 4-6, 11-14 to enable haulage of 59 Mt of waste rock to the tailings storage facility and 89 million tonnes to the stockpiles.

16.8 Open Pit Mine Equipment Selection

Mining equipment was selected based on overall mine profile and production rate. The equipment work schedule is based on two twelve hour shifts operating seven days a week. The open pit mobile equipment list is summarized in Table 16.7.

16.8.1 Loading

SNC-Lavalin provided a comparison of electric and diesel hydraulic shovels and Wellgreen Platinum selected electric shovels as the primary loading equipment. SNC-Lavalin provided a cost comparison study which was the basis for Wellgreen Platinum's selection of electrical shovels. Electrical shovels have low operating costs when used with large benches and consistent mining conditions, while diesel hydraulic loading equipment can provide lower operating costs in dynamic mining conditions requiring frequent moves. A large, 35 tonne diesel loader will support the loading requirement as a backup loading unit.

16.8.2 Haulage

The selected haulage fleet consists of large 227 tonne trucks, such as the Cat 793 class. Wellgreen Platinum and SNC-Lavalin reviewed the potential utilization of LNG retrofitted haulage trucks with GFS Corp (Natural Gas and Conversion Systems). GFS Corp research indicates:

- Caterpillar and Komatsu trucks, including the Cat 793 and Komatsu 830E, can be converted from 100% diesel to natural gas and diesel operation;
- A complete solution can be provided, from engine conversion to LNG storage;
- Conversion maintains OEM engine and truck performance; and
- Conversion maintains 100% diesel capability, so there is no engine de-rate under load.

In addition, Caterpillar has conducted research with LNG fueled haulage trucks that indicated there is potential to attain a 16% savings on fuel expenditures by using LNG. For the purpose of this study, SNC-Lavalin reduced haul truck fuel consumption costs by 16%.

16.8.3 Drilling

Electric drills capable of drilling single pass 15 m benches were selected as the primary drilling equipment. The fleet consists of two large production drills capable of drilling holes between 10 5/8" to 16", depending on blasting requirements, and a smaller support drill capable of drilling 6 ½" holes when required for blasting selectivity or highwall control.

Table 16.7: Open Pit Mine Equipment List

Type	Initial Quantity Year -1	Peak Quantity
Truck	6	31
Electric Shovel	1	2
Rotary Drill	0	2
Rotary Drill	1	1
Loader	1	1
Dozer	2	3
Dozer	2	2
Grader	1	1
Water Truck	1	1
Hydraulic Excavator	1	1
Low Bed Truck	1	1
Fuel & Lube Truck	1	1
Mechanics Service Trucks	1	1
Bus	1	1
Submersible Pumps	2	2
Submersible Pumps	1	1
Pickups	10	10

Source: SNC, 2015

Equipment replacement hours were determined for the major mining equipment fleet based on industry standards and experience. Time usage models were developed to estimate equipment replacement based on gross operating hours. Equipment hours and replacement are summarized in Table 16.8 and Source: SNC 2015

Table 16.8: Equipment Replacement Hours

Equipment Type	Replacement Hours
Loader	60,000
Electric shovel	75,000
Electric drill	52,000
Trucks	60,000
Dozers	50,000
Graders	50,000

Source: SNC 2015

Table 16.9: Major Equipment Replacement Schedule

Year	Trucks Cat793	Shovel P&H2800	Loaders 994	Drills 351E	Drills 271E	Dozers D10	Dozers RTD	Graders 16H	Minor Fleet
n-1	6	1	1		1	1	2	1	
1	1			1		1	1		1
2	2								
3	1	1							
4	1			1					
5	0								
6	0								
7	2					1	2	1	
8	6					1	1		
9	2								
10	2								
11	1	1							
12	2			1	1				
13	0	1							1
14	0					2		1	
15	1					1			
16	0								
17	1								
18									
19									
20									
21								1	

Source: SNC, 2015

16.9 Underground Mining Overview

The objective of the underground mining program was to provide high grade mill feed early in the life of mine plan. This comes from zones that would otherwise not be mined until late in the 2015 PEA base case life of mine plan. This could also be part of the Stage 5 open pit opportunity.

The underground mine design takes advantage of existing level development, ventilation and vertical development. This provides feed to the mill starting in year 3 of production with a relatively low capital requirement.

The selection of the mining methods took into consideration the following factors:

- A historical review of underground mining that had occurred from 1970 to 1973 by Hudson Yukon Mining, a subsidiary of Hudson Bay Mining & Smelting. The mining methods utilized at that time were as follows:
- Post Pillar Cut and fill;
- Shrinkage; and
- Development material.

- Bulk mining methods that provide provisions for ground support programs in order to prevent a detrimental influence on open pit mining above the underground workings; and
- Hudson Yukon Mining methods that concentrated on extracting areas in the mineral body with high grade and also able to minimize dilution.

The current study reviewed the following four underground mining methods:

- Shrinkage mining, but was eliminated due to geotechnical concerns. These openings would affect open pit mining, which was scheduled to operate concurrently with the underground activities;
- Block caving was considered as an alternative to a Stage 5 open pit scenario;
- Open stoping with backfill was chosen for those blocks amenable to bulk mining; and
- Post pillar cut and fill was the mining method chosen for shallow dipping, high grade mineralization zones.

This study assumes that the lateral development and the post pillar cut and fill production mining will be completed by one contractor who will provide his own mobile equipment. This contractor will also be responsible for the remote mucking of the open stope. A second contractor will drill and blast the open stopes and install the ground support cable bolting. The second contractor will be required to provide his own mobile equipment and grouting pumps.

16.10 Underground Geotechnical Considerations

16.10.1 Existing Excavations

As noted above, existing excavations are those areas that were developed and mined from 1970 to 1973. Additional excavations were developed in 2011 for the purposes of an underground exploration drilling program. The diamond drill stations that were developed in 2011 remain in good condition.

Hudson Yukon Mining stope excavations were mainly in the Gabbro/Massive Sulphide geological domains. These excavations continue to be accessible. Access drifts tend to be located in these areas also. When the excavations did intersect peridotite areas, pony sets and square sets were used to provide significant ground support. These timber sets have since become unstable; however, the ground above the timber, from a general perspective, remains standing. In 2011, ground failures were supported using new timber sets and split set bolting.

It is noted that the rock fabric and orientation of field stresses are not detailed sufficiently to allow the development of a definitive mine plan which would include a detailed assessment of stope lengths, hydraulic fill specifications and pillar requirements. For the purposes of the 2015 PEA, normal mine parameters were used to determine extraction and dilution estimates.

From visual observations provided by Wellgreen Platinum personnel it was determined that both the vertical and the horizontal stresses continue to be at a low level. Therefore, as part of the 2015 PEA, the ground support system of bolting, mesh and shotcrete has been provided to prevent unravelling and support wedges. In 1970, the mine had used timber sets to maintain access to the lateral access drifts.

More information is required determine the complete ground support design to manage the stress regime as mine development to depth continues.

16.10.2 Previous Mining

As noted above, Hudson Yukon Mining developed and mined some of the high grade massive sulphide mineral bodies from 1970 to 1973 with shrinkage and cut-and-fill methods. Commercial production commenced in 1972. The mined mineralization was trucked down from the mine to the millsite near the current lower camp, beside the Alaska Highway. Production ceased in 1973 due to falling metal prices, and discontinuous massive sulphide horizons. A total of 171,652 tonnes grading 2.23% Ni, 1.39% Cu, 1.30 g/t Pt, 0.92 g/t Pd, 171 ppb Au, 0.40 g/t Rh, 0.42 g/t Ru, 0.25 g/t Ir, 0.20 g/t Os, and 0.20 g/t Re were milled to produce 33,853 tonnes of concentrate, which was shipped to Sumitomo in Japan.

16.11 Mineralized Zones

16.11.1 LH1 Zone

The LH1 Zone is located between 577990 and 578090 m E and ranges from 1050 to 1150 m in elevation. N-S extent direction ranges from 10 to 55 m. It lies entirely within the Pyroxenite/Clinopyroxenite domains. Classification is mostly indicated resources.

16.11.2 LH2 Zone

The LH2 Zone is tabular in shape, dips to the southeast, and is located between 578225 and 578460 m E. Elevation ranges from 775 to 1015 m and width varies from 240 to 390 m in the N-S direction. It lies almost entirely within the Gabbro/Massive Sulphide domain. Classification is predominantly inferred resources.

16.11.3 LH3 Zone

The LH3 Zone tabular in shape, dips gently to the south, and is located between 577500 and 577725 m E. Elevation ranges from 1,325 to 1,390 m and width varies from 215 to 360 m in the N-S direction It lies mainly within the Gabbro/Massive Sulphide domain and extends a short distance into the bordering clinopyroxenite. Classification is mainly inferred resources.

16.11.4 LH4 Zone

The LH4 Zone is tabular in shape dipping go the southwest and is located between 578325 and 578465 m E. Elevation ranges from 750 to 860 m and width varies between 60 and 160 m in the N-S direction. It lies almost entirely within the Gabbro/Massive Sulphide domain. Classification is predominantly inferred resources.

16.11.5 BH1 Zone

The BH1 Zone follows the sedimentary contact in the eastern portion of the mineralized zone near the base of Pit 32. It extends approximately 300 m E-W along the contact and ranges in elevation from 1,115 to 1,285 m. It is narrow in the eastern portion with a width of 10 m and expands to a maximum of 65m in the west-central portion. Approximately 70% of the contained blocks are within the Gabbro/Massive Sulphide domain and are classified as indicated and inferred. About 10% of the blocks are within the Clinopyroxenite domain and are mostly classified as measured resources.

16.11.6 BH9 Zone

BH9 is located between 578290 and 578345 m E and ranges from 1,037 to 1,080 m in elevation. N-S extent direction ranges from 28 to 43 m. All of the enclosed blocks are within the Pyroxenite domain and are classified as indicated resources.

16.11.7 BH10 Zone

BH10 follows the sedimentary contact in the east central portion of the mineralized zone near the base of Pit 32. It is located between 577880 to 578070 E and ranges from 108 to 1,340 m in elevation. The zone is generally narrow, ranging from 10 to 15 m in the N-S direction. The enclosed blocks are mostly indicated and inferred Gabbro/Massive Sulphide (72%). Approximately 20% of the blocks are assigned to the Clinopyroxenite domain and are classified as measured resources.

16.11.8 BH11 Zone

BH11 is located between 578125 and 578210m E and ranges from 1,160 to 1,260 m in elevation. N-S extent direction ranges from 15 to 50 m. It lies almost entirely within the Clinopyroxenite domain. Classification is measured and indicated resources.

16.11.9 BC2 - PEA Block Caving Opportunity

BC2 is not included as part of the 2015 PEA production plan. It is considered to be an opportunity that extracts a significant portion of the remaining resource and is an alternative to mining a large Stage 5 open pit. It is located in the eastern portion of mineralized zone beneath pit 32. Approximately 11% of the contained tonnes are in the Gabbro/Massive Sulphide domain and 85% within Pyroxenite/Clinopyroxenite.

BC2 is "flattened" at a base elevation of 750 m and extended to the 1070 m level. The footprint is 136,300 m2 and widths vary from 40 m on the east side to 395 m near the centre.

16.11.10 BC56 - PEA Block Caving Opportunity

BC56 is not included as part of the 2015 PEA production plan. It is considered to be an opportunity that extracts a significant portion of the remaining resource and is an alternative to mining the Stage 5 open pit. It is located in the western portion of mineralized zone beneath pit 32. Almost all (95%) of the contained tonnes are in the Pyroxenite/Clinopyroxenite domains.

The footprint is 27,395 m^2 and width varies from 46 m on the west end to 144 m near the centre. It has a flattened base elevation of 1130 m and has a vertical extent ranging from 136 m to 165 m.

16.12 Planned Underground Support

16.12.1 Primary Support

The ramp access development is planned at 5.5 m high x 5.0 m wide to provide clearances for the haulage trucks and to meet the mine ventilation requirements. For this level of study, all development was assumed to be the same size with the exception of the infrastructure excavations.

Typically, the primary ground support is installed using a mechanized bolting machine. This type of bolter is both safe and cost effective. There are standardized pattern bolting arrangements for use in good ground conditions. However, if poor ground conditions are encountered, a detailed ground control assessment is required to design a more rigorous support. This support must be developed on a case by case basis.

For this PEA study, the typical bolting pattern consisted of 1.8 m fully grouted rebar, installed on a 1.2 m x 1.2 m pattern in the back and down the shoulders. Wire mesh screen will be installed in each round. Along with a good scaling program, the screen will eliminate the hazard of any smaller pieces of loose rock falling from between the bolts.

16.12.2 Secondary Support

Secondary ground support will be installed in development headings where the spans of the planned openings have increased from standard drift size or if adverse ground conditions dictate further support is required. Local ground conditions in the stopes may be require longer or additional supports such as the application of shotcrete reinforced with fibre and/or utilization of split set bolts in the walls.

16.12.3 Stope Backfilling

Stope backfill is used to maintain the stability of the hanging wall and the backs of the lateral development. Currently, there is insufficient volume of waste rock available for the backfill requirements. Therefore, stopes are backfilled using hydraulic fill with cemented hydraulic fill which includes other additives such as silica gel (Gelfill). The silica gel is used to improve the dewatering efficiency, increase density and to provide additional strength. The use of hydraulic fill will also improve recovery and dilution, which has been taken into consideration in the mine production plan calculations.

By providing stope backfill, subsidence is prevented and minimizes the potential failures that could have detrimental impact on open pit mining operations that will be running concurrently with the underground operations.

The fill plant will be operated such that tailings required for backfill will be converted to thickened slurry and pumped to the mine for use as fill.

16.13 Underground Mining Method Selection

The mining methods considered in the 2015 PEA production plan include:

- Open Stope; and
- Post Pillar Cut and Fill.

16.13.1 Open Stoping

Open stoping longitudinal retreat provides high productivity at low mining costs from a small number of working faces. Longitudinal retreat also minimizes operating waste development since a scram drift with draw points is not required.

Engineered Cemented Hydraulic fill that incorporates the use of Silica Gel is planned to provide geotechnical stability, preventing occurrence of subsidence above the excavation. In addition, it provides a free standing consolidated fill surface that minimizes dilution of adjacent blocks when they are being extracted.

Sublevels will be developed at intervals of 15 to 20 m, depending on the mineralization geometry. The sublevel sill drifts will be 5 m high and initially 5 m wide and then slashed to the mineralization boundaries. The minerals sublevels will provide access for drilling, blasting, ground support and mineral mucking.

Normally in transverse stopes, the width will be over 20 m to footwall by 30 m length and 20 m high with a cycle time of 109 days, including hydraulic fill with an average of 400 t/d of muck per stope.

Slope cycle times are shown in Table 16.10.

Table 16.10: Stope Cycle Time

Activity	Duration	Units
Backfill Barricade Removal	0	days
Bottom Sill Slash	0	days
Raisebore Preparation	0	days
Raisebore Pilot And Ream	0	days
Drilling Preparation	2	days
Drilling	8	days
Loading Preparation	2	days
Loading	9.5	days
Mucking Preparation	1	days
Mucking	42	days
Backfill Barricade Preparation	1	days
Backfill Barricade Construction	2	days
Backfill Barricade Cure	2	days
Plug Filling and Curing (Hydraulic Fill)	5	days
Backfill Body (Hydraulic Fill)	4	days
Backfill Interference Allowance	2	days
Backfill Body Cure	28	days
Total	**108.5**	**days**

Source: SNC, 2015

The mechanical availability will be 75% and the drills are capable of drilling 250 m per day. Maintenance will be required after drilling two stopes, so spare drills will have to be factored in.

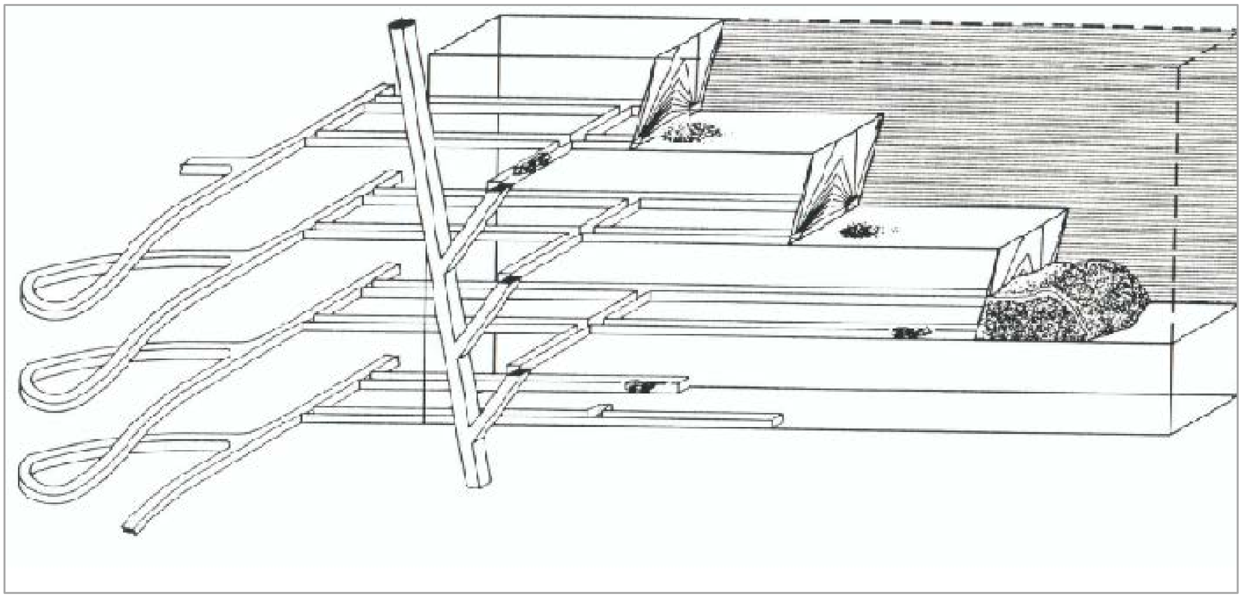
Blasthole drilling will use top hammer drills to drill 15 to 20 m long up holes from the extraction sill drift and, on other occasions, down holes from the upper sill to the lower extraction level. The maximum length of up holes is 20 m.

Down holes are projected to be 4.5” diameter, while the up holes are projected to be 3” diameter. The blast holes are drilled on a 2.5 m burden by 2.0 m spacing pattern and will be charged with EMMULSION and high explosive boosters and initiated with NONEL caps. A 0.50 kg/tonne powder factor has been assumed for LH blasting.

The broken mineral will be mucked from the bottom of the stope by remote control Load Haul Dump units (LHDs), loaded into trucks and hauled to surface. The mined out stopes will then be backfilled.

For the purposes of the 2015 PEA, a grade factor of 80% was assumed for open stoping with an extraction factor of 65% being applied. The extraction factor takes into account the requirement for rib pillars, whereas the grade factor takes into account dilution created by hanging wall sloughing and hydraulic fill.

Figure 16.11: Typical Open Stoping Schematic



[i] Source: Open Stope Mining in Canada. Australasian Institute of Mining and Metallurgy

Open stope mineralized zones are shown in Table 16.11 the table provides details of the open stoping mineralized areas.

Table 16.11: Open Stope Mineralized Zones

Area	Volume (Mm^3)	MTonnes	Grade Factor	Extract Factor	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
LH1_8050v2	0.29	0.86	80%	65%	0.25	0.486	0.019	0.766	0.477	0.23
LH3	0.93	2.80	80%	65%	0.657	0.61	0.025	0.949	0.823	0.136
LH2v3	2.14	6.54	80%	65%	0.525	0.441	0.028	0.656	0.475	0.077
BH1v2	0.38	1.15	80%	65%	0.295	0.539	0.019	0.583	0.32	0.151
BH9	0.04	0.11	80%	65%	0.409	0.258	0.018	0.433	0.509	0.065
BH10v2	0.36	1.09	80%	65%	0.515	0.517	0.023	0.602	0.51	0.138
BH11	0.13	0.36	80%	65%	0.394	0.209	0.017	0.32	0.464	0.058

Source: SNC, 2015

16.13.2 Mechanized Post Pillar Cut & Fill

Stope LH4v2 is planned to be extracted using mechanized Post Pillar Cut and Fill (MCF) mining because this mineralized zone is a high grade shallower dipping area. Table 16.12 provides the details of the LH4v2 zone.

Table 16.12: Post Pillar Cut and Fill Mineralized Zone

Area	Volume (Mm^3)	MTonnes	Grade Factor	Extract Factor	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
LH4v2	0.46	1.41	95%	75%	0.265	0.62	0.019	0.582	0.339	0.218

Source: SNC 2015

MCF is a lower productivity, higher cost mining method than open stoping, but provides highly selective mining with minimal dilution. Stopes can be sized with irregular backs and walls to match the mineral boundaries.

Drilling will be done by a two-boom electric hydraulic jumbo with an advanced rig control system that provides high quality drilling and blasting techniques that improves ground control and minimizes dilution. Each blast is projected to be four metres in length. After each blast, the area will be washed and scaled as well as bolted by a mechanical bolter fitted with a water scaler. Bolting will be with six foot rebar and Superswellex as secondary support when required.

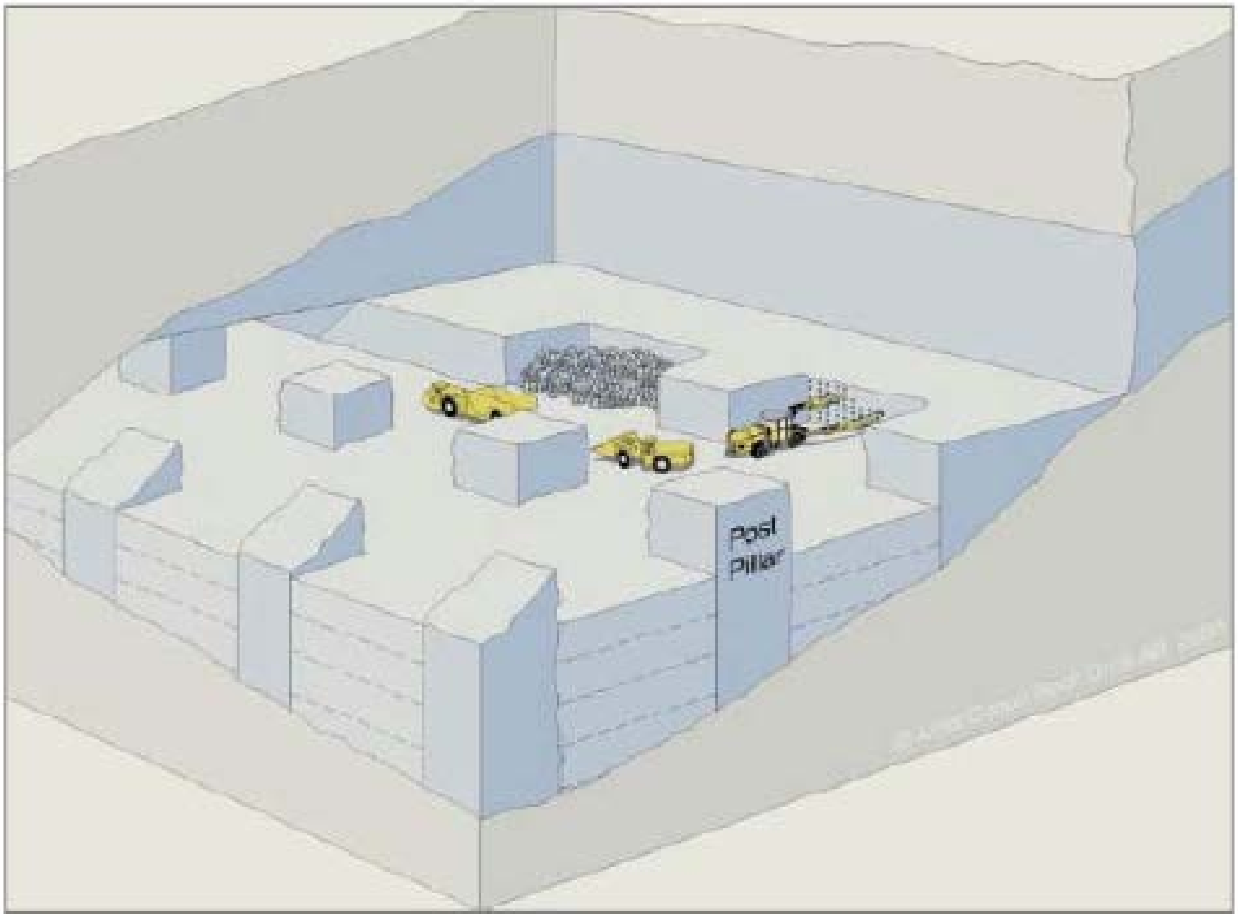
The broken mineral will then be mucked with LHDs into trucks and hauled to surface. The completed five metre high stope is then filled with hydraulic backfill or development waste. The last three feet of the pour is completed using engineered cemented fill to ensure that the next cut is established on a stable floor.

Once mining of the initial lift is completed, the ramp will access the next cut down and lift bench. The mined out area will then be filled with waste rock and hydraulic fill. To start the next lift, the access ramp would be slashed (breasted) at an appropriate gradient, up to plus 17%, to gain required elevation. The breasted waste rock would be left in place and is used as a ramp. Once the ramp is re-established, Post Pillar Cut and Fill mining would begin again working off of the waste rock backfill.

Haulage will be done with 50 t capacity trucks up the main decline to surface.

Post Pillar Cut and Fill requires a sufficient number of headings to attain a high production rate and efficient cycling of drill & blast, muck, scale/bolt/screen. In addition, the pillars from each cut must be positioned over the previous cuts to maintain maximum strength. Shotcrete posts will be utilized on an as required basis with appropriate instrumentation that monitors total load on the post.

Figure 16.12: Typical Post Pillar Cut and Fill Schematic



Source: ATLAS Copco Rock Drills AB, 2000

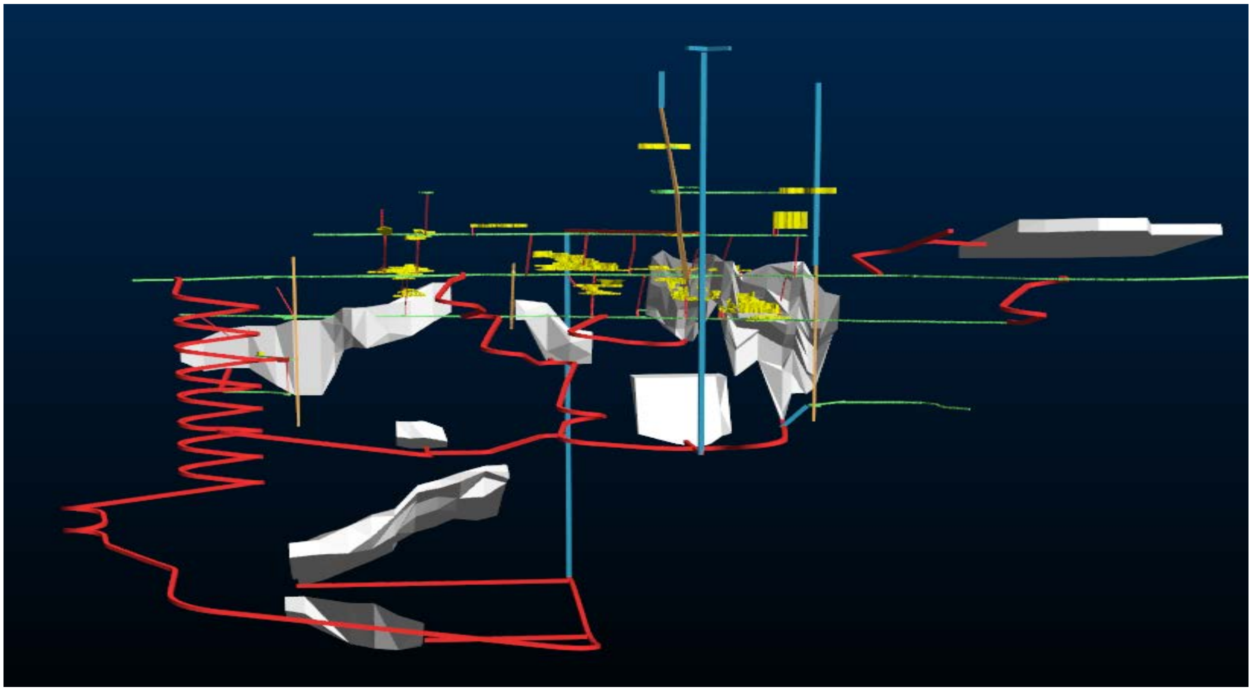
The Post Pillar Cut and Fill method is highly selective with a 95% grade factor applied. The extraction factor of 75% is lower due to the requirement for pillars, which are normally not recovered at a later date.

16.14 Underground Mine Design

16.14.1 Model Description

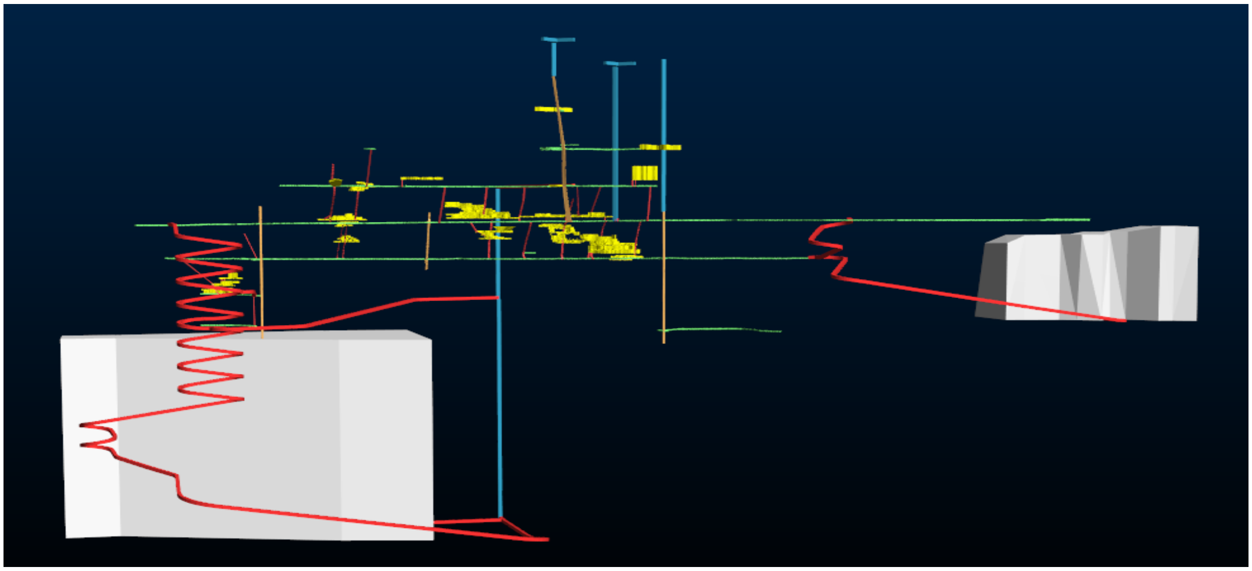
A 3D wire frame model of the underground was prepared in 5DP software to facilitate mine design.

Figure 16.13: Wellgreen Mine Design



Source: SNC, 2015

Figure 16.14: Block Caving Opportunity



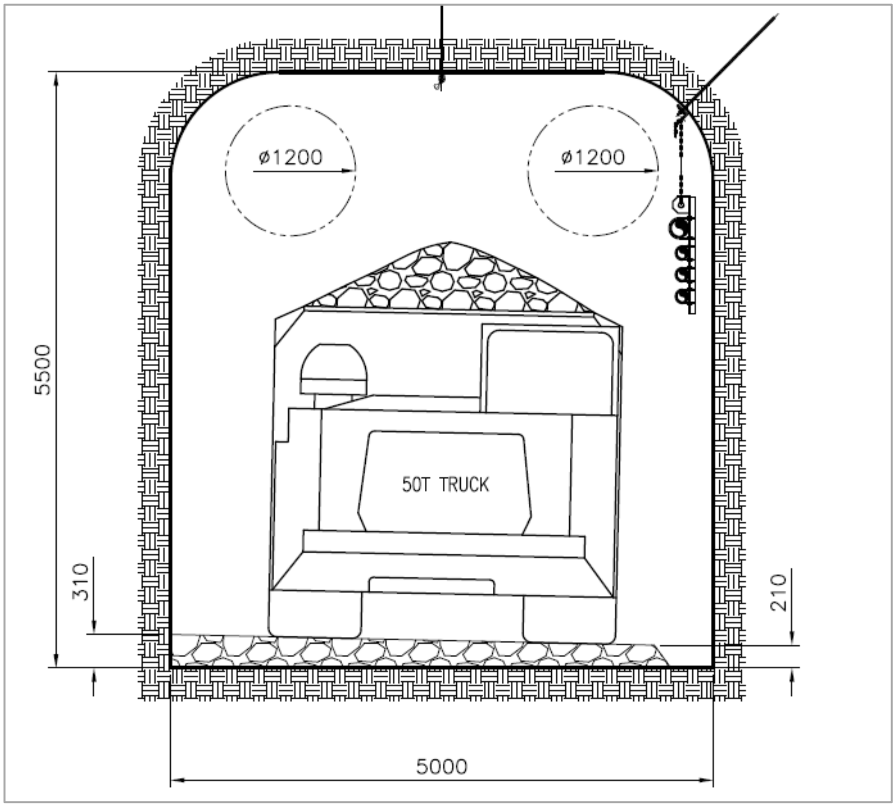
Source: SNC, 2015

16.14.2 Underground Access

Primary underground access will be through the existing mine portal using the existing level that will be re-conditioned for travelling and movement of material. All new lateral development ramps will be

5.5 m high and 5 m wide. The ramps will be driven at 18% to the bottom of the mine and will interconnect with other mineralization zones and old level workings. Small internal ramps will be driven, connecting old working and the mineralization zones.

Figure 16.15: Section of a Typical Development Profile



Source: SNC, 2015

16.14.3 Stope Access

The access drives will be driven from the new ramps to the working level. Level intervals will depend of the stoping application. Stope access is planned to be at 5.0 m wide by 5.0 m high.

16.14.4 LOM Development Requirements

This section presents an estimate of the development requirements for the LOMP. This information is used to support the mine capital and operating cost estimates in Section 21 and the development schedule.

Table 16.13 summarizes the scheduled mine development.

Table 16.13: Schedule of Mine Development Summary

Description	Unit	Total Qty
Ramp Development	metres	5,860
Level Development	metres	1,400
Rehab Development	metres²	14,400
Vertical Development	metres	1,135

Source: SNC, 2015

16.15 Underground Mine Production Schedule

Mine production tonnes were calculated by applying applicable recovery and dilution factors to resource estimates. Wireframe models of the individual zones were reviewed to determine the appropriate mining method with the associated recovery and dilution factors. Although detailed stope designs were not completed, the zones were reviewed to determine ventilation requirements, optimize the access ramp location and determine mobile equipment requirements. The overall mine production schedule was prepared based on individual production rates of each zone.

The following principal tasks were evaluated for each underground mineralized zone:

- Mine Method;
- Waste Development;
- Primary and Secondary Ground Support;
- Capital Infrastructure;
- Production Mucking;
- Ore and Waste haulage; and
- Backfilling.

16.15.1 Mine Production Rate

Development and production rates for the open stoping and post pillar cut and fill zones were developed by averaging production from four high level estimating processes:

- Taylors Rule;
- Long & Taylor estimates;
- The Half Vertical Tonnage; and
- AMC Estimate.

Table 16.14 summarizes the average production rate for each area.

Table 16.14: Average Production Rate for Each Zone

Average Production Rate		
Area	**Daily Tonnes**	**Years**
LH1_8050v2	531	2.9
LH3	6,851	0.7
LH2v3	2,149	5.4
LH4v2	1,599	1.8
BH1v2	811	2.5
BH9	168	1.2
BH10v2	503	3.9
BH11	298	2.2

Source: SNC, 2015

16.16 Underground Mine Plan

The LOM schedule was prepared together with the Open Pit schedule by prioritization of the mineral production from separate mining zones and combining the zones for an overall mine schedule.

16.16.1 Schedule Mine Productivities

Scheduled shifts in the mine will be 12 hours. Productive time per shift is assumed to be between 8 and 10 hours due to delays such as travel time, blast clearing times, daily preventative maintenance on equipment, unscheduled equipment downtime, and lunch breaks. The productivity estimations of the LHDs and mine trucks were determined from first principles cycle times using information in the equipment manuals and inputting travel distances to the load out locations.

16.17 Underground Mining Equipment

The list below shows the mobile equipment requirements for underground mining, which are to be provided by the contractor:

- Mobile Equipment;
- Trucks 50T;
- LHD ST14;
- Production Drills (ITH);
- Development Drill Jumbo;
- Secondary Breaking Drill (Commando);
- Cable Bolting;
- Bolters;
- Grader;
- Emulsion Loading System;
- Scissorlift;
- Boom Truck;

- Integrated Tool Carrier;
- Backhoe-Loader;
- Concrete/Shotcrete Transit Mixer Truck;
- Shotcrete Truck;
- Pickup;
- Service Truck;
- Electrician Truck;
- Lubrication Truck;
- Fuel Truck;
- Emergency Vehicle;
- Personnel Carrier (Bus);
- Personnel Carrier; and
- Cleanup/Construction LHD $3m^3$ bucket.

For safe operations, secondary breakers were added to eliminate mineral flow blockages and release trapped reserves above the draw point brow safely and effectively. These machines provide a solution for attacking high boulder hang-ups without endangering the mine workers.

The 50 t trucks with $7m^3$ LHDs were sized to meet the production rate that is required for the underground mining operations.

16.18 Underground Mining Personnel

Table 16.15 shows the average underground personnel requirement on site at full production.

Table 16.15: Average Mining Personnel On-Site

Description	Max Qty	Units	Schedule
Labour-Staff		Burden	25%
Mine Supervision			
Contractor Mine Captain	2	Persons	4x2
Contractor Mine Shift Supervisors	6	Persons	4x2
Mine Maintenance			
Contractor Maintenance Shift Supervisors	3	Persons	4x2
Contractor Maintenance Planner	3	Persons	4x2
Contractor H&S and Trainer	2	Persons	4x2
Engineering and Geology			
Senior Mine Engineer	2	Persons	2x2
Mine Engineers	2	Persons	2x2
Senior Surveyor	4	Persons	2x2
Mine Technicians	4	Persons	2x2
Mine Geologists	3	Persons	2x2
Ground Control Engineer/Vent/ Backfill	3	Persons	2x2
Staff Sub-total	**34**		
Contractors		Bonus	10%
Mine Operations -- Contractor			
ITH Operators (7-2 Drills)	21	Persons	4x2
Jumbo Operators	6	Persons	4x2
Scoop Operators	15	Persons	4x2
Truck Drivers	30	Persons	4x2
Blasters	9	Persons	4x2
Bolters	9	Persons	4x2
General Labourers / Construction	12	Persons	4x2
Backfill Plant Operators	12	Persons	4x2
Hourly Sub-total	**114**		
Mine Maintenance -- Contractor			
Mechanics and Welders	17	Persons	4x2
Servicemen	6	Persons	4x2
Electrician	9	Persons	4x2
Labourers	6	Persons	4x2
Hourly Sub-total	**38**		
Grand Total	**186**		

Source: SNC, 2015

The mine operating schedule is based on two 12-hours shifts per day, 365 days per year, with stope mucking and haulage crews working on holidays. The proposed mine roster will be a combination of contractors and company personnel. The hourly crews will be working on a 4-weeks on and 2-weeks off rotation and the salaried departments, such as administration services and technical services, will work a 2-weeks on and 2-weeks off rotation.

16.19 Underground Mining Support Services

16.19.1 Mineral and Waste Handling

During the project period and early production stages of the mine, both mineral and waste rock will be hauled to surface using 50 t mine rock trucks. During steady state production, a high percentage of the waste will be dumped into the stopes using LHDs and trucks, with the excess waste hauled to surface.

A surface stockpile will be established approximately 200 m from the portal. A crushing contractor will look after crushing at the mine and transportation to the mill.

16.19.2 Ventilation

The initial ventilation circuit has fresh air descending into the mine from surface ventilation raise through an existing raise and extended raise, and exhausting through the main ramp, old workings and RAR raise. The plan is to establish Fresh Air Raise (FAR) and Return Air Raise (RAR) fan houses and install four 500 HP fans to supply approximately 800,000 cubic feet per minute (cfm) to each major working area. The fresh air will be pushed down through raises and pick up in strategic bulkheads established on the existing levels for re-distribution to the planned working. Ventilation raises, ramp connections and between levels will be done to reduce pressure and improve the ventilation system.

The planned workings will be ventilated using collapsible ducting, steel ducting and appropriately sized booster fans and auxiliary fans. All fans are planned to be equipped with soft-starts, variable-frequency drive (VFD) and may be controlled remotely. Air flows in each zone will be controlled using ventilation raise/regulator set ups at the extremity of the levels. As the mine production ramps, internal and surface Fresh Air Raises will be constructed to provide sufficient air to all working faces (approximately 83,000 cfm). A Return Air Raise will be constructed and will be pulling air from the mine (approximately 850,000 cfm). The exhaust will be via the old workings, portal and return air raises.

A ventilation on demand system is planned to be installed to improve efficiency and decrease operating costs.

At the request of Wellgreen Platinum, the ventilation will not be heated during the cold weather season. Instead, a brine system will be installed, similar to that utilized at the Raglan Mine which was commissioned in 1996 and continues to be utilized. The freezing point of calcium chloride brine can be lowered to -51°C if the concentration of calcium chloride ($CaCl_2$) is correct. Brine is typically made on surface and transported underground. It would be prepared using batch mixing in a tank with an agitator or using a brine saturator to create brine from bulk salt.

It is recommended to keep all $CaCl_2$ brine at an average concentration of 20% for the following two reasons:

- To resist sudden temperature drops. If the brine is designed for a warmer temperature and the temperature unexpectedly drops, even for a short period of time, pipes can be blocked or damaged. Therefore, in the colder months the concentrate would be increased above 20%; and

- To resist corrosion. CaCl2 brine is most corrosive to metal at concentrations between 2% to 6% and is much less corrosive at higher concentrations.

Once mixed, the brine would be stored in a surface storage tank, allowing for large batches to be made at one time. The brine could then be distributed underground through pipes.

All mobile equipment would be fitted with heated, enclosed cabs to help protect workers from exposure to low temperatures. These operating conditions are similar to those at other underground mines in Canada.

In terms of health and safety, many companies operating in the arctic have used brine systems in the past, including the Raglan Mine in Northern Quebec which has been using brine since 1997. The Quebec Ministry of Labour completed studies regarding vapours and other aspects with minimal concerns. This information will be requested by the site in order to prepare training programs and ensure proper systems and personal protective equipment is in place before the use of brine commences.

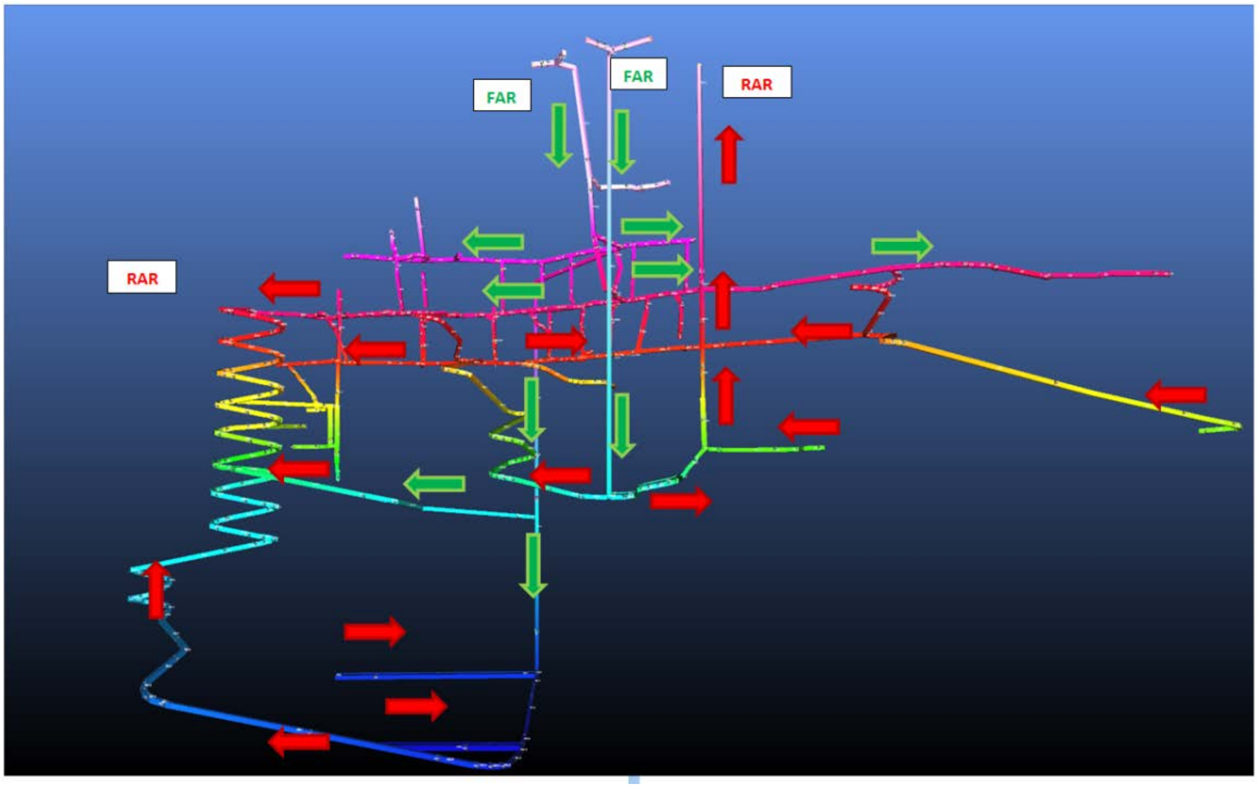
Table 16.16 presents the ventilation requirements based on the planned underground fleet. A factor of 100 cfm was used in total air flow calculation and sizing of the fans.

Table 16.16: Ventilation System

Main Systems	Function	Intake	Outlet	Geometry	Infrastructure
FAR1	Intake (Forced)	Surface up (old Raise)	Over Ramp Level into New Internal Fresh raise to LH2-LH4	4 m diameter and 5x5 m raise connection	2x500HP
FAR2	Intake (Forced)	Surface up (new raise)	LH1	5.2 m Diameter	2x500HP
RAR1	Exhaust	BH10 and under ramp level	Surface up	6 m Diameter	3x150HP
Ramp	Exhaust	LH2-LH4	Ramp level (surface lateral)	5.5 x 5.0m	Regulators

Source: SNC, 2015

Figure 16.16: Ventilation Schematic including Block Caving Opportunity



Source: SNC, 2015

Table 16.17: Equipment Detail for Ventilation

Equipment Detail	Units	Qty	HP	kW	Utilization	Total HP	Total kW	m^3/s	cfm
Production/Development Jumbo (2 boom)	ea.	7	99	74	10%	69	52	3.1	6,930
Rockbolter	ea.	3	74	55	20%	44	33	1.99	4,440
Production/ Development Load-Haul-Dump, 10 t	ea.	5	325	242	80%	1,300	969	58.16	130,000
Haulage Truck, 50 t - Production/ Development	ea.	10	600	447	80%	4,800	3,579	214.76	480,000
Grader	ea.	1	200	149	30%	60	45	2.68	6,000
Explosive Truck	ea.	1	128	95	20%	26	19	1.15	2,560
ANFO Loader	ea.	1	128	95	30%	38	29	1.72	3,840
Cassette Carrier	ea.	1	150	112	50%	75	56	3.36	7,500
Mechanics Truck	ea.	1	150	112	25%	38	28	1.68	3,750
Scissor Lift	ea.	2	150	112	25%	75	56	3.36	7,500
Supervisor/Engineering Vehicle	ea.	1	128	95	20%	26	19	1.15	2,560
Electrician Vehicle - Scissor Lift	ea.	1	128	95	30%	38	29	1.72	3,840
Forklift/Tractor	ea.	1	85	63	20%	17	13	0.76	1,700
Total						6,606	4,926	296	660,620
Losses	%	20%						59	132,124
Total Ventilation Requirements								**355**	**792,744**

Source: SNC, 2015

16.19.3 Compressed Air and Water Supply

The underground mobile drilling equipment, such as jumbos, rock-bolters and emulsion loaders, are to be equipped with their own compressors. Therefore, no reticulated compressed air system is required underground.

16.19.4 Mine Dewatering

Water volumes from underground are expected to be a normal volume (i.e., observations of the existing workings do not indicate significant water inflows). The water usually attributed to drilling will be reduced when efforts are made to conserve and recycle the drilling brine.

The mine dewatering system includes the following sumps:

- A level or drain sumps;
- An intermediate sumps;
- Vertical cone sump;
- A main sump;
- A settling sump; and
- Ramp sumps.

The drilling equipment, such as jumbos, rock bolters and exploration drills, would use a brine system in winter months, as described in the Ventilation section.

16.19.5 Mine Electrical Distribution

The major electrical power consumption in the mine would be from the following:

- Main and auxiliary ventilation fans;
- Drilling equipment;
- Mine dewatering pumps;
- Air compressors; and
- Maintenance satellite shop.

High voltage cables would enter the mine via the decline and be distributed to electrical sub-stations located near the production stopes. The power cables would be suspended from the back of development headings. All equipment and cables would be fully protected to prevent electrical hazards to the personnel. The primary power to the mine will be 4.16kV 3ph and reduced to 600V in level electrical sub-stations. At the rooms, the power will be reduced to 120V service to power the lighting and convenience receptacles.

16.19.6 Diesel Fuel Supply and Storage

Haulage trucks, LHDs, and all auxiliary vehicles would travel to the surface and use surface bulk fuel storage stations. The drills and rock bolters would fuel themselves from Sat-Stat fuel stations.

16.19.7 Explosives Supply and Storage

Explosives would be stored on surface in permanent magazines. Detonation supplies such as NONEL, blasting caps, and detonating cords would be stored in a separate magazine. Underground powder and cap magazines would be prepared near Area 2 production stopes. The daily explosive requirements would be used as storage in Day Boxes.

Emulsion would be used as the major explosive for mine development and production. Packaged emulsion would be used as a primer and for loading lifter holes in the development headings. Smooth blasting techniques may be used as required main access development headings, with the use of trim powder for loading the perimeter holes.

During the decline development, blasting in the development headings would be done at any time during the shift when the face is loaded and ready for blast. All personnel underground would be required to be in a designated Safe Work Area during blasting. During the production period, a central blasting system would be used to initiate blasts for all loaded development headings and production stopes at the end of the shift.

16.19.8 Underground Equipment Maintenance

An underground satellite maintenance shop is planned in the existing shop location on the 1800 Level. The purpose of this shop is to do equipment service only.

Major maintenance projects such as equipment overhauls and significant repairs and welding would be completed at the open pit maintenance shop or at a temporary shop established by the contractor on surface.

16.19.9 Communications

An underground communication system will be established using a leaky feeder system with a head-end in the new office complex. The leaky feeder cable will be installed cable will be installed throughout the mine with repeaters on surface to extend communications to the entire site.

A leaky feeder communication system would be used as the communication system for mine and surface operations. Telephones will be located at key infrastructure locations such as the electrical sub-stations, refuge stations, and main sump.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) would be supplied with an underground radio for contact with the leaky feeder network.

16.19.10 Consumables

All the consumables required for mining activities have been included in the unit cost of the respective mining activity.

16.19.11 Underground Transportation of Personnel and Materials

All mine supplies and personnel would access the underground via the main access decline. Two personnel vehicles would be used to shuttle employees from surface to the underground work areas and back to the surface during shift changes. Supervisors, engineers, geologists, and surveyors would use diesel powered jeeps as transportation underground. Mechanics and electricians would use the mechanic service trucks and maintenance service vehicles.

A boom truck with a 10 tonne crane would be used to move supplies, drill parts, and other consumables from surface to active underground workings.

16.19.12 Mine Safety

Portable refuge stations would be provided in the main underground work areas. The refuge chambers are designed to be equipped with compressed air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers would be capable of being sealed to prevent the entry of gases. The portable refuge chambers would be moved to new locations as the working areas advance, eliminating the need to construct permanent refuge stations.

Every vehicle would carry at least one fire extinguisher of adequate size and proper type. It is also recommended that underground heavy equipment be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system would be installed at the main intake raise to alert underground workers in the event of an emergency.

The main access decline would provide primary access and the ventilation raises with dedicated man-way would be equipped with ladders and platforms providing the secondary exit in case of emergency.

16.19.13 Overall Production Schedule

The overall underground and open pit production schedule is show in Table 16.18.

Table 16.18: Mine Production Schedule

Parameter	Unit	LOM Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Open Pit																				
Mineralization	M tonnes	392.0	0.0	11.7	15.8	16.1	24.3	14.6	17.6	22.5	23.5	23.2	24.7	40.1	30.2	30.2	43.4	23.6	23.1	7.5
Waste	M tonnes	295.9	8.2	9.5	11.8	11.6	18.9	39.2	36.3	14.4	10.1	7.6	6.6	24.1	34.0	34.0	17.5	7.1	4.2	0.7
Open Pit Strip Ratio	w:o	0.8	0.0	0.8	0.7	0.7	0.8	2.7	2.1	0.6	0.4	0.3	0.3	0.6	1.1	1.1	0.4	0.3	0.2	0.1
Total Open Pit	M tonnes	687.9	8.2	21.1	27.6	27.7	43.2	53.9	53.9	36.9	33.7	30.8	31.4	64.2	64.2	64.2	61.0	30.7	27.3	8.2
Underground Mineralization	M tonnes	9.5	0.0	0.0	2.5	2.5	1.8	1.0	0.9	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Mineralization Mined	**M tonnes**	**401.5**	**0.0**	**11.7**	**18.3**	**18.6**	**26.2**	**15.6**	**18.5**	**23.2**	**23.5**	**23.2**	**24.7**	**40.1**	**30.2**	**30.2**	**43.4**	**23.6**	**23.1**	**7.5**
Total Waste Mined	**M tonnes**	**295.9**	**8.2**	**9.5**	**11.8**	**11.6**	**18.9**	**39.2**	**36.3**	**14.4**	**10.1**	**7.6**	**6.6**	**24.1**	**34.0**	**34.0**	**17.5**	**7.1**	**4.2**	**0.7**
Total Mined	**M tonnes**	**697.4**	**8.2**	**21.1**	**30.1**	**30.2**	**45.0**	**54.8**	**54.8**	**37.6**	**33.7**	**30.8**	**31.4**	**64.2**	**64.2**	**64.2**	**61.0**	**30.7**	**27.3**	**8.2**

Source: SNC, 2015

17 Recovery Methods

17.1 Process Description

Process plant design uses a variety of head grades and recoveries, depending on the circuit being sized. The expected recoveries used for mine design and financial analysis are summarized above in the section “Mineral Processing and Metallurgical Testing”.

Comminution will consist of a primary crusher, followed by two-stage crushing in closed circuit with screens. This will produce feed for a single stage ball mill. Flotation concentrate will be thickened and vacuum filtered to produce concentrate for shipment by truck. Flotation tails will be discharged to the tails facility. Testing indicates that there is potential for improved recoveries of metals, particularly PGMs, by magnetic treatment of the final flotation tails. The exact nature of this process has not been fully evaluated and has not been included in the economic evaluation at this time.

Testing indicates that a flowsheet to produce a bulk copper nickel concentrate, supplemented by magnetic recovery of flotation tails will maximize recovery of valuable metals. Testing indicates that early activation of the nickel is needed to achieve maximum possible nickel recoveries. Previous reports that detailed sequential flotation in intermediate stages of grinding have not been replicated.

Testing efforts to produce a separate copper concentrate using sequential flotation resulted in nickel recoveries up to 10% lower than those realized from bulk flotation. In addition, bulk flotation followed by copper nickel separation also had a detrimental impact on overall recoveries. Therefore, for the purposes of this PEA, bulk flotation producing a bulk concentrate for direct sale has been pursued.

17.2 Process Design Criteria

17.2.1 Availability

In determining equipment sizes, it is necessary to use the instantaneous throughput, rather than the nominal mill capacity. This is calculated by dividing the nominal capacity by the availability. In this case, availability is the percentage of time that a part of the process is actively processing material. Industry standards have been used in this PEA. It is recommended that this factor be more fully defined at the pre-feasibility study. Availabilities used are: Crushing=50%, SAG milling=90%, Single Stage Ball Mill=95% and Filtration=50%.

17.2.2 Tonnage Basis

Annual production at the expected concentrate grades and recoveries will be 9,125,000 dry metric tonnes. The facility operates 365 days per year; therefore, the mill must process, on average, 25,000 dry metric tonnes per day (mtpd). This is the nominal, or name plate, plant capacity. In order to achieve this with 95% availability, the design capacity of the mill will be 1,096 dry metric tonnes per hour (mtph). This is rounded up to arrive at the design tonnage rate of 1,100 mtph. This equates to a design capacity of 26,400 dry metric tonnes per day. This capacity is used to size equipment in the facility.

17.2.3 Primary Crushing

The primary crusher will be a 60 – 89, 600-kW gyratory crusher. Run of mine material will be trucked directly to the gyratory crusher. Mineralized material will normally be directed to the crusher feed stockpile. This stockpile is covered to reduce snow and rain addition. The gyratory crusher will produce 165 mm product (6.5") and be capable of receiving material up to 1.2 m in size. A rock breaker will be used to break up larger material delivered from the pit.

17.2.4 Crushing

Crushing to produce ball mill feed will be performed in two stages. Both stages will be MP1000 or equivalent crushers drawing 1,000 kW each. The first crusher will receive material from a screen that removes material suitable for tertiary crushing and material at final size. The product from the secondary crusher and the mid-sized material will combine with the product from the tertiary crusher. This material will be screened to produce tertiary crusher feed and final ball mill feed, directed to the fine material bin. The tertiary feed will fall into a tertiary feed bin. A feeder will allow choke feeding of the tertiary crusher. Final crushed product will be directed to fine material bins that will feed the grinding circuit. The fine material bins will have a total capacity for 12 hours of grinding operation. Bulk density of the mineralization at 50% voids will be 1.61 tonnes per cubic meter. For 12 hours of production, or 13,500 tonnes, the required volume is 21,000 m^3.

17.2.5 Grinding

Grinding will be by single stage ball mills. Two mills will be required at this tonnage rate. The mills will operate in parallel. It is possible to operate the mills in series as well with minor piping changes. Optimization of the layout will be considered at the pre-feasibility stage. The design p80 is 75 microns (μ). In Crushing plant product is projected to be 12,700μ. The design circulating load is 300% to produce a cyclone overflow at 30% solids. The mills will 7.6 m diameter by 10.4 m mill drawing 10.5 megawatts of power. The mill will operate in closed circuit with cyclones. Feed from the fine material bins will be directed to the mill discharge pump box to avoid over grinding. Cyclone overflow will be 75μ.

17.2.6 Rougher Flotation

Rougher flotation will be via a bank of four 300 cubic meter tank cells. This will provide the 24 minutes retention time and a carrying capacity of less than 1 mtph/m^2. This is considered conservative and hence the circuit will be able to respond well if feed grade increases. Each cell will draw 300 kW. Total flotation concentrate tonnages have been calculated using a head grade of 0.5% nickel and 0.5% copper with 80% recovery of both. This assures that the subsequent equipment will not be undersized. Total flotation tailings tonnages have been calculated using a head grade of 0.2% nickel and 0.2% copper with 60% recovery of both to assure that downstream equipment will not be undersized. Using these criteria for design purposes, the mass flow to flotation concentrate is 165 mtph (279 m^3/hr), and the mass flow to flotation tailings is 1,030 mtph (2,784 m^3/hr).

17.2.7 Magnetic Separation

Magnetic separation units have not yet been sized for the project. It is anticipated that four units per line will be needed. It is estimated that the units will draw 50 kW each. Magnetic separation tails (non-magnetic material) will be directed to final tails. The design mass flow for magnetic separation is the case where rougher concentrate mass is at a minimum. This results in a magnetic concentrate mass flow of 124 mtph (216 m³/hr).

Magnetic concentrate (magnetic material) will be reground. Testing to determine the optimum grind size has not yet been completed. For design purposes, 40 microns with a work index of 19.0 has been used. This will require a 4.1 m x 5.1 m ball mill drawing 1,200 kW.

17.2.8 Magnetic Concentrate Flotation

Magnetic concentrate will be floated in four 20 m³ tank cells drawing 20 kW each.

17.2.9 Regrind

Magnetic concentrate and bulk rougher concentrate will be reground in a 400 kW tower mill.

17.2.10 Cleaner Flotation

Cleaner flotation will be in three stages. The tails from the first cleaner are final tails. Each subsequent stage may return to the previous stage or to the magnetic concentrate flotation feed. Each stage will consist of four 10 m³ tank cells (total of 12) drawing 10 kW each.

17.2.11 Dewatering

Testing to confirm dewatering equipment sizes has not yet been completed. An allocation of 500 kW for dewatering equipment is added to the equipment list. This includes thickening, vacuum filtration and ancillary equipment. The estimated mass of concentrate to be dewatered is 2.5% of the fresh feed or 27.4 mtph (48.0 m³/hr). At this time, a thickener diameter of 30 meters will be used. This is considered conservative.

Process Design Criteria is shown in Table 17.1

Table 17.1: Process Design Criteria

Parameter	Value	Units	Notes
Operational Constraints			
Operating Days Per Year	365	days	No annual shutdown
Hours Per Day	24	hours	
Tonnage			
Annual Tonnage	9,125,000	dmt	Per line tonnage
Nominal Daily Tonnage	25,000	dmt	1 to 3 lines at this rate
Mineralized Material			
Specific Gravity	3.22	none	
Nickel Grade - Low Grade	0.20	% Ni	Low grades and recoveries used for sizing of equipment downstream of rougher tailings
Nickel Grade - High Grade	0.50	% Ni	
Copper Grade - Low Grade	0.20	% Cu	High grades and high recoveries used for sizing of equipment downstream of rougher concentrate
Copper Grade - High Grade	0.50	% Ni	
Crushing			
Gyratory Crusher Availability	50%		Maximum of two gyratory lines
Crushing Rate	2,083	mtph	

Parameter	Value	Units	Notes
Crusher OSS	165	mm	
Crusher Size	60-89		Based on METSO
Maximum Size to Crusher	1.2	m	
Motor Size	600	kW	
Capacity	4100	mtph	
Reduction Ratio	7.3		Max for gyratory = 8
Number at 25,000 mtpd	1		
Number at 50,000 mtpd	1		
Secondary Crushing Availability	50%		
Crushing Rate	2,083	mtph	
Crusher CSS	15.875	mm	5/8" CSS for 5/8" screen
Crusher Size	MP1000		Or equivalent
Maximum Size to Crusher	330	mm	
Motor Size	1000	kW	
Capacity	2000	mtph	
Reduction Ratio	10.4		In two stages
Number at 25,000 mtpd	2		1 standard/coarse, 1 shorthead/fine
Number at 50,000 mtpd	4		2 standard/coarse, 2 shorthead/fine
Grinding			
Ball Mill Circuit Availability	95%		
Ball Mill Tonnage Rate	1096.5	mtph	
Ball Mill Circuit Discharge (P80)	75	um	
Grinding Circuit Feed (F80)	12,700	um	80% of screen opening size
Ball Mill Circuit Circulating Load	300%		
Bond Ball Wi	19.0	kWhr/mt	metric
Required Product Size	75.0	um	
Mill Diameter	25.0	feet	
Mill Length	33.5	feet	
Mill Type	overflow		
Mill Power	10,500	kW	
Bulk Concentrate			
Rougher Concentrate Cell Capacity	1096.5	mtph	
Cyclone Overflow Solids	30.0%		weight / weight
Volume Flow Rate	2899.0	m^3/hr	
Rougher Concentrate Mass - A	65.8	mtph	Low grade and low recovery for tails equipment sizing
Rougher Concentrate Mass - B	164.5	mtph	High grade and high recovery for conc. equipment sizing
Rougher Tailings Mass - A	1030.7	mtph	
Carrying Capacity	1	$mtph/m^2$	First approximation
Retention Time	24	minutes	
Total Cell Volume	1159.6019	m^3	
Total Surface Area	65.8	m^2	Required to achieve carrying capacity
Number of Cells	4		
Per Cell Volume	289.90048	m^3	
Cell Diameter	5.7	m	

Parameter	Value	Units	Notes
Per Cell Area	25.5	m²	
Total Cell Area	101.8	m²	Sufficient Capacity for crowders
Rougher Concentrate Nickel Grade Low Grade	1	% Ni	Used to calculate concentrate mass for conc. equipment
Rougher Concentrate Nickel Grade High Grade	2	% Cu	Used to calculate concentrate mass for tailings equipment
Nickel Recovery To Rougher Concentrate - High Recovery	75	% to Conc	Used to calculate concentrate mass for conc. equipment
Nickel Recovery To Rougher Concentrate - Low Recovery	60	% to Conc	Used to calculate concentrate mass for tailings equipment
Magnetic Concentrate			
% of Rougher Flotation Tails As Mag Con	12%		From XPS testing - preliminary value only
Mag Con Produced	123.7	mtph	Based on minimum mass of flotation concentrate
Mag Con Regrind			
Magnetic Concentrate Work Index	19.0	kWhr/mt	metric
Magnetic Concentrate Flotation			
% of Fresh Feed as Mag Conc. Flot Conc	0.20%		
Mag Rougher Concentrate Cell Capacity	123.7	mtph	
Con % Solids	40.0%		weight / weight
Volume Flow Rate	216.4	m³/hr	
Mag Con Flot Concentrate Mass	2.2	mtph	
Carrying Capacity	1	mtph/m²	First approximation
Retention Time	24	minutes	To be confirmed in PFS
Total Cell Volume	86.578947	m³	
Total Surface Area Required	2.2	m²	Based on carrying capacity
Number of Cells	4		
Per Cell Volume	21.6	m³	
Cell Diameter	2.4	m	
Per Cell Area	4.5	m²	
Total Cell Area Available	18.1	m²	Sufficient Capacity for crowders
Final Concentrate			
% of Fresh Feed as Final Concentrate	2.50%		

Source: Eggert, 2015

17.3 Major Equipment List

The major pieces of equipment that have been used to determine the capital and operating costs are listed in Table 17.2.

Table 17.2: Major Equipment

Equipment	Quantity	Size	Unit Power
Gyratory Crusher	1	60-89	600 kW
Conveyors	250 m	1.2 m	??
Primary Screen	2	2.4 m X 7.3 m	30 kW each
Secondary Screen	2	2.4 m X 7.3 m	30 kW each
Cone Crushers	2	MP1000	1000 kW each
Primary Ball Mills	2	7.6 m X 10.4 m	10.5 MW each
Rougher Flotation	4	300 m^3	300 kW each
Magnetic Flotation	4	20 m^3	20 kW each
Cleaner Flotation	12	10 m^3	10 kW each
Magnetic Regrind	1	4.1 m X 5.1 m	1,200 kW
Concentrate Regrind	1 (Isa, HIGS or Verti)	Unknown , Allocation	400 kW

Source: Eggert, 2015

18 Project Infrastructure

18.1 General

The Wellgreen project envisions construction of the following key infrastructure items during phase 1 (25,000 tpd):

- 13 km of 8 m wide access roads;
- 36 MW LNG fired power plant, expanded to 42 MW by year 5;
- LNG storage farm with 5 – 60,000 gallon bullets;
- LNG filling/dispensing system;
- LNG storage and dispensing for mine mobile equipment;
- Operations and construction camps;
- Mineral processing plant;
- Primary crusher;
- Secondary crushing and screening buildings;
- Haul truck shop;
- Warehouse and maintenance shop;
- Mine dry and administration building;
- ANFO storage and loading;
- Dual purpose fresh/firewater tank;
- Process water tank;
- Potable water skid and distribution;
- Sewage treatment plant; and
- Water treatment.

The following will be added during phase 2 (50,000 t/d) of the Wellgreen project:

- 27 MW LNG fired power plant;
- Additional LNG storage farm with 4 – 60,000 gallon bullets;
- Additional LNG filling/dispensing system;
- New process building containing grinding mills and rougher flotation;
- Duplicate screening building;
- Secondary and tertiary crushing building extension;
- Fresh/firewater tank extension; and
- Process water tank extension.

18.2 General Site Arrangement

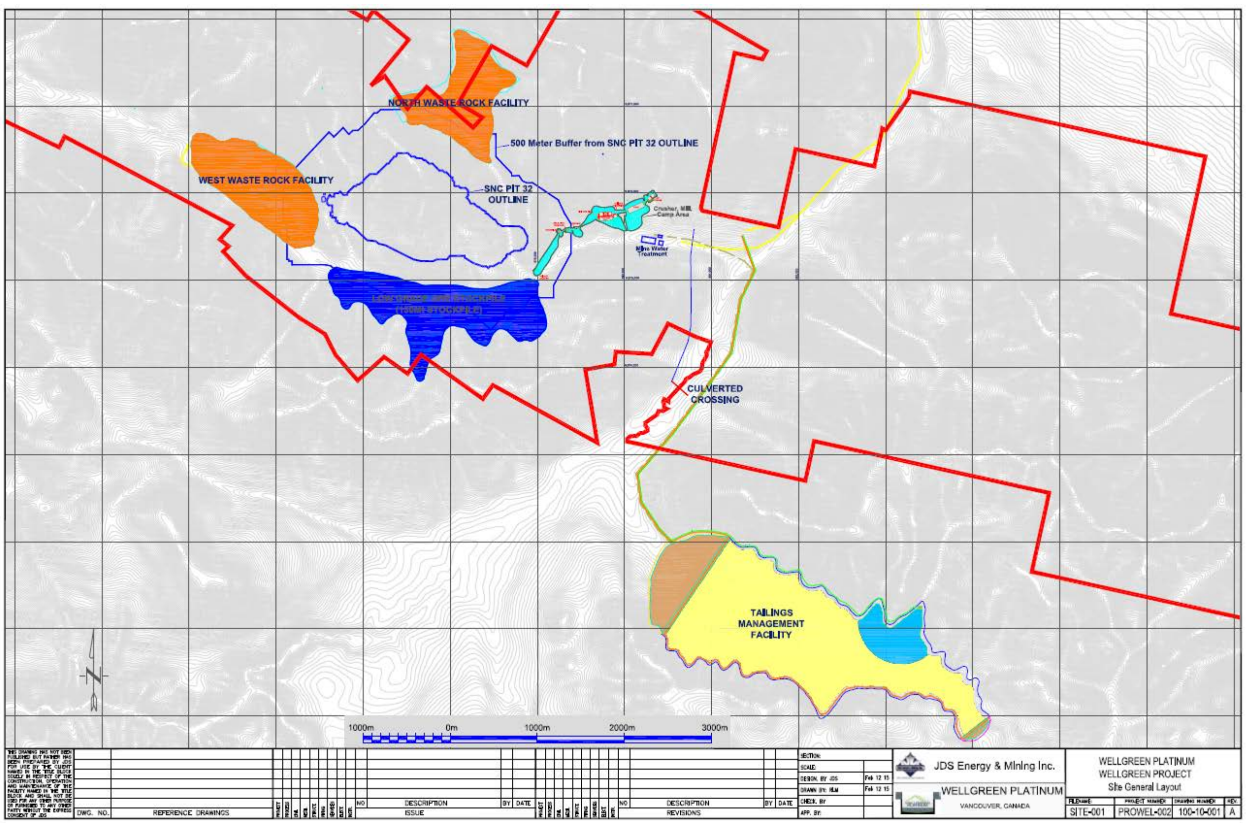
The overall site arrangement is shown in Figure 18.1.

The site has been configured for optimum construction access and operational efficiency. Primary builds are located to allow easy access from the existing mine access road and utilize existing topography to minimize bulk earthworks volumes. The primary crusher has been located as close as safely possible to the pit and at an elevation that facilitates material conveying. Existing roads are upgraded and reused wherever possible.

18.3 Site & Mine Access Road

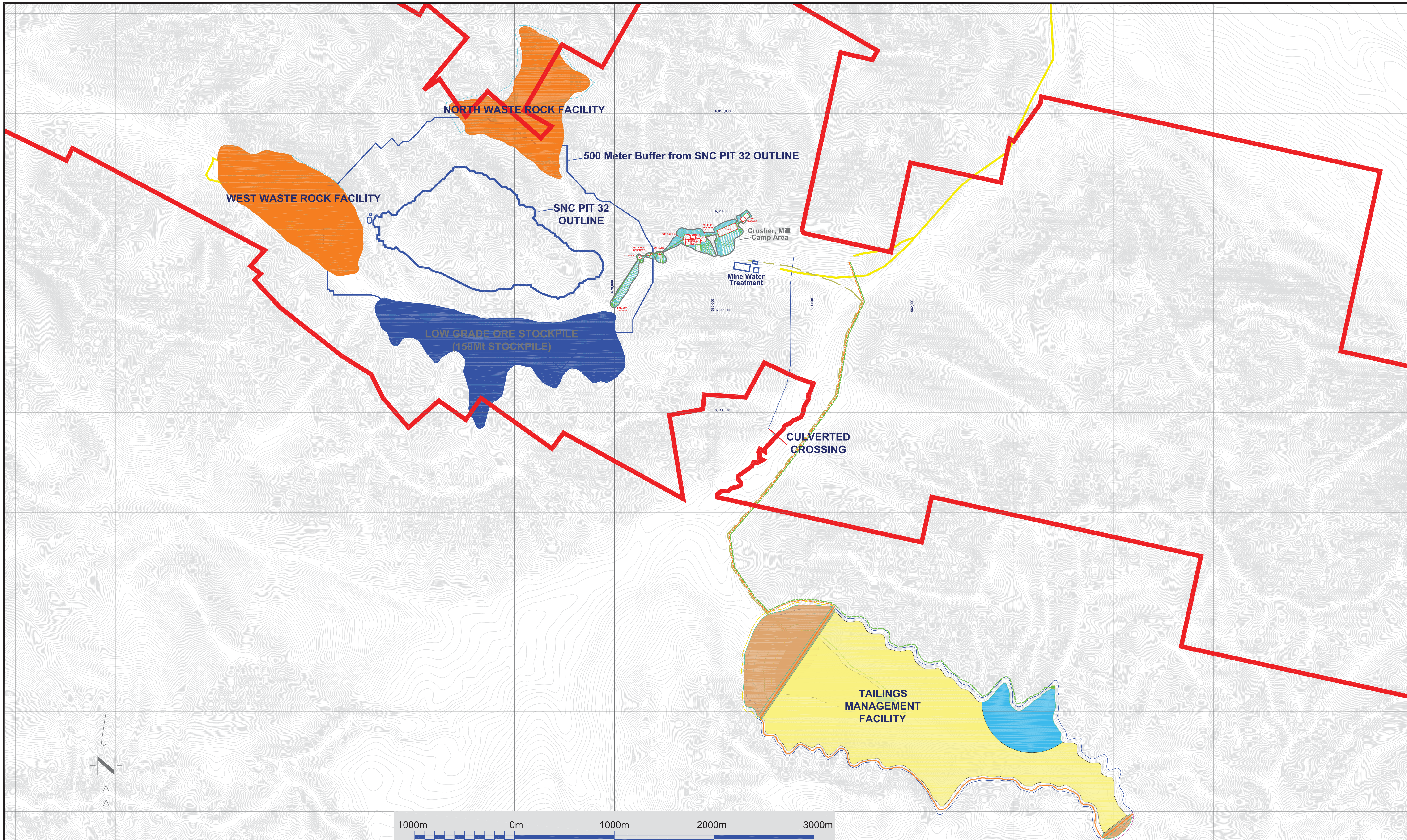
The existing 14 km mine access road from the Alaska Highway to site will receive significant upgrades to accommodate increased traffic. The road will be widened to 8 m with new gravel, grading and compaction. It will be suitable for transportation of mining equipment, fuel trucks, mobilization of construction equipment and ongoing operational requirements.

Figure 18.1: Overall Site Arrangement



Source: JDS, 2015

Overall Site Arrangement Preview Only –See 11x17 layout next page.



DWG. NO.	REFERENCE DRAWINGS

PROJECT | PROCESS | CIVIL | MECH. | STRUCT. | PIPING | SERVICES | ELECT. | INSTR.

NO	DESCRIPTION	BY	DATE

ISSUE

PROJECT | PROCESS | CIVIL | MECH. | STRUCT. | PIPING | SERVICES | ELECT. | INSTR.

NO	DESCRIPTION	BY	DATE

REVISIONS

SECTION:	
SCALE:	
DESIGN. BY JDS	Feb 12 15
DRAWN BY: RLM	Feb 12 15
CHECK. BY	
APP. BY:	

JDS Energy & Mining Inc.

WELLGREEN PLATINUM

VANCOUVER, CANADA

WELLGREEN PLATINUM
WELLGREEN PROJECT
Site General Layout

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
SITE-001	PROWEL-002	100-10-001	A

18.4 Power Supply

Wellgreen Platinum has signed a MOU with GE to provide products and services for the Wellgreen project, which includes complete power generation and the transmission network for the Wellgreen project. The MOU refers to GE's comprehensive electrical infrastructure technology for the mine processing equipment, transmission technology and control & automation equipment. This process is predicted to enhance project commissioning, start up of operations and decrease operating and capital expenditures due to increased efficiency, improved engineering designs and enhanced pricing formats.

LNG, for the purposes of the 2015 PEA, is considered to be delivered from Fort Nelson, British Columbia. In addition, there is potential to obtain LNG supply from Prudhoe Bay or Cook Inlet, Alaska (Figure 18.2). There are also several LNG suppliers that have expressed an interest to ship LNG from the lower main land, British Columbia, via ocean freight.

Figure 18.2: Potential LNG Sources

Source: Wellgreen, 2015

The haulage of LNG from Prudhoe Bay, Alaska is associated with the Alaska Interior Energy Project that is sponsored by the Alaska Industrial Development and Export Authority ("AIDEA"):

- Purpose is to to bring natural gas to residential and commercial customers in the Fairbanks North Star Borough (FNSB).
- The North Slope plant is designed for expansion created by industrial usage to other markets.
- Industrial usage has the potential to decrease unit production cost from $11.59 per Mcf to (Mcf) to $10.52 / Mcf (thousand standard cubic feet)

The Wellgreen project could eventually have the capability to be expanded to facilitate additional industrial users.

Phase 1 of the Wellgreen project will require 36 MW of peak power. GE has proposed nine Jenbacher J624 engines with generators (gensets) that will provide 4.4 MW each. Over subsequent years two additional engines will be installed to provide N+2 redundancy. To maintain un-interrupted power supply to the process plant, three days of storage capacity will be accomplished with 5 – 60,000 gallon LNG storage bullets.

An estimated additional 27 MW will be required during Phase 2. This would see the addition of six more J624 gensets and 4 more LNG storage bullets. The gensets will be constructed adjacent to the process plant to minimize high voltage cable runs and earthworks.

For the purposes of the 2015 PEA, the cost of the power is $0.14/kWh, which includes maintenance expenditures of $0.015/kWh.

18.5 Construction Power

Standalone diesel generators will supply 2 MW of power during site construction. These will be rented to reduce project capital costs.

18.6 Camp

A permanent Jack and Jill style camp facility with 200-person capacity will be installed and utilized over the entire life of mine. The existing 80-person exploration camp will remain on site to provide construction and operations overflow. A 100-person gang style construction camp will accompany the permanent camp during Phase 1 construction. To reduce capital cost the construction camp will be rented for the duration of Phase 1 construction and the permanent camp will be operated with double occupancy. This together with the exploration camp will provide approximately 580 potential rooms during Phase 1 construction. Both camps will share dinning, recreation and gym facilities. A permanent potable water and sewage system will be installed as part of the permanent camp. During phase 2 construction a 400-person construction camp will be rented.

18.7 Process Plant Building

The process plant is a 150 m x 50 m pre-engineered building. The preliminary layout utilizes a narrow footprint in order to minimize cut/fill volumes in a challenging geographical area. It contains milling, flotation, regrind, concentrate thickening, filter presses, concentrate storage/loadout, reagent storage and electrical rooms. A 12,500 t fine material bin will provide 12- hours of feed to the mill.

In order to double throughput during Phase 2 a second building will need to be erected containing duplicate grinding and rougher flotation circuits. Downstream equipment will be upsized in the existing process plant in an effort to reduce mass earthworks and building costs. A second fine material bin will also be constructed.

18.8 Screening Building

The mill feed screening build is an 18 m x 18 m pre-engineered structure. During the second phase of the project a second duplicate building will be erected.

18.9 Secondary and Tertiary Crushing Building

Secondary and tertiary crushers will be housed inside a 16 m x 36 m pre-engineered building. It is possible to simply extend this building during Phase 2 in order to increase crushing capacity.

18.10 Truck Shop

A large 100 m x 120 m pre-engineered building containing one large bay, one utility bay and mine dry will be used to service haul trucks and mining mobile equipment. As throughput and mining activities increase additional bays will be added in years 4, 9 and 17.

18.11 Maintenance, Warehouse, Mine Dry and Administration Building

The truck shop and warehouse will be contained in one common pre-engineered 15 m x 30 m building. The truck shop will house three separate bays: general maintenance; wash; and lube & oil. Each bay will have a dedicated 14’ x 14’ roll up to accommodate all vehicle sizes. Tire changing and large vehicle assembly will take place outdoors and utilize rough terrain mobile equipment. A general warehouse will be included within the same building in a separate partitioned bay.

The mine dry and administration building will be located inside the process plant to allow for quick access to/from each facility.

18.12 Communications & IT

The camp and offices will include a wired and wireless computer network and satellite phone system.

A hand-held radio system will be used for voice-communication between personnel in the field.

18.13 First Aid and Emergency Services

A qualified nurse or first-aid attendant will be provided on-site. The first aid room will be located besides the administration building. The ambulance and fire truck will be parked at the ready outside the process plant.

Buildings will be equipped with smoke, carbon monoxide and heat detectors, overhead sprinklers, hydrants / hoses and appropriate chemical fire extinguishers.

18.14 Bulk Explosives Storage and Magazines

Explosives will be stored at a secured and monitored site located approximately 800 m from the main plant and populated, high traffic areas. All infrastructure items include a storage silo, small truck shop, loading hopper, powder magazine and detonator magazine.

18.15 Bulk Fuel Storage and Delivery

The current mine plan uses LNG powered mining equipment with minor diesel subsidization. A 75,000 gallon LNG bullet will provide three days of fuel storage for the fleet. Dedicated loading and dispensing units will be included. Diesel fuel will be stored in two 85,000L dual wall fuel tanks located near the truck shop. The tanks will have an internal submersible pump capable of delivering 40 GPM to all site vehicles. Diesel will be delivered to mobile equipment by the fuel and lube truck. A small spill containment pad will be installed around the fueling station.

18.16 Fresh and Fire Water Tank System

The firewater tank will be dual purpose serving as a freshwater and firewater storage tank. Internal risers on all non-firewater suction lines will ensure a minimum volume of 470,000 L. This capacity will allow for approximately 2-hours of firefighting capability. Additional rings will be added to the tank during Phase 2 in order in increase capacity.

The buried firewater network will be pressurized by two pumps (one electric, one diesel stand-by). This network will be connected to all buildings requiring fire protection.

18.17 Process Water Tank

The process water tank is design with a 2-hour retention time. It will be a 12.5 m x 12.5 m steel tank with a total capacity of 1,500,000 L. Similar to the fresh/firewater tank, additional rings will be added during Phase 2.

18.18 Potable Water and Sewage Treatment

Potable water and sewage treatment systems will be included with the camp facilities. These will be permanent fixtures for the duration of the mine life.

18.19 Water Supply and Treatment

18.19.1 Water Supply

Site water needs adequate for drilling operations have historically been pumped from local creeks. With the addition of a larger camp to support mine construction and operations, potable water will have to be provided from new wells since the surface water in the local creeks freezes solid during the winter months.

Water supply for the mine's process plant will require the use of several different water sources and judicious storage and planning to ensure that sufficient water for mineral processing is available throughout the year. Both the pit and the tailings storage facility can provide significant collection and storage for water. The pit will collect surface water runoff as well as groundwater infiltration through the pit walls, once mining in the pit drops below the water table. Storm water from the watershed draining to the tailings storage facility is significant and will be important to the mine's operating water needs particularly in the early years of mine production before the pit drops below the water table and pit infiltration becomes a significant source of mine water for processing.

Until pit depth is such that water infiltration through the pit walls becomes a significant source of water for the mineral processing operations, a series of water well clusters will be required to supply sufficient make up water for the processing plant. Preliminary pit infiltration modelling has been based on a very limited data set and suggests that ground water entering the pit through wall infiltration will likely range from a low of approximately 235 cubic meters per day to a high of approximately 3,830 cubic meters per day. Pit infiltration rates are an important source of water for the operation and these preliminary estimates will require refinement in the next level of study.

Making full use of collected storm water in the pit, the tailings storage facility, and impacted runoff from the stockpile and waste rock facilities as well as recycling water decanted from the tailings itself will be a priority in order to minimize the use of fresh water from the well clusters. Judicious management of storage space for water in the pit, tailings storage facility, and collection pond will be critical to maintaining a sufficient process water supply without over use of the well cluster water. Storage of water is key in the seasonal swings from surplus water conditions (May/June), to the deficient drier times of the year (March and again in October).

18.19.2 Water Treatment

Preliminary estimates of the capital and operating costs for the Property's water treatment system were based upon a peak volumetric treatment rate of 600 cubic meters of water per hour. Only water destined for discharge will be treated and only during times of surplus availability. For the purposes of this study, influent water quality was assumed to be consistent with existing surface water sampling data collected for the site. It has been agreed that site specific discharge criteria will be used for the treatment limits for this site. Discharge water quality limits for the Property site remain to be quantitatively determined through negotiations with the regulatory authorities to determine specifically what the appropriate site-specific effluent discharge limits will be and will take existing background water quality into account. It is understood that these yet to be determined discharge water quality limits will be less stringent than the standard MMER standards and CCME guidelines due to existing background water quality at the site. Attenuation of runoff flows to the peak treatment rate will be achieved through the use of an appropriately sized collection pond and diversion channels. Pond sizing will be confirmed once runoff watersheds and storm runoff quantities are more clearly defined for the site in the next level of study taking into account stockpile and waste rock facility locations and extents.

For this PEA level study a peak water treatment rate of 600 cubic meters of water per hour is used in the estimation of both the capital costs for the water treatment system and for the operating cost estimates for on-going treatment activities. In most situations collected storm water, water recycled from the tailings storage facility, and water from pit dewatering activities will be added directly to the make-up water stream for use in the mill facility for processing material from the mine.

In times of significant excess water flows, for example during a significant storm event, water will be stored in the pit and in the tailings storage facility to attenuate the peak water treatment rate required.

Water treatment will consist of a collection (equalization) pond, treatment system, and a smaller polishing pond. The polishing pond will be the last water storage location that enables a final verification that discharge quality limits have been achieved prior to the water being released to the surrounding environment.

Based on a preliminary assessment performed by BIOTEQ, a hybrid treatment solution has been recommended that involves metal hydroxide and sulphide precipitation using a combination of traditional lime neutralization with BioteQ's ChemSulphide® process. Using lime and sulphide in tandem, hydroxide ensures the pH of the reactor is effective for precipitating metals such as aluminum and chromium, while the sulphide is effective at reaching the typically lower discharge requirements of other metals such as copper, cadmium, nickel, and arsenic. The resulting precipitate generated by this process is typically of a high metal content meaning that very little of it will require disposal since the high metal content precipitate is typically sent to a smelter for refining.

18.20 Freight

Freight will be delivered to site on the all-season access road and offloaded at the warehouse or other designated area.

18.21 Personnel Transportation

Construction and operations personnel will be transported from the Whitehorse airport to site via contracted coach bus. A maximum of 40 operations personal will be transported daily from Haines Junction to site.

18.22 Tailings Management Facility

Knight Piésold Ltd. (KP) provided preliminary costs (PEA level design) of the Tailings Management Facility (TMF), Option 1, using the existing mine production schedule in May 2014 (VA14-00783). In January 2015, KP was asked to provide updated preliminary costs for the TMF, Option 1, using the updated mine production schedule provided by JDS ("WG Mine Scheduler Jan 15 FINAL.xlsx", January 15, 2015).

18.22.1 TMF Design Basis

The design basis for this TMF study is summarized below in Table 18.1.

Table 18.1: TMF Design Basis

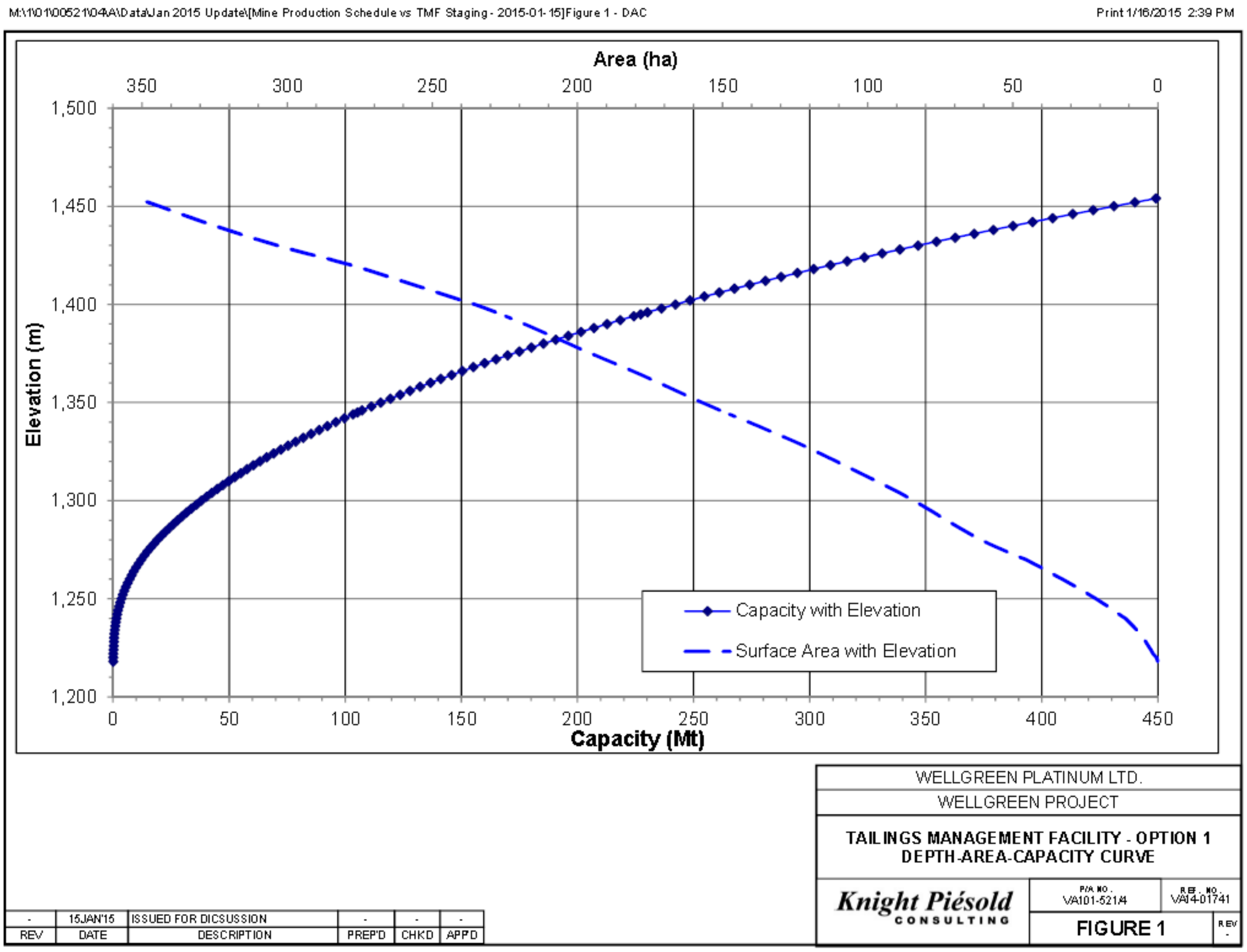
Parameter	Units	Value
Nominal Mineralization Throughput (Year 1 to 5)	tpd	~25,000
Nominal Mineralization Throughput (Year 6 to 24)	tpd	~50,000
Nominal Mineralization Throughput (Year 25)	tpd	~25,000
Design Life	yrs	25
Total Tonnes of Tailings	Mt	402
Tailings Final Settled Dry Density (average)	t/m^3	1.3
Final Required Storage Volume	Mm^3	310
1 Yr. Starter Dam Storage Volume	Mm^3	8.5

Source: KP, 2015

The TMF is sized to store a volume of approximately 300 Mm^3 for the total tailings production of 390 Mt at an assumed settled dry density of 1.3 t/m^3. The depth-area-capacity relationship for the TMF is shown on Figure 18.3. The TMF embankment is designed for six staged expansions as the level of the stored tailings and water increases. The TMF layouts for Stages 1 and 6 are shown on Figure 18.5.

Construction will be staged to distribute initial and sustaining capital costs. The staging and filling schedule used in the conceptual design and costing of the TMF is shown in Table 18.2 and on Figure 18.4.

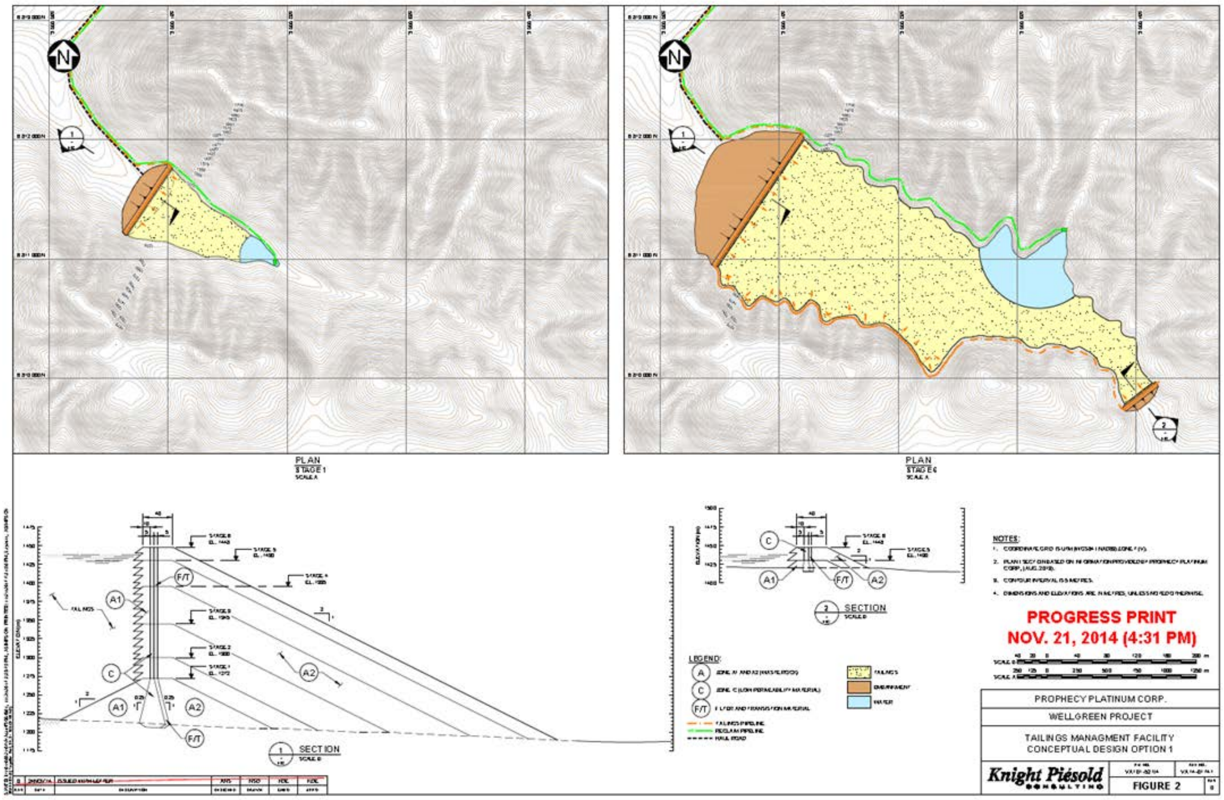
Figure 18.3: TMF Depth Area Capacity Curve



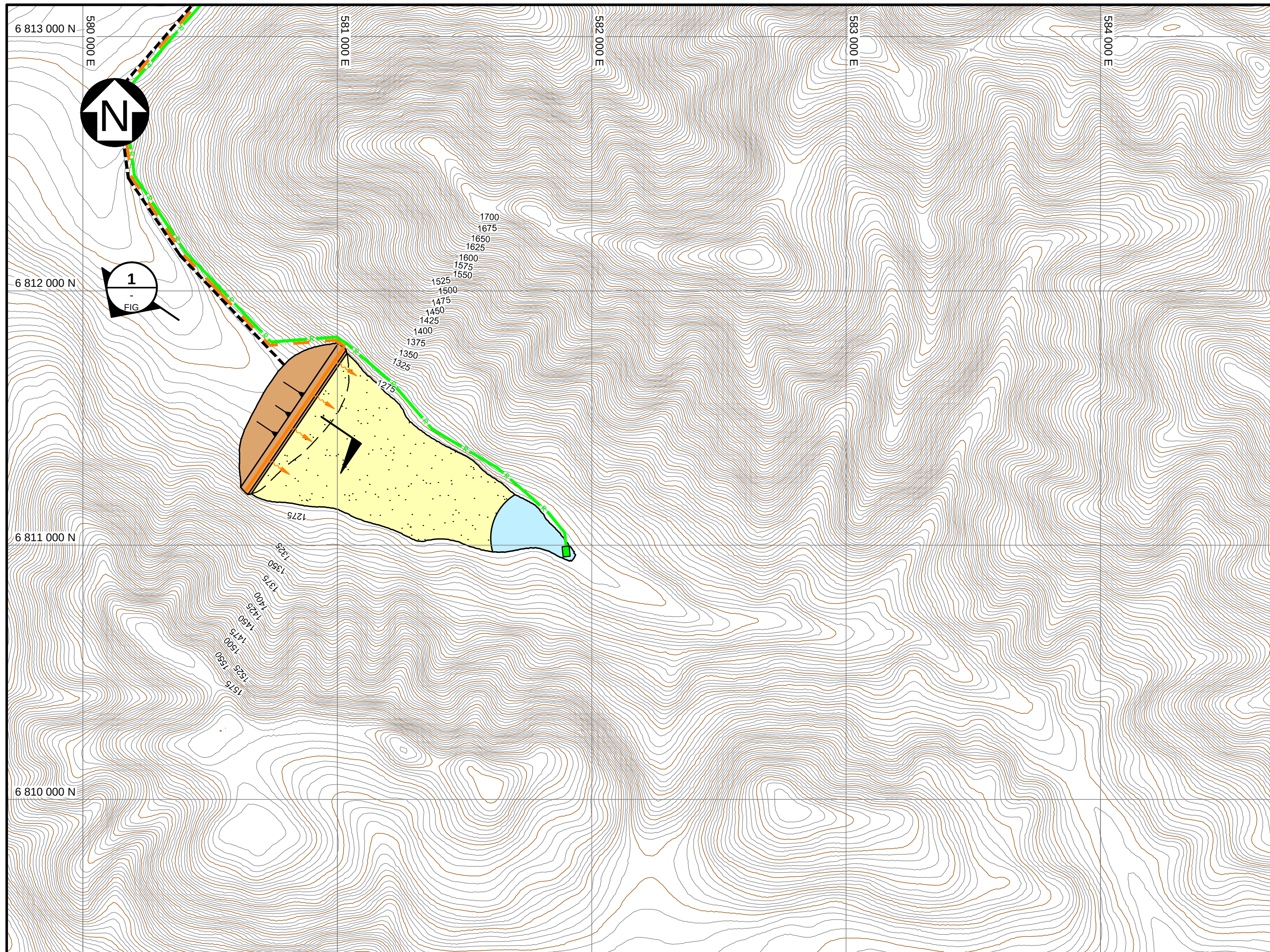
Source: KP, 2015

Figure 18.4: TMF Conceptual Design

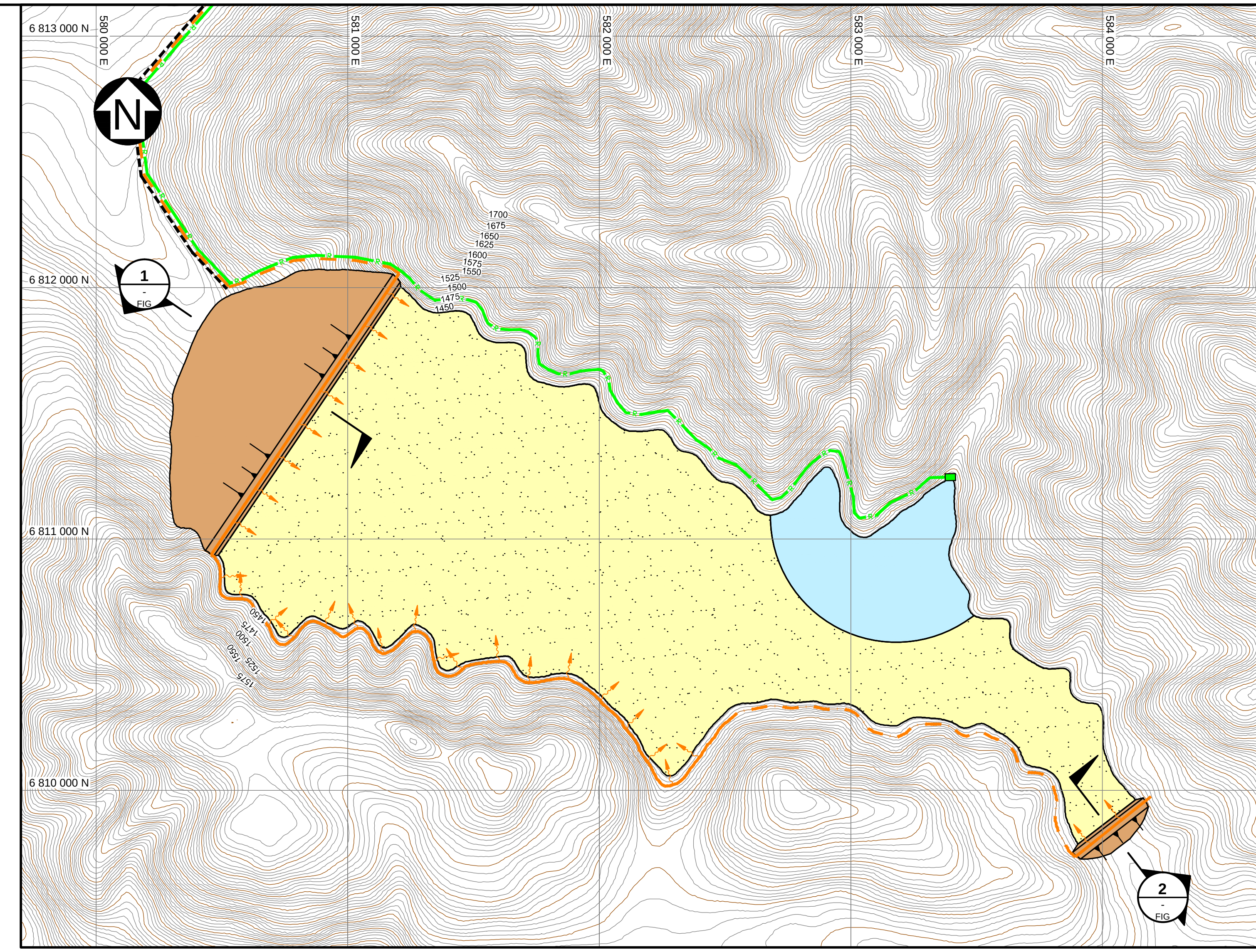
TMF Conceptual Design Preview Only –See 11x17 layout next page.



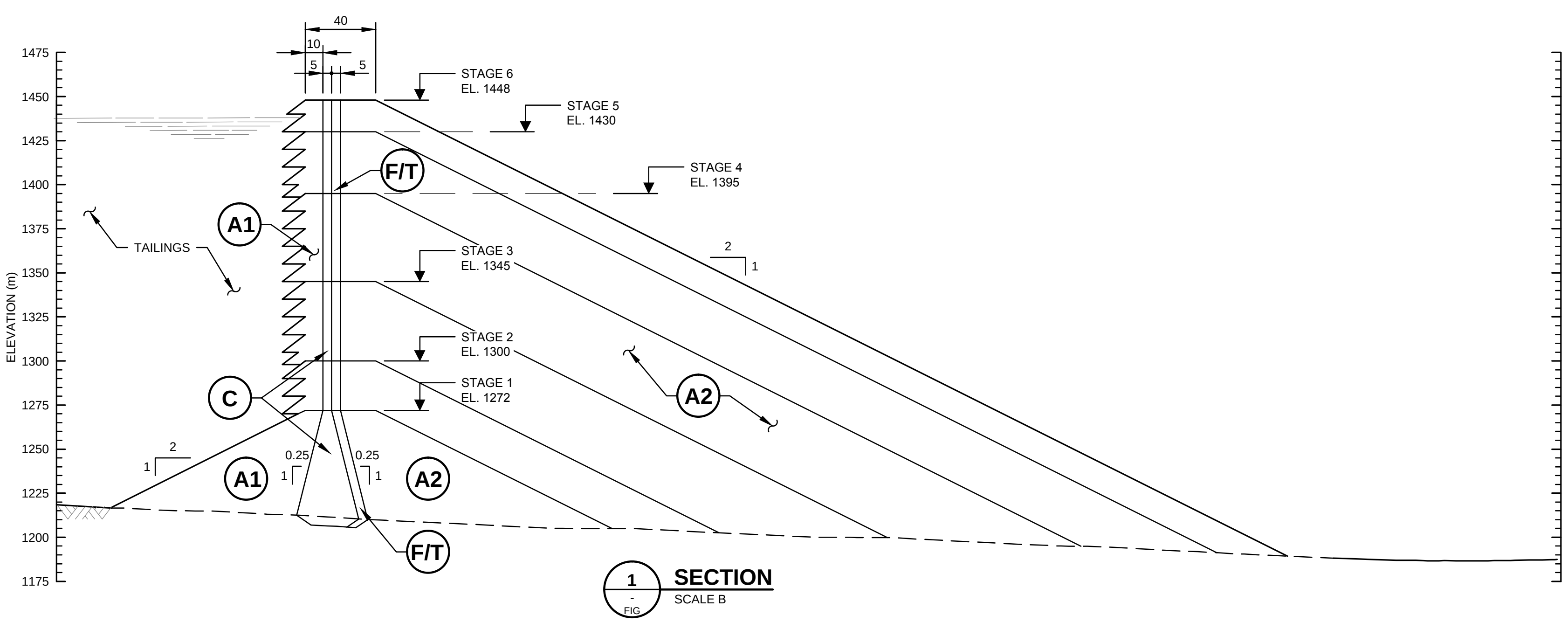
Source: KP, 2015



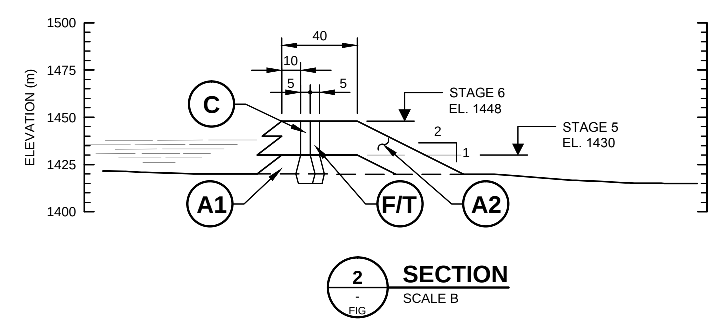
PLAN
STAGE 1
SCALE A



PLAN
STAGE 6
SCALE A



1 SECTION
SCALE B



2 SECTION
SCALE B

LEGEND:

- A — ZONE A1 AND A2 (WASTE ROCK)
- C — ZONE C (LOW PERMEABILITY MATERIAL)
- F/T — FILTER AND TRANSITION MATERIAL
- TAILINGS PIPELINE
- RECLAIM PIPELINE
- HAUL ROAD
- TAILINGS
- EMBANKMENT
- WATER

NOTES:

1. COORDINATE GRID IS UTM (WGS84 / NAD83) ZONE 7 (V).
2. PLAN / SECTION BASED ON INFORMATION PROVIDED BY PROPHECY PLATINUM CORP., (AUG. 2013).
3. CONTOUR INTERVAL IS 5 METRES.
4. DIMENSIONS AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

PROGRESS PRINT
NOV. 21, 2014 (4:31 PM)

SCALE B: 40 20 0 40 80 120 160 200 m
SCALE A: 250 125 0 250 500 750 1000 1250 m

PROPHECY PLATINUM CORP.

WELLGREEN PROJECT

TAILINGS MANAGMENT FACILITY
CONCEPTUAL DESIGN OPTION 1

Knight Piésold CONSULTING

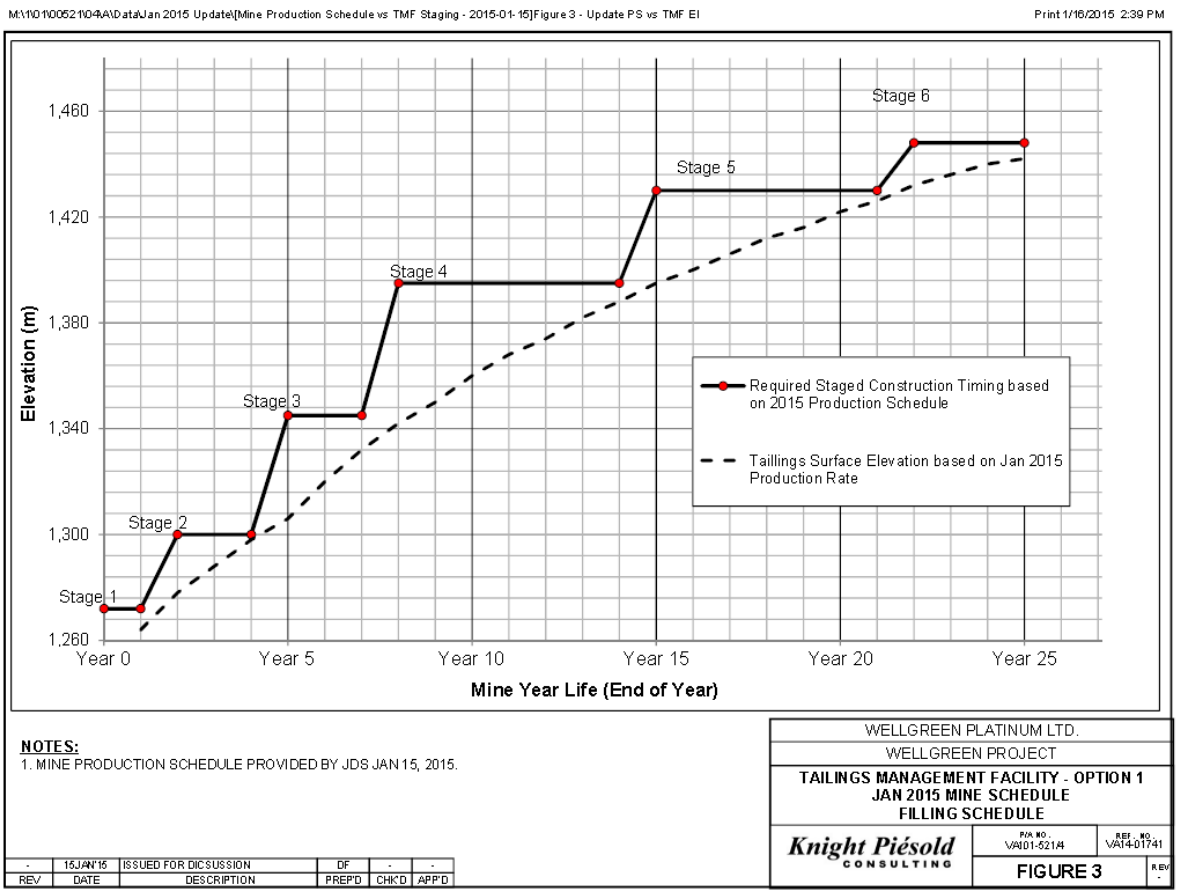
P/A NO.	REF NO.
VA101-521/4	VA14-01741

FIGURE 2 REV 0

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	CHK'D	APP'D
0	24NOV'14	~~ISSUED WITH LETTER~~	ANS	NSD	KDE	KDE

Figure 18.5: TMF Production Rate Filling Schedule

TMF Production Rate Filling Schedule –See 11x17 layout next page.



Source: KP, 2015

Table 18.2: Timing of Staged Construction

Stage	Stage Construction	Embankment Crest Elevation (m)	Tailings Storage
1	Year -1	1272	Year 1
2	Year 1	1300	Years 2 to 4
3	Year 4	1345	Years 5 to 8
4	Year 8	1395	Years 9 to 15
5	Year 15	1430	Years 16 to 21
6	Year 21	1448	Years 22 to 25

Source: KP, 2015

19 Market Studies and Contracts

19.1 Market Studies

The bulk concentrate to be produced at the Property is assumed to be of a quality that can be sold in the open market, however, no formal market studies were undertaken for this report.

Contracts for the smelting and refining of bulk concentrates are negotiated on an individual basis. Treatment charges typically depend on a variety of factors including the global smelter supply-demand balance, the grades of metals within the concentrate and the level of deleterious elements in the concentrate.

19.2 Concentrate Marketing

Wellgreen Platinum is currently contemplating a conventional flotation flowsheet that results in the production of a bulk concentrate containing nickel, copper, cobalt, PGMs and gold that would be sent to a nickel sulphide smelter.

There are currently believed to be at least seven large nickel smelters globally that could process the bulk concentrate from the Property: Jinchuan Group Co., Ltd. and Jilin Jien Nickel Co., Ltd. in China; Glencore plc in Sudbury, Canada; Stillwater Mining Company in the United States; Kalgoorlie Consolidated Gold Mines, in Australia; and Boliden Harjavalta Oy, in Finland These smelters are believed to be processing concentrates with average combined Ni-Cu-Co grades ranging from 8% to 21%. Wellgreen Platinum's concentrate is expected to have combined Ni-Cu-Co grades near the middle of this range and with low levels of deleterious elements.

The ability to market the project's bulk Ni-Cu-PGM concentrate will be driven by industry demand at the time of production.

19.2.1 Smelter Terms

Unlike concentrate markets such as copper, zinc, and lead, which are relatively standardized, bulk nickel-copper and nickel concentrate treatment terms vary considerably between smelters and contract terms are difficult to obtain due to confidentiality clauses in smelting agreements. The concentrate terms used in this PEA are conceptual in nature and are based on information from other nickel projects or contracts and informal discussions with concentrate marketing specialists.

This PEA assumes the smelter terms shown in Table 19.1 for a bulk concentrate from the Wellgreen project (in US dollars) based on smelting at a non-Chinese smelter.

Table 19.1: Bulk Ni-Cu Concentrate Smelter Term Assumptions

Bulk Concentrate	Unit	Assumptions
Average LOM Concentrate Grades		
Nickel	%	8.0
Copper	%	5.2
Cobalt	%	0.1
Platinum	g/t	5.9
Palladium	g/t	7.2
Gold	g/t	1.0
Moisture Content	%	8
Smelter Parameters		
Payables (subject to a minimum deduction as per below)		
Nickel	%	90
Copper	%	88
Cobalt	%	50
Platinum	%	80
Palladium	%	80
Gold	%	80
Minimum Deductions		
Nickel	%	1
Copper	%	0.25
Cobalt	%	0.25
Platinum	g/t	1
Palladium	g/t	1
Gold	g/t	1
Treatment & Refining Charges		
Bulk concentrate treatment charge	US$/DMT	225
Nickel refining	US$/lb Ni	0.65
Copper refining	US$/lb Cu	0.4
Cobalt refining	US$/lb Co	3
Platinum refining	US$/oz Pt	15
Palladium refining	US$/oz Pd	15
Gold refining	US$/oz Au	15
Freight & Marketing Charges		
Truck Freight	US$/wmt conc	43.48
Ocean Freight	US$/wmt conc	60
Port charge	US$/wmt conc	13
Survey, Umpire	US$/wmt conc	3.2
US Customs	US$/wmt conc	1.85
Total Freight & Marketing	US$/wmt conc	121.53
	US$/dmt conc	132.1
Insurance	US$/$1K value	0.495

Source: JDS, 2015

19.2.2 Other Transportation and Selling Costs

The amount of annual concentrate produced is based on the concentrate grade, mine production grade, recovery and moisture content. The Wellgreen project is expected to produce approximately 389,000 dry metric tonnes of concentrate per year, on average, over the life of mine. Transportation to the smelter is a function of the distance from the mine to the smelter and the modes of transportation (e.g. road, rail and ocean freighter). The concentrate would also be subject to port and insurance charges. Together, these costs are estimated to be approximately US$122/wet metric tonne of concentrate.

Future work will also continue to look at producing a separate copper concentrate to confirm that the bulk concentrate output flowsheet is the optimum approach and in case better concentrate terms are available for two concentrates instead of a single bulk concentrate.

19.2.3 Alternate Smelting Terms

There is currently a strong trend to use smelter terms based on the Chinese model. This model has the unique feature of not having treatment or refining charges, however, it has much lower payables than the model used in the 2015 PEA economics. Potential Chinese model values are shown in Table 19.2.

Table 19.2: Potential Chinese Smelter Terms

Bulk Concentrate	Unit	Assumptions
Smelter Parameters		
Payables (subject to a minimum deduction as per below)		
Nickel	%	68-72
Copper	%	30-35
Cobalt	%	30
Platinum	%	30-35
Palladium	%	30-35
Gold	%	30-35

Source: JDS, 2014

Chinese nickel smelters generally are not designed to treat Cu and PGMs in their concentrates and, as a result, give reduced credit for these metals. Smelters in North America and Europe are better set up to recover these by-products.

19.3 Contracts

No contractual arrangements for concentrate trucking, port usage, shipping, smelting or refining exist at this time but costs have been benchmarked against other comparable operations or studies from projects in the general region of the Wellgreen project. Furthermore, no contractual arrangements have been made for the sale of bulk or nickel and copper concentrates at this time.

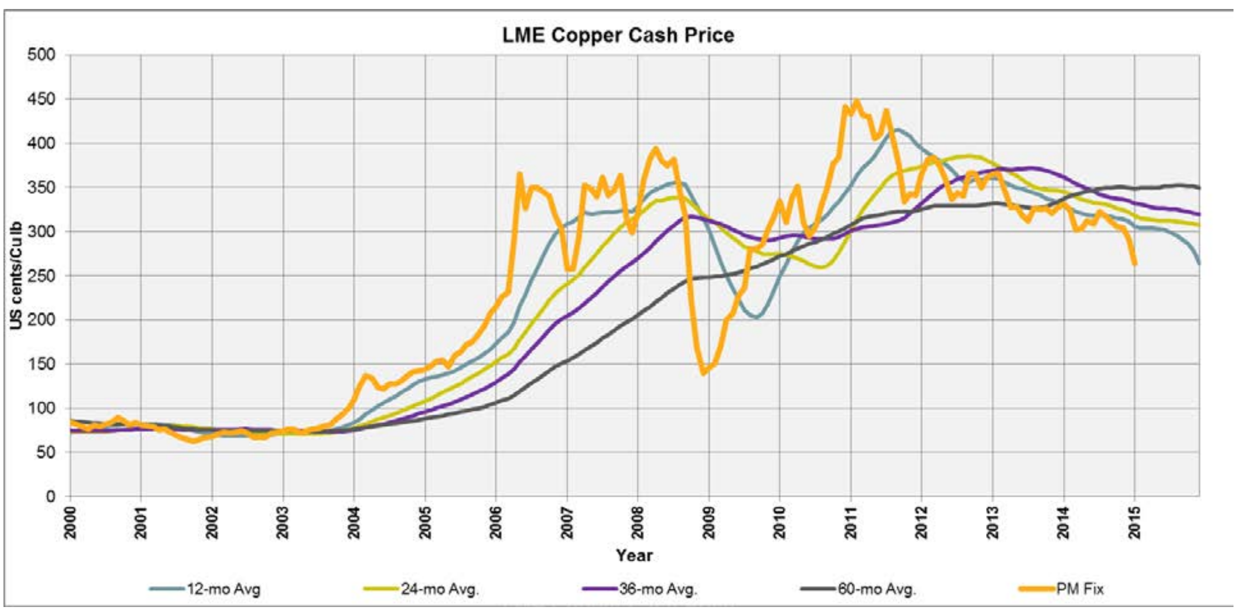
19.4 Commodity Price Forecasts and Assumptions

Metal prices and foreign exchange rates fluctuate continuously based on market sentiment and expectations of future supply, demand, and economic conditions. Figure 19.1 through 19.6 illustrate the changes in metal prices for the ten year period to December 2014 for the key metals from the Wellgreen project.

Figure 19.1: Nickel Price History

Source: JDS, 2015

Figure 19.2: Copper Price History

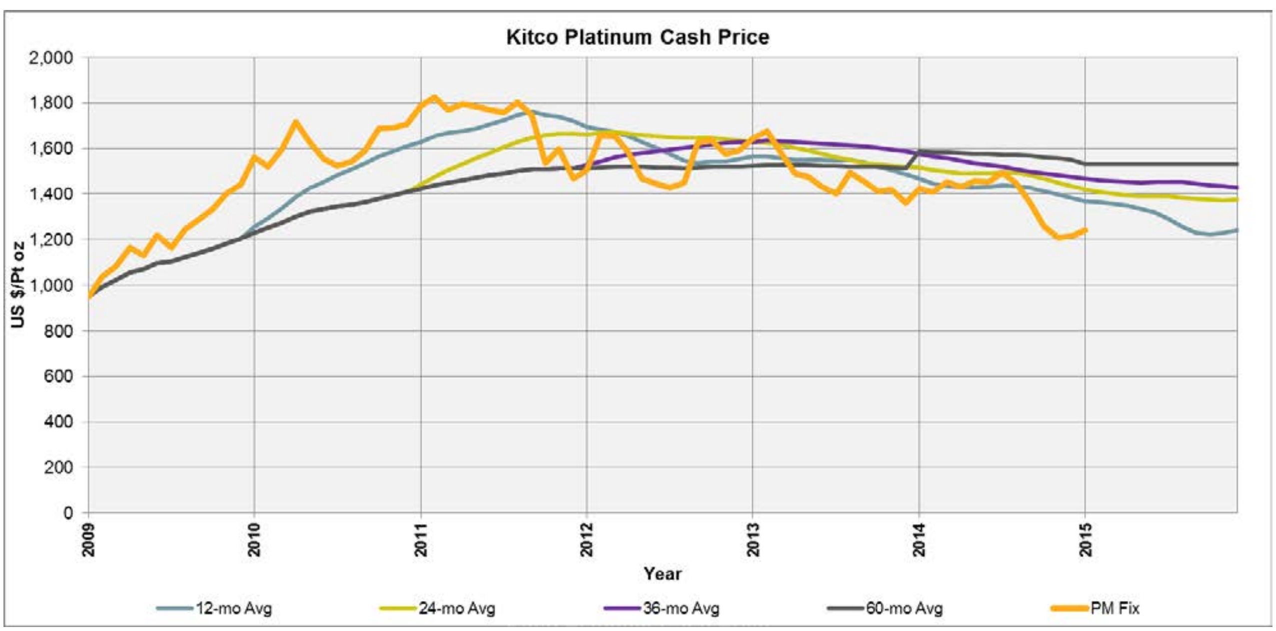


Source: JDS, 2015

Figure 19.3: Cobalt Price History

Source: JDS, 2015

Figure 19.4: Platinum Price History

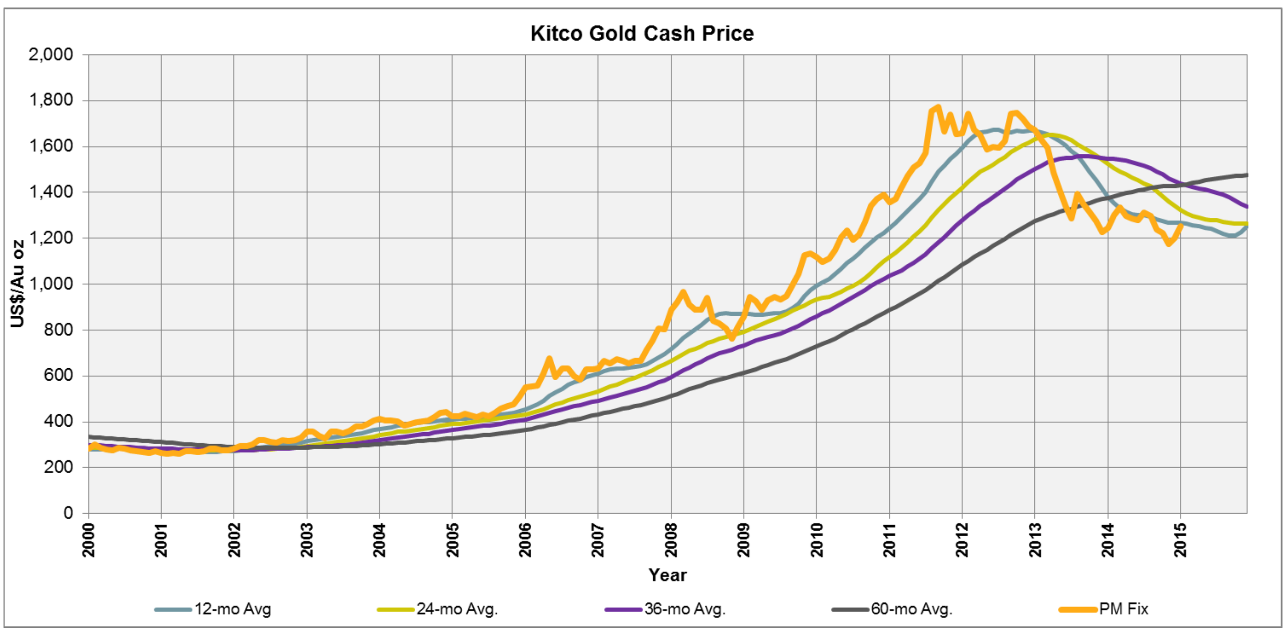


Source: JDS, 2015

Figure 19.5: Palladium Price History

Source: JDS, 2015

Figure 19.6: Gold Price History



Source: JDS, 2015

The Base Case pricing used in the economic analysis was derived based on a combination of spot prices, three-year trailing average monthly prices, long-term consensus analyst forecasts, and a review of the price assumptions used by peer group companies in recent economic analyses. In addition to the Base Case scenario, the economic analysis also evaluated spot, peer study average and long term consensus forecast metal price scenarios.

Table 19.3: Metal Price and Foreign Exchange Rate Used in Economic Analysis Scenarios

Parameter	Units	PEA Base Case	Peer Study Prices[1]	Long Term Consensus Forecast[2]	Spot Feb. 2, 2015
Nickel	US$/lb	8.00	8.82	8.74	6.83
Copper	US$/lb	3.00	3.30	3.18	2.51
Cobalt	US$/lb	14.00	14.00	12.93	13.38
Platinum	US$/oz	1,450	1,661	1,450	1,223
Palladium	US$/oz	800	797	950	773
Gold	US$/oz	1,250	1,356	1,148	1,273
Exchange Rate	C$/US$	0.900	0.900	0.877	0.800

Source: JDS & Wellgreen, 2015
[1] Mean price used by peers based on SEDAR filings over the past one year period
[2] Consensus analyst metal estimates for 2018 (2016 for cobalt) from Bloomberg, as at January 19, 2015

20 Environmental Studies, Permitting and Social or Community Impact

Environmental management issues associated with the Property are primarily, but not limited to, water quality and proximity to sensitive wildlife areas. Baseline environmental studies were initiated in 2012 and have been expanded upon over the past two years.

20.1 Existing Permits

Wellgreen Platinum does not hold any of the permits required to operate a mine. Wellgreen carries out its site and exploration activities under the following permits/licences:

- Class 3 Quartz Mining License, LQ00323;
- Class 3 Quartz Mining License, LQ0259;
- Surface Lease, 115 G11-003;
- Special Waste Permit, 41-229; and
- Waste Management Permit, 81-019.

20.2 Baseline Environmental Studies

Baseline environmental studies were first undertaken by Wellgreen Platinum in 2012. The area has undergone numerous baseline studies including the Alaska Highway Pipeline Project baseline work which runs adjacent to the property. In 2013, a comprehensive baseline program was outlined for the project.

Work undertaken to date includes:

- Weather;
- Hydrology, including initial hydrogeology;
- Monthly water quality;
- Initial Fishery studies; and
- Wildlife – aerial ungulate surveys and wildlife observations.

20.3 Weather

There is a weather station on site, located near the portal and upper camp. The data logger has been in place since 2013.

20.4 Hydrology

The main objective of the hydrologic monitoring program has been to characterize the timing and magnitude of stream flow at various locations within the project area. This data will be used to help make management, design and development decisions in the future. Five hydrometric stations were installed in October 2012 and are monitored monthly. Additional stations and further hydrologic monitoring may be required moving forward toward the development of a project proposal.

20.5 Aquatic Resources and Fishery Studies

In order to monitor potential changes related to the development of this project, water, sediment quality and aquatic biology baseline studies will be conducted in 2015. Monthly Water Quality sampling has been undertaken since 2012 at 14 locations in the project area. A total of 19 monthly datasets, from October 2012 to May 2014, of sample data have been gathered from locations on and tributaries to the Nickel, Quill, Swede Johnson and Arch Creeks representing the watershed catchments potentially affected by the Project.

These studies will continue and may be expanded upon in 2015 toward the development of a project proposal.

20.5.1 Wildlife Monitoring

Wildlife baseline information, including aerial moose and sheep surveys, have been collected since 2012 by Environment Yukon, Parks Canada and Kluane First Nation, with support from Wellgreen. Wellgreen has worked closely with Parks Canada, Department of Environment Yukon and Kluane First Nation to understand what additional baseline information is required and in collecting the data. The project area is located adjacent to Kluane National Park and Kluane Wildlife Sanctuary. Over 30 years of wildlife data has been collected for the park and sanctuary. In addition, Kluane First Nation has numerous years of ground based data for the Wellgreen project area. The information collected to date by all parties combined with close review and gap analysis will identify additional requirements will contribute to the comprehensive wildlife baseline study being conducted in 2015.

The wildlife monitoring program will need to be expanded in 2015 to include breeding bird surveys, raptor surveys, carnivore/den surveys, vegetation and habitat mapping, and any additional ungulate surveys identified.

20.6 Environmental Management

Wellgreen Platinum will be developing a number of management plans as part of Mine Licensing application process as the Wellgreen project moves toward development. These management plans include but are not limited to:

- Spill Response;
- Emergency Response;
- Reclamation and Closure;
- Wildlife;
- Environmental Monitoring;
- Explosives Management;
- Fuel Storage;
- Water Quality, Erosion and Sediment Control;
- Hazardous Materials and Waste Management;

- Heritage and Archaeological Sites Protection; and
- Access Management.

20.7 Site Reclamation and Closure

A site reclamation plan will be required as part of the design and project proposal submission. In this area, the expectation would be that all facilities would be removed from the site and that surface disturbance would be modified to minimize the impact upon wildlife and other land users. As part of the project design, the area of disturbance will be minimized and, as much as possible, there will be progressive reclamation work concurrent with operations. The site reclamation plan will be developed with input from Kluane First Nation that at a minimum meets the requirements outlined in the Yukon Government reclamation policy.

Financial assurance must be posted to secure the rehabilitation works, and the determination of the outstanding mine reclamation and closure liability associated with the Project technical features and structures must be sealed by a professional engineer who is licensed to practice in Yukon (Yukon Energy, Mines, and Resources 2006).

The Government of Yukon determines the amount and form of security to be provided by the proponent. The government will also ensure that security is maintained at all times. Financial security will comprise an initial payment, prior to commencement of development, and a periodic adjustment to ensure that full security is held for outstanding reclamation and closure liability throughout the development, operation, and closure of a mine site. Progressive reclamation may reduce the amount of financial security required to be provided, and maintained by the proponent.

The proponent must file an annual report stating what progressive reclamation has been accomplished and the results of environmental monitoring programs. The proponent will monitor to determine the effectiveness of the mitigation measures as progressive reclamation and closure work is completed. (Yukon Energy, Mines and Resources, 2006).

20.8 Environmental Assessment and Permitting

Before projects proceed to the licensing phase, they are first assessed through an EA. The Yukon Environmental and Socio-economic Assessment Board (YESAB) administer EAs in Yukon. The Wellgreen project will be subject to an EA under the YESAA.

20.8.1 Environmental Assessment

The project requires an Executive Committee screening because it is a quartz mining program that involves the movement of 250,000 t or more of rock. Projects assessed by the Executive Committee of YESAB generally require between one and three years (not more than 918 days, including time required for a government decision).

Detailed information requirements for this process are outlined in the Information Requirements for Executive Committee Project Proposal Submissions under the YESAA, which is available through the YESAB office.

Once assessments are complete, recommendations are forwarded to a decision body or bodies. The recommendations will be one of the following (YESAB 2011):

- The project will not have significant adverse effects and should proceed;
- The project will have significant adverse effects that cannot be mitigated and should not proceed;
- The project should proceed with terms and conditions that will mitigate the effects; and
- The project should be assessed at a higher level. (Note: This can only occur when the assessment was done at the Designated Office (DO) or Executive Committee level.)

In some cases, assessments may also recommend project audits or effects monitoring.

20.8.2 Licensing

The project will be subject to territorial legislation, and will require a number of permits and approvals. The project may also be subject to federal legislation, depending upon specific project features and details.

Quartz Mine License

All hard rock mining claims are administered through the Quartz Mining Act (QMA) in Yukon. The QMA enables the Government of Yukon to issue licenses and regulate mining developments. The Government of Yukon Department of Energy, Mines and Resources administer the Quartz Mine License (QML) following the EA. Although permits and licenses cannot be issued in advance of completing the assessment, regulatory processes can be initiated simultaneously while the assessment is underway (Yukon Energy, Mines and Resources 2010).

Water License

The Yukon Water Board is responsible for licensing the use of water and the discharge of wastes into waters within the Yukon Territory under the *Yukon Waters Act* (Yukon Water Board 2006). The project will require a Type A water license.

Storage Tank Systems Permit

All fuel storage is regulated under the Storage Tank Regulation of the *Yukon Environment Act*. All storage tanks require a Storage Tank Permit and must be installed according to territorial and federal standards.

20.9 Socio-economic Considerations

20.9.1 First Nations and Project Location

The Wellgreen project and all infrastructure are located on Crown Land and potentially settlement lands within the traditional territory of the Kluane First Nation. Kluane First Nation is a self-governing nation with a settled land claim agreement.

20.9.2 Communities

The primary communities affected by the project and related infrastructure are Burwash Landing, Destruction Bay, Haines Junction and Beaver Creek. The project is located in western Yukon, within the Whitehorse Mining District kilometers north of Burwash Landing.

20.9.3 Studies and Consultation

Wellgreen initiated engagement with Kluane First Nation and White River First Nation beginning in 2010. An exploration co-operation agreement (ECA) was signed with Kluane First Nation August 1, 2012 and regular ECA meetings are held between the company and Kluane First Nation. In addition quarterly meetings have been facilitated by Wellgreen with Kluane First Nation and regulatory agencies to support the baseline monitoring plans to support the permitting. Wellgreen attended two community meetings in Burwash related to employment opportunities in the mining industry and a moose management workshop put on by Kluane First Nation. See Section 4.7 above for greater detail.

In 2015, Wellgreen Platinum will undertake data collection towards a socio-economic assessment.

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21 Capital and Operating Cost Estimates

21.1 Capital Cost Estimate

The CAPEX of the Wellgreen project has been estimated based on the scope defined in previous sections of this PEA. The following parties have contributed to the preparation of Phase 1 (25,000 t/d) and Phase 2 (50,000 t/d) CAPEX estimates in the following specific areas:

JDS:

- Process plant;
- Plant infrastructure and services, including road diversion, LNG power plant, LNG fuel storage, ancillary buildings, construction camp, and permanent camp;
- EPCM and Indirect costs relating to the process plant and infrastructure; and
- Owner's Costs.

SNC:

- Conduct pit optimization and mine planning and design;
- Select mining equipment;
- Establish potentially mineable resources; and
- Tailings management facility haulage cost estimation.
- Estimate mining CAPEX.

KP:

- Tailings management facility design and cost estimation.

21.1.1 Open Pit

The initial open pit mobile equipment pre-production capital considers the pre-strip year and year one for the initial capital estimate. The total initial CAPEX is US$83 million. Expansion and sustaining capital total US$182,000,000, excluding escalation, contingency and freight. The surface mobile equipment CAPEX is summarized in Table 21.1

Table 21.1: Open Pit Mobile Capital Expenditure

Year	TOTAL CAPEX	Trucks	Shovel	Loaders	Drills	Drills	Dozers	Dozers	Graders	Minor Fleet
	(M$)	793	2800	994	351E	271E	D10	RTD	16H	
n-1	$58.8	6	1	1		1	1	2	1	
1	$24.3	1			1		1	1		1
2	$8.2	2								
3	$23.1	1	1							
4	$9.5	1			1					
5 - 6	$0	0								
7	$14.1	2					1	2	1	
8	$24.5	6					1	1		
9	$8.2	2								
10	$8.2	2								
11	$23.1	1	1							
12	$17.0	2			1	1				
13	$30.7	0	1							1
14	$4.9	0						2	1	
15	$6.2	1						1		
16	$0	0								
17	$4.1	1								
18 - 20	$0									
21	$1.0								1	
22 - 25	$0									
Total	**$265.4**									

Source: SNC-Lavalin, 2015

21.1.2 Underground

A significant portion of the lateral and vertical development is in mineralization, some of which is to be placed in the low grade stockpile and the remainder of which is direct feed to the mill. Therefore, much of the lateral and vertical development is not capitalized since it commences after the site has attained commercial production and is generating revenues. Operating and capital development costs are provided in Table 21.2. For the purpose of this PEA all future rock development has been considered an operating cost. In future detailed technical reports this will be split between capital expenditures and operating expenditures.

Table 21.2: Stage 1 CAPEX Development Expenditures

Stage I Underground Capitalized Operating and Capital Development Expenditures						
	Operating Development (C$M)			CAPEX (C$M)		
	Year -2	Year -1	Sub-Total	Year -2	Year -1	Sub-Total
5.5x5 Ramp Development	17.1	17.1	34.2			0.0
5.5x5 Level Development	4.6	4.6	9.1			0.0
5.5x5 FA Access Drift		2.2	2.2			0.0
Escape-way			0.0		0.3	0.3
6.0m dia Fresh Air Raise (Surface)	2.9	2.9	5.8			0.0
PRV station			0.0	0.1		0.1
PRV Type 1			0.0	0.0		0.0
Booster Pump Station			0.0		0.1	0.1
Ramp Sump			0.0	2.4		2.4
Typical Level Sump			0.0		1.7	1.7
Drain Holes			0.0		0.2	0.2
Clean Water Sump			0.0		1.6	1.6
Secondary Settling Sump			0.0		0.6	0.6
Main Sump (2 dirty and 1 clean)			0.0		1.0	1.0
Main Pumping Station			0.0		1.6	1.6
Main Fuel/Lube Station			0.0		0.4	0.4
Satellite Fuel/Lube Station			0.0	0.2		0.2
Sat Stat			0.0	0.1		0.1
Backfill			0.0	2.0		2.0
Backfill Station			0.0	1.2		1.2
Brine System			0.0	0.5		0.5
Explosives Magazine			0.0	0.3		0.3
Fuse Magazine			0.0		0.1	0.1
Main Supplies Storage Bay			0.0		0.4	0.4
Satellite Warehouse			0.0		0.2	0.2
Satellite Garage			0.0	0.3		0.3
Storage Bay			0.0	0.2		0.2
Main Wash Bay			0.0		0.1	0.1
24 Person Refuge Station			0.0		0.8	0.8
Portable Refuge Station			0.0		0.3	0.3
Comfort Station - Large			0.0		0.4	0.4
Surface FA Fan			0.0		3.0	3.0
Surface RA Fan			0.0		2.0	2.0
Booster Fan			0.0		1.1	1.1
Switch Room Type A (civil only)			0.0		1.0	1.0
Switch Room Type B (civil only)			0.0		1.1	1.1
Switch Room Type C (civil only)			0.0		0.7	0.7
Mine systems			0.0		0.8	0.8
Automation			0.0		0.7	0.7
Total Direct	24.5	26.8	51.3	7.2	19.9	27.1
Engineering	0.8		0.8			0.0
Freight			0.0		3.3	3.3
Commissioning			0.0		0.8	0.8
Spare Parts			0.0		0.8	0.8
Total Indirect	0.8	0.0	0.8	0.0	4.9	4.9
Contingency (18%)				1.3	3.6	4.9
Total Direct and Indirect	25.4	26.8	52.1	8.5	28.4	37.0

Source: SNC-Lavalin, 2015

21.1.3 Site Development

Phase 1 & 2

Site development costs carried in Phase 1 include costs for earthworks as well as access and site roads to satisfy both phases of the project. Cost estimates are based on historical project experience

21.1.4 Process Plant

Phase 1 & 2

The process plant design for Phase 1 incorporates the primary gyratory crushing, coarse plant feed material stockpiling, secondary and tertiary crushing, secondary & tertiary screening, ball milling, rougher flotation, regrind and three levels of cleaner flotation, concentrate dewatering, and thickened tailings disposal.

The process plant design for Phase 2 is similar to Phase 1; however, Phase 2 incorporates larger flotation cells by reducing the quantity. In addition, Phase 2 will utilize the existing Primary Crusher installed during Phase 1 and eliminate the associated equipment costs.

The estimate has been prepared based on new budget quotes for major mechanical equipment and high level estimates for bulk take-offs on earthworks, concrete, internal steel and major pipelines.

Factors have been applied to cover in-plant electrical distribution, instrumentation, piping, and allowances for minor mechanical equipment and platework. Estimates for reagent systems, utility supply (air/water), PLC control, and fire protection have been based on database pricing.

Table 21.3: Process Plant Capital Costs

Process Plant Capital Costs	Phase 1: Pre-Production (C$M)	Phase 2: Expansion (C$M)	Total Capital Costs (C$M)
Primary Crushing & Material Storage	15.7	1.3	17.0
Secondary & Tertiary Crushing	19.2	17.6	36.8
Tertiary Screening	3.8	3.1	6.9
Grinding Area & Magnetic Separation	67.4	54.4	121.8
Flotation Area & Regrind	29.6	27.4	57.0
Concentrate Dewatering	4.9	3.4	8.3
Tailings Thickening	4.3	3.7	8.1
Reagents	5.0	1.8	6.8
Process Plant Utilities	4.2	2.8	7.0
Total	**154.2**	**115.5**	**269.7**

Source: JDS, 2015

In addition, sustaining capital is required for the processing plant throughout the life of mine.

Table 21.4: Process Plant Sustaining Capital Costs

Description	Sustaining / Closure (C$M per year)	Total Capital Costs (C$M)
Year 2	0.5	0.5
Year 3	1.0	1.0
Year 4	1.0	1.0
Year 8 to 20	1.5	19.5
Year 21	1.3	1.3
Year 22	1.0	1.0
Year 23	0.5	0.5
Total		**24.8**

Source: JDS, 2015

Earthworks and Civil Works

Phase 1

Earthwork material take offs (MTOs) were based on AutoCAD models and by using limited topographical survey information, and thus require further review when detailed topographical data becomes available. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Northern BC/Yukon.

Phase 2

Earthwork costs have been carried in Phase 1 as all major work will be completed prior to commencing Phase 2.

Concrete

Phase 1 & 2

Concrete MTOs were based on preliminary layouts and/or included as estimated allowances based on similar plants. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Northern BC and Yukon.

Mechanical Equipment

Phase 1

The following major process equipment was sized based on the design criteria and budget quotes were obtained, as detailed below in Table 21.5.

Table 21.5: Summary of Quoted Equipment

Equipment Description	Quote Vendor	Estimate (C$M)
Primary Gyratory Crusher	Metso	2.2
Secondary Screen	Metso	0.3
Secondary Cone Crusher	Metso	3.3
Tertiary Cone Crushers	Metso	6.6
Tertiary Screen	Metso	0.6
Ball Mills	Metso	22.5
Cyclone Feed & Tailings Pumps	Weir	1.1
Cyclopacs	Krebs	1.2
Flotation Cells & Mag Separator	Outotec	7.6
Total Quoted Equipment		**45.4**

Source: JDS, 2015

Phase 2

The following major process equipment was sized based on the design criteria and budget quotes were obtained, as detailed below in Table 21.6.

Table 21.6: Summary of Quoted Equipment

Equipment Description	Quote Vendor	Estimate (C$M)
Secondary Screen	Metso	0.4
Secondary Cone Crusher	Metso	3.8
Tertiary Cone Crushers	Metso	7.6
Tertiary Screen	Metso	0.7
Ball Mills	Metso	25.8
Cyclone Feed & Tailings Pumps	Weir	1.0
Cyclopacs	Krebs	1.4
Flotation Cells & Mag Separator	Outotec	5.9
Secondary Screen	Metso	0.4
Total Quoted Equipment		**47.0**

Source: JDS, 2015

The following equipment for both Phase 1 and 2 were sized based on the design criteria and estimates were determined based on database pricing:

- Rock Breaker;
- Conveyors;
- Reclaim Apron Feeders;
- Regrind Mill;
- Regrind Cyclopacs;
- Overhead Cranes;

- Flotation Air Blowers;
- PSA/OSA Analyzers;
- Concentrate Thickeners;
- Concentrate Filter Press;
- Tailings Thickener; and
- Reclaim Water Barge.

Structural Steelwork

Phase 1 & 2

Structural steelwork MTOs were based on preliminary layouts and/or included as estimated allowances based on similar plants. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Northern BC and Yukon.

The unit rate includes supply, shop detailing, fabrication, surface preparation and final painting in the shop, transport to site, site erection and paint touch-up as required.

Platework

Phase 1

The following equipment was sized based on the design criteria:

- Mag Cleaner and Rougher Flotation Conditioning Tanks;
- Fresh/Fire Water Tanks;
- Process Water Tank; and
- Potable Water Tank.

The remaining mechanical bulks, such as pumps, vessels and receivers, were factored as a percentage of overall mechanical costs for each area. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Northern BC and Yukon.

The unit rate includes supply, shop detailing, fabrication, surface preparation and final painting in the shop, transport to site, site erection, and paint touch-up.

Phase 2

The costs included for Phase 2 platework includes allowances for expanding the following Phase 1 tanks to allow more capacity:

- Mag Cleaner and Rougher Flotation Conditioning Tanks;
- Fresh/Fire Water Tanks;
- Process Water Tank; and
- Potable Water Tank.

The remaining mechanical bulks, such as pumps, vessels and receivers, were factored as a percentage of overall mechanical costs for each area. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Northern BC and Yukon.

Piping, Electrical and Instrumentation

Phase 1 & 2

Piping, Electrical and Instrumentation costs were factored from mechanical equipment pricing for the crushing and process plant areas based on actual historical factors for similar plants in Northern BC and Yukon.

Installation

Phase 1 & 2

JDS has allowed a local labour rate of $100/hr based on similar projects in the area with a productivity factor of 1.20. Labours rates are based on a 50-hour work week, which is typical for remote projects.

The labour rate includes the following items:

- Base rate per hour
- Sick time;
- Holiday pay;
- Insurance;
- Health and welfare;
- Small tools and consumables;
- Safety gear and clothing;
- Site supervision;
- Mobilization and demobilization;
- Transportation – turnaround;
- Site and head office overhead; and
- Contractor mark-up & profit.

The estimate is based on the majority of the work being carried out under fixed price or re-measurable unit price contracts under a normal development schedule. No allowance is included for contracts on a cost plus or fast-track accelerated schedule basis.

The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works would be performed by experienced contractors, using a mix of local and out-of-town labour to achieve the required quality and meet the project schedule.

Indirect costs, including project contingency have been provided for in the CAPEX estimates. Indirect costs have been estimated based on a factor of the total direct costs established from similar projects.

21.1.5 On-Site Infrastructure

The on-site Infrastructure required to support the plant operations is shown below in Table 21.7.

Table 21.7: On-Site Infrastructure Capital Costs

Process Plant Capital Costs	Phase 1: Pre-Production (C$M)	Phase 2: Expansion (C$)	Total Capital Costs (C$M)
Water Supply & Distribution	4.2	0.1	4.3
Electrical Supply & Distribution	45.0	24.1	69.1
Bulk Diesel / LNG Storage	10.4	8.5	18.6
Assay Laboratory	1.4	0.0	1.4
Construction Camp	16.2	13.5	29.6
Permanent Camp / STP / WTP	9.7	1.4	11.1
Admin Offices & Ancillary Facilities	4.9	0.0	4.9
Plant Mobile Fleet	3.8	.4	4.2
Total	**95.5**	**47.6**	**143.2**

Source: JDS, 2015

Potable Water Supply and Storage

A potable water skid will be part of the permanent camp. It is sized to provide water throughout the mine life including construction, Phase 1 and 2 operations. A large potable water tank will provide 2 hours of retention time. Tank dimensions are 9.3 m diameter by 9.3 m tall.

Water Storage

During the first phase of construction the following tanks will be erected:

- 9.3 m diameter by 9.3m tall fresh/firewater tank.
- 12.5 m diameter by 12.5 tall process water tank.

Both tanks are designed with a 2 hour retention time. Both will be carbon steel with external insulation for freeze protection.

During the second phase of production, rings will be added to both tanks to increase their capacity. This is more economical than erecting two new tanks.

Site Power Supply/Fuel Storage

The project requires a dedicated power generating station capable of delivering 36MW of uninterrupted power. JDS has worked closely with GE during this PEA. Initial power generation infrastructure for phase 1 of the project includes:

- 9 GEJenbacher J624 reciprocating engines capable of delivering 4.4 MW each;
- 2 additional engines will be added after initial production to provide N+2 redundancy;
- 5 – 60,000 gallon LNG storage bullets; and
- LNG loading and dispensing system.

During phase 2 will see the plant throughput double; however power requirements will increase by 27MW. This will be accomplished with the addition of the following equipment:

- 6 GEJenbacher J624 reciprocating engines capable of delivering 4.4 MW each; and

- 4 – 60,000 gallon LNG storage bullets.

Further optimization is possible when the newest generation of GE model J920 LNG fired engines become available.

Ancillary Buildings/Camps

Phase 1

The following ancillary buildings are included in the CAPEX estimate:

- Main administration building with medical center and training room;
- Assay Laboratory;
- Plan maintenance warehouse;
- Plant truck shop complete with minor equipment;
- Mine Dry;
- Construction Camp – Rental of 100 beds;
- Construction Camp – Catering 377 workers; and
- Permanent Camp – Purchase of 200 beds.

The costs of ancillary and support buildings for Phase 1 were estimated based on historical unit rates per area for similar projects. In addition to the building structures, the cost includes the supply of the buildings electrics, fittings, and furnishings. The construction and permanent camp costs were based on recent budgetary quotations for rental and purchase options. Earthworks required for the project have been carried in the overall site development. The total cost was estimated at $32.1M.

The cost to supply power and water services to the buildings and camps form part of the water and electrical supply and distribution costs. In addition, reagent storage facilities are included in the process plant cost estimate.

Phase 2

The following ancillary buildings are included in the CAPEX estimate:

- Construction Camp – Rental of 400 beds; and
- Construction Camp – Catering 260 workers.

Costs of ancillary and support buildings are not required as Phase 2 will utilize the structures completed during Phase 1. The construction and permanent camp costs were based on recent budgetary quotations for rental and purchase options. Earthworks required for the project have been carried in the overall site development. The total cost was estimated at $14.9M.

Mobile Equipment

Mobile fleet required to support plant operations shown below in Table 21.8.

Table 21.8: Plant Support Mobile Equipment CAPEX Estimate

Process Plant Capital Costs	Phase 1: Pre-Production (C$M)	Phase 2: Expansion (C$)	Total Capital Costs (C$M)
Mobile Equipment (Various)	3.8	0.4	4.2
Total	**3.8**	**0.4**	**4.2**

Source: JDS, 2015

21.1.6 Tailings Management Facility

Preliminary (conceptual level) design capital cost estimates have been developed for the construction of the TMF. The unit rates are derived from KP's experience on the design and construction of waste and water management facilities for mines in Canada.

The preliminary costs have been separated into Initial Capital (Stage 1) and Sustaining Capital (Stage 2 to 6). The Initial Capital costs are assumed to be pre-production expenditures and will be incurred before production begins (i.e., Years -1 and earlier). A breakdown of the Initial and Sustaining Capital costs of the TMF are presented below in Table 21.9.

Table 21.9: TMF Cost Summary

Description	Initial Capital (Pre-production) (M$) Stage 1	Sustaining Capital (Production) (M$) Stage 2 to 6	TOTAL (M$)
TMF Construction	$24	$167	$191
Haulage	$10	$130	$140
Tailings & water Reclaim & WTP	$11	$8	$19
Total	**$45**	**$305**	**$350**

Source: KP & JDS, 2015

21.1.7 Tailings & Reclaim Pipelines/Ponds

Tailings & Reclaim Pipeline Distribution Systems

High level engineering included sizing of pipelines for tailings, reclaim water and make-up water. Existing topography information was used to determine optimal routing of lines to avoid farm land and take advantage of existing infrastructure such as access roads. The pipelines have generally been designed to be free draining. There is an intermediate low point on the common ROW for the reclaim water and tailings pipelines. Therefore, allowance has been included for a lined emergency dump pond in this area when the pipelines need to be drained. The pond is sized so that each line can be drained twice.

Pipeline costs were based North American carbon steel and HDPE pipe supply pricing and Turnkey installation unit rates. An allowance of 20% has been included on pipeline supply costs to allow for fittings and valves.

Water Collection & Polishing Ponds

Earthwork MTOs were based on rough take-offs and water requirements. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Northern BC and Yukon.

21.1.8 TMF - Material Haulage

The Tailings Management Facility (TMF) will be constructed over 6 stages with waste rock material hauled from the pit. An estimated 117Mt of material is required for construction purposes. Haulage costs were estimated based on cycle times and truck hourly costs. Assumptions for calculating the haulage costs are summarized in Table 21.10.

Table 21.10: TMF Haulage Cost Assumptions

Description	Unit	Amount
In situ Rock Density	t/m^3	2.90
Placed TMF density	t/m^3	2.23
Truck load	t/m^3	220
Haul Cycle	hr	0.68
Truck Cost	$/hr	386.54
Truck Productivity	t/Yr	6,591
Total Tonnes	Mt	117.1
Total Haulage Cost	**$M**	**139.9**

The Tailings Management Facility construction schedule is summarized in Table 21.11.

Table 21.11: TMF Construction Schedule

Period	Total Tonnes	Stage of Construction
N-1	8,153,462	Stage 1 Complete
Yr1	2,285,795	
Yr2	2,285,795	
Yr3	2,285,795	Stage 2 Complete
Yr4	0	
Yr5	4,435,885	
Yr6	4,435,885	
Yr7	4,435,885	
Yr8	4,435,885	Stage 3 Complete
Yr9	7,593,329	
Yr10	6,644,298	
Yr11	17,474,989	Stage 4 Complete
Yr12	12,874,154	
Yr13	12,874,154	Stage 5 Complete
Yr14	15,745,968	
Yr15	6,960,744	
Yr16	4,175,520	Stage 6 Complete
Total	**117,097,538**	

21.1.9 Indirect Costs

Phase 1

Indirect costs and Owner's costs total an estimated $45.9M, equal to 13.9% of the total direct costs. The various cost centres that comprise the indirect costs are described in the following sections.

Phase 2

Indirect costs and Owner's costs total an estimated $26.7M, equal to 15.6% of the total direct costs. The various cost centres that comprise the indirect costs are described in the following sections.

Heavy Construction Equipment

Phase 1

Heavy Construction Equipment costs have been calculated to be $4.8M, which equates to 1.5% of the direct costs less mining equipment. Costs are intended to cover an 80t crane and miscellaneous heavy equipment for the duration of the project to support the construction.

Phase 2

Heavy Construction Equipment costs have been calculated to be $2.6M, which equates to 1.5% of the direct costs less mining equipment. Costs are intended to cover an 80t crane and miscellaneous heavy equipment for the duration of the project to support the construction.

Field Indirect Costs

Phase 1

Field indirect costs have been calculated to be $19.3M, which equates to 6.0% of the direct costs less mining equipment. Costs are intended to cover the following:

- Temporary Construction Facilities: work areas and bays, roads, walks and parking areas, temporary buildings, temporary utilities for power and sewage, other minor temporary construction; and
- Construction Services: general and final clean-up, material handling and warehousing, craft training and testing, onsite services (soils exploration and soil testing, all labour and material costs, concrete testing and security), operation and maintenance of temporary facilities, surveying, pre-operational testing and start-up.

Phase 2

Field indirect costs have been calculated to be $10.3M, which equates to 6.0% of the direct costs less mining equipment. Costs are intended to cover the following:

- Temporary Construction Facilities: work areas and bays, roads, walks and parking areas, temporary buildings, temporary utilities for power and sewage, other minor temporary construction; and
- Construction Services: general and final clean-up, material handling and warehousing, craft training and testing, onsite services (soils exploration and soil testing, all labour and material costs, concrete testing and security), operation and maintenance of temporary facilities, surveying, pre-operational testing and start-up.

Freight and Logistics

Phase 1

Freight and logistics have been calculated to be $11.3M, which equates to 7.0% of the equipment and material costs less mining equipment. Costs include ocean freight and inland freight, this figure is based on factored historical data for similar projects.

Phase 2

Freight and logistics have been calculated to be $5.7M, which equates to 7.0% of the equipment and material costs less mining equipment. Costs include ocean freight and inland freight, this figure is based on factored historical data for similar projects.

Vendor Representatives

Phase 1

Vendor representatives have been calculated to be $2.1M, which equates to 2% of the equipment and material costs less mining. This figure is based on factored historical data for similar projects.

Phase 2

Vendor representatives have been calculated to be $1.6M, which equates to 2% of the equipment and material costs less mining. This figure is based on factored historical data for similar projects.

Start-Up & Commissioning / Capital Spares

Phase 1

Start-Up & Commissioning/Capital Spares have been calculated to be $4.2M, which equates to 4% of the equipment and material costs less mining equipment. This figure is based on factored historical data for similar projects.

Phase 2

Start-Up & Commissioning/Capital Spares have been calculated to be $3.3M, which equates to 4% of the equipment and material costs less mining equipment. This figure is based on factored historical data for similar projects.

First Fills

Phase 1

First fills have been calculated to be $2.1M, which equates to 2% of the equipment and material costs less mining equipment. This figure is based on factored historical data for similar projects.

Phase 2

First fills have been calculated to be $1.6M, which equates to 2% of the equipment and material costs less mining equipment. This figure is based on factored historical data for similar projects.

21.1.10 EPCM

Phase 1

EPCM services have been calculated to be $30.6M or 8% of the direct and indirect costs, which includes detailed engineering, procurement, project management and home office services as well as construction management. This was calculated on direct and indirect costs excluding mine equipment and mine development.

Phase 2

EPCM services have been calculated to be $15.8M or 8% of the direct and indirect costs, which includes detailed engineering, procurement, project management and home office services as well as construction management. This was calculated on direct and indirect costs excluding mine equipment and mine development.

21.1.11 Owner's Cost

Phase 1

For the purpose of this PEA estimate, $9.7M or 2.5% of the direct costs were selected to cover the Owner's Costs, which includes Insurance, Owner's team costs, pre-production, and Project development. This figure is based on factored historical data for similar projects.

Phase 2

No Owner's costs have been carried in Phase 2 of the project.

21.1.12 Contingency

The contingency reflects the potential growth in CAPEX within the same scope of work. The contingency includes variations in quantities, differences between estimated and actual equipment and material prices, labour costs and site-specific conditions. It also accounts for variation resulting from uncertainties that are clarified during detail engineering, when designs and specifications of the basic engineering scope are finalized.

Contingency is an amount of money allowed in an estimate for cost which, based on past experience, are likely to be encountered, but are difficult or impossible to identify at the time the estimate is prepared. It is an amount expected to be expended during the course of the project. Contingency does not include scope changes, force majeure, labour disruptions or lack of labour availability.

A contingency of 25% for all capital was used with the exception of mining equipment. The purpose of the contingency provision is to make allowance for uncertain elements of costs to cover such factors as:

- Limited information on site conditions, especially concerning sub-surface conditions and the engineering properties of excavated materials;
- Completeness and accuracy of quantity take-offs and estimate assembly and consolidation based on the level of engineering and design undertaken at study level;
- Accuracy of materials and labour rates (excluding extreme variations that would be covered under contingency);
- Accuracy of productivity expectations; and accuracy of equipment pricing.

Phase 1

Major cost categories (permanent equipment, material purchase, installation, subcontracts, pipelines, indirect costs and Owner's costs) were identified and analyzed. An overall contingency of $101.8M was obtained, representing 25% of the total CAPEX.

Phase 2

Major cost categories (permanent equipment, material purchase, installation, subcontracts, pipelines, indirect costs and Owner's costs) were identified and analyzed. An overall contingency of $53.4M was obtained, representing 25% of the total CAPEX.

21.1.13 Duties and Taxes

Local taxes on contractor-supplied materials and installation labour are not included in the estimate.

21.1.14 Escalation

No escalation costs have been included in Phase 1 or 2 of the project, all costs and prices are expressed in Q4 2014 dollars.

21.1.15 Sustaining Capital and Closure Costs

Ongoing capital requirement for the mine production period totals $970.5M over the mine life. This cost covers the phased construction of the tailings management facility, closure costs, mining, and process plant equipment to sustain the ongoing operation of the project.

21.1.16 Capital Cost Summary

All CAPEX costs are expressed in Q4 2014 CDN dollars. There are no allowances for escalation in Phase 1 or 2 of the project. The estimated costs include mine pre-stripping, mine development, site preparation, process plant, first fills, buildings, ancillary facilities, road works, power plant and utilities. The estimates are considered to have an overall accuracy of +/-30% and assume the project would be developed on an EPCM basis. Table 21.10 shows pre-production and sustaining CAPEX and Table 21.11 provides a summary of the Phase 1 CAPEX and Phase 2 Expansion estimates.

Table 21.12: Initial & Sustaining CAPEX Estimate

Capital Cost	Pre-Production (C$M)	Sustaining/Closure (C$M)	Total Capital Costs (C$M)
Mining Equipment	58.8	206.6	265.4
OP Mine OPEX during Pre-Production	16.1	0.0	16.1
UG Mine Development	0.0	37.0	37.0
Site Development	36.8	0.0	36.8
Process Plant	154.2	140.2	294.4
On-Site Infrastructure	89.7	53.4	143.2
Tailings Management Facility	35.5	175.1	210.5
TMF Waste Haulage	9.7	130.2	139.9
Indirects	45.2	27.4	72.6
EPCM	30.2	16.3	46.4
Owner's Costs	9.6	0.1	9.7
Closure	0.0	60.0	60.0
Subtotal	**485.9**	**846.3**	**1,332.2**
Contingency (25%)	100.3	118.1	218.4
Total Capital Cost	**586.2**	**964.4**	**1,550.6**

Source: JDS, 2015

Table 21.12: Initial & Sustaining CAPEX Estimate

Capital Cost	Pre-Production (C$M)	Sustaining/Closure (C$M)	Total Capital Costs (C$M)
Mining Equipment	58.8	206.6	265.4
OP Mine OPEX during Pre-Production	16.1	0.0	16.1
UG Mine Development	0.0	37.0	37.0
Site Development	36.8	0.0	36.8
Process Plant	154.2	140.2	294.4
On-Site Infrastructure	89.7	53.4	143.2
Tailings Management Facility	35.5	175.1	210.5
TMF Waste Haulage	9.7	130.2	139.9
Indirects	45.2	27.4	72.6
EPCM	30.2	16.3	46.4
Owner's Costs	9.6	0.1	9.7
Closure	0.0	60.0	60.0
Subtotal	**485.9**	**846.3**	**1,332.2**
Contingency (25%)	100.3	118.1	218.4
Total Capital Cost	**586.2**	**964.4**	**1,550.6**

Source: JDS, 2015

Table 21.13: CAPEX Estimate by Phase

Capital Cost	Phase 1 (C$M)	Phase 2 (C$M)	Sustaining / Closure (C$M)	Total Capital Costs (C$M)
Mining Equipment	58.8	0.0	206.6	265.4
OP Mine OPEX during Pre-Production	16.1	0.0	0.0	16.1
UG Mine Development	0.0	0.0	37.0	37.0
Site Development	36.8	0.0	0.0	36.8
Process Plant	154.2	115.5	24.7	294.4
On-Site Infrastructure	89.7	47.6	5.9	143.2
Tailings Management Facility	35.5	23.1	151.9	210.5
TMF Waste Haulage	9.7	10.6	119.6	139.9
Indirects	45.2	31.2	0.7	77.1
EPCM	30.2	16.2	0.5	46.4
Owner's Costs	9.6	0.0	0.1	9.7
Closure	0.0	0.0	60.0	60.0
Subtotal	**485.9**	**244.2**	**607.1**	**1,337.2**
Contingency (25%)	100.3	58.4	60.9	218.4
Total Capital Cost	**586.2**	**302.6**	**667.9**	**1,556.6**

Source: JDS, 2015

The following parameters and qualifications are made:

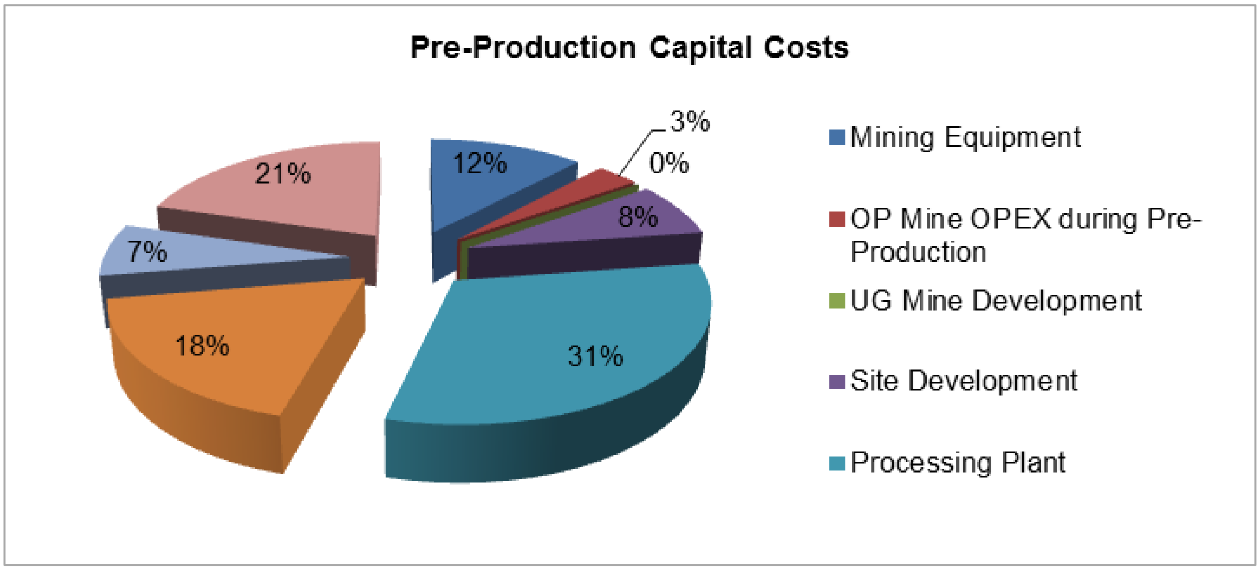
- Estimate was based on the fourth quarter of 2014 prices and costs; and
- No allowance has been made for exchange rate fluctuations over the life of the mine.
- Data for these estimates has been obtained from numerous sources, including:
- PEA-level engineering design;
- Budgetary equipment and infrastructure quotations;
- QP experience; and
- Data from recently completed similar studies and projects.
- The following assumptions were used in the CAPEX estimates:
- The detail of the design is discussed in the relevant sections of this report;
- Mining costs estimated by SNC are correct;
- Benchmarked plant CAPEX estimates are correct;
- Suitably qualified and experienced construction labour would be available at the time of execution of the project;
- Qualified construction personnel are available in the local community to assist the project;
- No geotechnical and drainage issues, therefore, no allowance for special ground preparation was made;
- Borrow sources for construction are available from within the mine limits;
- TMF costs estimated by KP are correct;
- TMF material haulage costs estimated by SNC are correct;
- A power and water supply capable of supplying the required demand of the processing plant is assumed to be available; and
- No extremes in weather would be experienced during the construction phase and as such, no allowances are included for construction-labour stand-down costs.

The following items are excluded from the estimate:

- Cost changes due to currency fluctuation;
- Force majeure issues;
- Sunk costs up to the project go-no go decision point. The costs that are excluded encompass pre-feasibility study costs, feasibility study costs, resource definition drilling, EIA work, metallurgical testing, hydrogeological and geotechnical drilling and testwork and all other work associated with a feasibility study and EIA ;
- Future scope changes;
- Project insurances;
- Project interest and financing cost;
- Land acquisition and compensation cost;
- Operational insurances such as business interruption insurance and machinery breakdown;
- Public road maintenance; and

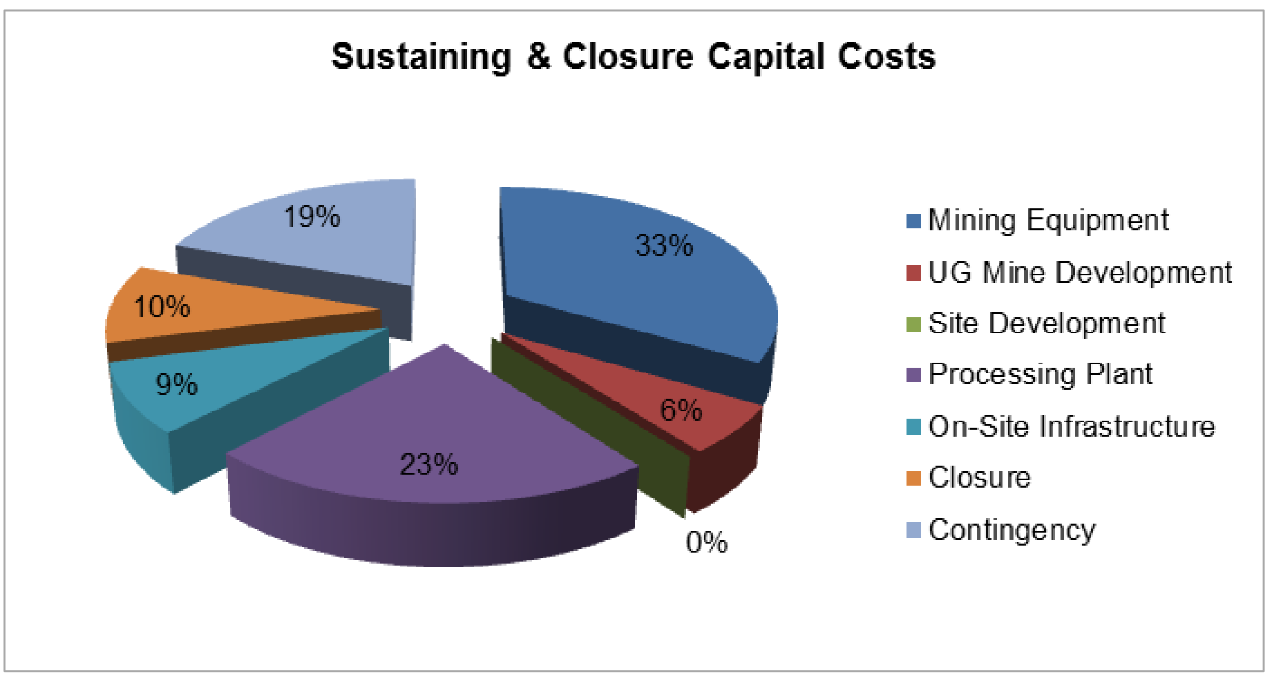
- Relocation or preservation costs, delays and redesign work associated with any antiquities and sacred sites.

Figure 21.1: Pre-Production Capital Costs



Source: JDS, 2015

Figure 21.2: Sustaining & Closure Capital Costs



Source: JDS, 2015

21.2 Operating Cost Estimates

Total life of mine operating costs amount to $7,169.2M. This translates to an average cost of $18.52/tonne processed. A breakdown of these costs is shown in 7 and Figure 21.3.

21.2.1 Open Pit

Open pit operating costs were developed from first principles based on equipment requirements and SNC-Lavalin experience with similar mining projects. Costs included in this estimate include labour, fuel consumption, power, consumables, maintenance, and overhaul costs. A breakdown of the operating cost is summarized in Table 21.14

Table 21.14: Open Pit Operating Cost

Function	(C$/t)
Drilling	0.45
Loading	0.16
Hauling	0.93
Dozing	0.20
Other	0.43
Total	2.16

Source: SNC, 2015

Manpower requirements for the open pit major mining equipment are listed in Table 21.13. Manpower is calculated based on four crews operating on a two week on, two off rotation and considers equipment availability. This excludes supervision, engineering, G&A, maintenance and mill staff.

Table 21.15: Open Pit Mobile Equipment Manpower

Year	Headcount
N-1	33
1	61
2	71
3	73
4	92
5	103
6	105
7	90
8	89
9	91
10	91
11	164
12	149
13	149
14	156
15	88
16	80
17	38
18 to 25	23

Source: SNC, 2015

Operating costs assume mobile equipment will be provided by the contractor during underground development and mining. Operating costs for the short term underground production program were developed by SNC, benchmarking various similar operations with a mark-up applied due to the use of contractor mining (Table 21.16).

21.2.2 Underground

A significant portion of the lateral and vertical development is in mineralization, some of which is to be placed in the low grade stockpile and the remainder of which is direct feed to the mill. Therefore, much of the lateral and vertical development is not capitalized since it commences after the site has attained commercial production and is generating revenues. Operating and capital development costs are provided in Table 21.16. For the purpose of this PEA all future rock development has been considered an operating cost. In future detailed technical reports this will be split between Capital and Operating.

Table 21.16: Underground Operating Cost

	Contractor Unit Mining Cost (CDN$/tonne)
Open Stope	48.4
Post Pillar C&F	54.5

Source: SNC, 2015

21.2.3 Process Plant

Process operating costs are summarized in Table 21.17.

Table 21.17: Process Operating Cost Estimate

Operating Costs	Unit	25 ktpd	50 ktpd
Labour	$/t	0.84	0.48
Power	$/t	4.16	4.16
Consumables	$/t	8.95	8.95
Total		**13.95**	**13.60**

Source: JDS, 2015

21.2.4 General and Administration

G&A costs are summarized in Table 21.18.

Table 21.18: General & Administration Cost Estimate

Operating Costs	Unit	25 ktpd	50 ktpd
Labour	$/t	0.50	0.31
Equipment	$/t	0.12	0.10
Materials	$/t	0.05	0.04
Expenses	$/t	0.39	0.27
Services	$/t	0.60	0.20
Total		**1.66**	**0.91**

Source: JDS, 2015

21.2.5 Operating Cost Summary

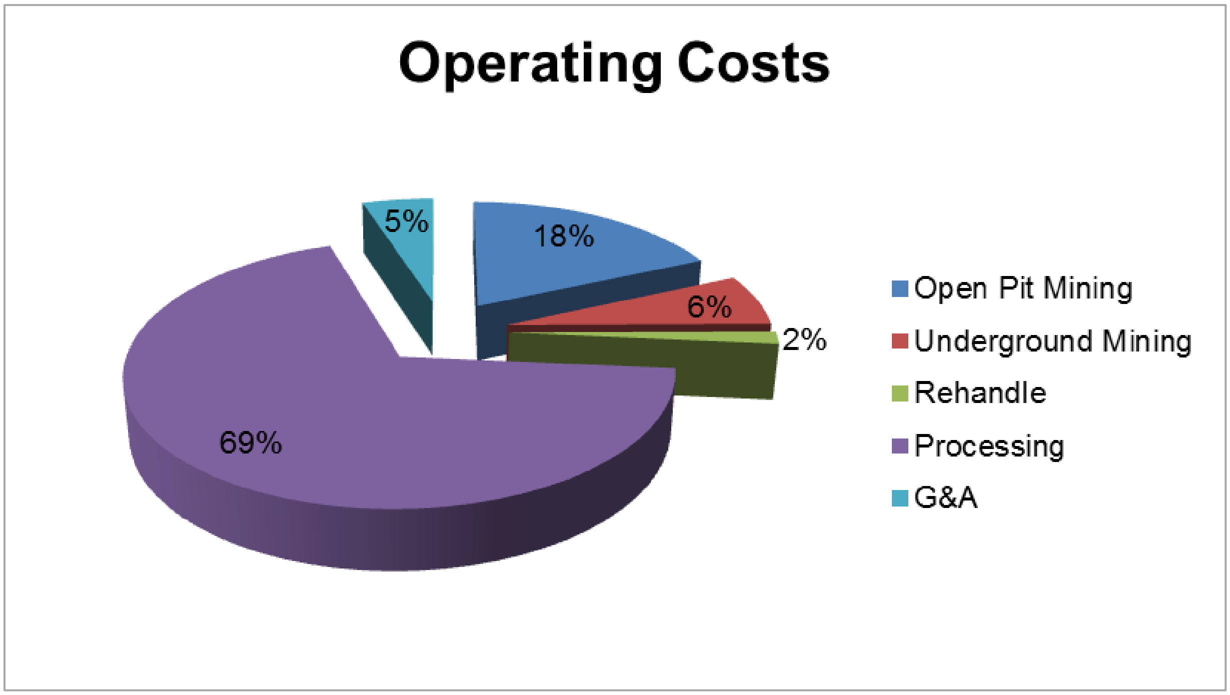
Total life of mine operating costs amount to $7,169.2M. This translates to an average cost of $18.52/tonne processed. A breakdown of these costs is shown in Table 21.19 and Figure 21.3.

Table 21.19: Summary of Operating Costs

Operating Costs	C$/ milled	C$/ mined	Average C$M/Yr	LOM C$M
Open Pit Mining‡	3.65	2.10	58.7	1,466.3
Underground Miningº	1.29	0.74	14.6	516.2
Re-handle*	0.31	0.18	5.5	125.5
Processing	13.64	7.85	231.6	5,474.0
G&A	0.99	0.57	16.2	399.2
Total	19.88	11.44	326.6	**7,981.2**

Source: JDS, 2015
(‡) Open Pit Mining Costs are based on $2.13/t mined and a 0.8 strip ratio
(º) Underground Mining Costs are based on $54.49/t mined
(*) Re-handle cost is based on $0.75/tonne re-handled. Total material re-handled amounts to 167.3M tonnes over the LOM.

Figure 21.3: Distribution of Operating Costs



Source: JDS, 2015

22 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, head grades, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This PEA contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment of skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of CAPEX and OPEX have been developed specifically for this project and are summarized in Section 21 of this report (presented in 2014 dollars). The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

Three metal price scenarios were utilized to prepare the economic analysis. In addition, a sensitivity analysis was completed of various factors including metal prices.

All revenues, costs and economic results are presented in Canadian dollars (C$) unless otherwise noted. Metal prices are stated in US dollars (US$). LOM plan tonnage and grade estimates are demonstrated in Table 22.1

Table 22.1: Life of Mine Plan Summary

Summary of Results	Unit	LOM Value
Mine Life	Years	25
Total Mineralized Resource	M tonnes	402
Total Waste	M tonnes	296
Total Material Mined	M tonnes	697
Strip Ratio	w:o	0.8
Processing Rate	LOM Average tpd	47,154
	LOM Average M tpa	17

Average Head Grades	Unit	Years 1-5	Years 6-16	Years 17-25 (Stockpiles)	LOM Value
Ni	%	0.32	0.27	0.21	0.26
Cu	%	0.31	0.15	0.08	0.14
Co	%	0.02	0.01	0.01	0.01
Pt	g/t	0.434	0.259	0.143	0.234
Pd	g/t	0.346	0.271	0.173	0.241
Au	g/t	0.087	0.045	0.025	0.042
Ni Eq*	%	0.65	0.48	0.33	0.44
Pt Eq*	g/t	2.47	1.80	1.26	1.67

(*) Metal equivalent grade calculation are based on metal prices used in the Base Case Scenario
Source: JDS, 2015

Other economic assumptions used in the economic analysis include the following:

- Discount rate of 7.5% (sensitivities using other discount rates have been calculated)
- Closure cost of $75M, including $15M of contingencies, was considered and occurs during Year 24 to Year 28;
- Nominal 2014 Canadian dollars;
- Revenues, costs and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Working capital was calculated as two months of operating costs (mining, processing, G&A) in Year 1 (assumed to be required in Year -1). The working capital is recuperated during the last year of production. Total working capital amounts to $35.5M;
- Results are presented on a 100% equity basis; and
- No management fees or financing costs have been considered.

The economic analysis excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.).

Table 22.2 outlines the metal price assumptions used in the economic analysis.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there are no guarantees that they will be realized if the project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 22.2: Metal Prices used in the Economic Analysis

Commodity	Unit	Base Case	Peer Base Case Prices	Long Term Consensus Forecast	Spot Prices as at Feb. 2, 2015
Nickel	US$/lb	8.00	8.34	8.74	6.83
Copper	US$/lb	3.00	3.21	3.18	2.51
Cobalt	US$/lb	14.00	14.00	12.93	13.38
Platinum	US$/oz	1,450	1,642	1,450	1,223
Palladium	US$/oz	800	775	950	773
Gold	US$/oz	1,250	1,350	1,148	1,273
F/X Rate	USD:CAD	0.90	0.93	0.88	0.80

Source: JDS, 2015

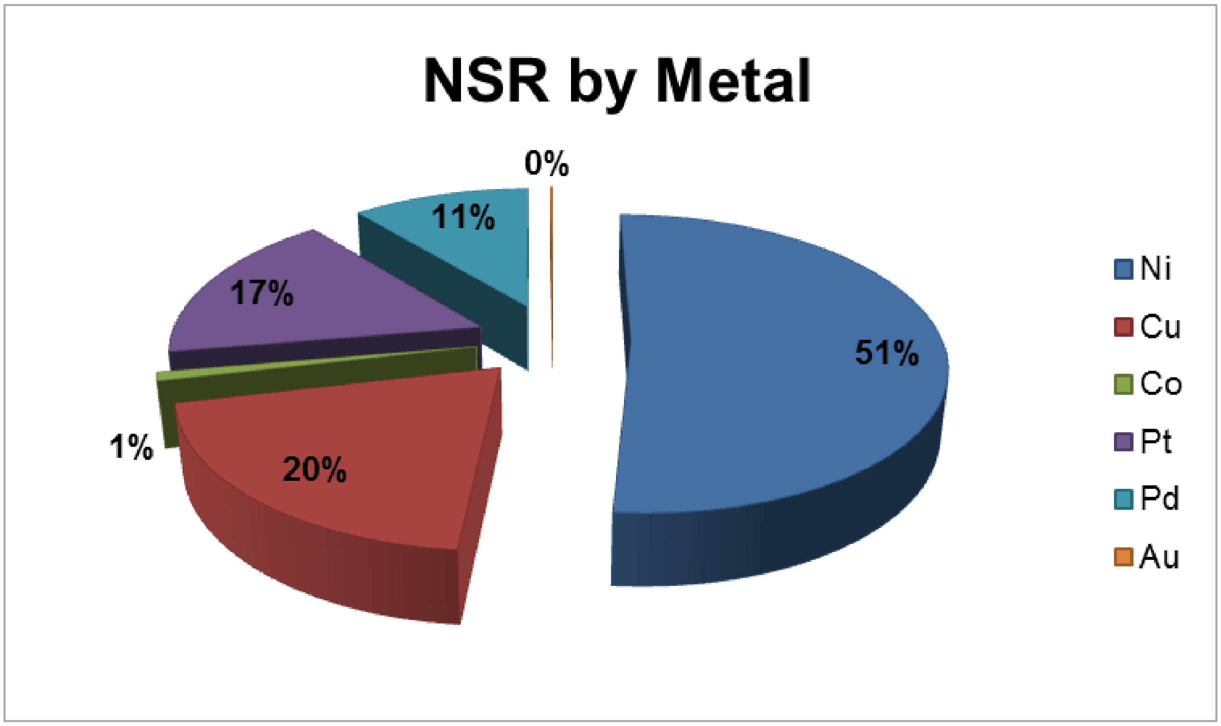
22.2 Revenues & NSR Parameters

Mine revenue is derived from the sale of nickel concentrate into the international marketplace. No contractual arrangements exist at this time. Details regarding the terms used in the economic analysis can be found in Section 19 of this report. Total smelter revenues amount to $15,507.2M over the 25-year mine life in the Base Case scenario.

Figure 22.1 demonstrates the distribution of revenues by metal.

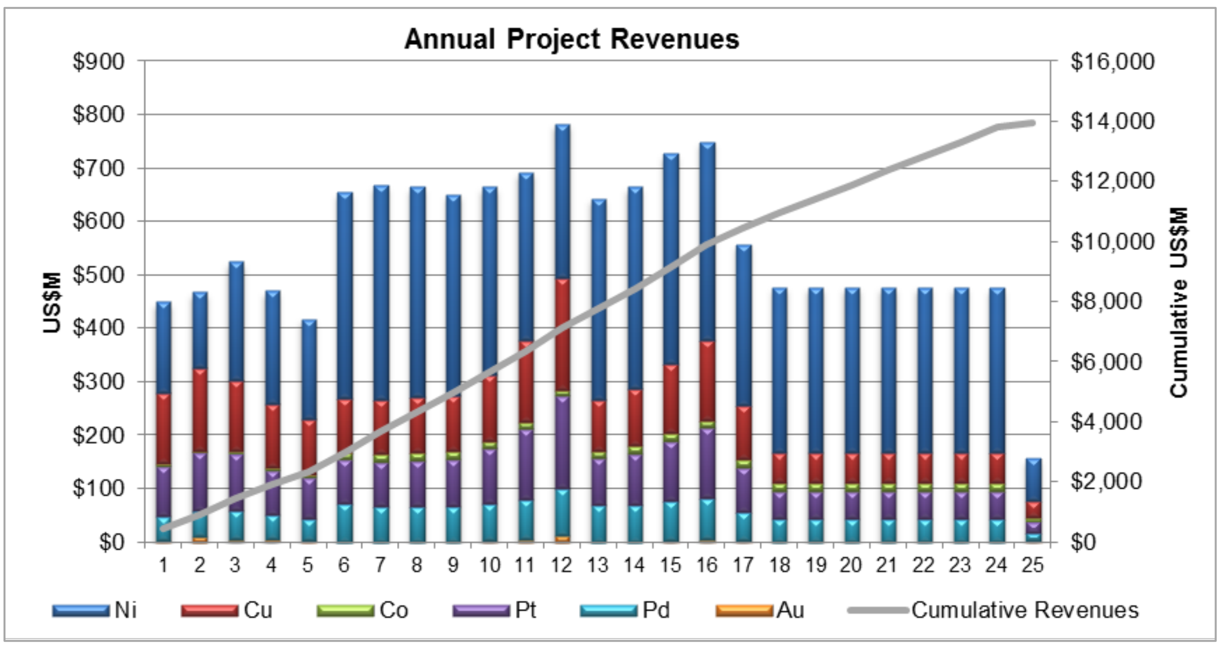
Total smelter revenues amount to $15,507.2M over the 25-year mine life in the Base Case scenario.

Figure 22.1: Life of Mine Net Revenues by Metal in the Base Case Scenario



Source: JDS, 2015

Figure 22.2: Annual Project Revenues in the Base Case Scenario



Source: JDS, 2015

22.3 Taxes

The Wellgreen project has been evaluated on an after-tax basis in order to provide a more indicative value of the potential project economics. A specialized mining tax professional was commissioned to review and assist in preparing a tax model for the post tax economic evaluation of the project with the inclusion of applicable federal and provincial income taxes. The tax calculations account for opening tax pools, Yukon Quartz Mining Royalties, provincial and federal income taxes. The tax calculations also assume appropriate capital cost allowance for each of the capital cost class. Total taxes for the life of the project amount to $2,265.4M for the Base Case scenario.

22.4 Economic Results

The Wellgreen project is economically viable with an after-tax internal rate of return (IRR) of 25.3% and a net present value using a 7.5% discount rate ($NPV_{7.5\%}$) of $1,216.9M using the Base Case metal prices.

Table 22.3 summarizes the economic results of the project for all metal price scenarios.

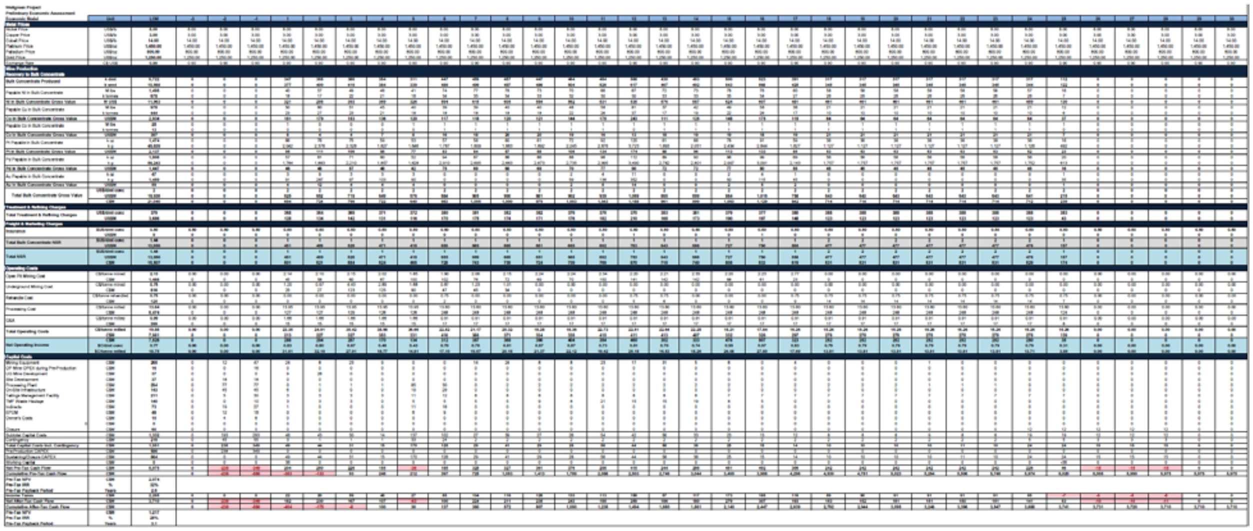
This preliminary economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Table 22.3: Summary of Economic Results

Summary of Results	Unit	Base Case Scenario	Peer Base Case Prices	Long Term Consensus Forecast	Spot Prices as a Feb. 2, 2015
Total LOM Pre-Tax Free Cash Flow	$M	5,975.3	6,451.2	8,112.8	4,716.9
Average Annual Pre-Tax Free Cash Flow	$M/Yr	239.0	258.0	324.5	188.7
LOM Income Taxes	$M	2,265.4	2,447.5	3,085.1	1,786.0
Total LOM After-Tax Free Cash Flow	C$M	3,710.0	4,003.8	5,027.7	2,930.9
Average Annual After-Tax Free Cash Flow	$M/Yr	148.4	160.2	201.1	117.2
Discount Rate	**%**	**7.5**	**7.5**	**7.5**	**7.5**
Pre-Tax NPV	**$M**	**2,073.6**	**2,934.1**	**2,966.0**	**1,500.0**
Pre-Tax IRR	**%**	**32.4**	**41.6**	**41.5**	**25.8**
Pre-Tax Payback	**Years**	**2.6**	**2.0**	**2.0**	**4.4**
After-Tax NPV	**$M**	**1,216.9**	**1,749.6**	**1,769.3**	**859.1**
After-Tax IRR	**%**	**25.3**	**32.1**	**32.1**	**20.4**
After-Tax Payback	**Years**	**3.1**	**2.4**	**2.4**	**6.3**

Source: JDS, 2015

Figure 22.3: Cash Flow Model



Source: JDS, 2015

Cash Flow Model Preview Oonly – See 11x17 layout next page.

22.5 Sensitivities

A sensitivity analysis was performed on the Base Case metal pricing scenarios to determine which factors most affect the project economics. The analysis revealed that the project is most sensitive to metal prices and foreign exchange rate, followed by head grade and operating costs. The project showed least sensitive to capital costs. Table 22.4 along with Figure 22.7 outline the results of the sensitivity test performed on the after-tax $NPV_{7.5\%}$ for the Base Case evaluated.

The project was also tested under various discount rates. The results of this sensitivity test are demonstrated in Table 22.5.

Table 22.4: Sensitivity Results for Base Case NPV

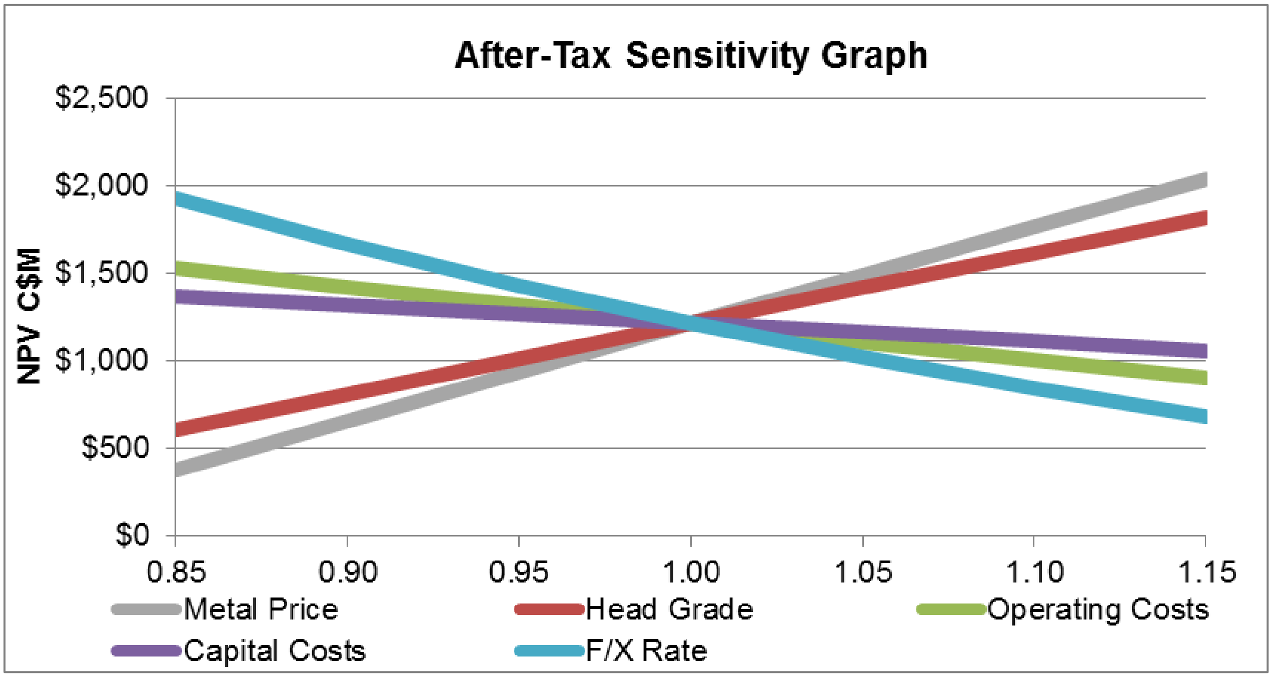
	After-Tax $NPV_{7.5\%}$ (C$M)						
Variable	**-15%**	**-10%**	**-5%**	**100%**	**+5%**	**+10%**	**+15%**
Metal Price	379	663	941	1,217	1,492	1,765	2,039
F/X Rate	1,928	1,665	1,430	1,217	1,024	848	686
Head Grade	606	811	1,014	1,217	1,419	1,620	1,821
Operating Costs	1,530	1,426	1,322	1,217	1,112	1,007	901
Capital Costs	1,373	1,321	1,269	1,217	1,165	1,113	1,061

Source: JDS, 2015

Wellgreen Project
Preliminary Economic Assessment

Economic Model	Unit	LOM	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Metal Prices																																			
Nickel Price	US$/lb	**8.00**	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00
Copper Price	US$/lb	**3.00**	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Cobalt Price	US$/lb	**14.00**	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00	14.00
Platinum Price	US$/oz	**1,450.00**	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00	1,450.00
Palladium Price	US$/oz	**800.00**	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00	800.00
Gold Price	US$/oz	**1,250.00**	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00
Exchange Rate	C$:US$	**0.90**	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
Mine Production																																			
Recovery to Bulk Concentrate																																			
Bulk Concentrate Produced	**k dmt**	**9,722**	**0**	**0**	**0**	**347**	**368**	**386**	**354**	**311**	**447**	**459**	**457**	**447**	**464**	**484**	**568**	**430**	**453**	**500**	**523**	**391**	**317**	**317**	**317**	**317**	**317**	**317**	**317**	**112**	**0**	**0**	**0**	**0**	**0**
	k wmt	**10,568**	**0**	**0**	**0**	**377**	**400**	**419**	**384**	**339**	**486**	**499**	**497**	**486**	**504**	**526**	**617**	**467**	**492**	**543**	**568**	**425**	**345**	**345**	**345**	**345**	**345**	**345**	**344**	**122**	**0**	**0**	**0**	**0**	**0**
Payable Ni in Bulk Concentrate	M lbs	**1,495**	0	0	0	40	37	49	46	41	74	77	76	73	70	66	67	72	73	78	76	60	58	58	58	58	58	58	57	16	0	0	0	0	0
	k tonnes	**678**	0	0	0	18	17	22	21	18	34	35	34	33	32	30	30	33	33	35	34	27	26	26	26	26	26	26	26	7	0	0	0	0	0
Ni in Bulk Concentrate Gross Value	**M US$**	**11,963**	**0**	**0**	**0**	**321**	**298**	**393**	**369**	**326**	**594**	**615**	**605**	**584**	**562**	**531**	**535**	**576**	**587**	**624**	**607**	**481**	**461**	**461**	**461**	**461**	**461**	**461**	**459**	**130**	**0**	**0**	**0**	**0**	**0**
Payable Cu in Bulk Concentrate	M lbs	**978**	0	0	0	50	60	51	45	40	39	39	40	40	48	58	81	37	42	49	58	38	21	21	21	21	21	21	21	12	0	0	0	0	0
	k tonnes	**444**	0	0	0	23	27	23	21	18	18	18	18	18	22	26	37	17	19	22	26	17	10	10	10	10	10	10	10	6	0	0	0	0	0
Cu in Bulk Concentrate Gross Value	**US$M**	**2,934**	**0**	**0**	**0**	**151**	**179**	**153**	**136**	**120**	**117**	**118**	**120**	**121**	**144**	**175**	**243**	**111**	**125**	**148**	**175**	**115**	**64**	**64**	**64**	**64**	**64**	**64**	**64**	**37**	**0**	**0**	**0**	**0**	**0**
Payable Co in Bulk Concentrate	M lbs	**28**	0	0	0	1	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0
	k tonnes	**13**	0	0	0	0	0	0	0	0	1	1	1	1	1	1	0	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Co in Bulk Concentrate Gross Value	**US$M**	**397**	**0**	**0**	**0**	**8**	**4**	**4**	**7**	**8**	**16**	**18**	**20**	**20**	**19**	**16**	**13**	**18**	**19**	**19**	**18**	**19**	**21**	**21**	**21**	**21**	**21**	**21**	**21**	**7**	**0**	**0**	**0**	**0**	**0**
Pt Payable in Bulk Concentrate	k oz	**1,474**	0	0	0	66	76	75	59	53	57	58	60	61	72	92	120	61	66	78	91	59	36	36	36	36	36	36	36	16	0	0	0	0	0
	k g	**45,839**	0	0	0	2,042	2,376	2,328	1,827	1,646	1,787	1,809	1,863	1,892	2,245	2,876	3,723	1,893	2,051	2,434	2,844	1,827	1,127	1,127	1,127	1,127	1,127	1,127	1,126	492	0	0	0	0	0
Pt in Bulk Concentrate Gross Value	**US$M**	**2,137**	**0**	**0**	**0**	**95**	**111**	**109**	**85**	**77**	**83**	**84**	**87**	**88**	**105**	**134**	**174**	**88**	**96**	**113**	**133**	**85**	**53**	**53**	**53**	**53**	**53**	**53**	**52**	**23**	**0**	**0**	**0**	**0**	**0**
Pd Payable in Bulk Concentrate	k oz	**1,808**	0	0	0	57	61	71	60	52	94	87	86	86	88	96	112	89	90	96	99	69	56	56	56	56	56	56	56	20	0	0	0	0	0
	k g	**56,243**	0	0	0	1,784	1,883	2,210	1,857	1,628	2,910	2,695	2,663	2,673	2,735	2,995	3,490	2,782	2,801	2,997	3,091	2,143	1,757	1,757	1,757	1,757	1,757	1,757	1,752	613	0	0	0	0	0
Pd in Bulk Concentrate Gross Value	**US$M**	**1,447**	**0**	**0**	**0**	**46**	**48**	**57**	**48**	**42**	**75**	**69**	**68**	**69**	**70**	**77**	**90**	**72**	**72**	**77**	**80**	**55**	**45**	**45**	**45**	**45**	**45**	**45**	**45**	**16**	**0**	**0**	**0**	**0**	**0**
Au Payable in Bulk Concentrate	k oz	**47**	0	0	0	3	9	3	3	3	0	0	0	0	2	4	11	0	0	2	4	1	0	0	0	0	0	0	0	1	0	0	0	0	0
	k g	**1,459**	0	0	0	91	287	97	103	90	0	0	0	0	59	136	352	0	0	50	115	45	0	0	0	0	0	0	0	35	0	0	0	0	0
Au in Bulk Concentrate Gross Value	**US$M**	**59**	**0**	**0**	**0**	**4**	**12**	**4**	**4**	**4**	**0**	**0**	**0**	**0**	**2**	**5**	**14**	**0**	**0**	**2**	**5**	**2**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1**	**0**	**0**	**0**	**0**	**0**
Total Bulk Concentrate Gross Value	**US$/dmt conc**	**2**	**0**	**0**	**0**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**0**	**0**	**0**	**0**	**0**
	US$M	**18,936**	**0**	**0**	**0**	**625**	**652**	**719**	**649**	**576**	**884**	**905**	**900**	**881**	**902**	**939**	**1,069**	**865**	**899**	**984**	**1,016**	**758**	**643**	**643**	**643**	**643**	**643**	**643**	**641**	**215**	**0**	**0**	**0**	**0**	**0**
	C$M	**21,040**	**0**	**0**	**0**	**694**	**724**	**799**	**722**	**640**	**983**	**1,005**	**1,000**	**979**	**1,003**	**1,043**	**1,188**	**961**	**999**	**1,093**	**1,129**	**842**	**714**	**714**	**714**	**714**	**714**	**714**	**712**	**239**	**0**	**0**	**0**	**0**	**0**
Treatment & Refining Charges																																			
Total Treatment & Refining Charges	**US$/dmt conc**	**379**	**0**	**0**	**0**	**368**	**364**	**369**	**371**	**372**	**380**	**381**	**382**	**382**	**379**	**375**	**370**	**383**	**381**	**379**	**377**	**380**	**388**	**388**	**388**	**388**	**388**	**388**	**388**	**383**	**0**	**0**	**0**	**0**	**0**
	US$M	**3,686**	**0**	**0**	**0**	**128**	**134**	**142**	**131**	**116**	**170**	**175**	**174**	**171**	**176**	**182**	**210**	**165**	**173**	**190**	**197**	**148**	**123**	**123**	**123**	**123**	**123**	**123**	**123**	**43**	**0**	**0**	**0**	**0**	**0**
Freight & Marketing Charges																																			
Insurance	**$US/dmt conc**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**	**0.50**
	US$M	**9**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1**	**0**	**0**	**0**	**1**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Total Bulk Concentrate NSR	**$US/dmt conc**	**1.44**	**0**	**0**	**0**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**1**	**0**	**0**	**0**	**0**	**0**
	US$M	**13,956**	**0**	**0**	**0**	**451**	**469**	**526**	**471**	**418**	**655**	**669**	**665**	**651**	**665**	**692**	**783**	**643**	**666**	**727**	**750**	**558**	**477**	**477**	**477**	**477**	**477**	**477**	**476**	**157**	**0**	**0**	**0**	**0**	**0**
Total NSR	**$US/dmt conc**	**1.44**	**0**	**0**	**0**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**1**	**0**	**0**	**0**	**0**	**0**
	US$M	**13,956**	**0**	**0**	**0**	**451**	**469**	**526**	**471**	**418**	**655**	**669**	**665**	**651**	**665**	**692**	**783**	**643**	**666**	**727**	**750**	**558**	**477**	**477**	**477**	**477**	**477**	**477**	**476**	**157**	**0**	**0**	**0**	**0**	**0**
	C$M	**15,507**	**0**	**0**	**0**	**501**	**521**	**584**	**524**	**465**	**728**	**743**	**739**	**724**	**739**	**769**	**870**	**715**	**740**	**808**	**833**	**619**	**531**	**531**	**531**	**531**	**531**	**531**	**529**	**174**	**0**	**0**	**0**	**0**	**0**
Operating Costs																																			
Open Pit Mining Cost	C$/tonne mined	**2.13**	0.00	0.00	0.00	2.14	2.10	2.15	2.02	1.85	1.90	2.06	2.15	2.24	2.24	2.34	2.20	2.21	2.33	2.20	2.23	2.77	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	C$M	**1,466**	0	0	0	45	58	60	87	100	102	76	72	69	70	150	141	142	142	68	61	23	0	0	0	0	0	0	0	0	0	0	0	0	0
Underground Mining Cost	C$/tonne mined	**0.75**	0.00	0.00	0.00	1.20	0.97	4.43	2.89	1.68	0.87	1.23	1.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	C$M	**516**	0	0	0	25	27	123	125	90	47	45	34	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Rehandle Cost	C$/tonne rehandled	**0.75**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.75	0.00	0.00	0.00	0.00	0.00	0.75	0.75	0.00	0.00	0.00	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.00	0.00	0.00	0.00	0.00
	C$M	**125**	0	0	0	0	0	0	0	0	2	0	0	0	0	0	5	6	0	0	0	9	14	14	14	14	14	14	14	7	0	0	0	0	0
Processing Cost	C$/tonne milled	**13.64**	0.00	0.00	0.00	13.95	13.95	13.95	13.95	13.95	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	13.60	0.00	0.00	0.00	0.00	0.00
	C$M	**5,474**	0	0	0	127	127	129	126	126	248	248	248	248	248	248	248	248	248	248	248	248	248	248	248	248	248	248	248	124	0	0	0	0	0
G&A	C$/tonne milled	**0.99**	0.00	0.00	0.00	1.66	1.66	1.66	1.66	1.66	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.91	0.00	0.00	0.00	0.00	0.00
	C$M	**399**	0	0	0	15	15	15	15	15	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	8	0	0	0	0	0
Total Operating Costs	**C$/tonne milled**	**19.88**	**0.00**	**0.00**	**0.00**	**23.35**	**24.91**	**35.42**	**38.98**	**36.66**	**22.82**	**21.17**	**20.32**	**18.28**	**18.36**	**22.73**	**22.51**	**22.64**	**22.28**	**18.21**	**17.84**	**16.26**	**15.26**	**15.26**	**15.26**	**15.26**	**15.26**	**15.26**	**15.26**	**15.26**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
	C$M	**7,981**	**0**	**0**	**0**	**213**	**227**	**327**	**353**	**331**	**416**	**386**	**371**	**334**	**335**	**415**	**411**	**413**	**407**	**332**	**326**	**297**	**278**	**278**	**278**	**278**	**278**	**278**	**278**	**139**	**0**	**0**	**0**	**0**	**0**
Net Operating Income	**C$M**	**7,526**	**0**	**0**	**0**	**288**	**294**	**257**	**170**	**134**	**312**	**357**	**368**	**390**	**404**	**354**	**460**	**302**	**333**	**476**	**507**	**323**	**252**	**252**	**252**	**252**	**252**	**252**	**250**	**35**	**0**	**0**	**0**	**0**	**0**
	$C/dmt conc	**0.77**	**0.00**	**0.00**	**0.00**	**0.83**	**0.80**	**0.67**	**0.48**	**0.43**	**0.70**	**0.78**	**0.81**	**0.87**	**0.87**	**0.73**	**0.81**	**0.70**	**0.74**	**0.95**	**0.97**	**0.83**	**0.79**	**0.79**	**0.79**	**0.79**	**0.79**	**0.79**	**0.79**	**0.31**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
	$C/tonne milled	**18.75**	**0.00**	**0.00**	**0.00**	**31.51**	**32.19**	**27.91**	**18.77**	**14.81**	**17.10**	**19.57**	**20.18**	**21.37**	**22.12**	**19.42**	**25.18**	**16.53**	**18.26**	**26.08**	**27.80**	**17.69**	**13.81**	**13.81**	**13.81**	**13.81**	**13.81**	**13.81**	**13.71**	**3.85**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
Capital Costs																																			
Mining Equipment	C$M	**265**	0	12	47	24	8	23	9	0	0	14	24	8	8	23	17	31	5	6	0	4	0	0	0	1	0	0	0	0	0	0	0	0	0
OP Mine OPEX during Pre-Production	C$M	**16**	0	0	16	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
UG Mine Development	C$M	**37**	0	0	0	9	28	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Site Development	C$M	**37**	0	18	18	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing Plant	C$M	**294**	0	77	77	0	1	1	1	85	30	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	0	0	0	0	0	0	0
On-Site Infrastructure	C$M	**143**	0	45	45	6	0	0	0	19	29	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Tailings Management Facility	C$M	**211**	0	5	30	3	3	3	3	11	12	8	8	8	8	8	8	8	8	7	7	7	7	7	7	7	7	7	7	7	0	0	0	0	0
TMF Waste Haulage	C$M	**140**	0	0	10	3	3	3	0	5	5	5	5	9	8	21	15	15	19	8	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirects	C$M	**73**	0	19	27	1	0	0	0	11	16	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
EPCM	C$M	**46**	0	12	18	0	0	0	0	6	9	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner's Costs	C$M	**10**	0	4	6	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0	C$M	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Closure	C$M	**60**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	12	12	12	12	12	0	0
Subtotal Capital Costs	C$M	**1,332**	0	193	293	46	43	30	14	137	102	27	39	27	26	54	42	56	33	23	13	12	8	8	8	9	8	8	19	19	12	12	12	0	0
Contingency	C$M	**218**	0	45	55	3	1	1	1	33	24	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	5	5	3	3	3	0	0
Total Capital Costs Incl. Contingency	**C$M**	**1,551**	**0**	**238**	**349**	**49**	**44**	**31**	**15**	**170**	**126**	**29**	**41**	**29**	**28**	**56**	**44**	**58**	**36**	**25**	**15**	**14**	**10**	**10**	**10**	**10**	**11**	**10**	**24**	**24**	**15**	**15**	**15**	**0**	**0**
Pre-Production CAPEX	C$M	**586**	0	238	349	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining/Closure CAPEX	C$M	**964**	0	0	0	49	44	31	15	170	126	29	41	29	28	56	44	58	36	25	15	14	10	10	10	10	11	10	24	24	15	15	15	0	0
Working Capital	C$M	**0**	0	0	0	36	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-36	0	0	0	0	0
Net Pre-Tax Cash Flow	**C$M**	**5,975**	**0**	**-238**	**-349**	**204**	**250**	**226**	**155**	**-36**	**185**	**328**	**327**	**361**	**376**	**298**	**415**	**244**	**298**	**451**	**492**	**309**	**242**	**242**	**242**	**242**	**242**	**242**	**226**	**46**	**-15**	**-15**	**-15**	**0**	**0**
Cumulative Pre-Tax Cash Flow	**C$M**		**0**	**-238**	**-586**	**-383**	**-133**	**93**	**248**	**212**	**397**	**725**	**1,053**	**1,413**	**1,789**	**2,088**	**2,503**	**2,746**	**3,044**	**3,495**	**3,988**	**4,296**	**4,539**	**4,781**	**5,023**	**5,264**	**5,506**	**5,748**	**5,974**	**6,020**	**6,005**	**5,990**	**5,975**	**5,975**	**5,975**
Pre-Tax NPV	**C$M**	**2,074**																																	
Pre-Tax IRR	**%**	**32%**																																	
Pre-Tax Payback Period	**Years**	**2.6**																																	
Income Taxes	**C$M**	**2,265**	**0**	**0**	**0**	**22**	**20**	**59**	**48**	**27**	**85**	**104**	**116**	**126**	**133**	**113**	**156**	**57**	**117**	**173**	**185**	**116**	**89**	**90**	**91**	**91**	**91**	**91**	**85**	**-7**	**-5**	**-5**	**-5**	**0**	**0**
Net After-Tax Cash Flow	**C$M**	**3,710**	**0**	**-238**	**-349**	**182**	**230**	**167**	**107**	**-63**	**100**	**224**	**211**	**235**	**243**	**185**	**259**	**186**	**180**	**279**	**307**	**193**	**153**	**152**	**151**	**151**	**150**	**151**	**141**	**53**	**-10**	**-10**	**-11**	**0**	**0**
Cumulative After-Tax Cash Flow	**C$M**		**0**	**-238**	**-586**	**-404**	**-175**	**-8**	**100**	**36**	**137**	**360**	**572**	**807**	**1,050**	**1,235**	**1,494**	**1,680**	**1,861**	**2,140**	**2,447**	**2,639**	**2,792**	**2,944**	**3,095**	**3,246**	**3,396**	**3,547**	**3,688**	**3,741**	**3,731**	**3,720**	**3,710**	**3,710**	**3,710**
Pre-Tax NPV	**C$M**	**1,217**																																	
Pre-Tax IRR	**%**	**25%**																																	
Pre-Tax Payback Period	**Years**	**3.1**																																	

Figure 22.4: Sensitivity Graph on Base Case Economic Results



Source: JDS, 2015

Table 22.5: Discount Rate Sensitivity

Discount Rate	Pre-Tax NPV	After-Tax NPV
0%	5,975.3	3,710.0
5%	2,898.1	1,744.3
7.50%	2,073.6	1,216.9
10%	1,502.4	850.9
12%	1,167.6	636.0

Source: JDS, 2015

23 Adjacent Properties

Any adjacent mineral properties have no bearing on the project or this report.

24 Other Relevant Data and Information

There is no other relevant data or information for this report.

25 Interpretation and Conclusions

Industry standard mining and processing methods were used in this report. Sufficient information and data was available to the QPs for a PEA-level study and the goal of producing a PEA study, prepared in accordance with 43-101 guidelines, was achieved. The preliminary economic results, based on the assumptions highlighted in this report, show a positive outcome.

It is important to note that this result is only preliminary and could change significantly as more information is gathered and market conditions change. This assessment includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

25.1 Risks

As with almost all mining ventures, there are a large number of risks and opportunities that can influence the outcome of the Wellgreen project. Most of the risks are based on a lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal price, exchange rates, etc.). The following section identifies the most significant potential risks currently known for the Wellgreen project, almost all of which are common to mining projects at this early stage of project development.

Subsequent higher-level engineering studies would be needed to further refine these risks and opportunities, identify new ones, and define mitigation or opportunity implementation plans. While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the Wellgreen project.

Table 25.1 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches. The most significant potential risks associated with the Wellgreen project are the ability to convert inferred resources to indicated and measured, geotechnical stability of pit walls and tailings facility, lower metal recoveries than those projected, the ability to produce a marketable concentrate, waste storage capacity, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal prices. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

External risks are, to a certain extent, beyond the control of the project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates.

Table 25.1: Internal Project Risks

Risk	Explanation	Potential Impact	Possible Risk Mitigation
Recoveries	Flotation recoveries and corresponding concentrate grades need further investigation.	If life-of-mine recovery of Ni, Cu or PGM's is lower than projected, project economics could be negatively impacted.	Conduct a full suite of tests to confirm assumptions.
Permit Acquisition	The ability to secure a mining permit is of paramount importance as is the negotiation with current stakeholders.	Failure to secure a mining permit would stop the project.	The development of close relationship with the communities and government along with a thorough EIA and project design that gives appropriate consideration to the environment and local people is required.
Development Schedule	The development could be delayed for a number of reasons and could impact project economics depending on metal prices at the time.	If delays in schedule result development during a period of lower metals prices project economics could be reduced due to lower revenues than projected.	If an aggressive schedule is to be followed, PFS field work should begin ASAP.
Inability to upgrade inferred resources to measured or indicated	The PEA mine plan uses 50% inferred resources which cannot be used at a higher level of study	If some of the inferred resources cannot be upgraded to indicated then the mineable tonnage would be reduced of what is presented here and project economics could be negatively affected	A well planned definition drilling campaign, renewed geostatistical analysis and resource estimation needs to be undertaken to determine the amount of inferred resource that can be converted or new material added.
TMF Location and Stability	The geotechnical condition of the soils under the TMF embankment and rock storage facilities must be investigated to confirm the location suitability and design adequacy.	A change in facility design or having to move the waste storage facilities could significantly impact both OPEX and CAPEX.	Conduct field investigations at the next level of study.

Risk	Explanation	Potential Impact	Possible Risk Mitigation
Smelter Terms	Smelter terms used in the study are only preliminary and could significantly affect the project economics if the terms (payable %, deductions and/or penalties, TC/RC's) change.	A reduction in the net smelter return would have a direct effect on project economics. Low concentrate grade and/or the presence of deleterious elements in the concentrates could impact the desirability of the concentrate and the price smelters are willing to pay.	Conduct advanced metallurgical test work to confirm any deleterious elements and verify concentrate composition.
Geotechnical Characterization	Open pit slope angles were prepared based on limited availability of structural and geotechnical data. The final pit walls are planned to be high and a change in the slope angle could have a large impact on economics. The suitability of the selected UG mining methods and assumptions also needs to be confirmed with more study.	Presence of unfavorably oriented structures, weak rock masses or hydraulic gradients behind pit walls may result in shallower slope angles being required.	Conduct geotechnical site investigation program and produce 3D structural model at the next level of study.
Waste Storage Capacity	The ability to have access to sufficient areas for tailings, waste rock and mill feed stockpile material is critical to the success of the project.	Large waste and stockpile facilities are required for the LOM plan. Failure to utilize the locations selected could increase operating costs and change operational plans.	Thorough analysis of the waste and stockpile facilities from permitting, geotech, hydrogeology and logistics needs to be conducted.

Source: JDS 2015

25.2 Opportunities

Table 25.2 identifies what are currently deemed to be the most significant opportunities for the Wellgreen project and their potential benefit. The most significant potential opportunities associated with the Wellgreen project are the improved metallurgical recoveries by secondary processing and additional metallurgical and process testing, exotic PGM and silver credits, reduced waste mined with steeper pit walls, expansion of the pit and/or block caving as an alternative to extend production beyond the base case a pit expansion and possible connection to grid power. Additional details on potential opportunities are found in Sections 25.2.1 to 25.2.4.

Table 25.2: Project Opportunities

Opportunity	Explanation	Potential Benefit
Metallurgical Recoveries	Improvements could potentially be made to process recoveries and/or concentrate grade and marketability	The NPV of the project may be improved with optimization of metallurgical recoveries and concentrate grade. The sensitivity of the project with respect to changes to process recovery is similar to the project's sensitivity to changes in processed head grades which has been included in the sensitivity analysis of this PEA.
PGM and silver Credits	PGMs other than Pt and Pd such as Rh could potentially be recovered and add value to the concentrate. Silver may also be recoverable.	Including silver and more PGMs into the bulk concentrate may provide another stream of revenue and improve project economics.
Geotech Characterization	Slope parameters could be re-adjusted and reconfirmed.	Supportive geotech drilling could provide information to steepen the final pit slopes and reducing the strip ratio for the LOM.
Stage 5 Pit Expansion	The existing pit could potentially be expanded and production increased from the remaining 66% of the resource model.	A larger pit and increased production could potential increase the mine life and increase the NPV of the Wellgreen project.
Block Caving	Block caving could extract a significant portion of the remaining resource and is an alternative to mining a Stage 5 open pit expansion.	Longer mine life and reduced surface disturbance foot print by not expanding the pit and waste dumps.
Ni/Cu/PGM	Metal price has the biggest single impact on the project economics.	The impact is shown in the economic sensitivity section.
Yukon Grid Power Expansion	The power line currently terminates at Haines Junction, approximately 140 km from the project site. Grid power expansion is possible.	Grid power would provide significant savings versus liquefied natural gas power generation.

Source: JDS 2015

25.2.1 Secondary Processing

Secondary processing is not considered in the PEA economic modelling because capital and operating expenditures have not been evaluated to a PEA-level of accuracy, and sufficient detail has not been established regarding the composition of the secondary feed. This section is only provided to describe a potential opportunity.

Average recoveries to the bulk concentrate for the life of mine are expected to be 61% for platinum, 73% for palladium and 60% for gold, or 66% for precious metals combined. While continued metallurgical testing work will try to improve metal recoveries using conventional flotation, the secondary processing of certain tails (containing the remaining 20-30% of the PGMs and gold) represents an opportunity to increase total PGM recoveries and potentially add to the economics of the Wellgreen project.

The original interest in the potential for secondary processing was targeted at enhancing recovery of PGMs from the Peridotite/Dunite and Clinopyroxenite/Pyroxenite domains since conventional flotation processing showed decreased recovery in the PGMs compared to the Gabbro/Massive Sulphide domains.

As noted in the mineral processing discussion, after the initial flotation process, the initial rougher flotation tailings are passed over a magnetic separation unit which produces a smaller volume of magnetic concentrate material. The magnetic separator concentrate is reground and reports to a flotation circuit which produces the magnetic separator concentrate, which is added to the cleaner flotation circuit, and magnetic separator flotation tail.

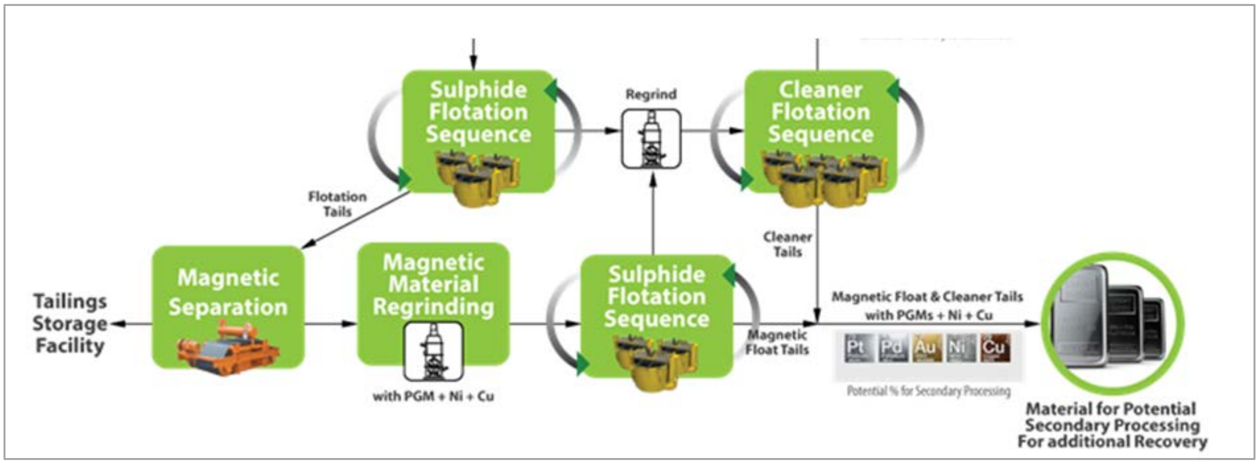
Therefore, there are three tailings streams that were initially considered for secondary processing:

- Rougher Tailings (produced from the initial magnetic separation process);
- Cleaner Flotation Tailings; and
- Magnetic Separation Flotation Tailings.

Rougher tailings produced from the non-magnetic material passing out of the magnetic separation process is a significant percentage of the mass, and limited testing was undertaken to determine whether an upgraded product could be developed using $CuSO_4$ flotation or gravity separation. Neither showed significant concentration from these tests. Therefore, it does not appear there is an opportunity in this stream for secondary processing, and this material would be transported to the tailings storage facility. Additional research beyond this initial limited testing will be conducted on the rougher tailings as part of future studies to determine if secondary processing opportunities exist.

Figure 25.1 shows two streams from the three metallurgical domains that are considered to have excellent potential for secondary treatment include the first cleaner flotation tails and the magnetic concentrate flotation tails.

Figure 25.1: Potential Secondary Mineral Processing Feed



Source: Eggert, Wellgreen Platinum, 2015

Preliminary testing of secondary processing methods was conducted by SGS Lakefield in 2014 under the supervision of the metallurgical QP John Eggert, with assistance from Dr. David Dreisinger. The following four secondary processes were tested:

Roast-Leach

The samples were roasted at elevated temperature in a muffle furnace and then subjected to cyanidation. The NaCN addition will initially be based on the base metal content of the feed and then controlled by periodic titration over a 24 hour leach retention time. The cyanidation leach solution and residue will be fully analyzed (Au, Ag, Pt, Pd, Cu, Ni, Co, Zn, Fe as well as Stot and S= on the leach residue) to gauge PGM extraction and roasting efficiency. Reagent additions were measured and reported.

Platsol Testing

Batch Platsol pressure leach test work was prioritized to maximize precious metal extraction in determining the amenability of the concentrate sample to the Platsol process. The following baseline parameters were applied:

- Temperature – 230° C;
- Oxygen pressure – 6.8 atm (100 psig);
- Residence time – 120 minutes;
- Concentrate regrind (as received);
- Pulp density – as required for auto-thermal operation or as determined from head analysis; and
- Chloride addition – 20 g/L.

Sampling and assaying of the autoclave test products included:

- Solids – Au, Ag, Pt, Pd, Cu, Ni, Fe, Stot, S=, WRA; and
- Solutions – Au, Ag, Pt, Pd, Cu, Ni, Fe, Fe^{2+}, SO_4, Free acid.

Chlorination

The samples were subjected to a wet chlorination leach. Typical conditions included:

- 20% solids;
- 85º C;
- Chlorine sparged;
- pH controlled with HCl; and
- 6-hour leach.

The leach solution and residue were analyzed for Au, Ag, Pt, Pd, Cu, Ni, Co, Zn, and Fe.

Intensive Cyanidation

The samples were subjected to intensive cyanidation tests, typical conditions included:

- Fine grind (10-15 μm);
- 20 g/L NaCN;
- Oxygen sparged; and
- 24-hour leach.

The leach solution and residue were analyzed for Au, Ag, Pt, Pd, Cu, Ni, Co, Zn, and Fe.

The Platsol hydrometallurgy process provided the highest precious metal recoveries, ranging from 84% to 98%, excluding silver. Table 25.3 summarizes the results from six Platsol tests. At this time, the results for chlorination, cyanidation and roast-leach were not sufficiently successful to warrant further testing.

Table 25.3: Summary of Platsol Tests

Well -10 Test 004 Cln Test #1 Cleaner Tails									
Product	**Amount**	**Distribution (%)**							
	(mL,g)	**Au**	**Pt**	**Pd**	**Ag**	**Cu**	**Ni**		
PLS	1194	83.8	**98.2**	**98.2**	99.1	99.4	99.4		
Residue	66	16.2	1.8	1.8	0.9	0.6	0.6		
		100.0	100.0	100.0	100.0	100.0	100.0		
Well -10 Test 004 Cleaner Test #1 Magnetic Tails									
PLS	1181	92.2	**82.6**	**94.0**	100.0	95.4	96.9	2.5	99.4
Residue	100	7.8	17.4	6.0	0.0	4.6	3.1	97.5	0.6
		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
LCT-3 Combined 1st Cleaner Tails									
PLS	1169	92.5	**96.1**	**96.1**	90.7	99.2	97.1	5.0	80.8
Residue	94	7.5	3.9	3.9	9.3	0.8	2.9	95.0	19.2
		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
LCT-3 Combined Magnetic Tails									
PLS	1151	92.0	**92.0**	**92.0**	86.9	91.1	96.9	4.2	95.5
Residue	100	8.0	8.0	8.0	13.1	8.9	3.1	95.8	4.5
		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
VW102 Concentrate 1									
PLS	996	99.3	**96.2**	**96.2**	0.0	98.0	97.6	88.3	80.3
Residue	20	0.7	3.8	3.8	100.0	2.0	2.4	11.7	19.7
		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
VW 101 U/F Concentrate 1									
PLS	1205	90.1	**90.1**	**90.1**		97.0	97.9	5.8	90.8
Residue	66	9.9	9.9	9.9		3.0	2.1	94.2	9.2
		100.0	100.0	100.0		100.0	100.0	100.0	100.0

Source: SGS, 2014

The various tailings streams created by each of the three metallurgical domains were then reviewed to determine the potential production associated with the secondary opportunity which is noted in Table 25.3.

Table 25.4: Summary of Platsol Test Results

Gabbro/Massive Sulphide	Mass Pull %	Cu %	Ni %	Pt %	Pd %	Au %	S %	MgO %
Cleaner Tail	14.3	1.9	6.3	9.3	6.8	6.9	18.6	TBD
MagneticTail	8.0	0.3	1.4	3.2	2.6	1.9	6.0	TBD
Subtotal	22.3	2.2	7.6	12.4	9.4	8.8	24.6	TBD
Clinopyroxenite/Pyroxenite								
Cleaner Tail	15.0	4.3	10.0	16.4	11.1	9.6	25.9	14.1
Magnetic Tail	8.5	0.2	1.8	3.3	2.5	2.5	9.0	TBD
Subtotal	23.5	4.5	11.8	19.7	13.5	12.0	34.8	14.1
Peridotite/Dunite								
Cleaner Tail	7.0	8.9	9.1	10.7	13.5	7.5	13.0	5.6
Magnetic Tail	12.5	6.90	3.1	12.7	12.3	6.6	15.0	7.6
Subtotal	19.5	15.8	12.1	23.3	25.8	14.1	28.1	13.1

Source: SGS, 2014

Based on the encouraging recovery results from the initial Platsol hydrometallurgical testing, additional material from each of the three geo-metallurgical domains will be tested using hydrometallurgical processes. Work will also be completed on projected capital and operating costs to determine whether the use of hydrometallurgical processing as a secondary processing approach will improve the economics of the Wellgreen project. This work will compare the benefits of increasing the total PGM and base metal recoveries to the production of a concentrate.

No economic value was attributed to this secondary processing opportunity in this PEA.

A possible production schedule associated with the potential secondary processing opportunity during the first 10 years of operations is shown in Table 25.5.

Table 25.5: Summary of Secondary Production

	Average Years 1-5	Average Years 6-10
t/a	2,109,885	4,233,905
t/d	5,780	11,600
Ni (%)	0.150	0.113
Cu (%)	0.050	0.025
Pt (g/t)	0.320	0.176
Pd (g/t)	0.184	0.164
Au (g/t)	0.041	0.018
Ni (lbs)	6,270,496	11,889,857
Cu (lbs)	2,084,717	2,085,728
Pt (oz)	19,544	21,580
Pd (oz)	11,216	20,096
Au (oz)	2,523	2,249

Source: Wellgreen Platinum, 2015

25.2.2 Recoveries, Exotic PGM's and Other Metallurgical Opportunities

Other metallurgical opportunities are:

- Additional optimization testing could potentially improve metal recoveries to bulk concentrates. Additional work in this area is recommended;
- Although generally present in significant quantities in the concentrate, minimal data is available for silver. However, when silver has been measured, it has typically been by methods other than fire assay, except in the analysis of the concentrates performed by XPS. Note: Methods commonly used for base metals, such as ICP and AA, tend to under-report precious metals, due to the nugget effect. Therefore, silver recoveries might improve the project economics at minimal expense; and
- Historical results indicate that total PGM grades could increase by approximately 10-25% if exotic PGMs, such as rhodium, iridium and osmium, were included. These exotic PGMs were recovered in concentrates by HudBay in the 1970s and they have consistently shown up in the metallurgical test work. Additional work is recommended to look at bringing the exotic PGMs into future economic assessments.

25.2.3 Stage 5 Pit Opportunity

The Stage 5 pit takes into account PEA base case underground extraction that has removed various high grade mineralized zones. It also considers the replacement of these zones with hydraulic fill at a specific gravity of 2.0 which is considered to be "waste" in the optimized pit shell. The pit shell (Strip ratio = 1.90) is summarized as follows:

Table 25.6: Stage 5 Pit

Waste (Mt)	Production (Mt)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)
1,031.1	553.4	0.24	0.12	0.02	0.21	0.22	0.04

Source: SNC 2015

A bench by bench analysis & production scenarios were then completed with the following criteria:

- A low grade stockpile was established that consisted of grades from 0.6 g/t Pt Eq to 1.0 g/t Pt Eq and a high grade stockpile that is comprised of Stage 5 material grading from 1.0 g/t Pt Eq to 1.4 g/t Pt Eq;
- Potential production would commence after the PEA base case Stage 4 pit and underground extraction was completed;
- Processing of PEA base case stockpiles would be deferred until the Stage 5 open pit was completed; and
- Stage 5 production would commence in Year 17 with mineral processing rates being considered at 50,000 t/d, 75,000 t/d and 100,000 t/d.

The following tables review the potential production rate associated with this PEA opportunity.

Table 25.7: PEA Base 25ktpd to 50ktpd with Stage 5 Pit Opportunity at 50 kt/d

	PEA Base Case Mining Stages 1-4	PEA Opportunity Mining Stage 5	Stockpiles
Years	1 to 16	17 to 36	37 to 54
Mill (kt/d)	42.2	50.0	50.0
Average Annual Production			
Nickel (Mlbs)	73.1	80.1	57.4
Copper (Mlbs)	55.3	54.8	19.3
Cobalt (Mlbs)	3.4	3.9	3.1
Platinum (koz)	89.5	92.4	40.2
Palladium (koz)	103.5	111.1	60.6
Gold (koz)	15.9	18.1	7.2
3E (koz)	208.9	221.6	107.9

Source: SNC, 2015

Table 25.8: PEA Base 25ktpd to 50ktpd with Stage 5 Pit Opportunity at 75 kt/d

	PEA Base Case Mining Stages 1-4	PEA Opportunity Mining Stage 5	Stockpiles
Years	1 to 16	17 to 29	30 to 42
Mill (kt/d)	42.2	75.0	75.0
Average Annual Production			
Nickel (Mlbs)	73.1	120.5	82.1
Copper (Mlbs)	55.3	82.6	28.4
Cobalt (Mlbs)	3.4	5.9	4.4
Platinum (koz)	89.5	139.3	58.5
Palladium (koz)	103.5	167.7	87.2
Gold (koz)	15.9	27.2	10.5
3E (koz)	208.9	334.2	156.1

Source: SNC, 2015

Table 25.9: PEA Base 25ktpd to 50ktpd with Stage 5 Pit Opportunity at 100 kt/d

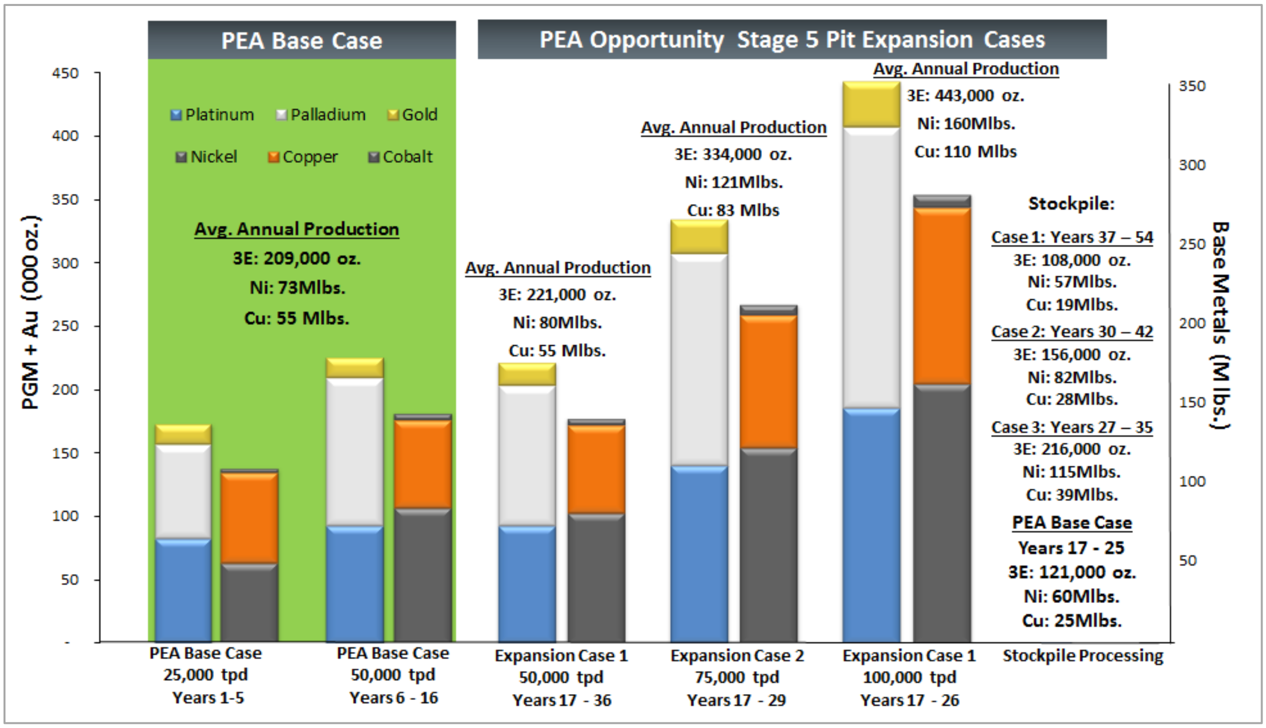
	PEA Base Case Mining Stages 1-4	PEA Opportunity Mining Stage 5	Stockpiles
Years	1 to 16	17 to 26	27 to 35
Mill (ktpd)	42.2	100.0	100.0
Nickel (Mlbs)	73.1	160.1	114.7
Copper (Mlbs)	55.3	109.6	38.6
Cobalt (Mlbs)	3.4	7.9	6.2
Platinum (koz)	89.5	184.8	80.4
Palladium (koz)	103.5	222.3	121.1
Gold (koz)	15.9	36.1	14.3
3E (koz)	208.9	443.2	215.9

Source: SNC, 2015

As the production rate for the Stage 5 opportunity increases, there is a corresponding increase in nickel, platinum and palladium and gold production that would enable the project to become a significant global producer of these metals. However, it must be noted that operating and capital expenditures associated with this PEA opportunity have not been estimated and therefore it is not possible to generate an economic assessment. In addition, the production rate relies on utilization of inferred resources that are not considered to demonstrate economic viability.

Figure 25.2 Illustrates the various PEA Stage 5 Opportunity production scenarios.

Figure 25.2: 2015 PEA Base Case Production & Expansion Opportunities



Source: SNC, 2015

25.2.4 Block Caving

Block Caving is not included as part of the PEA production plan. It is considered to be an opportunity that extracts a significant portion of the remaining resource and is an alternative to mining a large Stage 5 open pit

The minimum width of mineralization to consider block caving is 80 metres and the minimum length is 200 metres for the purposes of the PEA opportunity. Table 16.15 summarizes the two zones that are considered to be an opportunity for block caving. The block caving system has certain advantages as compared to mining a Stage 5 open pit, the most significant of which is a large decrease in waste rock storage requirements. Block caving opportunities are shown in Table 25.10.

Table 25.10: Block Caving Opportunities

Area	Volume (Mm^3)	Tonnes (Mt)	Footprint (Mm^3)	Grade Factor	Extract Factor	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Au g/t
BC2	43.6	125.0	0.1	85%	100%	0.249	0.205	0.015	0.331	0.274	0.074
BC56	3.5	10.1	0.0	85%	100%	0.251	0.362	0.017	0.491	0.308	0.147

Source: SNC, 2015

26 Recommendations

JDS recommends that the project progress to a Pre-feasibility Study (PFS) level, with the necessary work conducted in two phases, and with Phase 2 contingent on the success of Phase 1.

The key areas for follow up work of Phase 1 of the pre-feasibility program in 2015 that JDS recommends Wellgreen Platinum pursue are listed below:

- Conduct initial drilling within the pit models designed to further upgrade Inferred Mineral Resources to Measured & Indicated Mineral Resources and test extensions of mineralization within the pit where it is unclassified, with the cost of such activities estimated to be $**[3.5million]**;
- Implement additional metallurgical test programs in order to optimize recoveries from the main geo-metallurgical domains and conduct more detailed testing and assessment of potential secondary processing options, with the cost of such activities estimated to be $**[200,000]**;
- Commence evaluation of the cost and benefits of bringing the exotic PGMs such as rhodium, osmium, iridium and ruthenium into the mineral resource estimate, with the cost associated with such an evaluation estimated to be $**[200,000]**;
- Conduct additional geotechnical work to improve understanding of pit slopes and mine infrastructure, with the cost of such work estimated to be $**[200,000]**; and
- Conduct open pit trade-off studies, with the cost of such work estimated to be $**[100,000]**.

In aggregate, the total cost of Phase 1 of the PFS activities is estimated to be $4.1 million. If Phase 1 is successful, Wellgreen Platinum should consider pursuing Phase 2 of the PFS activities, which will be comprised of various activities such as drilling, sampling, assaying, geotechnical studies, metallurgical testwork and engineering studies in order to further de-risk the Wellgreen project. It is estimated that the costs associated with completing Phase 2 may be in the range of $5 million to $10 million. However, a more definite estimate can by necessity only be made after Phase 1 is completed and a decision is taken by Wellgreen Platinum to pursue Phase 2.

Further details details on recommendations are found in Section 26.1.

26.1 Metallurgical & Processing

The following Metallurgical and Processing work is recommended for the PFS and FS levels:

- Analysis of historic assay results, including testing for exotic PGMs, suggests that these elements can increase the Pt+Pd grade by more than 50% in material classified as gabbro, and approximately 15-17% for material classified as clinopyroxenite/pyroxenite and peridotite. Previous metallurgical testing indicates that exotic PGMs are recovered to concentrate in flotation, with recovery levels similar to platinum. Therefore, it is believed that the exotic PGM recoveries will be similar to those of platinum;

- No economic value was attributed to the exotic PGMs in the Wellgreen PEA, and additional testing will be conducted in the next round of studies to better quantify the grade and economic contribution of the PGMs;
- Average recoveries to the bulk concentrate for the life of mine are expected to be 61% for platinum, 73% for palladium and 60% for gold, or 66% for precious metals combined. While continued metallurgical testing work will try to improve metal recoveries using conventional flotation, the secondary processing of certain tails (containing the remaining 20-30% of the PGMs and gold) represents an opportunity to increase total PGM recoveries and potentially add to the economics of the Wellgreen project;
- There are a number of avenues for testing that have not yet been fully evaluated. The ability to recover PGMs to a magnetic concentrate must be fully evaluated to determine if there is potential for increasing recoveries;
- All testing to date indicates that talc and similar minerals are liberated early in flotation. Further testing to reduce the recovery of these minerals to concentrates is needed;
- Tests to size a SAG mill and High Pressure Grinding Rolls (HPGR) have not been undertaken. This should be done at the Pre-Feasibility or Feasibility study stage;
- Testing to determine the work index for concentrates to be reground is needed;
- An evaluation of the relative performance of the flotation circuit if the feed is SAG product, crushing followed by single stage ball mill product; wet screening ahead of crushing followed by single stage ball mill product and high pressure grinding roll product should be undertaken. The sensitivity of the mineralization to over-grinding indicates that a better understanding of the impacts of crushing and grinding be evaluated;
- Additional testing to establish grade recovery curves for the clinopyroxenite zones may be necessary;
- Confirmatory testing to establish product properties, such as specific gravity of concentrates, etc., is necessary at more detailed stages such as a PFS. These tests can also be used to further refine the flowsheet.
- Testing to establish dewatering criteria are needed; and
- A simulation of the crushing plant is necessary to confirm the layout, screen sizes, openings, etc.

26.2 Open Pit Geotechnical

As the Wellgreen Project advances to the pre-feasibility study (PFS) and feasibility study (FS) levels of design, geotechnical specific drilling, testing and engineering will be required to support the pit slope and waste dump facility designs. The following geotechnical work will be necessary at the PFS and FS levels:

- A full geotechnical characterization program including geotechnical specific core drilling and discontinuity orientation, laboratory strength testing of core samples and engineering to support a PFS or FS pit slope design;
- Geotechnical characterization of shallow foundation materials beneath the proposed waste rock dumps and stockpile by drilling and/or backhoe test pits;

- Drilling and installation of monitoring instrumentation to determine the extents and temperatures of permafrost near waste dumps and infrastructure;
- Sufficient hydrogeological characterization to determine potential pit inflows and provide reasonable estimates of phreatic surface behavior in pit walls during and after mining; and
- Structural mapping and development of a site geologic structural model incorporating the major fault and shear structures at the site.

26.3 Open Pit Mining

- Increase the amount of measured and indicated material in the Resource through additional diamond drilling;
- Complete a trade-off study evaluating the ultimate pit shell with updated parameters;
- Design detailed pit phases with ramp access that target high grade material;
- Consider in-pit dumping by phasing the East and West side pits to reduce haulage distances and the environmental footprint;
- Generation of power via utilization of trolley assisted mine haulage trucks that are transported to the crusher loaded and to the pit empty;
- Geotechnical drilling to optimize pit slope and waste dump angles;
- Trade-off study to evaluate the benefits of an in-pit crusher;
- Haulage optimization study between the 1540 waste dump, 1720 waste dump and tailings facility; and
- Further develop the application and cost of LNG retrofitted trucks.

27 List of Abbreviations

Units of measurement used in this report conform to the SI (metric) system. A complete list of abbreviations is shown in Table 27.1.

Table 27.1: Units of Measure & Abbreviations

°C	**degree Celsius**
°F	degree Fahrenheit
A	ampere
a	annum
Ag	silver
Au	gold
bbl	barrels
C$ or CAD	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
Co	Cobalt
Cu	copper
d	day
dia.	diameter
dmt	dry metric tonne
dwt	dead-weight ton
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
G	giga (billion)
GAAP	Generally Acceptec Accountaing Practices
g	gram
g/L	gram per litre
g/t	gram per tonne
Gal	Imperial gallon
gpm	Imperial gallons per minute
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
ha	hectare
hp	horsepower
hr	hour
HRIA	Heritage Resource Impact Assessment
in	inch
in^2	square inch
J	joule
k	kilo (thousand)

kcal	kilocalorie
kg	kilogram
km	kilometre
km/h	kilometre per hour
km^2	square kilometre
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
L/s	litres per second
LSA	Local Study Area
M	mega (million)
m	metre
µ	micron
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
MASL	metres above sea level
MBCA	Migratory Birds Convention Act
µγ	microgram
min	minute
mm	millimetre
MMER	Metal Mining Effluent Regulations
MOE	Saskatchewan Ministry of the Environment
mph	miles per hour
Mt	Million tonnes
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
NaCN	Sodium Cynanide
Ni	nickel
OPEX/CAPEX	Operating Cost / Capital Cost
opt, oz/st	ounce per short ton
oz	Troy ounce (31.1035g)
Pb	lead
Pd	paladium
PEA	Preliminary Economic Assessment
PGM	Platinum Group metals
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
Pt	platinum
s	second
st	short ton
stpa	short ton per year
stpd	short ton per day

t	metric tonne
TOR	Terms of Reference
t/a	metric tonne per year
t/d	metric tonne per day
TSF	Tailings Storage Facility
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
VMS	Volcanogenic Massive Sulphide
W	watt
wmt	wet metric tonne
yd^3	cubic yard
yr	year

Source: JDS, 2015

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APPENDIX A

QP CERTIFICATES

PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

JDS Energy & Mining Inc.

CERTIFICATE OF AUTHOR

I, Michael E. Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report, Wellgreen Project, Yukon Territory, Canada", with an effective date of February 2, 2015, (the "Technical Report") prepared for Wellgreen Platinum Ltd.;

2. I am currently employed as a Senior Project Manager with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver British Columbia, V6C 2T6;

3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

4. I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over seven years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I visited the Wellgreen project September 17-18, 2013;

8. I am responsible for Sections 1 (except 1.4 to 1.9), 2, 3, 15, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27 and 28 of this Technical Report;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

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

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: February 2, 2015
Signing Date: March 18, 2015

(signed and sealed) "Michael E. Makarenko, P.Eng.

Michael E. Makarenko, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Ronald G. Simpson, P.Geo.
GeoSim Services Inc.
1975 Stephens St.
Vancouver, BC, Canada V6K 4M7
Tel: (604) 803-7470
E-mail: rsimpson@geosimservices.com

I, Ronald G. Simpson, P.Geo., do hereby certify:

- I am employed as a Professional Geoscientist with GeoSim Services Inc.
- This certificate applies to the technical report titled "Preliminary Economic Assessment Technical Report, Wellgreen Platinum Project, Yukon Territory, Canada" with an effective date of February 2, 2015 (the "Technical Report").
- I am a Professional Geoscientist (19513) in good standing with the Association of Professional Engineers and Geoscientists of British Columbia. I graduated with a Bachelor of Science in Geology from the University of British Columbia, May 1975.
- I have practiced my profession continuously for 40 years. I have been directly involved in mineral exploration, mine geology and resource estimation with practical experience from feasibility studies.
- As a result of my experience and qualifications, I am a qualified person as defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101").
- I visited the property on September 17, 2013.
- I am responsible for Sections 1.4, 1.5, 1.7, 4 through 12, and 14 of the Technical Report.
- I am independent of Wellgreen Platinum Ltd. as described in Section 1.5 of NI 43-101.
- My prior involvement with the Property that is the subject of this Technical Report was as author of a previous Technical Report dated September 8, 2014.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: February 2, 2015
Signing Date: March 18, 2015

(signed and sealed) "Ronald G. Simpson, P. Geo."

Ronald G. Simpson, P.Geo.

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CERTIFICATE OF QUALIFIED PERSON

I, Michael Levy, MSc, PE, PG do hereby certify that:

1. I am Principal Geotechnical Engineer of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled “Preliminary Economic Assessment Technical Report, Wellgreen Project, Yukon Territory, Canada” with an Effective Date of February 2, 2015 (the “Technical Report”).
3. I graduated from the University of Iowa in 1998 with a B.Sc. in Geology and from the University of Colorado in 2004 with a M.Sc. in Civil-Geotechnical Engineering. I am a registered Professional Engineer in the states of Colorado (#40268) and California (#70578) and a registered Professional Geologist in the state of Wyoming (#3550). I have 16 years of experience in civil and mining geotechnical projects ranging from conceptual through feasibility design levels, construction and operations support. I specialize in the design and management of large mine slopes.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I have visited the Wellgreen Project on September 11-12, 2013.
6. I am responsible for the preparation of Sections 16.6 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: February 2, 2015

Signing Date: March 18, 2015

(signed and sealed) “Michael Levy, MSc, PE, PG”

____

Michael Levy, MSc, PE, PG

Eggert Engineering Inc
158 David Street
Sudbury, Ontario
P3E 1T4

CERTIFICATE OF AUTHOR

I, John Eggert, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report, Wellgreen Project, Yukon Territory, Canada”, with an effective date of February 2, 2015 (the “Technical Report”), prepared for Wellgreen Platinum Ltd.
2. I am currently employed as the President of Eggert Engineering Inc. whose office is at 158 David Street, Sudbury, Ontario, P3E 1T4.
3. I am a graduate of Queen’s University at Kingston with a B.Sc. in Mining Engineering, 1990. I have practiced my profession continuously since 1990.
4. I have worked in operations, technical and managerial positions in Canada. I have been an independent engineer for six years. I have performed mill designs, metallurgical accounting, cost estimations, operations management, due diligence reviews and report writing for mining projects in Canada, the USA and Mexico.
5. I am a Registered Professional Engineer in Ontario, licence number 90397597.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have not visited the Wellgreen Project.
8. I am responsible for Sections 1.6, 1.9, 13 and 17 of the Technical Report.
9. I am independent of Wellgreen Platinum Ltd. as described in Section 1.5 of NI 43-101.
10. My prior involvement with the Wellgreen Project that is the subject of the Technical Report relates to metallurgical testing that was performed on the project in 2013 and 2014.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
12. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: February 2, 2015

Signing Date: March 18, 2015

(signed & sealed) “John Eggert, P.Eng.

John Eggert, P.Eng.

Sustainable Mine Development
Global Mining & Metallurgy
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40 Larch Street, Suite 300
Sudbury (Ontario)
Canada P3E 5M7
Tel: 705-222-0164

CERTIFICATE OF AUTHOR

I, George B. Darling, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report, Wellgreen Project, Yukon Territory, Canada”, with an effective date of February 2, 2015, (“the Technical Report”) prepared for Wellgreen Platinum Ltd.;

2. I am currently employed as a Director of Technical Mine Services with SNC-Lavalin inc. with an office at Suite 300 – 40 Larch Street, Sudbury, Ontario, P3E 5M7;

3. I am a graduate of Queen’s University with a B. Sc. in Mining Engineering, 1976. I have practiced my profession continuously since 1976;

4. I have worked in technical, operations and management positions at mines in Canada, New Caledonia and Australia. I have been an independent consultant for over nine years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Mining Engineer in Ontario (#10497014);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I have not visited the Wellgreen project site;

8. I am responsible for Sections 1.8, 16 except 16.6;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

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

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: February 2, 2015

Signing Date: March 18, 2015

(signed and sealed) “*George B. Darling, P. Eng.*”

George B. Darling, P. Eng.