Amended Feasibility Study Update

Marathon Palladium & Copper Project

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Revision #0

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May 31, 2024

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1. SUMMARY

1.1 Introduction

The Technical Report for the Marathon Palladium-Copper project (the “Marathon Project” or “Project”) located just outside the Town of Marathon on the shores of Lake Superior in Ontario, Canada was prepared by Generation Mining Limited (the “Company” or “Gen Mining”), G Mining Services Inc. (“GMS”) along with contributions from Wood Canada Limited (“Wood”), Knight Piésold Ltd., P&E Mining Consultants Inc. and JDS Energy and Mining, Inc.

Gen Mining currently owns a 100% interest in the Marathon Project. The Project is managed and operated by Gen Mining’s 100%-owned subsidiary Generation PGM Inc. (“Gen PGM”). In this document, Gen PGM and Gen Mining will be used interchangeably for simplicity.

This Technical Report summarizes the current progress and latest results following the 2021 Feasibility Study with updated designs and construction costs, additional geotechnical site investigations and metallurgical testing results for the Marathon Project. This Technical Report also presents updated Mineral Resource and Mineral Reserve estimates for the Marathon Property. The Technical Report outlines the Feasibility Study update with the development of an open pit mine, processing facilities and related infrastructure both on site and off site.

This Technical Report was prepared pursuant to the requirements of Canadian National Instrument 43-101 (“NI 43-101”). The reported Mineral Resource and Mineral Reserves estimates in this Technical Report were prepared in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) Standards (2014) on Mineral Resources and Reserves, Definitions and Guidelines (2019).

All dollar amounts are in Canadian dollars and stated on a 100% project ownership basis unless otherwise noted.

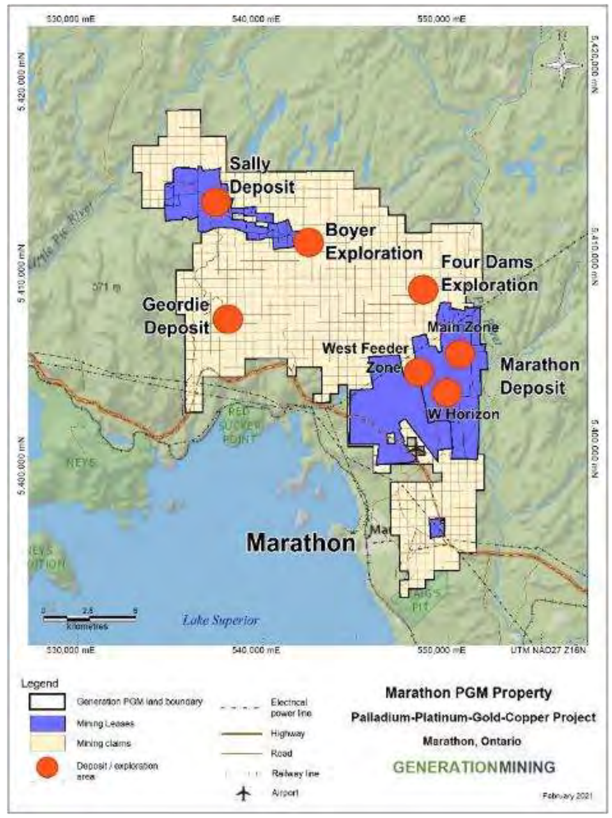
1.2 Property Location

The Marathon Project is located approximately 10 km north of the Town of Marathon, Ontario, adjacent to the Trans-Canada Highway No. 17 on the northeast shore of Lake Superior (Figure 1.1) Thunder Bay, a major industrial city in the area with a population of 113,524 people (2021 Census, Statistics Canada), is located approximately 300 km westward along Highway 17. Marathon has a population of 3,138 (2021 Census, Statistics Canada). Property access is by a gravel road from Highway 17 (Figure 1.2), which lies just north of Marathon and immediately south of the Property.

Figure 1.1: Regional Location

Source: *Marathon PGM Corp. (2006).*

Figure 1.2: Local Property Map



Source: *Generation Mining (2022).*

1.3 Land Tenure

The Property consists of a total of 21,883 ha, including 46 leases and 933 claim cells.

The Property is subject to net smelter return royalties ranging from 1 to 4%. Within the Mineral Reserve footprint, only the top northern extent of the Marathon deposit (specifically on the North pit) is subject to a net smelter return royalty of 4%.

On January 26, 2022, Gen Mining completed the acquisition of the remaining 16.5% interest in the Project from Stillwater Canada Inc., a subsidiary of Sibanye Stillwater Limited. The Company now holds 100% of the Marathon Project, and the joint venture agreement dated July 10, 2019, between Stillwater and Gen PGM has been terminated.

1.4 Property Description

Access to the Property is directly off the Trans-Canada Highway No. 17 and is accessible via gravel road. The Property is characterized by moderate to steep hilly terrain with a series of interconnected creeks and lakes surrounded by dense vegetation. Outcrops are common on the Property and overburden is generally minimal (0.5 m) with the deepest areas ranging from 3 to 10 m in thickness. The general elevation around the mine site is slightly higher than the overall regional topography. Ground surface elevations around the proposed site range from approximately 260 to over 400 masl with a gradual decrease in elevation from north to south and west to east.

The vegetation consists of northern hardwood and conifer trees as well as areas with muskeg, which are bogs or wetlands common to boreal forest regions. The Project area is bounded to the east by the Pic River and Lake Superior to the south and west.

The climate is typical of the northern Canadian Shield with long winters and short, warm summers. Average annual precipitation in Marathon was 759 mm for the period 2015-2019 (Pukaskwa station, which is approximately 15 km south of the Property). On average, annual snowfall is between November and April with a peak average snow depth of 45 cm in March. The annual average temperature is 1.4°C with the highest average monthly temperature of 15°C in August and lowest in January of -14°C (Marathon Airport 2015-2019).

Electrical power and telephone communication are present at the Property and in the Town of Marathon, which is linked to the Ontario power grid. The construction of the East-West Tie transmission project was completed in 2022. This is a 450 km double-circuit 230 kV transmission line connecting the Lakehead Transfer Station in the Municipality of Shuniah near the city of Thunder Bay to the Wawa Transfer Station located east of the Municipality of Wawa. It will also connect to the Marathon Transformer Station.

The Marathon airport is located immediately north of the Town of Marathon and runs adjacent to Highway 17 near the southwest corner of the Property.

1.5 History

The Marathon Property was explored by various companies over the past 60+ years. During this time, a total of 193,057 m of drilling was completed, with most of the drilling delineating the Marathon deposit. Most of the drilling (567 holes and 103,834 m) was completed by Marathon PGM Corp. between 2004 and 2009 to expand the Mineral Resource and for condemnation holes outside of the proposed open pit area.

The Marathon Property went through various ownership changes during the history of the Project. The most recent history including Gen Mining started on July 11, 2019, when Gen Mining (through its wholly-owned subsidiary) completed the acquisition of a 51% initial interest in the Property, from Stillwater Canada Inc., a wholly owned subsidiary of Sibanye Stillwater Limited, and entered into a joint venture agreement with respect to the Property. Following the acquisition of the Project, Gen Mining retained P&E to complete an updated Mineral Resource estimate and Preliminary Economic Assessment on the Marathon Project. The NI 43-101 Technical Report - Updated Mineral Resource Estimate and Preliminary Economic Assessment of the Marathon Deposit, Thunder Bay Mining District, Northwestern Ontario, Canada (effective date of

January 6, 2020) was filed in February 2020. An amended Technical Report was filed in July 2020, which contained no material amendments to the original Technical Report filed in February 2020. On November 30, 2020, Gen Mining completed all the requirements under the joint venture agreement to increase its interest in the Property and Joint Venture to 80%. Following the increase in ownership to 80%, Sibanye Stillwater Limited did not continue funding the Joint Venture and its position decreased to 16.5%. Gen Mining purchased Sibanye Stillwater's ownership interest and completed the acquisition of the outstanding portion from Sibanye Stillwater Limited acquiring 100% interest in the Property on January 26, 2022.

On March 3, 2021, the Company announced the results of the Feasibility Study completed by G Mining Services. The 2021 Feasibility Study outlined an open pit mining operation with a rate of return over a 13-year mine life with the base case financials as follows: after-tax IRR of 30%, $NPV_{6\%}$ of \$1.07 billion, and a payback of 2.3 years.

On December 22, 2021, the Company announced that it had agreed to enter into a definitive Precious Metal Purchase Agreement with Wheaton Precious Metals Corp. ("Wheaton PMPA") Pursuant to the Precious Metal Purchase Agreement, Wheaton Precious Metals Corp. will pay Gen Mining total upfront cash consideration of \$240 million, \$40 million of which was paid on an early deposit basis (March and September 2022) prior to construction, with the remainder payable in four staged installments during construction, subject to various customary conditions being satisfied.

On May 19, 2022, the public hearings conducted by the Joint Review Panel for the Environmental Assessment of the Company's Marathon Project were concluded. The Joint Review Panel process is the highest standard of environmental assessment review in Canada. The Project's Environmental Impact Statement and other evidence were subject to a rigorous review by the Joint Review Panel with more than 50 participants. The Joint Review Panel report was delivered on August 3, 2022, with recommendations to the Federal and Provincial governments. The Honourable Steven Guilbeault, Federal Minister of Environment and Climate Change, and The Honourable David Piccini, Ontario Minister of the Environment, Conservation and Parks, each announced on November 30, 2022, that the Company's Marathon Project may proceed, subject to conditions set out in the Federal decision statement and the provincial approval order, respectively. The decision Statement and approval order were made following a thorough, multi-year, joint Federal and Provincial environmental assessment process, with input received from Indigenous groups, the public, federal government departments including the Ministry of Environment Canada and Climate Change, Fisheries and Oceans Canada, Natural Resources Canada and Transport Canada, and provincial government departments including Ministry of Northern Development, Mines, Natural Resources and Forestry, the Ministry of Environment, Conservation and Parks, the Ministry of Transportation, the Ministry of Labour, and the Technical Standards and Safety Authority.

In August 2022, Gen PGM entered into an agreement with Hycroft Mining Holding Corporation ("Hycroft") for the purchase of an unused, surplus SAG mill and, in August 2022, Gen PGM entered into an agreement with Hycroft for the purchase of an unused, surplus SAG mill and an unused, surplus ball mill, which was subsequently amended to include purchase of the main transformer and substation equipment for the process plant. This equipment is included in the detailed design of the Project.

On November 14, 2022, the Biigtigong Nishnaabeg community ratified the Community Benefit Agreement. This agreement between Gen PGM and Biigtigong Nishnaabeg describes the benefits the Biigtigong Nishnaabeg community will receive from the Project and details how the Project's impact on the community will be mitigated. It includes commitments from the Company regarding environmental management, employment, training and education, business opportunities, social and cultural support, and financial participation.

No previous mining activity has taken place on the Property.

1.6 Geological Setting and Mineralization

The Marathon Property is situated along the eastern margin of the Proterozoic Coldwell Complex, which is part of the Keweenawan Supergroup of igneous, volcanic and sedimentary rocks (Figure 1.3).

The Marathon deposit is hosted by the Two Duck Lake Gabbro, a late intrusive phase of the Eastern Gabbro (Figure 1.4). The Eastern Gabbro is a composite intrusion and occurs along the northern and eastern margin of the Coldwell Complex, which intrudes the much older Archean Schreiber-Hemlo Greenstone Belt. The entire Coldwell Complex is believed to have intruded over a relatively short period of time between 1108 and 1094 Ma.

The Marathon deposit consists of several large, thick and continuous zones of disseminated sulphide mineralization hosted within the Two Duck Lake Gabbro. The mineralized zones occur as shallow dipping sub-parallel lenses that follow the basal gabbro contact and are labeled as footwall, main, hanging wall zones and the W-Horizon. The Main Zone is the thickest and most continuous zone. For 418 drill hole intersections with mineralized intervals greater than 4 m thick, the average thickness is 42 m, and the maximum is 205 m.

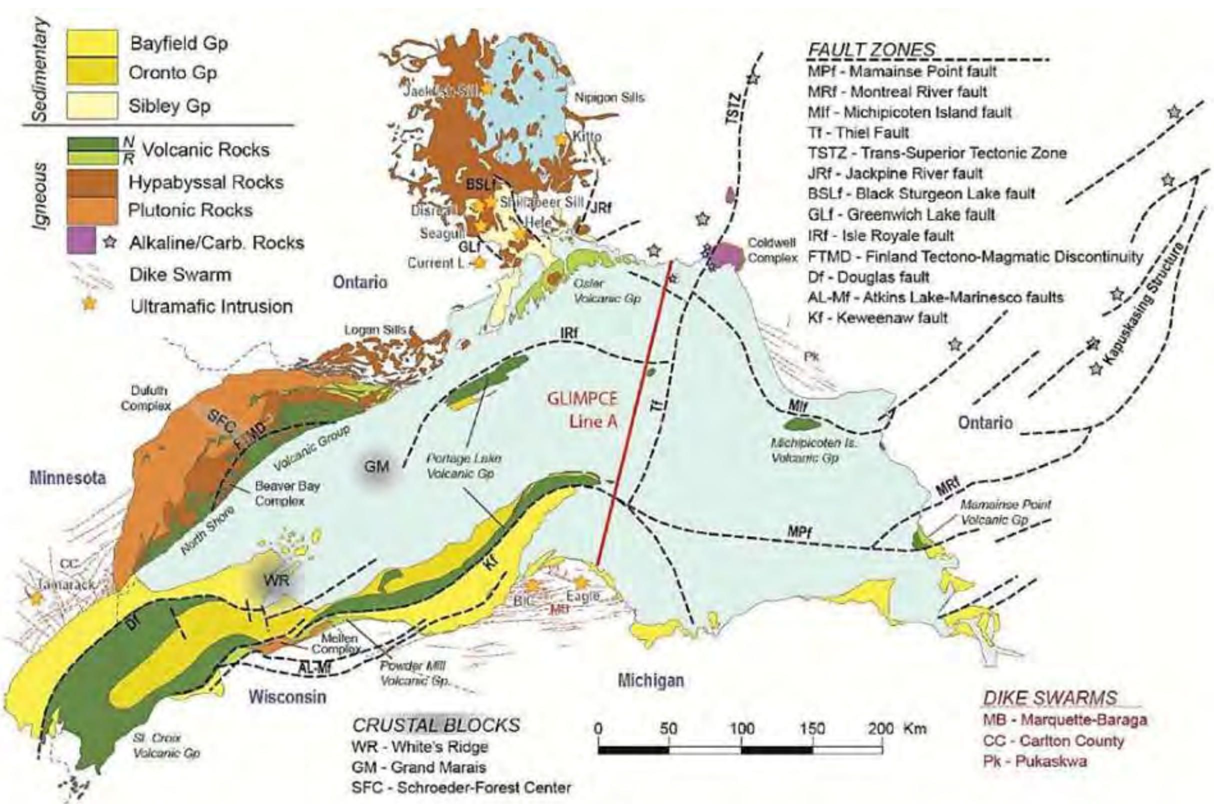
Sulphides in the Two Duck Lake Gabbro consist predominantly of chalcopyrite, pyrrhotite and minor amounts of bornite, pentlandite, cobaltite and pyrite. The proportions of sulphide minerals as determined in a QEMSCAN survey of a bulk sample are 2.75% pyrrhotite, 0.79% copper-iron sulphides (chalcopyrite and bornite), 0.09% pentlandite and trace amounts of pyrite, galena, and sphalerite.

The relative proportions of pyrrhotite and chalcopyrite vary significantly across the Marathon deposit; however, in general, the sulphide assemblage changes gradually up section from the base to the top of mineralized zones. Sulphides at the base of the Two Duck Lake Gabbro consist predominantly of pyrrhotite and minor chalcopyrite but the relative proportion of chalcopyrite increases up section to nearly 100% chalcopyrite near the top. In the W-Horizon, sulphides consist mainly of chalcopyrite and bornite and minor to trace amounts of pentlandite, cobaltite, pyrite and pyrrhotite. In general, the variations in the chalcopyrite to pyrrhotite ratio across the deposit, and from bottom to top of the deposit, correlates with variations in the copper/palladium ratio, with the highest concentrations of palladium occurring in samples with copper-rich sulphide assemblages.

The model that best explains the Marathon deposit is based on the accumulation of sulphides in basins and troughs of a magma conduit which underwent significant upgrading of copper and Platinum Group Metals content by the process of multistage dissolution grading that was described for similar disseminated mineralization in the Noril'sk region, Russia by Kerr and Leitch (2005).

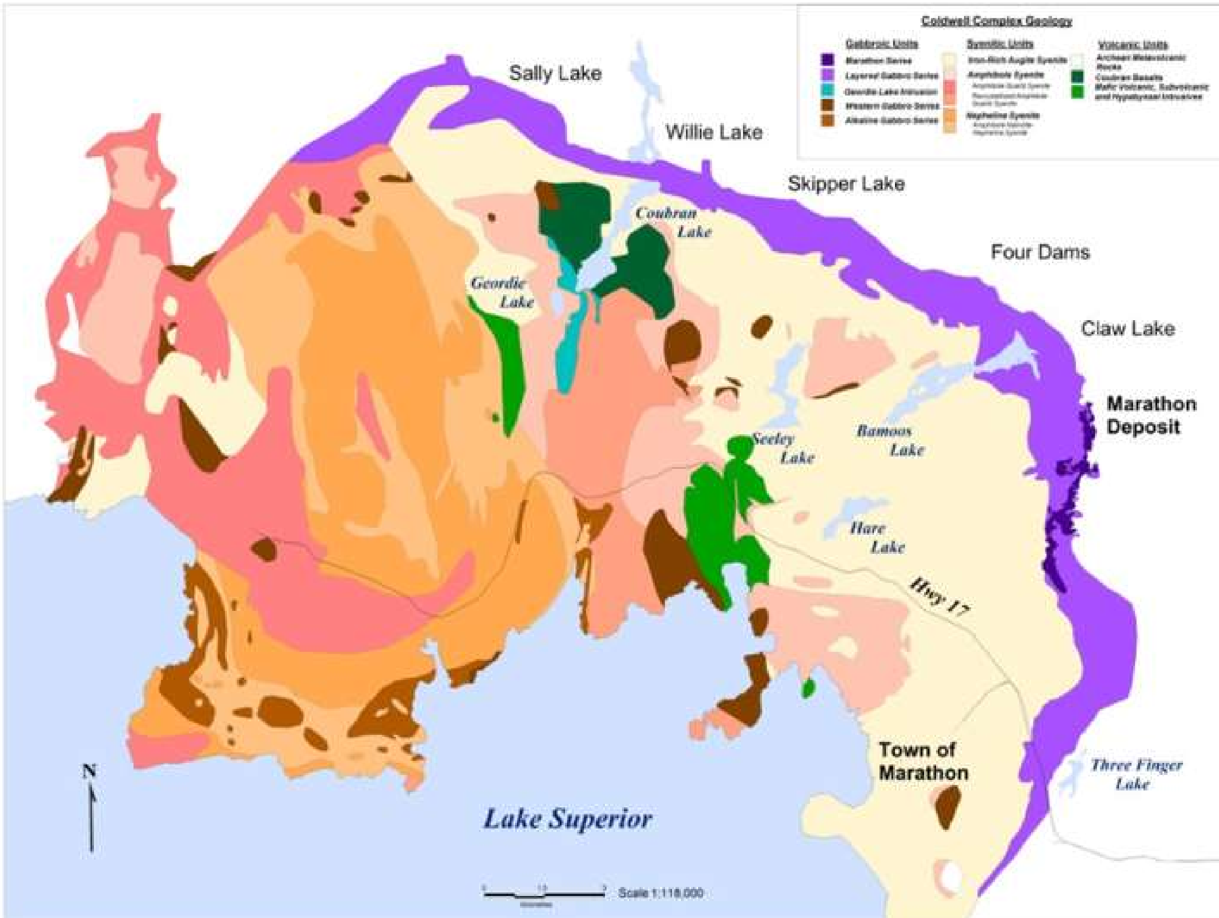
In addition to the Marathon deposit, the Property hosts other Platinum Group Metals deposits / mineralization in four additional areas – Geordie, Sally, Boyer and Four Dams.

Figure 1.3: Regional Structural Geology



***Source**: Miller and Nicholson, 2013.*

Figure 1.4: Coldwell Complex Geology



***Source**: Modified after Walker et al. (1993).*

1.7 <u>Deposit Types</u>

The Marathon deposit is one of several mafic to ultramafic intrusive bodies in the Mid-continent Rift System that host significant copper, nickel or Platinum Group Metals sulphide mineralization. These intrusions include the Yellow Dog peridotite (Eagle Deposit), the Tamarack Deposit, the Current Lake Intrusive Complex (Thunder Bay North Deposit), and the numerous intrusions located along the base of the Duluth Complex.

The intrusion and deposition of sulphides within magma conduits has recently been accepted as the dominant mineralization process chosen to explain rift related deposits and has been proposed for the Marathon, Thunder Bay North and the Eagle Deposits. The magma conduit model has grown in favour since it was proposed to explain deposits in the Noril'sk region and the deposits at Voisey's Bay, Newfoundland and Labrador, Canada.

Comparisons between the Mid-continent Rift System and the Voisey's Bay and Noril'sk settings point to several similarities that suggest that the Mid-continent Rift System is a likely setting for Ni-Cu mineralization. The continental rifting and associated voluminous igneous activity in all three regions formed in response to the rise of a hot plume of mantle material from deep in the Earth, fracturing the overlying continental crust. In the Mid-continent Rift System, melting of the plume produced more than 2 million cubic km of mostly basalt lava flows and related intrusions.

1.8 Exploration

In 2018, Stillwater partnered with PACIFIC (a consortium of industry, government, and academic partners) and completed a production-scale passive seismic survey of the Marathon deposit which resulted in a 3D velocity inversion model.

In 2019, exploration work by Gen Mining consisted of geologic mapping and prospecting at the Boyer zone and the northern extension of the Geordie Deposit. Three trenches were completed at Boyer exposing the continuation of mineralization at surface. A passive seismic survey was completed at Sally to help define deep high-density targets for potential drill testing. Borehole EM surveys were completed by Crone Geophysics on diamond drill holes SL-19-72, M-19-536 and M-19-537.

In 2020, to compliment the previous seismic surveys, a magnetotelluric survey was conducted over a portion of the Marathon deposit and an area immediately west of the Marathon deposit as well as over the Sally deposit and the immediate surrounding area.

In 2021, a high-resolution LiDAR and aerial photography survey was carried out over the entire property. Field mapping programs were carried out at the Four Dams, Willie Lake, and Redstone prospects as well as the area immediately west of the Marathon deposit. Three trenches were completed at the Marathon deposit to better define the ore-footwall contact in areas of lower confidence.

No significant exploration work was carried out in 2022.

1.9 Drilling

In 2019, Gen Mining completed a 12,434.5 m exploration drilling program on the Marathon Property. The program tested several high-priority targets along a strike length of more than 25 km.

In 2020, Gen Mining completed 12 holes totalling 5,068 m. The drilling was focused on the Feeder Zone conduit associated with the Main Marathon deposit and the northern limb of the W-Horizon. This drilling followed the successful completion, in 2019, of drill holes M-19-537 and M-19-538 which intercepted the down dip continuation of the Main Marathon deposit for the first time. The 2020 drilling filled a 300 m gap between the historical drilling and the 2019 drilling south of the 5,404,900N fault. Additional targets included the conductive zone west of the Marathon deposit identified in the 2020 MT survey and the down dip extension of high-grade Platinum Group Metals mineralization in the W-Horizon.

In 2021, Gen Mining completed 22 holes totalling 9,875.2 m, of which 11 holes (5,735.2 m) were completed at the Central Feeder Zone and followed up on mineralization defined as part of the 2020 drill program. An additional 11 holes (4,140.0 m) were drilled at the Biiwobik Prospect, testing the Chonolith and Powerline West occurrences.

In 2022, Gen Mining completed 48 holes totalling 7326.9 m. The majority of the program was aimed at de-risking Mineral Resources and gaining confidence in the Mineral Reserve in the North, Central and South Pits. An additional 741 m (2 holes), were completed to test continuity between the Main Zone and Central Feeder Zone. Finally, 125.1 m of drilling was completed as a means of extending hole M-21-551, which was drilled in 2021 but had to be abandoned due to technical issues prior to reaching its target depth.

1.10 Sample Preparation, Analysis and Security

The core and trench cut sampling protocol (preparation, analysis and security procedures) instituted and used by past Project operator Marathon PGM Corp. in each of their drilling and other rock sampling programs were identical to those reported in prior NI 43-101 Technical Reports on the Property.

Prior to 2011, all drill core samples were sent for preparation and analysis to Accurassay in Thunder Bay. From 2011 to 2022, all drill core samples were sent for preparation to ALS Minerals in Thunder Bay and subsequent analysis at the ALS Vancouver facility.

Marathon PGM Corp. continued with a robust Quality Assurance/Quality Control (QA/QC) program that had been implemented by that company in the mid-2000s. The QA/QC program consisted of the insertion of reference materials, field blanks and duplicate pair monitoring. All data from the 2009 and 2011 drill programs were examined by P&E. Drill data prior to 2009 were previously examined by P&E and accepted for use in previous Mineral Resource estimates.

P&E has reviewed the corresponding laboratory QC data for Gen Mining’s 2019-2022 drilling programs, including standards, blanks and duplicates, and does not consider that the laboratory QC data indicates issues with data accuracy, contamination or precision.

P&E considers the sampling methods from the current and past drilling programs to be satisfactory. P&E considers the data to be of good quality and acceptable for use in the current Mineral Resource estimates for the Marathon, Geordie and Sally deposits.

1.11 Data Verification

The Project was visited by Mr. David Burga, P.Geo. of P&E, an independent Qualified Person as defined by NI 43-101 on April 4, 2012 and he collected 10 verification samples from nine holes. The samples were taken by Mr. Burga to AGAT Labs in Mississauga, ON for analysis. Copper, silver and nickel were analyzed using 4-acid digest with AAS finish. Gold, platinum and palladium were analyzed using lead collection fire assay with ICP-OES finish.

A site visit to the Project was undertaken by Mr. Bruce Mackie of Bruce Mackie Geological Consulting Services ("Mackie") on May 4, 2019. As part of the site visit, 12 verification samples from nine diamond drill holes intervals were taken by Mr. Mackie, P.Geo. and submitted to Activation Laboratories Ltd. in Thunder Bay and analyzed for Au, Ag, Pt, Pd and Cu.

For both site visits (Burga and Mackie), drill logs for the sections reviewed were found to be appropriately detailed and present a reasonable representation of geology, alteration mineralization and structure. No discrepancies in the sample tag numbers within the core trays and the intervals quoted in the aforementioned Excel spreadsheets were noted.

Based on the results of the Investigation, Messrs. Burga and Mackie are of the professional opinion that the mineralized drill hole assay results and corresponding drill hole logs reported by Stillwater and Marathon PGM that were the subject of their investigations are verifiable and accurate and portray a reasonable representation of the types of mineralization encountered on the Marathon and Geordie deposits.

Based on the review from P&E, there is good correlation between the independent verification samples and the original analyses in the Company database.

Based upon the evaluation of the QA/QC program undertaken by the Company, as well as database verification carried out by P&E, it is P&E's opinion that the data is robust and suitable for use in the Mineral Resource estimates for the Marathon, Geordie and Sally deposits.

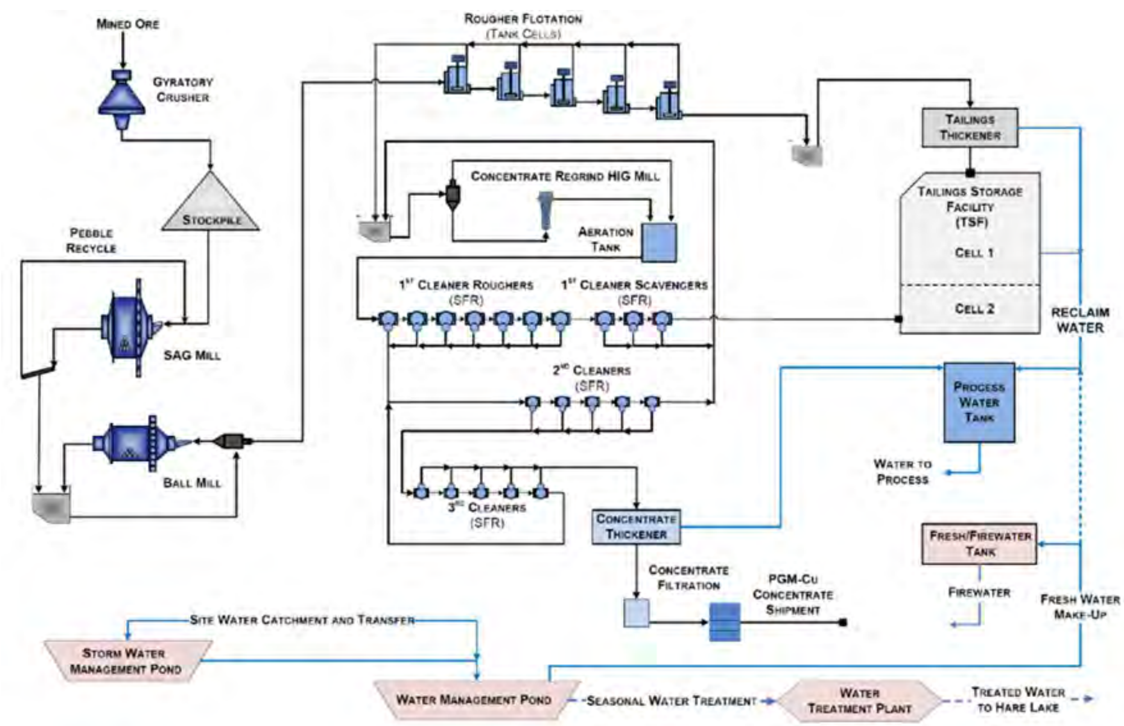
Process QP has reviewed the metallurgical test results and the composite samples that were selected for metallurgical testing and considers it suitable for this level of study and support the process design in this report.

1.12 Mineral Processing and Metallurgical Testing

Metallurgical testing and process flowsheet definition for the Marathon Project dates back to 1960. Historical testing has allowed for a thorough review of concepts and criteria to optimize process plant design and metallurgical performance. Tests included crushing, grinding, as well as batch, cycle and mini-pilot plant-scale flotation testing. The focus of the 2020 metallurgical testwork programs was to initially validate then to optimize the process flowsheet and associated criteria with the priority of maximizing palladium and copper recovery. The 2020 metallurgical testing, along with data from historical results, were used to shape and optimize the process flowsheet. The 2020 metallurgical testwork (in-lab work) was completed at SGS Canada Inc. ("SGS") in Lakefield, Ontario spanning the period June 2020 to December 2020. Additional metallurgical testing was undertaken at SGS during 2022, including specific grinding energy testing for concentrate regrind mill sizing, and additional locked cycle testing on metallurgical drill holes completed by Gen Mining in Q2-Q3 2023 to refine GeoMet model recovery estimation for payable metals.

The processing strategy (Figure 1.5) established from the 2020 test programs has been further optimized through the plant engineering undertaken by Wood in 2022-2023. The process flowsheet and selection of equipment is an improvement relative to previous designs with improved operability of the circuit and higher palladium and copper recovery.

Figure 1.5: Optimized Process Flowsheet



1.12.1 Metallurgical Recovery

Determination of a predictive curve for metal recovery to a combined Cu-PGM concentrate was initially established as part of the 2020 metallurgical testing program. Metal recovery estimates as a function of head grade have been refined in more recent Q4 2022 testwork with separate GeoMet model equations for copper, palladium, platinum, gold and silver summarized in Table 1.1. Based on the outcome of 2022 testwork and improved metal recovery, a previously considered PGM-scavenger circuit to reprocess the rougher tailings coarse fraction is excluded from current planning.

Table 1.1: GeoMet Equations for metal recovery to final concentrate

Parameter	GeoMet Formula	Maximum Value
%Rec Cu to Final Conc	= 97.55 x (% Cu head grade) $^{0.0239}$	94% Rec Cu
%Rec Pd to Final Conc	= 89.14 x (g/t Pd head grade) $^{0.0203}$	90% Rec Pd
%Rec Pt to Final Conc	= 104.51 x (g/t Pt head grade) $^{0.2034}$	84% Rec Pt
%Rec Au to Final Conc	= 116.51 x (g/t Au head grade) $^{0.1822}$	86% Rec Au
%Rec Ag to Final Conc	= 50.82 x (g/t Ag head grade) $^{0.6090}$	68% Rec Ag
%Mass Pull to Final Conc	= 0.625 x $e^{(2.899 \text{ x } \%\text{Cu head grade})}$	2.0% Mass Pull

The process plant metallurgical recovery (at the average head grade) is estimated at an average of 88.0% palladium, 93.5% copper, 75.3% platinum, 71.5% gold and 66.4% silver.

1.12.2 Recovery Methods and Plant Design

The Marathon Project process design is based on 2020-2022 metallurgical test programs and operational design criteria focused on Platinum Group Metals and copper recovery. In 2022, the Company finalized an agreement with Hycroft for the purchase of an unused, surplus SAG mill and an unused, surplus ball mill. The process plant flowsheet includes a conventional comminution circuit consisting of a SAG mill, followed by a ball mill (an “SAB” circuit). With the added capacity of the Hycroft mills, the pebble crusher (included in the 2021 FS) is no longer required. The current processing plant will support a throughput of 10.1 Mt/y (27,700 t/d), an increase or 10% from the 2021 Feasibility Study of 9.2 Mt/y (25,200 t/d).

After the comminution circuit, the flowsheet includes a flotation circuit, followed by concentrate dewatering and tailings impoundment. Cu-PGM flotation includes a rougher flotation circuit followed by regrinding rougher concentrate and a three-stage cleaner circuit. The 2021 Feasibility Study flotation circuit design was revised to replace the Direct Flotation Reactors previously included with conventional open tank cells for the roughers followed by Woodgrove Staged Flotation Reactors for the cleaning circuit to de-risk the start-up and early years of the operation.

The processing plant will produce a Cu-PGM concentrate.

1.13 Mineral Resource Estimate

The Mineral Resource estimate presented herein has been prepared following the guidelines of the Canadian Securities Administrators’ NI 43-101 and Form 43-101F1 and in conformity with generally accepted “CIM Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines (2019).

The Mineral Resource estimate in Table 1.2 was completed by Gen Mining and reviewed by P&E. The Authors are not aware of any known permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

Table 1.2: Pit Constrained Mineral Resource Estimates for the Marathon, Geordie and Sally Deposits (Effective date December 31, 2022)

Mineral Resource Classification	Tonnes	Pd		Cu		Pt		Au		Ag	
	k	g/t	koz	%	M lbs	g/t	koz	g/t	koz	g/t	koz
Marathon Deposit											
Measured	158,682	0.60	3,077	0.20	712	0.19	995	0.07	359	1.75	8,939
Indicated	29,905	0.43	412	0.19	124	0.14	136	0.06	59	1.64	1,575
M+I	**188,587**	**0.58**	**3,489**	**0.20**	**836**	**0.19**	**1131**	**0.07**	**418**	**1.73**	**10,514**
Inferred	1,662	0.37	20	0.16	6	0.14	7	0.07	4	1.25	67
Geordie Deposit											
Indicated	17,268	0.56	312	0.35	133	0.04	20	0.05	25	2.4	1,351
Inferred	12,899	0.51	212	0.28	80	0.03	12	0.03	14	2.4	982
Sally Deposit											
Indicated	24,801	0.35	278	0.17	93	0.2	160	0.07	56	0.7	567
Inferred	14,019	0.28	124	0.19	57	0.15	70	0.05	24	0.6	280
Total Project											
Measured	158,682	0.60	3,077	0.20	712	0.19	995	0.07	359	1.75	8,939
Indicated	71,974	0.43	1,002	0.22	350	0.14	316	0.06	140	1.5	3,493
M+I	**230,656**	**0.55**	**4,079**	**0.21**	**1,062**	**0.18**	**1,311**	**0.07**	**499**	**1.67**	**12,432**
Inferred	28,580	0.39	356	0.23	143	0.1	89	0.04	42	1.45	1,329

***Notes**:*

1. *Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*
2. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
3. *The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
4. *The Marathon Mineral Resource is reported within a constrained pit shell at a NSR cut-off value of $15/t.*
5. *Marathon NSR (C$/t) = (Cu % x 88.72) + (Ag g/t x 0.47) + (Au g/t x 44.69) + (Pd g/t x 58.63) + (Pt g/t x 28.54) - 3.37.*
6. *The Marathon Mineral Resource estimate was based on metal prices of US$1,800/oz Pd, US$3.50/lb Cu, US$1,000/oz Pt, US$1,600/oz Au and US$20/oz Ag, and an exchange rate of 1.30 C$: 1 US$.*
7. *The Sally and Geordie Mineral Resources are reported within a constraining pit shell at a NSR cut-off value of $13/t.*
8. *Sally and Geordie NSR (C$/t) = (Ag g/t x 0.48) + (Au g/t x 42.14) + (Cu % x 73.27) + (Pd g/t x 50.50) + (Pt g/t x 25.07) – 2.62.*
9. *The Sally and Geordie Mineral Resource estimate was based on metal prices of US$1,600/oz Pd, US$3.00/lb Cu, US$900/oz Pt, US$1,500/oz Au and US$18/oz Ag, and an exchange rate of 1.30 C$: 1 US$.*
10. *Contained metal totals may differ due to rounding.*

1.13.1 Mineral Resource Estimate – Marathon Deposit

Mineral Resources for the Marathon deposit reported herein have been constrained within an optimized pit shell. The results within the constraining pit shell are used solely for the purpose of reporting Mineral

Resources and include Measured, Indicated and Inferred Mineral Resources. Pit-Constrained Mineral Resources are reported using a NSR cut-off value of $15 /t. Wireframe modeling utilized Seequent Leapfrog Geo™ software. Mineral Resource estimation was carried out using Datamine Studio RM software. Variography was carried out using Snowden Supervisor™. Pit optimization was carried out using Whittle.

The modeled Marathon mineralization domains extend along a corridor 2,000 m wide and 3,500 m in length. An orthogonal block model was established with the block model limits selected so as to cover the extent of the mineralized structures, the proposed open pit design, and to reflect the general nature of the mineralized domains. The block model consists of separate variables for estimated grades, rock codes, percent, bulk density and classification attributes. A sub-celled block model was used to accurately represent the volume and tonnage contained within the constraining mineralized domains. The block size used in the estimate is 5 m (easting), 10 m (northing), 5 m (elevation) with no rotation assumed.

The Mineral Resource estimate was constrained by mineralization domains that form hard boundaries between the respective composite samples. Block grades were estimated in a single pass with Inverse Distance Cubed (ID3) interpolation using a minimum of four and a maximum of 12 composites with a maximum of three samples per drill hole. Composited samples were selected within a 200 m x 200 m x 50 m diameter search envelope oriented to the dip and dip direction of the mineralization. The Datamine Dynamic Anisotropy method was used to estimate dip and dip direction values for each block. This has allowed the search ellipse to be optimized to the dip and dip direction of the mineralization. For each grade element, an uncapped Nearest Neighbor model was also generated using the same search parameters. An NSR block model was subsequently calculated from the estimated block grades.

Blocks were classified algorithmically based on the local drill hole spacing within each domain. All blocks within 70 m of four or more drill holes were classified as Measured and blocks within 120 m of three or more drill holes were classified as Indicated. All additional estimated blocks were classified as Inferred.

P&E considers that the information available for the Marathon deposit is reliable, demonstrates consistent geological and grade continuity, and satisfies the requirements for a Mineral Resource estimate.

1.13.2 Mineral Resource Estimate – Geordie and Sally Deposits

Mineral Resource estimates were generated by P&E for the Geordie and Sally deposits. The methodologies to create the block models were similar to those used for the Marathon deposit. The GEOVIA GEMS™ V6.8.2 database was used for the Geordie and Sally deposits Mineral Resource estimates.

1.14 Mineral Reserve Estimate – Marathon Deposit

The Mineral Reserve estimate was prepared by GMS (Table 1.3). The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM definitions (2014) and is suitable for public reporting. As such, the

Mineral Reserves are based on Measured and Indicated Mineral Resources which were considered for optimization purposes with mining dilution factors applied. The Mineral Reserve does not include any Inferred Mineral Resources which were classified as waste for reporting purposes.

The resource model (Subsection 1.13.1) was provided to GMS as a regularized block model with a standard SMU block size of 5 m x 10 m x 5 m.

Open pit optimization was conducted in GEOVIA Whittle™ to determine the optimal economic shape of the open pit with pit slopes applied according to Knight Piésold feasibility level pit slope design study. The conclusions of this study have been used as an input to the pit optimization and design process.

The Marathon Project uses an NSR value for the mineralization cut-off grade. The marginal cut-off grade corresponds to the ore-based cost. However, an elevated NSR cut-off value was applied of $16.90/t of ore (US$13.00/t). These elevated NSR cut-off values applied to select blocks prior to dilution will provide some operating margin and cover the impact of mining dilution.

A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic cut-off grade. The block contacts are then used to estimate a dilution skin around ore blocks to estimate an expected dilution during mining. The dilution skin consists of 1.0 m of material in a north-south direction (across strike) and 1.0 m in an east-west direction (along strike). The dilution is therefore specific to the geometry of the ore body and the number of contacts between ore and waste. The ore body consists of two styles of mineralization. There are massive-mineralized envelopes such as for the Main Zone which incur relatively little dilution and other narrower mineralized envelopes (namely the W-Horizon) that incur higher mining dilutions with this estimation technique.

Table 1.3: Marathon Project Open Pit Mineral Reserve Estimates

Mineral Reserves	Tonnage	Pd		Cu		Pt		Au		Ag	
	kt	g/t	koz	%	M lb	g/t	koz	g/t	koz	g/t	koz
Proven	114,798	0.65	2,382	0.21%	530	0.2	744	0.07	259	1.68	6,191
Probable	12,863	0.47	193	0.20%	55	0.15	61	0.06	26	1.53	635
P&P	**127,662**	**0.63**	**2,575**	**0.21%**	**586**	**0.2**	**806**	**0.07**	**285**	**1.66**	**6,825**

Notes:

1. *Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) were used for Mineral Reserve classification.*
2. *Mineral Reserve estimate completed by Alexandre Dorval, P.Eng., of GMS, an independent QP, as defined by NI 43-101.*
3. *Mineral Reserves were estimated at a cut-off value $16.90 NSR/t of ore.*
4. *Mineral Reserves were estimated using the following long-term metal prices: Pd = US$1,500/oz, Pt = US$1,000/oz, Cu = US$3.50/lb, Au = US$1,600/oz and Ag = US$20/oz, and an exchange rate of 1 US$: 1.30 C$. The pit designs are based on a pit shell selected at a revenue factor of 0.74.*
5. *A minimum mining width of 5 m was used.*
6. *Bulk density of ore is variable and averages 3.1 t/m^3.*
7. *The average strip ratio is 2.6:1.*
8. *The average mining dilution factor is 9%.*
9. *Numbers may not add due to rounding.*

1.15 Mining Methods

Mining methods will employ conventional open pit, truck and shovel operating practice. Three pits will be mined over the 13-year mine life with an additional two years of pre-commercial production (or pre-production) mining to be undertaken where waste material is being mined for construction and ore will be stockpiled ahead of plant commissioning. The fleet will be owner-operated and will include outsourcing of certain support activities such as explosives manufacturing and blasting. Production drilling and mining operations will take place on a 10 m bench height. The primary loading equipment will consist of hydraulic face shovels (29 m^3 bucket size) and large front-end wheel loader (19 m^3 bucket size). The loading fleet is matched with a fleet of 246 t haulage trucks. A fleet of 90 t and 45 t excavators will be used to excavate the limited volume of overburden material and will also be allocated to mining of the narrow-thickness ore zones mainly associated with the W-Horizon in the South Pit to mitigate additional dilution.

Mining production at peak capacity is 43 Mt/y (118,000 t/d).

The Marathon deposit is well defined and characterized by ore material outcropping on surface, wide, and moderately dipping mineralized zones. The mine plan includes the development of three open pits aligned generally in a north – south orientation (North pit, Central pit and South pit) over a total approximate strike length of 3 km. Each of the pits have been designed and included pit wall push backs or phases to allow for extraction over the 13-year mine life. The designs include in-pit dumping for the South and Central pits.

The open pit operation includes a waste rock dump immediately to the east of the open pits and an ore stockpile (peak utilization of approximately 10 Mt) to the west of the pits, proximal to the crusher location.

1.16 Production Profile

A high-level summary of the project's production rates by stage is presented in Table 1.4.

Table 1.4: High-Level Production Profile

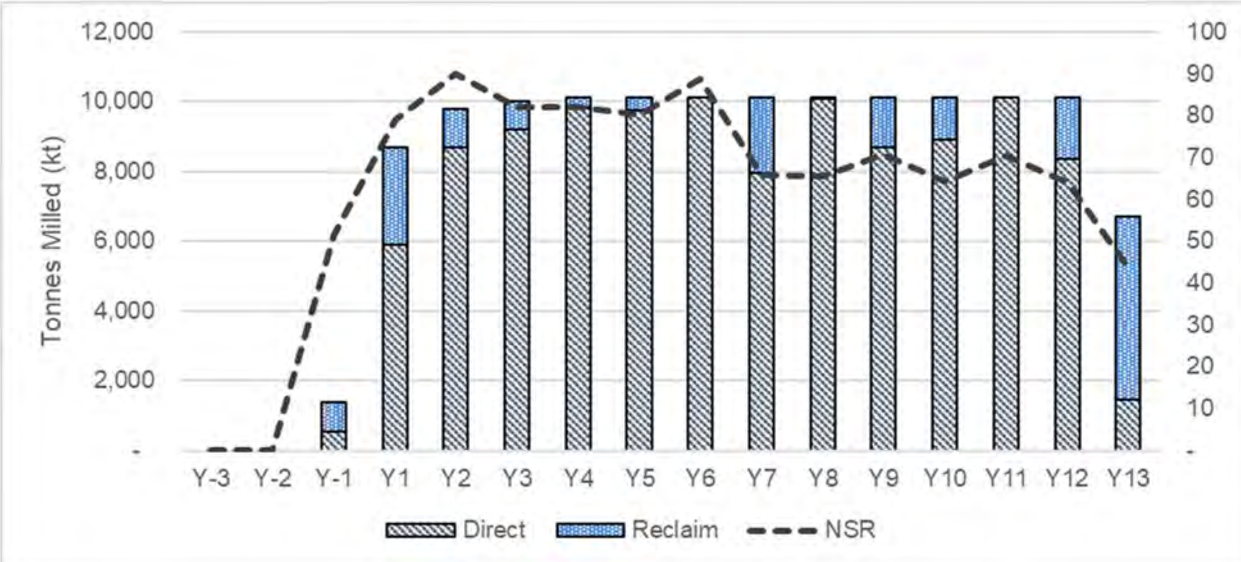
Operating Data	Units	Pre-Production	Operations	Total
Mine Life	years	2.5	12.5	15.0
Total Milled Tonnes	Mt	2.6	124	127
Total Mined Tonnes	Mt	19.5	440	460
Strip Ratio	waste:ore	4.43	2.56	2.60
Metal Production[1]	**Units**	**Recovered Metals**	**Payable Metal**	**% of Revenue**
Palladium	k oz	2,266	2,122	58%
Copper	M lbs	548	517	29%
Platinum	k oz	607	485	7%
Gold	k oz	204	158	4%
Silver	k oz	4,529	3,156	1%
***Note**: [1] including pre-production period and operations phase.*				

1.16.1 Milling Schedule

Operating life for the Project is approximately 13 years. Design milling capacity is 10.1 Mt/y. (27,700 t/d) with a ramp up from 9.2 Mt occurring following a powerline upgrade scheduled for 2027. Annual mill feed tonnage is kept constant with mined ore direct from the pits and rehandled ore from stockpiles to fill plant capacity (Figure 1.6). Reclaim of the low-grade material that was previously stockpiled occurs typically during the last 2 years of operations.

Medium and high-grade ore will be stockpiled for the first 2 years of mining until it is rehandled to the mill as higher grade ore is prioritized. The peak stockpile capacity is approximately 14 Mt. All material is milled by the end of project life.

Figure 1.6: Mill Production Profile



1.16.2 Mine Production Profile – Key Metals

The profile of recovered metal production is summarized by metal in Figure 1.7 to Figure 1.9. The mine production profile is summarized in Table 1.5.

Figure 1.7: Palladium – Recovered Metal

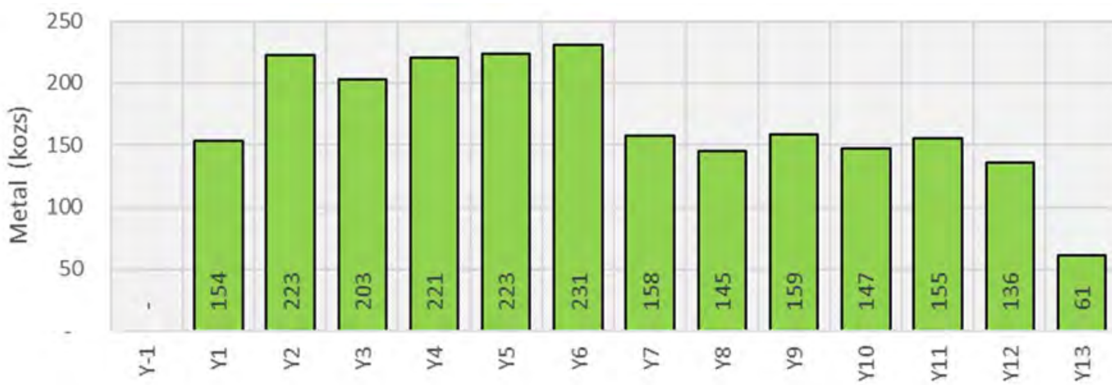


Figure 1.8: Copper – Recovered Metal

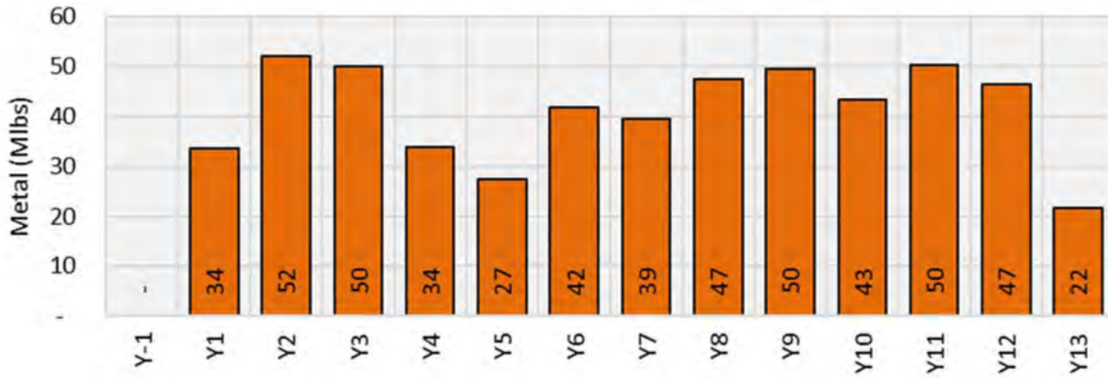


Figure 1.9: Platinum, Gold and Silver – Recovered Metal

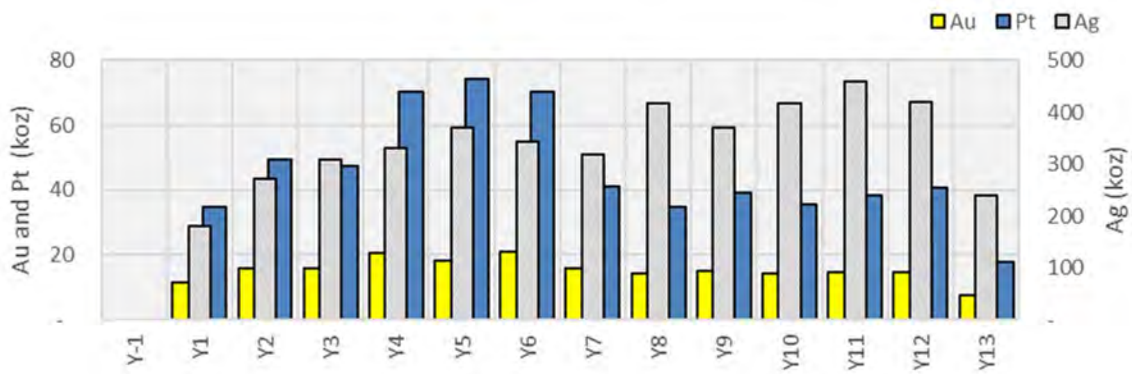


Table 1.5: Life-of-Mine Production Profile

		Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Total
Total Tonnage	**Mt**	**0.3**	**4.9**	**9.8**	**32.5**	**43.0**	**41.7**	**41.8**	**41.9**	**41.5**	**41.0**	**43.0**	**43.0**	**39.4**	**22.2**	**12.1**	**1.9**	**459.7**
Total Waste	Mt	0.3	4.8	7.8	23.4	31.4	30.7	30.0	31.2	30.4	31.8	31.8	33.5	29.6	11.2	3.7	0.4	**332.1**
Overburden	Mt	0.0	0.5	0.4	0.9	0.2	0.3	0.2	0.1	0.4	0.1	0.0	0.0	0.0	0.0	0.0	0.0	**3.2**
NAG	Mt	0.3	4.4	6.6	19.9	28.1	27.5	25.7	28.6	29.1	29.4	29.9	31.2	25.8	8.5	2.1	0.2	**297.3**
PAG	Mt	0.0	0.4	1.2	3.5	3.3	3.2	4.3	2.6	1.3	2.4	1.9	2.3	3.8	2.7	1.6	0.3	**34.8**
Strip Ratio	W:O		66.2	4.0	2.6	2.7	2.8	2.5	2.9	2.7	3.5	2.8	3.5	3.0	1.0	0.4	0.3	**2.60**
Ore Tonnage	**Mt**	**0.0**	**0.1**	**1.9**	**9.1**	**11.6**	**11.1**	**11.8**	**10.6**	**11.1**	**9.2**	**11.2**	**9.5**	**9.7**	**10.9**	**8.4**	**1.5**	**127.7**
Cu Grade	%	0.07	0.07	0.20	0.23	0.23	0.23	0.15	0.13	0.19	0.19	0.21	0.24	0.21	0.23	0.25	0.37	**0.21**
Ag Grade	g/t	1.33	1.33	1.70	1.27	1.29	1.48	1.52	1.68	1.54	1.60	1.84	1.71	1.93	2.04	2.00	2.57	**1.66**
Au Grade	g/t	0.04	0.04	0.07	0.07	0.06	0.07	0.08	0.08	0.08	0.07	0.06	0.07	0.06	0.06	0.07	0.08	**0.07**
Pt Grade	g/t	0.13	0.13	0.22	0.20	0.19	0.19	0.25	0.27	0.25	0.18	0.14	0.17	0.16	0.16	0.18	0.17	**0.20**
Pd Grade	g/t	0.41	0.41	0.66	0.75	0.70	0.67	0.69	0.75	0.75	0.57	0.48	0.57	0.52	0.52	0.52	0.55	**0.63**

1.17 Project Infrastructure

The existing regional infrastructure provides the Project with a number of logistical opportunities for project execution and operations including the availability and movement of personnel, materials, equipment and consumables to site, and the transport of Cu-PGM concentrate by rail or highway to third party smelters.

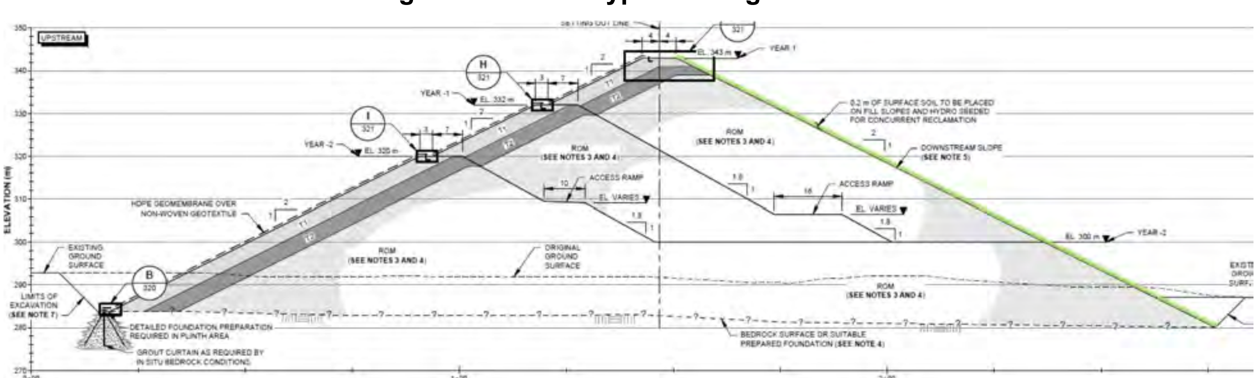
Project design for the Feasibility Study update has considered access roads, processing facilities, workshops, warehouse, administrative buildings, water treatment, explosive plant, communication systems, power and power transmission lines, water management and environmental controls. Off-site infrastructure (including transload concentrate facility, assay lab and accommodation units) to support the Project and operation have been included.

1.18 Tailings Storage Facility

The Tailings Storage Facility ("TSF") and associated water management facilities have been designed to meet the requirements of the Lakes and River Improvement Act ("LRIA") Ministry of Natural Resources and Forestry (MNRF, 2017) and the Canadian Dam Association guidelines ("CDA", 2021). The TSF is located west of the processing plant and generally south-west of the open pits.

The TSF design methodology includes for perimeter embankments being raised using downstream construction with run-of-mine rockfill (Figure 1.10). The embankment will be primarily founded directly on bedrock or competent overburden. The majority of TSF area provides for robust foundation conditions primarily consisting of exposed bedrock. A thin intermittent layer of glacial drift (sand and gravels) is present within localized areas. The upstream transition and filter zones are graded to the tailings and a high-density polyethylene geomembrane is included on embankment face to minimize seepage. The embankments will be raised in stages through the life of mine to provide the required storage capacity for tailings and temporary water management. The embankment stability exceeds the factor of safety requirements outlined in LRIA and CDA guidelines for all stages of mine life (construction, operation and closure).

Figure 1.10: TSF Typical Design Section

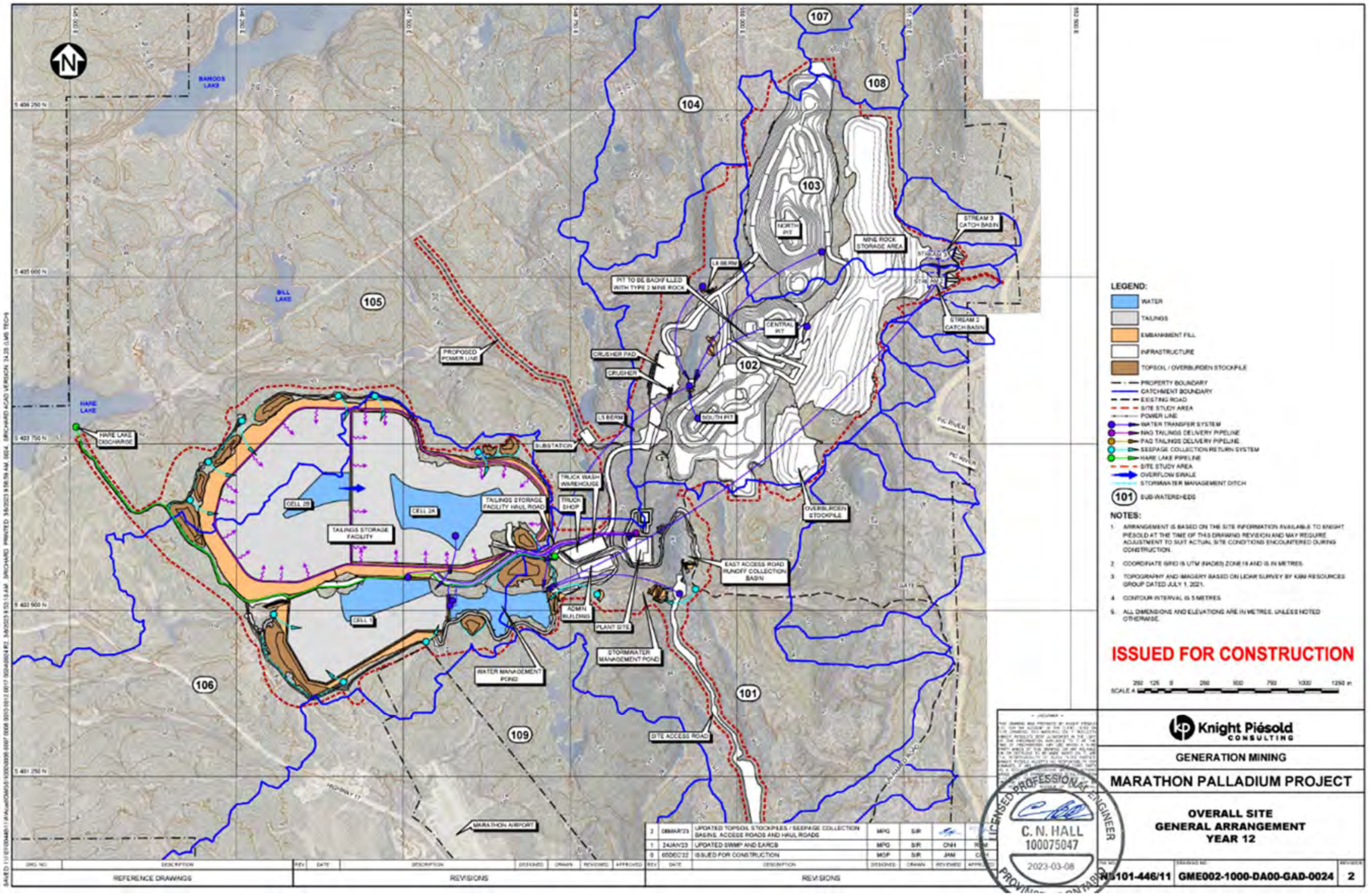


The TSF arrangement includes two storage cells. Cell 1 and Cell 2A will provide storage for the initial production years; Cell 2A and 2B will provide storage for the remaining production years. Potentially acid generating (PAG or Type 2) material will be stored in Cell 2A (designed to ensure PAG material is saturated for closure conditions and in perpetuity).

The TSF will provide permanent, secure confinement for approximately 120 Mt of tailings material and 30 Mt of PAG mine rock. The available storage capacity within the TSF has been aligned with production profile requirements for the life of mine.

The water management facilities (Figure 1.11) associated with the TSF include a Water Management Pond and a Stormwater Management Pond. The Water Management Pond is located east of Cell 1 and will be the source of plant operating water, manage contact water from the site and allow for seasonal discharge to Hare Lake as required. An additional collection basin, the East Access Road Collection Basin (EARCB) is planned east of the plant site and below the main site access road. Together, the Stormwater Management Pond and East Access Road Collection Basin will manage contact water from the plant area.

Figure 1.11: General Arrangement of Site, TSF and Water Management system



1.19 Market Studies and Contracts

1.19.1 Metal Price Data

The following information outlines the considerations used for determining the metal price assumptions for the Economic Analysis.

Table 1.6: Commodity Prices and Exchange Rates [1]

Description	FX Rate	Palladium	Copper	Platinum	Gold	Silver
	C$:US$	US$/oz	US$/lb	US$/oz	US$/oz	US$/oz
3-Year Trailing Average [2]	1.297	$2,219	$3.67	$980	$1,791	$22.47
2-Year Trailing Average [2,3]	1.277	$2,235	$4.11	$1,026	$1,800	$23.33
December 31, 2022 Value	1.355	$1,789	$3.80	$1,074	$1,825	$23.95
Assumption used in Economic Analysis[4]	1.35	$1,800	$3.70	$1,000	$1,800	$22.50

Notes:
1. Source: *FactSet.*
2. Nominal price.
3. 2-year trailing shown for reference.
4. Lesser of 3-year trailing average and December 31, 2022, rounded.

1.19.2 Concentrate Sale

Gen Mining has run a competitive tender process with multi-metallic international smelters that are capable of recovering PGMs. Firm term sheets have been received from domestic and international smelters with competitive treatment charges, refining charges (TC/RC) and payability terms reflecting the high value per tonne and potential for higher margins than traditional clean copper concentrates.

Final payment terms will be based on prevailing metal prices from the London Metals Exchange (copper) and the London Bullion Market Association (palladium, platinum, gold and silver), subject to payabilities and minimum deductions. The economic model assumes an average of TC/RCs and payability terms between smelters where the product is envisioned to be sold. A summary of the payment terms and costs is presented in Table 1.7 and Table 1.8.

Table 1.7: Payable Metals in Concentrates

Payable Element	Approximate Net Payable Rates	Minimum Deductions
Palladium	95%	2.6 g/t
Copper	96.5%	1.1%
Gold	75%	1 g/t
Platinum	77%	2.6 g/t
Silver	75%	30 g/t

Table 1.8: Treatment and Refining Charges

Element	Treatment Charge	Refining Charge
Palladium	-	US$24.50/oz
Copper	US$79/dmt	US$0.079/lb
Gold	-	US$5.00/oz
Platinum	-	US$24.50/oz
Silver	-	US$0.50/oz

1.20 Environmental Studies, Permits, and Social or Community Impacts

The Environmental Assessment for the Project was approved on November 30, 2022 in accordance with the Canadian Environmental Assessment Act (CEAA, 2012) and Ontario's Environmental Assessment Act through a Joint Review Panel pursuant to the Canada-Ontario Agreement on Environmental Assessment Cooperation (2004).

As of the effective date of this Technical Report, the Project is in the process of obtaining various federal, provincial and municipal permits, approvals and licences required to construct and operate the Project.

A total of 16 Indigenous groups were identified by the Crown (Canada and Ontario) as having a potential interest in the Project. Of the 16 Indigenous groups, seven groups indicated that they were interested in participating in consultation processes related to the Project. The seven groups are Biigtigong Nishnaabeg First Nation, Pays Plat First Nation, Mitchipicoten First Nation, Ginoogaming First Nation, Superior North Shore Métis – MNO, Jackfish Métis – Ontario Coalition of Indigenous Peoples and Red Sky Métis Independent Nation.

1.21 Communities Proximal to the Project

The Project is situated within the geographic territory of the Robinson Superior Treaty area. It is also within lands claimed by Biigtigong Nishnaabeg, as it asserted exclusive Aboriginal Title. In November 2022, a Community Benefits Agreement was completed between Biigtigong Nishnaabeg and the Company for the development and operation of the Project.

The Town of Marathon is the closest population centre to the Project site. The town has a population of approximately 3,200 and is located about 10 km to the south of the site. The site lies partially within the municipal boundaries of the Town of Marathon, as well as partially within the unorganized townships of Pic, O'Neil and McCoy.

1.22 Capital and Operating Costs

The summary of the Project's capital and operating costs are presented in Table 1.9 and Table 1.10.

Table 1.9: Capital Costs

Capital Costs	Units	Value
Initial Capital[1]	$M	1,112
Pre-production Revenue	$M	(156)
Leased Equipment[2]	$M	(58)
Initial Capital (Adjusted)[1]	**$M**	**898**
LOM Sustaining Capital	$M	424
Total Capital Cost (Adjusted)	**$M**	**1,322**
Closure Costs	$M	72

Note:
[1] Refer to Non-IFRS Financial Measures described in Section 2.
[2] Lease drawdowns net of lease payment during the construction and pre-production periods.

Table 1.10: Operating Costs

Category	Total Costs ($ M)	Unit Cost [1] ($/t milled)
Mining	1,432	11.45
Processing	1,087	8.70
G&A	334	2.67
Concentrate Transport Costs	230	1.84
Treatment & Refining Charges	286	2.29
Royalties	12	0.09
Total Operating Cost	**3,381**	**27.04**

Note:
[1] Refer to Non-IFRS Financial Measures described in Section 2.

1.23 All-In Sustaining Cost Summary

The AISC, which includes closure, reclamation and sustaining capital costs but excludes the impact of the Wheaton PMPA is presented in Table 1.11 and averages US$813/oz PdEq over the LOM.

Table 1.11: AISC Cost Summary

Category	Total Costs
Total Operating Cost	$3,381 M
Closure & Reclamation	$72 M
Sustaining Capital	$424 M
All-in Sustaining Cost (AISC)	**$3,878 M**
All-in Sustaining Cost (AISC)[1]	**US$813/oz PdEq**

Note:
[1]Refer to Non-IFRS Financial Measures in Section 2.

1.24 Execution Plan

The Project execution strategy is currently anticipated to employ an integrated Engineering Procurement and Construction Management and Commissioning team, which has formed the basis of the construction cost estimate. Engineering and procurement are expected to be performed by various contractors given responsibility for specific areas and scope. Throughout the execution and commissioning phases, the Project management team will consist of employees of the Company and consulting firms with experience in implementing similar sized projects. The Project construction period is estimated at 24 months. Estimated construction labour is to average approximately 520 full-time equivalents over the construction period and a peak of approximately 800 full-time equivalent contractors and employees on the Project.

The Project team will manage and execute the engineering, procurement, and construction, provide Project control, staff for start-up and operation, and commission both the mine and process areas. In parallel to construction, an Operational Readiness Plan will be developed. This plan will establish all of the critical operating systems and operating procedures to allow for efficient start-up and ramp-up to commercial production.

1.25 Economic Analysis

The economic analysis is carried out in nominal terms (i.e., without inflation factors) as of the effective date of the Technical Report and in Canadian dollars without any project financing but inclusive of the Wheaton PMPA equipment financing and costs for closure bonding. The economic results are calculated as of the beginning of Q2 Year -3, which corresponds to the start of the pre-production initial capital phase (over 13 quarters), including engineering and procurement, with all prior costs treated as sunk costs but considered for the purposes of taxation calculations. The economic results such as the net present value (NPV) and internal rate of return (IRR) are calculated on an annual basis.

Key results and assumptions used in the FS are summarized in Table 1.12 and Table 1.13.

Table 1.12: Key Economic Input Assumptions

Price Assumptions	Units	Value
Palladium	US$/oz	$1,800
Copper	US$/lb	$3.70
Platinum	US$/oz	$1,000
Gold	US$/oz	$1,800
Silver	US$/oz	$22.50
Exchange Rate	C$/US$	1.35
Diesel Fuel	$/L	1.17
Electricity	$ / kWh	0.07

***Note**: Commodities listed in order of revenues.*

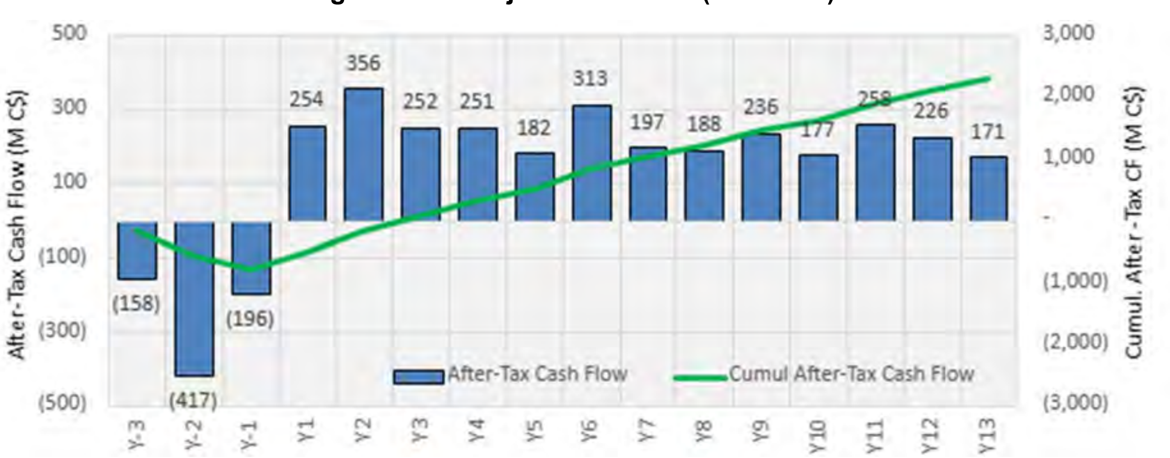
Table 1.13: Economic Analysis

Economic Analysis	Units	Value
Pre-tax Undiscounted Cash Flow	$M	3,387
Pre-tax NPV6%	$M	1,798
Pre-tax IRR	%	31.9
Pre-tax Payback	years	2.0
After-tax Undiscounted Cash Flow	$M	2,285
After-tax NPV6%	$M	1,164
After-tax IRR	%	25.8
After-tax Payback	years	2.3

1.25.1 Project Cash Flow (After Tax)

A summary of the LOM cash flow is presented in Figure 1.12.

Figure 1.12: Project Cash Flow (After-Tax)



1.25.2 Sensitivities

The Project has significant leverage to palladium and copper prices. The after-tax valuation sensitivities for the key metrics are shown below.

Table 1.14: Economic Sensitivity Tables

Palladium Price US$/oz	1,400	1,600	1,700	1,800	1,900	2,000	2,200
NPV6% ($M)	696	930	1,047	**1,164**	1,282	1,400	1,634
Payback (y)	3.3	2.9	2.5	**2.3**	2.2	2.0	1.9
IRR (%)	18.5%	22.3%	24.0%	**25.8%**	27.5%	29.1%	32.3%

Copper Price US$/lb	2.5	3.0	3.5	3.7	3.9	4.5	5.0
NPV6% ($M)	836	972	1,109	**1,164**	1,219	1,386	1,522
Payback (y)	3.0	2.6	2.4	**2.3**	2.2	2.0	1.9
IRR (%)	21.1%	23.1%	25.0%	**25.8%**	26.5%	28.7%	30.4%

After-Tax Results	OPEX Sensitivity				
	+30%	+15%	0%	-15%	-30%
NPV 6% ($M)	1,031	1,085	**1,164**	1,274	1,411
Payback (y)	2.7	2.5	**2.3**	2.1	2.0
IRR (%)	23.4%	24.4%	**25.8%**	27.4%	29.2%

After-Tax Results	CAPEX Sensitivity				
	+30%	+15%	0%	-15%	-30%
NPV 6% ($M)	932	1,048	**1,164**	1,281	1,397
Payback (y)	3.3	3.0	**2.3**	1.9	1.3
IRR (%)	18.4%	21.6%	**25.8%**	31.6%	40.1%

Discount Rate Sensitivity	NPV (After-Tax) ($M)
0%	2,285
5%	1,303
6%	**1,164**
8%	925
10%	731

Foreign Exchange Rate C$:US$	NPV (After-Tax) ($M)
1.25	928
1.30	1,046
1.35	**1,164**
1.40	1,284
1.45	1,403

Fuel Price Sensitivity	NPV (After-Tax) ($M)
0.90	1,197
1.00	1,185
1.10	1,173
1.17	**1,164**
1.30	1,148
1.40	1,136

Power Price Sensitivity ($/kWhr)	NPV (After-Tax) ($M)
0.05	1,207
0.06	1,186
0.07	**1,164**
0.08	1,143
0.09	1,121
0.10	1,100

1.26 Interpretations and Conclusions

The completion of this Feasibility Study update has confirmed the technical and economic viability of the Marathon Project, based on an open pit mining operation with a production rate of approximately 42 Mt/y and an SAB / flotation plant operating at up to 10.1 Mt /y.

1.27 Risks and Opportunities

Table 1.15 outlines the significant risks and uncertainties that could reasonably be expected to affect the reliability of confidence in the projected economic outcome for the Feasibility Study update. Table 1.16 outlines the significant opportunities that could reasonably be expected to have a positive impact on improving the Project economics in the future.

Table 1.15: Risks

Risk Category	Description	Potential Impact[1]
Mineral Resource Estimate	Until the operation commences, and operational grade reconciliation is undertaken, there is some level of uncertainty related to the predictability of the Mineral Resource estimate	▪ Reduction in Mineral Resources available for conversion to Mineral Reserves
Environment Assessment Conditions and Permitting	There is uncertainty associated with the precise timing for the approval of permits required to build, and operate the Project as designed and there are EA conditions which are required to be completed prior to construction commencing	▪ A delay to the start date for project construction ▪ A delay to the start of operations or future operations continuity
Project Financing	There is uncertainty with the Company securing timely and/or adequate Project financing	▪ Delay (short-term or long-term) in the start date of the Project
COVID-19	The resurgence, or unexpected impact of the COVID-19 pandemic is uncertain	▪ Reduced efficiency of the construction workforce or delayed construction schedule
Construction Costs	Construction costs are based on the current designs; final designs and construction methodology may change	▪ Increased construction costs
Operating Costs	Operating efficiency, operating time, productivity, and consumables are assumed based on provisional budgetary quotations along with similar benchmark operations; any reduction in operating efficiency or increased consumables will increase operating costs	▪ Increased operating costs
Processing Plant Metallurgical Recovery	The plant metallurgical recovery models are based on laboratory scale testing. Actual metallurgical recovery and mass pull of the operating plant may be different to the predicted model	▪ Less payable metal or increase in plant operating costs
Labour and Skilled Resources	There is a national and international shortage of unskilled, skilled, and technical expertise in mining.	▪ Increased labour costs ▪ Increase in remote employees with an increase in camp requirements

Metal Prices and Exchange Rates	For each payable element and the exchange rate, the economic assumptions are sensitive (both positively and negatively impacted) by metal prices and changes in C$/US$ exchange rates	▪ Variability in economic results with changing metal prices. ▪ Strengthening of the C$ as compared to the US$ will negatively impact economic results
***Note**:* [1] *This is not intended to outline all potential impacts, simply the impacts that could reasonably be expected to occur in the event the risk item results in an impact.*		

Table 1.16: Opportunities

Opportunity	Description	Potential Impact[1]
Mineral Resource Estimate	Unrealized local variability due to grade interpolation smoothing may lead to opportunities to extract somewhat more metal from fewer tonnes	▪ Higher value per tonne of ore.
Plant Throughput	2022 metallurgical tests indicated variability in material hardness; the process design criteria has allowed for the higher than average material hardness	▪ Decreased material hardness would support an increase in throughput, de-risking the production profile, and an opportunity to advance metal production and cash flow
Exploration Success on the Property	With the conversion of the Property resources to reserves or new exploration success, would be expected to increase material feed to the plant and increase either mine life beyond the 13 years or allow for increased throughput over the same operating life.	▪ Increased reserves would increase production which would imply increased value and cash flow. ▪ Increased mine life would extend employment opportunities and increase operating cash flow
Trolley Assist ("TA") or the 'next generation' powered mining fleet	The concept of TA was evaluated with equipment suppliers / dealers but was not included in the Base Case operating design. TA would conceptually increase up-ramp truck speed and allow for additional tonnage (with a reduced cycle time) or reduce capital requirements. Mining fleet manufactures are testing battery and fuel cell mining equipment with viable options being marketed within the life of mine of the operation.	▪ Improved operating efficiency and lower mine operating costs ▪ Reduction in the generation of GHG from operations (reduced diesel consumption).
Automation of the mining fleet	With the truck fleet being relatively small, autonomous haulage is not expected to be viable; however, the automation of drills and dozers would improve operating efficiency or reduce operating costs.	▪ Reduced operating costs on a $/t basis
***Note**:* [1] *This is not intended to outline all potential benefits but those that could reasonably be expected to occur or possibly realized.*		

1.28 Recommendations

- With the demonstrated and positive economic analysis, progress to the next phase of Project development including project financing, advancing required permits to allow for the Property to be developed through construction and into production. The total cost of the next phase of the project up to commercial production is estimated at $1,112 M.
- Advance on the EA conditions as outlined by the Federal and Provincial agencies per the positive EA decision report and progress the permitting activities to allow for construction to start as soon as financing is available.
- Progress the study for the power line and connection requirements for the increased plant throughput and the future electrification of the operation.
- Continue with implementation of an Independent Tailing Review Panel for the oversight during the Tailing Storage Facility life cycle.
- Execute off-take agreements with smelters.

2. INTRODUCTION

2.1 Introduction

The Technical Report for the Marathon Palladium-Copper Project (the "Marathon Project" or "Project") located just outside the Town of Marathon on the shores of Lake Superior in Ontario, Canada was prepared by G Mining Services Inc. ("GMS") along with contributions from Generation Mining Limited (the "Company" or "Gen Mining"), Wood Canada Limited ("Wood"), JDS Mining Ltd. ("JDS"), Knight Piésold Ltd. ("KP") and P&E Mining Consultants Inc. ("P&E").

Gen Mining currently owns a 100% interest in the Marathon Project. The Project is managed and operated by Gen Mining's 100%-owned subsidiary Generation PGM Inc. ("Gen PGM").

Gen Mining commissioned the aforementioned consultants to prepare and issue this Technical Report in accordance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' current "Standards of Disclosure for Mineral Projects" under the provisions of National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101 CP and Form 43-101F1.

The objective of this Technical Report is to provide the results of an updated feasibility study ("FS") for the development of an open pit mine at the Marathon Project, including processing facilities and related infrastructures. The initial FS was issued on March 21, 2021 (effective date March 3, 2021) in a Technical Report titled "*Feasibility Study Marathon Palladium & Copper Project Ontario, Canada*" prepared by GMS. This Technical Report summarizes the results of the updated economics of the Project, and also presents an updated Mineral Resource and Mineral Reserves Estimates for the Marathon Property (the "Marathon Property" or "Property").

2.2 Scope and Terms of Reference

The scope of this Technical Report and FS includes the geology and Mineral Resources of the Marathon Property, including the following deposits: Marathon, Geordie, and Sally. The Mineral Reserves, mining, infrastructure, processing, and financial analysis sections of this Technical Report considers only the Marathon deposit (the "Marathon Deposit").

The monetary units are in Canadian dollars, unless otherwise stated.

Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards).

Estimates of Mineral Resources and Mineral Reserves follow industry best practices as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM, 2019). Classification of Mineral Resources and Mineral Reserves conform to CIM Definition Standards (CIM, 2014).

2.3 Non-IFRS Measures

The Company has included certain non-IFRS financial measures in this Technical Report such as initial capital cost, cash operating costs and AISC per palladium equivalent ounce (“PdEq”), unit operating costs, EBITDA, and Free Cash Flow, which are not measures recognized under IFRS and do not have a standardized meaning prescribed by IFRS. For the reconciliation of cash costs and AISC, on both a per tonne and PdEq basis, refer to the tables set forth in the Capital and Operating Cost Summary in Sections 1, 21 and 22. Non-IFRS measures do not have any standardized meaning prescribed under IFRS, and therefore, they may not be comparable to similar measures employed by other companies. The data presented is intended to provide additional information and should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS. These measures do not have any standardized meaning prescribed under IFRS, and therefore may not be comparable to other issuers.

- Initial Capital includes all costs incurred from the as of the Effective Date of the Technical Report (excluding historical sunk costs) until the point where commercial production is achieved, including expenses related to engineering, equipment purchases and installation, process plant and mine infrastructure construction, and any other costs associated with putting the Project into operations.
- Initial Capital (Adjusted) includes all costs mentioned above in addition to adjustments for pre-commercial production revenue and equipment financing (net of payments, interest and fees incurred prior to commercial production).
- Operating Costs include mining, processing, general and administrative and other, concentrate transportation costs, treatment and refining charges, and royalties.
- AISC include Operating Costs, closure, and reclamation, and sustaining capital. For the full reconciliation of cash costs and AISC.
- LOM Average AISC includes LOM AISC divided by LOM PdEq.
- LOM Average Operating Cost includes LOM Operating Costs divided by LOM PdEq.
- Free Cash Flow includes total revenue less Operating Costs, working capital adjustments, equipment financing, initial capital, sustaining capital and closure costs
- Palladium Equivalence ounces uses the formula PdEq oz = Pd oz +(Cu lb x 3.7 US$/lb + Pt oz x US$1,000/oz + Au oz x US$1,800/oz + Ag oz x US$22.5/oz) / US$1,800 Pd/oz

2.4 Source of Information and Data

Previous Reports issued on the Marathon Project include the following:

- G Mining Services Inc. Technical Report, Feasibility Study of the Marathon Palladium & Copper Project, Ontario, Canada for Generation Mining Ltd., effective date March 3, 2021.
- P&E Mining Consultants Inc.: (Amended) Technical Report, Updated Mineral Resource Estimate and Preliminary Economic Assessment of the Marathon Deposit Thunder Bay Mining District, Northwestern Ontario, Canada for Generation Mining Ltd., effective date January 6, 2020.
- Nordmin Engineering Ltd.: Marathon PGM-Cu Feasibility Study (Draft Report), document dated March 14, 2014 for Stillwater Canada Inc.
- Micon International Limited: Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada, dated January 8, 2010.
- Micon International Limited: Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada, dated February 2, 2009.
- P&E Mining Consultants Inc., 2006b: Technical Report and Preliminary Economic Assessment of the Marathon PGM-Cu Property, Marathon Area, Thunder Bay Mining district, Northwestern Ontario, Canada, June 30, 2006, revised July 8, 2006.
- P&E Mining Consultants Inc., 2006a: Technical Report and Resource Estimate on the Marathon PGM-Cu Property Marathon Area, Thunder Bay Mining District, Northwestern Ontario, Canada for Marathon PGM Corporation, dated March 24, 2006.
- Documents listed in Section 27 - References

2.5 Technical Report Responsibilities

The Technical Report and FS responsibilities of the engineering consultants are as follows:

Table 2.1: Consultants Used and Area of Responsibility

Consultant Company	Area of Responsibility
GMS	Overall integration, Mineral Reserve Estimate, mining methods, concentrate logistics, economic analysis, operating costs pertaining to mining and G&A
JDS	Infrastructure, and power capital cost estimates, and project execution plan and schedule
LQ Consulting and Management Inc. (“LQ”)	Project execution plan and schedule

Wood	Recovery methods, processing plant capital and operating cost
KP	Tailings Storage Facility, water balance, geotechnical studies (mine rock storage piles, open pit and local infrastructure and foundations)
P&E	Property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, and Mineral Resource Estimates and adjacent properties
Haggarty Technical Services ("HTS")	Metallurgical testing, recovery methods, processing plant design and operating cost
WSP	Environment, permitting and communities and social aspects

2.6 Summary of Qualified Persons

The authors and co-authors of each section of the Technical Report, who acting as a QP as defined by NI 43-101, take responsibility for those sections of the Technical Report is summarized in Table 2.2 as outline in detail in Section 28 "Certificate of Author" attached to this Technical Report.

The overall effective date of this Technical Report is December 31, 2022.

Table 2.2: Qualified Persons (QPs)

Name of Qualified Person	Company	Technical Report Section[1]
Mr. Carl Michaud, ing.	GMS	1, 2, 3, 19, 21, 22, 25, 26
Mr. Alexandre Dorval, P.Eng.	GMS	15, 16, 21, 22, 25, 26
Mr. Jean-Francois Maille, P.Eng.	JDS	18, 21, 24, 25, 26
Mr. Craig Hall, P.Eng.	KP	18, 20, 21, 25, 26
Mr. Eugene Puritch, P.Eng., FEC, CET	P&E	1,14,23,25,26
Ms. Jarita Barry, P.Geo.	P&E	1,11,12,25,26
Mr. Fred Brown, P.Geo.	P&E	1,14,25,26
Mr. David Burga, P.Geo.	P&E	1,9,10,12,25,26
Mr. William Stone, PhD, P.Geo.	P&E	4,5,6,7,8,10,23,25,26
Mr. Ben Bissonnette, P.Eng.	Wood	1.11, 1.12, 13, 17, 21.8.2, 25.3, 26.10, 26.11, 27
Mr. Sumit Nair, P.Eng.	Wood	1.22, 18.1, 18.2.6, 18.2.6.1, 18.2.6.2, 18.3.1, 18.3.2, 18.3.3, 18.3.4, 18.3.5, 18.4.3, 18.4.4, 18.4.5, 21.1, 21.5, 21.6.6, 26.10

[1] Co-authored sections may be listed multiple times. Specific subsections being QP'd can be found on the QP certificates.

2.7 Site Visit

The following independent QPs as defined by NI 43-101 visited the site as described below:

Table 2.3: QP Site Visit Dates

Name of Qualified Person	Consultant Company	Site Visit Date
Carl Michaud, ing.	GMS	January 2023
Craig Hall, P.Eng	KP	April 2011 and March 2012
David Burga, P.Geo.	P&E	April 2012
Eugene Puritch, P.Eng, FEC, CET	P&E	Various visits between 2005 and 2010
Jean-Francois Maille, P.Eng	JDS	October 2022

2.8 Units of Measure, Abbreviations and Nomenclature

The units of measure presented in this Technical Report, unless noted otherwise, are in the metric system.

A list of the main abbreviations and terms used throughout this Report is presented in Table 2.4.

Table 2.4: List of Main Abbreviations

Abbreviations	Full Description
Ag	Silver
As	Arsenic
Au	Gold
Ba	Barium
Bi	Bismuth
°C	Degrees Celsius
C	Carbon
Ce	Cerium
cm	Centimetre(s)
CAD or C$	Canadian Dollar
Co	Cobalt
Cr	Chromium
CRM	Certified Reference Material
Cu	Copper
dB	Decibel
dmt	Dry Metric Tonne
°F	Degrees Fahrenheit
F	Fluorine
ft	Feet
FA	Fire Assay
Fe	Iron
FEL	Front End Loader
FS	Feasibility Study
G	Giga – (000,000,000)
g	Gram(s)
gpt or g/t	Grams per tonne
g/L	Gram(s) per litre
G&A	General & Administration
gpm	Gallons per minute (US)
GPS	Global Positioning System
ha	Hectares
Hg	Mercury
h or hr	Hour
h/d or hr/d	Hours per day
h/y or hr/y	Hours per year

Abbreviations	Full Description
H/wk or hr/wk	Hours per week
hp	Horsepower
HQ	HQ sized core (63.5 mm diameter)
Hz	Hertz
IRR	Internal Rate of Return
IP	Induced Polarization
ISO	International Organization for Standardization
Ir	Iridium
k	Thousand(s)
k	Kilo(s) (000's)
kg	Kilogram(s)
kg/t	Kilograms per tonne
kV	Kilovolts
km	Kilometer(s)
km^2	Square Kilometre(s)
km/h	Kilometer per hour
kPa	Kilopascal
kV	Kilovolts
kW	Kilowatts
kWh	Kilowatts per hour
LOM	Life of Mine
L or l	Litre(s)
µm	Micron(s)
M	Mega or Millions (000,000's)
masl	Metres above sea level
m	Metre(s)
m/min	Metre(s) per minute
m/s	Metre(s) per second
m^2	Square metre(s)
m^3	Cubic metre(s)
Mg	Magnesium
mg	Milligram(s)
mg/L	Milligram(s) per litre
Mi	Hoek-Brown material constant
mm	Millimeter(s)
ml	Milliliter(s)
min	Minute(s)

Abbreviations	Full Description
Mn	Manganese
Mo	Molybdenum
MPa	Megapascal
Mt	Million tonnes
Mtpd	Metric tonnes per day
Mtpy	Metric tonnes per year
MVA	Megavolt-ampere
MW	Megawatt
Nb	Niobium
Ni	Nickel
NAG	Non Acid Generating or non Potentially Acid generating
NPI	Net Profit Interest
NPV	Net Present Value
NSR	Net Smelter Return
NQ	Drill Core Diameter (47.6 mm)
Ø	Diameter
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
OSA	On-stream analyzer
oz	Troy Ounce (31.10348 grams)
PAG	Potentially Acid Generating
PEA	Preliminary Economic Assessment
Pb	Lead
Pd	Palladium
PFS	Pre-feasibility Study
PGM	Platinum Group Metals
PLC	Programmable Logic Controller
ppb	Parts per billion
ppm	Parts per million
psi	Pounds per square inch
Pt	Platinum
PV	Present Value
Rb	Rubidium
RC	Reverse Circulation
Rh	Rhodium
RoM or ROM	Run-of-mine

Abbreviations	Full Description
rpm	Revolutions per minute
S	Sulphur
SAG	Semi-Autogenous Grinding
Sb	Antimony
Se	Selenium
Sec	Second(s) (time)
Si	Silicon
Sm	Samarium
Sn	Tin
t	Tonne(s) (1,000 kg) (metric ton)
t/y or tpy	Tonne(s) per year
t/d or tpd	Tonne(s) per day
t/h or tph	Tonne(s) per hour
t/m^3	Tonne(s)s per cubic metre
Te	Tellurium
Th	Thorium
UCS	Unconfined Compressive Strength
USD or US$	United States Dollar
V	Vanadium
V	Volt
VAT	Value Added Tax
wk	Week
wmt	Wet Metric Tonne
XRF	X-ray Fluorescence
yr	Year
Y	Yttrium
Yb	Ytterbium
Zn	Zinc
Zr	Zirconium

3. RELIANCE ON OTHER EXPERTS

The Technical Report has been compiled by the QPs based on information prepared and/or reviewed by the independent QPs. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the QPs at the time of the preparation of the Technical Report.
- Assumptions, conditions, and qualifications as set forth in this Technical Report.
- Data, reports, and other information supplied by the Company, including, Jean-Paul Deco, Generation Mining, Manager Business Development and Concentrate Marketing with respect to concentrate marketing, and other third-party sources that has been vetted and verified.

The QPs of this Technical Report believe that the basic assumptions contained in the information indicated above are factual and accurate and that the interpretations are reasonable. The QPs of this Technical Report have, to the extent applicable, relied on this data and have no reason to believe that any material facts have been withheld. The QPs of this Technical Report have taken all appropriate steps, in their professional judgement, to ensure that the work, information, or advice from the above indicated information is sound and the QPs do not disclaim any responsibility for this Technical Report.

In preparing the Technical Report, the QPs that prepared the following section have relied upon certain work, opinions, and statements of experts. The QPs consider the reliance on other experts as described herein, as being reasonable based on their knowledge, experience, and qualifications. The QPs that authored this Technical Report disclaim responsibility for such expert report content:

- Section 22: Liam Fitzgerald, Partner, Tax, PwC, worked on the tax section of the FS Model reviewed by PricewaterhouseCoopers LLP for the Company based on a draft of the FS Model for the Marathon Project provided by GMS for inclusion in the financial analysis and the tax narrative for the Technical Report. Mr. Fitzgerald's information is used in support of the financial analysis.

The results and opinions expressed in this Technical Report are conditional upon the information provided by the Experts listed as being current, accurate and complete as of the effective date of the Technical Report.

The authors wish to emphasize that they are QPs only in respect of the areas in this Technical Report identified in their "Certificates of Qualified Persons" submitted with this Technical Report to the Canadian Securities Administrators.

Except for the purposes contemplated under provincial securities laws, any other use of this Technical Report by any third party is at the party's sole risk.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

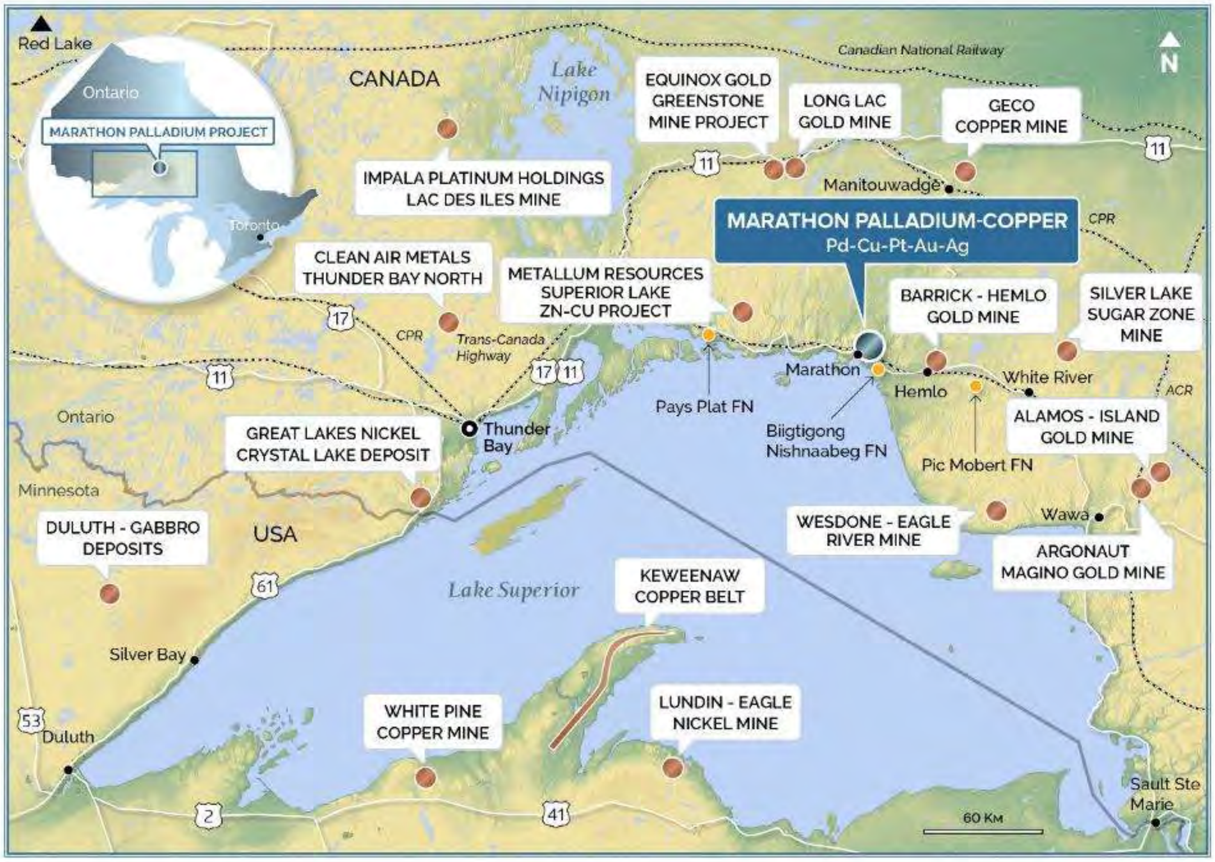
The Marathon Property is located approximately 10 km north of the Town of Marathon, Ontario, which is situated adjacent to the Trans-Canada Highway No. 17 on the northeast shore of Lake Superior. Marathon has a population of 3,138 (2021 StatsCan census). The Property is approximately 300 km east of Thunder Bay, along Highway 17 and 400 km northwest of Sault Ste. Marie along the same Highway 17.

Local access to the Property is by a gravel road from Highway 17 (Figure 4.1 and Figure 4.2), which lies just north of the Town of Marathon and immediately south of the Property. The centre of the proposed Project footprint sits at approximately 48°45’N Latitude, 86°19’W Longitude. The primary industry supporting the Town of Marathon is mining.

Figure 4.1 : Regional Location Map

Source: Marathon PGM Corp. (2006).

Figure 4.2: Regional Mining Activity Map



Source: *Generation Mining Limited (2023).*

4.2 Project Ownership

In 2010, the Property was acquired by Stillwater Mining Company ("Stillwater") from Marathon PGM Corporation ("Marathon PGM") (TSX: MAR) for US$118 million. At that time, Stillwater was a palladium and platinum mining company with headquarters located in Littleton, Colorado, USA. Stillwater mined PGMs from the Stillwater igneous complex in south central Montana known as the J-M Reef and recovered metals from spent catalytic converters. In 2017, Stillwater was acquired by Sibanye Gold Limited (NYSE: SBSW) for US$2.2 billion.

On July 10, 2019, Gen Mining, through Generation PGM Inc. ("Gen PGM"), completed the acquisition of a 51% initial interest in the Marathon Property and entered into a joint venture agreement with Stillwater Canada Inc. ("Stillwater Canada"). Gen Mining paid $3 million in cash and issued 11,053,795 common shares of Gen Mining at a deemed price of $0.2714 per common share (totalling $3 million), for total consideration of $6 million. Pursuant to the joint venture agreement, Gen Mining had the right to increase its interest in the Property to 80% by sole funding $10 million in exploration, evaluation and development expenditures and preparing a preliminary economic assessment ("PEA") within four years.

On February 19, 2020, Gen Mining filed a PEA and by November 2020 had incurred $10 million in exploration and evaluation expenditures fulfilling the Ownership Increase Right. On November 27, 2020, Gen Mining increased its ownership interest to 80% in the Marathon Property. On December 14, 2020, Stillwater Canada elected to forego its proportionate share of joint venture funding and dilute pursuant to the provisions of the joint venture agreement. On July 21, 2021, Stillwater Canada elected to not exercise its ownership increase right to expand its ownership in the Marathon Property to 51%.

On December 8, 2021, Gen Mining entered into an acquisition agreement with Stillwater Canada pursuant to which Gen Mining would acquire Stillwater Canada's remaining 16.5% interest in the Marathon Project. On January 26, 2022, Gen Mining completed the acquisition whereby Gen Mining issued 21,759,332 common shares of the Company to Stillwater Canada. The Company now holds 100% of the Marathon Property, and the joint venture agreement dated July 10, 2019, between Stillwater Canada and the Company has been terminated in accordance with its terms.

As of the effective date of this Technical Report, Gen Mining is the operator of the Project.

4.3 Property Description and Tenure

The original Marathon Property held by Stillwater Canada from 2010 to 2019 has since been enlarged by Gen Mining through the periodic staking of unpatented mining claims. Gen Mining staked an additional 215 claim blocks totalling 4,558 ha during the summer of 2019. This staking increased the land position to 46 leases and 933 claims for a total of 21,883 ha (218.83 km^2) (Figure 4.3).

The 46 leases are located in Seeley Lake, Pic, O'Neill, Grain and Martinet Lake Townships and total 4,810.2 ha. Claim information, recorded dates and expiry dates are listed in the Appendix A.

All claims have been renewed to their respective anniversary dates ranging from 2026 to 2028. Assessment work by the Company will have to be applied by these dates to retain the claims in good standing. The claims are registered in the name of Gen PGM, a subsidiary of Gen Mining.

In 2014, Stillwater Canada initiated the conversion of mineral claims comprising surveyed area CLM509 to a mining lease with surface rights. The survey area CLM509 is west of the Marathon Deposit. The timing for the conversion of CLM509 extended through the implementation of the Ministry of Energy, Northern Development and Mines ("MENDM") new MLAS system in 2017. The lease was granted on November 25, 2020 (lease number 110068). However, since the conversion was initiated with legacy claims, there is no connection to current claim cells in the new MLAS system referencing which cells comprise the new lease 110068.

4.3.1 Ontario Mineral Tenure

The Ontario claims information presented in this section is valid as of the effective date of this Technical Report. Crown lands are available to licensed prospectors for the purpose of mineral exploration. A licensed prospector must first stake a mining claim to gain the exclusive right to explore on Crown land. Claim staking is governed by the Ontario Mining Act and is administered through the Provincial Mining Recorder and Mining Lands offices of the MENDM.

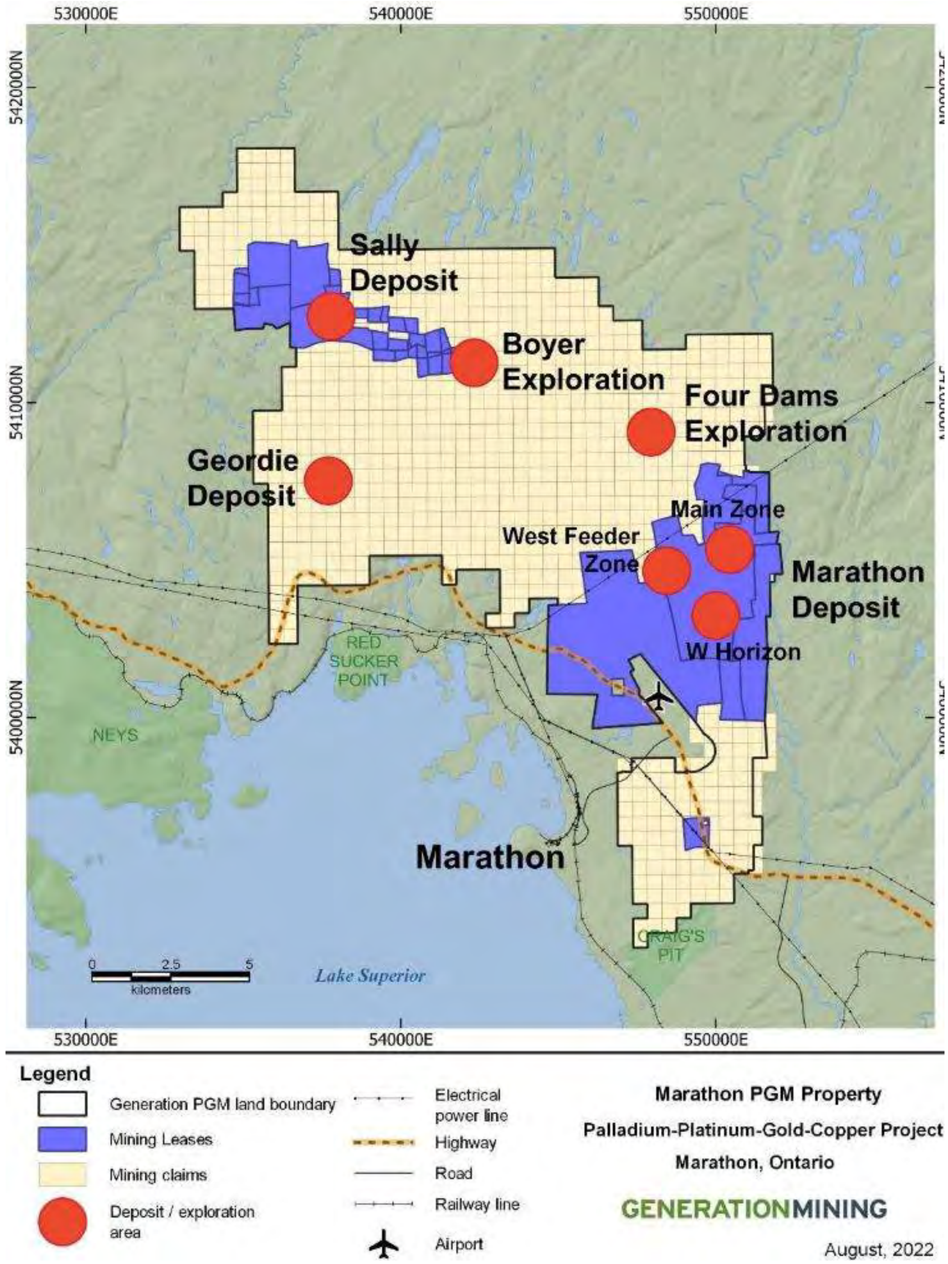
Mining claims can be staked either in a single unit or in a block consisting of several single units. In un-surveyed territory, a single unit claim is laid out to form a 16 ha (40-acre) square with boundary lines running 400 m (1,320 ft) astronomic north, south, east, and west. Multiples of single units, up to a maximum of 16 units (256 ha), may be staked with only a perimeter boundary as one block claim.

Upon completion of staking, a recording application form is filed with payment to the Provincial Recording Office. All claims are liable for inspection at any time by the MENDM. A claim remains valid as long as the claim holder properly completes and files the assessment work as required by the Mining Act and the Minister approves the assessment work. A claim holder is not required to complete any assessment work within the first year of recording a mining claim. To keep an unpatented mining claim current, the mining claim holder must perform $400 worth of approved assessment work per mining claim unit, per year.

Immediately following the initial staking date, the claim holder has two years to file one year's worth of assessment work. Claims are forfeited if the assessment work is not completed.

A claimholder may prospect or carry out mineral exploration on the land under the claim. However, the land covered by these claims must be converted to leases before any development work or mining can be performed. Mining leases are issued for 21-year terms and may be renewed for further 21-year periods. Leases can be issued for surface and mining rights, mining rights only or surface rights only. Once issued, the lessee pays an annual rent to the province. Furthermore, prior to bringing a mine into production, the lessee must comply with all applicable federal and provincial legislation.

Figure 4.3: Marathon Deposit Claim Location Map

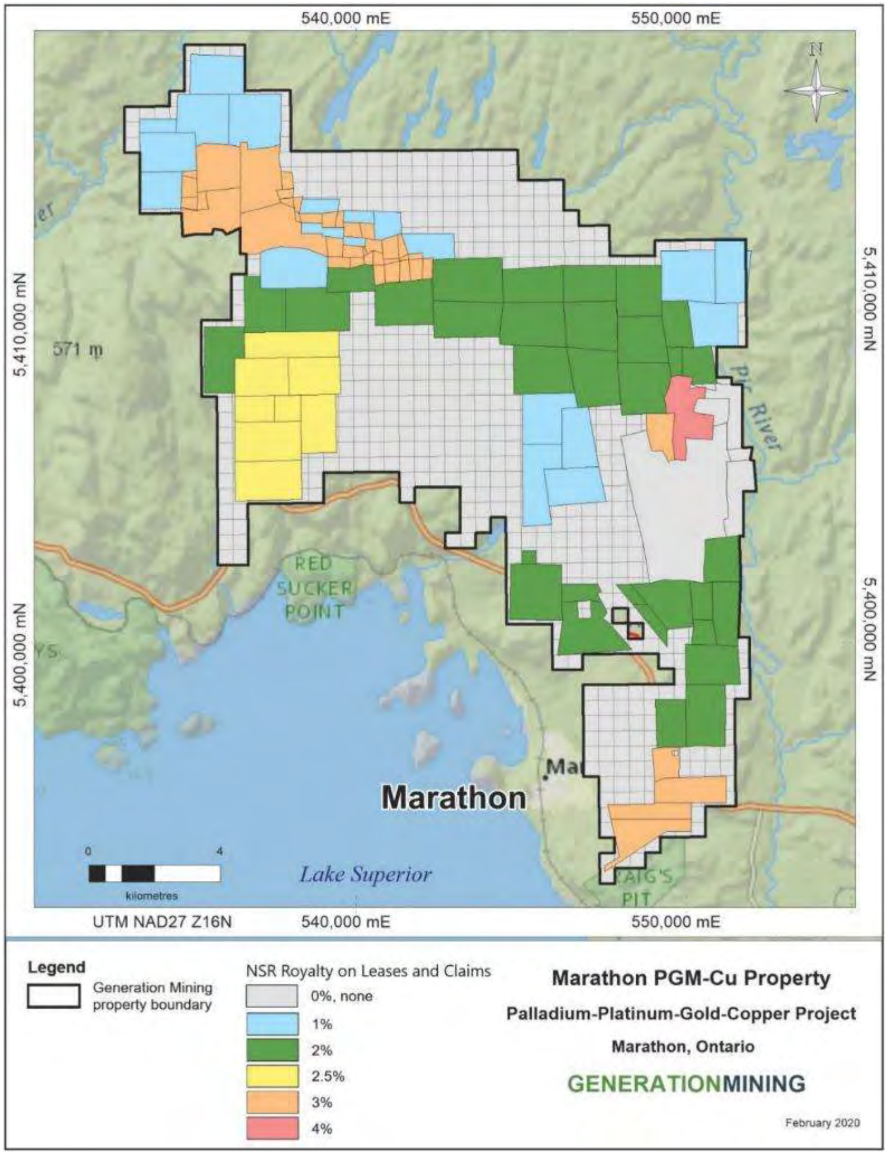


Source: *Gen Mining (2022).*

4.4 Royalties

The Property is subject to NSR royalties ranging from 1% to 4% (Figure 4.4). In particular, the top northern extent of the Marathon Deposit (specifically on the North pit) is subject to a NSR royalty of 4%. A complete summary of the encumbrances can be found in Table 4.1.

Figure 4.4: Summary Royalty ("NSR") Map



Source: *Gen Mining (2019).*

Table 4.1: Royalties and Agreements

Party	Date	NSR Value	Details
Marathon Project Area – Royalty Agreements			
Fenwick/ Leishman	Aug. 16, 2005	3%	Royalty in favour of Kenneth Fenwick and Don Leishman on mining claims TB 1247007, TB 1247010-11. Gen Mining has the right at any time to acquire up to one-third of the royalty (up to an aggregate of 1% of the royalty) upon a payment of $500,000 for every 0.5% of the royalty purchased.
Seafield	Nov. 2, 2004	2%	Royalty in favour of Seafield Resources Ltd. on mining claim TB 1205330. Gen Mining has the right at any time to acquire up to half of the royalty (up to an aggregate of 1% of the royalty) upon a payment of $1,000,000.
Dunlop	Mar. 21, 2006	3%	In favour of W. Bruce Dunlop on mining claims TB 104122 and TB 104118-104121 inclusive. Gen Mining has the right at any time to acquire up to one half of the royalty (an aggregate of 1.5%) upon payment of $500,000 for every 0.5% of the royalty purchased.
Gionet	May 2007	1%	With a right of first refusal on the sale of the royalty in favour of Brian D. Gionet and Michael Dorval on mining claims 4208442 and 3014935.
Michano/ Gionet	Apr. 21, 2005	2%	In favour of Michano/Gionet on mining claims TB 3012177, TB 3006862, TB 3012173, TB 3019790, TB 4204047-49. Gen Mining has the right at any time to acquire up to one half of the royalty (an aggregate of 1%) upon payment of $1,000,000.
Benton	Mar. 25, 2009	4% and $0.05/t waste mgmt fee	Certain conditions of which were modified by the Benton Resources/Stillwater Mining Co. Agreement dated December 16, 2010 - 2% NSR and $0.05/t waste manage fee in favour of Teck Resources on mining claims 1240016, TB101224-25, TB101578-81, TB101583, TB103572-75, TB103583-84, TB106983, TB103657 and TB107641.
Michano/ Gionet/ Dorval	Jul. 12, 2011	2%	On mining claims TB 4246277, TB 4242127, and TB 4246285. Gen Mining has the right at any time to acquire up to one half of the royalty (an aggregate of 1%) upon payment of $1,000,000.
Michano/ Gionet	Jul. 12, 2011	2%	On mining claims TB 4246283-84. Gen Mining has the right at any time to acquire up to one half of the royalty (an aggregate of 1%) upon payment of $1,000,000.
Yozipovic	Nov. 14, 2011	2%	On mining claim TB3006106. Gen Mining has the right at any time to acquire the 2% NSR from the vendor for a fee of $1,000,000.
Sally Project Area – Royalty Agreements			
Benton/ Gold Royalties Corp.	Dec. 13, 2011	1%	Pursuant to the Benton Agreement dated March 25, 2009, certain conditions of which were modified by the Benton Resources/Stillwater Mining Co. Agreement dated December 16, 2010 - 1% NSR in favour of Stephan Stares on mining leases CLM 121-124, TB101845-47, TB101849-50, TB101864-66, TB101869-71, TB101891-905, TB101910, TB101915-17, TB101924, TB108223-24 and mining claims 4204476-78, 4207280-83, 4209025-26, 1240550-55, 1240548-49, 1240017-

Party	Date	NSR Value	Details
			19, 4207863, 4207856-59, 4207860-61, 4203971-72, 1245401, and 1246640-43,. Note: In December 2011, Stares sold one half of the subject royalty (an aggregate of 0.5%), excluding TB120016, to Gold Royalties Corp. and one-half to Kalt.
Benton "Newmont Royalty"	Dec. 16, 2010	2%	In favour of Newmont (Franco-Nevada) on mining leases CLM 121-124, TB101845-47, TB101849, TB101850, TB101864-66, TB101869-71, TB101891-905, TB101910, TB101915-17, TB101924, and TB108223-24. Note: an annual report to Franco-Nevada is required on the Par Lake property.
Benton	Dec. 16, 2010	1%	In favour of Benton Resources on mining leases CLM 121-124, TB101845-47, TB101849-50, TB101864-66, TB101869-70, TB101871, TB101845, TB101891-905, TB101910, TB101915-17, TB101924, TB108223-24 and mining claims 4204476-78, 4207280-81, 4207282-83, 4209025-26, 1240550-55, 1240548-49, 1240017-19,4207863, 4207856-61, 4203971-72, 1245401, and 1246640-43. Royalty is only payable commencing on and from the date that a minimum aggregate combined total of 2,500,000 ounces of platinum, palladium and gold is produced from the above-mentioned leases and claims.
Geordie Project Area			
Wahl	Jul. 8, 2008	2%	In favour of Rudy Wahl, on mining claims 3015131-33. Gen Mining has the right at any time to acquire up to one half of the royalty (an aggregate of 1%) after commencement of commercial production and payment of $1,000,000.
Discovery	Mar. 3, 2008	2.5%	Pursuant to underlying Agreements of record that remained in effect subsequent the acquisition of Discovery PGM Corp. by a predecessor of Stillwater Canada, the Geordie Lake property is encumbered by a 2.5% NSR in favor of Superior Prospects Inc. and Melvin Joa (in aggregate) on mining claims 1184283, 1184297, 1209682-84, and 1237697-99.
Gryphon/ L.E.H. Ventures	Jun. 3, 1999	0%	Gryphon Metals Corp. retains the right upon the completion and presentation of a definitive FS on the Geordie Lake Property to back into a 12.5% interest on the property by paying Stillwater Canada a total of 31.25% of all exploration and development costs incurred on the property to that point.

4.5 Indigenous Community Participation

In connection with the certain financial commitments to an Indigenous community, and at the request of that community, a royalty agreement has been executed and registered on title to the Property. The terms of this agreement are confidential but the anticipated costs of this agreement are included in general and administration costs as described in Section 21 – Capital and Operating Costs.

4.6 Comment of Property Description and Location

To the extent known to the QP, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that have not been discussed in this Technical Report. To the extent known to the QP, permits have been acquired or are reasonably expected to be able to be acquired to conduct the work proposed for the Property. Details on permitting can be found in Section 20 – Environmental Studies, Permitting and Social or Community Impact.

The Environmental Assessment ("EA") for the Project was approved on November 30, 2022 in accordance with the Canadian Environmental Assessment Act ("CEAA", 2012) and Ontario's Environmental Assessment Act ("EA Act") through a Joint Review Panel ("JRP") pursuant to the Canada-Ontario Agreement on Environmental Assessment Cooperation (2004).

As of the effective date of this Technical Report, the Project is in the process of obtaining various Federal, Provincial and municipal permits, approvals and licenses as required to construct and operate the Project.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Access and Infrastructure

The Property is located at latitude 48°45’ N and longitude 86°19' W. The Property is accessed by paved and gravel roads, approximately 10 km north of the Town of Marathon (Figure 5.1). Regional infrastructure is considered very good with the Trans-Canada highway, Canadian Pacific railway (“CPR”) and a municipal airport all in close proximity to the Marathon Property.

The local site access will be developed off the Camp-19 Road during the construction period to minimize water impacts.

Figure 5.1 : Access Road Photograph

Source: Gen Mining (2019).

There are sufficient surface rights available for mining operations, inclusive of the processing plant and associated tailings and waste rock storage facilities.

5.2 Climate

The Project lies in the sub-arctic region. The Property’s climate is typical of northern areas within the Canadian Shield with long winters and short but warm to hot summers. However, Marathon experiences cooler summers and warmer winters compared to other more remote northerly communities in northwestern Ontario due to its proximity to Lake Superior. The annual average temperature is 1°C with the highest average monthly temperature of 15°C in August and lowest in January of -15°C. Extreme minimum temperatures at the Marathon Airport ranged from -41.7°C to +2.1°C and maximum temperatures ranged from +2.6°C to +28.5°C (Environment Canada). The average annual precipitation for the Pukaskwa Station (located 15 km south of the Marathon Airport) is 759 mm, which compares well to the Marathon Airport data for 1988-1999 (840 mm).

Operations in this climate typically require covered buildings. Adverse weather conditions are rarely severe enough to halt an open pit operation for any more than a few hours during low-traction surface conditions during winter storms. Winter conditions are not expected to be noteworthy nor to have a significant impact on annual production.

5.3 Local Resources

Thunder Bay is the largest regional city and is largely the hub for the communities north of Lake Superior. Thunder Bay is approximately 300 km west of the Marathon Property with good connections via the Trans-Canada highway.

The Town of Marathon has a population of 3,138 (2021 census) and is the closest municipality to the Project. There are several active mines in the general area and therefore some local mining services are available in the Town of Marathon. The Marathon airport is located immediately north of the Town of Marathon and runs adjacent to Highway 17, near the southwest corner of the Property. Marathon Municipal Airport (CYSP) operates as a Registered Airport (Aerodrome class) under the Canadian Aviation Regulations (CARs; Subsection 302). The airport is used by private aircraft owners and commercial helicopter companies. As of the effective date of this Technical Report, no commercial flight service is available.

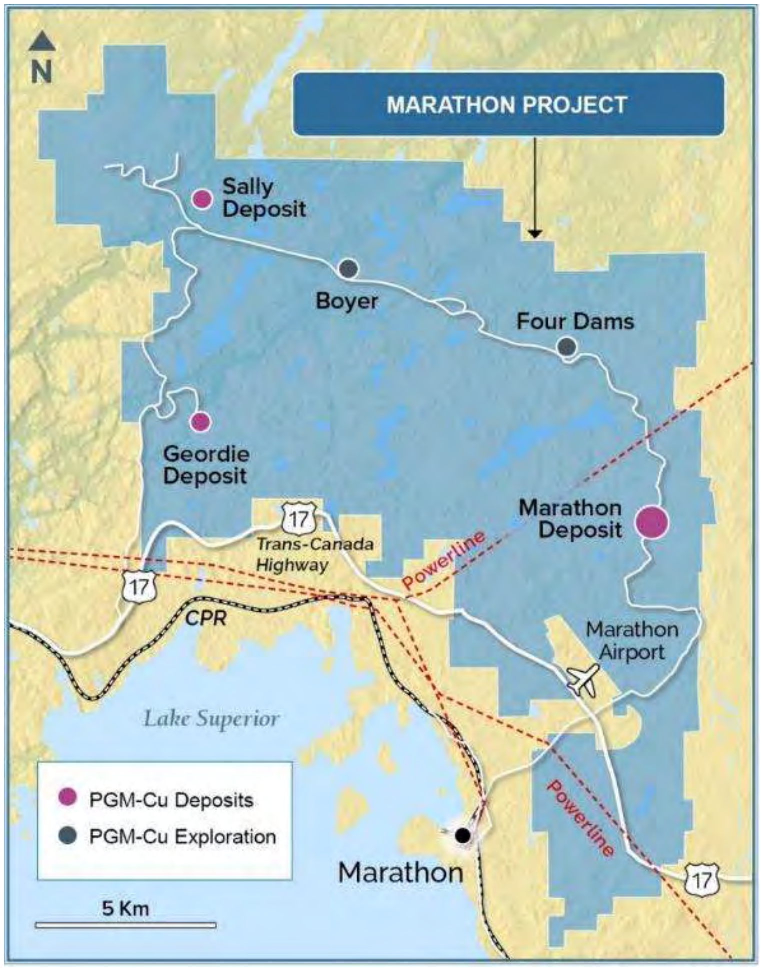
Electric power from Ontario power grid is readily available for the Project with a new East-West tie crossing the southern limits of the Property. In addition to this line, the Manitouwadge ("M2W") high voltage power line transects the Property northwest of the Marathon Deposit.

A rail line runs a few km south of the Property (Figure 5.2) and shallow water dock facilities are available at Marathon and Heron Bay. Telephone and mobile communication infrastructure are readily available regionally and in close proximity to the Property boundary.

Water is readily available on site from various sources including local lakes and creeks in the area.

Land-use activities in the area include hunting, fishing, trapping and snowmobiling. The existing Property access road is used by anglers to access the Pic River and by snowmobile users in the winter. Sport fishing activity is focused on the Pic River, which contains a variety of warm water fish species and in Hare and Bamoos Lakes located northwest of the Property. Pukaskwa National Park is located near the mouth of the Pic River approximately 20 km downstream of the Property.

Figure 5.2: Access, Topography, Physiography Marathon Project Map



Source: *Gen Mining (2020).*

5.4 Physiography

The Property is located in an area of moderate to steep, hilly terrain typical of glaciated areas of the Canadian Shield (Figure 5.3). The surrounding terrain is typical boreal forest cover with significant topographic relief characterized by relatively flat plateaus, truncated at steep cliffs adjacent to a series of creeks and ponds. The vegetation consists of northern hardwood and conifer trees as well as muskeg areas, which are bogs or wetlands common to all boreal forest regions. The land is not used for agriculture. Wildlife includes black bear, wolves, moose, rabbits and various migratory birds.

The site is bounded to the east by the Pic River (Figure 5.4) and Lake Superior to the south and west. The Project site is drained by a total of six primary sub-watersheds, four of which drain to the Pic River whereas the remaining two drain directly to Lake Superior. All other small creeks in the area drain into the Pic River. The interior of the Project site is isolated from both the Pic River and Lake Superior by steep relief (i.e., topography) and therefore many of the higher elevation lakes and streams of this area are fishless. In the instances where fish do occur, the community is limited to small-bodied (forage) fish (EcoMetrix, 2012).

The general elevation around the Project site is slightly higher than the overall regional topography. Ground surface elevations in the area of the Property range from approximately 260 m to over 400 m above sea level with a gradual decrease in elevation from north to south.

Occasional outcrops of gabbro are present on the Property and overburden which consists of boulder till with gabbro and mafic volcanic boulders, ranges from 0.1 m to 10 m in thickness with a typical thickness of 0.3 to 1m.

Figure 5.3: Topography Photograph

Source: *Generation Mining (2022)*

Figure 5.4: Pic River Photograph

Source: *Stillwater Canada (2012).*

5.5 Sufficiency of Surface Rights

There is sufficient surface area for all the required facilities as stated in this Technical Report, including topsoil stockpiles, ore and waste rock storage facilities, tailings storage facilities, processing plant sites and other associated infrastructure.

6. HISTORY

6.1 Exploration History

The Marathon area exploration for copper and nickel deposits started in the 1920s and continued until the 1940s with the discovery of titaniferous magnetite and disseminated chalcopyrite occurrences.

Exploration on the Marathon Property and within the Coldwell Complex began in the early 1960s but it became more consistent since 2003. There have been several changes in the approach to exploration over the years, which reflects the evolving understanding of the geology and the deposit model. Early exploration was focused on iron and copper. With the rise in PGM prices, the Project was re-evaluated for its PGM potential, thus expanding the zones of interest. Additionally, the development of access over time has allowed for more continuous and lower cost exploration, especially along the northern margin of the Coldwell Complex.

6.1.1 Summary 1964 - 2022

During the past five decades, several phases of exploration and economic evaluation, including geophysical surveys, prospecting, trenching, diamond drilling programs, geological studies, Mineral Resource estimates, metallurgical studies, mining studies, and economic analyses were carried out on the Property. These studies have each enhanced the knowledge base of the Marathon Deposit. The following historical summary of work is taken, in part, from an internal Nordmin Marathon PGM-Cu FS dated March 14, 2014.

In 1963, Anaconda Copper acquired the Property and carried out systematic exploration work, including diamond drilling of 32,741 m in 151 drill holes from 1964 to 1966. This culminated in the discovery of a large copper-PGM deposit (the Marathon Deposit). Many of the holes were drilled in areas off the present Property. Anaconda carried out a test pitting program that recovered 23 t of mineralized material, which was sent for testing to its Extraction Metallurgy Research Division (“EMRD”) facilities. Anaconda conducted a number of metallurgical tests intermittently from 1965 to 1982. Anaconda’s primary objective was to improve metallurgical recoveries of copper and increase the copper concentrate grade. Anaconda discontinued further work on the Property in the early 1980s due to low metal prices at the time.

In 1985, Fleck Resources Ltd. (“Fleck”) purchased a 100% interest in the Property with the objective of improving the economics by focusing on the PGM values of the Marathon Deposit. Fleck carried out an extensive program, which included re-assaying the Anaconda drill core, further diamond drilling, surface trenching of the mineralized zones, bulk sampling and a pilot plant testing at Lakefield Research Limited (“Lakefield”). Fleck drilled a total of 3,627 m in 37 diamond drill holes.

In 1986, H.A. Symons carried out a FS for Fleck based on a 9,000 tpd conventional flotation plant with marketing of copper concentrate. The study indicated a low internal rate of return (“IRR”). In 1987, Kilborn

Limited carried out a PFS for Fleck that included preliminary results from the Lakefield pilot plant tests (Kilborn Limited, 1987). The study envisaged a 13,400 tpd conventional flotation plant with marketing of copper concentrate but the study indicated a low IRR, later confirmed by Teck Corporation ("Teck").

In late 1987, Teck prepared a Preliminary Economic Feasibility Report on Fleck's Marathon Project based on a conventional open pit operation and concluded that the Project was uneconomic due to low metal prices at that time.

In 1987, Euralba Mining Ltd. ("Euralba"), an Australian junior mining company, entered into a joint venture agreement with Fleck.

In 1989, BHP Engineering Pty Ltd. ("BHP") carried out a PFS for Euralba, compiled some 2,500 samples of drill core and had them assayed at Lakefield. Euralia retained Geostat Systems International ("Geostat") to develop a Mineral Resource block model of the Marathon Deposit that was used by BHP to design an optimized open pit. BHP considered several metallurgical processes, including an on-site smelter process.

In 1998, Fleck changed its name to PolyMet Mining Corp.

In 2000, Geomaque Exploration Ltd. ("Geomaque") acquired certain rights to the Property through an option agreement with PolyMet. Under the terms of the November 7, 2000 option agreement, Geomaque could earn a 50% interest in the Property by spending $2,750,000 on exploration or completing a FS by October 31, 2004. The terms of the option agreement also allowed Geomaque to earn an additional 10% interest on the Property by making a payment of $1 million within three months of the fourth anniversary of the option agreement.

Geomaque and its consultants carried out a study of the economic potential of the Project. The study included a review of the geology and drill hole database, interpretation of the mineralized zones, statistics and geostatistics, computerized block model, Mineral Resource estimation, open pit design and optimization, metallurgy, process design, environmental aspects, capital and operating cost estimates and financial modeling. Geomaque also completed 15 diamond drill holes totalling 3,158 m; however, the results were not available for incorporation in the study. The internal Geomaque study was presented as a NI 43-101 compliant Technical Report titled "Marathon Palladium Project Preliminary Assessment and Technical Report" dated April 9, 2001.

Marathon PGM acquired the Property from PolyMet in December 2003 and carried out exploration and various studies from 2004 through 2010. On December 23, 2003, Roscoe Postle Associates Inc. ("RPA") was retained by Marathon PGM to prepare an independent Technical Report on the Project including an independent updated Mineral Resource estimate. The purpose of the Technical Report was to provide an independent assessment of the Property in relation to an initial public offering by Marathon PGM. As part of their assignment, RPA prepared a Mineral Resource estimate of the Marathon Deposit using the same drill hole database that Geomaque used for its 2001 Mineral Resource estimate. In addition to the drill hole database, RPA used the assay database from trenches on the Marathon Deposit that were excavated by Anaconda and Fleck.

Marathon PGM funded programs of advanced exploration and diamond drilling on a continuous basis between June 2004 and 2009. Approximately 617 holes and 113,030 m were drilled from 2004 to 2009 to expand the Mineral Resource and for condemnation holes outside of the proposed open pit area. In 2006, a Technical Report titled “Technical Report and Resource Estimate on the Marathon PGM-Cu Property, Marathon” was prepared by P&E Mining Consultants (“P&E”). In 2007, P&E authored a second Technical Report titled “Updated Technical Report and Preliminary Economic Assessment on the Marathon PGM-Cu Property, Marathon Area” for Marathon PGM. An internal study on the Mineral Resource update of the Geordie Palladium-Copper Property was produced on June 4, 2008 and filed on SEDAR. A FS was published in 2008 and updated in January 2010 by Micon/Metchem titled “Technical Report on the Updated Mineral Resource Estimate and Updated Feasibility Study for the Marathon PGM-Cu Project” Stillwater and Marathon PGM closed an agreement on December 1, 2010. Stillwater subsequently formed a Canadian corporation, Stillwater Canada. In March 2012, Mitsubishi Corp Mining Ltd. of South Africa (formerly called Coal of Africa Limited) purchased a 25% interest in Stillwater Canada. In March 2014, Nordmin Engineering Ltd. provided Stillwater Canada with an internal FS on the Property. Stillwater Canada drilled a total of 45 holes totalling 10,285 m.

From 2011 to 2017, Stillwater Canada developed trail access and conducted a systematic approach to prospecting, geological mapping, trenching, geophysics and diamond drilling. Stillwater Canada also re-logged over 150 drill holes. A total of 45 holes were drilled and 9,767 m of core was recovered from the holes.

In 2017, Stillwater was acquired by Sibanye Gold Limited (NYSE: SBSW) and renamed Sibanye-Stillwater (NYSE: SBSW).

On July 10, 2019, Gen Mining had (through its wholly-owned subsidiary Gen PGM) completed the acquisition of a 51% initial interest in the Marathon Property and entered into a joint venture agreement with Stillwater. The Company paid $3 million in cash and issued 11,053,795 common shares of Gen Mining at a deemed price of $0.2714 per common share (totalling $3 million), for total consideration of $6 million. Pursuant to the joint venture agreement Gen Mining had the right to increase its interest in the Marathon Property to 80% by funding $10 million in exploration, evaluation, and development expenditures and preparing a PEA within four years (the “Ownership Increase Right”). On February 19, 2020, the Company filed a PEA and in November 2020 incurred $10 million in exploration and evaluation expenditures fulfilling the Ownership Increase Right. On November 27, 2020, the Company increased its ownership interest to hold an 80% interest in the Marathon Property. On December 14, 2020, Stillwater elected to forgo its proportionate share of joint venture funding and dilute pursuant to the provisions of the joint venture agreement (“Dilution Provisions”). On July 21, 2021, Stillwater elected to not exercise its ownership increase right to expand its ownership in the Marathon Property to 51%. Pursuant to the Dilution Provisions, Gen Mining held an 83.5% and Stillwater a 16.5% interest in the joint venture as of September 30, 2021.

On December 8, 2021, Gen Mining entered into a binding acquisition agreement (the “Acquisition Agreement”) with Stillwater, pursuant to which Gen Mining would acquire Stillwater’s remaining 16.5% interest in the Marathon Project for total consideration of 21,759,332 common shares of Gen Mining. On January 26, 2022, the Company completed the acquisition of Stillwater’s interest and currently owns a 100% interest in the Marathon Property. As a result of this transaction, Sibanye-Stillwater, a leading international precious metals mining company, with a diverse portfolio of operations in the United States and Southern Africa, now holds 32,813,127 common shares of the Company, representing approximately 18.2% of the Company’s issued and outstanding common shares as at February 28, 2023.

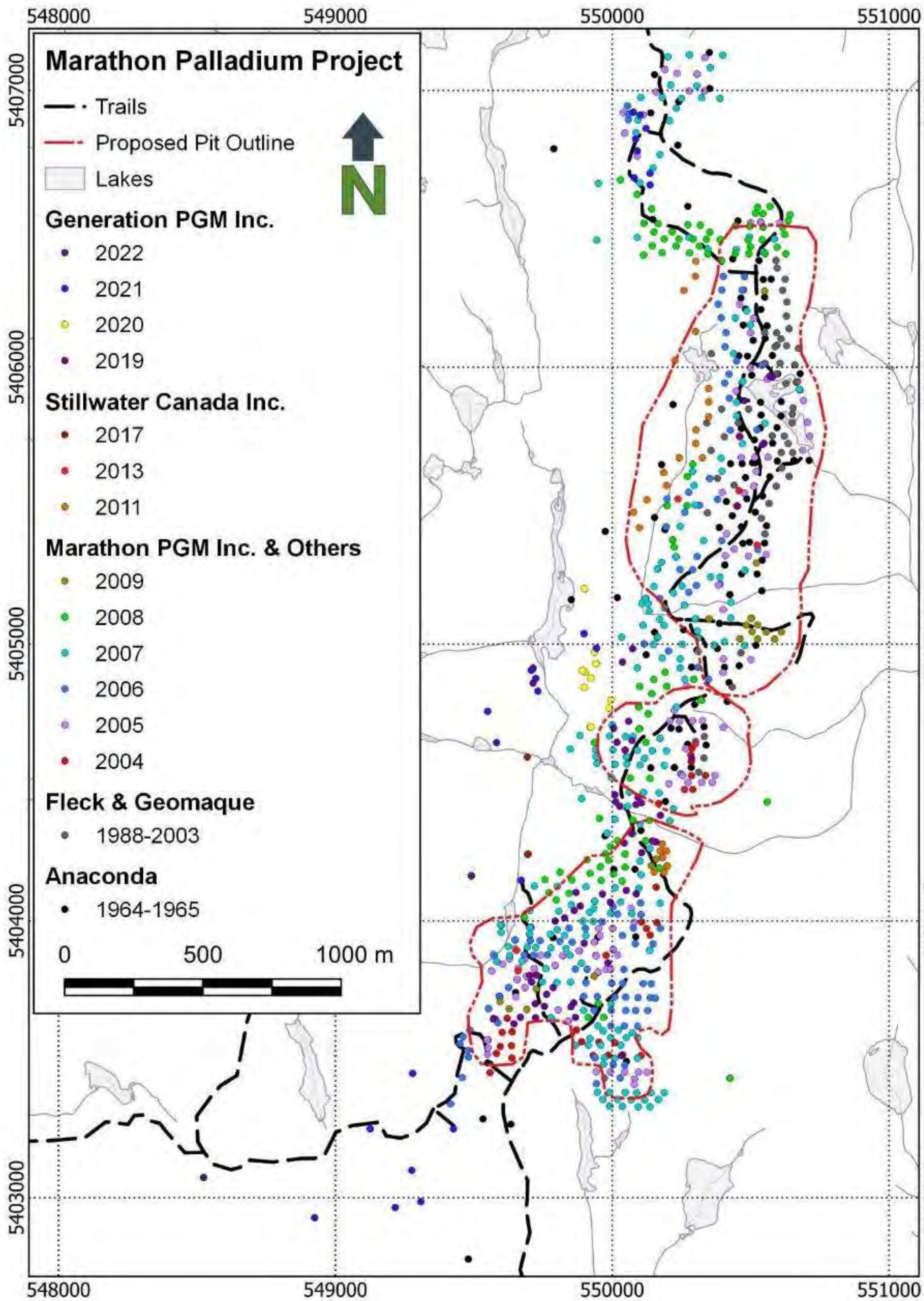
6.2 Drilling

A summary of the diamond drilling prior to July 11, 2019 on the Project is listed in Table 6.1 and also shown in Figure 6.1. All historical drill holes (prior to 2022) were previously surveyed in UTM NAD 27 Zone 16N. In 2022, all historical UTM NAD27 information was converted to UTM NAD83.

Table 6.1: Summary of Historical Drilling and Trenching on the Marathon Property, 1964-2022

Company	Year	No. of Holes / Trenches	Total Length (m)
Drilling Data			
Anaconda	1964-1966	151	32,741
Fleck	1980s	37	3,627
Geomaque	2000	15	3,158
Marathon	2004	32	4,080
Marathon	2005	102	14,602
Marathon	2006	108	21,799
Marathon	2007	205	39,781
Benton	2005-2007	50	9,198
Various - Geordie	1987-2010	61	9,647
Various - Sally	1991-2017	82	16,975
Marathon	2008	99	21,239
Marathon	2009	21	2,333
Stillwater Canada	2011	35	6,553
Stillwater Canada	2013	6	1,400
Stillwater Canada	2017	22	5,925
Generation Mining	2019	39	12,809
Generation Mining	2020	12	5,068
Generation Mining	2021	22	10,000
Generation Mining	2022	50	8,068
Sub-Total		**1,149**	**229,002**
Trenching by Location			
Marathon	2004-2021	111	7,832
Sally-Redstone	1991-2017	25	1,857
Skipper-Boyer	2017-2021	13	824
Four Dams-Lacobeer	2013	20	994
Total		**169**	**11,507**

Figure 6.1: Diamond Drill Hole Locations, Marathon Deposit, Organized by Exploration Companies



Source: Gen Mining (2022).

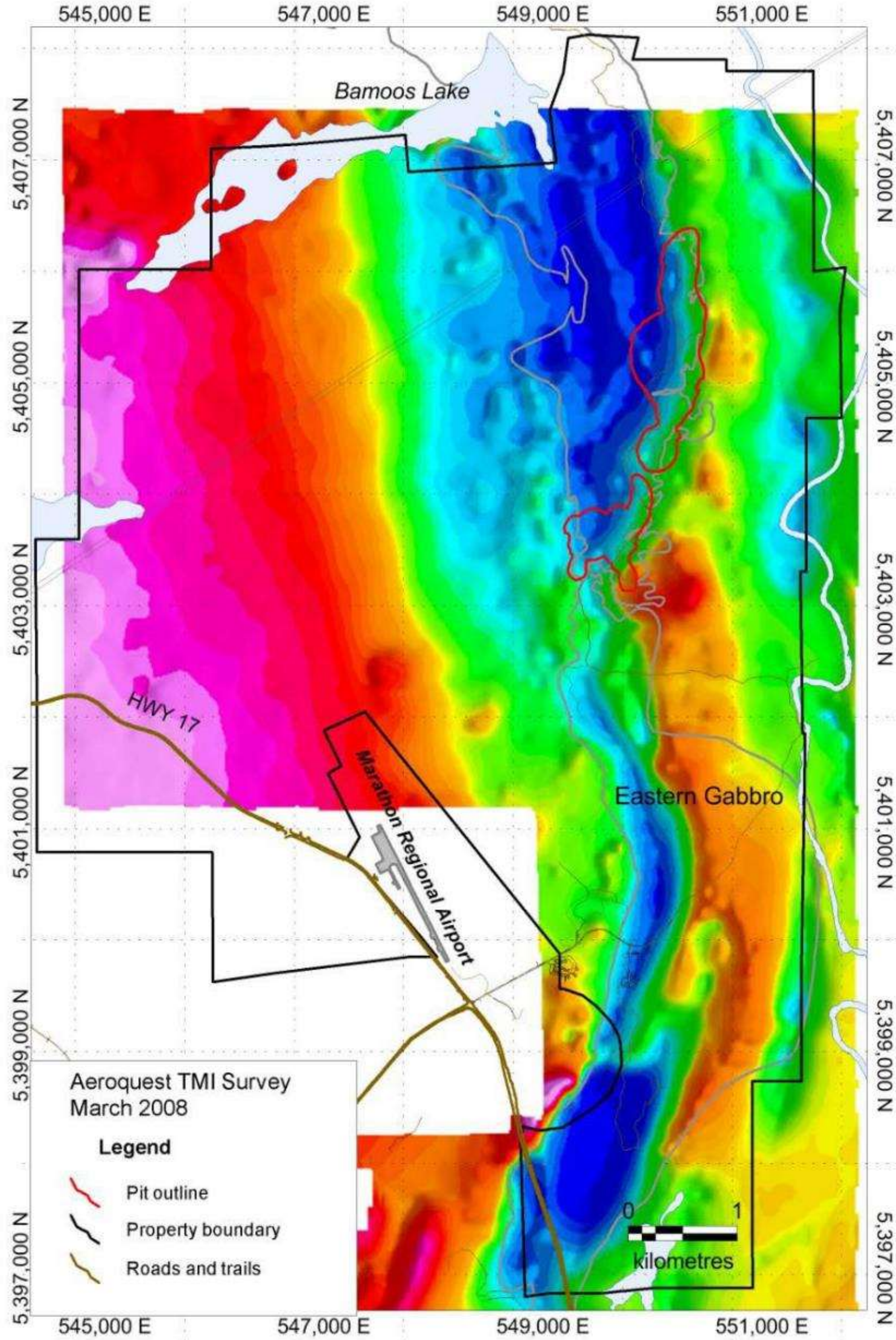
6.3 Historical Geophysical Surveying

Several geophysical surveys have been conducted over the Property and are summarized in Table 6.2.

Table 6.2: Summary of Geophysical Surveys

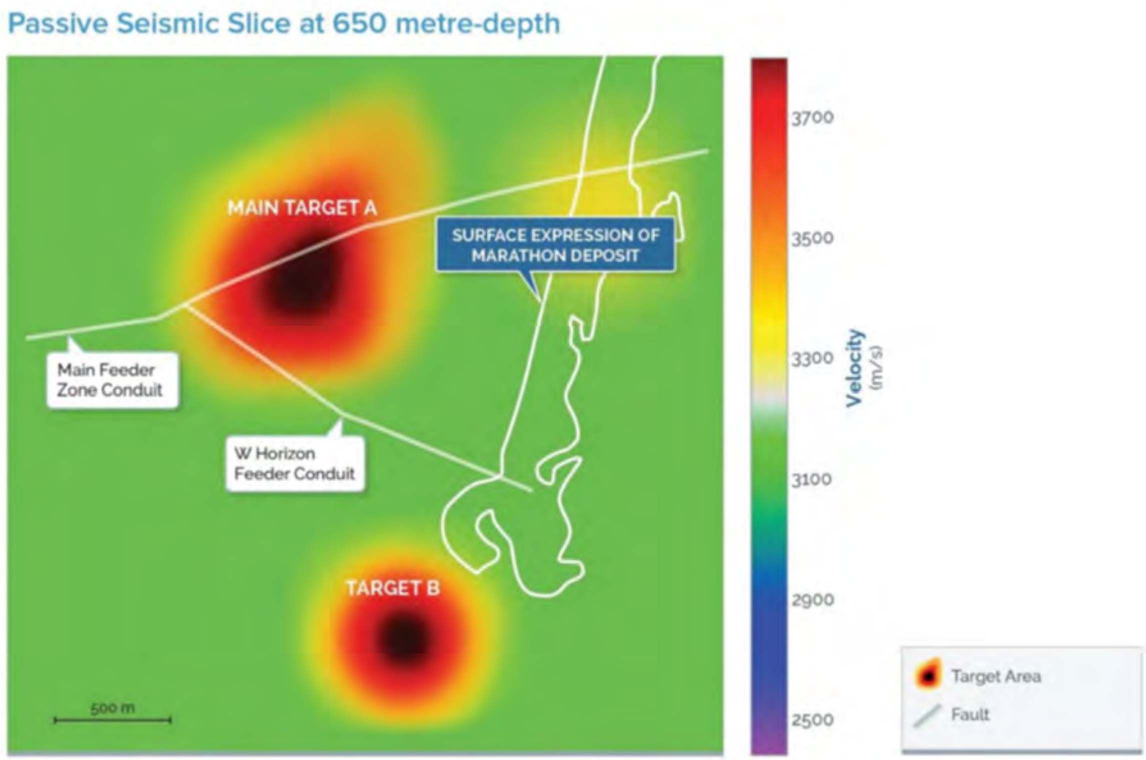
Year	Survey Type	Purpose
2005	IP/Resistivity & Magnetics by JVX	Delineate disseminated sulphide zones believed to contain copper and PGM mineralization.
2007	Geophysical Survey Report: Insight Section Array Induced Polarization and Resistivity Surveys. February 2007 Insight Geophysics Inc.	Acquire high density apparent resistivity and chargeability measurements from near surface to depths up to 500 m.
2007	Heliborne VTEM and Cesium Magnetometer Geophysical Survey October to December 2007	Identify sub-surface conductive and magnetic features of interest for follow-up ground exploration and drilling
2007	Geophysical Survey Report: Insight Section Array Induced Polarization and Resistivity Surveys May 2007 Insight Geophysics Inc.	Determine the geometry of the source producing the negative magnetic trend with the possibility of outlining any embayment that could be favourable to hosting wider zones of the targeted mineralization. (Figure 6.2)
2008	Heliborne AeroTEM System EM and Magnetic Survey Superior Block March 2008 by Aeroquest International	
2011	Heliborne High Resolution Aeromagnetic and Spectrometric Survey June 2011 Geo Data Solutions GDS Inc.	Data used to guide exploration over the Marathon Property.
2012	Gravity Survey of the Marathon PGM-Cu Deposit August 2012	Model the eastern gabbro at depth and identify a potential magma source below the Coldwell complex.
2015	Hole to hole 3D Borehole IP, July 2015, Abitibi Geophysics	Attempt to define a conductive zone within the higher sulphide portion of the high-grade PGM Zone.
2016	Surface pulse-EM survey, Oct 2016, Crone Geophysics	Confirm and model the conductive zones below the W-Horizon.
2017	Borehole EM Survey at Sally, Jun 2017, Crone Geophysics	Search for off hole conductors in holes SL-17-58 & SL-17-59.
2018	Passive seismic tomography survey, Aug 2018, PACIFIC	Delineate the likely conduits for the magma that originally formed the Main Zone and W Horizon Deposits. (Figure 6.3 and Figure 6.4)
2018	High resolution ground gravity survey, Oct 2018, Abitibi Geophysics	
2019	Borehole EM Survey at Marathon and Sally, Nov 2019, Crone Geophysics	Search for off hole conductors in hole M-19-536, M-19-537, SL-19-78.
2017 - 2020	Borehole EM Survey at Marathon, Aug-Oct 2020, Crone Geophysics	Search for off hole conductors in holes M-20-539, M-20-543 & M-20-547.

Figure 6.2: Magnetometer Survey Results Over the Marathon Property



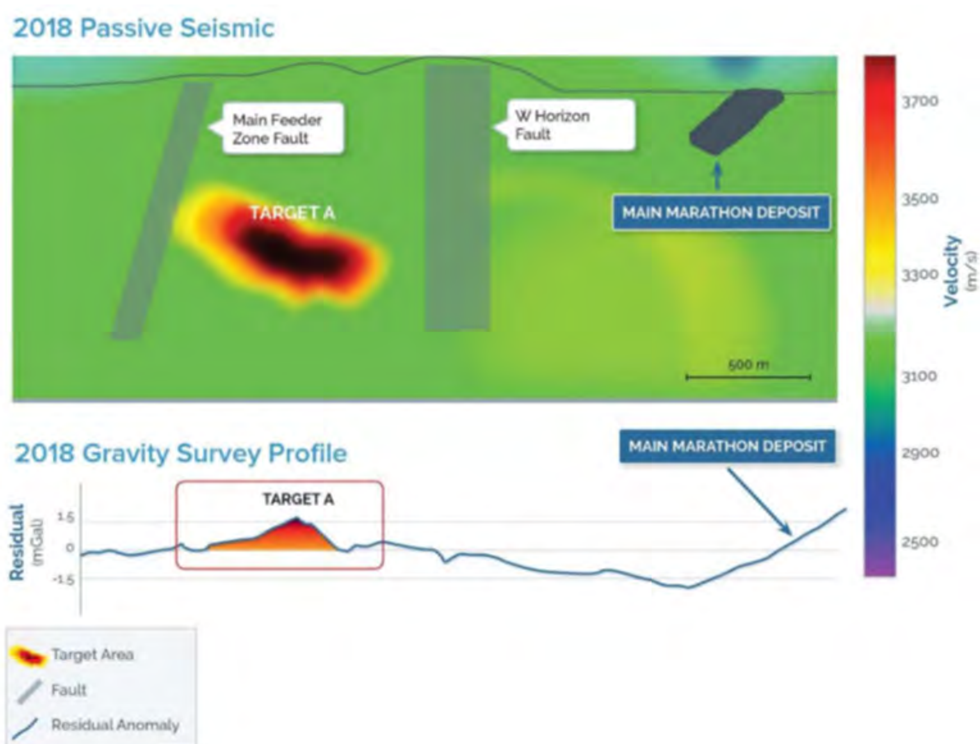
Source: Stillwater Canada Inc. (2014).

Figure 6.3: Seismic Data Revealing Potential Feeder Zones



Source: *Generation Mining Limited (2019).*

Figure 6.4: Seismic Data Profile on Potential Feeder Zones



Source: *Gen Mining (2019).*

6.4 Geological Mapping

As part of the 2005 summer exploration program, a detailed geological survey was carried out over the same grid that was established for the geophysical surveying. Approximately 15.0 line-km of mapping and prospecting was conducted. The results of the geological mapping program were incorporated into the existing geological database. Geological mapping also continued through 2007-2009 summer exploration programs. Geological mapping was carried out between 2014-2018 at Sally, Four Dams to Boyer to update historic mapping into the current geological legend. Additional geological mapping was carried out in the summer of 2021 to cover the areas between Marathon and Four Dams, Boyer and Sally, and Sally and Redstone that had not yet been mapped into the current geological legend. At the conclusion of this program, the entire Marathon Horizon from the main deposit area to the western extent of the Property had been mapped to modern standards. Additional mapping between 2021 and 2022 was focused on proposed infrastructure for the mining operations to support detailed engineering and ensure that no mineralization of economic potential exists in these areas.

6.5 Historical Mineral Resource Estimates

Historical Mineral Resource estimates on the Marathon Deposit are summarized in Table 6.3. The estimates are difficult to compare due to different cut-off grades, metal prices and recovery assumptions. Additional details regarding previous Mineral Resource estimates can be found in the 2021 Feasibility Study report.

Table 6.3 : Historical Mineral Resource Estimates - Marathon Deposit*

Estimator and Date	Tonnes (M)	Pd (g/t)	Pt (g/t)	Cu (%)	Cut-off Value
Anaconda, 1984	31.3	1.34 combined		0.47	N/A
Kilborn, 1986	42.6	1.51	0.41	0.46	N/A
Kilborn, 1987	36.9	1.10	0.27	0.38	$12/t NSR
Geostat, 1988	29.4	1.02	0.26	0.36	$16/t NSR
Geomaque, 2001	21.3**	1.32	0.34	0.40	0.8 g/t Pd
RPA, 2004	62.5**	0.79	0.20	0.30	0.15% Cu
P&E, 2006	68.3**	0.91	0.25	0.32	$12/t NSR
Micon, 2009	114.8**	0.78	0.23	0.24	$10.50/t NSR
P&E, 2020	202.8**	0.55	0.19	0.20	$13/t NSR

**Marathon Deposit only, not including Sally, Geordie or other*

*** Measured + Indicated Resources.*

The historical Mineral Resource estimates summarized in Table 6.3 are historical in nature and, as such, are based on prior data and reports prepared by previous operators and are not in compliance with NI 43-101. A Qualified Person has not done the work necessary to verify the historical estimates as current estimates under NI 43-101, and the estimates should not be relied upon. There can be no assurance that any of the Mineral Resources, in whole or in part, will ever become economically viable. They are listed for information and historical reference purposes only, as they demonstrate the development history of the Marathon Deposit. Gen Mining is not treating the historical estimates as current Mineral Resources or Mineral Reserves. The Company has completed the necessary work to establish a current Mineral Resource on the Marathon Property as presented in Section 14 of this FS Report.

6.6 Historical Mineral Reserve Estimates

Historical Mineral Reserve estimates on the Marathon Deposit are summarized in Table 6.4. The estimates are difficult to compare due to different cut-off grades, metal prices and recovery assumptions.

Table 6.4 : Historical Mineral Reserve Estimates - Marathon Deposit

Estimator and Date	Tonnes (M)*	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	Cut-off Value
Micon, 2010	91.5	0.83	0.24	0.09	0.25	--	US$10.50/t NSR
GMS, 2021	117.7	0.62	0.20	0.07	0.21	1.41	US$13/t NSR

**** Proven + Probable Reserves.***

The historical estimates referred to Table 6.4 are historical in nature and are not compliant with and not meant as defined in National Instrument 43-101, but used only as these terms were understood at the time. A Qualified Person has not done sufficient work to classify the historical estimates as current Mineral Reserves, and the issuer is not treating the historical reserve estimates as current Mineral Reserves. Whereas the Authors of this current Technical Report considers these figures as relevant and reliable, they are not meant to be understood as anything beyond a very preliminary order of magnitude approximation of the potential tonnage and tenor of mineralized material that occurred in situ at Marathon. The Company has completed the necessary work to establish a current Mineral Reserve on the Marathon Property as presented in Section 15 of this FS Report.

6.7 Feasibility Studies

6.7.1 Micon 2010 Feasibility Study

Marathon PGM retained Micon to update a FS on the Marathon Property. The prior FS had been completed in December 2008. As part of this update of the FS, Micon prepared an updated Mineral Resource estimate, a new open pit mine design and new mine schedule, and a new Mineral Reserve estimate. The Technical Report presented the updated Mineral Resource and Reserve estimates and discussed the results of the updated FS for the Marathon Property. The effective date of the updated FS was November 24, 2009 (Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada, dated January 8, 2010).

The design of a 22,000 tpd process plant comprised primary crushing, secondary crushing, high pressure grinding rolls (“HGPR”), ball milling, flotation, concentrate dewatering and process solids (tailings) disposal. The concentrator was designed to produce a copper sulphide flotation concentrate containing PGMs and gold. The total capital cost estimate was $495 million, comprising $351 million of pre-production capital and $144 million of sustaining and closure capital. The estimated total average LOM unit operating cost was $16.64/t.

The Micon FS demonstrated the potential to generate strong cash flow under appropriate metal price assumptions of US$2.91/lb Cu, US$1,346.65/oz Pt, US$321.44/oz Pd, US$819.22/oz Au, US$14.10/oz Ag, and an exchange rate of C$/US$ = 1.099. The base case results showed that the Project generated an IRR of 21.2% before-tax and 17.4% after-tax. The undiscounted payback period was 4.4 years, and the discounted cash flow was positive after 6 years. The NPV at a 6% discount rate was $250.7 million after-tax. The sensitivity studies demonstrates that the Project was quite sensitive to adverse changes in price assumptions and moderately sensitive to changes in operating cost or capital expenditure. Micon recommended that Marathon PGM proceed with the development of the Marathon Deposit.

6.7.2 GMS 2021 Feasibility Study

Gen Mining retained GMS to produce a FS on the Marathon Property. As part of this FS, P&E prepared an updated Mineral Resource estimate and GMS produced new open pit mine designs and new mine schedule, and a new Mineral Reserve estimate. A new process plant design by Ausenco was also produced as part of the FS. The Technical Report presented the updated Mineral Resource and Reserve estimates and discussed the results of the FS for the Marathon Property. The effective date of the FS was March 3, 2021 (Feasibility Study for the Marathon Palladium & Copper Project, Ontario, Canada, dated March 23, 2021).

The design of a 25,200 tpd process plant comprised primary crushing, SAG milling, ball milling, pebble crusher, flotation, PGM scavenger circuit, concentrate dewatering and process solids (tailings) disposal.

The concentrator was designed to produce a copper-palladium concentrate. The total capital cost was estimated at $1,377 million, comprising $888 million of pre-production capital and $489 million of sustaining and closure capital. The estimated total average LOM unit operating cost was $23.61/t.

The GMS FS completed on the Property demonstrated the potential to generate strong cash flow under the metal price assumptions of US$3.20/lb Cu, US$1,000/oz Pt, US$1,725/oz Pd, US$1,400/oz Au, US$20/oz Ag, and an exchange rate of C$/US$ = 1.28. The base case results showed that the Project generated an IRR of 38.6% before-tax and 29.7% after-tax. The undiscounted payback period was 2.3 years. The NPV at a 6% discount rate was $1,068 million after tax. The sensitivity studies demonstrated that the Project was quite sensitive to adverse changes in price assumptions and moderately sensitive to changes in operating cost or capital expenditure. GMS recommended that Gen Mining proceed with the development of the Marathon Deposit.

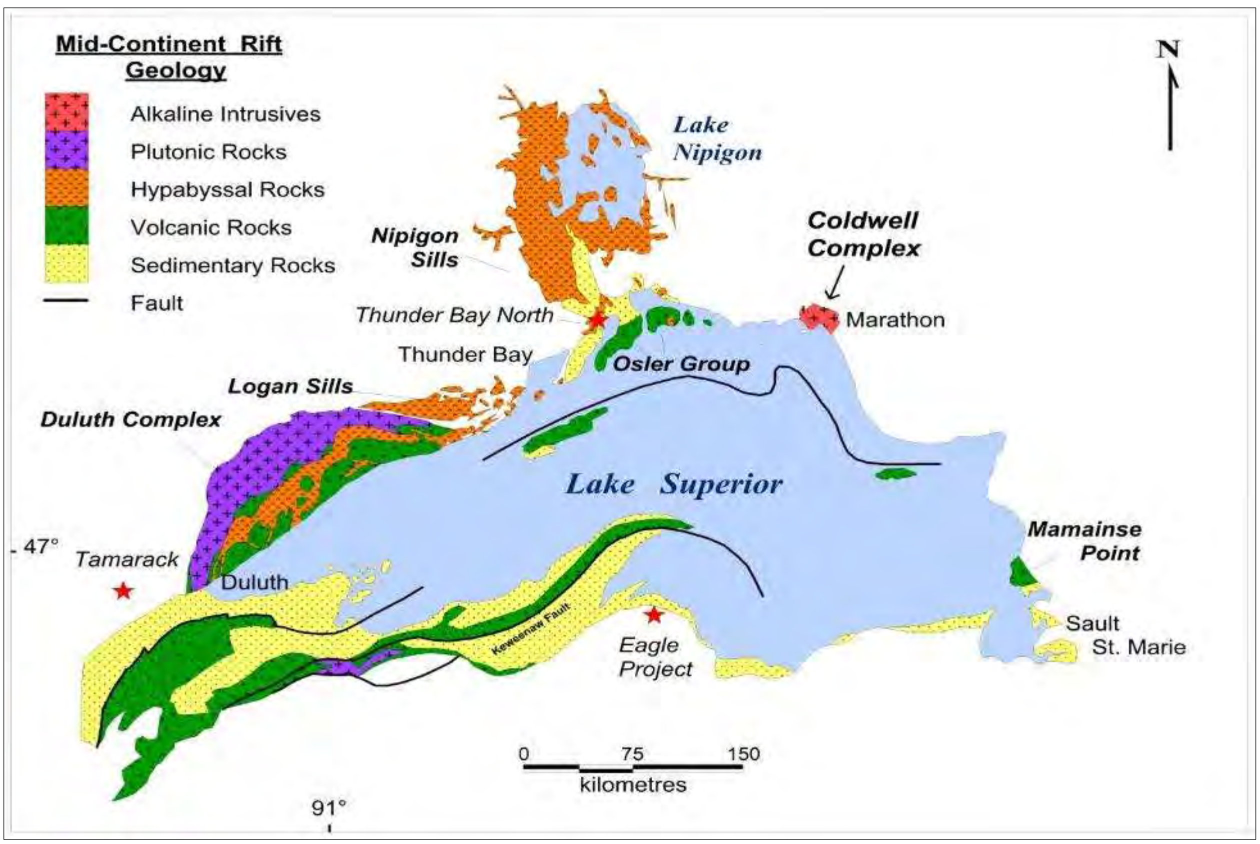
7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Marathon Deposit is hosted by the Two Duck Lake ("TDL") Gabbro, a late intrusive phase of the Eastern Gabbro. The Eastern Gabbro has recently been described as a composite intrusion by Good et al. (2012) and occurs along the northern and eastern margin of the Proterozoic Coldwell Complex which intrudes the much older Archean Schreiber-Hemlo greenstone belt (Figure 7.1). The sub-circular Coldwell Complex has a diameter of 25 km and a surface area of 580 km^2 and is the largest alkaline intrusive complex in North America (Walker et al. 1993).

The Coldwell Complex is believed to have intruded over a relatively short period of time near the beginning of the main stage of the Mid-Continent Rift magmatism that occurred between 1108 and 1094 Ma (Heaman and Machado, 1992 and Heaman et al., 2007).

Figure 7.1: Regional Geology of the Mid-Continent Rift in the Lake Superior Area



Source: *Marathon PGM Corp. (2010).*

7.1.1 Geology of the Coldwell Complex

The Coldwell Complex was first described as a lopolith by Puskas (1967) and as three intrusive centres by Mitchell and Platt (1977). The intrusive centres were later described as three superimposed rings by Currie (1980). Detailed mapping across the Coldwell Complex by Walker et al. (1993) supported the multiple intrusive centre model of previous interpretations. Walker et al. also proposed that the Coldwell Complex has a sub-horizontal structure or stratigraphy.

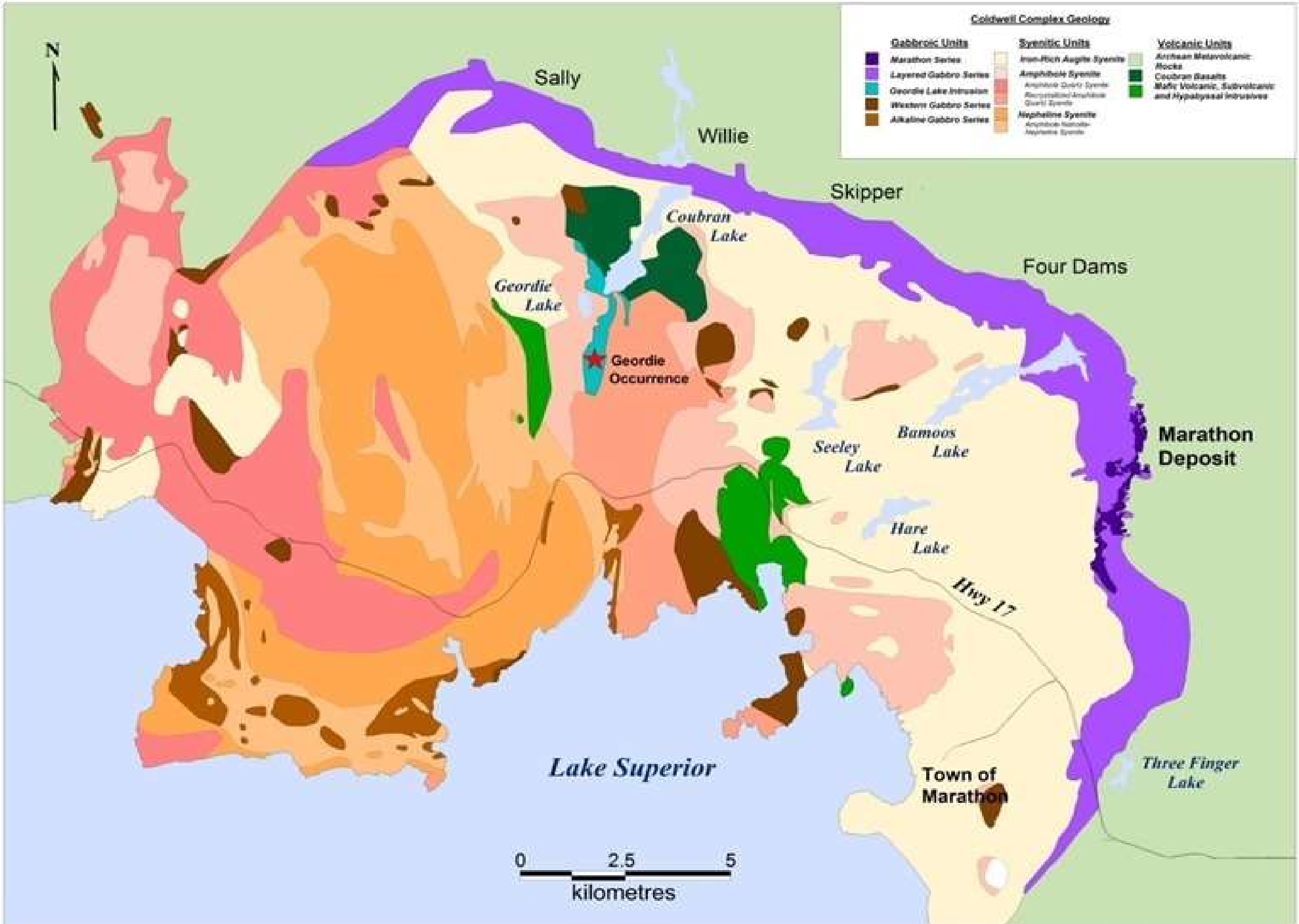
The major rock units of each magmatic centre of the Coldwell Complex Figure 7.2, include the following:

- Centre I: Eastern and Western Gabbros, Amphibole Quartz Syenite, Iron rich Augite Syenite, Monzodiorite and mafic volcanic and subvolcanic rocks.
- Centre II: Amphibole Nepheline Syenite and Alkaline Gabbro.
- Centre III: Quartz Syenite and Amphibole Quartz Syenite.

Work by Kern et al. (2012) and Kulakov et al. (2012) suggests Centres I and III were intruded prior to Centre II. These two studies presented comprehensive paleomagnetic data from the Coldwell Complex and included measurements from intrusive syenitic to gabbroic rocks of Centres I, II and III. The results of Kern et al. indicate that paleomagnetic signatures for Centres I and III are statistically indistinguishable, and that rocks of Centre II were emplaced after the magnetic reversal that occurred about 1103-1104 Ma. The study by Kulakov et al. examined the package of volcanic rocks located in the centre of the Coldwell Complex and determined that the paleomagnetic signature for the basalts is very similar to that for intrusive rocks of Centres I and III as determined by Kern et al. and is consistent with a deposition age of 1107 Ma.

Most recently passive seismic surveys were completed at the Marathon Deposit in 2018 and 2019 and at the Sally Deposit in 2019. This technique measures wave velocity contrasts between lithologies based on density variation. The density contrast between basement Archean footwall, syenites, gabbros and oxide melagabbros is sufficient to distinguish between lithologies. The survey results showed a large sub-horizontal, undulating high-velocity zone dipping to the west, extending from the Eastern Gabbros (Good et al., 2020). In 2019, an exploration drill hole tested the velocity model with a 1,000 m deep hole through the syenites of Center I. The stratigraphy intersection started with syenites from surface to 300 m depth followed by layered series gabbro showing inward dipping layering and flattening to a sub-horizontal sheet (Good et al, 2020). This does not support an outer ring dyke structure or a larger lopolith with basalt roof pendants. The Coldwell Complex most likely formed by intrusions of alkaline gabbro or syenite sills into a basalt pile. The features of the Coldwell Complex, sub-horizontal emplacement, circular shape of the complex and coincident gravity high, are most consistent with emplacement within a volcanic caldera (Good et al., 2020).

Figure 7.2: Geology of the Coldwell Complex



Note: *Shows the locations of the Marathon Deposit and the Geordie Deposit. Geology modified after Walker et al. (1993).*
Source: *Marathon PGM Corp. (2010).*

7.1.2 Geology of the Eastern Gabbro

The Eastern Gabbro forms part of a very large magmatic system and contains numerous Cu-PGM occurrences along its entire length. It is up to 1,500 m thick and strikes for 33 km around the eastern margin of the Coldwell Complex (Figure 7.12). It is considered the oldest intrusive phase of the Coldwell Complex and was interpreted to have formed by multiple intrusions of magma into restricted dilatant zones within volcanics, possibly associated with ongoing caldera collapse (Walker et al, 1993; Shaw, 1997 after work by Puskas (1967 and 1970); and Currie (1980). Shaw (1997) concluded the Eastern gabbro consists of evolved basaltic magma with a sub-alkaline parentage).

The magnetic signature of the Eastern Gabbro in the area of the Marathon Deposit is shown in Figure 7.3, which highlights the segmented or discontinuous character of various phases of the Eastern Gabbro.

The Eastern Gabbro is overlain by massive to layered augite syenite (Puskas, 1970; and Walker et al., 1993). The layering in the gabbro and the augite syenite dip moderately towards the center of the complex.

7.1.2.1 Historic Classification of the Eastern Gabbro

Puskas (1970) subdivided the Eastern Gabbro into three groups: the Outer Border Zone of chilled gabbro; the Inner Border Zone A of massive gabbro; and the Inner Border Zone B of layered gabbro. Based on detailed regional mapping, Walker et al. (1993) subdivided the Eastern Gabbro into three dominant intrusive bodies: the Eastern Layered Gabbro Series; the TDL Gabbro; and the Malpa Lake Gabbro. Further detailed study of two stratigraphic sections through the Layered Gabbro Series by Shaw (1997) resulted in the definition of at least three intrusive phases separated by thick zones of xenolith-laden massive gabbroic bodies. The lower zone consists of a fine-grained chill (Sequence I) that grades upward into modally layered gabbro at the metre scale (Sequence II) to the centimetre scale (Sequence III).

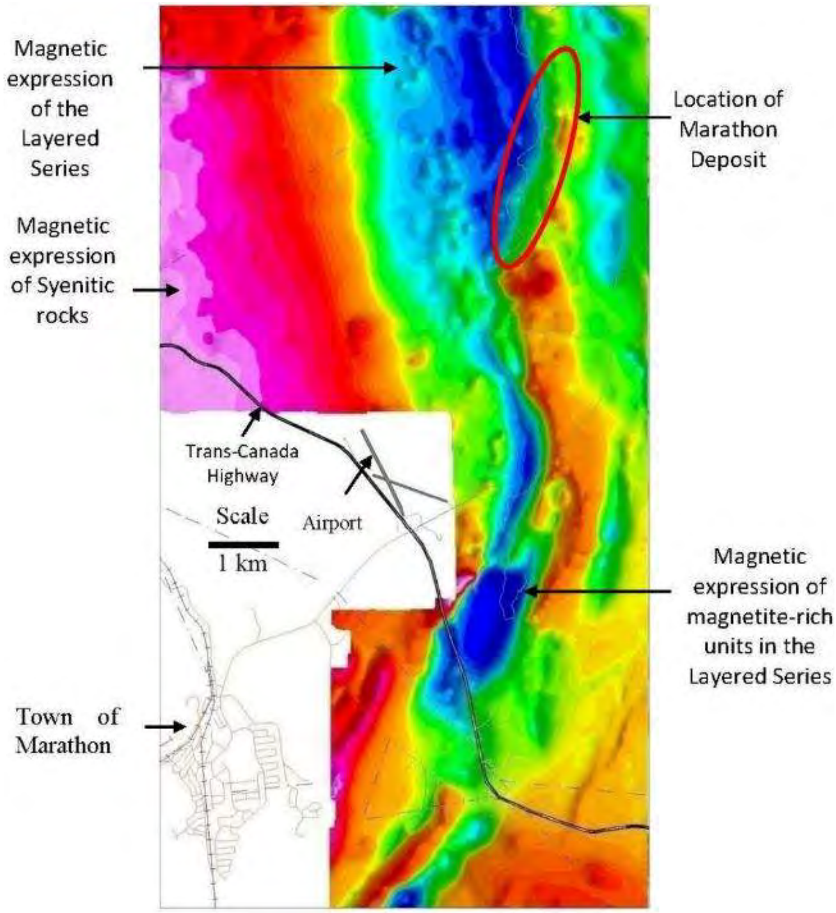
7.1.2.2 New Classification of the Eastern Gabbro

A new classification of the Eastern Gabbro, as proposed by Good et al. (Economic Geology 2015), includes the Fine Grained Series, Layered Series and Marathon Series. The new classification is based on distinctive petrographic features, geochemical characteristics and cross-cutting relationships. The three series largely maintain the subunits of the Eastern Gabbro as presented by Puskas (1970) and Shaw (1997) but with the main differences that the units are not necessarily co-genetic. The Marathon Series is the youngest intrusive phase and is defined here to include all mafic and ultramafic intrusive rocks that host copper and PGM mineralization in the vicinity of the Marathon Deposit. The Fine Grained Series is the oldest phase and is equivalent to the outer boundary chill gabbro of Puskas or Sequence I rocks of Shaw. The Layered Gabbro Series matches the Inner Zones A and B of Puskas or Sequences II and III of Shaw (Table 7.1).

Table 7.1: New Classification Scheme for the Eastern Gabbro

Good et al.'s Classification for Eastern Gabbro				Previous Classification Schemes		
Series	**Unit**	**No. of Sub-units**	**Relative Age**	**Puskas, 1970**	**Wilkinson, 1983**	**Shaw, 1997**
Fine Grained Series	Metabasalt	4	Oldest	Outer border zone of chilled gabbros	Fine Grained Gabbro	Layered Gabbro Series I
	Peridotite	2				
Layered Series	Olivine gabbro	2		Inner Border Zone B of Layered gabbro	Banded Gabbro	Layered Gabbro Series II and III
	Oxide augite melatroctolite	1				
Marathon Series	Gabbroic anorthosite	1		Inner Border Zone A of massive gabbro	Mottled Gabbro	
	Wehrlite	4				
	Augite troctolite	7				
	Oxide melatroctolite	2			Magnetite Olivinite	
	Two Duck Lake Gabbro	6			Heterogeneous Gabbro	Two Duck Lake Gabbro
	Apatitic clinopyroxenite	3	Youngest			

Figure 7.3: Total Magnetic Image Over Eastern Boundary of the Coldwell Complex



Source: *Marathon PGM Corp. (2010).*

7.1.3 Detailed Geology of the Marathon PGM-Cu Property

The Property geology is defined to a large extent by the intrusive cross-cutting relationships between the Marathon Series and the earlier Fine-Grained Series, and by the complicated nature of the basal contact with the partially melted Archean rocks. The geology of the Property is shown on a plan map (Figure 7.4).

The TDL Gabbro is the dominant host rock for Cu-PGM mineralization. Additional accumulations of Cu-PGM mineralization are associated with oxide ultramafic intrusions of the Marathon Series that consist of clinopyroxene ± olivine ± magnetite ±apatite cumulate rocks. These ultramafic bodies occur predominantly in the hanging wall of the Marathon Deposit and were formerly referred to as Layered Magnetite Olivine Cumulates.

7.1.4 Archean Country Rock and Rheomorphic Intrusive Breccia

The footwall of the Marathon Deposit comprises Archean intermediate pyroclastic rocks that have undergone partial melting as a result of the heat of intrusion of the Eastern Gabbro. At the contact with the

Eastern Gabbro, the footwall is referred to as Rheomorphic Intrusive Breccia (“RIB”). The RIB/gabbro contact is not a simple contact as blocks of RIB material occur within the gabbroic series and intrusions of gabbro extend deep below the footwall contact. Also, a few thin near vertical promontories of RIB extend into the gabbroic series (Figure 7.4).

In a detailed study of the RIB, Uldis Abolins (1967) described the breccia as a matrix supported heterogeneous mixture of angular and sub-rounded fragments composed of fine to coarse grained gabbroic material, quartzite, pyroxenite and layered quartz pyroxenite. A distinguishing feature of the RIB is the common occurrence of elongate curved pyroxenite fragments. Abolins estimated the composition of the breccia matrix to be close to that of a quartz norite.

Locally, the footwall forms basins and ridges under the TDL Gabbro. This paleo surface played an important role in the formation of the Marathon Deposit by encouraging accumulation of sulphides through physical processes such as settling out of sulphide droplets in the magma conduit (refer to Section 8 – Deposit Types).

The Archean country rock varies along strike from the Marathon Deposit to the north and includes amphibolite, granodiorite, mafic to felsic volcanics and metasediments; however, in all areas RIB can be observed in surface mapping and drill core.

7.1.5 Metabasalt (Fine Grained Series)

The most abundant rock type in the hanging wall overlying the Marathon Deposit is metabasalt. Layering can be detected at the metre scale by gradational change in grain size. Contacts with other gabbro units are sharp.

The metabasalt consists of equigranular clinopyroxene, olivine, plagioclase and minor magnetite. Intergranular angles are near 120° indicating the fine metabasalt is re-crystallized. Re-crystallization would require very high temperature metamorphism perhaps of pyroxene hornfels grade. Metamorphism occurred during intrusion of Layered Series and TDL Gabbro.

An important and remarkable feature of metabasalt is the extremely low level of secondary alteration. In a survey of 50 thin sections, only a few sections contained serpentine alteration of olivine and one section contained amphibole alteration of olivine. Tremolite was not observed. Trace to less than trace amounts of secondary minerals such as chlorite and muscovite occur in the vicinity of olivine or cross-cutting fractures.

Locally, the occurrences of flattened pipe shaped features that resemble amygdules imply the unit originated as basaltic flows that were recrystallized during pyroxene hornfels grade metamorphism.

A common feature within metabasalt, particularly close to intrusions of TDL Gabbro, is the formation of 1-2 cm sized zoned amoeboid shaped blebs with either a clinopyroxene or olivine core or a thin plagioclase rich rim. This texture is interpreted to have formed either by migration of material from the TDL magma along a very fine 3-D network or by pyroxene hornfels metamorphism related to intrusion of the TDL magma.

An early intrusive peridotite associated geochemically with the early metabasalt phase occurs at the Sally Deposit and is adjacent to mineralization. The peridotite is composed of medium to coarse grained euhedral clinopyroxene, fine to medium sized euhedral olivine, interstitial tabular to irregular plagioclase and minor accessory magnetite. There are multiple occurrences emplaced as vertical pipes to horizontal sills and change orientation over short distances. They can be up to 100 m thick and they are homogenous with little compositional variation.

7.1.6 Layered Olivine Gabbro and Oxide Augite Melatroctolite (Layered Series)

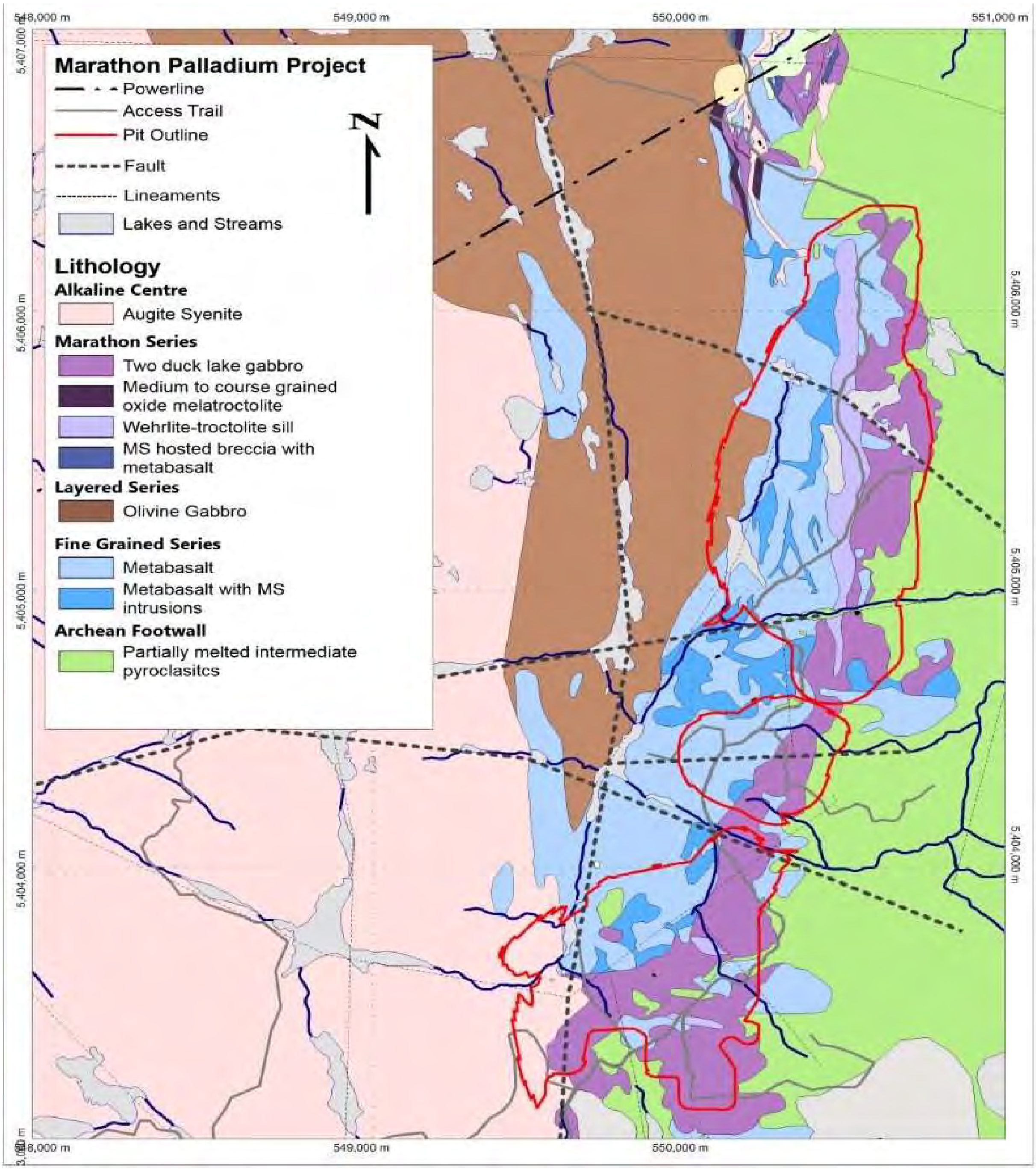
The Layered Series makes up the majority of the Eastern Gabbro and only occurs along the western edge of the Property. It is compositionally, geochemically and texturally similar along the entire strike length of the complex. The Layered Series is dominated by massive to modally layered olivine gabbro with lesser amounts of inter-layered thick units of oxide augite melatroctolite. Contacts between these units are typically gradational.

The olivine gabbro is medium to coarse grained and is characterized by intergranular texture, plagioclase alignment, and modal layering. The modal layering is defined by a gradational increase in the abundance of plagioclase and ranges in composition from olivine melagabbro to olivine gabbroic anorthosite. The lower contact of modal layers is not sharp but shows strong contrast. The modal layers are variable on a decimetre to metre scale and may show continuous to lenticular rhythmic layering. Cross-bedded, wavy or convoluted layering may also be present.

The olivine gabbro has an intergranular texture and is composed of, in decreasing order of abundance, plagioclase, clinopyroxene, olivine, magnetite and apatite. Medium to coarse grained plagioclase is euhedral to subhedral, whereas olivine and clinopyroxene crystals are medium grained and subhedral. The gabbro includes up to 10%, fine grained, euhedral and interstitial apatite and up to 10% interstitial magnetite. Alteration of plagioclase and mafic minerals to sericite and chlorite or actinolite, respectively, is weak to moderate.

The oxide augite melatroctolite is texturally similar and gradational to the layered olivine gabbro and is distinguished by abundant magnetite (15 to 25 modal %). The oxide augite melatroctolite occurs as discontinuous and irregular pods and lenses within the layered olivine gabbro. The unit is typically medium to coarse grained and may exhibit plagioclase alignment.

Figure 7.4: Geological Map of the Marathon Deposit



Note: *Mapping by geologists of Marathon PGM Corp and Stillwater Canada Inc, 2012.*
Source: *Gen Mining (2021).*

7.1.7 <u>Wehrlite-Troctolite Sill (Marathon Series)</u>

The Wehrlite-Troctolite ("WT") Sill located immediately above the main mineralization-bearing TDL Gabbro (Figure 7.5 and Figure 7.6) is an important marker horizon and is thought to have important implications with regard to the origin of the Marathon Deposit mineralization. Further, of equal or greater significance, the excellent continuity of the unit across a total of 128 carefully logged drill holes negates the possibility of numerous post mineralization faults as proposed by Dahl et al. (2001). The sill is 30 to 50 m thick and is composed of an upper wehrlite and lower augite troctolite unit and does not contain any significant sulphides.

The WT Sill is an excellent marker horizon and provides the only evidence for normal faulting along the surface lineaments located near 5,404,900 N and 5,404,500 N as illustrated in Figure 7.5.

The WT Sill occurs along the entire strike length of the Marathon Deposit and forms an important marker horizon above the Main Zone of mineralization. This relationship changes at the south end of the Marathon Deposit (near 5,403,800 N) where the dip of the sill is sub-horizontal and the TDL Gabbro cross cuts the sill to form the southwest limb of the Marathon Deposit.

The wehrlite typically occurs immediately above the augite troctolite unit. The wehrlite consists of, in decreasing order of abundance, olivine, clinopyroxene, plagioclase, and magnetite. Olivine and clinopyroxene are medium to very coarse grained but olivine is generally subhedral and clinopyroxene is anhedral. Plagioclase is interstitial and medium to coarse grained, and magnetite is anhedral to subhedral. Plagioclase comprises 5–25% of the rock. Thin layers of coarse-grained oxide wehrlite commonly occur within the wehrlite.

The augite troctolite is distinguished by the presence of coarse-grained olivine, clinopyroxene and magnetite oikocrysts. The nature of plagioclase varies from euhedral laths to anhedral, interstitial networks; the latter feature giving the augite troctolite a mottled appearance.

7.1.8 Two Duck Lake Gabbro (Marathon Series)

The TDL Gabbro is the host rock for the Marathon Deposit. It occurs as a massive and poorly layered unit approximately 50 to 250 m thick that strikes near north for greater than 6 km (Figure 7.5, Figure 7.6 and Figure 7.7) and in general dips west at angles from 5 to 45°. The TDL Gabbro intruded the Fine-Grained Series beneath the WT sill and near the basal contact with Archean Footwall. The TDL Gabbro is intruded by very thin dykelets of RIB that are partial melt derivatives of the Archean basement and also by late north-northwest trending quartz syenite dykes.

The modal mineralogy of a composite sample that is representative of the Marathon Deposit mineralization (and TDL Gabbro) was determined in a QEMSCAN survey by XPS (Kormos, 2008). A total of nine aliquots of material were analyzed. In decreasing order of abundance, the composite sample comprised 42.0% plagioclase, 25.7% clinopyroxene, 7.8% amphibole, 5.5% iron oxides, 4.6% olivine, 2.6% other silicates (quartz, epidote, talc, and serpentine), 2.2% orthoclase, 0.7% biotite, and the remainder of various sulphides (pyrrhotite, chalcopyrite and pentlandite). Orthopyroxene is rare and where present occurs as late reaction rims on olivine (Good, 1993).

The TDL Gabbro is distinguished from other gabbro types by cross-cutting relationships and mineral textures resulting from the respective crystallization histories. In TDL Gabbro, plagioclase crystallized first and forms elongate laths that are surrounded by ophitic textured clinopyroxene or olivine. Pegmatitic textured TDL Gabbro occurs locally as pods within coarse grained gabbro or as rims on Fine Grained Series xenoliths. Mineralized pegmatite makes up less than about 5% of all mineralized zones. The composition of pegmatitic TDL Gabbro was compared to that of coarse grained, TDL Gabbro by Good (1992), and found to be similar.

An important aspect of TDL Gabbro relative to other Cu-PGM deposits, such as at the Lac des Iles Mine, is the fresh unaltered nature of primary minerals and textures. There is some local development of secondary minerals such as chlorite, amphibole, serpentine and calcite but the abundance of these minerals is not greater than about 10% for the Marathon Deposit (Kormos, 2008).

There is only a minor fluctuation in mineral compositions across the TDL Gabbro (Good and Crocket, 1994a; Ruthart, 2013). Plagioclase crystals are normally zoned with compositions between 52% and 65% anorthite; however, the main mineralized zone (“Main Zone”) typically exhibits replacement at grain margins by a more calcic plagioclase (69-79% anorthite). The average olivine composition is 56.9% forsterite and 540 ppm Ni. Clinopyroxene and orthopyroxene lie, respectively, within the fields of augite and hypersthene with Mg numbers between 0.6 and 0.7.

7.1.9 Oxide Ultramafic and Apatitic Clinopyroxenite Intrusions (Marathon Series)

The thickest accumulations of magnetite rich oxide melatroctolite are located between approximately 5,404,500 N and 5,405,200 N and occur to the north and south of the normal fault along the surface lineaments located near 5,404,900 N.

Oxide ultramafic intrusions frequently contain disseminated chalcopyrite and pyrrhotite and make up an important but very irregular component of the Marathon Series. The intrusions typically occur as discontinuous sills and irregular pods that crosscut Fine Grained Series, the WT sill, and the TDL Gabbro. The intrusions are less than 200 m in strike length and up to 100 m thick but are commonly a few to tens of metres thick and less than 50 m along strike. The size, irregular shape and mineralogy of these intrusions resemble the oxide ultramafic intrusions (“OUI”) that occur in the Duluth Complex (Ripley et al., 1998) and Sept Isles Intrusive Suite (Tollari et al., 2008).

The numerous cumulate phases and combinations thereof in oxide ultramafic intrusions are best described using the cumulate terminology of Miller et al. (2002). For example, the intrusive units vary in composition from oxide melatroctolite (FOCpA to FCOpA) to apatitic clinopyroxenite (CCoFAp to CCFoAp) to apatitic olivine clinopyroxenite (COFAp to OCFAp). Magnetite content varies from 5% in the clinopyroxenite to 25% in the oxide melatroctolite. Semi-massive or massive bands of magnetite are common and vary from 2 to 50 cm in thickness. Apatite is ubiquitous and varies in abundance from 5% to 30%. Massive apatite cumulate bands up to 30 cm thick are rare but found in apatitic clinopyroxenite.

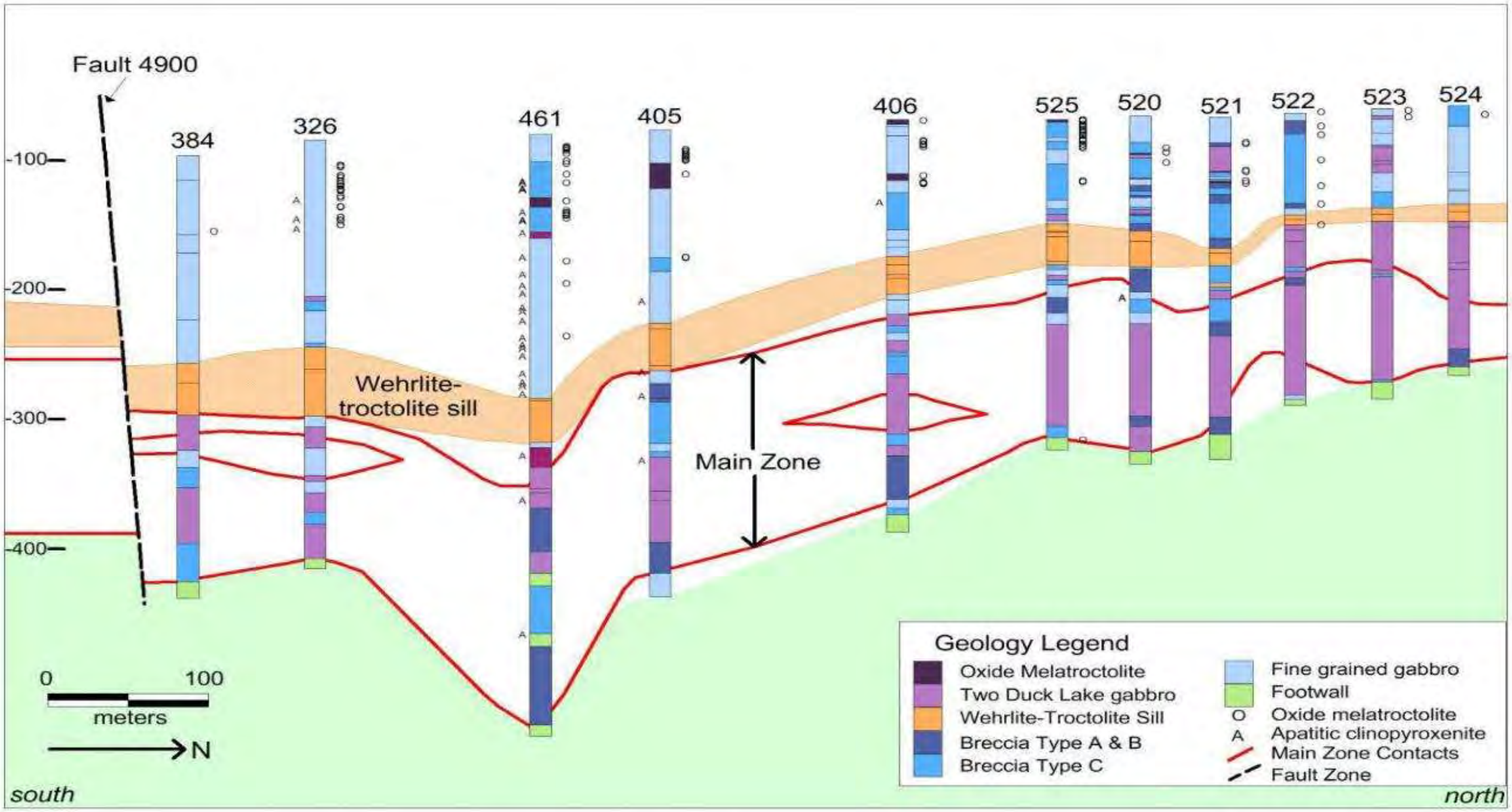
In general, these intrusions occur throughout the stratigraphy at the Marathon Deposit, however, units located high up in the stratigraphy are predominantly oxide melatroctolite and have higher overall magnetite content. These oxide melatroctolite intrusions are typically intermixed with plagioclase-rich gabbro bands (PcOf to PFoc), which display ophitic and (or) flow aligned textures. Units lower down in the stratigraphy are composed primarily of apatitic clinopyroxenite and apatitic olivine clinopyroxenite. Compositional zonation is not evident within the lower intrusions.

7.1.10 Breccia Units (Marathon Series)

The TDL Gabbro intruded along planes of weakness in earlier metabasalt and the Archean pyroclastic or rheomorphic footwall breccia to form numerous sills and intrusive breccias. Four types of intrusive breccias are recognized at the Marathon Deposit: Type A consists of TDL Gabbro matrix and angular xenoliths of fine grained series; Type B is similar to type A but also includes xenoliths of footwall material; Type C consists of metabasalt that is cut by multiple thin dykelets of TDL Gabbro, or higher up in the stratigraphic section, typically oxide melatroctolite; and type D consist of TDL Gabbro matrix and angular xenoliths of WT Sill only observed south of the 5,404,500 N fault. In general, the main body of TDL Gabbro progresses outward from a central uniform gabbro without xenoliths to breccia Type A and lastly to breccia Type C near the upper contact with metabasalt. Breccia Type B typically occurs along the basal contact, however, not always present. However, it should be noted that the distribution of breccia units is not regular, and reversals are common, as illustrated for example, by the distribution of breccia units down drill holes 461 and 514 in Figure 7.5.

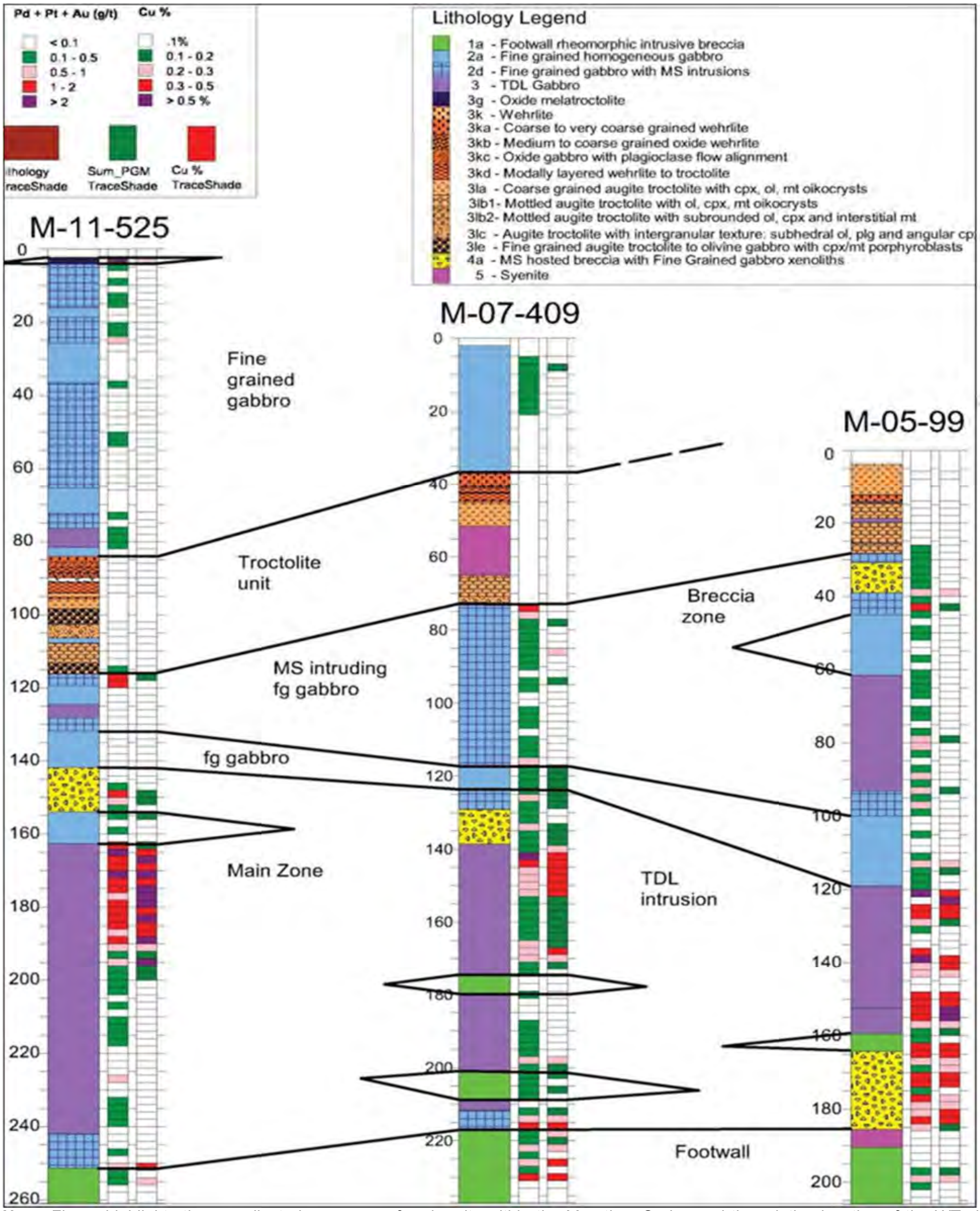
Breccia Types A, B, C and D typically contain sulphide-bearing TDL Gabbro, or higher up in the stratigraphy, sulphide-bearing oxide melatroctolite. Hence, breccia units are an important host rock for Cu-PGM mineralization.

Figure 7.5: Longitudinal View through the Central Portion of the Marathon Deposit (Looking West)



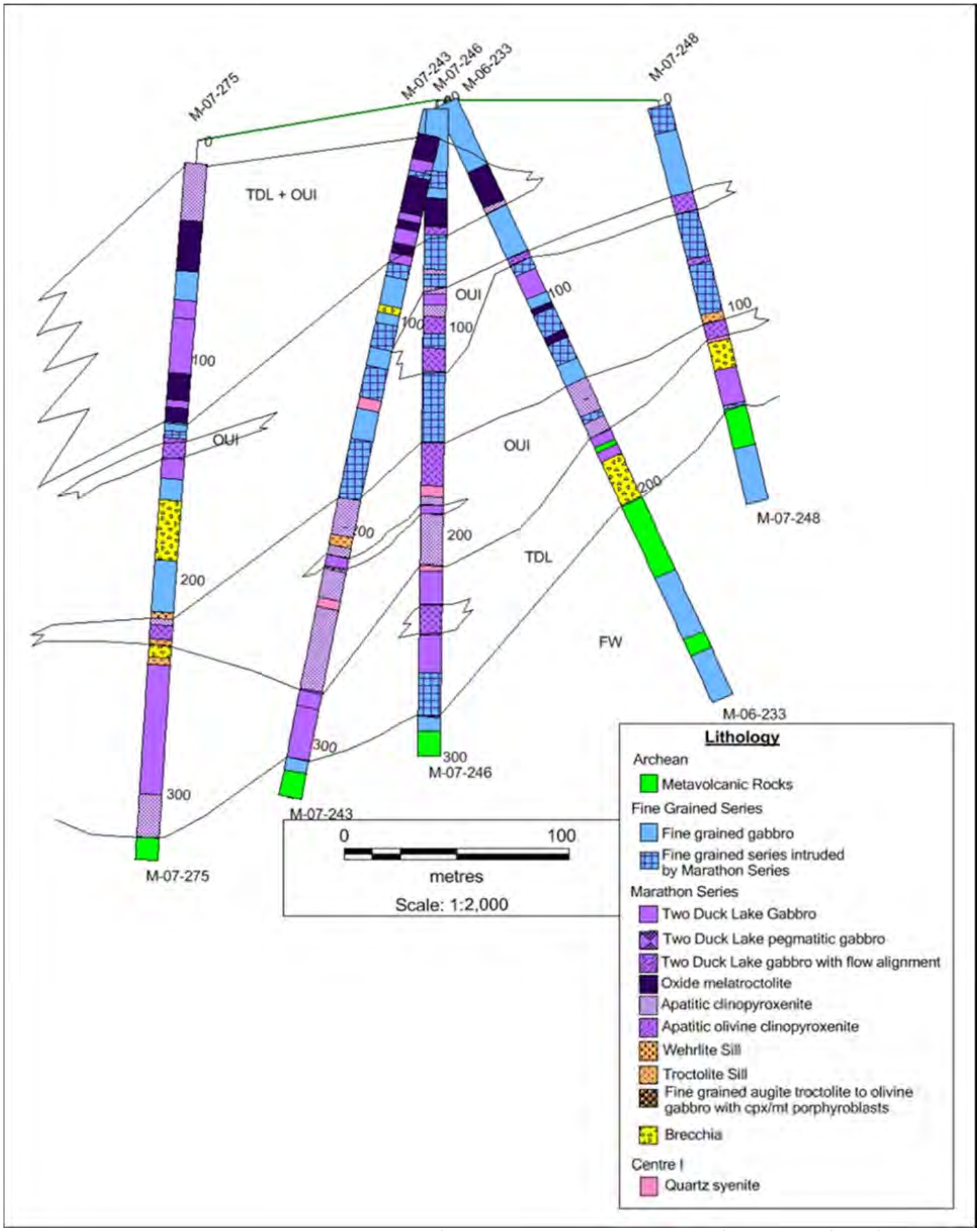
Note: *Figure highlights the complicated sequence of rock units within the Marathon Series and the relative location of the WT sill above the Main Zone of TDL Gabbro. Note the offset along the normal fault close to 5,404,900N. Note the distribution of apatitic clinopyroxenite immediately above the central portion of the Main Magma conduit as indicated by the position of hole M08-461. Hole numbers indicated without prefix example 525 is M-11-525. Note that for Figure 7.5, breccia Types A and B are described as Breccia with Marathon Series matrix, and breccia type C is described as Fine-grained gabbro with Marathon Series intrusions.*
Source: *Stillwater Canada (2014).*

Figure 7.6: Vertical Cross Section Through the Main Zone at Section 5,405,450 N (Looking North)



__Note:__ Figure highlights the complicated sequence of rock units within the Marathon Series and the relative location of the WT sill above the Main Zone mineralization. Note that hole M-11-525 is also located in the longitudinal projection in Figure 7.5.
Source: *Stillwater Canada (2012).*

Figure 7.7: Vertical Cross Section at 5,403,750 N (Looking North)



Note: *Figure shows the irregular but complicated nature of the oxide +/- apatite bearing ultramafic intrusions (OUI) of the Marathon Series.*
Source: *Stillwater Canada (2012).*

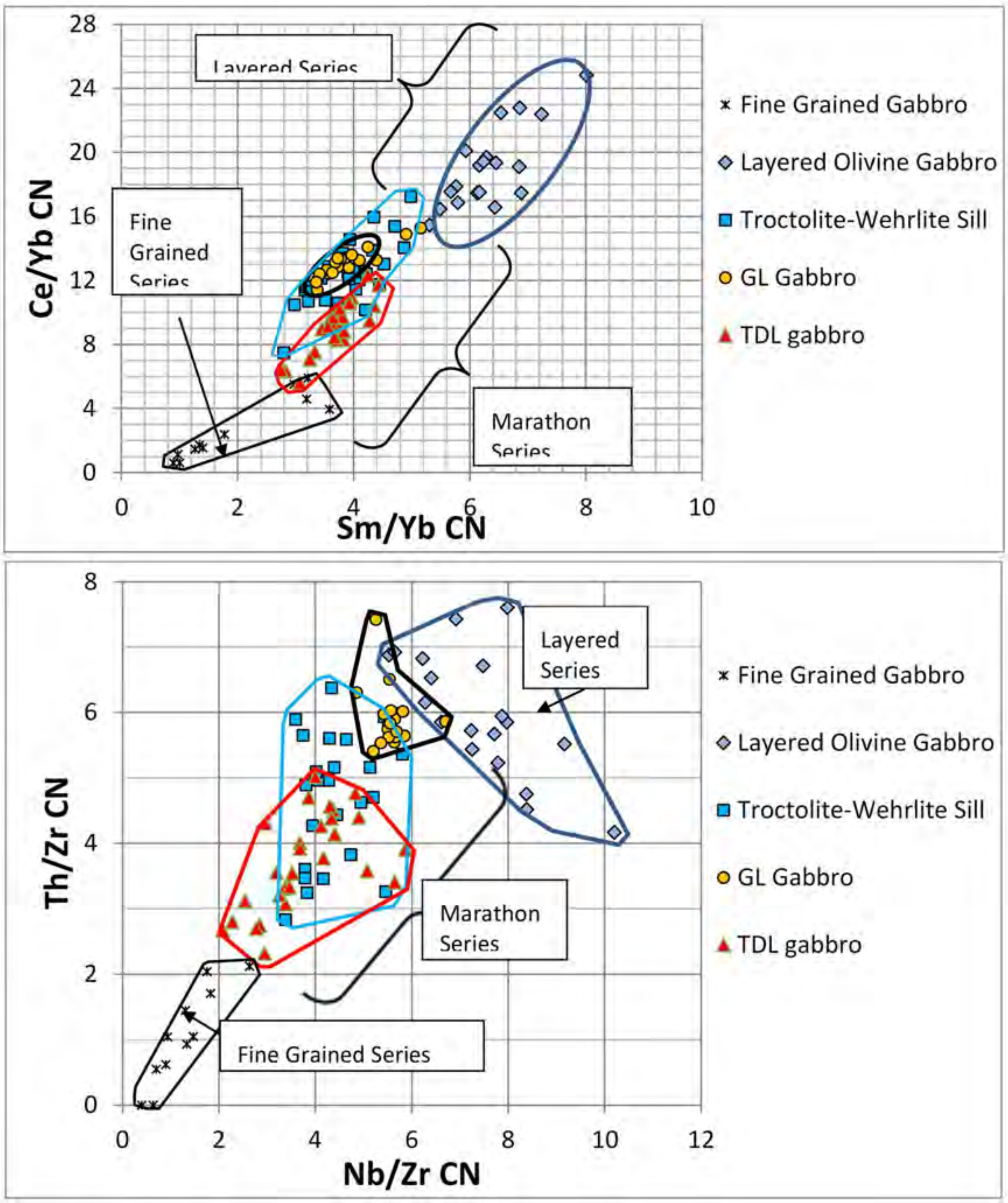
7.2 Geochemical Discrimination Diagrams for the Eastern Gabbro

Trace element data, together with cross-cutting relationships, provide clear evidence that the Eastern Gabbro is a composite intrusion. Each of the three magmatic series (Fine Grained, Layered and Marathon Series) previously characterized by textural, petrographic and cross-cutting relationships have recently been shown to have distinctive trace element signatures that can only be explained by intrusion of distinct magma types.

Pearce element diagrams (Figure 7.8) are very useful as discrimination diagrams because they neatly characterize the three intrusive series of the Eastern Gabbro into separate fields. In each figure, rock units of the Marathon Series plot in a field that lies between those for Fine Grained and Layered Series with the Fine Grained Series having lower Ce/Yb, Sm/Yb, Th/Zr and Nb/Zr and conversely, the Layered Series having higher Ce/Yb, Sm/Yb, Th/Zr and Nb/Zr (“Ce” = cerium, “Yb” = ytterbium, “Sm” = samarium, “Th” = thorium, “Zr” = zirconium, “Nb” = niobium).

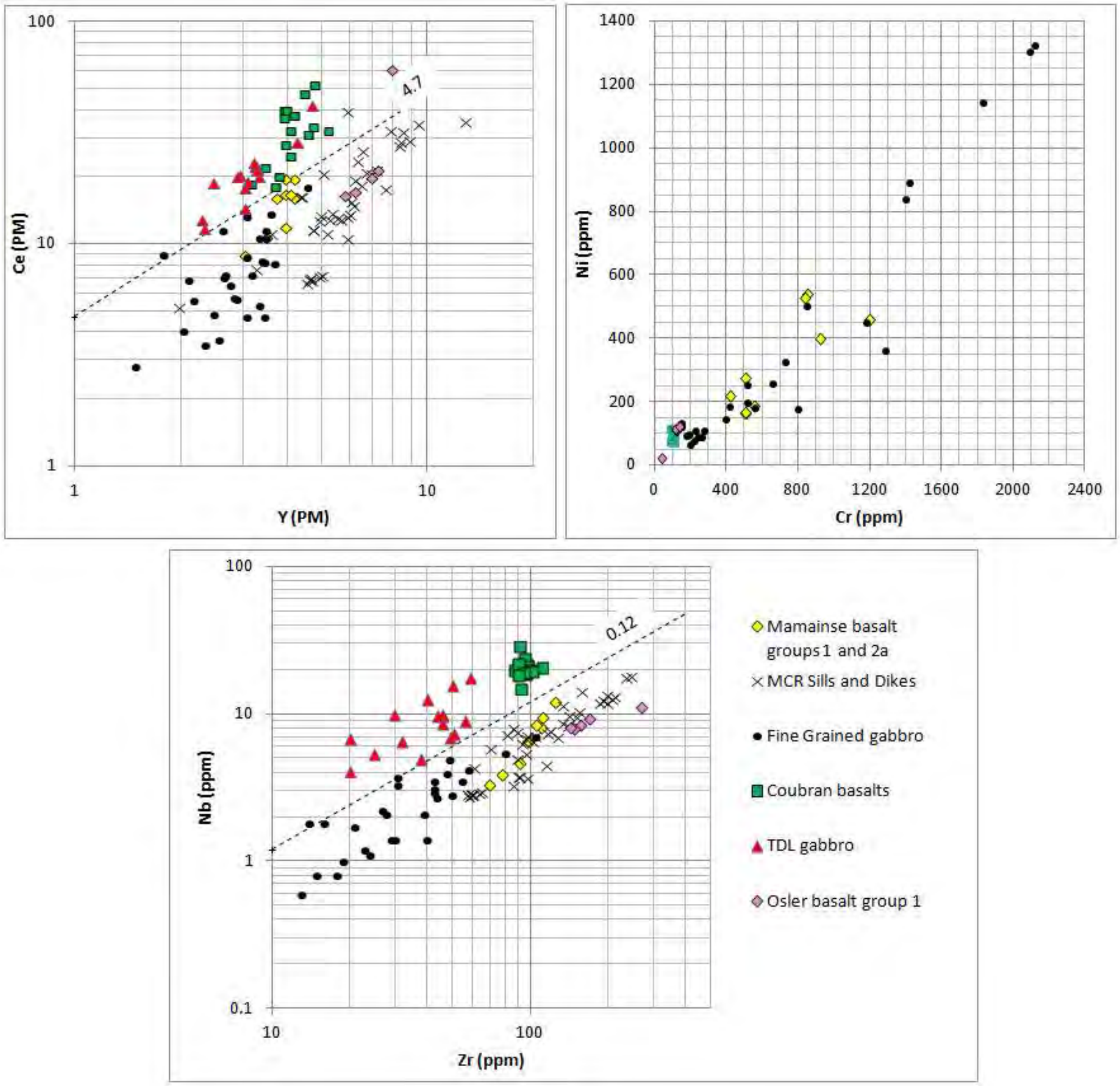
In Figure 7.9, three prominent units from the Coldwell Complex are compared to other Mid-continent Rift Systems (“MRS”) related intrusive and extrusive rock units located along the north shore of Lake Superior (Figure 7.1 and Figure 7.2). In Figure 7.9, the representative samples of TDL Gabbro are compared to Fine Grained Series, Coubran basalt and MRS related intrusive sills and dykes of the Logan and Nipigon Sills located near Thunder Bay, Ontario (after Hollings et al. 2011). It is interesting that the data for the Fine Grained Series overlie the fields for the Nipigon and Logan sills, whereas the rocks of the Marathon Series have somewhat higher Ce/Yb, Sm/Yb, Th/Zr and Nb/Zr. Since the Fine Grained series is the earliest intrusive phase in the CC, then the similarity of the Fine Grained Series to the Logan and Nipigon sills suggests that timing of the two events were simultaneous.

Figure 7.8: Pearce Element Ratio Diagrams for the Three Major Intrusive Suites in the Eastern Gabbro Suite



***Note**: These diagrams very nicely characterize the units into three groups that could be considered as least evolved (Fine Grained Series) to most evolved (Layered Series). Note the element in the denominator for axes on both figures is considered to be the least incompatible, respectively. Some data for TDL Gabbro after Ruthart (2013). Ratios are chondrite normalized after Sun and McDonough (1989).*
***Source**: Stillwater Canada (2012).*

Figure 7.9: Comparison of TDL Gabbro and Coubran Basalt to Intrusive and Extrusive Rocks of Mid-continent Rift



***Note**: Comparison of Coldwell Units (TDL Gabbro and basaltic flows north of Coubran Lake) to Mid-continent Rift related intrusive Sills (Nipigon sills) in the vicinity of Thunder Bay and basalt flows from Mamainse Point located along the eastern shoreline of Lake Superior and Osler basalt. Data for Nipigon Sills after Hollings et al. (2011), and Mamainse Point after Lightfoot et al. (1999). Some data for TDL Gabbro after Ruthart (2013). Ratios are chondrite normalized after Sun and McDonough (1989).*
***Source**: Stillwater Canada (2014).*

7.3 Mineralized Showings and Occurrences

7.3.1 Mineralized Zones

The Marathon Deposit consists of several large, thick and continuous zones of disseminated sulphide mineralization hosted within the TDL Gabbro (Figure 7.10 and Figure 7.11). The mineralized zones occur as shallow dipping sub-parallel lenses that follow the basal gabbro contact and are labeled as footwall, main, hanging wall zones and the W Horizon. The Main Zone is the thickest and most continuous zone. For 393 drill hole intersections with mineralized intervals greater than 4 m thick, the average thickness is 42.8 m and the maximum thickness is 205.1 m. Figure 7.11 and Figure 7.12 illustrate the location of the main mineralized areas on the Property.

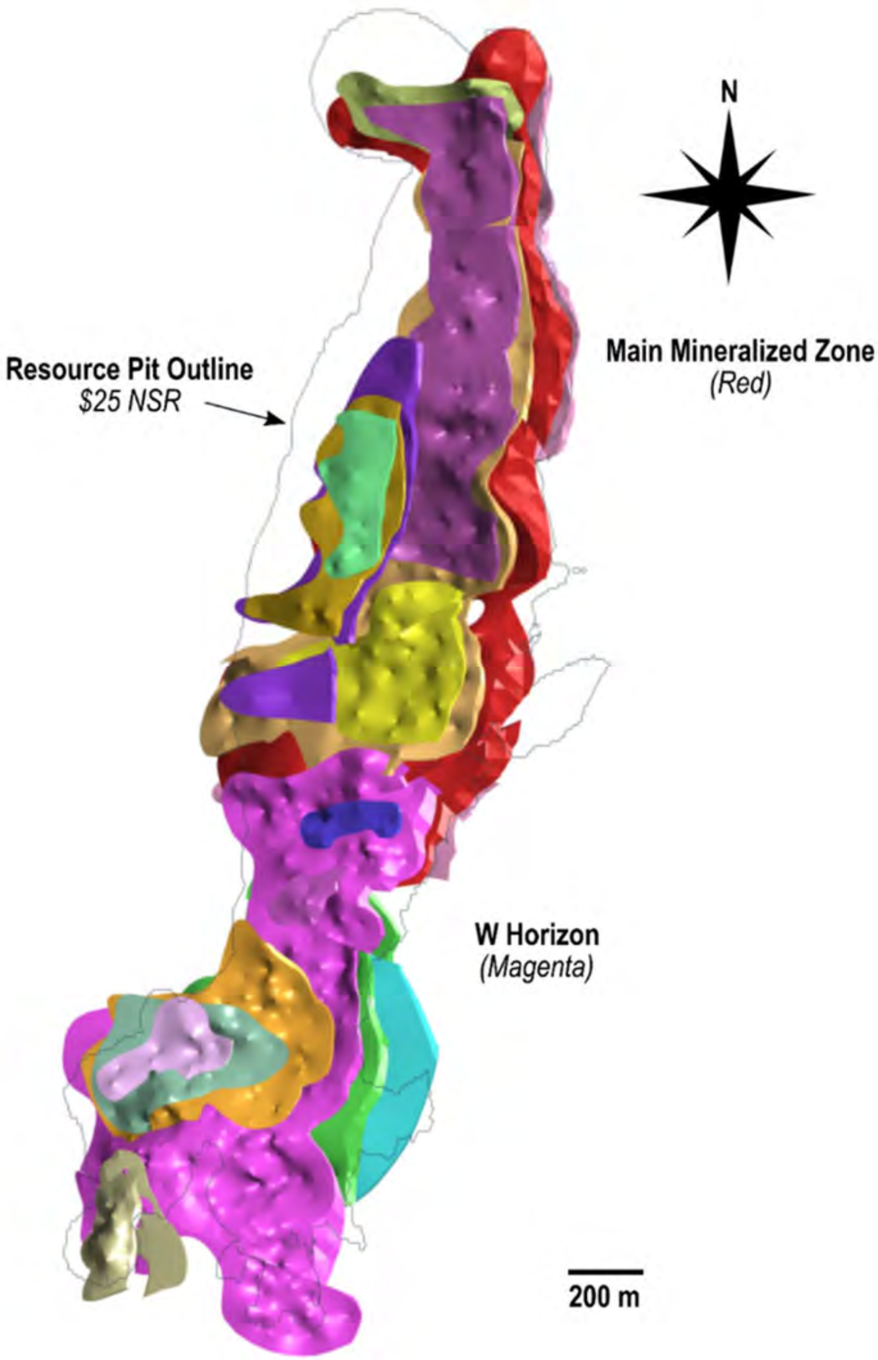
This section describes Cu and PGM occurrences located in the vicinity of the Marathon Deposit; for instance, the Geordie and Sally Deposits, and other occurrences located along the outer margin of the Coldwell Complex.

Each of these occurrences displays at least some of the many characteristics described at the Marathon Deposit. Given that these prospects share a common origin, then similarities between them are expected. However, in detail, there is much dissimilarity in the respective petrography or metal compositions that imply, for instance, that a dominant intrusive or mineralization forming process at one location might have played a minimal role at another. These factors are assessed at every locale and used to determine deposit significance and relevant exploration criteria.

Mineralized domains have been defined by drilling and 3-D modeling at several, but not all, locations. These mineralized domains are displayed with the Marathon Deposit in Figure 7.13. The figures are reproduced to the same scale in order to illustrate their relative size, and each body is oriented in their true position with north pointing toward the top of the page.

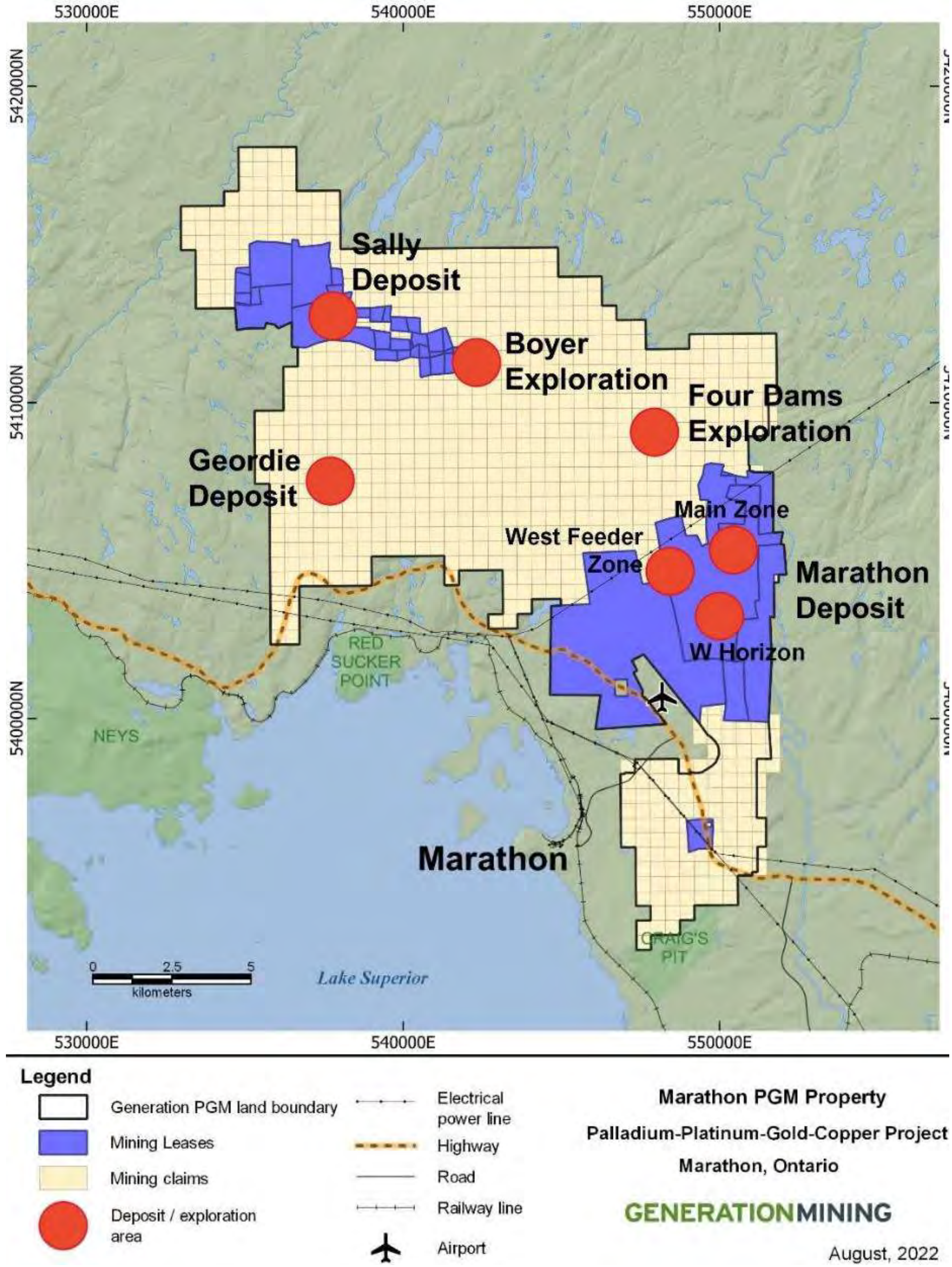
There are significant differences in the Cu and PGM abundances between the various deposits in the Coldwell Complex. These differences are best illustrated in the plot of Cu vs. Pd (Figure 7.14). For instance, the distribution of Cu and Pd at Sally closely matches the distribution observed at the Marathon Deposit. The abundance of Cu relative to Pd is much higher at Four Dams compared to other deposits. Samples such as those at Four Dams North have Cu/Pd ratios of 20,000 to 200,000 and greater than 200,000 at Four Dams South. The distribution of Cu and Pd at Geordie shows a strong positive correlation and the average Cu/Pd (6,500) is slightly higher than the average Cu/Pd at the Marathon Deposit (3,800). Similarly, at Redstone, there is a strong positive correlation and the average Cu/Pd (22,000) is greater than at either Geordie or Marathon.

Figure 7.10: Plan View of the Marathon Deposit Mineralized Zones



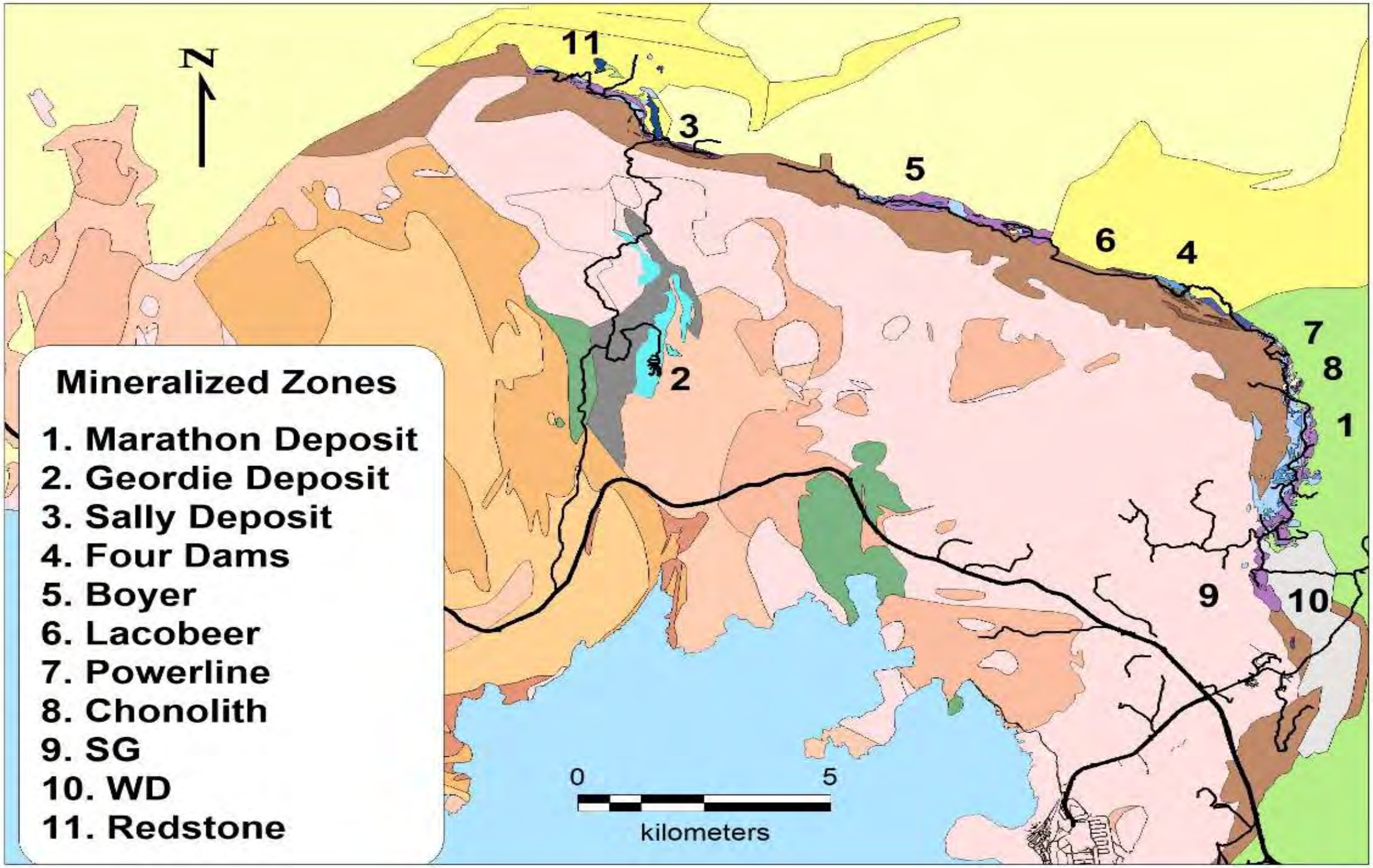
Source: *Gen Mining (2022)*

Figure 7.11: Locations of Mineralized Deposits and Those Areas Identified for Exploration



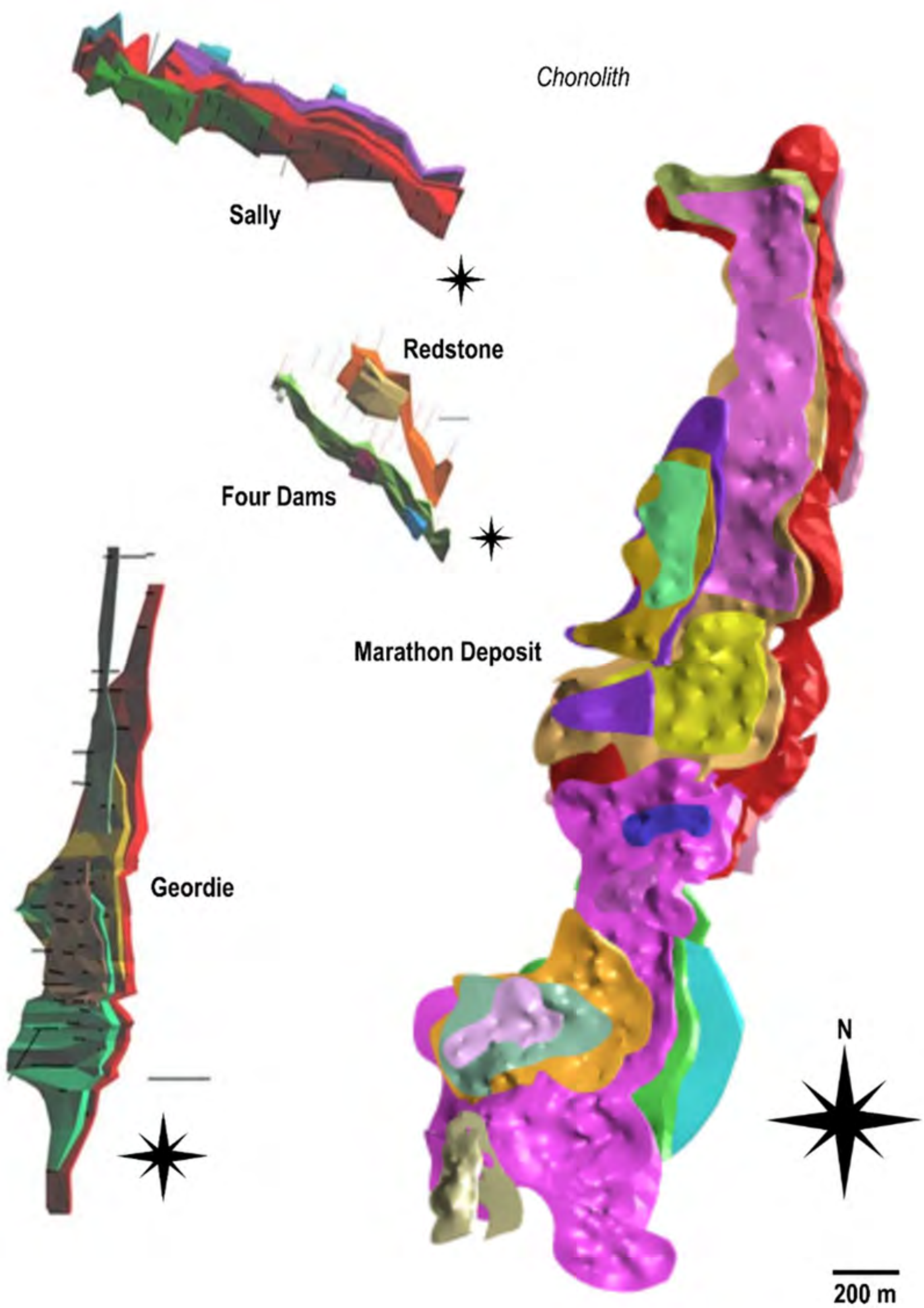
Source: *Gen Mining (2022).*

Figure 7.12: Geology Map of the Coldwell Complex and Location of all Known Cu-PGM Occurrences as of January 1, 2020



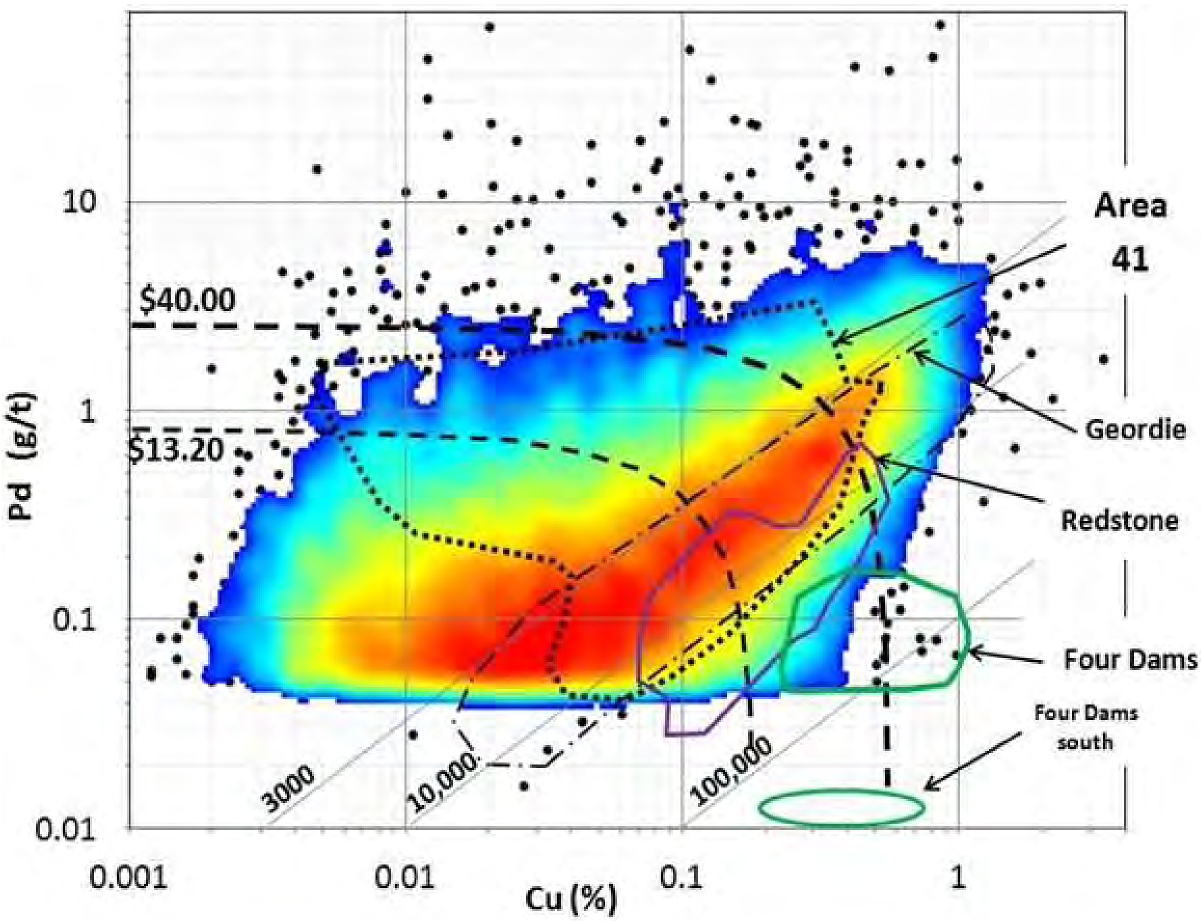
Source: *Gen Mining (2019).*

Figure 7.13: Scaled 3-D Models of Coldwell Mineralized Domains Compared to Marathon Deposit



Note: *The scaled 3-D models are oriented correctly with north pointing up as shown by individual north arrows. Trace of drill holes at each location except for the Marathon Deposit are indicated by faint grey lines.*
Source: *Gen Mining (2023).*

Figure 7.14: Comparison of Cu vs. Pd for Coldwell Complex Deposits



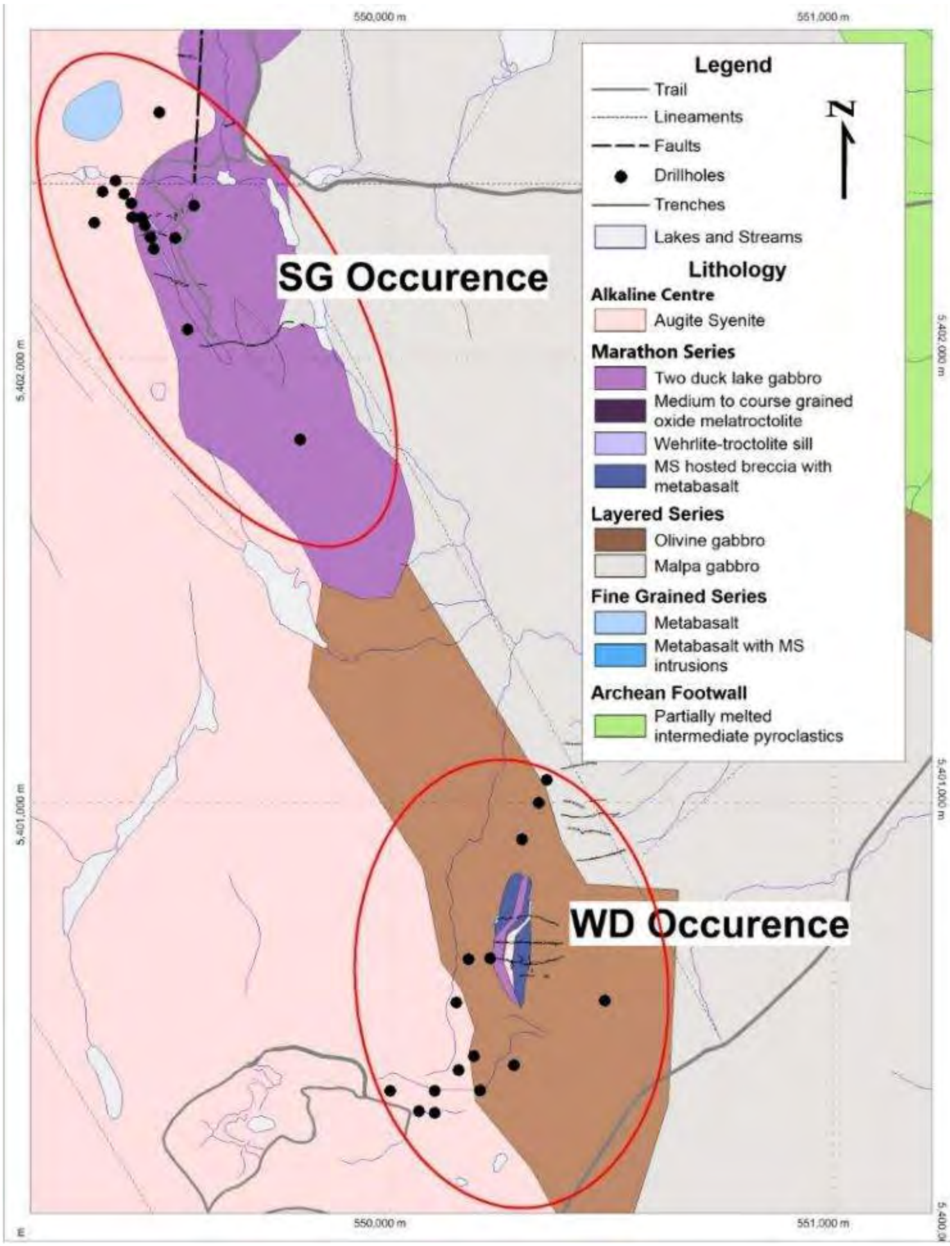
Notes: *The coloured contours represent the point density map for Marathon Deposit assays (black dots). Fields for assays from other occurrences are represented by individual curves. Dashed curves labelled as $13.20 and $40.00 represent calculated NSR $/t values using the 2010 Mineral Resource estimate metal prices and process recoveries. Diagonal blue lines represent constant Cu/Pd values, for example 3,000.*
Source: *Micon (2010).*

7.3.2 <u>SG and WD Occurrences</u>

The SG and WD occurrences are located south of the Marathon Deposit as shown in Figure 7.12 and Figure 7.15. These zones are hosted by TDL Gabbro, but unlike at the Marathon Deposit where mineralization occurs directly above the footwall, mineralized TDL Gabbro at the SG and WD zones occur along the west margin of the Eastern Gabbro close to the contact with the overlying Augite Syenite. The depth to footwall and nature of the contact in this area are unknown.

The change in stratigraphy south of the Marathon Deposit is interpreted to be related to faulting at 5,402,350 N resulting in the footwall offset to the east by approximately 2 km. A southeast trending fault connects the SG and WD zones; both zones also encompass additional converging faults (Figure 7.15). The area between these two zones lacks exploration due to thick overburden which makes prospecting, trenching and drilling difficult.

Figure 7.15: Lithology Map Showing the SG and WD Occurrences



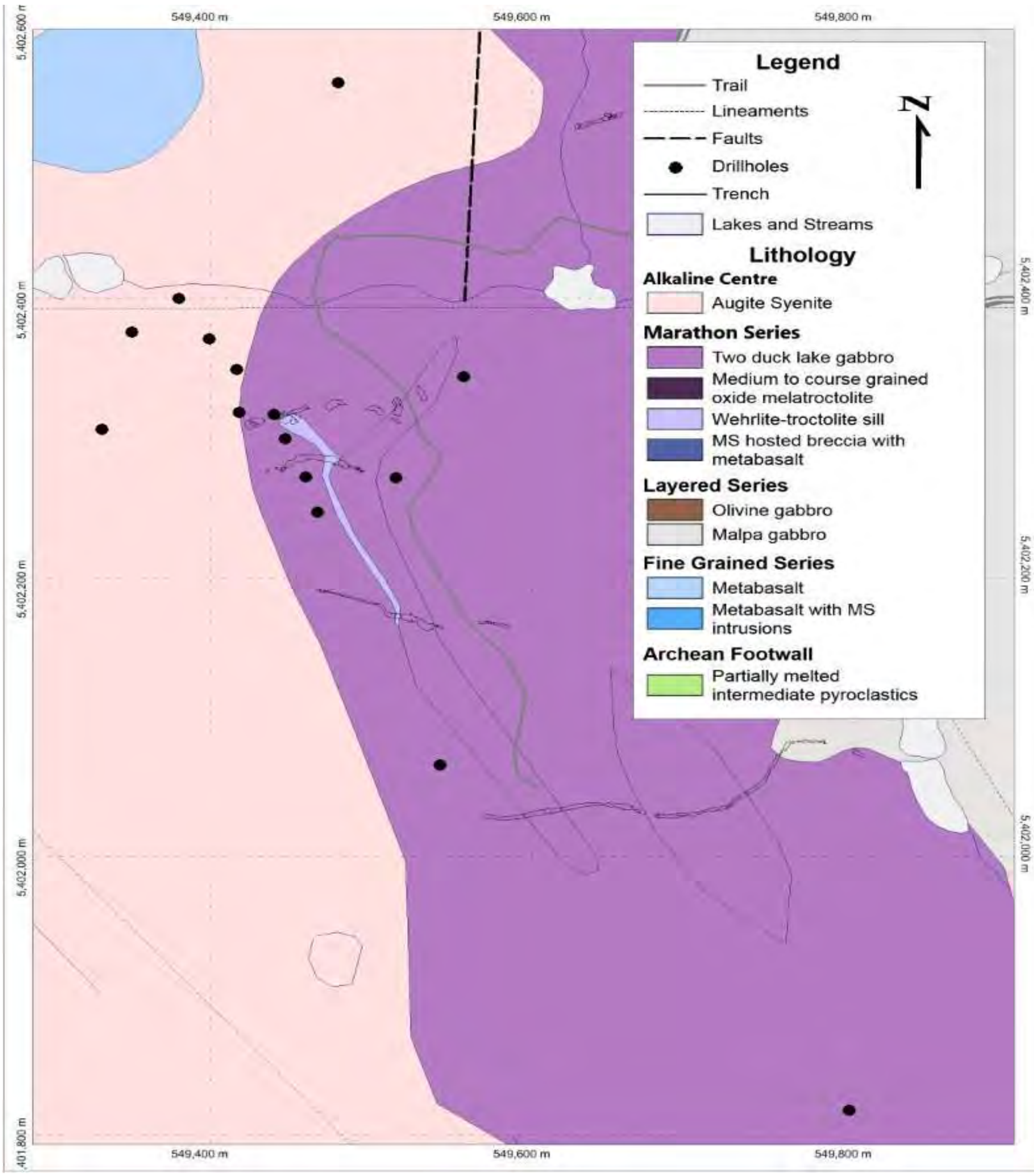
Source: *Gen Mining (2021).*

7.3.2.1 SG Zone

The SG Zone is characterized by near-surface mineralization in TDL Gabbro (Figure 7.16), similar to that at the Marathon Deposit. Previous work included 16 drill holes, 56 grab samples and 600 m of outcrop stripping. The mineralized zone has a strike of 160° to 170°, dips at 30-45° west and extends for 120 m along strike.

The SG Zone includes a thick sequence of TDL Gabbro. Mineralization typically occurs in zones where TDL Gabbro is intermixed with lenses of oxide ultramafic rocks. The best drill hole intersection to date is shallow with an average grade of 1.33 g/t total PGM and 0.27% Cu over 18 m.

Figure 7.16: SG Occurrence Showing Lineaments, Trenches and Drill Holes

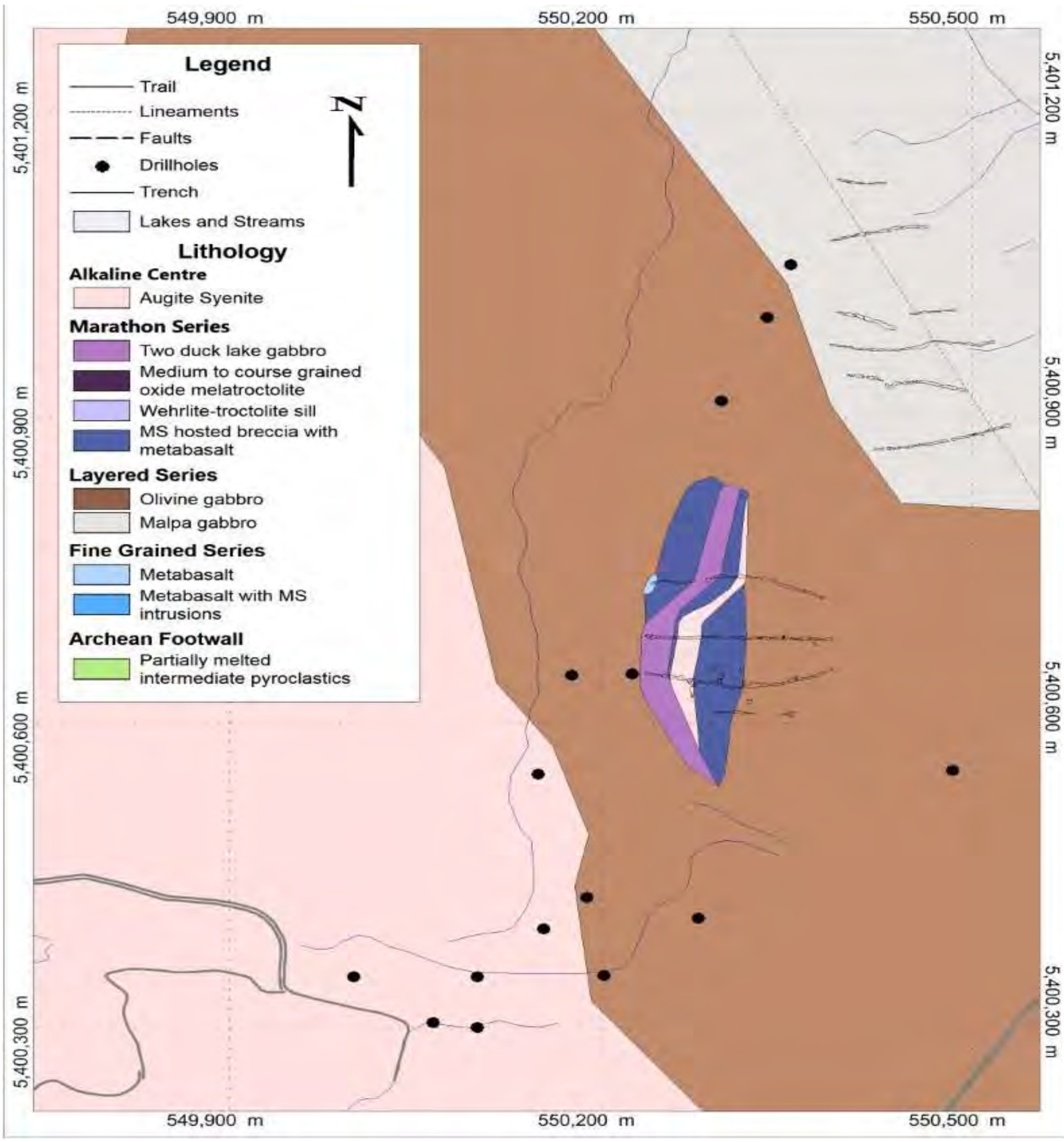


Source: *Gen Mining (2021).*

7.3.2.2 WD Zone

The WD Zone is located southeast of the SG Zone (Figure 7.17). Previous work included 15 drill holes, 1,000 m of outcrop stripping and channel sampling, and 48 grab samples. Mineralization in this area occurs at two stratigraphic positions: TDL Gabbro and Layered Series Gabbro. These two mineralized zones are easily classified using Cu/Pd ratios. The Cu/Pd ratio for mineralization in the Layered Series is much higher than for mineralization in the TDL Gabbro owing to the negligible Pd values and higher average copper content in the Layered Series rocks. Strike length for the mineralized zones is 100 m in the Layered series and 150 m in the TDL Gabbro. Both zones are open to the north. All mineralization strikes north-south. The Marathon Series mineralization dips steeply west at 70°. Dip for Layered Series mineralization is shallow at 45° west.

Figure 7.17: WD Occurrence Showing Lineaments, Trenches and Drill Holes



Source: *Gen Mining (2021).*

7.3.3 Biiwobik Prospect

7.3.3.1 Chonolith Zone

The Chonolith Zone is a continuous extension of the north end of the Main Zone, as confirmed by drilling in 2021. In general, the Main Zone follows the footwall contact north along the edge of the proposed Main pit and at 5,406,300 N changes direction and continues down dip to the west. The mineralization continues for 350 m west before turning north where it is interpreted to connect to a 200 m deep channel of mineralization referred to as the Chonolith (Figure 7.18). The Chonolith Zone is up to 120 m thick and begins in the north at a depth of 200 m. The north-south trending section of the Chonolith is 500 m long and has been cut by 10 drill holes, including six holes drill by Gen Mining in 2021. The best intersection in this area returned 1.3 g/t total PGM and 0.6% Cu over 95 m. The section of the Chonolith that strikes west and connects with the Main Zone inside the proposed open pit was intersected by a total of 10 drill holes. The best intersection in this area was 1.28 g/t total PGM and 0.41% Cu over 50 m.

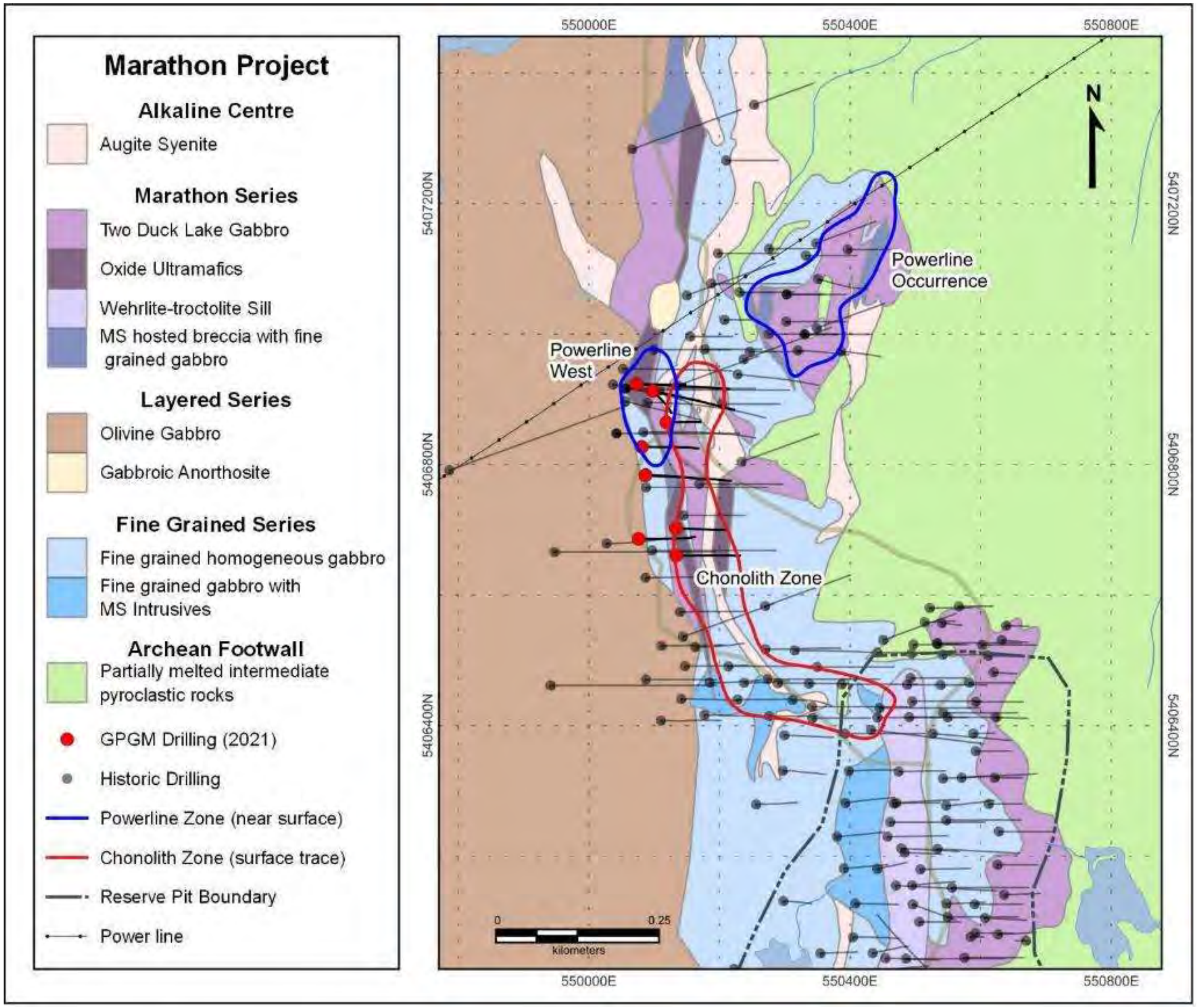
7.3.3.2 Powerline West

The Powerline West Occurrence directly overlies the Chonolith Zone and is interpreted to be the western extension of the main Powerline Occurrence that occurs 200 m to the east. The mineralization is near surface (<100 m vertical depth) and hosted predominantly within a mix of oxide rich ultramafic pods and fine grained to pegmatitic TDL Gabbro. The best intersection to date returned 1.78 g/t PGM and 0.46% Cu over 46 m.

7.3.3.3 The Powerline Occurrence

The Powerline Occurrence, located northeast of the Chonolith Zone, consists of a flat lying bowl-shaped body of TDL Gabbro that sits in a trough in the footwall (Figure 7.18). The Chonolith Zone and Power Line Occurrence are separated by a shift in the footwall to the east and a syenite dyke. The Power Line Zone consists of multiple lenses, including intervals such as 0.44 g/t total PGM and 0.2% Cu over 18 m.

Figure 7.18: North End of the Marathon Deposit Showing the Biiwobik Prospect including the Chonolith and Power Line Occurrences



Source: *Gen Mining (2023).*

7.3.4 Geordie Deposit

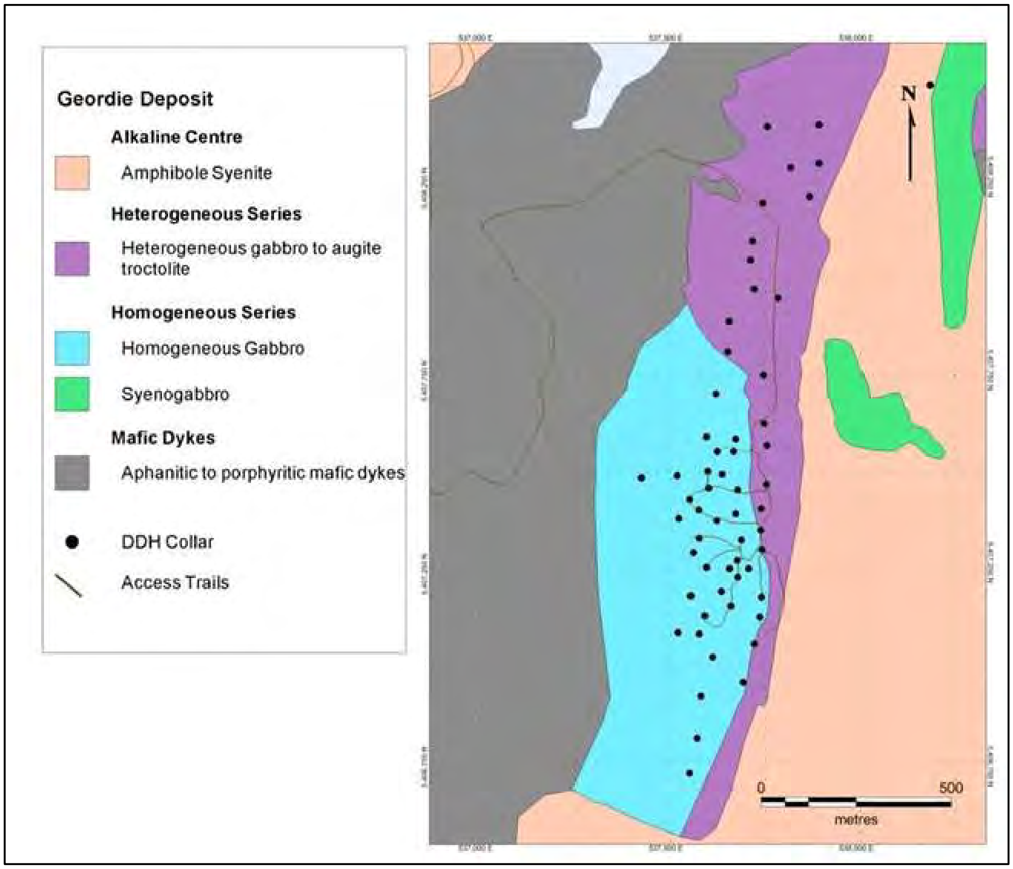
The Geordie Deposit is located near the centre of the Coldwell Complex (Figure 7.12). Mineralization occurs along the base of the Geordie Intrusion, a large, layered gabbro with a basal zone of heterogeneous augite troctolite and gabbro. A simplified geology map of the Geordie Deposit is shown in Figure 7.19 and a cross-section through the middle of the deposit is shown in Figure 7.20.

Exploration on the Geordie Deposit includes 69 diamond drill holes totalling 12,234 m, trenching, mapping, magnetic and radiometric airborne survey and soil sampling.

The sulphides consist predominantly of chalcopyrite and bornite, and minor pyrite, millerite, cobaltite, siegenite, sphalerite and galena. Sulphides are disseminated with angular to blebby grain shapes. Thin veins of chalcopyrite occur near the base of the intrusion and in the underlying syenite. The mineralization occurs within a thick continuous basal zone that dips 45 to 60° and traced over a strike length of 1.7 km. Minor thin discontinuous zones occur higher up in the stratigraphy.

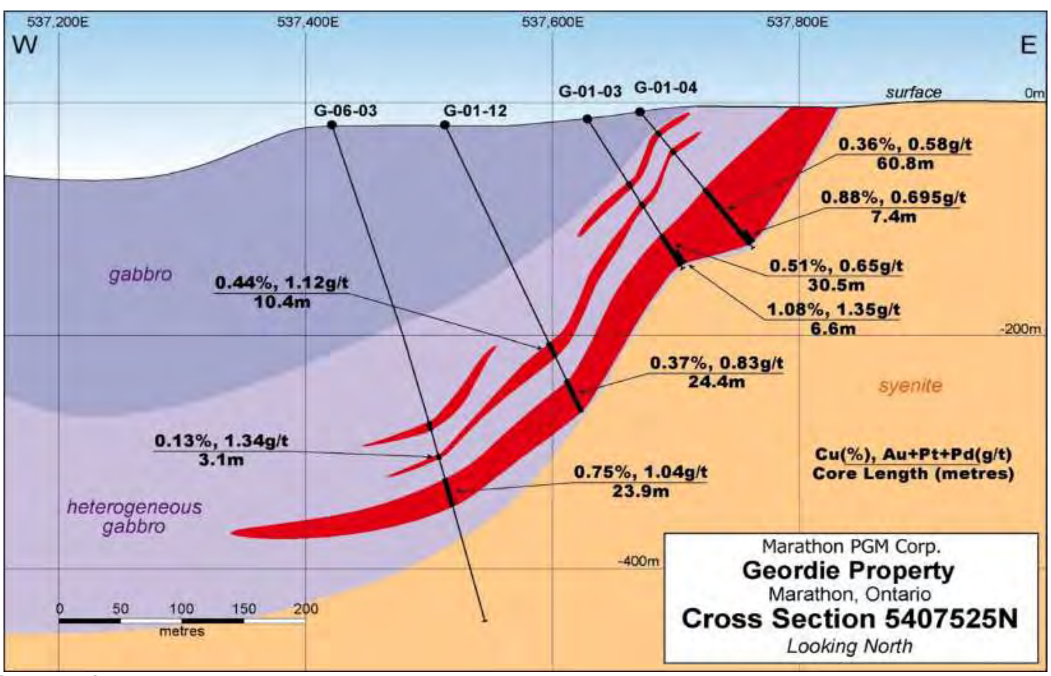
Drilling has outlined a series of sub-parallel mineralized zones within the gabbroic/troctolite body. Mineralization is mainly chalcopyrite with lesser amounts of bornite, pyrite, magnetite, and supergene chalcocite. Associated with concentrations and disseminated grains of chalcopyrite are a wide variety of PGM and precious-metal tellurides, bismuthinites and alloys. The abundance of Pt is very low; however, for samples with greater than 45 ppb Pt or Pd (three times the detection limit of Pd) the average Pd/Pt is 11. There is a strong positive correlation between Cu and Pd and the average ratio for Cu/Pd is 6,500.

Figure 7.19: Geologic Map of the Geordie Deposit



Source: *Stillwater Canada (2014).*

Figure 7.20: Vertical Cross Section at the Geordie Deposit (Looking North)



Source: *Stillwater Canada (2014).*

7.3.5 Four Dams Prospect

The Four Dams Prospect is located 4 km northwest of the Marathon Deposit on the northern rim of the Coldwell Complex (Figure 7.12). Four Dams is subdivided into three mineralized zones, as follows: Four Dams North, Four Dams South and Lacobeer Lake (Figure 7.21).

The Four Dams North mineralization occurs in a 100 m thick lens of Marathon Series ultramafic rocks that strikes northwesterly for 350 m and dips 60° to the southwest. The intrusion has a thin marginal zone of melagabbro and a core of apatitic clinopyroxenite to apatitic wehrlite.

Sulphides in the Four Dams North Zone include disseminated to blebby chalcopyrite with lesser pyrrhotite and trace bornite. The mineralization includes intervals such as 0.16 g/t PGM and 0.39% Cu over 74 m, and 0.23 g/t PGM and 0.40% Cu over 85 m. Higher PGM grades occur in the central apatitic wehrlite zone.

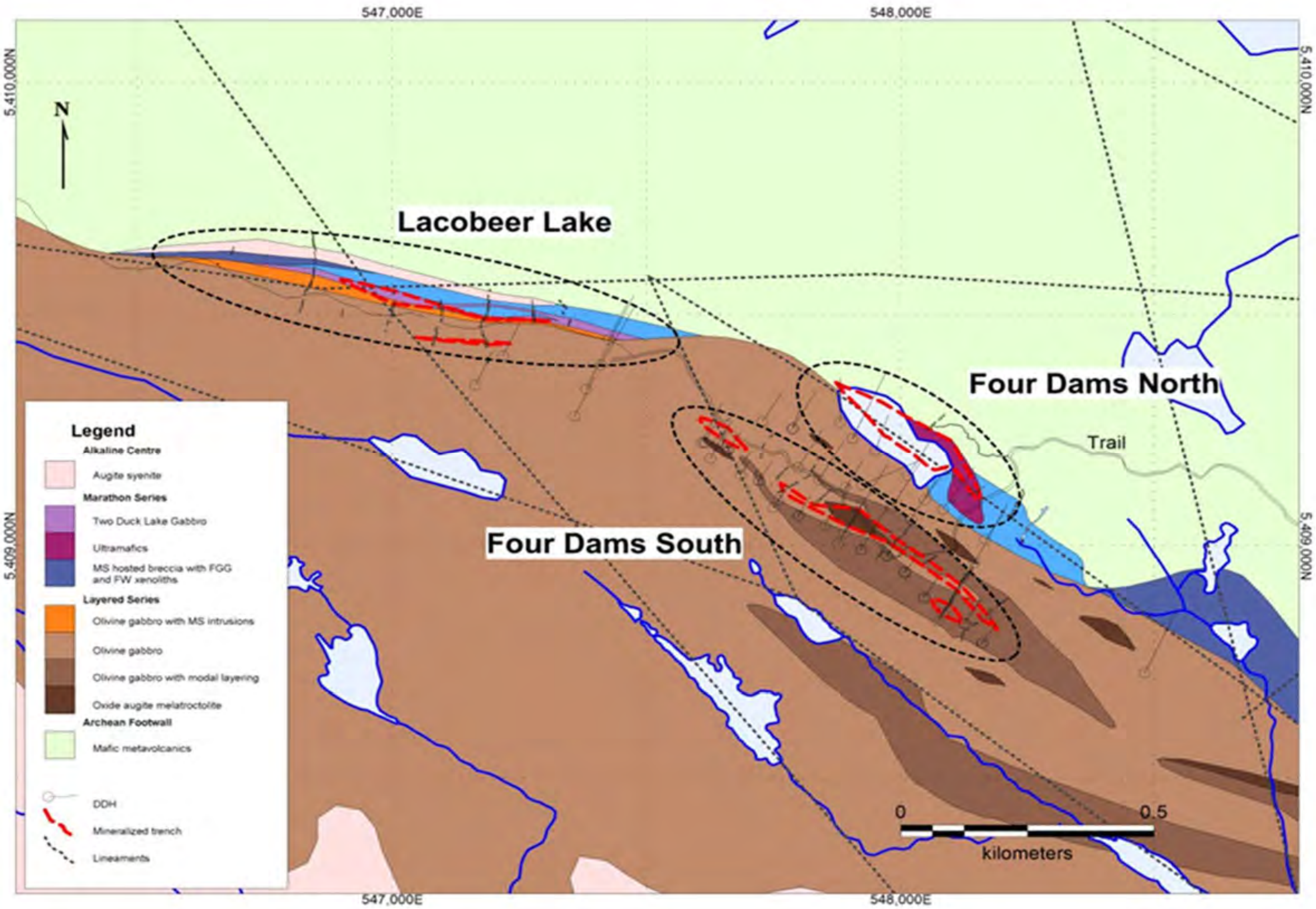
The Four Dams South mineralization is hosted by the Layered Series rocks, located approximately 150 m south of the Four Dams North mineralization. The mineralization occurs in homogeneous or modally layered olivine gabbro inter layered with magnetite rich lenses.

The Four Dams South Zone is continuous for 700 m along strike, dips 40° to the southwest, and pinches and swells from thicknesses of up to 50 m down to 4 m. The Zone was defined by 32 short diamond drill holes in 2013 with a best intersection of 0.33% Cu over 48 m. It contains only trace Pd.

The sulphide minerals consist of fine to medium grained disseminated pyrrhotite and chalcopyrite, which are associated with actinolite and albite alteration. The Four Dams South mineralization is believed to be a result of hydrothermal remobilization.

The Lacobeer Zone is poorly defined owing to thick overburden. Work to date includes five trenches with only one of them intersecting mineralization. The zone is inferred to be a maximum of 25 m thick on surface with complicated textural relationships within Marathon Series gabbros. Best grab samples from prospecting included 2.6 g/t PGM and 0.53% Cu.

Figure 7.21: Three Mineralization Zones at Four Dams



Note: *Mineralized surface zones were determined using projected drill hole data (Four Dams North) and surface sampling.*
Source: *Micon (2010).*

7.3.6 Sally Deposit

The Sally Deposit occurs along the northern margin of the Eastern Gabbro (Figure 7.12). The Sally Deposit strikes east-southeast, dips at 45-50° south and extends for over 1.2 km along strike. The Sally Deposit is open to the east and west. P&E completed an initial Mineral Resource estimate of the Sally Deposit in 2019, which is presented in Subsection 14.2.13 of this Technical Report.

A total of 65 holes have been drilled in the Sally Deposit area, of which 48 were drilled into Sally Main Zone (Figure 7.22). The drilling at Sally Main Zone is considered to be sufficient to define the thickness and continuity of the mineralized envelope. Closer spaced drilling will be required to define and characterize zones of higher-grade material.

Drilling has thus far intersected four main mineralized horizons at Sally, referred to in descending order from top to bottom, as Zones 1 to 4 (Figure 7.23):

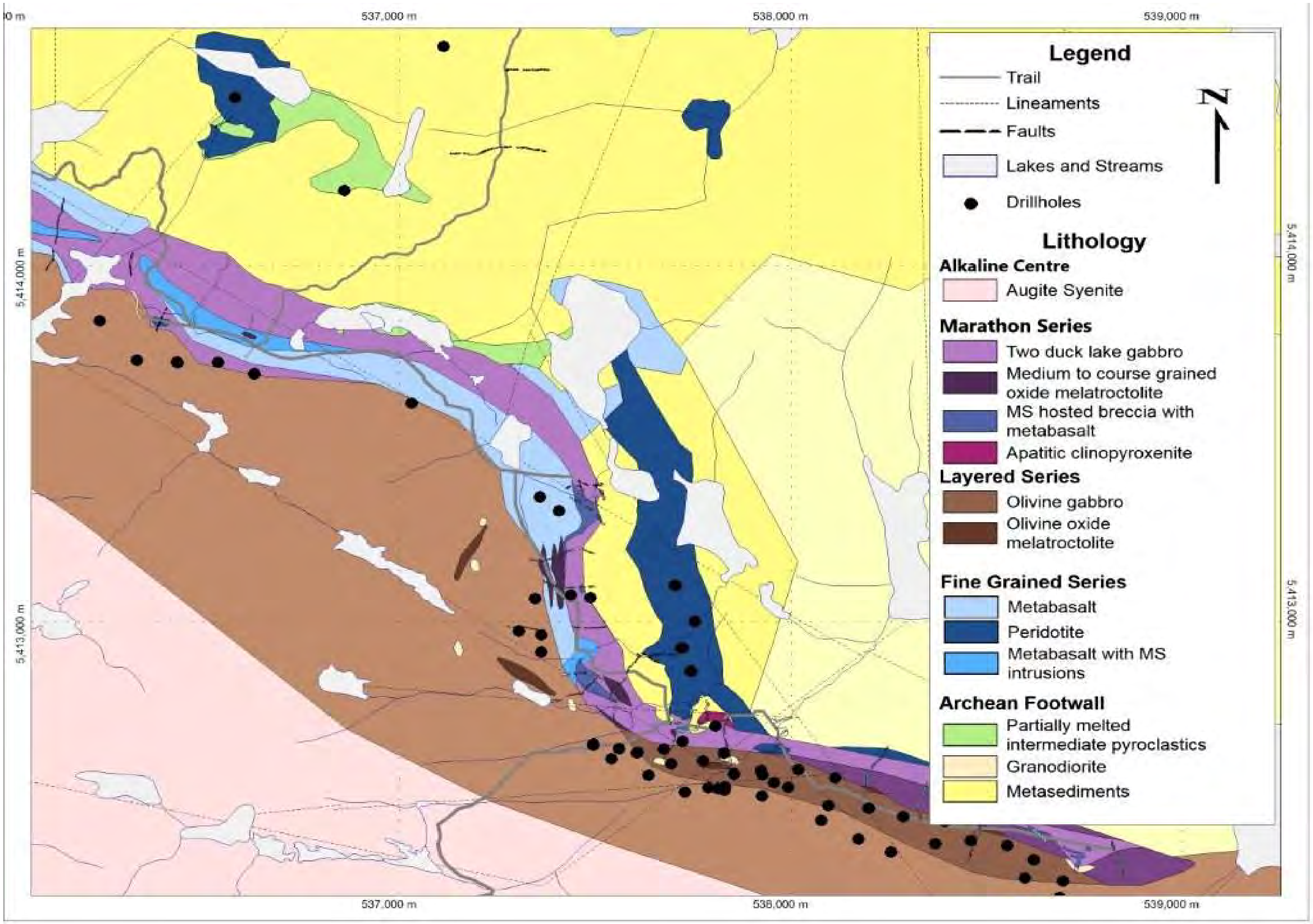
Zone 1: The uppermost mineralized zone contains Cu and trace amounts of Pd, and is commonly less than 10 m thick. Zone 1 is hosted by fine grained early intrusion TDL Gabbro that is intermixed with Marathon Series oxide melatroctolite.

Zone 2: The second mineralized zone is hosted by TDL Gabbro and clinopyroxenites that generally includes xenoliths of the Fine Grained Series. This second mineralized zone is typically 40 to 50 m thick and contains some of the highest Pd grades in the Sally Deposit, particularly at the contact between the Marathon Series (Breccia unit A) and the peridotite unit of the Fine Grained Series. Grab samples include sample K008054 which returned 188.3 g/t PGM+Au and 9.11% Cu.

Zone 3: Zone 3 occurs below the peridotite unit and is referred to as the Main Zone because it is normally over 40 m thick and is the most continuous over the strike length of the Sally Deposit, except at the far west end where mineralization is cut by multiple faults. The mineralization is hosted by TDL Gabbro.

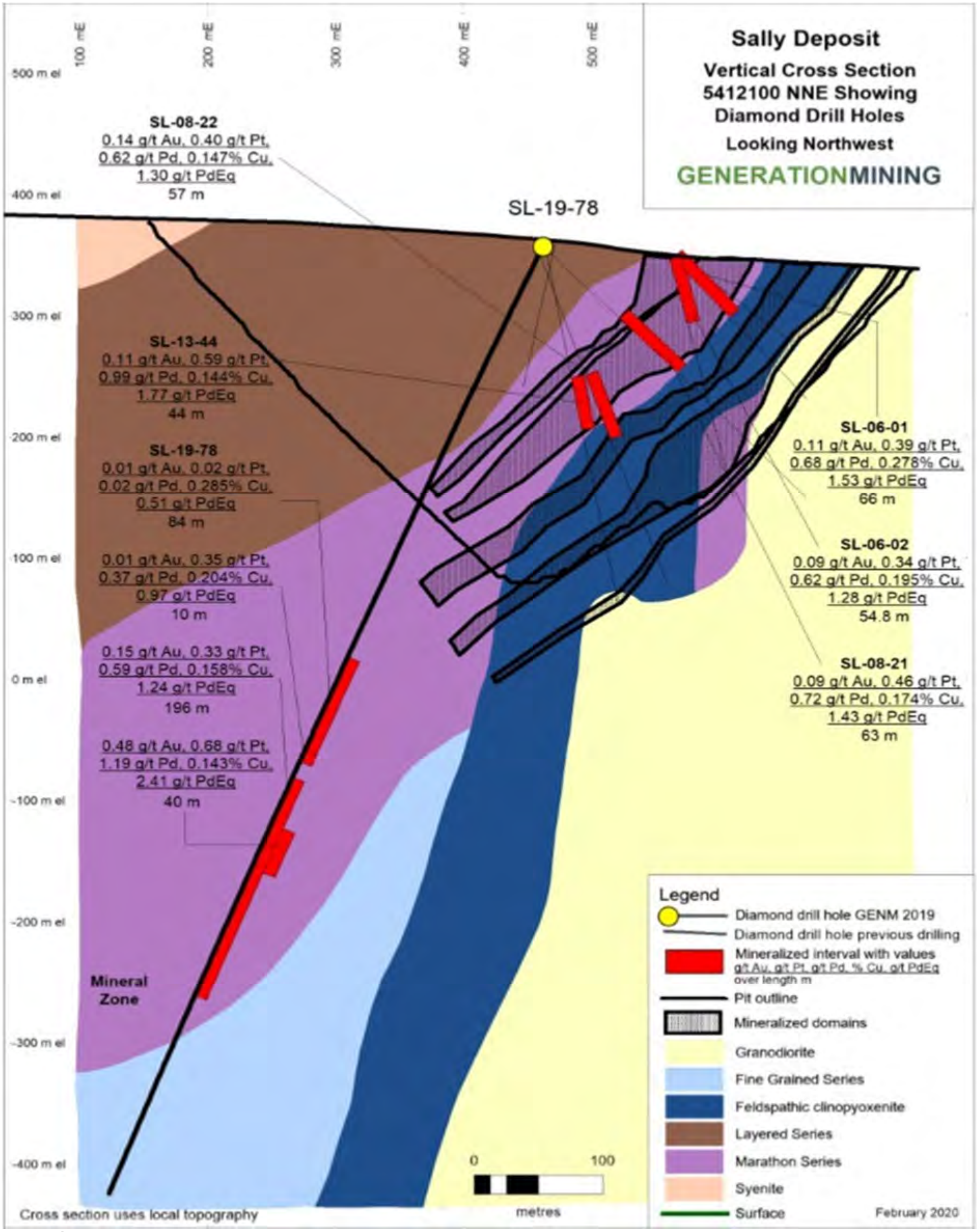
Zone 4: Zone 4 occurs below the Main Zone, where Fine Grained Series and/or Archean footwall are crosscut by Marathon Series intrusions. Mineralization contains Cu and Pd values that are similar to the Main Zone, but has increased pyrrhotite content, and thus is considered to be lower tenor.

Figure 7.22: Geology Map of Sally with Drill Hole Collars



Source: *Gen Mining (2021).*

Figure 7.23: Vertical Cross Section of Sally Showing Stratigraphy of Geological Units and Mineralization



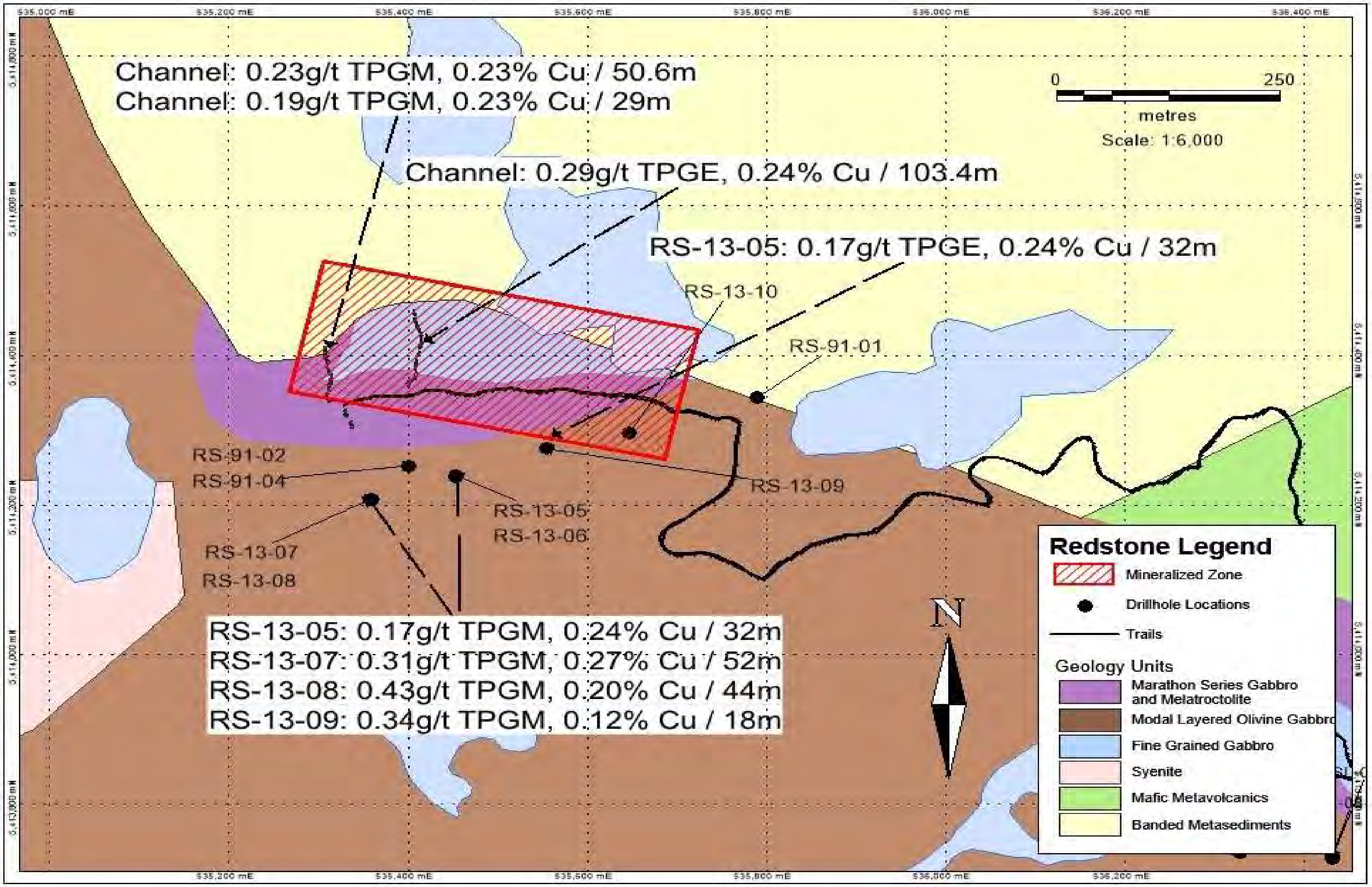
Source: *Gen Mining (2020).*

7.3.7 Redstone Prospect

The Redstone Prospect is situated along the outer margin of the Eastern Gabbro in the northwest corner of the Caldwell Complex (Figure 7.12). The mineralized zone strikes near east-west, dips between 30° and 45° south and is continuous along strike for 450 m (Figure 7.24). The zone extends down dip for at least 200 m and is open to the west.

The mineralization consists of disseminated chalcopyrite, pyrrhotite and trace bornite hosted in a complicated assemblage of Marathon Series rocks. The upper portion of the sequence is dominated by oxide melatroctolite with minor TDL Gabbro and the lower zone is composed predominantly of Marathon Series breccia units. The lower breccia units are composed of TDL Gabbro intermixed with oxide melatroctolite and numerous xenoliths of the Fine-Grained Series and/or metavolcanic footwall.

Figure 7.24: Geology of the Redstone Prospect with 2013 Drill Hole and Surface Channel Assays



Source: *Stillwater Canada (2014).*

7.3.8 The W Horizon

The W Horizon forms a nearly continuous sheet of mineralization that strikes north-south for 1.5 km from section 5,403,100 N to section 5,404,600 N and continues down dip for over 700 m. The zone is open at depth. It ranges in thickness from 0.40 to 108.0 m and occurs near the top of the mineralized zones. The zone is difficult to identify in drill core because it commonly contains only trace sulphides (chalcopyrite and bornite). Continuity of the W Horizon between drill holes is shown by minimum PGM abundances of 1 g/t and by Cu/(Pt+Pd) ratios less than 3,500.

Several very high-grade lenses ranging from 30 to 200 m in length occur within the W Horizon. The best intersections to date included 107 g/t PGM+Au, 1.04 g/t Rh and 0.02% Cu over 2 m (hole M07-239) and 45.2 g/t PGM+Au and 0.49% Cu over 10 m (hole M07-306).

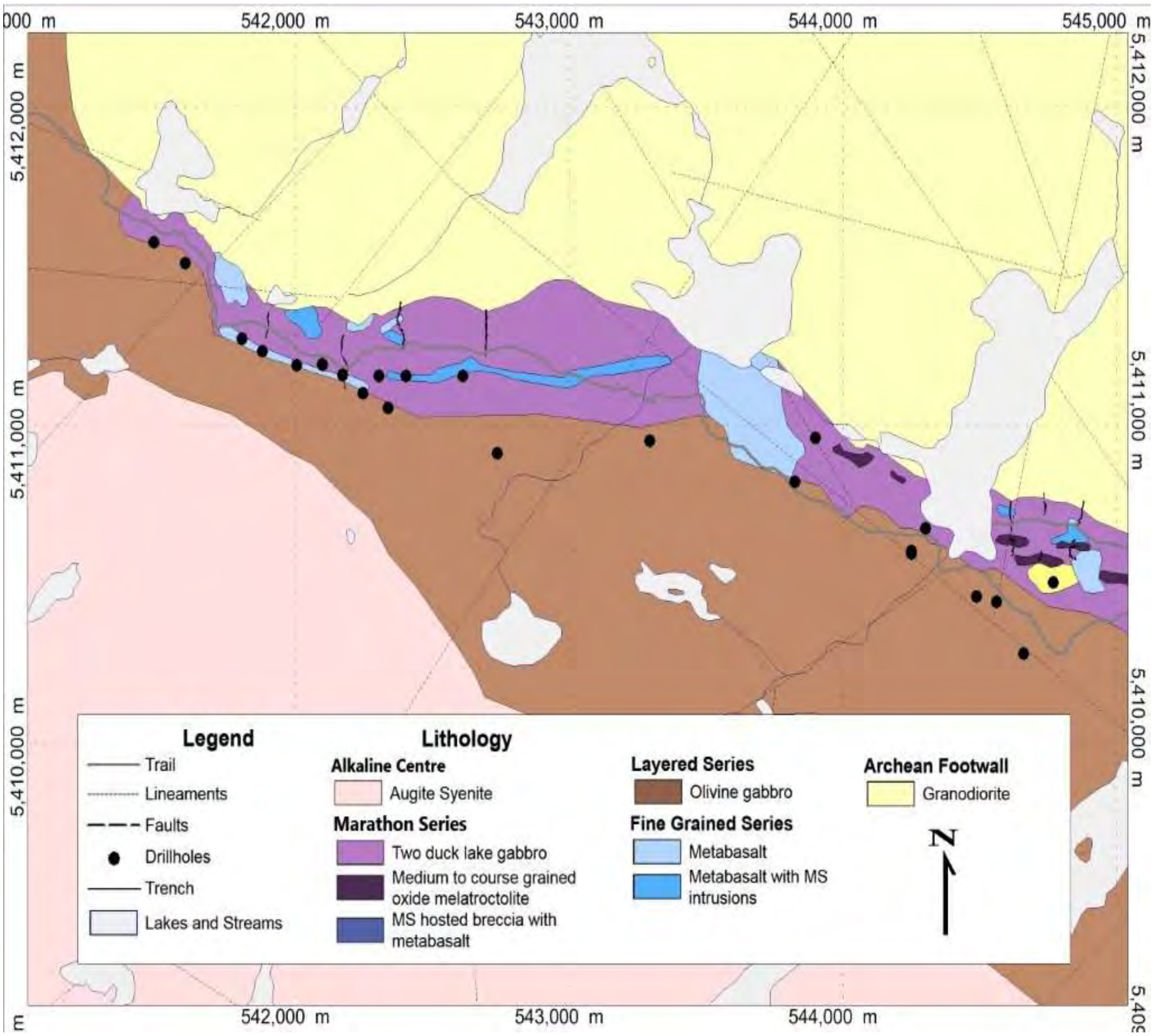
7.3.9 Boyer Prospect

The Boyer Prospect is located 10 km north and along strike from the Marathon Deposit to the roughly east-west trending northern margin of the Caldwell Complex and 4 km east of the Sally Deposit. It was discovered in 2016 and work completed includes surface mapping, six trenches and 14 diamond drill holes. Channel samples from various trenching programs included total PGM+Au values up to 3.1 g/t over 2.02 m, 0.82 g/t over 21.78 m, and 1.11 g/t over 7.69 m, with surface grab samples yielding up to 6.78 g/t.

The Boyer area has the largest intrusion of Two Duck Lake Gabbro outside of the Marathon and Sally Deposits and has a prominent reversely magnetized signature. The TDL intrusion has a strike length of 3 km extending from the Skipper Zone to the east and is up to 150 m thick. It dips 20° to 45° to the south. The Boyer Prospect area along with the drill holes can be seen in Figure 7.25.

The TDL gabbro at Boyer is similar to Marathon and Sally Deposits, but there is an increased proportion of pegmatitic material, brecciation and fragments of troctolites, wehrlites and dunnites. The TDL gabbro intrudes through the metabasalt along the basal granodiorite footwall.

Figure 7.25: Geology Map of Boyer Zone with Drill Hole Collars



Source: *Gen Mining (2021).*

7.4 Sulphide Mineralization

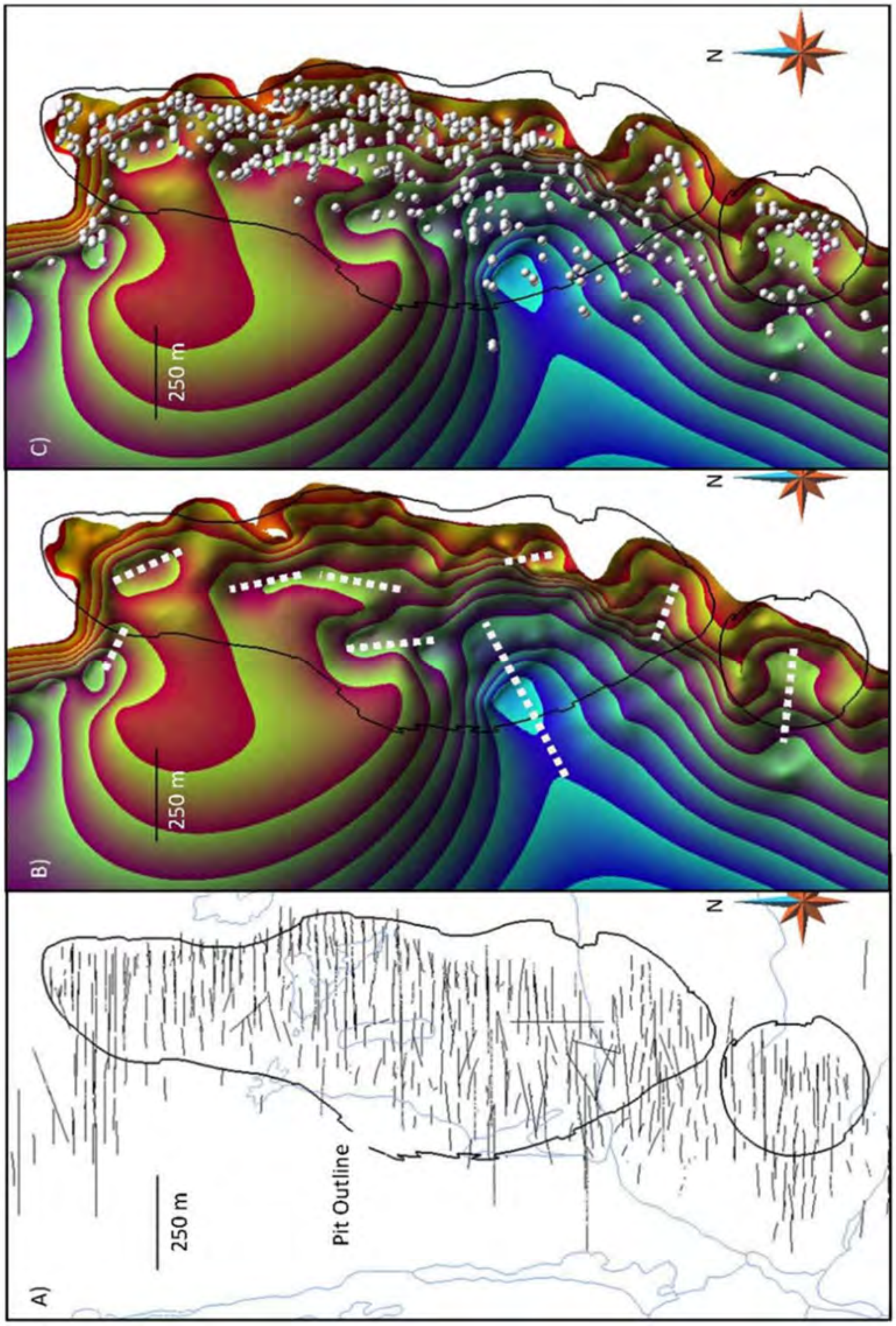
Sulphides in the TDL Gabbro consist predominantly of chalcopyrite, pyrrhotite and minor amounts of bornite, pentlandite, cobaltite, and pyrite. They occur in between primary silicates and to a lesser extent in association with secondary calcite and hydrous silicates such as chlorite and serpentine (Watkinson and Ohnenstetter, 1992). Chalcopyrite occurs as separate grains or as replacement rims on pyrrhotite grains. Some chalcopyrite is intergrown with highly calcic plagioclase (An70 to An80) in replacement zones at the margins of plagioclase crystals (Good and Crocket, 1994).

The modal mineralogy of a composite sample that is representative of the Marathon Deposit mineralization (and TDL Gabbro) was determined in a QEMSCAN™ survey by XPS (Kormos, 2008). A total of nine aliquots of material were analyzed. In decreasing order of abundance, the sulphide component of the composite sample consists of 2.75% pyrrhotite, 0.79% Cu-Fe sulphides (chalcopyrite and bornite), 0.09% pentlandite and trace amounts of pyrite, galena and sphalerite.

The relative proportions of pyrrhotite and chalcopyrite vary significantly across the Marathon Deposit. In general, the sulphide assemblage changes gradually up section from the base to the top of mineralized zones. Sulphides at the base of the TDL Gabbro consist predominantly of pyrrhotite and minor chalcopyrite and the relative proportion of chalcopyrite increases up section to nearly 100% chalcopyrite near the top. In the W Horizon, sulphides consist mainly of chalcopyrite and bornite and minor to trace amounts of pentlandite, cobaltite, pyrite and pyrrhotite.

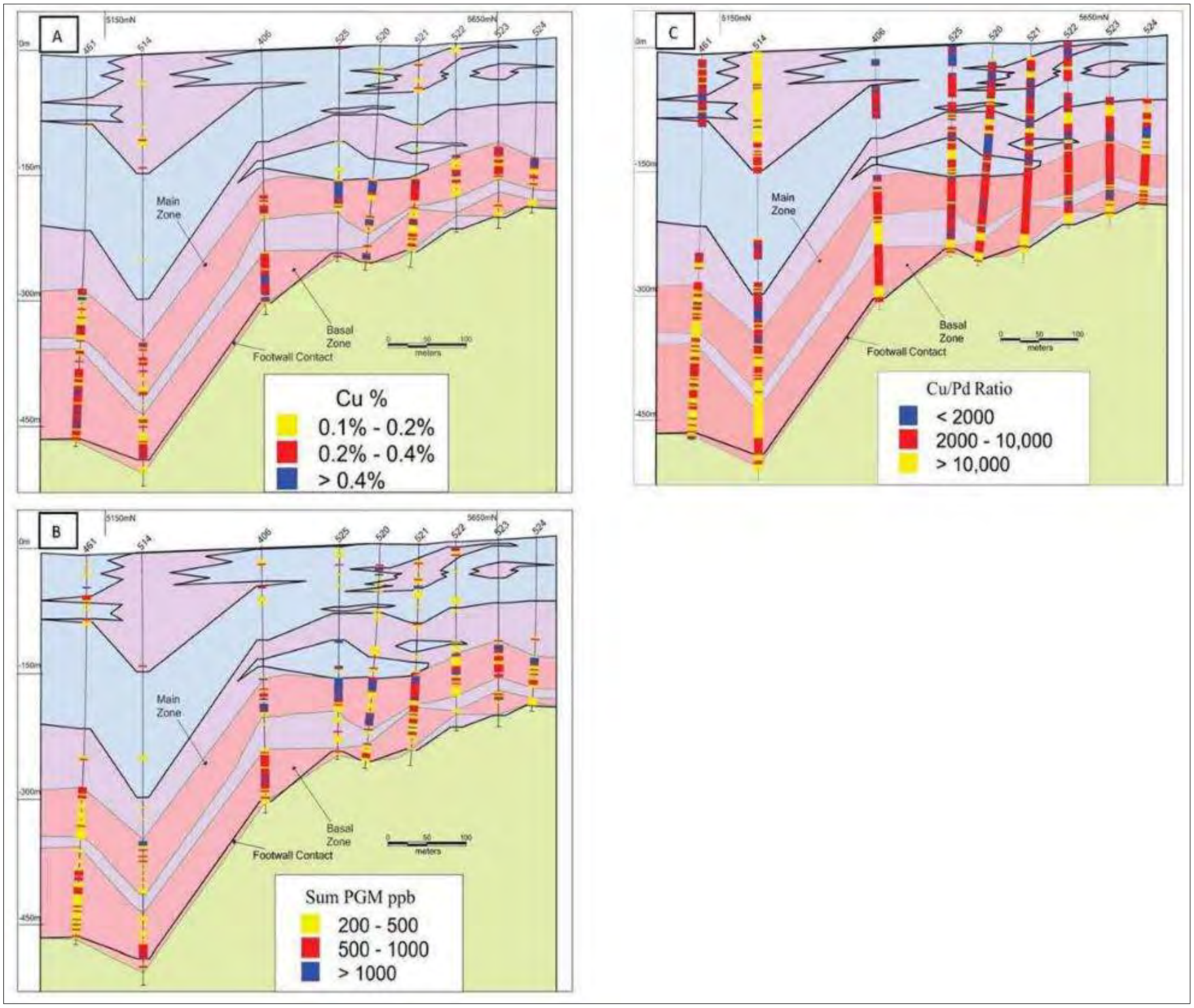
There is a relationship between mineralization and the paleo topography of the footwall contact as demonstrated in Figure 7.26. For example, mineralization is best developed within basins or troughs (b and c) of the footwall and thins or pinches out above prominent footwall ridges. It is important to note that although the mineralized zones are almost continuous from the north to south extents of the Marathon Deposit, assays with the best grades (combined Pd+Cu recalculated and presented as NSR) in Figure 7.27, fall along trends that mimic the alignment of troughs or ridges.

Figure 7.26: Plan Views of the Proposed Pit Outline (2010) Beneath the Marathon Main Zone



Note: *Figure A) includes all diamond drill holes and outlines for small lakes and streams. Figure B) includes the contoured 3-D surface model for the footwall contact. The white dashed lines highlight the trough axes in the footwall. Figure C) includes white spheres that represent drill hole assays that are filtered to show only those with NSR values greater than $75/t.*
Source: *Marathon PGM Corp. (2010).*

Figure 7.27: Marathon Deposit North-South Vertical Cross Section Along the Western Edge of the Main Open Pit (Looking West)



__Note:__ Figures show the Main and Footwall zones hosted within TDL Gabbro. Detailed geology along the drill stems for this section is located in Table 7.5. Numbers along the top of drill stems are drill hole numbers (example, M11-514). Numbers at top of figure are deposit section indicator (example 5150 m N corresponds to 5405150 m N, NAD 27 Zone 16N). Figures A, B and C contain assay values along the drill stem for Cu, Pd and Cu/Pd, respectively.
__Source:__ Marathon PGM Corp. (2010).

7.4.1 Platinum Group Minerals

The following summary was prepared from the detailed petrographic and SEM studies conducted at Lakehead University by Liferovich (2006, 2007). Two sample groups from the Main Zone and W Horizon are described and compared. A total of 2,304 grains from 55 thin sections were analyzed and 39 different platinum group minerals and gold, silver alloys were identified.

The grain size distribution for PGM in the Main Zone is similar to that in the W Horizon (Table 7.2). In general, approximately 60% of PGM grains are less than 5 microns.

The type and proportion of host minerals for the PGM are presented in Table 7.3. The dominant host minerals for the PGM in both areas are sulphides and other PGM. Similar proportions occur within the boundaries of plagioclase crystals but note that the 25% proportion is by count and not by volume (mass) and it is expected that the volume percent of grains in plagioclase margins is less than 25% because included grains are smaller. The relatively high proportion (38%) of PGM in hydrous silicates (chlorite and serpentine) in the Main Zone contrasts with the much lower proportion in the W Horizon (4.3%).

The suite of PGM in the Main Zone is very different from that of the W Horizon (Table 7.4). Indeed, of the 12 dominant PGM that comprise 85% of the PGM reported in the W Horizon, none were found in the Main Zone. Conversely, of the 10 dominant minerals found in the Main Zone (91% of all PGM found), only 2.6% occurred in the W Horizon. This remarkable difference in the ranges of PGM for the two zones implies different conditions of PGM mineral crystallization.

The finding from Lakehead is supported by work completed in 2014 (Cabri L.) and 2016 (Ames et al.). The two studies apply two various techniques to separate sulphide phases from silicates, Cabri's work utilized hydro separation while Ames used energy pulse disaggregation. Both methods then took the separated grains, mounted them on a thin section and completed mineral identification by SEM. Both studies observed that the main Pd mineralogy at the Main Marathon Deposit was dominantly antimony-arsenide, arsenides, bismuthides, telluride and stannite. Only the Ames study contained samples from the W Horizon but found a very different mineral assemblage: arsenides, sulphides, antimony-arsenides, plumbide, and tellurides. There was also a higher variety of palladium, platinum and rhodium species in the W Horizon relative to the Main zone.

Table 7.2: Size Distribution for PGM Minerals in the Main Zone Compared with the W Horizon

Zone	No. of Grains	< 5 Microns (%)	5-10 Microns (%)	10-20 Microns (%)	>20 Microns (%)
Main	573	64.9	16.9	12.5	5.7
W Horizon	1,731	58.3	27.1	9.6	5.0

Source: *Ruthart (2013).*

Table 7.3: Proportion of PGM Minerals Spatially Associated with Silicates, Sulphides or Other PGMs

Zone	No. of Grains	Plagioclase Boundaries (%)	Sulphides (%)	Other PGMs (%)	Hydrous Silicates (%)
Main	573	22.4	34.9	4.36	38
W Horizon	1,731	25	53.7	16.5	4.3

Note: *This does not represent volume percent as grains included in plagioclase boundaries are smaller than those located elsewhere.*
Source: *Ruthart (2013).*

Table 7.4: Dominant PGM Mineral Phases in the Main Zone Compared to the W Horizon

Mineral	Formula	W Horizon	Main Zone
Zvyagintsevite	$(Pd,Pt,Au)_3Pb$	41.8%	-
Palladinite	(Pd,Cu,Au)O	15.5%	-
Telargpalite	$(Pd,Ag)_3Te$	5.5%	-
Skaergaardite	PdCu	3.9%	-
Kotulskite, Pb-rich	Pd(Te,Bi,Pb)	3.8%	-
Isoferroplatinum	$(Pt,Pd)_3(Fe,Cu)$	3.7%	-
Keithconnite, Pb-rich	$Pd_{3-x}(Te,Pb,Sb)$	3.5%	-
Tetraferroplatinum	PtFe	3.4%	-
Plumbopalladinite	Pd_3Pb_2	1.2%	-
Vysotskite	PdS	1.2%	-
Laflammeite	$Pd_3Pb_2S_2$	1.1%	-
Atokite, Pb-rich	$(Pd,Pt)_3(Sn,Pb)$	0.9%	-
Au, Ag and alloys		7.0%	3.3%
Stilwaterite	Pd_8As_3	0.4%	0.9%
Arsenopalladinite	$Pd_8(As,Sb,Pb)_3$	0.3%	1.7%
Cotunnite, Ru-rich	$(Pb,Ru)Cl_2$	-	2.1%
Hessite	Ag_2Te	-	3.7%
Hollingworthite	(Rh,Pt,Pd)AsS	0.2%	5.6%
Sperrylite	$PtAs_2$	1.1%	6.3%
Kotulskite	Pd(Te,Bi)	-	9.9%
Sobolevskite	PdBi	0.1%	10.1%
Mertierite-II	$Pd_8(Sb,As,Pb)_3$	0.3%	16.1%
Kotulskite-Sobolevskitess	$Pd_2Te(Bi,Pb)$	0.2%	34.9%

***Note**: A total of 2,304 grains from 55 thin sections were analyzed from the two zones.*
Other minerals with less than 1% distribution in both zones were excluded from this list.
***Source**: Ruthart (2013).*

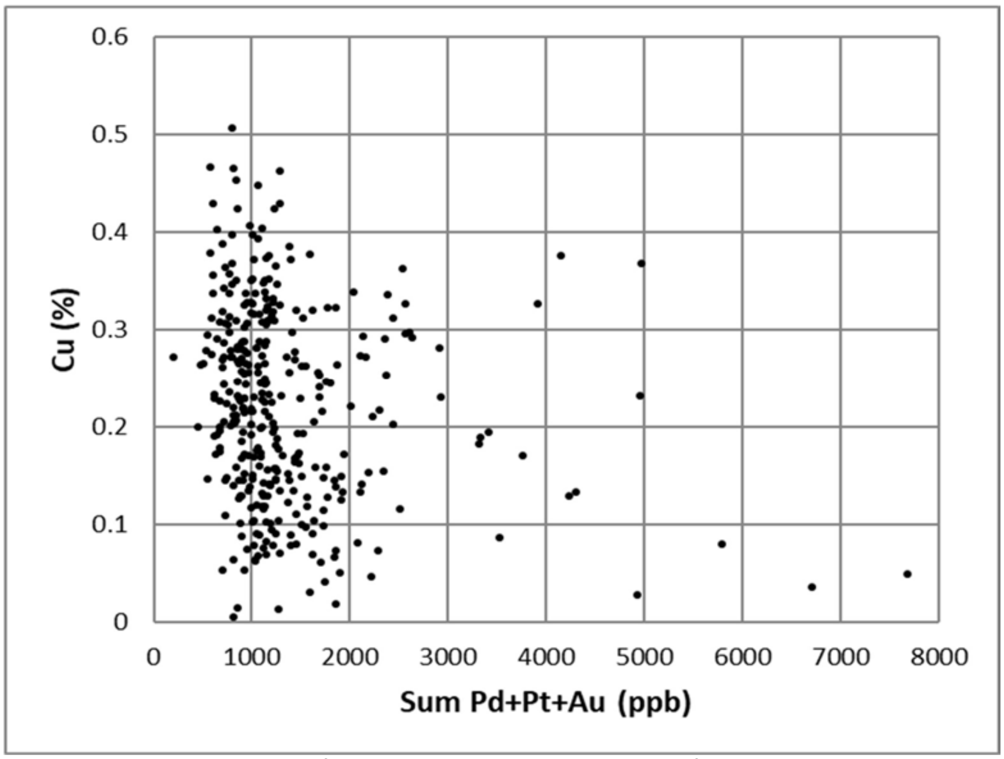
7.4.2 Distribution of Cu, Ni and PGM within the Marathon Deposit

A very prominent feature of the Marathon Deposit is the local and extreme enrichment of PGM with respect to Cu and Ni. For example, high-grade samples from the W Horizon that contain between 25 and 50 g/t Pd (1 g/t = 1 part per million) might also contain very low concentrations of Cu and Ni (<0.02%). The separation of PGM from Cu is observed throughout the Marathon Deposit, but is most common near the top of the mineralized zone. In the southern half of the Marathon Deposit, PGM enrichment is most prominent in the W Horizon.

The separation of PGM from Cu is shown by the very poor correlation between Cu and the sum of PGM for the average of 356 intersections in the Marathon Deposit (Figure 7.28). The disparity in the relative behavior of PGM and Cu and Ni is unusual for contact type magmatic sulphide deposits. Barrie et al. (2002) attributed the PGM enrichment to high temperature zone refining process, but this process is inconsistent with mass balance calculations and the close correlation between Pd and the other PGM metals.

An understanding of the separation of PGM from Cu is important to define the model for deposition of the Marathon Deposit. In this section, the trends for S, Cu, Ni and PGM concentrations in these zones are described and three mechanisms for metal concentration during magmatic processes are proposed.

Figure 7.28: Plot of Cu Vs. the Sum of Pd+Pt+Au for Average Values of 356 Diamond Drill Hole Intersections (NSR Cut-off of $15/t)



***Note**: Each point represents an intersection of between 4 and 160 m thickness. All of the points represent 14,485 m of drill core or approximately 8,000 samples.*
***Source:** Marathon PGM Corp. (2010).*

7.4.3 <u>Metal Ratios for the Marathon Deposit</u>

Inter element ratios for metals that show positive and significant correlation are calculated for a subset of samples representative of the Marathon Deposit (Table 7.5).

Table 7.5: Calculated Ratios for Cu, Ni and the PGM Metals

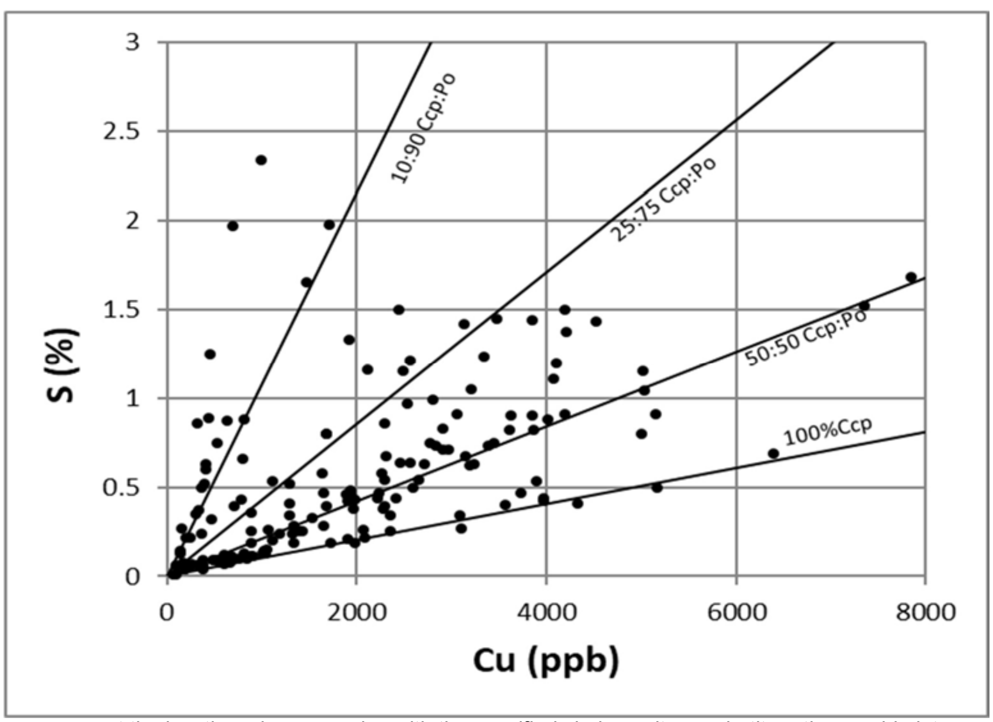
Ratio	Average	Standard Deviation	Minimum	Maximum	No. of Samples
Cu/Ni	14.5	2.8	8.2	21	40
Pd/Pt	2.99	1.02	0.83	9.2	8,663
Pd/Rh	40	19	10	84	32
Pd/Ir	910	636	147	2,573	28
Pd/Au	9.6	6.6	0.3	80	8,663

Note: *Cu/Ni ratio calculated for samples with >3,000 ppm Cu. Pd/Pt ratio calculated for intersection data. Pd/Rh and Pd/Ir calculated using high precision and high accuracy data by Good (1993) and 10 high grade samples analyzed by Activation Labs.*
Source: *Gen Mining (2019).*

7.4.4 Distribution of Cu in TDL Gabbro

The sulphide assemblage in the Marathon Deposit comprises predominantly chalcopyrite and pyrrhotite with minor pentlandite and bornite. Chalcopyrite is the dominant copper mineral and bornite occurs locally, particularly in the W Horizon. In general, sulphides at the base of the Main Zone are composed of pyrrhotite and the proportion of chalcopyrite increases up section. On average, the majority of mineralized samples contain greater than 25% chalcopyrite and less than 75% pyrrhotite as shown in Figure 7.29. Samples with the highest concentrations of PGM fall along or close to the curve representing 100% chalcopyrite.

Figure 7.29: Sulphur vs. Copper for Samples Representative of Marathon Deposit Mineralization



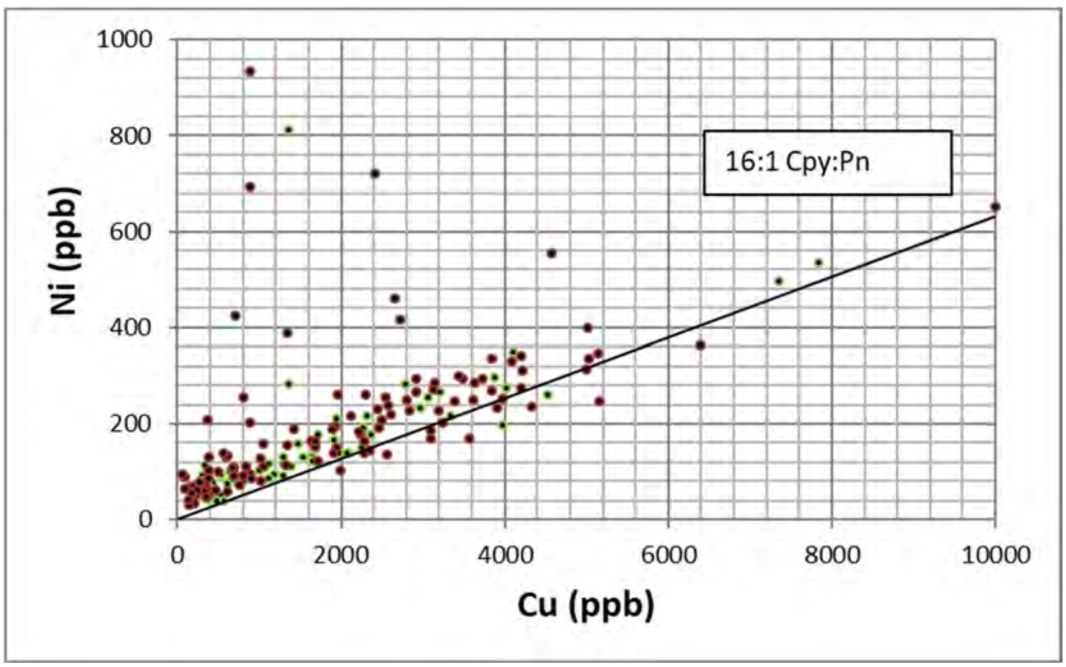
Note: *The lines represent the location where samples with the specified chalcopyrite: pyrrhotite ratios would plot.*
Source: *Marathon PGM Corp. (2010).*

7.4.5 Distribution of Ni Relative to Cu

Pentlandite is the dominant nickel-bearing mineral and is present as a minor component of the sulphide assemblage. Based on whole rock data for Ni vs. Cu, as shown in Figure 7.30, the chalcopyrite to pentlandite ratio for mineralized samples is relatively constant at approximately 16:1. For whole rock data where Cu is >3,000 ppm, the Cu/Ni ratio is relatively constant at 14.5. A small proportion of samples in the Marathon Deposit reveals that the abundance of nickel is normally less than approximately 1,200 ppm and rarely greater than 1,500 ppm (Figure 7.30).

In Figure 7.30, the abundance of nickel, where the abundance of copper is 0%, corresponds to the amount of nickel (60-100 ppm) held by olivine and clinopyroxene. The nickel content of olivine, as measured by Good (1993) for samples in the Main Zone and Ruthart (2013) for samples in the W Horizon, is between 400 and 600 ppm.

Figure 7.30: Plot of Ni Against Cu for a Subset of Main Zone Samples for which S (wt %) was Determined

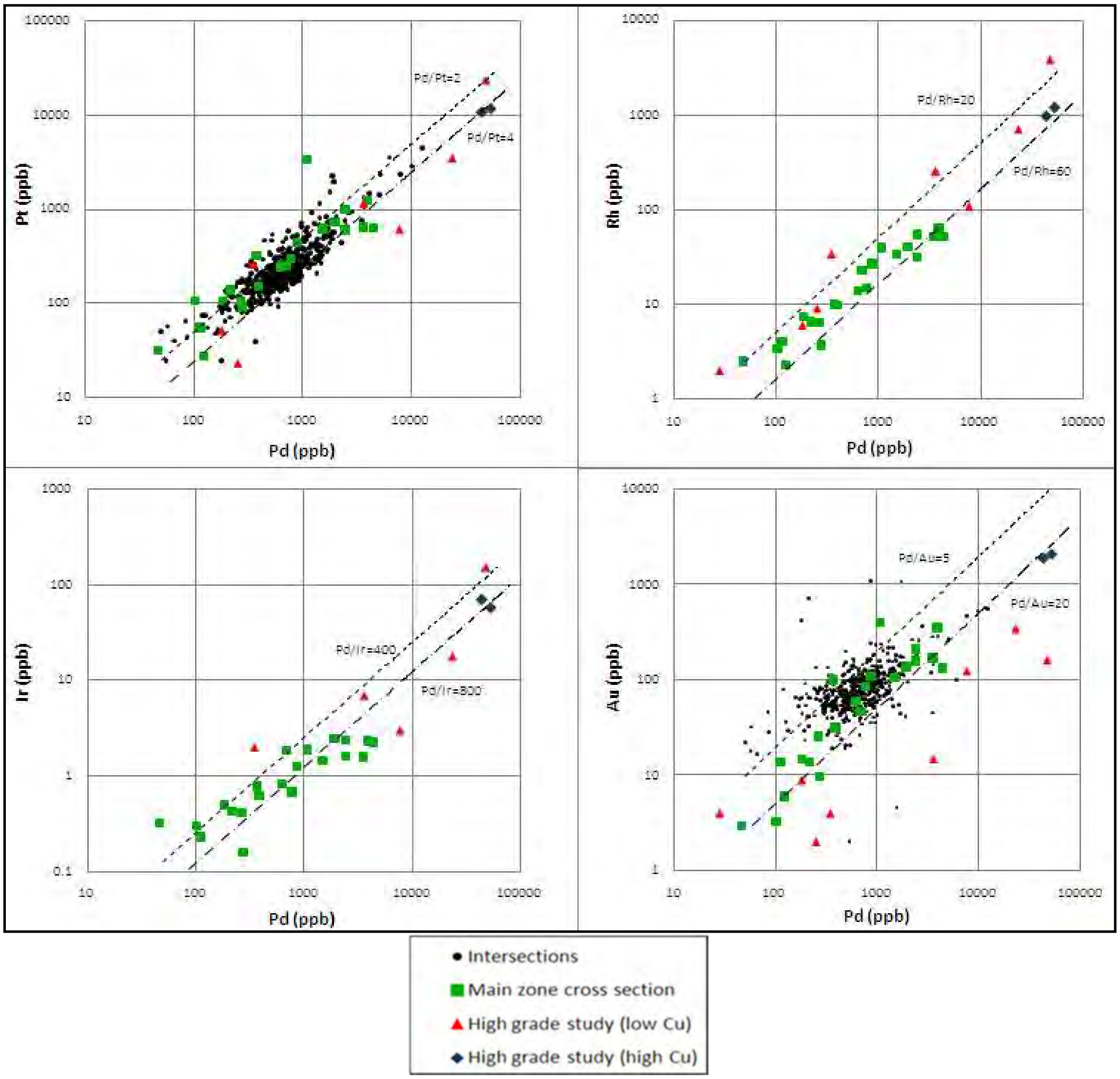


Note: *In general, the nickel content increases with increasing Cu. The majority of samples lies along a trend parallel to a calculated line representing samples with 94% chalcopyrite and 6% pentlandite or an approximate ratio of 16:1.*
wt % = weight percent.
Source: *Marathon PGM Corp. (2010).*

7.4.6 Distribution of PGMs

There is a strong and positive correlation between Pd and the other PGM metals (Pt, Rh and Ir) and Au for all types of mineralization in the Marathon Deposit (Figure 7.31).

Figure 7.31: Plot of Pd vs. Rh, Ir and Au for Representative Sample Groups of the Marathon Deposit



Note: *Intersections are averages of drill core intervals of between 4 and 160 m of mineralization. Main Zone cross-section samples were analyzed by Good (1993). 10 high-grade study samples are subsamples of 2 m thick, high-grade intersections (analyzed by Activation Labs). Low Cu samples represent 50 cm splits from interval at 184-186 m in hole M-07-237 which contained 121 ppm Cu. High Cu samples are 10 cm of quartered core that were selected from the interval between 152-156 m in hole M-07-306 which contained 0.8% (8,000 ppm) Cu. The Main Zone cross section samples and high-grade study samples are considered to be high precision and high accuracy analyses.*
Source: *Marathon PGM Corp. (2010).*

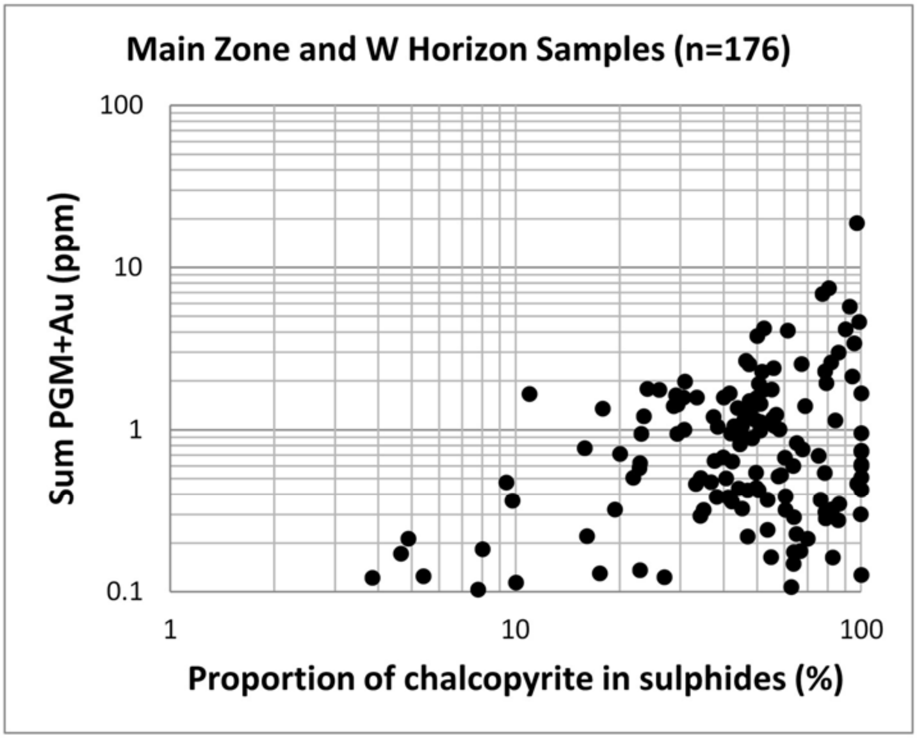
7.4.7 Relationship Between Sulphide Assemblage and PGM

The composition of the sulphide assemblage is in general indicative of PGM enrichment. For example, a pyrrhotite rich sulphide assemblage is typically poor in PGM whereas chalcopyrite rich (up to 100%) or bornite-bearing sulphide assemblages are typically high in PGM. This general field relationship is verified in Figure 7.32 where the values for the sum of PGM+Au are highest in samples with high calculated proportions of chalcopyrite in total sulphides. Note this relationship is different than that shown in

Figure 7.35 where it shown that there is no correlation between Cu and Pd. Also note that the increasing proportion of chalcopyrite is not always a sign of increasing PGM+Au.

That there is a relationship between chalcopyrite and total PGM+Au, but no correlation between copper and Pd implies multiple concentrating mechanisms acted to concentrate Cu and PGM+Au.

Figure 7.32: Sum of Pt+Pd+Au vs. Calculated Proportion of Chalcopyrite in Sulphide Assemblage



Note: *Data set is representative of Main Zone and W Horizon.*
Source: *Marathon PGM Corp. (2010).*

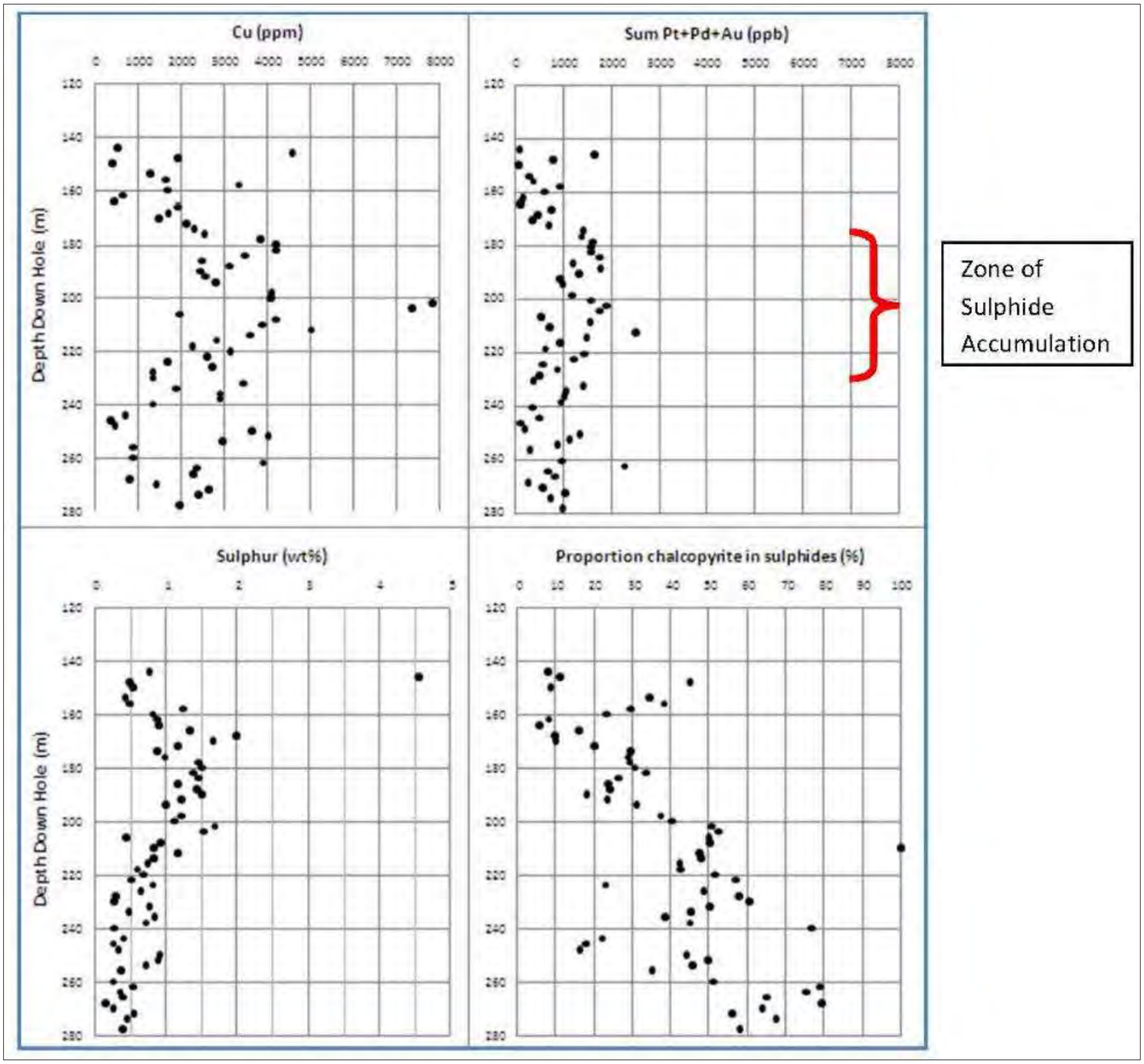
7.4.8 Variations of Cu, PGM, Sulphur and Chalcopyrite Across Mineralized Zones

Two different trends are shown by metal variation plots across mineralized zones in Figure 7.33 and Figure 7.34.

In Figure 7.33, the abundances of S and PGM increase systematically up section and can be attributed to the simple accumulation of sulphides. The change in the abundance of Cu is less obvious, but there is a systematic decrease in the proportion of chalcopyrite in the sulphide assemblage. In summary, the abundance of sulphides and PGM are increasing, but sulphides are becoming more pyrrhotite rich.

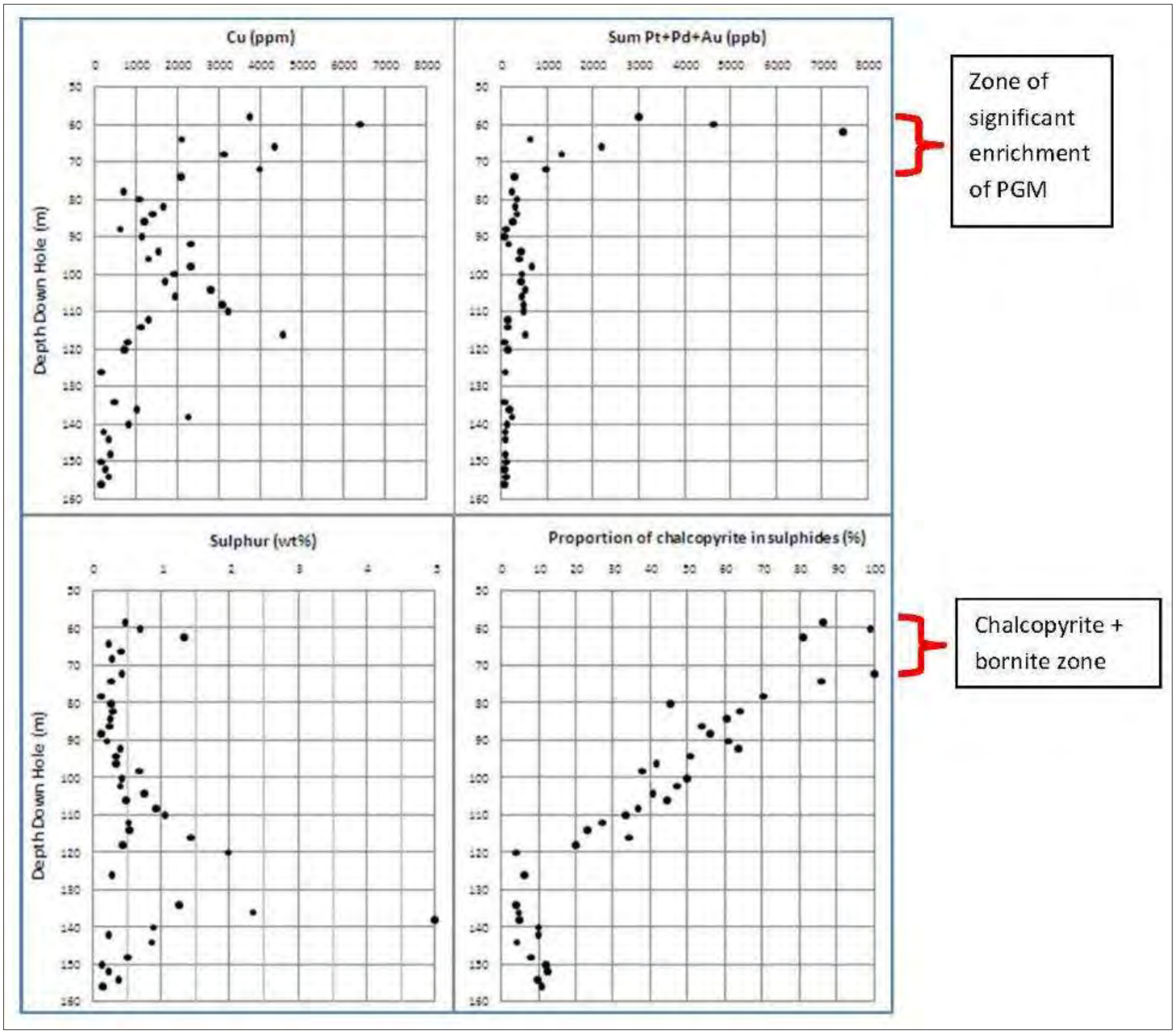
In Figure 7.34, the abundance of Cu and the proportion of chalcopyrite increase up section, the abundance of S stays flat or decreases, and the Pd stays low but increases dramatically in the uppermost 12 m where the samples contain the highest proportion of chalcopyrite.

Figure 7.33: Metal Variation Down Diamond Drill Hole MB-08-10



Note: *Each sample represents 2 m of split drill core. It shows elevated PGM and Cu with increasing sulphur (sulphides) regardless of proportion of chalcopyrite.*
Source: *Marathon PGM Corp. (2010).*

Figure 7.34: Metal Variation Down Diamond Drill Hole G9



Note: *Each sample is 2 m of split drill core. It shows significant PGM enrichment in zones with highest proportion of chalcopyrite.*
Source: *Marathon PGM Corp. (2010).*

7.4.9 Mechanisms for Cu-PGM Concentration in the Marathon Deposit

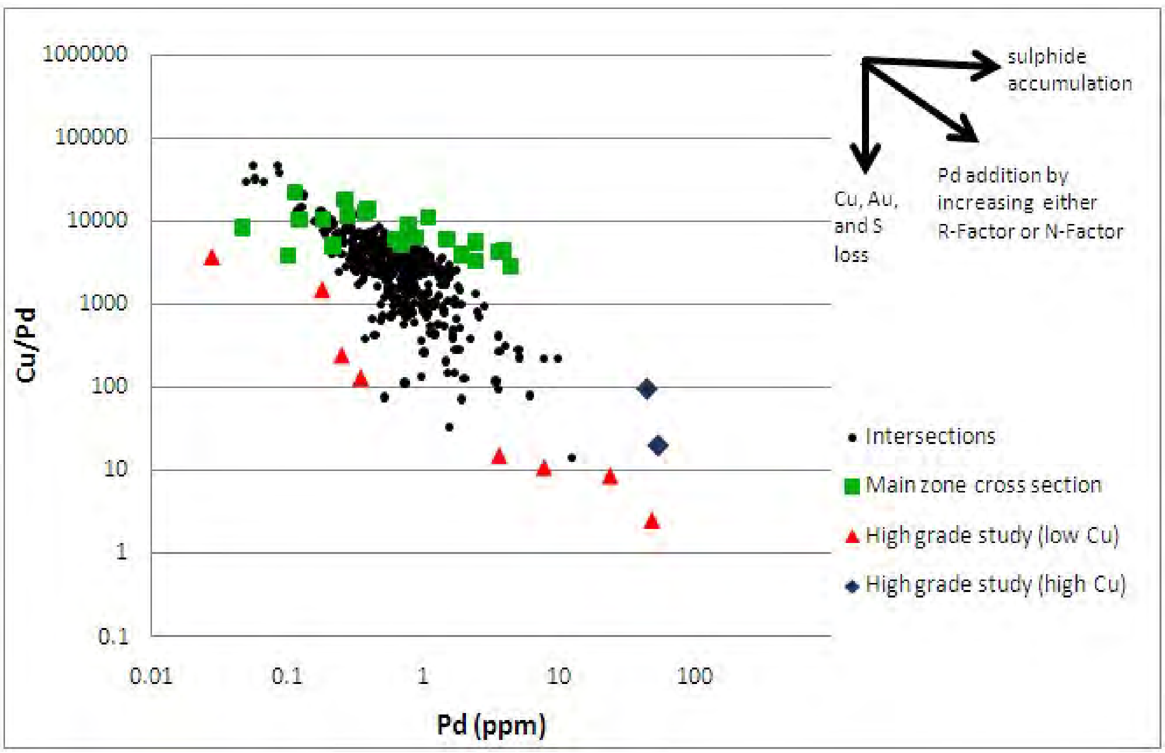
At least three mechanisms for sulphide and PGM precipitation have been proposed for the Marathon Deposit including hydrothermal (Watkinson and Ohnenstetter 1992), magmatic (Good and Crocket (1994a) and zone refining (Barrie 2002). A hydrothermal mechanism at low or intermediate temperatures (<600°C) is not possible owing to the near total absence of hydrous minerals in the W Horizon and the significant correlations between Pd-Pt, Pd-Rh and Pd-Ir. The high temperature zone refining mechanism suggested by Barrie (2002) is compelling but there is insufficient experimental evidence to use PGM correlation as support for or against the model and the implied redistribution and concentration of PGM by zone refining does not fit with a mass balance calculation. It is viewed that there is too much PGM and too little gabbro for a zone refining mechanism to have played a significant role.

Based on petrographic and geochemical evidence, it seems most likely that more than one process operated at high temperatures (>700°C) to concentrate metals in the Marathon Deposit. Three possible mechanisms include:

- Accumulation of sulphide liquid in fluid dynamic traps in the magma conduit.
- Ongoing interaction of sulphides with magma that is flowing through the conduit (R-factor).
- Removal of S, Cu, and Au from the sulphide assemblage.

The effects of the three mechanisms on the abundance of Cu and Pd are shown in Figure 7.35. The effect of accumulating sulphides is shown by the trend for the Main Zone samples (green squares). The effect of the R-factor is the rapid increase in Pd relative to Cu (pulls samples toward the lower right corner of Figure 7.35). The intersection data (dots) represent the average affects due to sulphide accumulation and R-factor enrichment. Finally, the removal of Cu in PGM enriched zones (W Horizon) is shown by the downward displacement of the samples from the low Cu, high grade zone (red triangles). The removal of Au is inferred from the Pd-Au variation diagram in Figure 7.31.

Figure 7.35: Dominant Mechanism Diagram for Cu and PGM Concentration



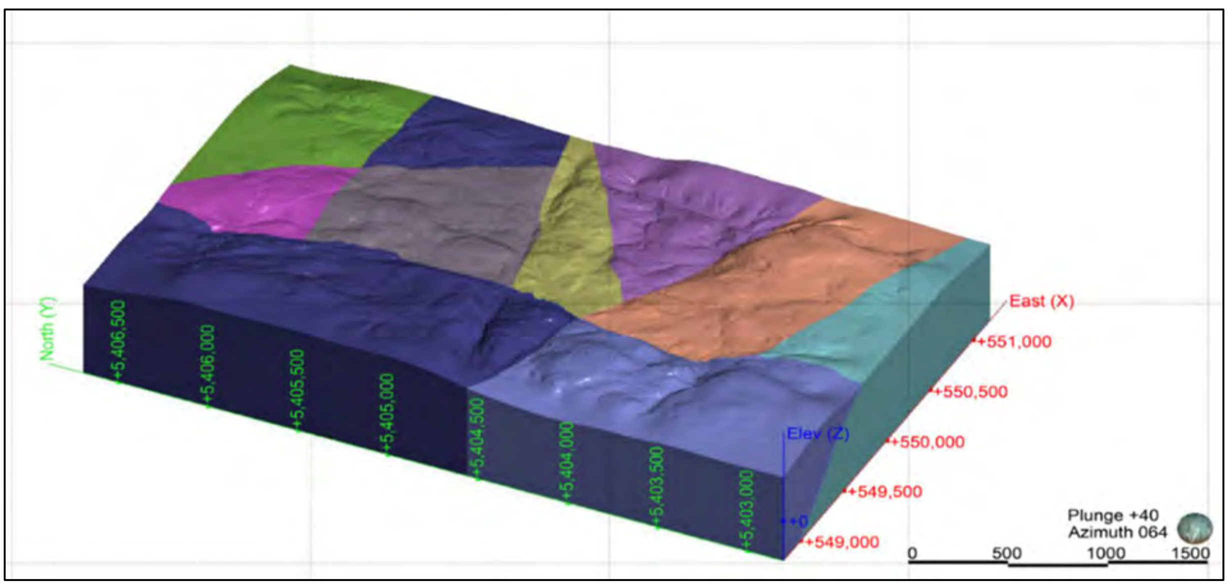
Note: *Figure highlights the effects on metal values of the three dominant mechanisms proposed to explain the concentration of Cu and PGM in the Marathon Deposit.*
Source: *Marathon PGM Corp. (2010).*

7.5 Marathon Structure

Based on the interpretation of drill hole contacts and surface geology, a structural model has been developed that incorporates 11 fault blocks for the Marathon Deposit (Figure 7.36). Lineament structures are prominent throughout the Coldwell Complex and are associated with brittle faulting. They occur in two series; radial and concentric. The radial faults extend from a central location on the southwest of the Coldwell Complex below Lake Superior. There are five prominent large continuous faults that extend beyond the lithological boundary of the Coldwell Complex, as well as less prominent secondary and tertiary splays off the larger faults. The concentric faulting occurs as discontinuous faulting perpendicular to the radial faulting. The two series of lineaments create a mosaic of various blocks throughout the Coldwell Complex but is most prominent along central, east and northern margin. From surface geology, it is recognized that there is offset between the blocks; however, only at the Marathon Deposit is the offset measurable due to the continuous WT sill. The radial faulting at the Marathon Deposit are normal faults, with up to 50 m offset and rotation of north side down of between 7-12°. The radial faulting is less defined as they do not pass through the Marathon Deposit but based on exploration drilling dip towards the center of the Coldwell Complex at shallower angle than 65°.

Faulting is believed to be a critical component to the emplacement of the Marathon Deposit. The thickest drill intercepts of the Marathon Series are adjacent to known faulting and surface lineaments. Mineralization is also thickest within footwall embayments which show a spatial relationship and similar orientation to known lineaments and faulting. Faulting acts as the structural control for magma emplacement within a conduit setting.

Figure 7.36 : Modeled Marathon Fault Blocks



Source: *Gen Mining (2022).*

8. DEPOSIT TYPES

8.1 Deposit Type Magma Conduit Model

The Marathon Deposit is one of several mafic to ultramafic intrusive bodies in the MRS System that host significant copper, nickel or PGM sulphide mineralization. These intrusions include the Yellow Dog peridotite (Eagle Deposit), the Tamarack Deposit, the Current Lake Intrusive Complex (Thunder Bay North Deposit), and the numerous intrusions located along the base of the Duluth Complex.

Intrusion and deposition of sulphides within magma conduits has recently become the dominant mineralization forming process chosen to explain the rift related deposits. For example, a magma conduit deposit model has been proposed for the Marathon Deposit by Good (2010), Thunder Bay North by Goodgame et al. (2010) and the Eagle Deposit (Ding et al., 2012). The magma conduit model has grown in favour since it was proposed to explain deposits in the Noril'sk region, Siberia by Naldrett et al. (1995) and Naldrett and Lightfoot (1999) and the deposits at Voisey's Bay by Li and Naldrett (1999). Further, an important contribution to the understanding of magma conduits and the formation of very high tenor PGM deposits was presented by Kerr and Leitch (2005). They derived a sophisticated geochemical model for an open system multiple stage process expected in a magma conduit. This model was applied to explain the extreme PGM concentrations found in the W Horizon at the Marathon Deposit by Good (2010).

8.2 Magma Conduit Model for Marathon Mineralization

In the magma conduit deposit model, the present exposure of the TDL and Eastern Gabbro series represents only a fraction of the magma that was generated in the mantle and made its way up through the crust. Most of the magma actually passed through the magma conduits and erupted on the surface as basaltic volcanic flows. The gabbroic units and associated Cu-PGM mineralization represent material that crystallized or settled out of the magma as it moved through the conduit.

It is envisaged that a very large volume of magma, perhaps greater than 10,000 times the volume of gabbro present in-situ, passed through the conduit and formed the TDL Gabbro. On the basis of mass balance calculations and considering the TDL Gabbro is less than 250 m thick, only a very large magmatic system such as this can explain the excessive enrichments of platinum metals with up 45 g/t of combined Pt, Pd and Au over 10 m or the accumulations of disseminated sulphide layers that are up to 160 m thick. Similarly, in the case of the oxide ultramafic intrusions, very large volumes of magma are required to deposit the very thick layers (tens of metres) of massive magnetite (>75% magnetite).

In the magma conduit model, fluid dynamic factors that affected magma flow are relevant to exploration. Features such as pooling of TDL magma in basins within the footwall or brecciation of Eastern Gabbro by TDL magma as it stopes its way upward during ascent are important examples of how the magma flow was slowed resulting in the precipitation of the denser sulphide liquid from the magma. Conversely above ridges or crests in the footwall, where TDL Gabbro thins and the magma velocity increased, sulphides were unable to settle out of the magma and mineralized horizons thin or pinch out. Accumulation of sulphide by fluid dynamic processes can explain the bulk of the mineralization in the Marathon Deposit and metal trends such as that shown in diamond drill hole MB-08-10. Metal trends show increasing Cu and PGM+Au with increasing total sulphides regardless of the proportion of chalcopyrite in the sulphide assemblage.

After sulphides settled out of the magma, a second process acted to upgrade the sulphides with PGM+Au, particularly in the upper portions of the mineralized zone (as described in drill hole G9). The upgrading occurred as magma passed through the conduit and interacted with sulphides in the crystal pile possibly by stirring up early formed sulphides. This process of sulphide upgrading was used to describe the extreme enrichments of PGM relative to Cu in disseminated sulphides at the Noril'sk deposits by Naldrett et al. (1995). Naldrett et al. described the mathematical model whereby the ratio of magma in the conduit that interacted with sulphides to the amount of sulphides is referred to as the R factor. Under conditions where the R-Factor is very high, continued interaction of fresh magma with sulphides will continue to increase the grade of PGM while the Cu concentration remains constant. Very high PGM concentrations in the W Horizon such as 45 g/t over 10 m (hole M07-306) and metal trends such as the gradual increase in the proportion of chalcopyrite and the matching rapid increase in PGM+Au are interpreted to be a result of continuous upgrading.

A third process of PGM upgrading by sulphide dissolution (after Kerr and Leitch, 2005) is envisaged to have occurred in the W Horizon to account for samples with extreme PGM content and only trace Cu. For example, in many instances the PGM enrichment of up to 75 ppm Pd occurs in samples with only 0.01% to 0.02% Cu. These levels of Pd when re-calculated to abundances in 100% sulphides correspond to untenable concentrations of between 2% and 4% Pd in 100% sulphide. The sulphide dissolution process involves the progressive removal of Cu and S from the pre-existing sulphides when they interact with magma that is sulphur under saturated. The Pd and Pt remain behind with the remnant sulphides. Evidence of Au loss in samples of the W-Horizon imply that Au was also removed along with Cu and S by this same process.

8.3 Comparison of Marathon Deposit with Mid-Continent Rift-Related Deposits

There are many striking petrologic and geochemical similarities between the TDL Gabbro and the Partridge River Intrusion, located at the base of the Duluth Complex, Minnesota (Good and Crockett, 1994). The Partridge River intrusion is the best described gabbroic intrusion in the Duluth Complex and is host to the Minnamax (Babbit) and Dunka Road Cu-Ni-PGM Deposits. The relevant features described from the Partridge River Intrusion that are also observed in the TDL Gabbro, include the following:

- The textures and abundance of minerals in the Partridge River Intrusion and the inferred crystallization path are remarkably similar to those of the TDL Gabbro.

- The compositions of plagioclase, pyroxene, and olivine are restricted relative to other mafic intrusions and overlie values for the TDL Gabbro.
- The coherent behavior of Zr, Rb, and Y, indicative of control by variable proportions of intercumulus liquid, is consistent with observations in the TDL Gabbro.
- Chalcopyrite and PGM are intergrown with calcic plagioclase that replaces less calcic plagioclase.
- Pyrrhotite, but not pentlandite, is replaced by chalcopyrite.
- Sulphides are predominantly interstitial to unaltered plagioclase, olivine, and pyroxenes and chalcopyrite and PGM are associated with Cl-enriched biotite and apatite, and altered minerals, such as chlorite, epidote, and calcite.
- Variable Cu/Ni ratios within deposits and between deposits and a trend of increasing ratios with increasing Cu are indicative of chalcophile element fractionation as shown for the TDL Gabbro.
- The occurrence of more than one type of disseminated sulphide zone, one being relatively sulphur rich is analogous to the main and basal sulphide zones in the TDL Gabbro.

The many similarities between the Partridge River Intrusion and the TDL Gabbro imply that they formed by analogous processes. Four mechanisms have previously been proposed to account for features observed in the Partridge River Intrusion:

- Chalockwu and Grant (1990) proposed that the magma of the Partridge River Intrusion was emplaced as a plagioclase plus olivine crystal mush that crystallized in situ.
- Grant and Chalockwu (1992) provided geochemical and isotopic evidence implying that the Partridge River Intrusion consists of a mechanical mixture of cumulus plagioclase, olivine, and intercumulus liquid which were not in equilibrium with each other.
- Foose and Weiblen (1986), and Ripley (1986) proposed various mechanisms for the mixing of magmas of similar compositions, but at different stages of crystal fractionation, to account for compositional irregularities.
- Finally, an external source for sulphur is well documented in the available literature. Andrews and Ripley (1989) argue that sulphur assimilation occurred prior to intrusion of the host gabbro. These mechanisms are, to some extent, analogous to those proposed in the model for the formation of the Marathon Deposit.

8.4 Comparisons of Mid-Continent Rift, Voisey Bay and Noril'sk Deposits

Comparisons between the MRS and the Voisey Bay and Noril'sk settings point to several similarities that suggest that the Mid-Continent Rift is a likely setting for Ni-Cu mineralization. The continental rifting and associated voluminous igneous activity in all three regions formed in response to the rise of a hot plume of mantle material from deep in the earth, fracturing the overlying continental crust. In the Mid-Continent Rift, melting of the plume produced more than two million km^2 of mostly basalt lava flows and related intrusions.

In all three regions, basalts derived from the mantle plume are enriched in trace elements, particularly in comparison to the most common basalts erupted on earth, those formed at rifts in the oceans. Like basalts in the Noril'sk region, early basalts of the Mid-Continent Rift have compositions characterized by relatively high abundances of magnesium, chromium, nickel, and platinum, and relatively low abundances of sulphur. Such metal-rich but sulphur-poor basalt magmas can carry metals (such as Ni, Cu, and PGM) to high levels in the crust because sulphur is not available to form a separate sulphide liquid that would scavenge metals from the magma while it is still deep below the surface. If these metal-rich basalts encounter a source of sulphur near the surface, and sulphur is incorporated into the basalt magma, they would be ripe for sulphide mineral formation.

8.5 Marathon Deposit Model Conclusions

A possible model for the emplacement and crystallization history of the TDL magma and genesis of sulphides is proposed as outlined below.

Step one: Crystallization of plagioclase and olivine occurred in a deep magma chamber prior to emplacement into its present site. Due to density differences, plagioclase did not settle out of the magma column but much of the olivine did. During crystallization and sporadic replenishment with unfractionated magma, the magma chamber becomes compositionally stratified.

Step two: Sulphur migrated out of the country rock into the magma chamber resulting in the formation of sulphide droplets. The Ni/S ratio of the sulphide droplets will be high in the lower layers of the chamber and low in the upper layers of residual magma.

Step three: The Two Duck Lake intrusion and sulphide deposit is formed when magma is forced out of the deep chamber upward into its present site. The more fractionated, plagioclase-rich upper layers become mixed with the less fractionated lower layers by the turbulent movement out of the deep chamber. The sulphide droplets grow as they come into contact with other droplets during transport. At the time of intrusion, the crystal mush consists of plagioclase crystals of nearly uniform composition, interstitial silicate magma, and droplets of sulphide liquid; there was little, if any, crystal-free magma in the chamber.

Step four: After intrusion, some minor settling of plagioclase crystals occurred, and plagioclase formed a framework for crystallization of the interstitial melt. The crystal mush cooled rapidly thereby inhibiting post-cumulus processes, such as complete internal equilibration of the system. A very small amount of volatile-rich interstitial melt migrated toward the center of the intrusion, crystallized granophyre, and released water into the surrounding gabbro, resulting in the formation of pegmatite.

Step five: Subsolidus reactions occurred involving local migration of components in deuteric fluid. This process results in features such as the replacement of pyrrhotite by chalcopyrite and the deposition of PGM in association with hydrous silicates; the last to form are microscopic chalcopyrite, calcite, and chlorite veinlets. The numerous documented features presumably reflect reactions that occur as the temperature decreases and the fluid evolves.

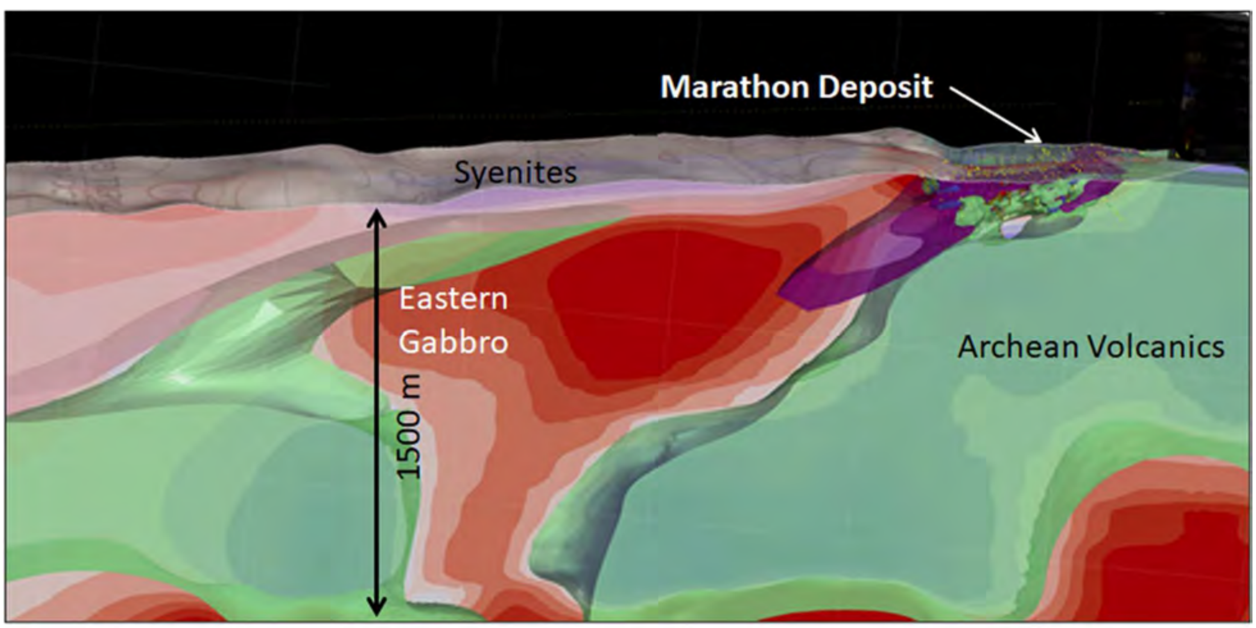
9. EXPLORATION

9.1 Exploration Work by Gen Mining

Shortly after acquiring the Marathon Property from Stillwater Canada in July 2019, Gen Mining initiated follow up of exploration targets previously developed by Stillwater Canada. Exploration work by Gen Mining began to focus on the exploration for high-grade mineralization, as either density accumulated semi-massive to massive sulphides or highly enriched PGM zones like the W Horizon. Geological controls on these types of higher-grade mineralization require focused exploration along feeder conduits and provide opportunities for target generation at greater depths.

Exploration for density accumulated semi-massive to massive sulphides was in part guided by a partnership, established in 2018, between Stillwater Canada and PACIFIC, a consortium of industry, government, and academic partners to conduct fundamental and applied research to develop passive seismic techniques for mineral exploration. The purpose of this work was to image the Coldwell Complex to a depth of 2 km, modeling the geometry of the Eastern Gabbro as a guide for future exploration. A production-scale survey was completed at the Marathon Deposit and resulted in a 3-D velocity inversion model (Figure 9.1). The survey successfully imaged the stepping boundary between the Archean Footwall and Eastern Gabbro. The resolution of the passive seismic survey was also much improved over the gravity survey and is the first model showing the west dipping footwall contact to the Coldwell Complex.

Figure 9.1: Passive Seismic 3-D Velocity Inversion Showing the Marathon Deposit Relative to the Coldwell Complex



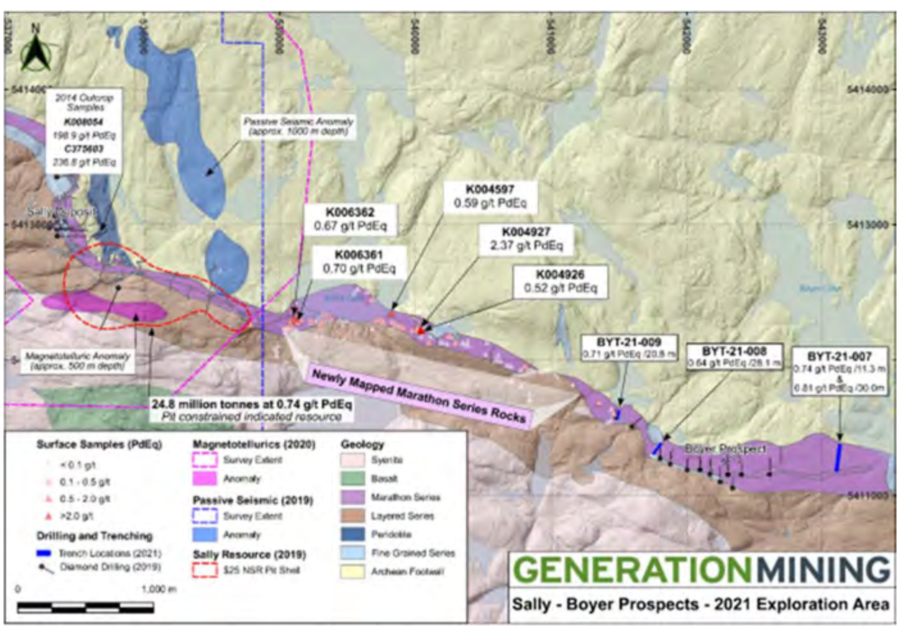
***Source**: Gen Mining (2021).*

In 2019, exploration work by Gen Mining consisted of geologic mapping and prospecting at the Boyer zone and the northern extension of the Geordie Deposit. Three trenches were completed at Boyer, exposing the continuation of mineralization at surface. A passive seismic survey was completed at the Sally Deposit to help define deep high-density targets for potential drill testing. Borehole EM surveys were completed by Crone Geophysics on diamond drill holes SL-19-72, M-19-536 and M-19-537.

In 2020, to compliment the previous seismic surveys, a magnetotelluric ("MT") survey was completed over a portion of the Marathon Deposit and an area immediately to the west, which was believed to be underlain by one of the feeder zone conduits of the Marathon Deposit. A MT survey was also completed over the Sally Deposit and immediate environs. The 2020 MT survey at Marathon delineated one target for drill testing. MT targets at Sally are, as of the effective date of this Technical Report, under development. Borehole EM surveys were completed by Crone Geophysics on diamond drill holes M-20-539, M-20-543 and M-20-547.

In 2021, a field mapping program was carried out over four grids on the Property. The Four Dams, Willie Lake and Redstone grids were planned to help better understand the continuity of the favorable Marathon Series horizon in areas where modern mapping was sparse or non-existent. A small condemnation mapping program was carried out on the hilltop immediately west of the Marathon Deposit, on which critical site infrastructure is planned as part of the proposed mine development. The Willie Lake program was particularly successful in defining a broad, continuous sequence of Two Duck Lake gabbro extending from the Boyer Prospect to the Sally Deposit and extending the mapped prospective horizon by nearly 2 km. The best grab sample on this grid returned 2.08 g/t PGM and 0.17% Cu (Figure 9.2).

Figure 9.2: Boyer-Sally Area - 2021 Summer Mapping Program



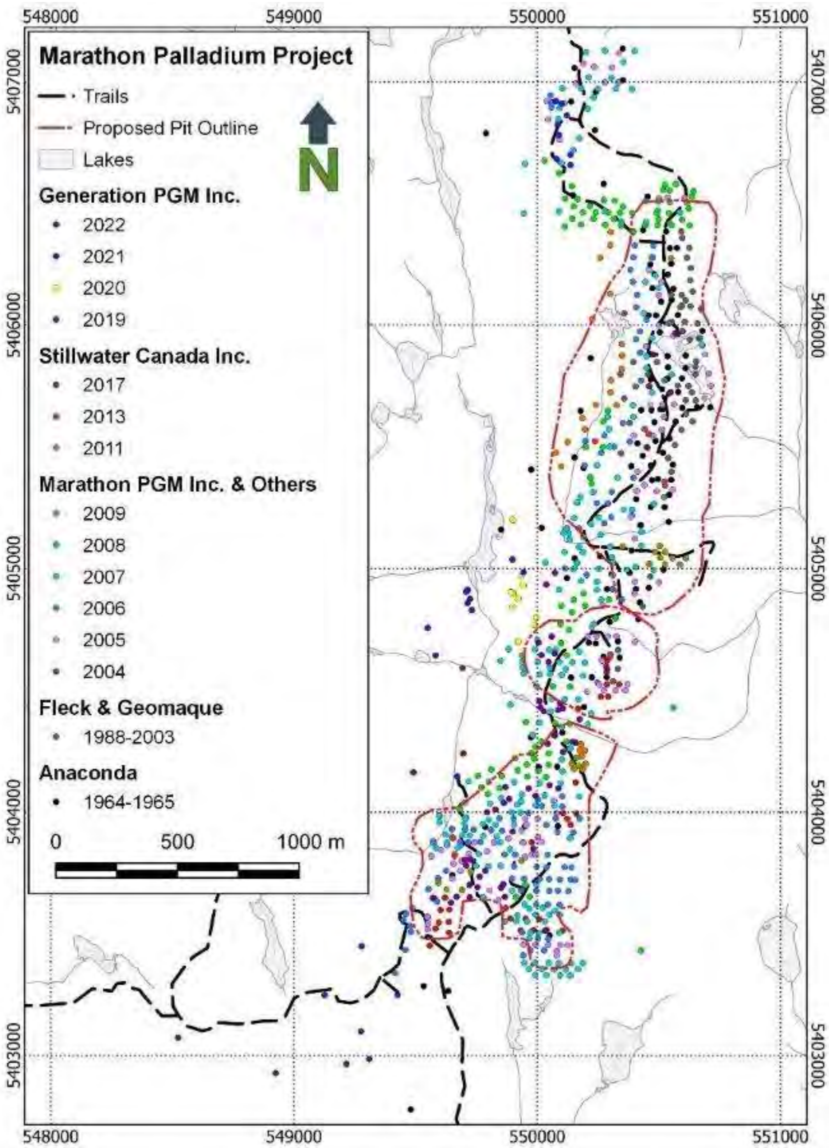
Source: *Gen Mining (News release November 8, 2021).*

10. DRILLING

This section describes the drilling activities completed by Gen Mining in 2019 through 2022 that occurred at the Marathon Deposit and other surrounding targets. Drilling activities prior to 2019 are discussed in Section 6 - History. Collar locations of all previous and current drill holes at the Marathon Deposit are included in Figure 10.1.

The results from the 2020 drilling program, which were not included in the 2020 Mineral Resource estimate (effective date of June 30, 2020), and the 2021 and 2022 drilling programs are included in the 2023 Mineral Resource estimate (effective date of December 31, 2022).

Figure 10.1: Diamond Drilling by Year at the Marathon Project



Source: *Gen Mining (2022).*

10.1 2019 Exploration Drilling Program

The Property had been under-explored for the past several years during a time of unprecedented low palladium prices. The Company's goal in 2019 was to confirm historical results, evaluate the potential to expand known resources, and drill test several exploration targets. In 2019, the Company completed 39 drill holes totalling 12,434.5 m, which included 1,023 m of confirmation drilling. Results were consistent and validated historical drill results (Table 10.1).

Table 10.1: 2019 Drilling Program

Deposit	Target	Holes Drilled	Meters Drilled
Marathon	Confirmation/Infill	5	1,023
Marathon	West Feeder Zone near Main Zone	6	3,484
Boyer	Greenfield exploration drilling	14	3,063
Geordie	Two offsets	8	2,587
Sally	High- grade samples and massive sulphides	6	2,278
Total		**39**	**12,435**

Drilling of various geophysical targets within the West Feeder Zone, and approximately 1.4 km west of the Marathon Deposit, confirmed that Target A, as described in Subsection 6.3 and shown in Figure 6.3 and Figure 6.4, most probably represents a high-density olivine and magnetite rich phase of the Layered Series Gabbro. Drill holes M-19-537 and M-19-538, which were drilled approximately 350 m west of the Marathon Deposit, intersected significant widths (102 m and 80 m, respectively) of Marathon Series rocks down dip from the Marathon Deposit. Results from holes M-19-537 and M-19-538 confirmed the continuation of the Marathon Deposit to the south side of the 5,404,900 N fault, which is believed to have provided a locus for the feeder conduit to the Marathon Deposit and the north part of the W Horizon, which hosts high-grade PGM mineralization.

Additionally, Gen Mining completed 14 holes totalling 3,063 m on the Boyer Zone: six holes totalling 2,278 m in the Sally Deposit area, and eight holes totalling 2,587 m in the Geordie area.

Drilling at Sally significantly extended the mineralized zone along strike and down dip. Boyer, which had not been previously drilled, is currently a 500 m long prospective horizon displaying anomalous subeconomic PGM concentrations (from results to date). No significant mineralization was intersected at Geordie, where the program focused on reconnaissance drill testing of gabbroic intrusions proximal and similar to the gabbro hosting the Geordie Deposit.

Data from the 2019 drill program were incorporated where appropriate in the 2020 Mineral Resource estimate for the Marathon Deposit; however, they were not incorporated in the Sally Deposit 2020 Mineral

Resource estimate. In 2019, no drilling was completed within the Geordie Deposit Mineral Resource domains.

10.2 2020 Exploration Drilling Program

In 2020, the Company completed 12 holes totalling 5,068 m (Table 10.2). The drilling was focused on the feeder zone conduit associated with the Main Marathon Deposit and the northern limb of the W Horizon. This drilling followed the successful completion of drill holes M-19-537 and M-19-538, which intercepted the down dip continuation of the Main Marathon Deposit for the first time. The 2020 drilling filled a 300 m gap between the historical drilling and the 2019 drilling south of the 5404900 N fault. Additional targets included the conductive zone west of the Marathon Deposit identified in the 2020 MT survey and the down dip extension of high-grade PGM mineralization in the W Horizon.

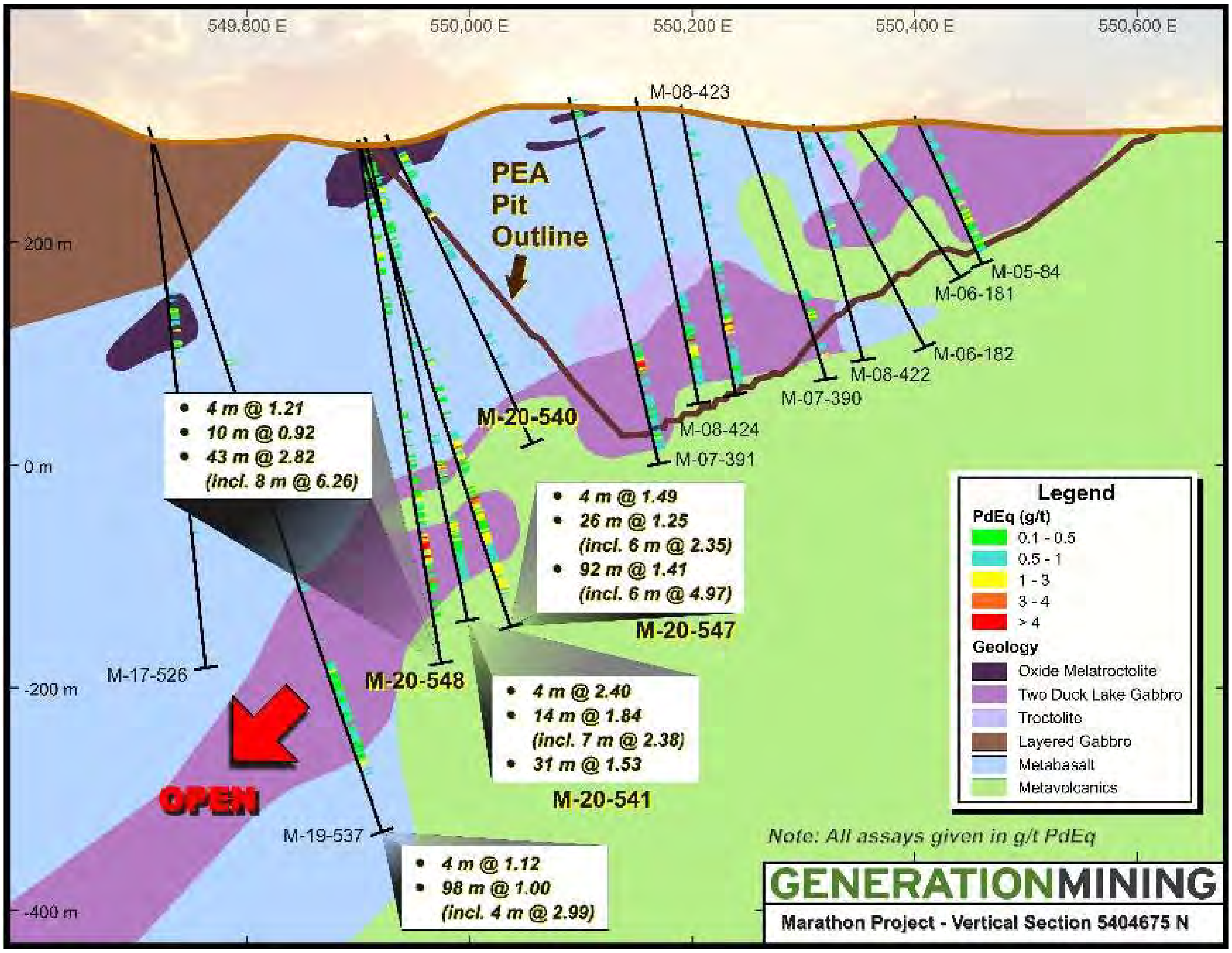
Table 10.2: 2020 Drilling Program

Deposit	Target	Holes Drilled	Meters Drilled
Marathon	MT Target	1	711
Marathon	West Feeder Zone	7	2,988
Marathon	W-Horizon	4	1,369
Total		**12**	**5,068**

The principal aim of the 2020 exploration drill program was to test for the potential of near-surface, ramp accessible mineralization. No PGM mineralization was intersected in hole M-20-539 that tested the MT target north of the 54048900N fault; however, significant intervals of PGM mineralization were intersected in drill holes testing the West Feeder Zone and extensions to the W-Horizon south bracketed by the 5404900N and 5404500 N faults. Assay results from selected drill holes (shown as g/t PdEq) are shown in a vertical section and in a plan view of the West Feeder Zone area (Figure 10.2 and Figure 10.3). True widths approximate down-hole lengths. The calculation is as follows:

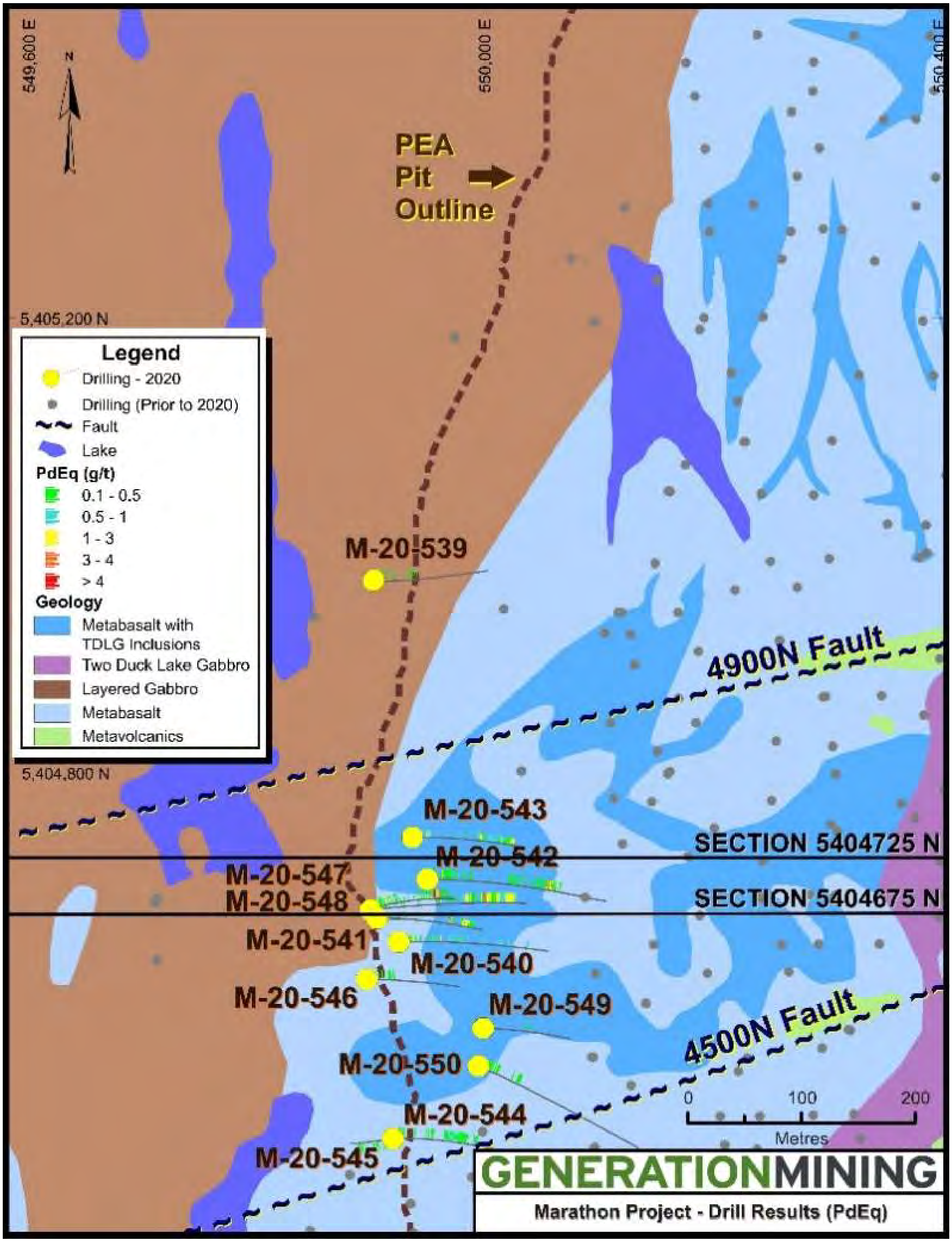
The palladium equivalent calculation expressed as g/t is the sum of the theoretical in situ value of the constituent metals (Au+Pt+Pd+Cu) divided by the value of 1 g of Pd. The calculation makes no provision for expected metal recoveries or smelter payables. The following commodity prices were used: US$1,300/oz Au, US$900/oz Pt, US$1,275/oz Pd and US$3.00/lb Cu.

Figure 10.2: Marathon Deposit – Vertical Section 5404675



Source: *Gen Mining (News release January 5, 2021).*

Figure 10.3: Marathon Deposit – Plan Views of the West Feeder Zone (Drill Results PdEq)



Source: *Gen Mining (News release January 5, 2021).*

10.3 2021 Exploration Drilling Program

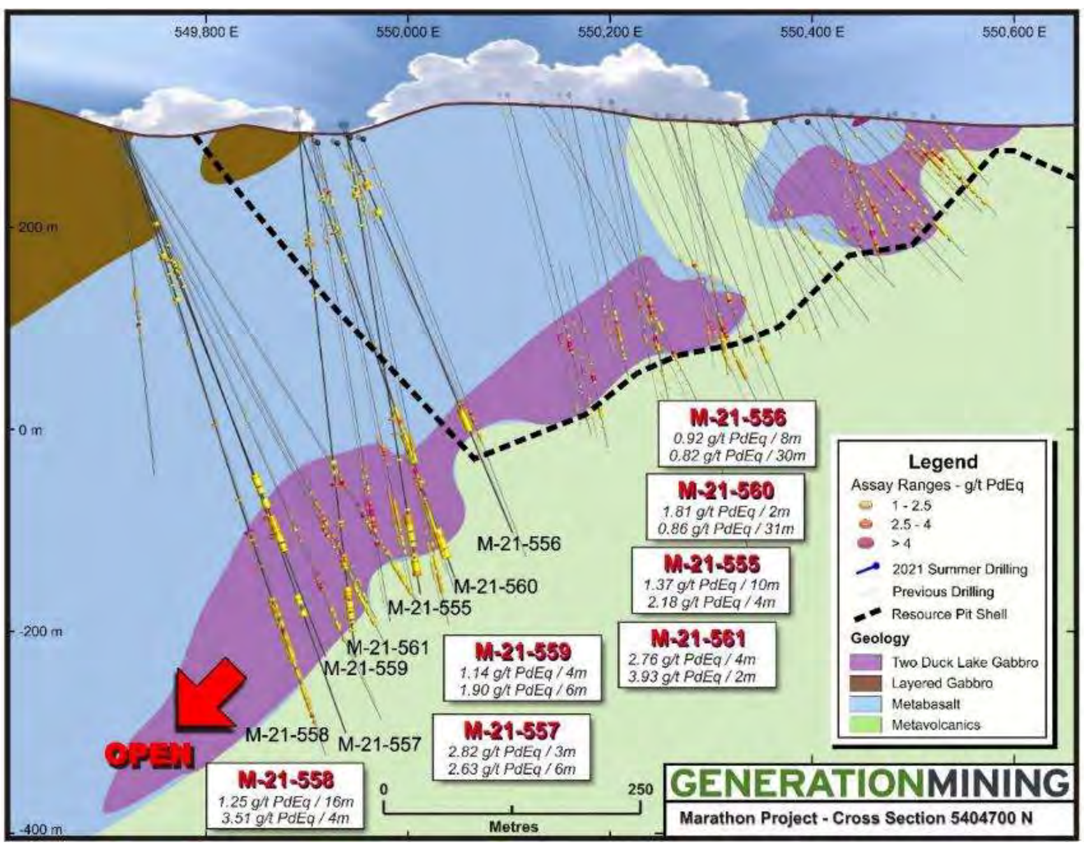
In 2021, the company completed 22 holes totalling 9,875.2 m on the Marathon Deposit and the Biiwobik Prospect (Chonolith and Powerline Zones) to the north of the Marathon Deposit (Table 10.3).

Table 10.3: 2021 Drilling Program

Deposit	Target	Holes Drilled	Metres Drilled
Marathon	Central Feeder Zone	11	5,735
Marathon	Biiwobik Prospect	11	4,140
Total		**22**	**9,875**

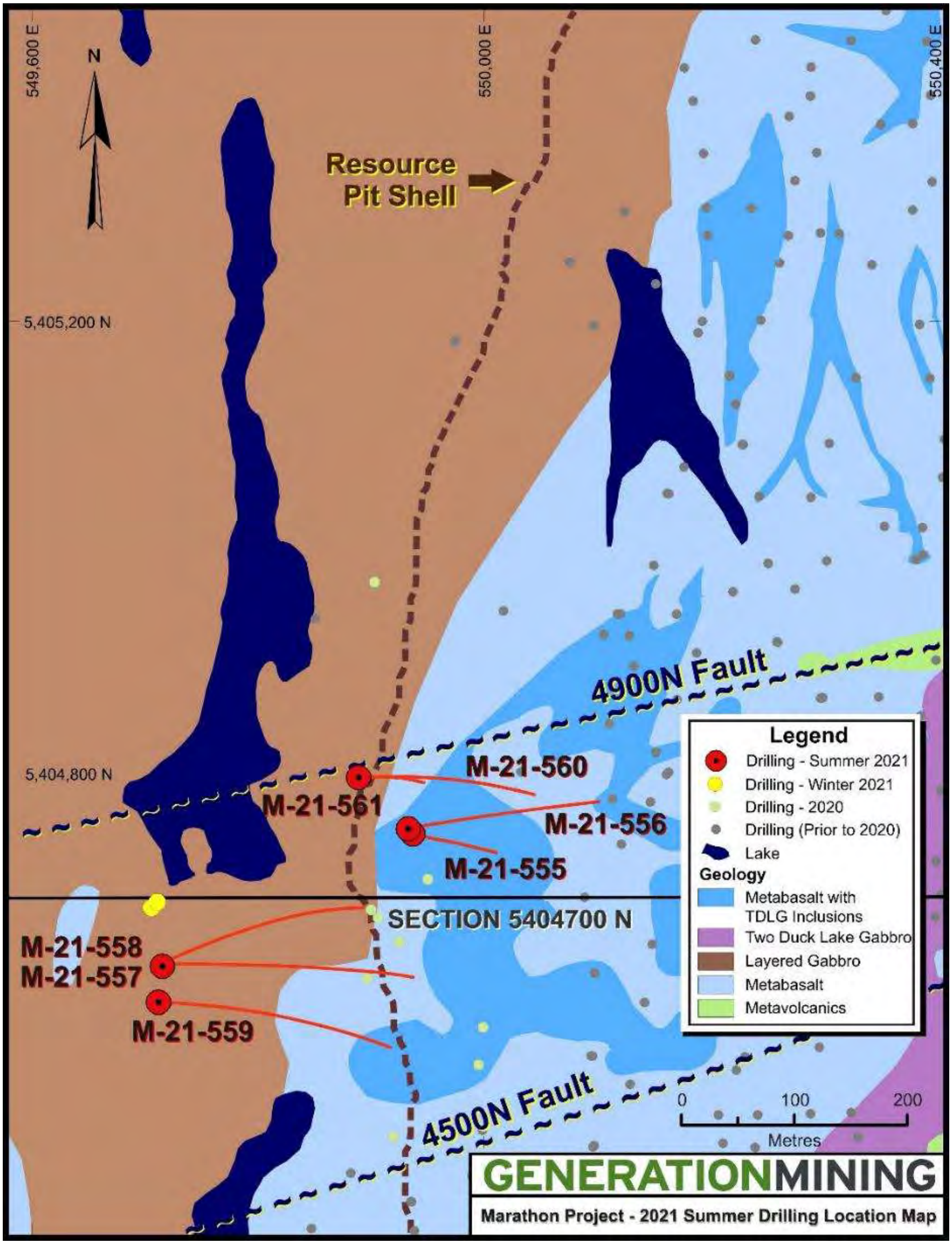
Following the success of the 2020 drilling program, 11 holes totalling 5,735.2 m targeted the potential downdip feeder channels for both the Main Zone and W-Horizon. Assay results from selected drill holes (shown as g/t PdEq) are shown in a vertical section (Figure 10.4) and plan view of the West Feeder Zone area (Figure 10.5). True widths approximate down-hole lengths.

Figure 10.4: Marathon Deposit – Vertical Section 5404700



Source: *Gen Mining (News release August 17, 2021).*
Note: *The Mineral Resource pit shell shown is that from the 2021 FS*

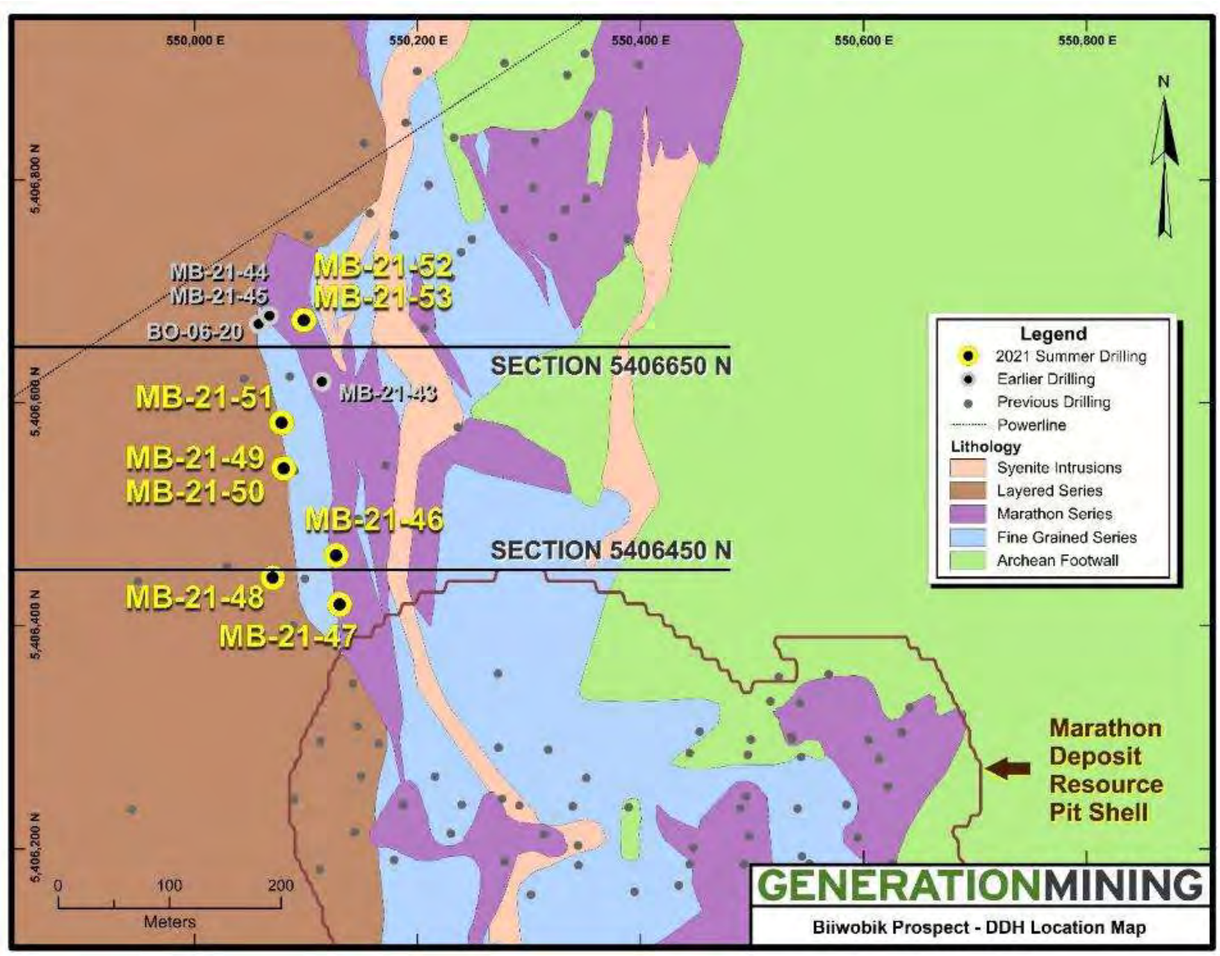
Figure 10.5: Marathon Deposit – Plan Views of the Central Feeder Zone – 2021 Drill Locations



Source: *Gen Mining (News release August 17, 2021).*

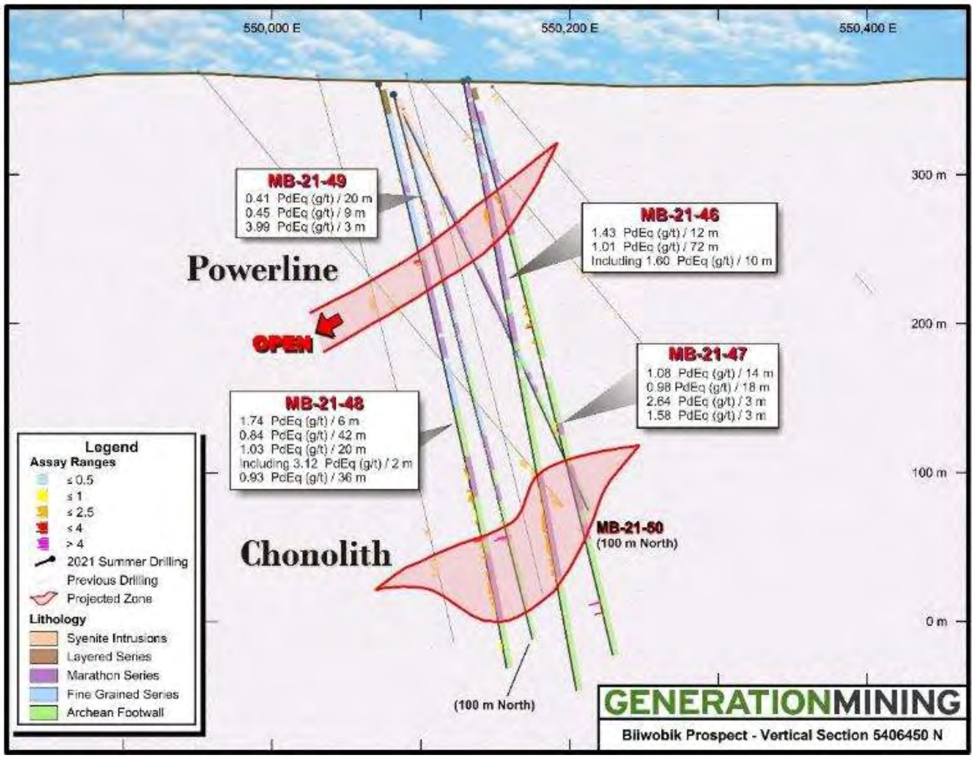
An additional 11 holes totalling 4,140 m focussed on the Biiwobik Prospect, which includes the Chonolith Zone and the newly defined Powerline West occurrences. The Chonolith Zone represents a tube-like feeder channel that is presumed to be continuous and extends approximately 300 m N-NW from the northern extent of the Marathon Deposit. Wide mineralized intercepts in holes MB-21-43 (0.82 g/t PdEq over 135 m) and MB-21-46 (1.10 g/t PdEq over 72 m) demonstrated the economic potential of this zone. The Powerline West Occurrence represents near surface mineralization (<100 m vertical depth) that overlies the Chonolith Zone, and is highlighted by MB-21-44 (1.08 g/t PdEq over 80 m) and MB-21-45 (1.78 g/t PdEq over 46 m). Both the Chonolith and powerline West Occurrences remain open down dip and along strike. Assay results from selected drill holes (shown as g/t PdEq) are shown in a plan view of the West Feeder Zone area and vertical sections 5406450N and 5406650N in Figure 10.6 to Figure 10.8. True widths approximate down-hole lengths.

Figure 10.6: Biiwobik Prospect – Plan View of the 2021 Drill Locations



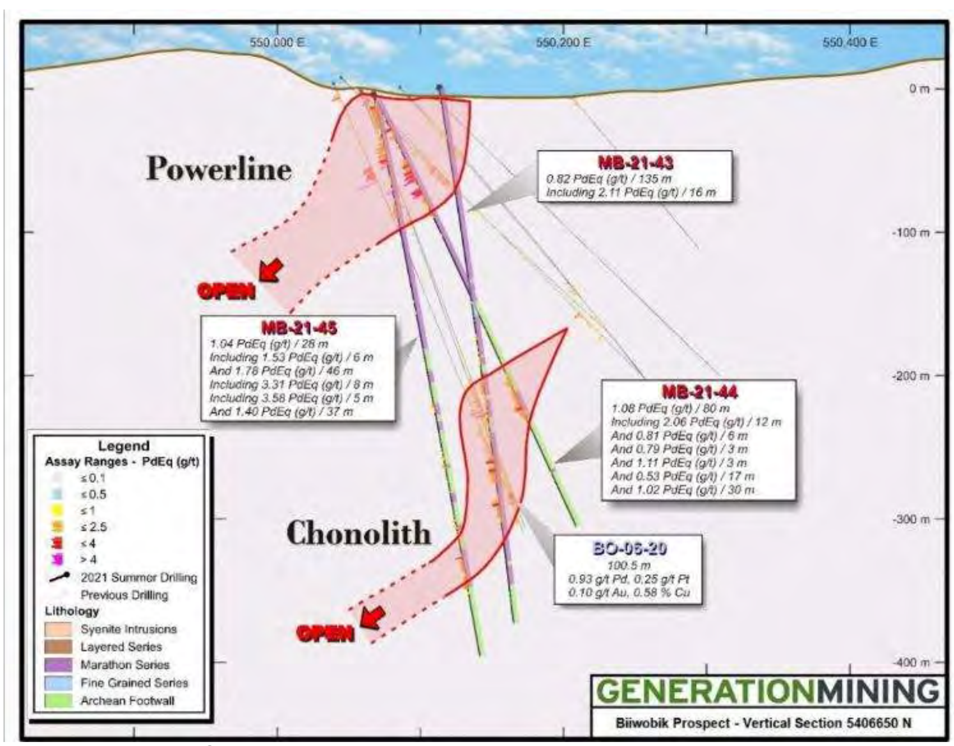
__Source__: Gen Mining (News release November 8, 2021).
__Note__: The Marathon Deposit Mineral Resource pit shell shown is that from the 2021 FS

Figure 10.7: Biiwobik Prospect – Vertical Section 5406450



***Source**: Gen Mining (News release November 8, 2021).*

Figure 10.8: Biiwobik Prospect – Vertical Section 5406650



***Source**: Gen Mining (News release September 2, 2021).*

10.4 2022 Resource Definition Drilling Program

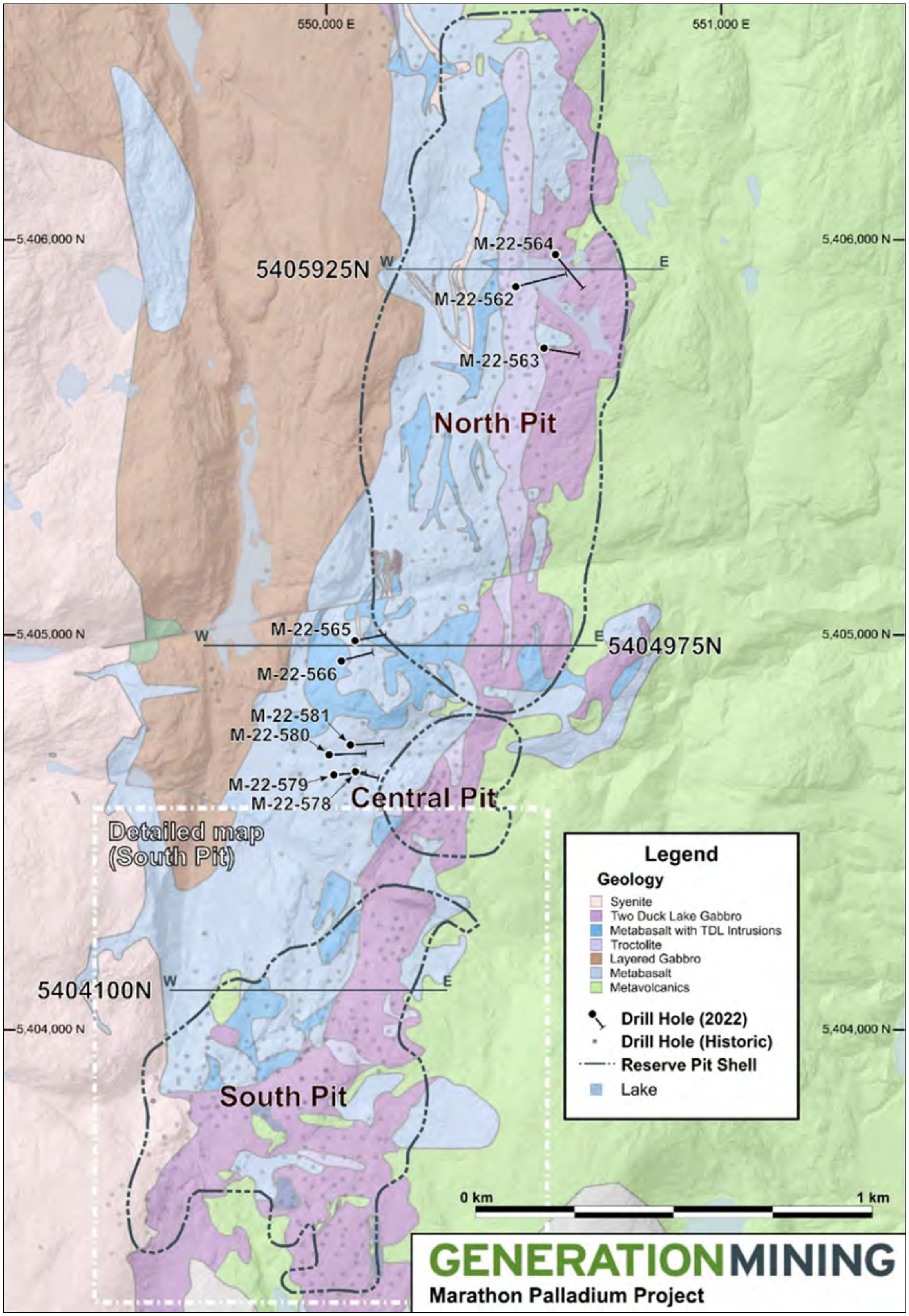
In 2022, the Company completed a drilling campaign focused within the Marathon Mineral Resource in areas of lower drill density (Table 10.4).

Table 10.4: 2022 Drilling Program

Deposit	Target	Holes Drilled	Metres Drilled
Marathon	North Pit	5	821
Marathon	Central Pit	12	2,210
Marathon	South Pit	31	4,296
Marathon	Central Feeder	2	741
Marathon	M-21-551 Extension	1	125
Total		**51**	**8,193**

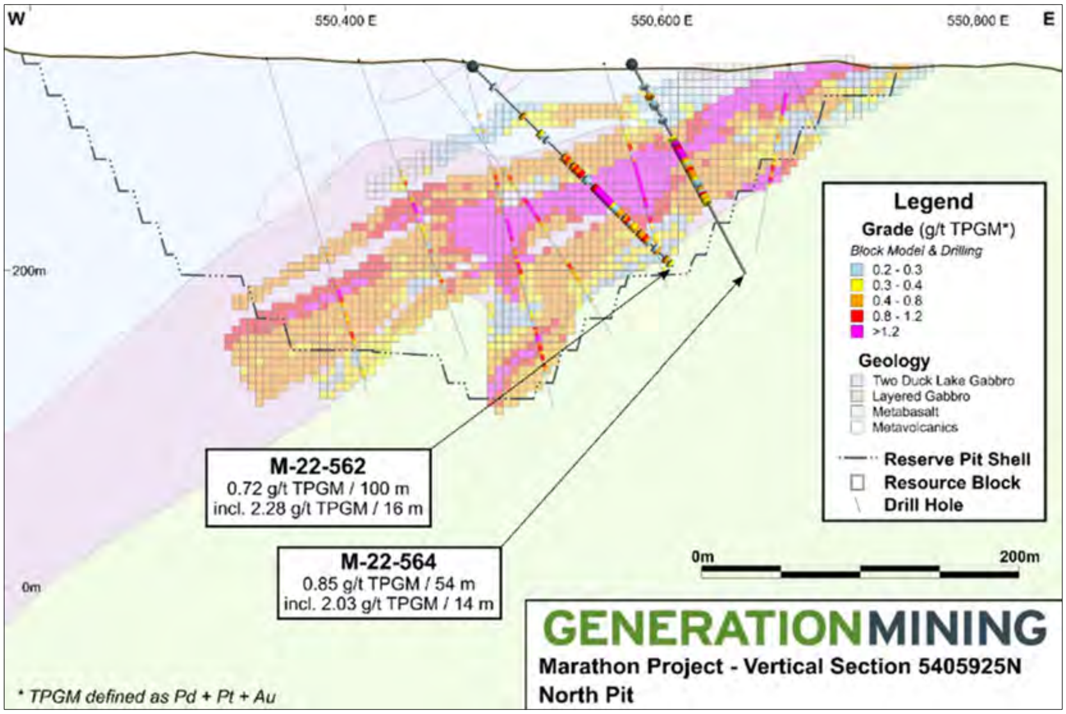
The majority of the 2022 drill program was aimed at de-risking the Mineral Resources by testing and confirming high-grade mineralization scheduled in the first three years of mining. Assay results from selected drill holes in the North and Central Pit (shown as g/t Total PGM) are shown in plan view and vertical sections 5405925N and 5404975N in Figure 10.9 to Figure 10.11. Assay results from selected drill holes in the South Pit (shown as g/t Total PGM) are shown in plan view and vertical section 5404100N in Figure 10.12 and Figure 10.13. Total PGM is the sum of Pd, Pt, and Au in g/t, and makes no provisions for Cu.

Figure 10.9: North and Central Pit Areas - Plan View of the 2022 Drill Locations



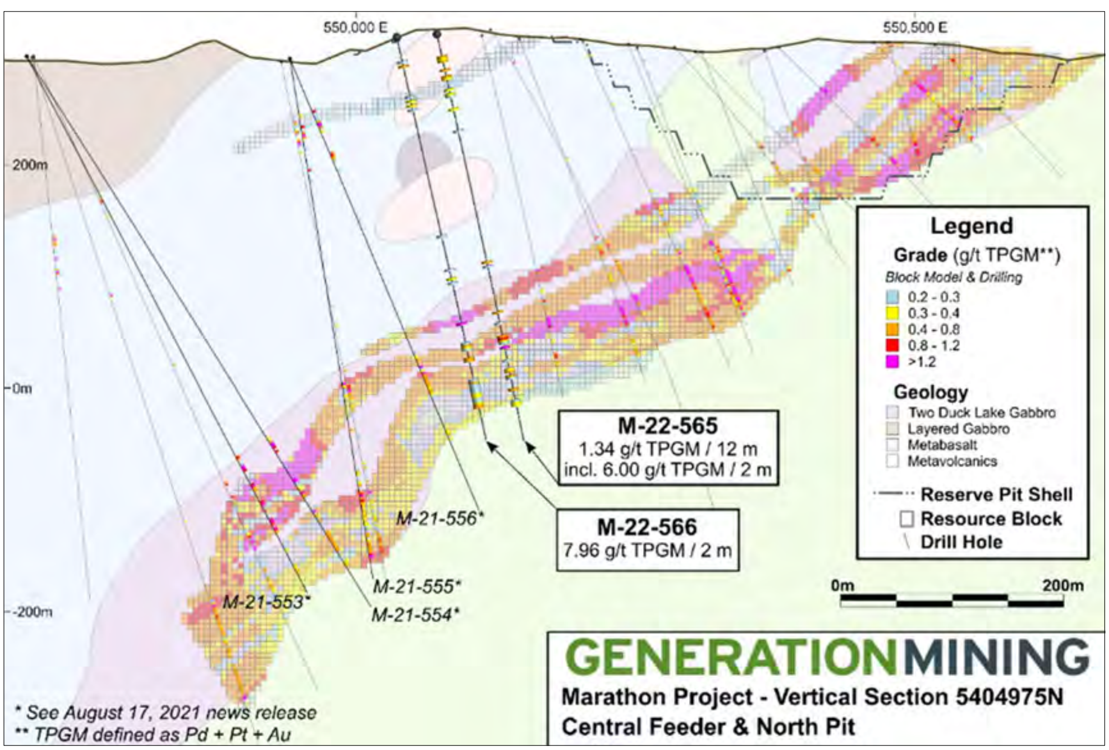
***Source**: Gen Mining (2022)*

Figure 10.10: North Pit - Vertical Section 5405925N



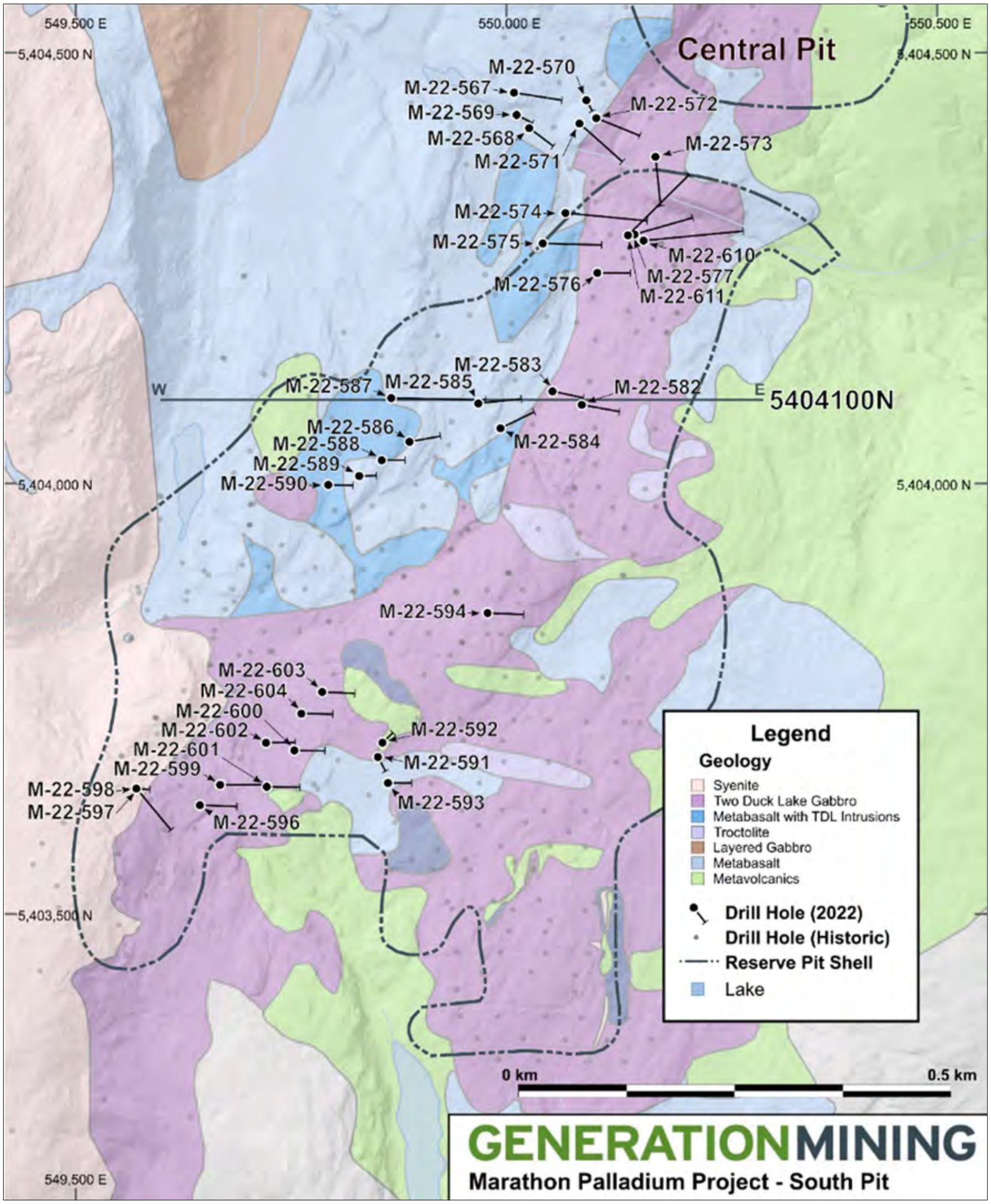
***Source**: Gen Mining (2022)*

Figure 10.11: Central Feeder - Vertical Section 54049750N



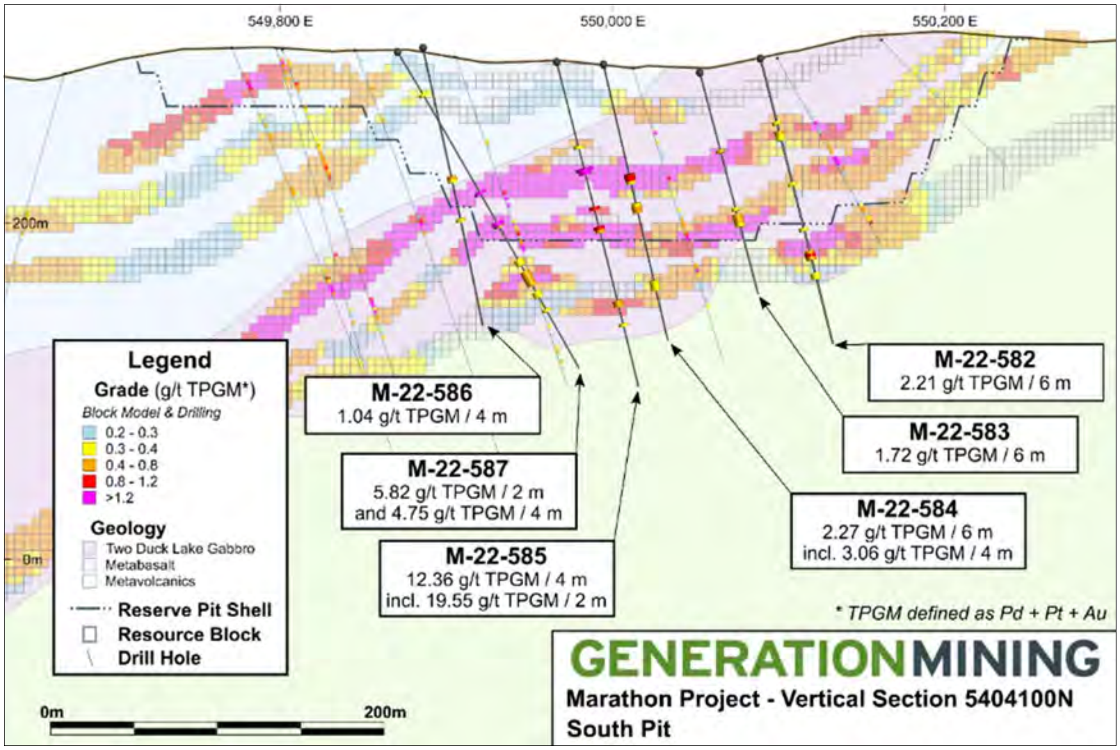
***Source**: Gen Mining (2022)*

Figure 10.12: South Pit Area - Plan View of the 2022 Drill Locations



***Source**: Gen Mining (2022).*

Figure 10.13: South Pit - Vertical Section 5404100N



***Source**: Gen Mining (2022)*

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation, Analysis and Security

The following section of this Technical Report is largely taken from the 2010 Technical Report completed by Python Mining Consultants Inc. ("Python") and the 2014 internal FS report completed by Nordmin Engineering Ltd. ("Nordmin") and outlines sampling protocol (preparation, analysis and security procedures) instituted and used by Marathon PGM in each of their drilling and other rock sampling programs since at least 2007. These protocols are identical to those reported in earlier NI 43-101 Technical Reports issued by Marathon PGM and Gen Mining on the Property.

11.1.1 Sampling Method and Approach

During the earlier drilling campaigns, drill core was logged and sampled on the Property. In more recent campaigns (including the Company's 2019, 2020, 2022 drilling programs), drill core was transported from the Property to a drill core logging facility in the Town of Marathon. A geologist was responsible for logging the drill core and marking sample intervals. Samples were collected at 1 or 2 m intervals in all significant mineralized zones and from known mineralized rock units. Some samples were shortened to less than 1 m at the logging geologist's discretion and samples from known non-mineralized zones were sampled at up to 3 m intervals. Sampling was continuous wherever possible to minimize potential continuity problems during Mineral Resource modeling. At least two samples were collected before and after each mineralized domain in order to estimate dilution.

The beginning and end of each sample was marked with a wax crayon, and then a sample tag was placed at the beginning of each sample. The drill core was also marked with a line along the length of the drill core to indicate where the drill core was to be cut in half. The drill core was then cut using a wet saw with a diamond blade. One half was placed in a sample bag and sent for assay and the other half remained in the box as a permanent record or in some cases to be utilized for additional metallurgical test work. The duplicate samples were prepared by splitting the remaining halved core leaving only quartered drill core in the box.

Each sample bag had a numbered identification ("ID") tag placed inside along with the sample before being sealed. The sample ID number was also written on the outside of the sample bag. The position of the samples on the remaining half drill cores was marked with a corresponding ID tag. Samples were then grouped into batches before being placed into rice bags. Each rice bag was also sealed and labeled before being dispatched. From 2011 to present, samples were no longer grouped in batches, instead each drill hole had its own batch.

The sealed rice bags were kept on site in a secure storage area until the batch was ready to be shipped, at which point they were delivered directly by Company personnel or placed on pallets and shipped via courier to the ALS prep facilities in Thunder Bay. Upon arrival at the lab, ALS personnel would check each rice bag to ensure all seals were in place and there was no sign of tampering, and report back any damaged or missing samples.

11.1.2 Laboratory Protocols

Prior to 2011, all drill core samples were sent for preparation and analysis to Accurassay in Thunder Bay. From 2011 to 2022, all drill core samples were sent for preparation to ALS Minerals in Thunder Bay and subsequent analysis to the ALS Vancouver facility.

During the 2006 to 2010 drilling campaigns, samples were delivered either by Marathon PGM personnel or shipped via courier to Accurassay's facilities (acquired by AGAT Laboratories ("AGAT") in 2017) in Thunder Bay, Ontario. When samples were deemed to be high priority, they were transported from the Property by helicopter to the Greyhound Bus Lines station in the Town of Marathon, from where they were shipped via bus to Accurassay in Thunder Bay. Upon receipt of the samples, Accurassay personnel would ensure that the seals on rice bags and individual samples had not been tampered with.

Accurassay (now AGAT) is independent of Gen Mining and provides analytical services to the mining and mineral exploration industry. Accurassay has been accredited for analysis of Au, Pt, Pd, Cu, Ni and Co under ISO/IEC Guideline 17025 by the Standards Council of Canada and is registered under ISO 9001:2000 quality standard.

In 2011, Stillwater Canada changed assay labs and initiated analyses at ALS Chemex Labs Ltd. ("ALS Minerals") in Thunder Bay. ALS Minerals used a similar lab protocol with the exception that PGM analyses were conducted by ICP-AES instead of Atomic Absorption used at Accurassay.

At the time of delivery, the laboratory acknowledged receipt of the sample shipment being in good order and logged all samples into their Laboratory Information Management System ("LIMS"). Samples were both prepared and analyzed at the Accurassay or the ALS Minerals laboratory in Thunder Bay, Ontario.

All samples were analyzed for Cu, Ni, Ag, Au, Pt and Pd. Rh was requested on samples within an intersection of two or more consecutive samples with an NSR value greater than $8/t, as well as the two samples on either side of the intersection, even though the values were likely to be below detection limit. The two samples outside of the mineralized intersection were requested for dilution information purposes. Rh analysis was not completed for the 2022 drill program.

11.1.3 Sample Preparation

11.1.3.1 Accurassay

The samples provided to Accurassay by Marathon PGM were drill core samples, rock samples and pulp samples. The samples were dried, if necessary, crushed to approximately minus 10 mesh and split into 250 to 450 g sub-samples using a Jones Riffler. The sub-samples were then pulverized to 90% passing 150 mesh using a ring and puck pulverizer and homogenized prior to analysis. Silica sand cleaning between each sample was performed to prevent cross-contamination.

Fire Assay Precious Metals

For flame atomic absorption spectroscopy ("AAS") determinations, preliminary concentration for Au, Pt and Pd by fire assay (lead collection) was the preferred method. The standard operating procedure for fire assaying at Accurassay involved weighing, fluxing, fusion and cupellation of each sample.

A 30.2 g sample mass was routinely used, although select sample masses may have been altered to accommodate sample chemistry, if required.

A furnace load consisted of 23 or 24 samples with a check done every 10^{th} sample (by client ID), along with a laboratory blank and a Quality Control Standard. Duplicate checks were performed on pulverized samples.

Samples provided to Accurassay by Marathon PGM did not require preliminary treatment and were mixed directly with the assay flux and fused. Accurassay used a premixed basic flux purchased from Reliable Industrial Supply. The composition of the flux is as follows: Litharge (PbO), 50.4%, soda ash (dense), 35.9%, borax, 10%, and silica flour, 3.6%. It is standard practice for laboratories to use a premixed flux and adjust the ingredients when necessary.

Samples were typically fused for 1¼ hr at 1,800 to 2,000°F. The fusion time may have been increased if needed.

Samples were typically cupelled for 50 minutes at 1,900°F. The cupellation time may have been increased if needed.

Digestion – Precious Metals

Precious metal beads were digested using a nitric/hydrochloric acid digestion at Accurassay and bulked up with a 1% lanthanum oxide ("La_2O_3") solution and distilled water. The use of lanthanum in the concentration of 0.2-1.0% is an acceptable practice and complies with accepted published methods. A final volume of 3 ml was used for analysis.

Digestion – Base Metals

For flame AAS determinations of Cu, Co, Ni, Pb, and Ag at Accurassay, an acid digestion consisting of aqua regia (one part nitric to three parts hydrochloric acid) was the preferred method. A sample mass of 0.25 g and a final volume of 10 ml was used for analysis. For samples requiring a full assay digestion (high grade); a sample mass of 2.5 g and a final volume of 250 ml was used. A full assay was required whenever the concentration of any given element was greater than 1% for any of the above noted elements.

Flame Atomic Absorption Spectrometric Measurement

Accurassay used a Varian AA240FS with manual sample introduction for the determination of Au, Pt and Pd. A Varian 220FS or 240FS with SIPS and auto-diluter was used for the determination of base metals.

Calibration certified reference materials (CRMs) were made from 1,000 ppm certified stock solutions. Quality assurance ("QA") solutions were made up from separately purchased 1,000 ppm certified stock solutions. All stock solutions were prepared commercially by ISO certified suppliers.

Reporting

Laboratory reports were produced using Accurassay's LIMS program. All duplicate assays were reported on the certificate of analysis. Quality control ("QC") CRM and blanks were not reported unless requested by the client.

Control Charts for Quality Control Certified Reference Materials

All Accurassay data generated for QC CRM, blanks and duplicates were retained with the client's file and used in the validation of results. For each QC CRM, control charts were produced to monitor the performance of the laboratory. Warning limits were set at +/-2 standard deviations, and control limits were set at +/-3 standard deviations. Any data points for the quality control CRM that fell outside the warning limits, but within the control limits, required 10% of the samples in that batch to be re-assayed. If the results from the re-assays matched the original assays the data were validated, if the re-assay results did not match the original data, the entire batch was rejected, and new re-assays were performed. Any QC CRM that fell outside the control limits was automatically re-assayed and all of the initial test results were rejected.

Certified Reference Materials

Accurassay's in-house CRMs used for Au, Pt, Pd and Rh was made up from a rock source provided to Accurassay by a third party. The CRM names were APG1 and APP7. The CANMET CRM used for the analysis of Au, Pt, Pd and Rh were WMS-1 and WMG-1. All CRMs used to certify base metal values were provided by CANMET. The following CRMs were used: CZN3, RTS-2, and RTS-3.

Two in-house CRMs (MPG1 and MPG2) were used for control of Au, Pt, Pd and Cu determinations. The CRMs were made up from a composite of core sample reject material provided to Accurassay by Marathon

PGM from the Marathon Deposit and were representative of the metal abundances in the Coldwell Complex deposits. The values for MPG1 and MPG2 were developed by Accurassay and verified through round-robin analysis with other laboratories in Canada.

The QA sample was made in the laboratory from certified stock solutions purchased from an ISO 9000 certified supplier. The solution was made from a completely different lot number than the solutions used to calibrate CRM. The QC CRMs were used to monitor the processes involved in analyzing the samples. The QA samples were used to verify the initial calibration of the instruments and monitor the calibration throughout the analysis.

It should be noted that although a CRM or QA standard may not have been listed by batch number on the control charts, a CRM and QA sample was run with each batch.

The values for APG1 and APP7 were developed by Accurassay and verified through round-robin analysis with other laboratories in Canada. The values for CANMET certified reference materials were obtained from their respective certificates of analysis.

11.1.3.2 ALS Minerals

Since 2011, all drill core samples were sent to ALS Minerals sample preparation facility in Thunder Bay. Pulp sample material was then sent to the Vancouver ALS facility for analysis. ALS Minerals is independent of Gen Mining and operates with a quality management system and complies with the requirements of ISO 9001:2008. The quality management system of ALS is audited both internally and by external parties.

The samples were prepared and sent for multi-element analyses (Table 11.1).

Table 11.1: Sample Analysis Methods

Procedure	Description	Element Analyzed and Range (ppm)		
Prep 31	Crush to 70% less than 2 mm, riffle split off 250 g, pulverize split to better than 85% passing 75 microns.			
PGM-ICP23	Pt, Pd and Au by fire assay and ICP-AES finish. 30 g nominal sample weight.	Pt 0.005-10 Pd 0.001-10 Au 0.001-10		
ME-ICP41	Aqua Regia Digestion – first pass exploration tool, dissolution of base metals.	Ag 0.2-100 W 10-10,000 Ca 0.01%-25% La 10-10,000 Sb 2-10,000 Zn 2-10,000 Cd 0.5-1,000 Mg 0.01%-25% Sc 1-10,000 Co 1-10,000 Mn 5-50,000 Sr 1-10,000	Al 0.01%-25% Cr 1-10,000 Mo 1-10,000 Th 20-10,000 As 2-10,000 Cu 1-10,000 Na 0.01%-10% Ti 0.01%-10% B 10-10,000 Fe 0.01%-50% Ni 1-10,000 Tl 10-10,000	Ba 10-10,000 Ga 10-10,000 P 10-10,000 U 10-10,000 Be 0.5-1,000 Hg 1-10,000 Pb 2-10,000 V 1-10,000 Bi 2-10,000 K 0.01%-10% S 0.01%-10%
OG46-OL	Aqua regia is a powerful solvent for sulphides, which dissolves Ag and base metals but may not completely dissolve more resistive elements. Minimum sample weight 0.5 g.	Ag 1-1,500 ppm As 0.001-60 Cd 0.001-10 Co 0.0005-30 Cu 0.001-50 Fe 0.01-100	Mn 0.01-60 Mo 0.001-10 Ni 0.001-30 Pb 0.001-20 S 0.01-10 Zn 0.001-30	
S-IR08 OL for S > 10%	Total sulphur by combustion furnace.	Total S 0.01% - 50%		

Source: *Geochemistry Service Schedule (2022).*

11.1.4 Conclusions

It is is this section report Author's opinion that the sample preparation, analysis, and security measures taken at the Marathon, Geordie and Sally Deposits were adequate.

11.2 Marathon Deposit Quality Assurance/Quality Control

11.2.1 2009 and 2011 Programs

Marathon PGM continued with a robust QA/QC program that had been implemented by that company in the mid-2000s. The QA/QC program consisted of the insertion of reference materials, field blanks and duplicate pair monitoring.

Two CRMs, named MPG1 and MPG2, were prepared by Accurassay in Thunder Bay. Material was sourced from the Marathon Project. 375 samples were analyzed for the characterization of MPG1, and 325 samples were analyzed for the characterization of MPG2. Mean and standard deviation values were calculated for each reference material.

All data from the 2009 and 2011 drill programs were examined by the Authors. Drill data prior to 2009 was previously examined by the Authors and accepted for use in previous Mineral Resource estimates.

11.2.1.1 Performance of Reference Materials

For the 2009 data, there were 31 data points for MPG1 and 18 data points for MPG2. All data points fell between +/- two standard deviations from the mean for Au, Cu, Pd and Pt.

For the 2011 data, there were 35 data points for MPG1 and 32 data points for MPG2. All data points fell between +/- two standard deviations from the mean.

11.2.1.2 Performance of Blank Material

The blank material used for the 2009 and 2011 programs was commercially prepared nepheline syenite sand. There were 49 data points in 2009 and 68 in 2011. All blank results were below five times the detection limit for the commodity in question.

11.2.1.3 Performance of Duplicates

There were 81 pulp duplicate pairs analyzed at ALS Chemex for Au, Pt and Pd for the 2011 drilling program. All duplicate pairs were plotted on a simple scatter graph. The precision on the gold pulp pairs was acceptable, with less precision (as is to be expected) on the very low grades. Both platinum and palladium demonstrated excellent precision at the pulp level. There were no duplicates available for copper.

11.2.2 Surface Trench Samples

The Marathon Deposit database contains 4,479 surface trench sample assays collected from channels that were cut by saw along lines spaced 30 to 50 m apart along approximately 2 km strike length. The channels were cut in approximately straight lines located close to and perpendicular to the base of the Marathon Deposit during the years 1985, 1986, 2005 to 2009 and 2021.

After a comparison of the trench samples with the diamond drill holes in the same vicinity, the channel samples were included in the Mineral Resource estimate. In a report titled, "Trench vs. Core Assay Data in the Marathon Deposit Main Zone," authored by D. Good, Ph.D., P. Geo. (March 18, 2012), it was clearly shown that channel samples should not be excluded from the database since a sampling bias could not be observed. The test sample set included channel samples cut from a relatively Pd-rich zone of the Main Zone, and when compared to the core samples drilled in the immediate vicinity, there was no sampling bias demonstrated. The Author of this technical report section has reviewed the report by Dr. Good and has accepted the methodology and conclusions.

11.2.3 Gen Mining 2019 to 2022 Drilling Programs

11.2.3.1 Performance of Certified Reference Materials

The analyses for elements Au, Pt, Pd, Ag and Cu for CRMs MPG1 and MPG2 are plotted in Figure 11.1 to Figure 11.10.

A few minor outliers beyond the set control limits can be noted; however, the overall performance of both CRMs for all elements was excellent and no bias or temporal variation in the 2019-2022 data were noted.

Figure 11.1: Performance of CRM MG1 for Au

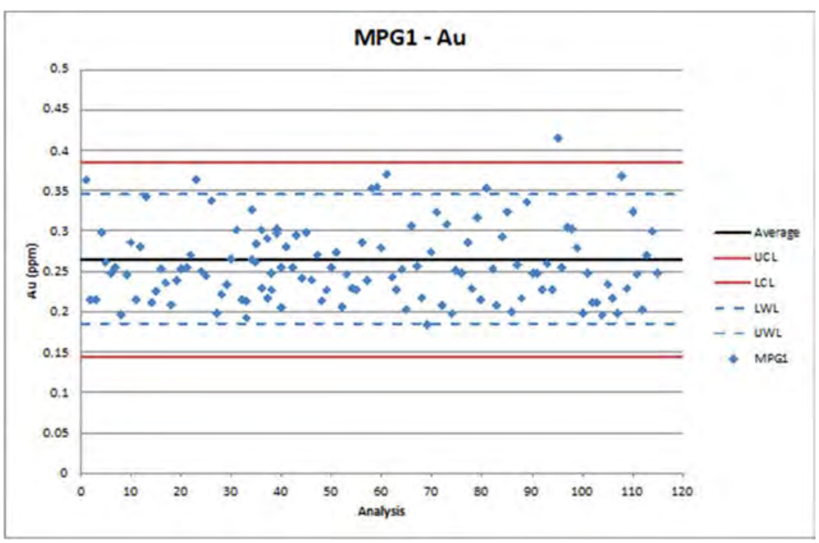


Figure 11.2: Performance of CRM MPG1 for Pt

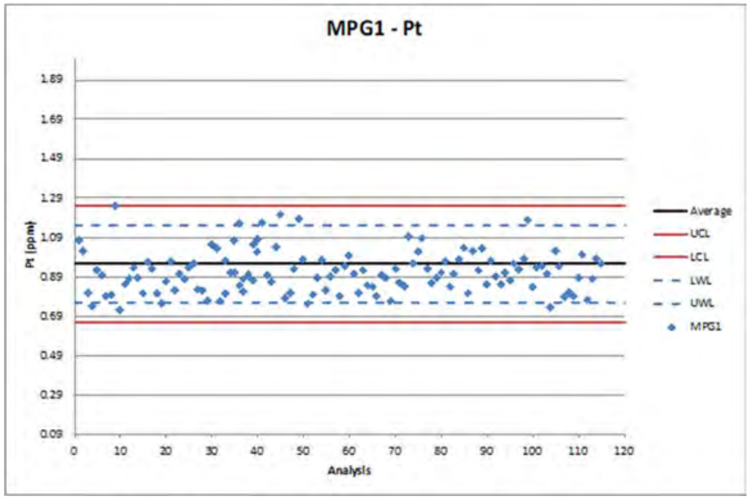


Figure 11.3: Performance of CRM MPG1 for Pd

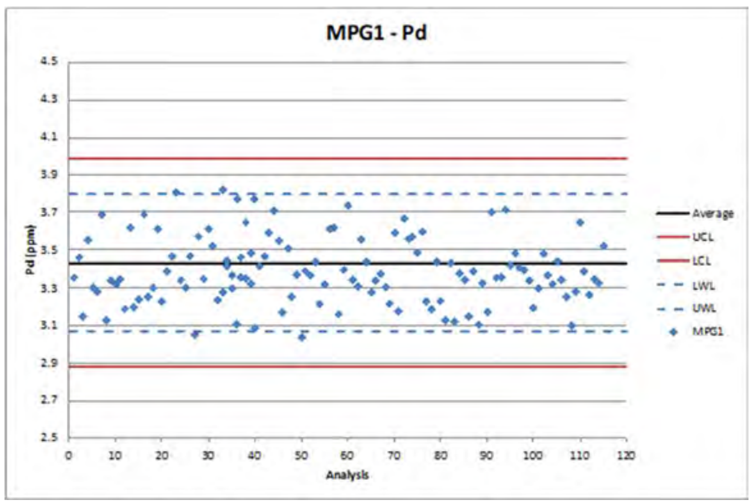


Figure 11.4: Performance of CRM MPG1 for Ag

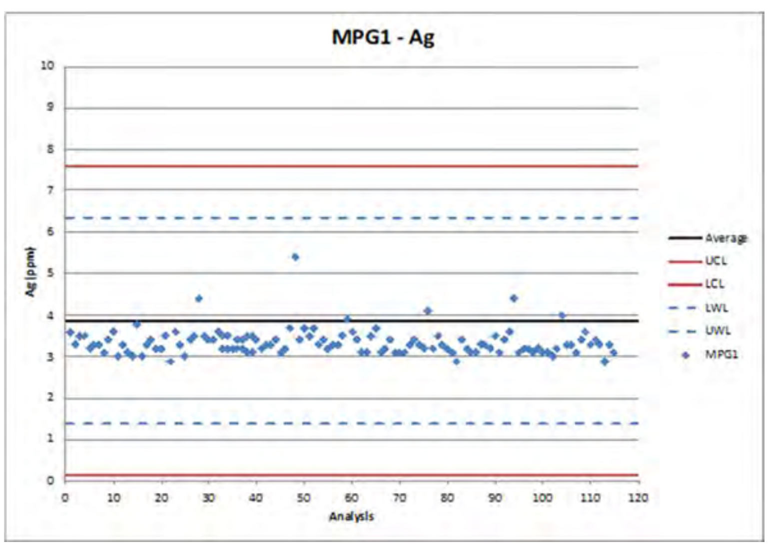


Figure 11.5: Performance of CRM MPG1 for Cu

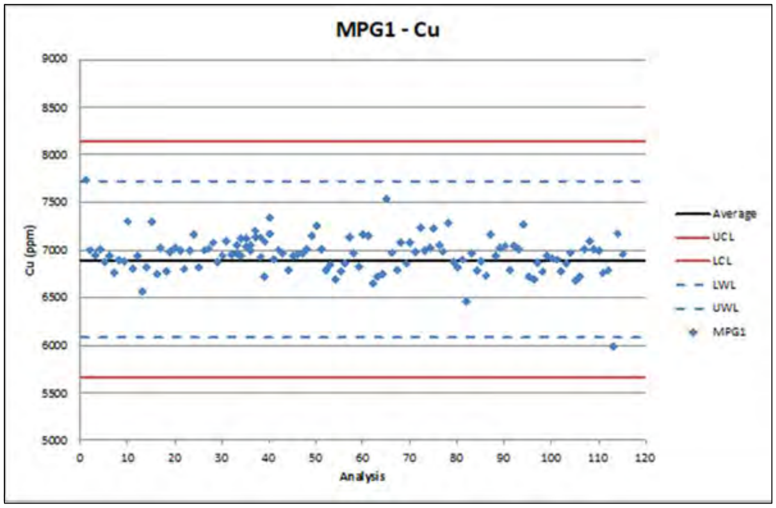


Figure 11.6: Performance of CRM MPG2 for Au

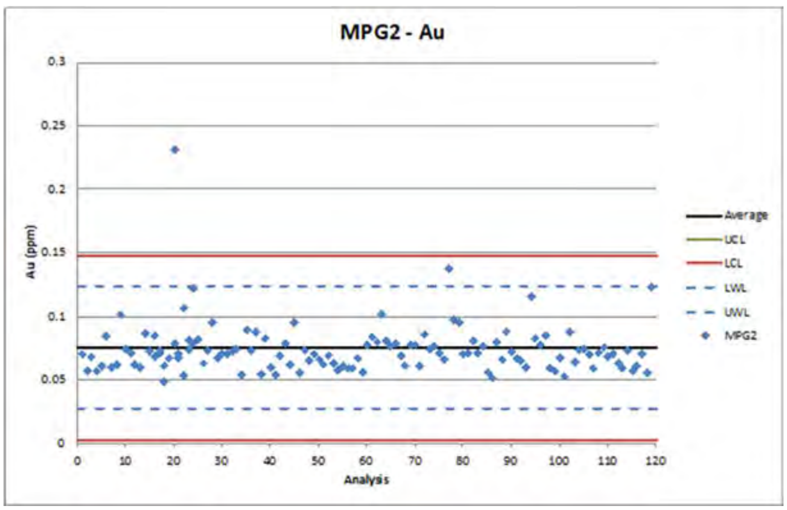


Figure 11.7: Performance of CRM MPG2 for Pt

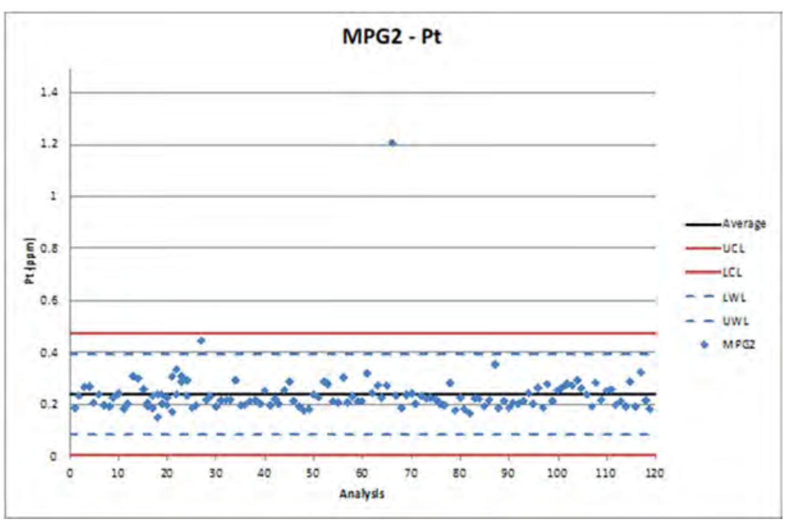


Figure 11.8: Performance of CRM MPG2 for Pd

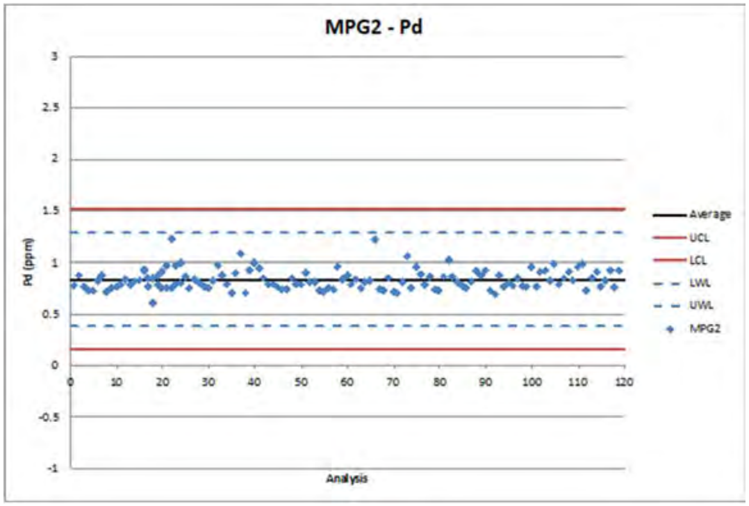


Figure 11.9: Performance of CRM MPG2 for Ag

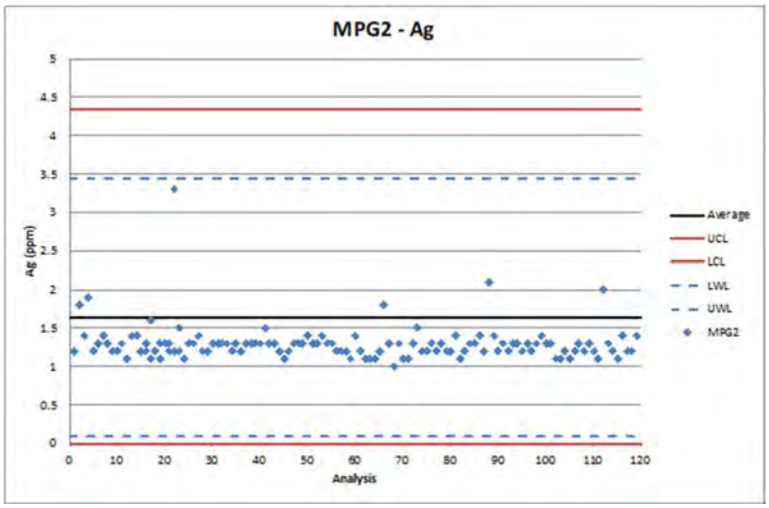
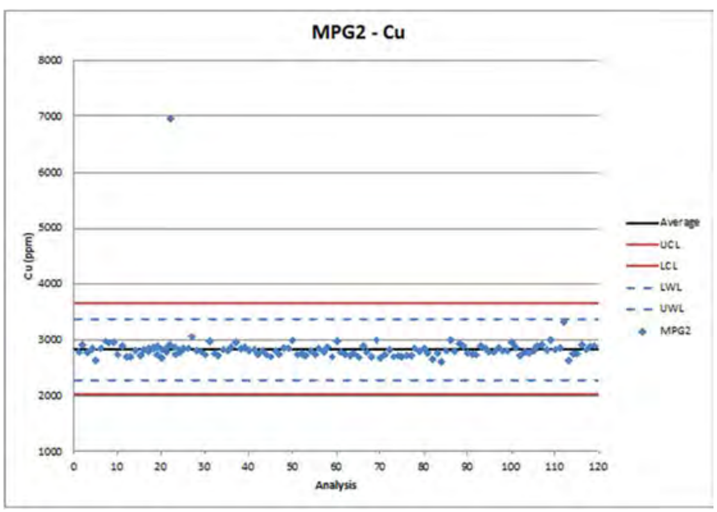


Figure 11.10: Performance of CRM MPG2 for Cu



11.2.3.2 Performance of Blank Material

The results of the blank sample analyses were considered excellent, with the vast majority of the Au, Pt, Pd, Ag and Cu determinations falling below the respective upper working limit of two times the standard deviation of the mean of each element (Figure 11.11 to Figure 11.15). The occasional result falling above the upper working limit was not considered to be of material impact to the Mineral Resource Estimate and contamination was not considered to be an issue with the 2019, 2020 and 2022 data.

Figure 11.11: Performance of Blanks for Au

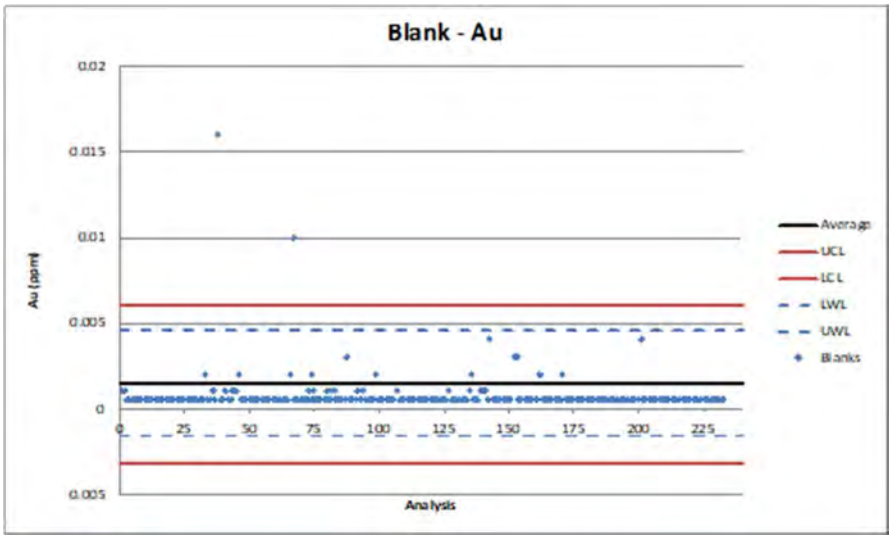


Figure 11.12: Performance of Blanks for Pt

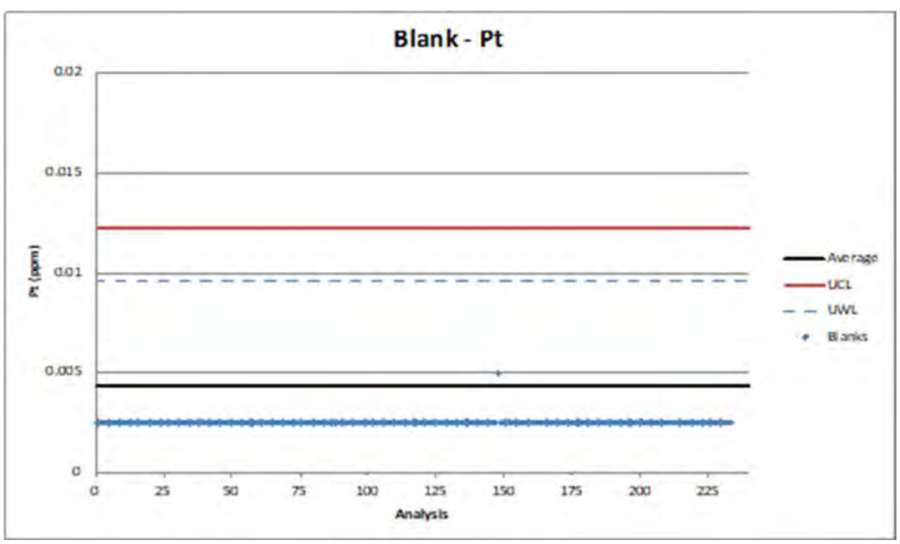


Figure 11.13: Performance of Blanks for Pd

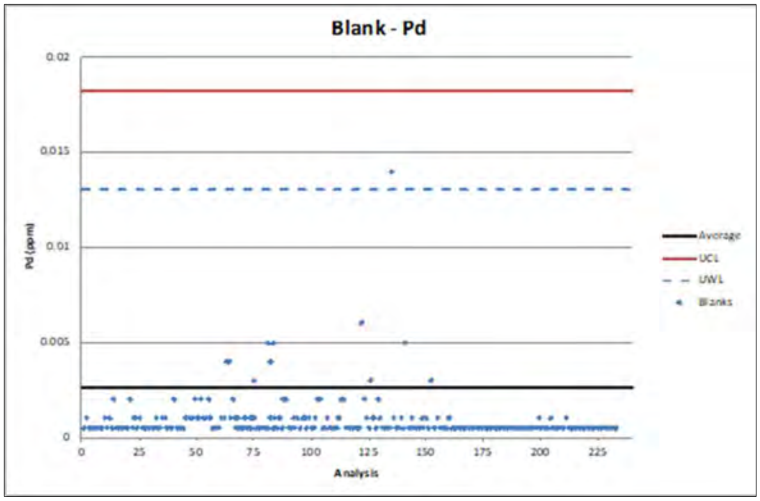


Figure 11.14: Performance of Blanks for Ag

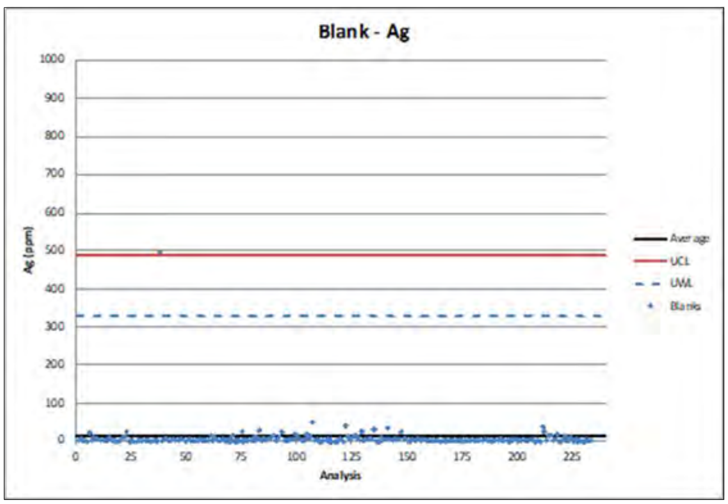
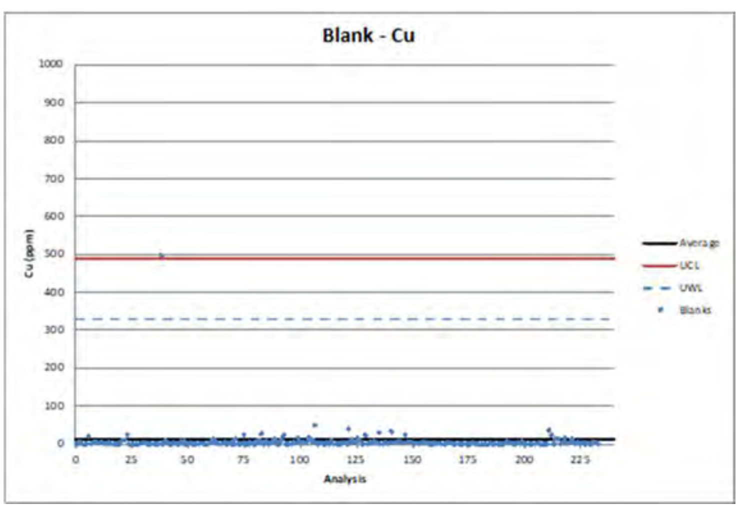


Figure 11.15: Performance of Blanks for Cu



11.2.3.3 Performance of Duplicates

The field duplicate data for Au, Pt, Pd, Ag and Cu were plotted on scatter plots and compared with the laboratory duplicate data (Figure 11.16 to Figure 11.20). Precision for all elements is shown to increase with the reduction in grain size from field to lab, as expected, and precision at laboratory level, as demonstrated by R^2 values; all of which considered satisfactory by the Author of this technical report section.

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Figure 11.16: Field and Laboratory Duplicates for Au

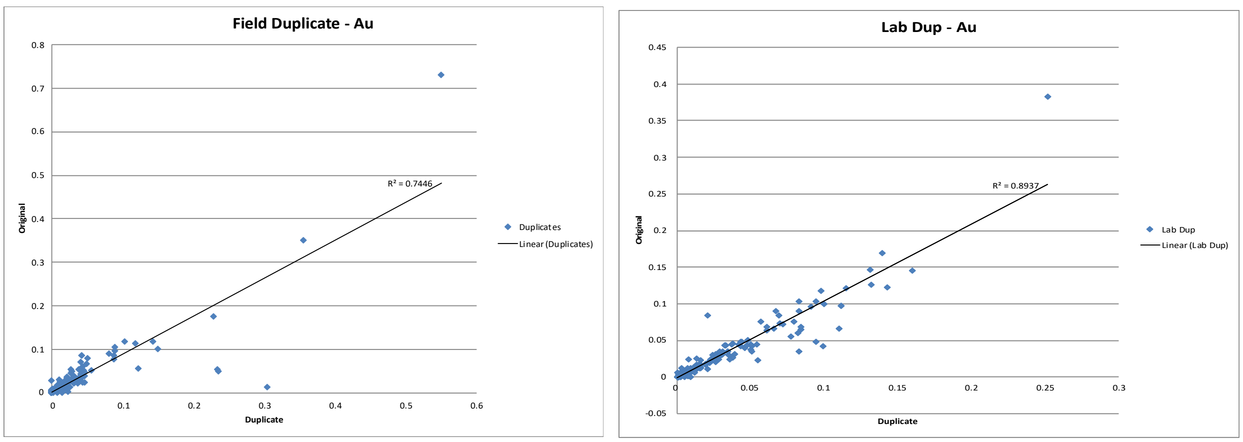


Figure 11.17: Field and Laboratory Duplicates for Pt

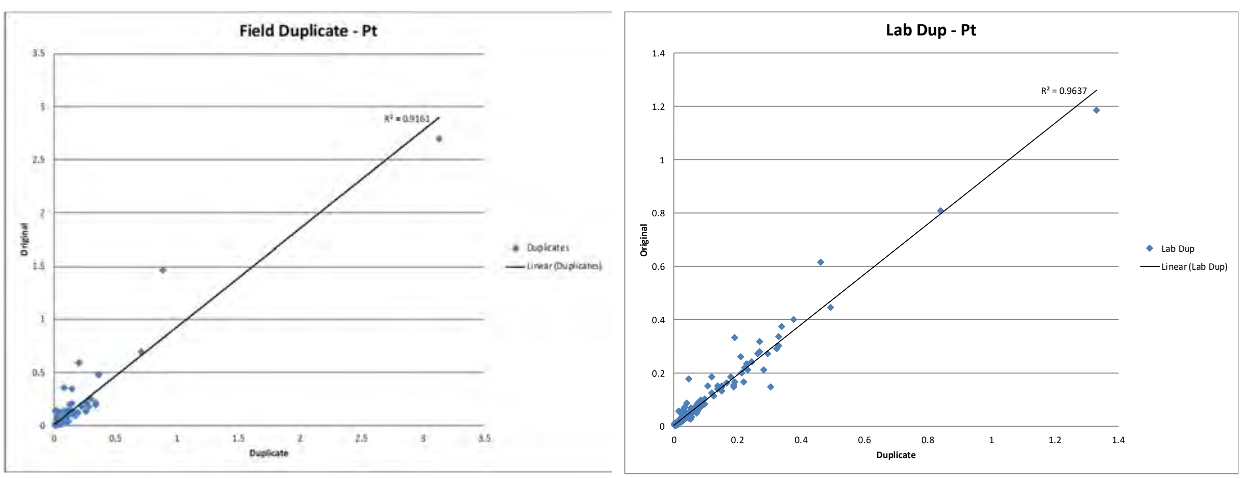


Figure 11.18: Field and Laboratory Duplicates for Pd

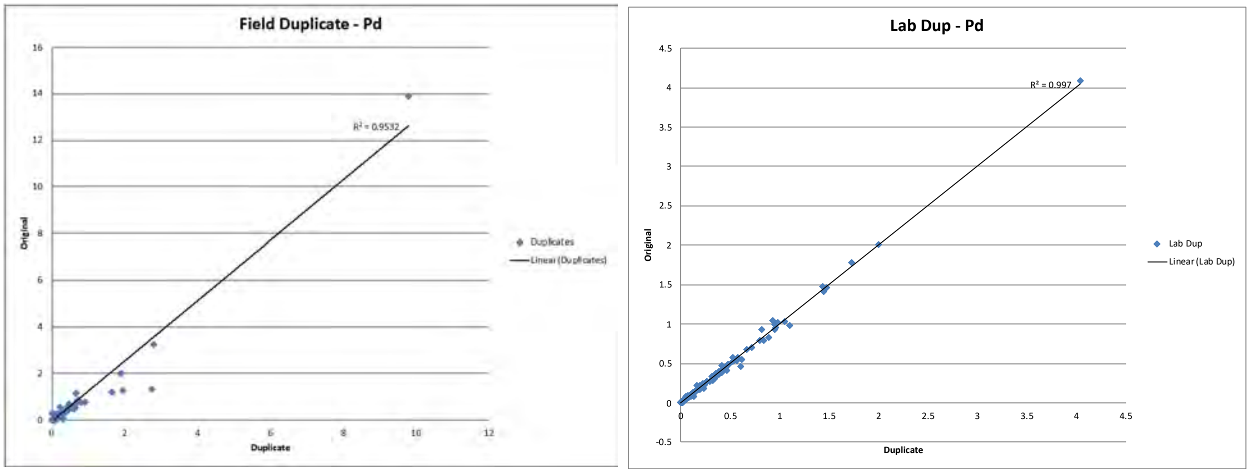


Figure 11.19: Field and Laboratory Duplicates for Ag

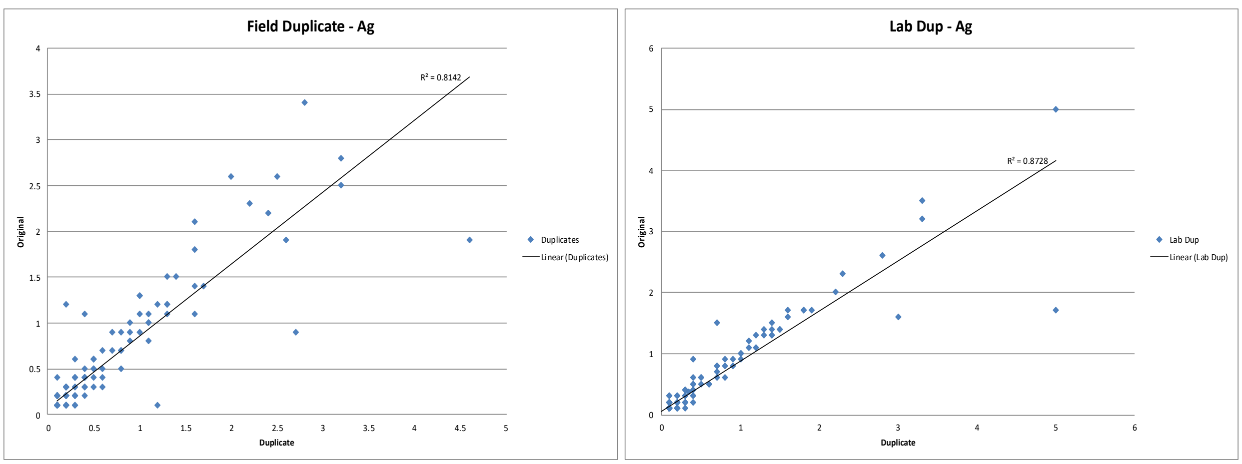
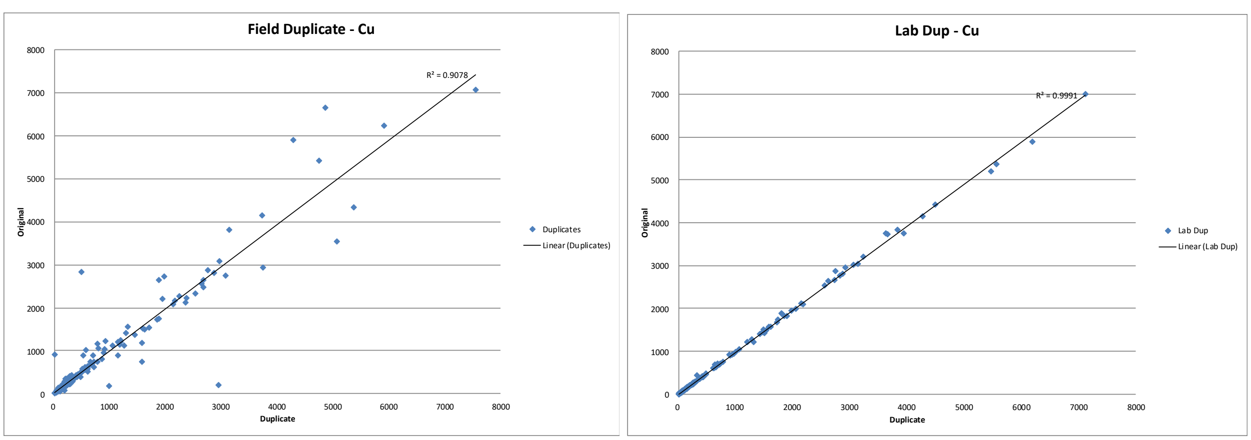


Figure 11.20: Field and Laboratory Duplicates for Cu



11.2.3.4 Laboratory Quality Control

The Author of this technical report section has reviewed the corresponding laboratory QC data from 2019 to 2022 drilling programs, including CRMs, blanks and duplicates, and does not consider that the laboratory QC data indicates issues with data accuracy, contamination or precision.

11.2.4 Conclusions

The Author of this technical report considers the Marathon Deposit data to be of good quality and acceptable for use for Mineral Resource estimation.

11.3 Geordie Deposit Quality Assurance/Quality Control

11.3.1 2010 Program

11.3.1.1 Performance of Certified Reference Materials

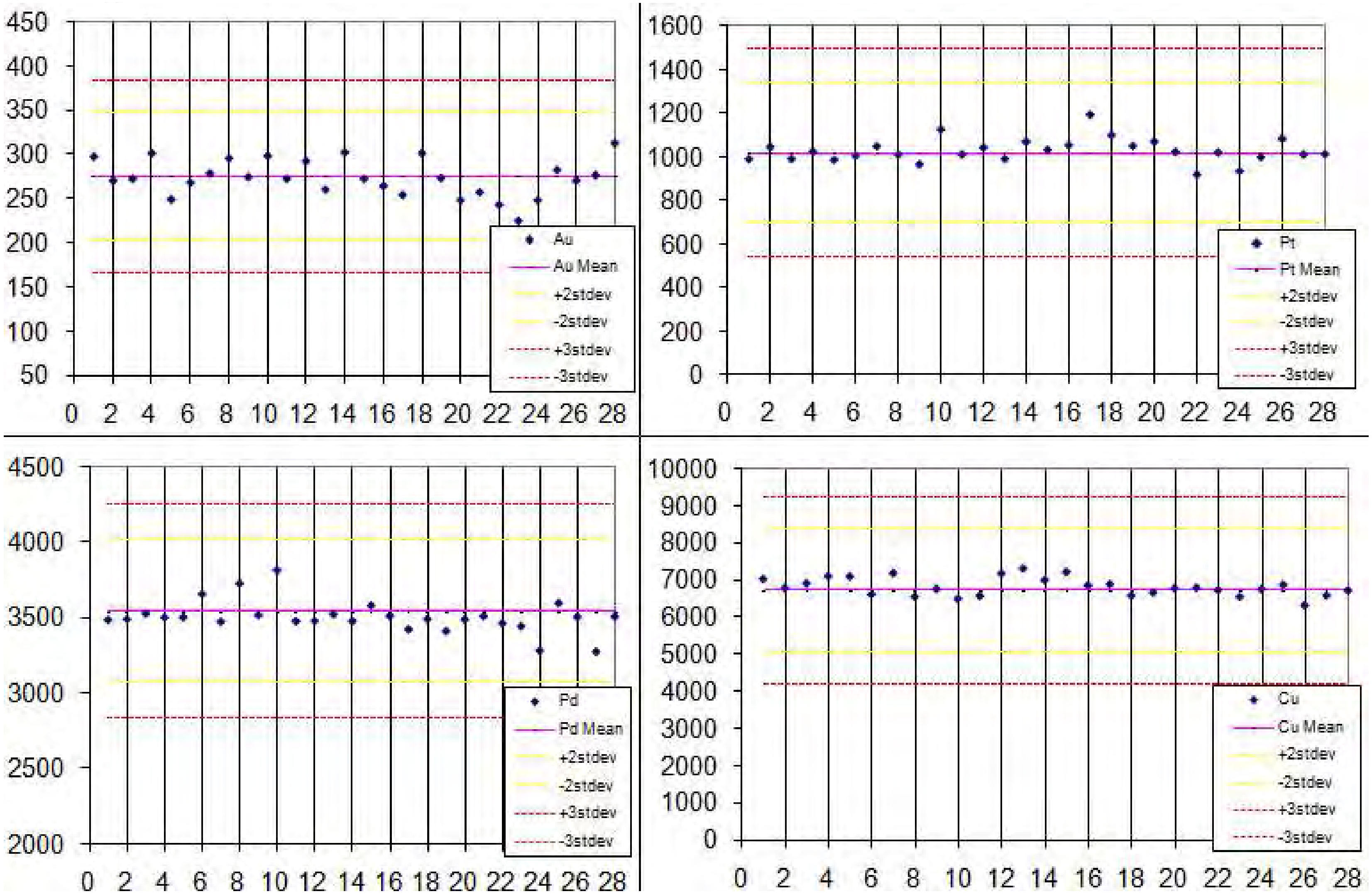
All data generated for QC CRMs, blanks and duplicates were used in the validation of results. For each QC standard, control charts were produced to monitor the performance of the laboratory. Warning limits were set at ±2 standard deviations, and control limits were set at ±3 standard deviations. If two consecutive data points for the QC CRM fell outside the warning limits, but within the control limits, 10% of the samples in that batch were to be re-assayed. If the results from the re-assays matched the original assays the data was validated, if the re-assay results did not match the original data the entire batch was rejected, and new re-assays were performed. Any QC CRM that fell outside the control limits was automatically re-assayed and all of the initial test results were rejected.

As can be noted in the control charts shown in Figure 11.21 and Figure 11.22, none of the Cu, Au or Pd results fell outside of the warning limit and only one of the Pt results fell between the warning limit and the control limit. Consequently, no action was considered necessary.

The results of the MPG1 CRM tests are shown in Figure 11.21. All values are in ppb except Cu in ppm. As shown in the figure, no determination fell outside of the 2x detection limit (warning) boundary and there was no sample drift during the period.

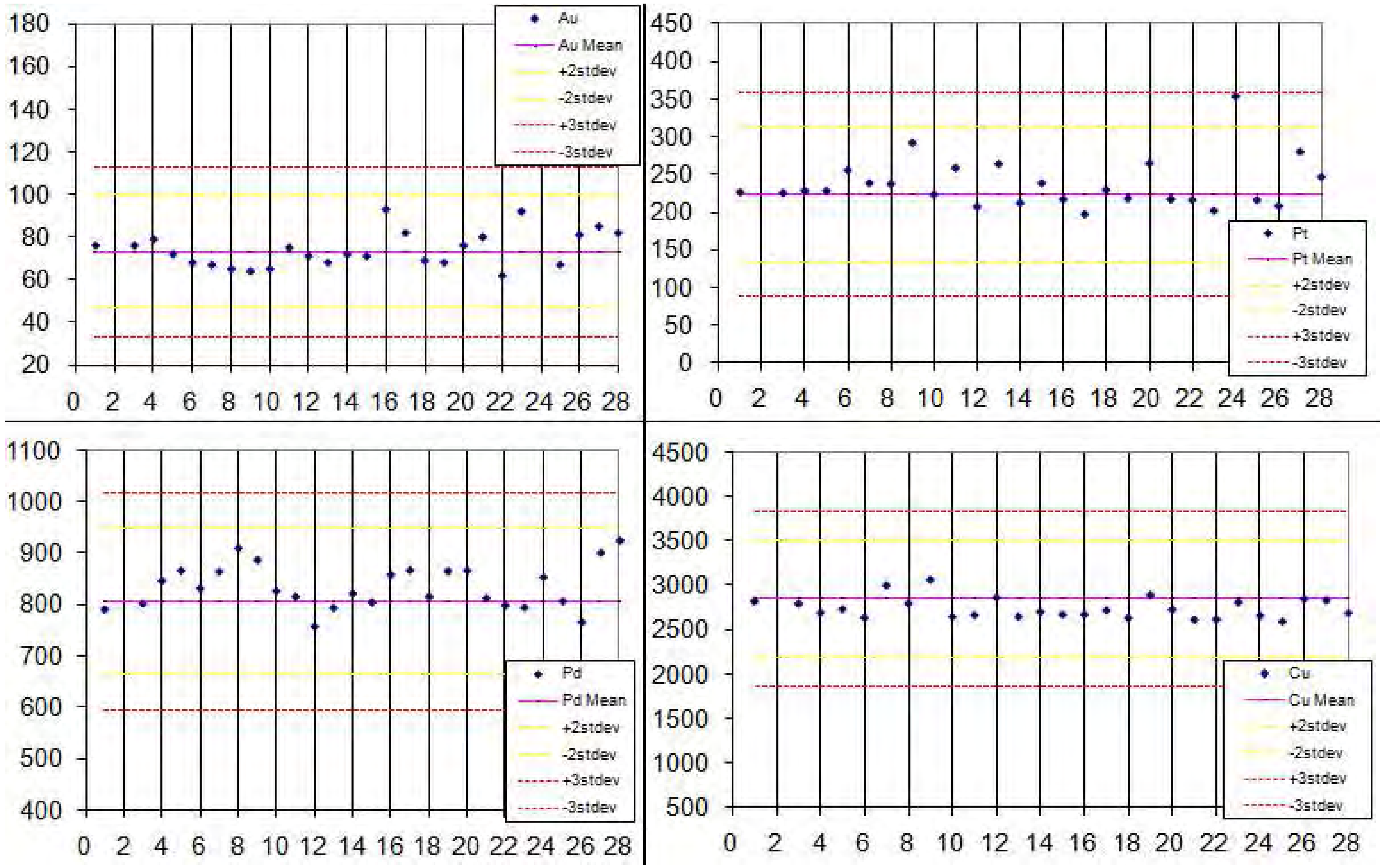
The results of the MPG2 CRM tests are shown in Figure 11.22. All values are in ppb except Cu in ppm. As shown in the figure, only one determination fell outside of the 2x detection (warning) limit boundary and there was no sample drift evident during the period observed. No action was taken for the batch where Pt falls outside of the warning limit.

Figure 11.21: Determinations for In House Standard MPG1



Source: Python (2010).

Figure 11.22: Determinations for In House Standard MPG2



Source: *Python (2010).*

11.3.1.2 Performance of Blank Material

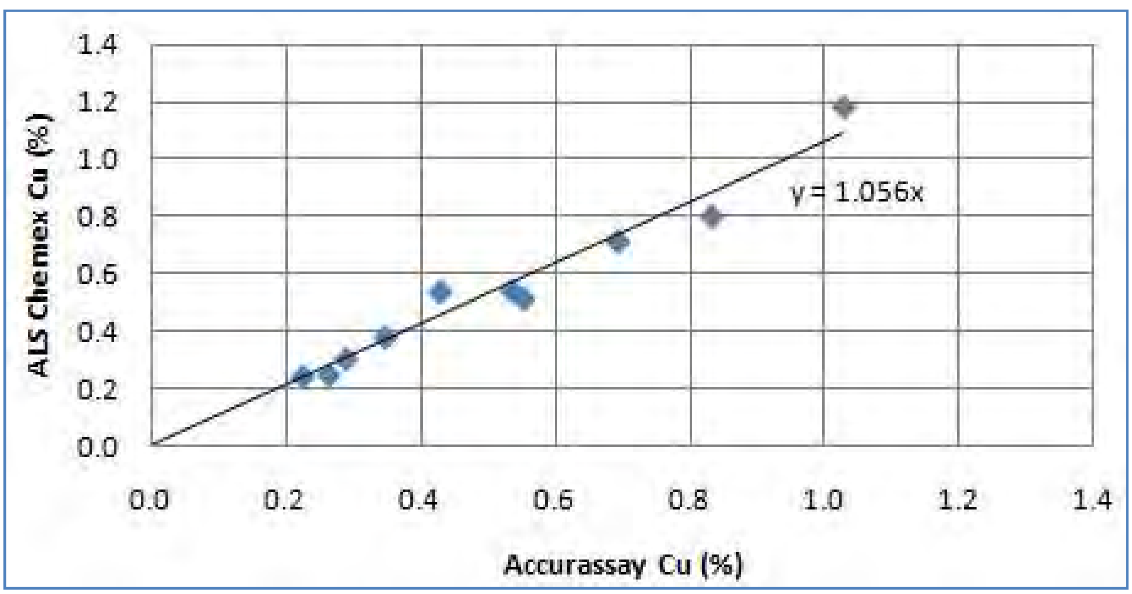
Every sample batch (consisting of 22 or 23 samples) shipped to Accurassay, contained a single blank sample. The blank material comprised 40 g of pulverized nepheline syenite, obtained from "B and L" in Thunder Bay. To verify the quality of the blank material, 10 samples were tested at ALS Chemex to ensure the viability of this material.

The results of the 56 blank sample analyses were considered excellent, with all of the Au, Pt and Pd determinations at or below the detection limits of 5, 15 and 10 ppb, respectively. Three blank Cu determinations returned results of 6, 7 and 46 ppm (greater than three times the detection limit of 1 ppm), however, these elevated results were still considered acceptable levels of contamination and of no material impact. Therefore, no action was necessary for these three batches.

11.3.1.3 Performance of Pulp Duplicates

To further verify the accuracy of Cu determinations carried out by Accurassay, a total of 10 pulp samples selected from the two main host rocks (units 3a and 3b), with a varying range of Cu grades, were submitted to ALS Minerals in Thunder Bay for comparison analysis. Results of the duplicate analyses are shown in Figure 11.23 and Table 11.2. Two samples returned 15% to 25% higher values from the ALS Minerals; however, the results are considered acceptable.

Figure 11.23: Comparison Chart of ALS and Accurassay Cu Results



***Source**: Python (2010).*

Table 11.2: Duplicate Pulp Analyses from Accurassay and ALS Chemex

Hole_ID	Sample No.	From (m)	To (m)	Zone	Cu % (AA)	Cu % (ALS)	ALS-AA (%)
Heterogeneous Gabbro (unit 3a)							
G10-01	870004	10.00	12.00	MZ	0.55	0.51	-7.4%
G10-02	870059	66.00	68.00	MZ	0.35	0.38	9.2%
G10-03	870090	42.00	44.00	HW	0.29	0.30	4.5%
G10-04	870149	142.00	144.00	MZ	0.43	0.54	25.1%
G10-13	870620	184.00	186.00	MZ	0.69	0.71	2.8%
Heterogeneous Gabbro (unit 3b)							
G10-03	870084	32.00	34.00	MZ	0.54	0.54	1.1%
G10-07	870258	60.00	62.00	HW	0.26	0.25	-6.4%
G10-10	870433	180.00	182.00	MZ	1.03	1.19	15.2%
G10-11	870504	186.00	188.00	MZ	0.22	0.24	7.0%
G10-13	870618	180.00	182.00	MZ	0.83	0.80	-3.8%

Note: *ALS-AA (%) = % difference in values of ALS compared with AA.*

11.3.2 Conclusions

The Author of this technical report considers the Geordie Deposit data to be of good quality and acceptable for use for Mineral Resource estimation.

11.4 Sally Deposit Quality Assurance/Quality Control

The QA/QC from the 2013 drill program through 2019 was established by means of an internal quality management system with a rotating sequence of duplicates, blanks and CRMs that are inserted for every 15th sample.

The blanks were created in-house using granular nepheline syenite sand purchased from Bell and Mackenzie Ltd. (Thunder Bay). Baggies of blank material were prepared in a clean environment.

11.4.1 2013 Drilling Program

11.4.1.1 Performance of Certified Reference Materials

Two CRMs (MPG1 and MPG2) were prepared and certified by Accurassay Laboratories in 2008 and used during the 2013 through 2019 drilling programs. The certified results for CRMs MPG1 and MPG2 are shown in Table 11.3 and Table 11.4.

The CRMs were prepared from sample rejects collected from drilling the Property in 2007 and 2008. The preparation and certification procedures used for MPG1 and MPG2 are described in an article by Wesley M. Johnson, in the Geostandards Newsletter, Vol. 15, No. 1, April 1991, p. 23 to 31, entitled “Use of Geochemical Reference Materials in A Quality Control/Quality Assurance Program”.

Table 11.3: CRM MPG1

Element	Average (ppb)	Standard Deviation (ppb)
Pd	3,538	236
Pt	1,019	160
Au	275	36
Cu	6,715	835
Ni	444	33
Co	70	5

Table 11.4: CRM MPG2

Element	Average (ppb)	Standard Deviation (ppb)
Au	70	13
Pt	223	45
Pd	805	71
Cu	2,853	329
Ni	318	28
Co	85	8

The analyses for elements Au, Pt, Pd, Ag and Cu for standards MPG1 and MPG2 are plotted in Figure 11.24 to Figure 11.33.

The mean value, standard deviation and lower and upper working limits (two standard deviations from the average) of both the MPG1 and MPG2 standards are presented in Table 11.5 and Table 11.6.

Table 11.5: CRM MPG1 Control Limits

	Au (ppm)	Pt (ppm)	Pd (ppm)	Ag (ppm)	Cu (ppm)	Ni (ppm)	S (%)
Average	0.261	0.914	3.334	3.320	6,982.89	375.495	1.115
Standard Deviation	0.056	0.101	0.203	0.268	339.049	19.2712	0.0593
Lower Working Limit	0.149	0.712	2.928	2.784	6,304.792	336.9526	0.9964
Upper Working Limit	0.372	1.116	3.740	3.856	7,660.98	414.037	1.233

Note: *Ag = silver, Au = gold, Cu = copper, Ni = nickel, Pd = palladium, Pt = platinum, S = sulphur.*

Table 11.6: CRM MPG2 Control Limits

	Au (ppm)	Pt (ppm)	Pd (ppm)	Ag (ppm)	Cu (ppm)	Ni (ppm)	S (%)
Average	0.0835	0.2503	0.8337	1.2396	2,860.879	277.6593	1.1777
Standard Deviation	0.0409	0.0883	0.0992	0.2043	130.0568	13.0896	0.0612
Lower Working Limit	0.0017	0.0737	0.6353	0.831	2,600.7653	251.4801	1.0553
Upper Working Limit	0.1653	0.4270	1.0322	1.6482	3,120.993	303.8386	1.3002

Note: *Ag = silver, Au = gold, Cu = copper, Ni = nickel, Pd = palladium, Pt = platinum, S = sulphur.*

As noted in Figure 11.24, there are some outliers beyond the upper control limit (example point 5229); however, individual outliers were isolated to a specific element and did not fail for all tested elements in the same sample. In addition, inspection of the internal CRM data determined by routine ALS Minerals procedure verified the analyses were sound and no further action was taken. There is a strong confidence for the analysis as data fell within the 95% confidence interval as seen in Figure 11.24 to Figure 11.33, and there was no systematic bias either above or below the recommended values, nor was there temporal variation in the data.

Figure 11.24: Performance of CRM MPG1 for Au

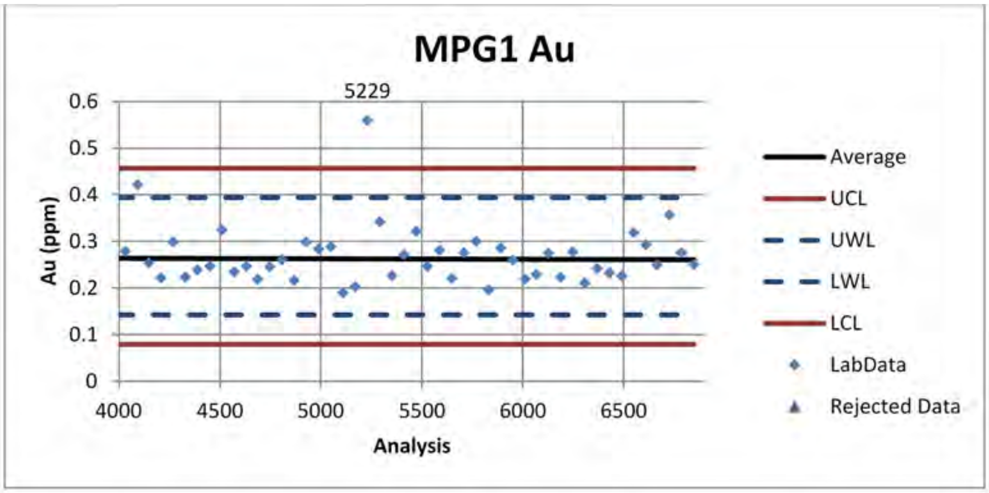


Figure 11.25: Performance of CRM MPG1 for Pt

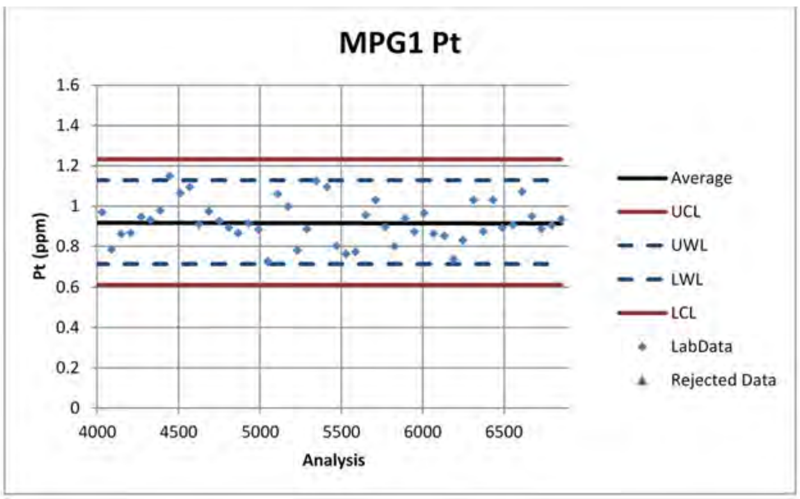


Figure 11.26: Performance of CRM MPG1 for Pd

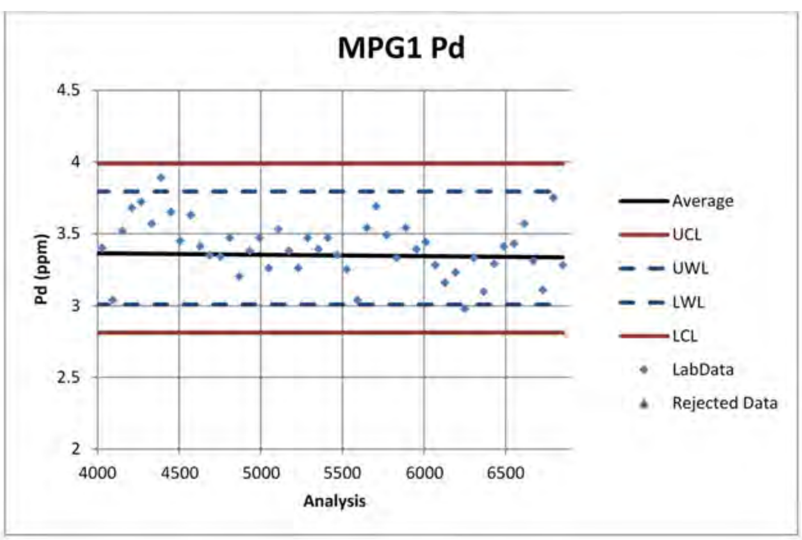


Figure 11.27: Performance of CRM MPG1 for Ag

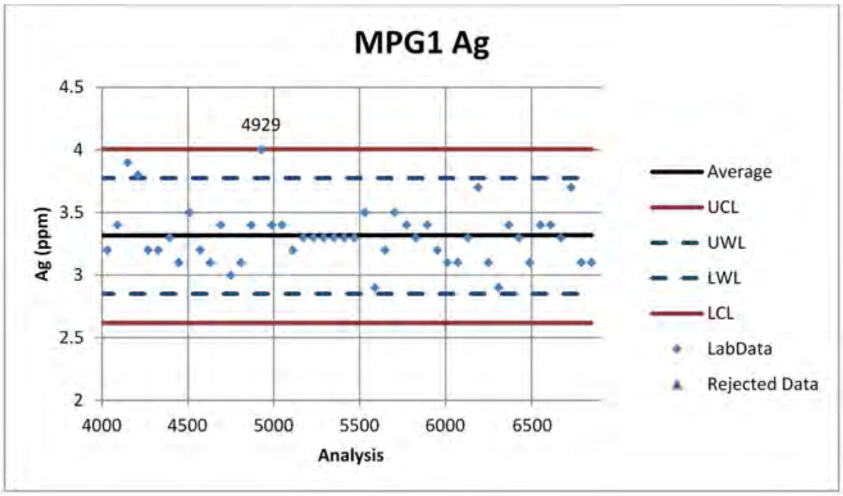


Figure 11.28: Performance of CRM MPG1 for Cu

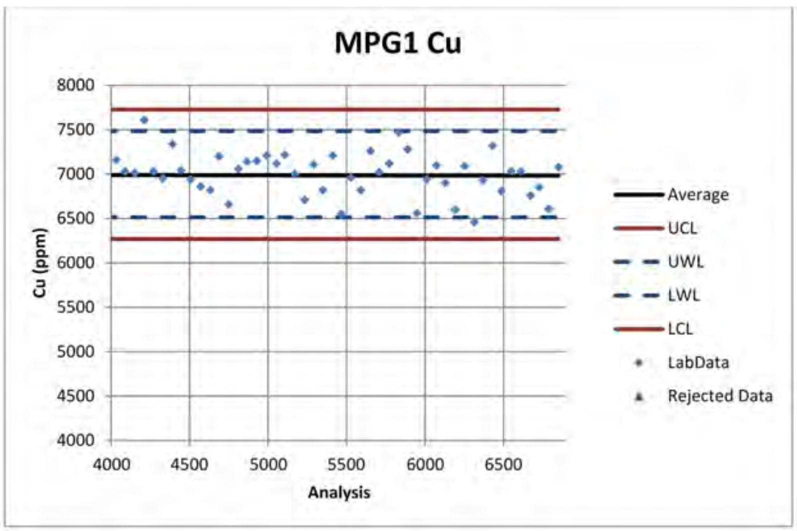


Figure 11.29: Performance of CRM MPG2 for Au

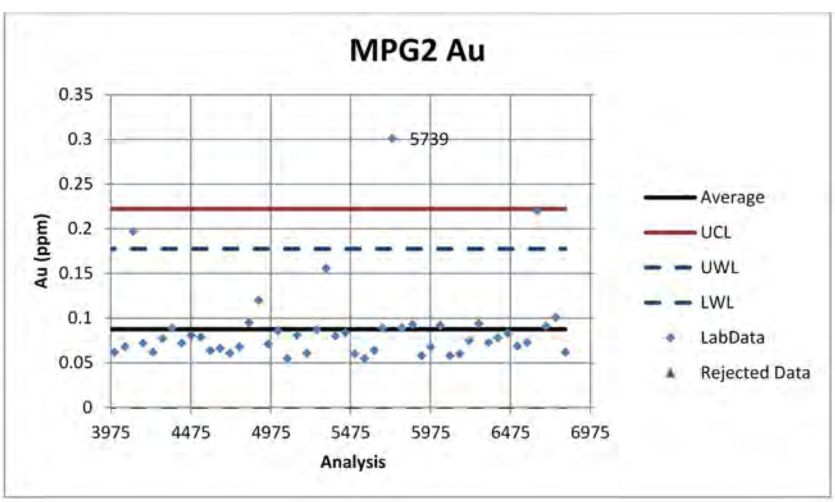


Figure 11.30: Performance of CRM MPG2 for Pt

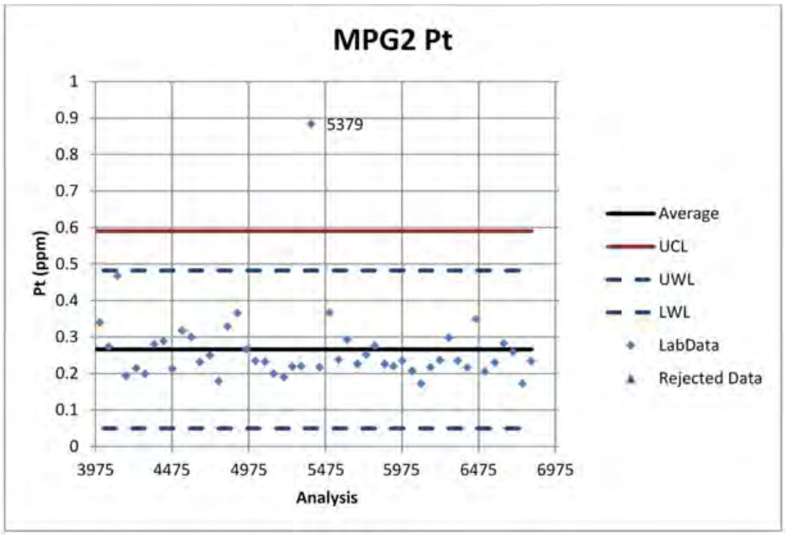


Figure 11.31: Performance of CRM MPG2 for Pd

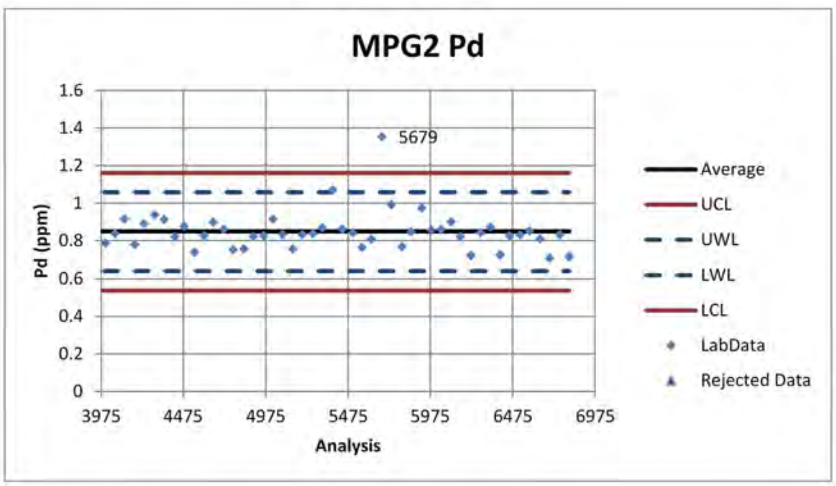


Figure 11.32: Performance of CRM MPG2 for Ag

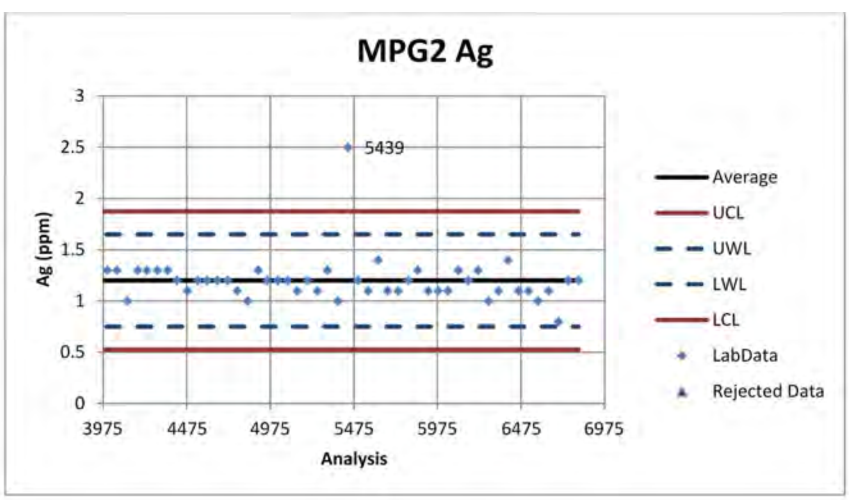
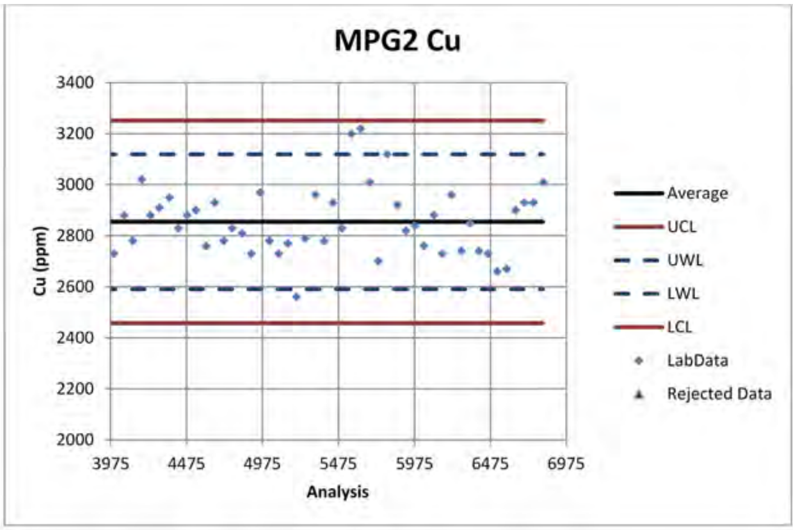


Figure 11.33: Performance of CRM MPG2 for Cu



11.4.1.2 Performance of Blank Material

The blanks were created in-house using granular nepheline syenite sand purchased from Bell and Mackenzie Ltd. (Thunder Bay). Baggies of blank material were prepared in a clean environment.

The mean value, standard deviation and upper working limits (two standard deviations from the average) of the blank material are presented in Table 11.7.

The results of the blank sample analyses were considered excellent, with the vast majority of the Au, Pt, Pd, Ag and Cu determinations falling below the respective upper working limit of two times the standard deviation of the mean of each element (Figure 11.34 to Figure 11.38). The occasional result falling above the upper working limit (as with sample 8621 in Figure 11.35) was not considered to be of material impact to the Mineral Resource estimate and contamination was not considered to be an issue in the 2013 data.

Table 11.7: Blank Control Limits

	Au (ppm)	Pt (ppm)	Pd (ppm)	Ag (ppm)	Cu (ppm)	Ni (ppm)	S (%)
Average	0.0013	0.0028	0.0011	0.1096	8.0593	2.9765	0.0176
Standard Deviation	0.0019	0.0009	0.0012	0.0327	8.7975	10.5980	0.0414
Upper Working Limit	0.0051	0.0046	0.0034	0.1751	25.6543	24.1726	0.1004

Note: *Ag = silver, Au = gold, Cu = copper, Ni = nickel, Pd = palladium, Pt = platinum, S = sulphur.*

Figure 11.34: Performance of Blanks for Au

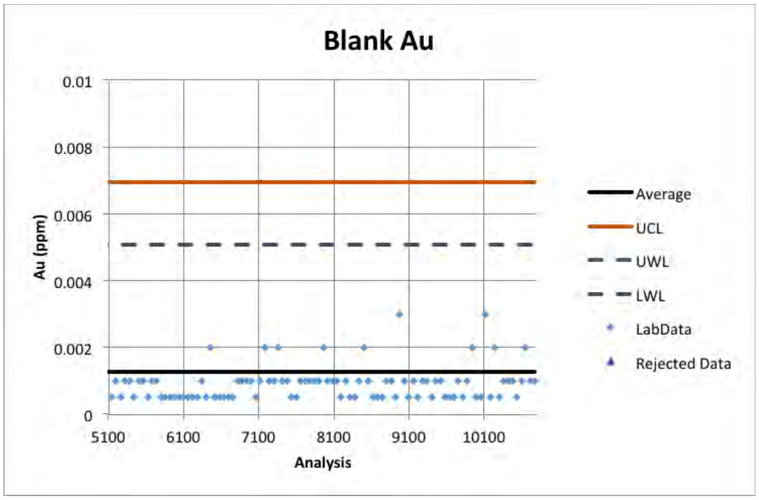


Figure 11.35: Performance of Blanks for Pt

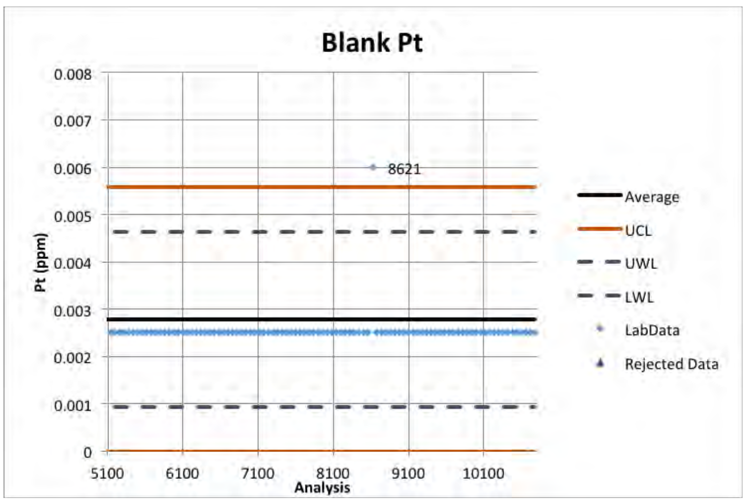


Figure 11.36: Performance of Blanks for Pd

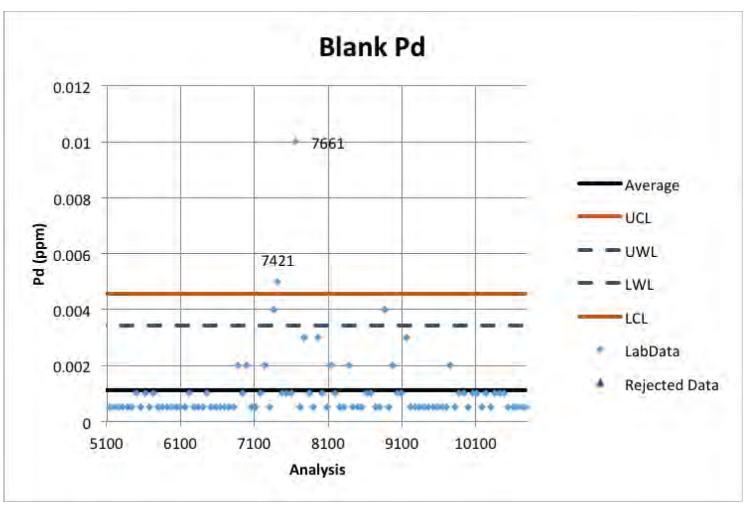


Figure 11.37: Performance of Blanks for Ag

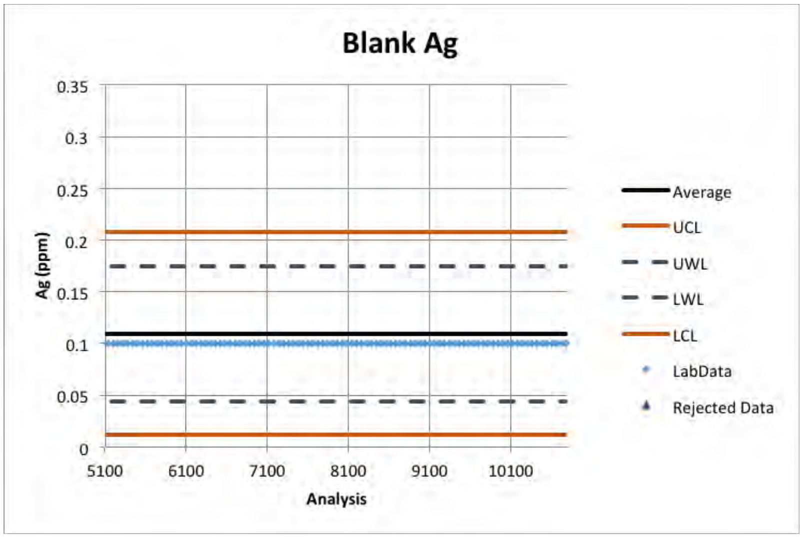
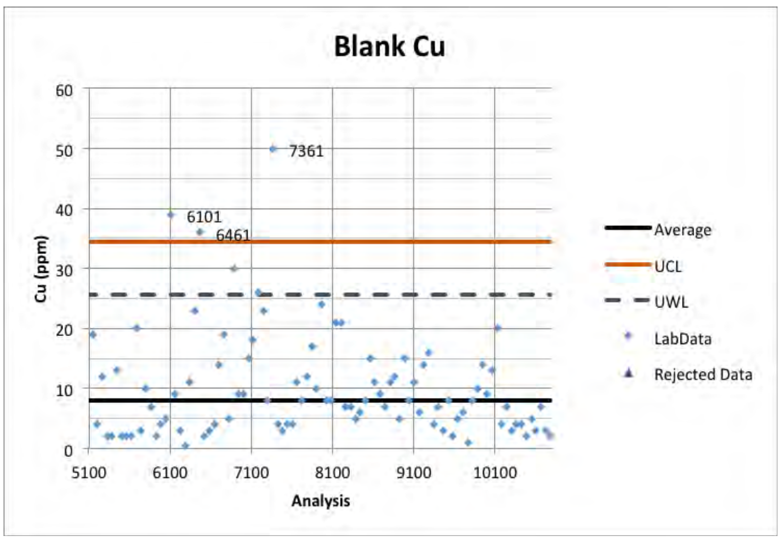


Figure 11.38: Performance of Blanks for Cu



11.4.1.3 Performance of Field Duplicates

The field duplicate data is represented in Table 11.8 and the duplicate sample results are plotted in Figure 11.39 through Figure 11.43 for each element including: Au, Pt, Pd, Ag, Cu, Ni and S. A best-fit line is calculated for each element, as well as the R^2 value. There is a strong confidence in the data, with all R^2 values greater than 89%.

Table 11.8: Field Duplicate Control Limits

	Au (ppm)	Pt (ppm)	Pd (ppm)	Ag (ppm)	Cu (ppm)	Ni (ppm)	S (%)
Average	0.0195	0.0489	0.0854	0.3417	825	124.1472	0.3048
Standard Deviation	0.0391	0.1054	0.1771	0.3329	909.051	164.5228	0.3599
R^2	0.899	0.8933	0.9508	0.911	0.9551	0.9874	0.944

Note: *Ag = silver, Au = gold, Cu = copper, Ni = nickel, Pd = palladium, Pt = platinum, S = sulphur.*

Figure 11.39: Performance of Field Duplicates for Au

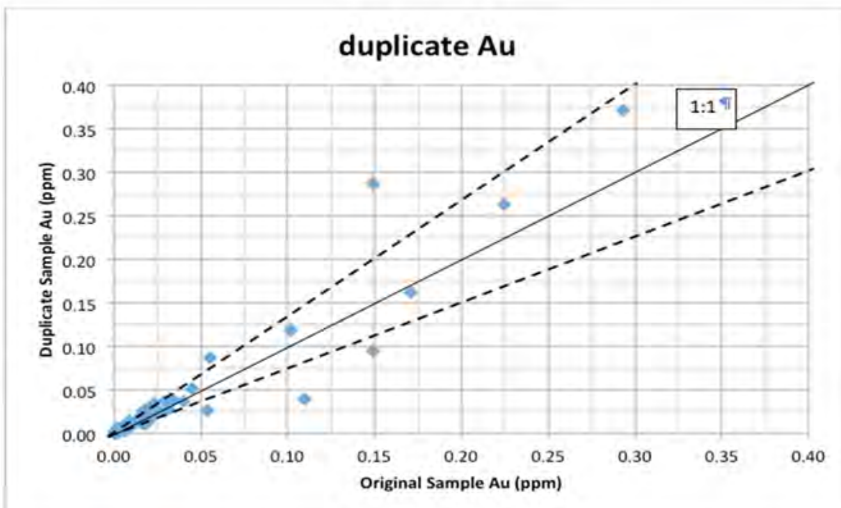


Figure 11.40: Performance of Field Duplicates for Pt

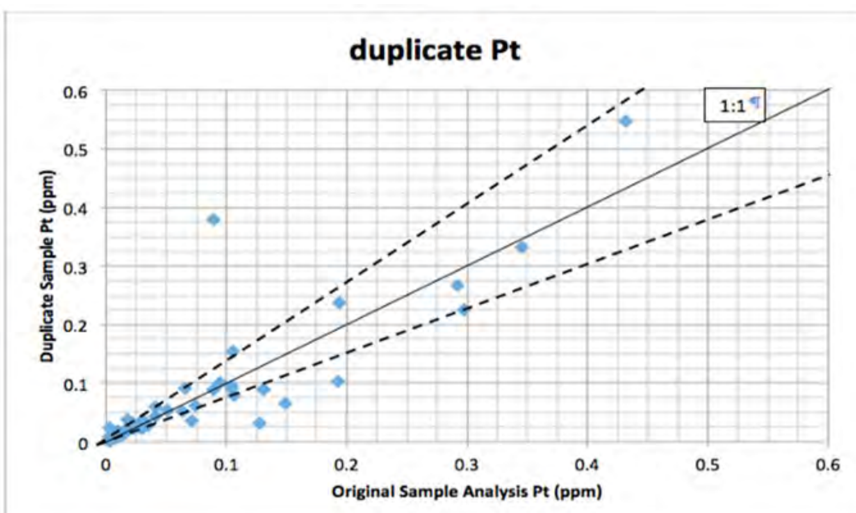


Figure 11.41: Performance of Field Duplicates for Pd

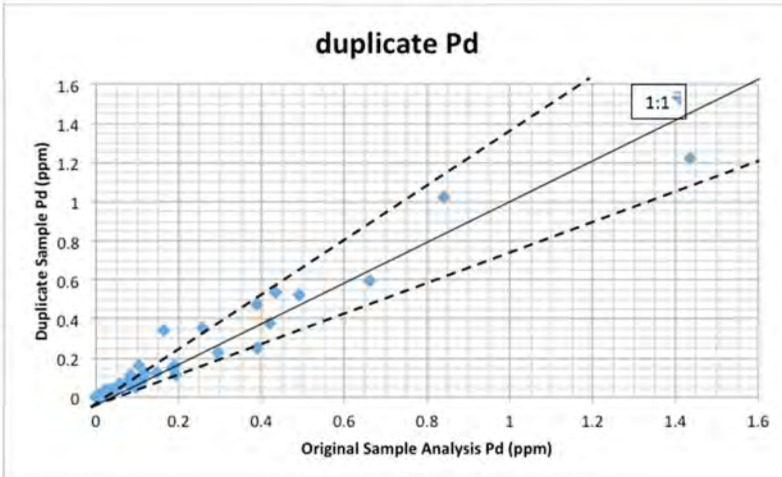


Figure 11.42: Performance of Field Duplicates for Ag

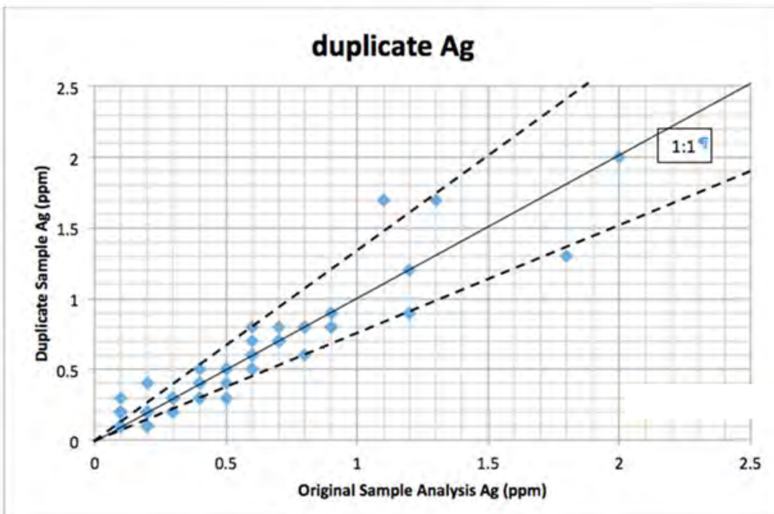
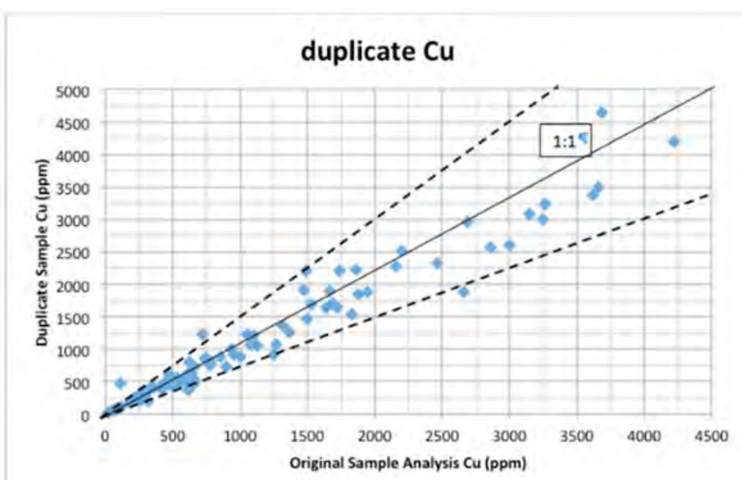


Figure 11.43: Performance of Field Duplicates for Cu



11.4.2 2017 Drilling Program

11.4.2.1 Performance of Certified Reference Materials

The analyses for elements Au, Pt, Pd, Ag and Cu for CRMs MPG1 and MPG2 are plotted in Figure 11.44 to Figure 11.53.

Some outliers beyond the set control limits can be noted; however, the overall performance of both CRMs, for all elements, was excellent and no bias or temporal variation in the 2017 data was noted.

Figure 11.44: Performance of CRM MPG1 Au

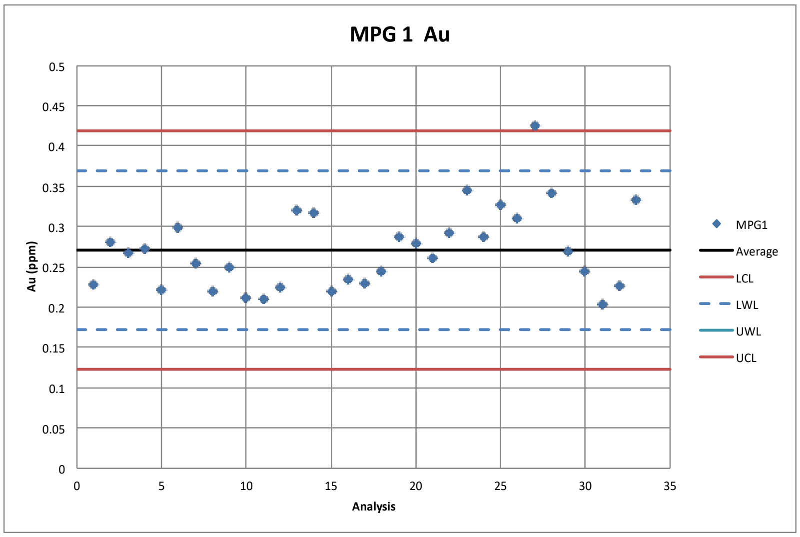


Figure 11.45: Performance of CRM MPG1 for Pt

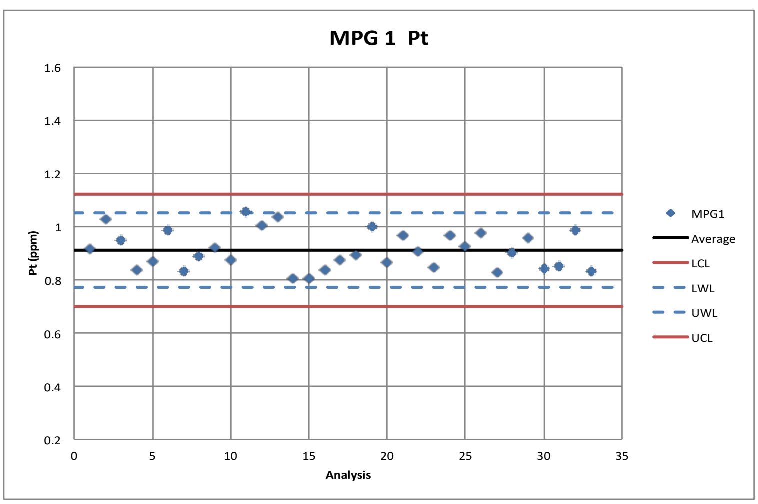


Figure 11.46: Performance of CRM MPG1 for Pd

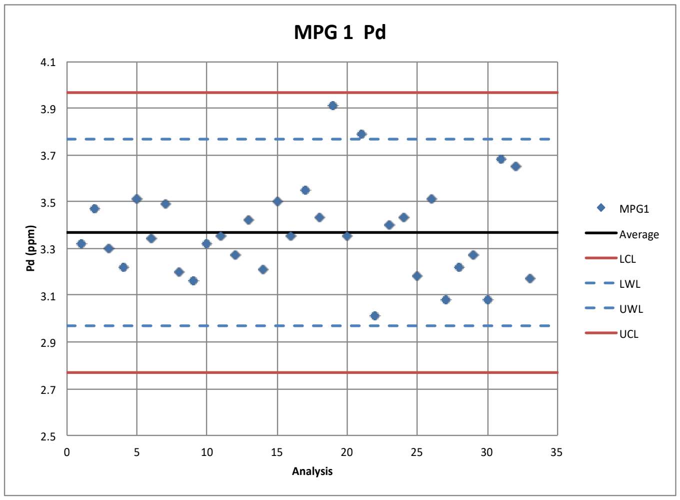


Figure 11.47: Performance of CRM MPG1 for Ag

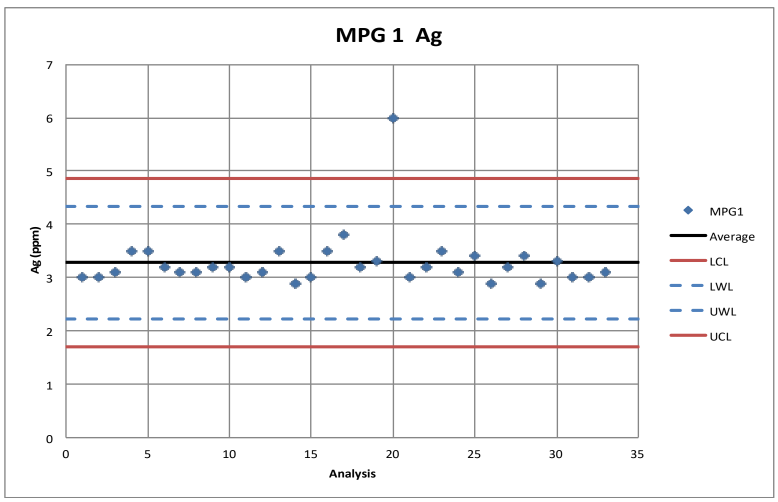


Figure 11.48: Performance of CRM MPG1 for Cu

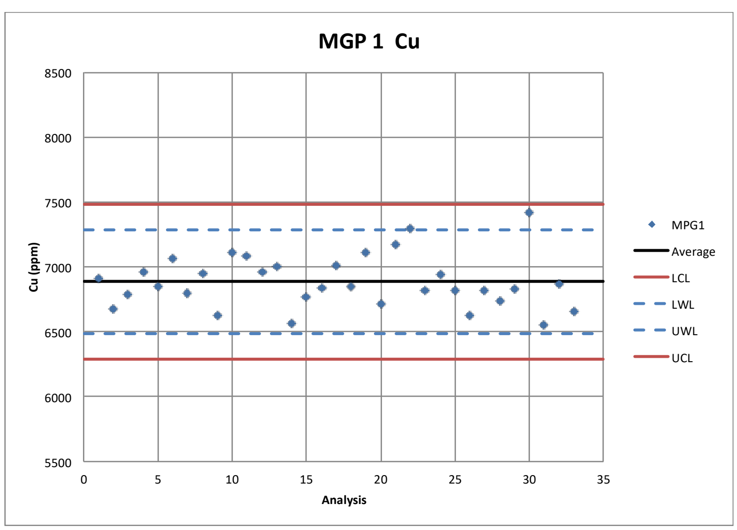


Figure 11.49: Performance of CRM MPG2 for Au

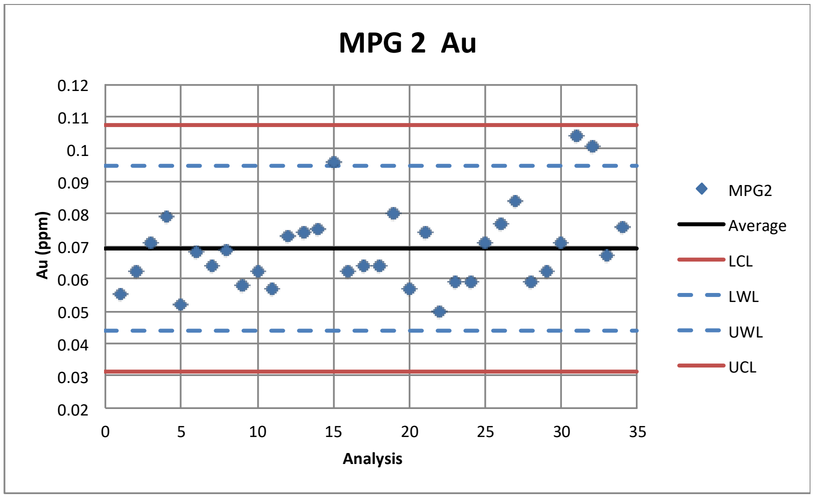


Figure 11.50: Performance of CRM MPG2 for Pt

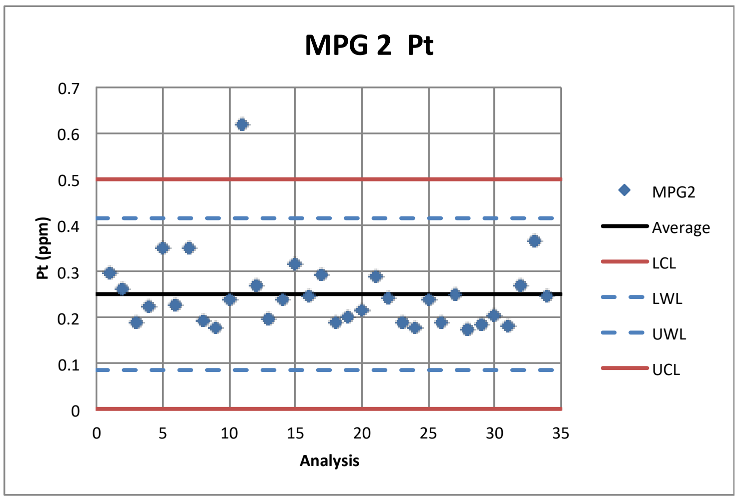


Figure 11.51: Performance of CRM MPG2 for Pd

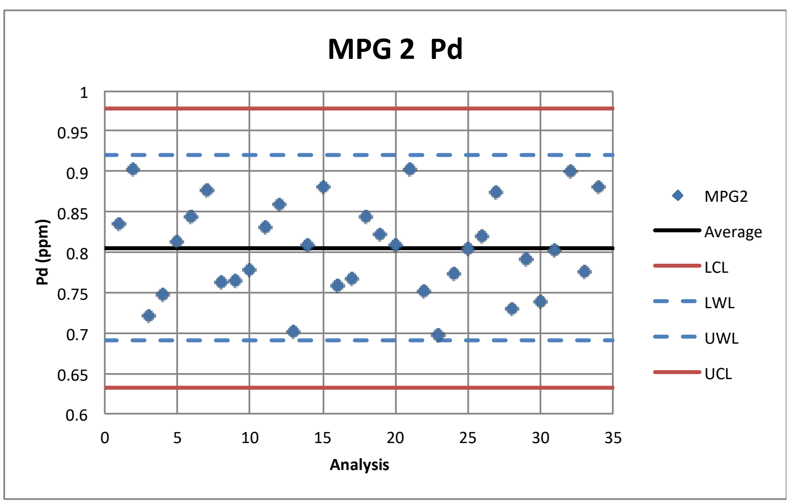


Figure 11.52: Performance of CRM MPG2 for Ag

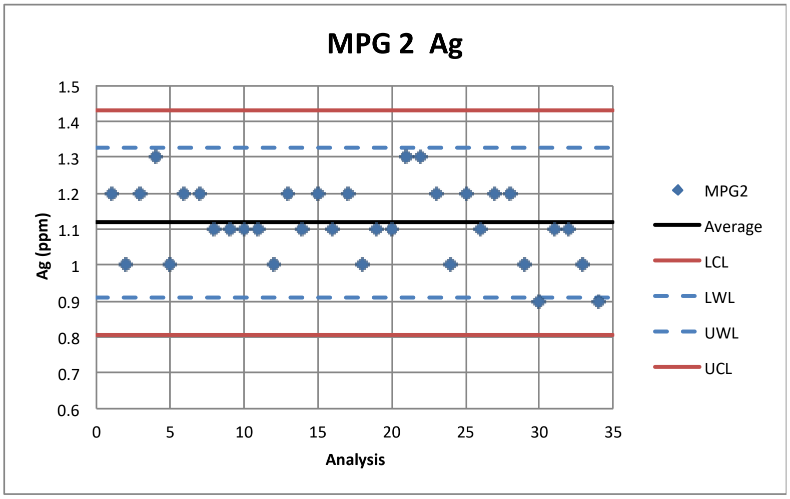
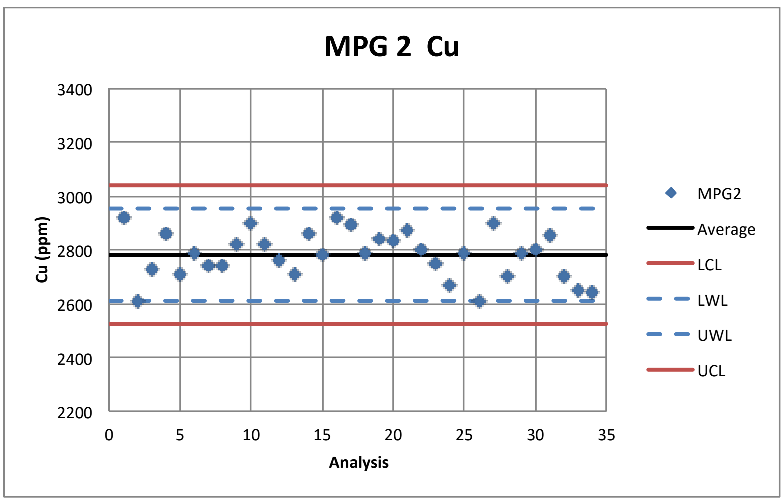


Figure 11.53: Performance of CRM MPG2 for Cu



11.4.2.2 Performance of Blank Material

The results of the blank sample analyses (Figure 11.54 to Figure 11.58) were considered excellent, with the vast majority of the Au, Pt, Pd, Ag and Cu determinations falling below the respective upper working limit of two times the standard deviation of the mean of each element. The occasional result falling above the upper working limit was not considered to be of material impact to the Mineral Resource estimate and contamination was not considered to be an issue with the 2017 data.

Figure 11.54: Performance of Blanks for Au

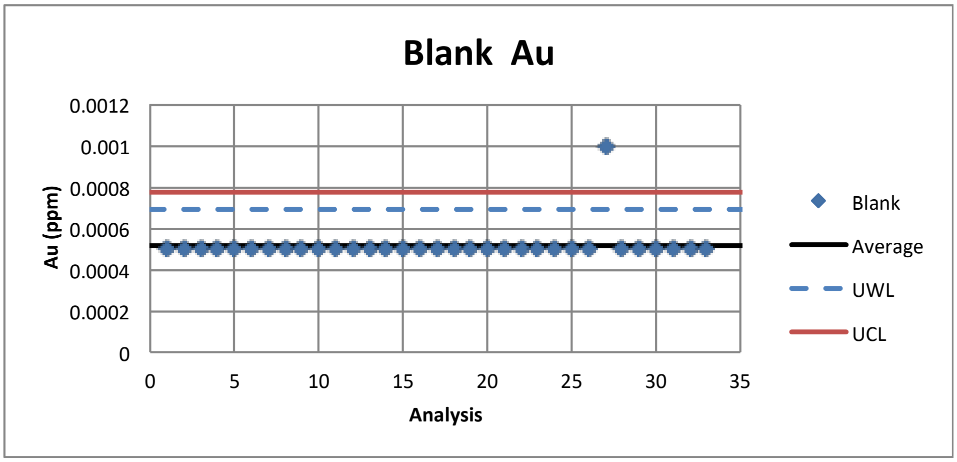


Figure 11.55: Performance of Blanks for Pt

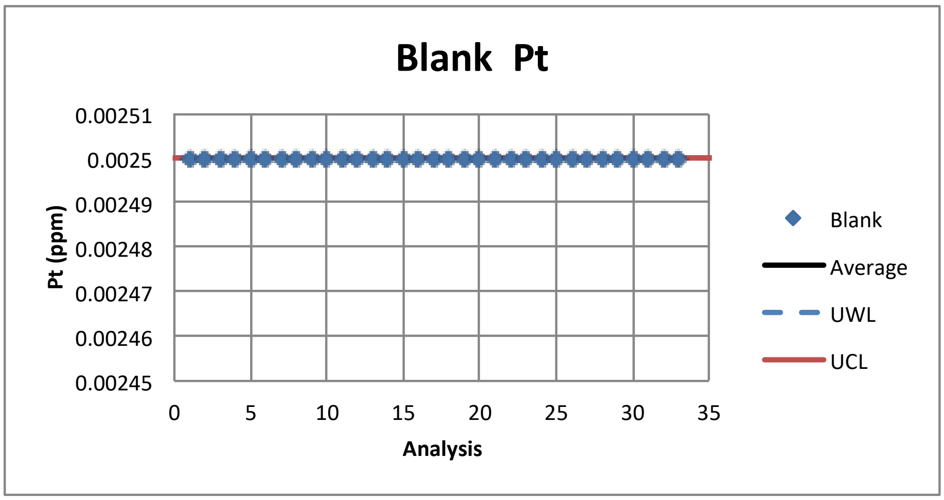


Figure 11.56: Performance of Blanks for Pd

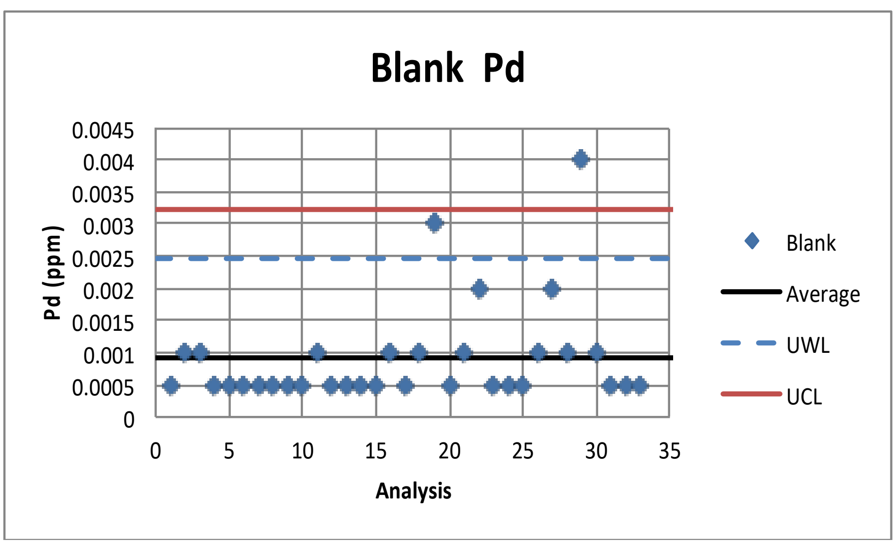


Figure 11.57: Performance of Blanks for Ag

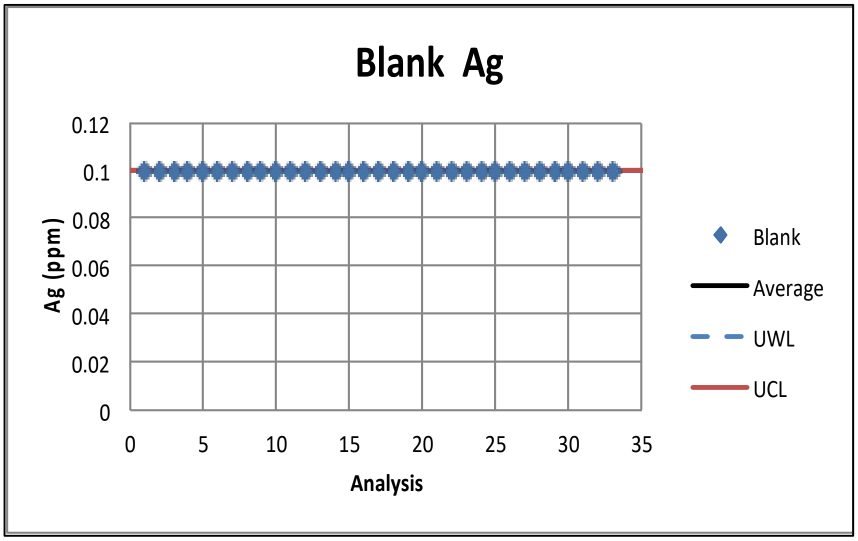
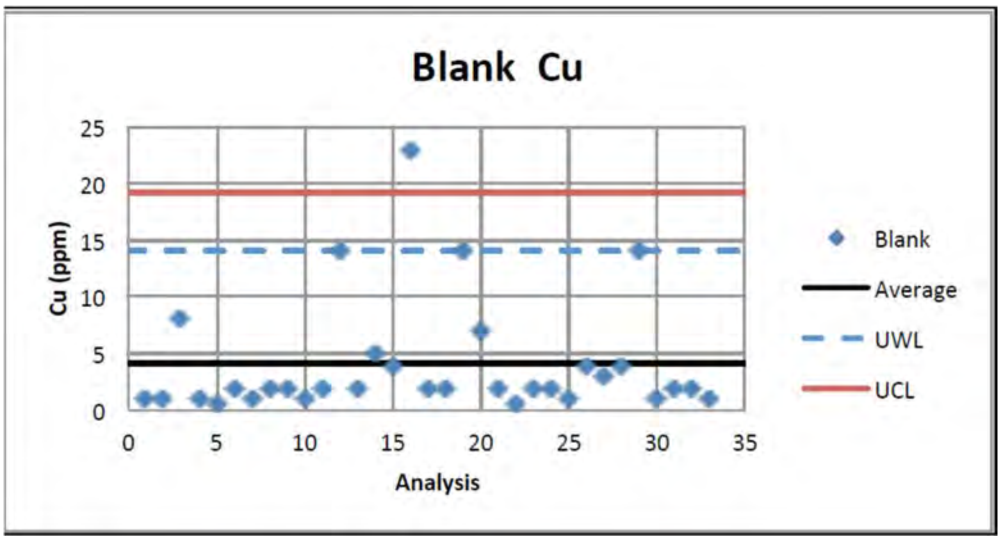


Figure 11.58: Performance of Blanks for Cu



11.4.2.3 Performance of Field Duplicates

The field duplicate data for Au, Pt, Pd, Ag and Cu were plotted on scatter plots and precision for all elements was considered acceptable by The Author of this technical report.

11.4.3 2019 Drilling Program

11.4.3.1 Performance of Certified Reference Material

The analyses for elements Au, Pt, Pd, Ag and Cu for CRMs MPG1 and MPG2 are plotted in Figure 11.59 to Figure 11.68.

Some outliers beyond the set control limits can be noted; however, the overall performance of both CRMs for all elements was excellent and no bias or temporal variation in the 2019 data was noted.

Figure 11.59: Performance of CRM MPG1 Au

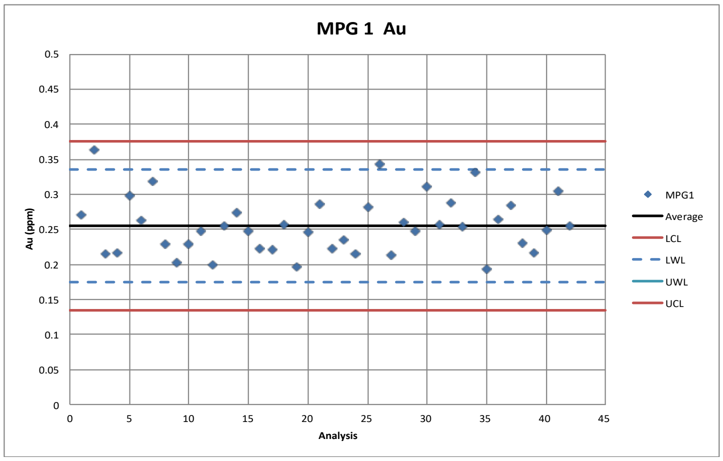


Figure 11.60: Performance of CRM MPG1 for Pt

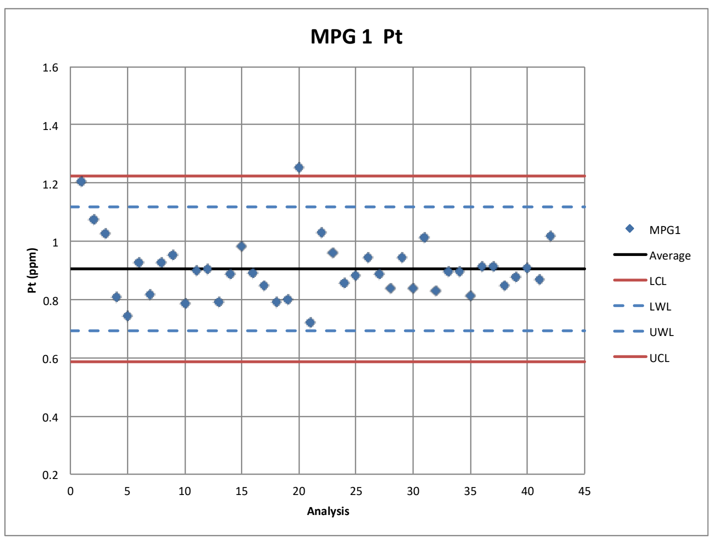


Figure 11.61: Performance of CRM MPG1 for Pd

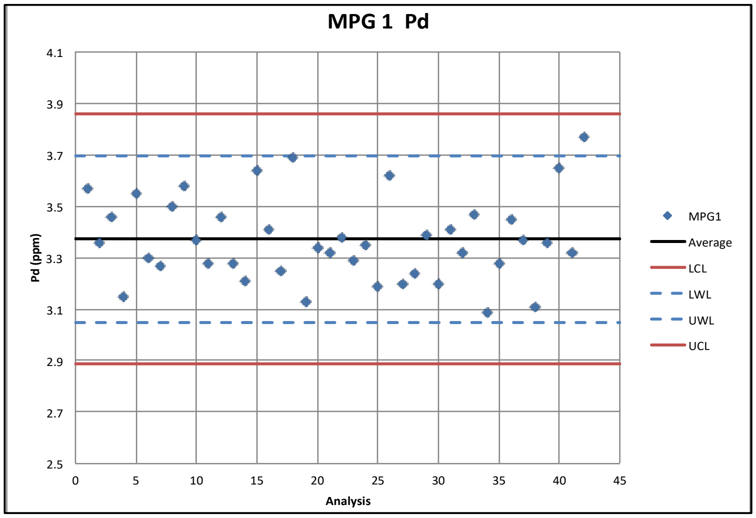


Figure 11.62: Performance of CRM MPG1 for Ag

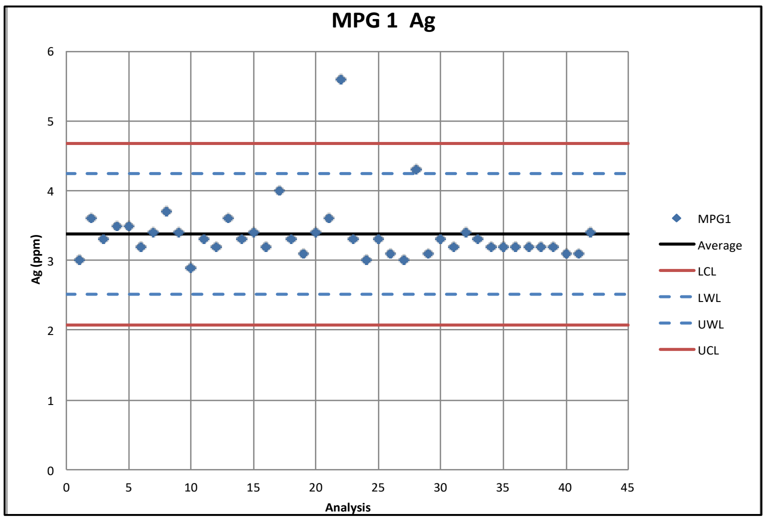


Figure 11.63: Performance of CRM MPG1 for Cu

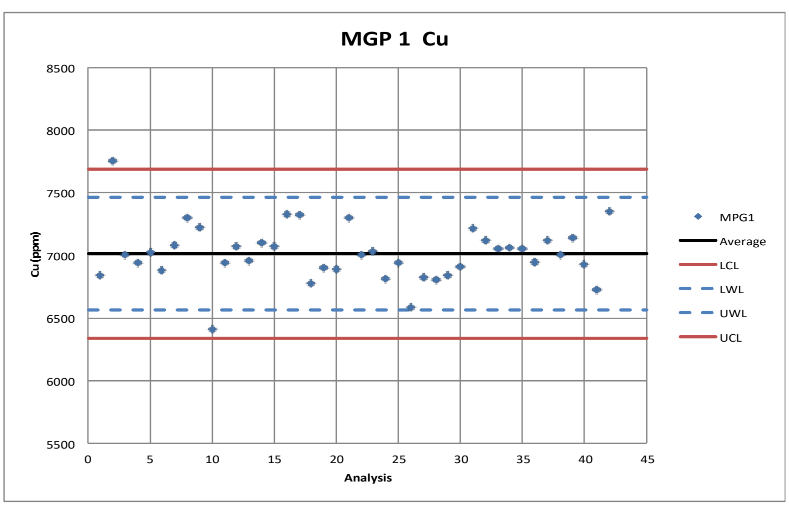


Figure 11.64: Performance of CRM MPG2 for Au

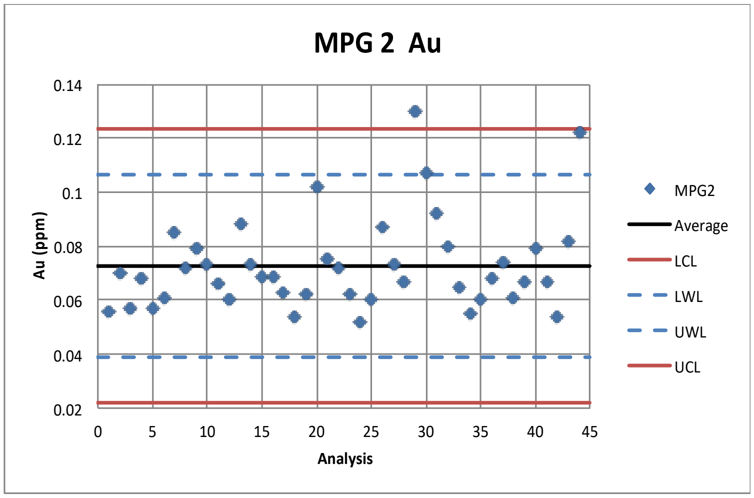


Figure 11.65: Performance of CRM MPG2 for Pt

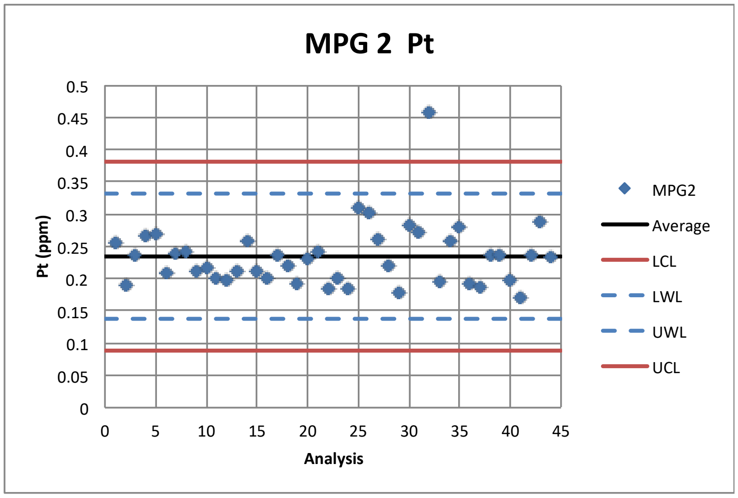


Figure 11.66: Performance of CRM MPG2 for Pd

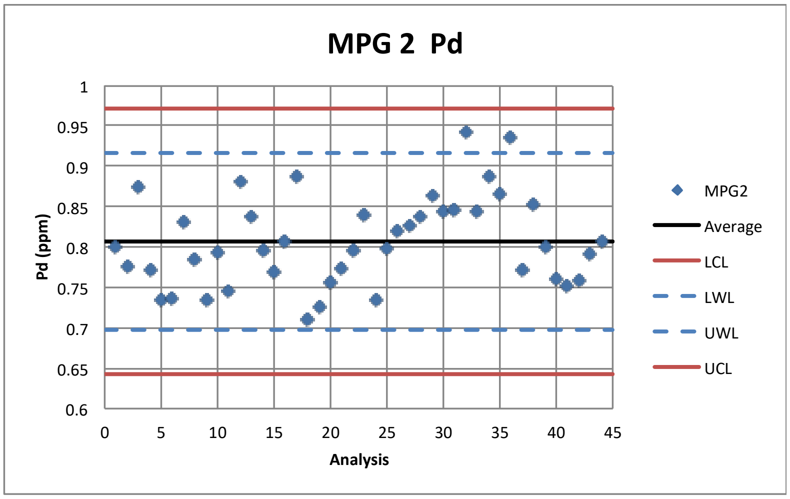


Figure 11.67: Performance of CRM MPG2 for Ag

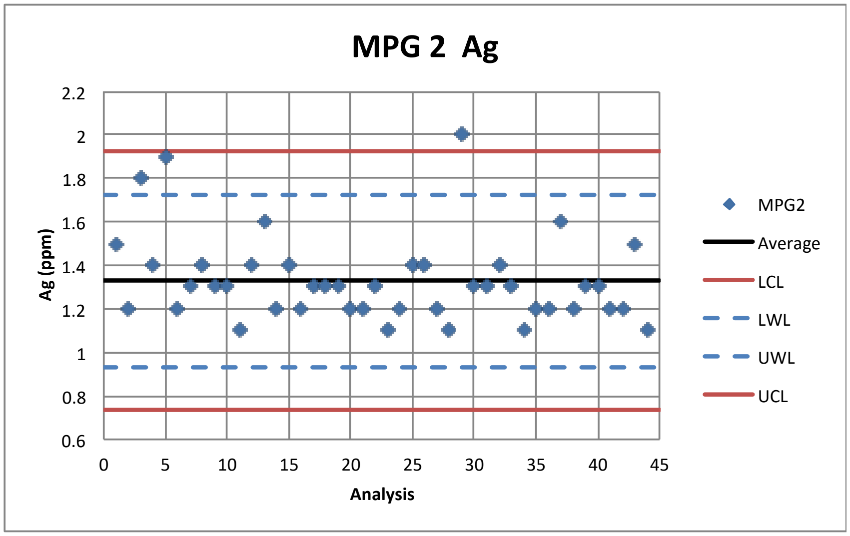
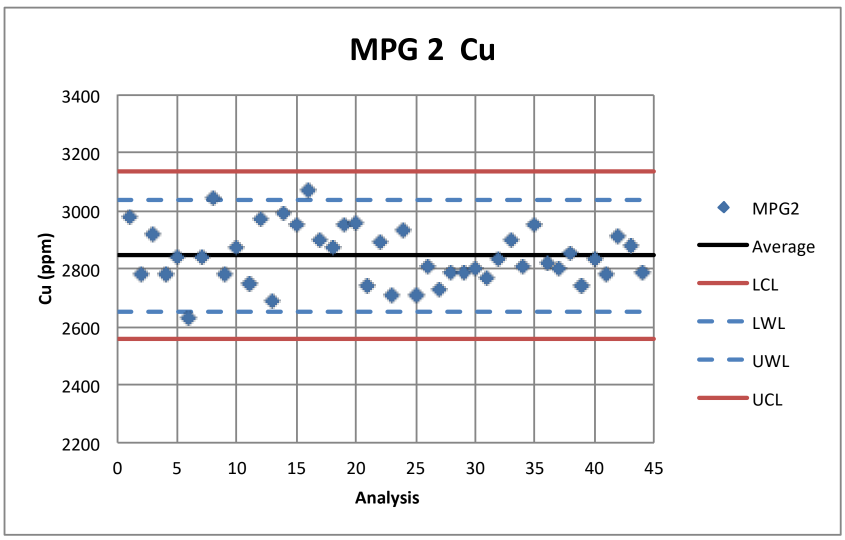


Figure 11.68: Performance of CRM MPG2 for Cu



11.4.3.2 <u>Performance of Blank Material</u>

The results of the blank sample analyses were considered excellent, with the vast majority of the Au, Pt, Pd, Ag and Cu determinations falling below the respective upper working limit of two times the standard deviation of the mean of each element (Figure 11.69 to Figure 11.73). The occasional result falling above the upper working limit was not considered to be of material impact to the Mineral Resource estimate and contamination was not considered to be an issue with the 2019 data.

Figure 11.69: Performance of Blanks for Au

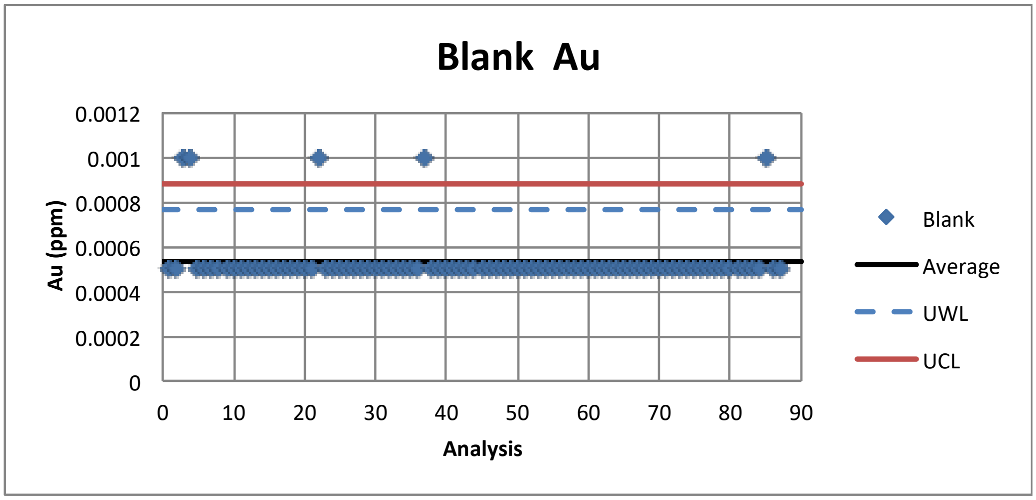


Figure 11.70: Performance of Blanks for Pt

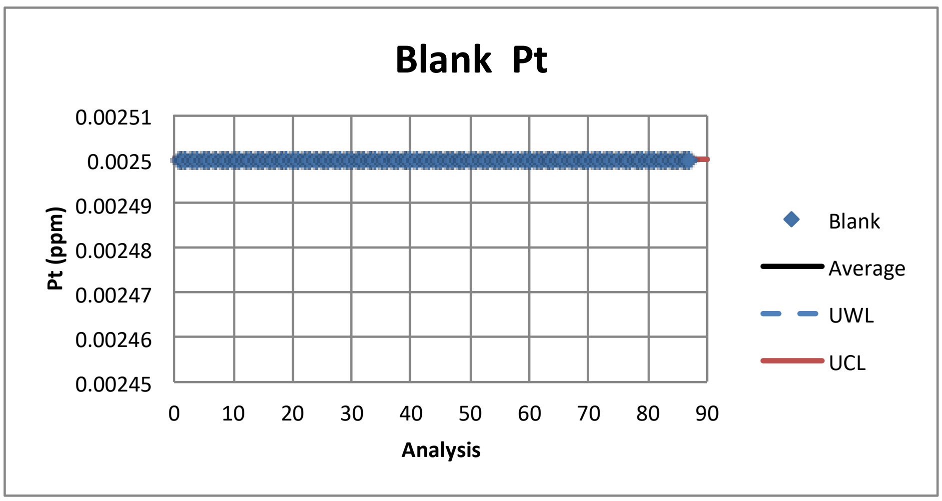


Figure 11.71: Performance of Blanks for Pd

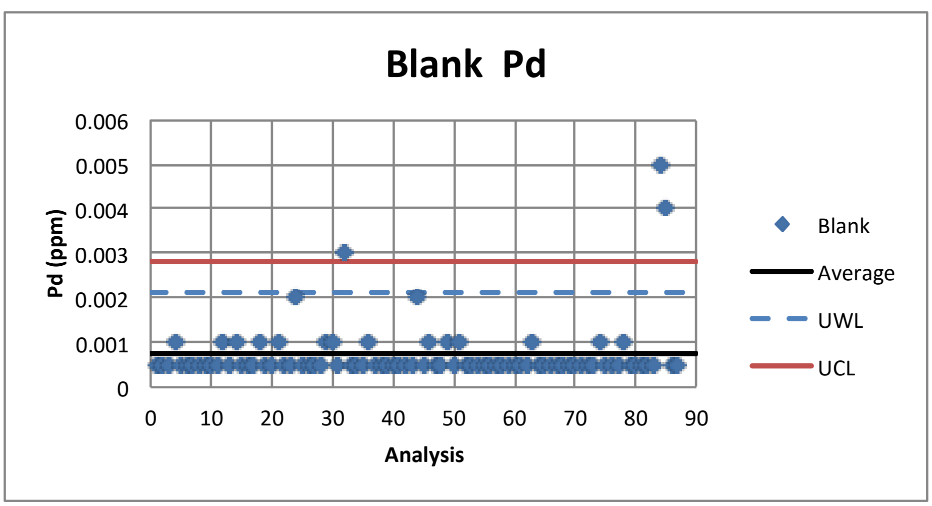


Figure 11.72: Performance of Blanks for Ag

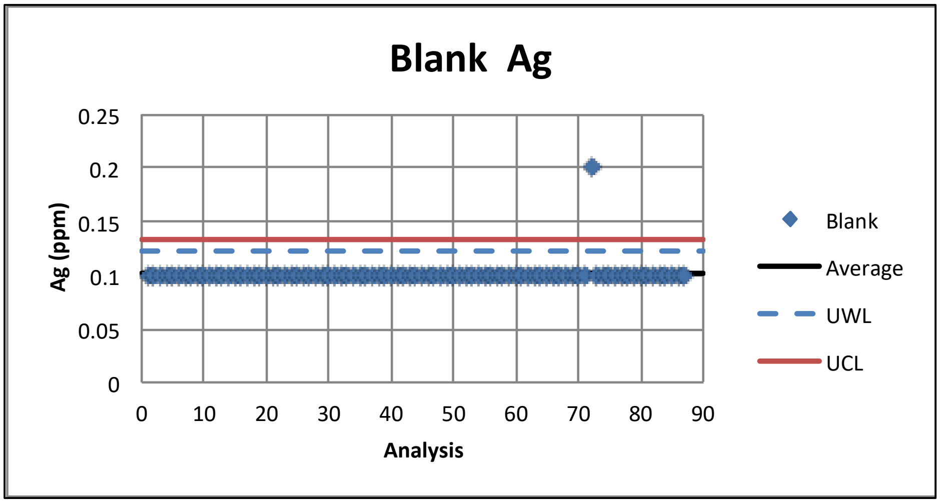
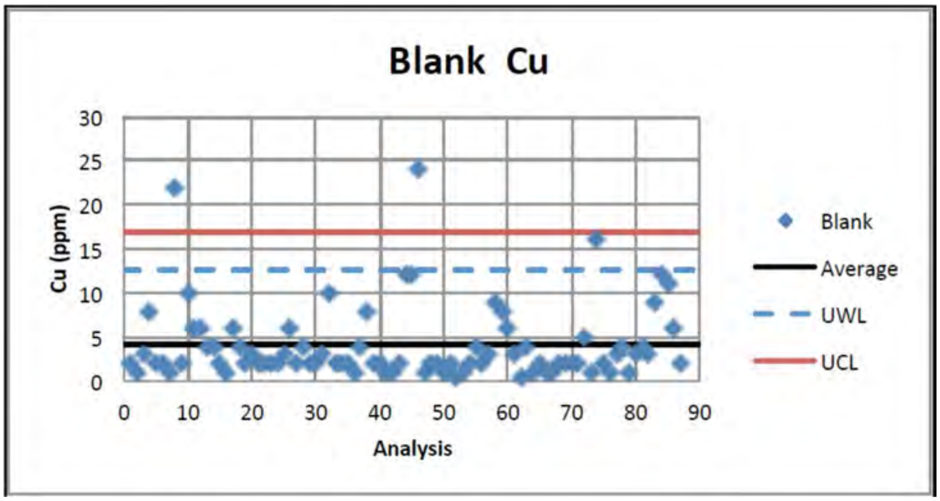


Figure 11.73: Performance of Blanks for Cu



11.4.3.3 Performance of Field Duplicates

The field duplicate data for Au, Pt, Pd, Ag and Cu were plotted on scatter plots and precision for all elements was considered acceptable by The Author of this technical report section.

11.4.4 Conclusions

The Author of this technical report section considers the Sally Deposit data to be of good quality and acceptable for use for Mineral Resource estimation.

12. DATA VERIFICATION

12.1 April 2012 Site Visit Independent Sampling

The Property was visited on April 4, 2012 by Mr. David Burga, P.Geo., of P&E, an independent QP as defined by NI 43-101. Mr. Burga collected 10 samples from nine holes. Samples were collected by sawing in half the half core that was remaining in the core box.

The samples were placed in plastic bags, given a unique sample ID and taken by Mr. Burga to AGAT in Mississauga, Ontario for analysis.

Copper, silver and nickel were analyzed using four-acid digest with AAS finish. Gold, platinum and palladium were analyzed using lead collection fire assay with ICP-OES finish.

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

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

Results of the independent site visit samples are presented in Figure 12.1 through Figure 12.4.

Figure 12.1: P&E Site Visit Results for Palladium

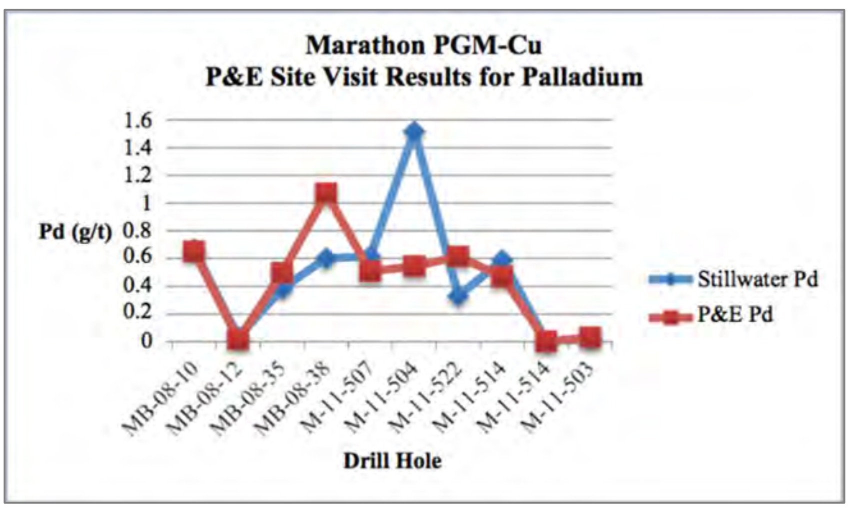


Figure 12.2: P&E Site Visit Results for Platinum

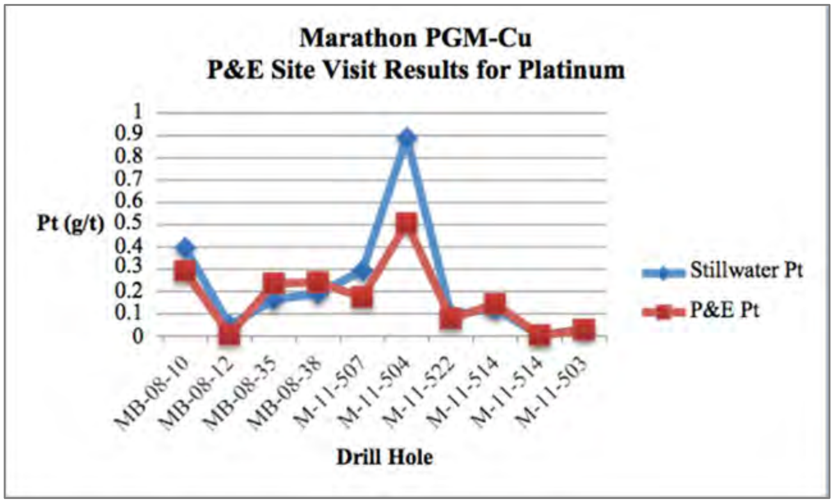


Figure 12.3: P&E Site Visit Results for Copper

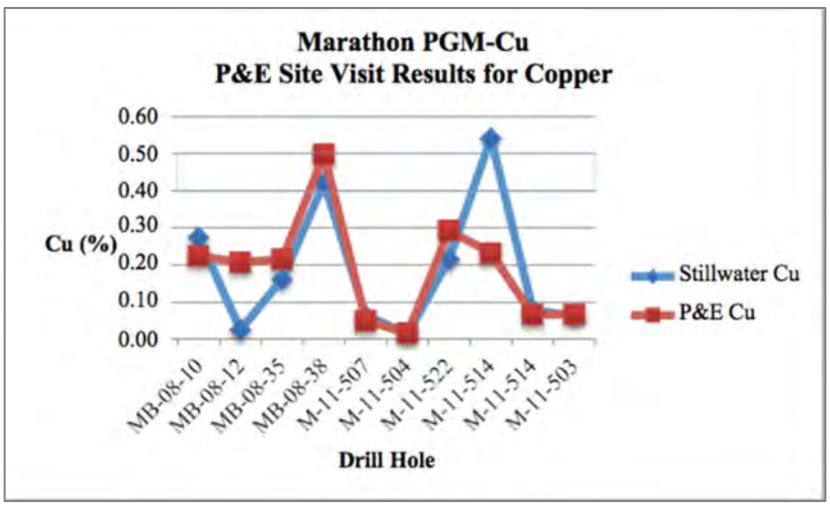
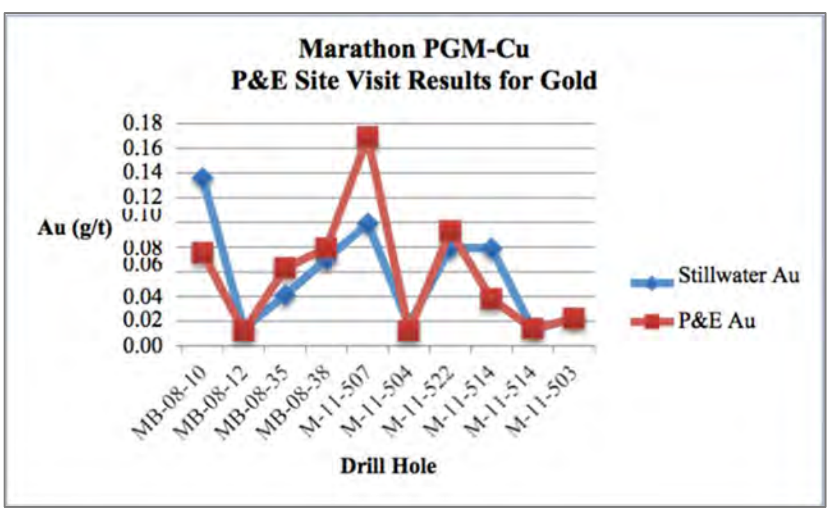


Figure 12.4: P&E Site Visit Results for Gold



12.2 May 2019 Site Visit and Independent Sampling

A site visit to the Property was undertaken by Mr. Bruce Mackie, P.Geo., of Mackie, an independent QP as defined by NI 43-101, on May 4, 2019. As part of the site visit, confirmation samples from selected drill core intervals were taken by Mr. Mackie and submitted to Activation Laboratories Ltd. in Thunder Bay. This work was aided by Mr. John McBride, P.Geo., a Senior Project Geologist employed at that time by Stillwater Canada.

12.2.1 Data Verification and Drill Core Examination

During the site visit, 12 mineralized drill hole intercepts were inspected by Mr. Mackie (listed in Table 12.1).

Prior to the inspection, the core was located and laid out at the main core storage facility in the Town of Marathon. This work was performed by Mr. John McBride of Stillwater Canada. It should be noted that while the mineralized drill hole intercepts were provided in advance to save time during the site visit, the specific intervals that were to be re-sampled by Mr. Mackie were not provided in advance.

Table 12.1: Drill Hole Intercepts Inspected

Zone	Hole No.	From (m)	To (m)	Interval (m)
Main Zone	M-05-49	20.0	34.0	14.0
Main Zone	M-05-49	80.0	90.0	10.0
Main Zone	M-11-520	176.0	189.0	13.0
Main Zone	M-11-520	211.0	227.0	16.0
BR Zone	M-06-178	3.0	17.0	14.0
Southern Resource Zone	M-17-528	43.0	55.0	12.0
Southern Resource Zone	M-17-529	70.0	80.0	10.0
Sally Zone	SL-17-71	31.0	49.0	18.0
Sally Zone	SL-17-72	264.0	284.0	20.0
Sally Zone	SL-17-72	310.0	320.0	10.0
Geordie	G-00-08	158.01	168.50	10.5
Geordie	G-10-17	216.00	234.00	18.0
Total				**165.5**

Source: *Mackie (2019).*

The 12 intercepts were selected from nine diamond drill holes based largely on the following criteria: availability of core (much of the mineralized core from historic drilling from the Core Area of the Marathon Deposit was taken for metallurgical testwork), intercepts ranging from low-grade (<0.5 g Pd/t), medium-grade (0.5 to 1.0 g Pd/t) and high-grade (>1.0 g Pd/t). In addition, drill core intercepts selected were from five different zones. The Core Area is defined as the area of the Property from which the historic Mineral Resource estimates were estimated (the Main Zone, BR Zone, and Southern Resource Zone) and the Sally and Geordie Deposits. Finally, the selection included drill core from several different drill campaigns carried out between 2005 and 2017 by both Marathon PGM and Stillwater Canada.

Mr. Mackie's inspection of the mineralized drill hole intercepts comprised the following tasks:

- Drill hole numbers were verified, and initial and final depths of the mineralized intercepts were reviewed.
- Measurement of core sample lengths and verification of sample numbers and tags.
- Validation of the descriptive geology with emphasis on the reported visual estimates of pyrite, chalcopyrite, pyrrhotite, chalcocite and magnetite content reported by Marathon PGM and Stillwater Canada.
- Validation, using original Accurassay and ALS Chemex assay certificates, of Pd, Pt, Au, and Cu assays reported for the mineralized intercepts in MS Excel™ files: Marathon Assays and Core.xlsx and Geordie Assay Range for Due Diligence.xlsx provided by Stillwater Canada.

Mr. Mackie's visual estimates of pyrite, chalcopyrite, pyrrhotite, chalcocite and magnetite content generally agree with those reported by Marathon PGM and Stillwater Canada for the 12 mineralized drill hole intercepts reviewed.

Drill logs for the sections reviewed were found to be appropriately detailed and presented a reasonable representation of geology, alteration, mineralization and structure.

No discrepancies in the sample tag numbers within the drill core trays and the intervals quoted in the above-mentioned Excel spreadsheets were noted. Nor were any discrepancies observed in the Pd, Pt, Au, and Cu values quoted from those in the original assay certificates.

Based on the results of the investigation, Mr. Mackie is of the opinion that the mineralized drill hole assay results and corresponding drill hole logs reported by Stillwater Canada and Marathon PGM (for drill holes M-05-49, M-11-520, M-06-178, M-17-528, M-17-529, SL-17-71, SL-17-72, G-00-08, and G-10-17 that were the subject of the investigation) are verifiable and accurate and portray a reasonable representation of the types of mineralization encountered on the Marathon and Geordie Deposits.

12.2.1.1 Confirmation of Sampling

Twelve samples were taken for due diligence to verify the presence of palladium, platinum, gold, and copper in the drill core. In addition, a sample of both the high- and low-grade standards used by Stillwater Canada in its 2017 drill program were also taken for analyses. The sample intervals were selected by Mr. Mackie without prior knowledge given to Gen Mining. The samples collected consisted of sawn quarter core. All verification samples duplicated the original sample intervals. In all instances the original sample interval was visible in the core box. Each verification sample was indicated with a Bruce Mackie sample identification tag that was placed in the core box. Mr. Mackie collected each sample and placed them in clearly identified plastic bags with a unique sample number tag.

The verification samples remained in the custody of Mr. Mackie until he delivered them in person in a sealed container to Activation Laboratories Ltd. (Actlabs), an accredited assay laboratory, in Thunder Bay.

The samples were prepared and analyzed using similar methodologies employed by Stillwater Canada during its 2017 diamond drilling campaign: sample preparation Code RX1, gold, platinum and palladium analyses by fire assay followed by ICP-MS (Code 1C-EXP2) and trace element analyses by partial "aqua regia" digestion with an ICP-MS finish (Code UT-1M). In addition, the Specific Gravity of each of the core samples was determined by Pycnometer (Nitrogen).

Table 12.2 gives the intervals sampled and Table 12.3 summarizes the results of the confirmation sampling.

Table 12.2: Confirmation of Sample Intervals

Zone	Hole Number	From (m)	To (m)	Interval (m)	Lab / Year	Lab Certificate Number
Sally	SL-17-71	41.0	43.0	2.0	ALS/2017	TB17177687
Sally	SL-17-72	276.0	278.0	2.0	ALS/2017	TB17210631
Sally	SL-17-72	314.0	316.0	2.0	ALS/2017	TB17210631
Southern Resource	M-17-529	72.0	74.0	2.0	ALS/2017	TB17233256
Southern Resource	M-17-528	45.0	47.0	2.0	ALS/2017	TB17220588
BZ Zone	M-06-178	7.0	9.0	2.0	Accurassay/2006	200641225
Main Zone	M-11-520	183.0	185.0	2.0	ALS/2011	TB11168362
Main Zone	M-11-520	217.0	219.0	2.0	ALS/2011	TB11168362
Main Zone	M-05-49	22.0	24.0	2.0	Accurassay/2005	200541214
Main Zone	M-05-49	84.0	86.0	2.0	Accurassay/2005	200541214
Geordie	G-00-08	160.1	161.1	1.0	Accurassay/2000	200041175
Geordie	G-10-17	222.00	224.00	2.0	Accurassay/2010	201040690

Table 12.3: Confirmation of Assay Results

Survey By	From (m)	To (m)	Length (m)	Au (g/t)	Pd (g/t)	Pt (g/t)	Cu (ppm)
DDH SL-17-71 Mineralized Intercept Sally Zone							
Stillwater	41.0	43.0	2.0	0.200	0.633	0.245	3,330
Mackie	41.0	43.0	2.0	0.195	0.591	0.246	3,510
DDH SL-17-72 Mineralized Intercept Sally Zone							
Stillwater	276.0	278.0	2.0	0.124	1.310	0.850	529
Mackie	276.0	278.0	2.0	0.065	1.190	0.587	225
Stillwater	314.0	316.0	2.0	0.252	1.085	0.658	1,920
Mackie	314.0	316.0	2.0	0.263	1.790	0.924	2,840
DDH M-17-529 Mineralized Intercept Southern Resource							
Stillwater	72.0	74.0	2.0	0.136	0.815	0.239	3,510
Mackie	72.0	74.0	2.0	0.101	0.750	0.235	3,530
DDH M-17-528 Mineralized Intercept Southern Resource							
Stillwater	45.0	47.0	2.0	0.190	0.274	0.129	2,770
Mackie	45.0	47.0	2.0	0.103	0.113	0.101	2,530
DDH M-06-178 Mineralized Intercept BZ Zone							
Marathon	7.0	9.0	2.0	0.963	2.230	0.727	2,352
Mackie	7.0	9.0	2.0	0.152	1.750	0.583	852
DDH M-11-520 Mineralized Intercept Main Zone Resource							
Stillwater	183.0	185.0	2.0	0.055	0.616	0.139	3,480
Mackie	183.0	185.0	2.0	0.053	0.599	0.120	2,940
DDH M-11-520 Mineralized Intercept Main Zone Resource							
Stillwater	217.0	219.0	2.0	0.160	1.160	0.244	4,680
Mackie	217.0	219.0	2.0	0.092	0.935	0.275	3,860
DDH M-05-49 Mineralized Intercept Main Zone Resource							
Marathon	22.0	24.0	2.0	0.005	0.755	0.530	190
Mackie	22.0	24.0	2.0	0.013	0.461	0.430	190
DDH M-05-049 Mineralized Intercept Main Zone Resource							
Marathon	84.0	86.0	2.0	0.039	0.321	0.106	1,410
Mackie	84.0	86.0	2.0	0.043	0.327	0.071	2,340
DDH G-00-08 Mineralized Intercept Geordie							
Marathon	160.1	161.1	1.0	0.141	2.125	0.107	9,980
Mackie	160.1	161.1	1.0	0.092	1.700	0.092	8,670
DDH G-10-17 Mineralized Intercept Geordie							
Marathon	222.0	224.0	2.0	0.065	0.981	0.065	5,163
Mackie	222.0	224.0	2.0	0.052	0.824	0.051	5,860
MPG-1 High Grade Standard 2017 Drill Program							
Stillwater				0.275	3.538	1.109	6,715
Mackie				0.240	3.550	0.868	7,070
MPG-2 Low Grade Standard 2017 Drill Program							
Stillwater				0.073	0.805	0.223	2,853
Mackie				0.119	1.110	0.245	2,800

Note: *DDH = diamond drill hole.*
Source: *Mackie (2019).*

The Authors of this technical report section considers that there is good correlation between the independent verification samples and the original analyses in the Company database.

12.3 Marathon Deposit Assay Database Verification

In September 2019, verification of assay data entry was performed on 7,022 assay intervals for Cu, Au, Ag, Pt and Pd. Only a few data entry errors were observed and corrected. The 7,022 verified intervals were checked against assay laboratory certificates from Accurassay Laboratories of Thunder Bay, Ontario, ALS Chemex of Vancouver, B.C., ACME Analytical Laboratories Ltd. of Vancouver, B.C., Bell White Analytical Laboratories of Haileybury, Ontario, and XRAL Laboratories of Don Mills, Ontario. The checked assays represented 51% of the data to be used for the Mineral Resource estimate and approximately 13% of the entire database.

12.4 Geordie Deposit Database Verification

In January 2020, the Authors of this technical report section conducted verification of the Geordie Project drill hole assay database for gold, platinum, palladium, silver and copper, by comparison of the database entries with assay certificates, supplied to P&E by Gen Mining, in pdf files.

Assay data ranging from 1987 through 2010 were verified for the Geordie Project. Sixty-nine percent (69%), representing 3,163 out of 4,558 samples, of the database was checked for gold, platinum, palladium, silver and copper, which included 82% (1,047 out of 1,277 samples) of the constrained drilling assay data.

Only two minor errors for gold and one minor error for palladium were encountered during verification of the Geordie Deposit database, which are of no material impact to the Mineral Resource estimate.

12.5 Geordie Deposit Site Visit and Due Diligence Sampling

Due diligence sampling was not considered necessary on the Geordie Deposit for verification purposes, due to the extensive verification sampling already undertaken over a number of drilling programs.

12.6 Sally Deposit Database Verification

In January 2020, the Authors of this technical report section conducted verification of the Sally Deposit drill hole assay database for gold, platinum, palladium, silver and copper, by comparison of the database entries with assay certificates, supplied to P&E by Gen Mining, in Portable Document Format. Assay data ranging from 2007 through 2017 were verified for the Sally Deposit.

Fifty-seven percent (57%), representing 5,182 out of 9,119 samples, of the database was checked for gold, platinum and palladium, which included 50% (1,275 out of 2,529 samples) of the constrained drilling assay data.

Fifty-three percent (53%), representing 4,874 out of 9,119 samples, of the database was checked for copper, which included 50% (1,275 out of 2,529 samples) of the constrained drilling assay data.

Thirty-seven percent (37%), representing 3,325 out of 9,119 samples, of the database was checked for silver, which included 41% (1,029 out of 2,529 samples) of the constrained drilling assay data.

No errors were encountered during verification of the Sally Deposit database.

12.7 Sally Deposit Site Visit and Due Diligence Sampling

Due diligence sampling was not considered necessary for the Sally Deposit verification purposes, due to the extensive verification sampling already undertaken on the Property over a number of drilling programs.

12.8 Conclusion

Based upon the evaluation of the QA/QC program undertaken by Generation Mining, as well as the data verification work carried out by the Authors, it is the opinion of the Authors of this technical report section that the data is robust and suitable for use in the current Mineral Resource Estimate.

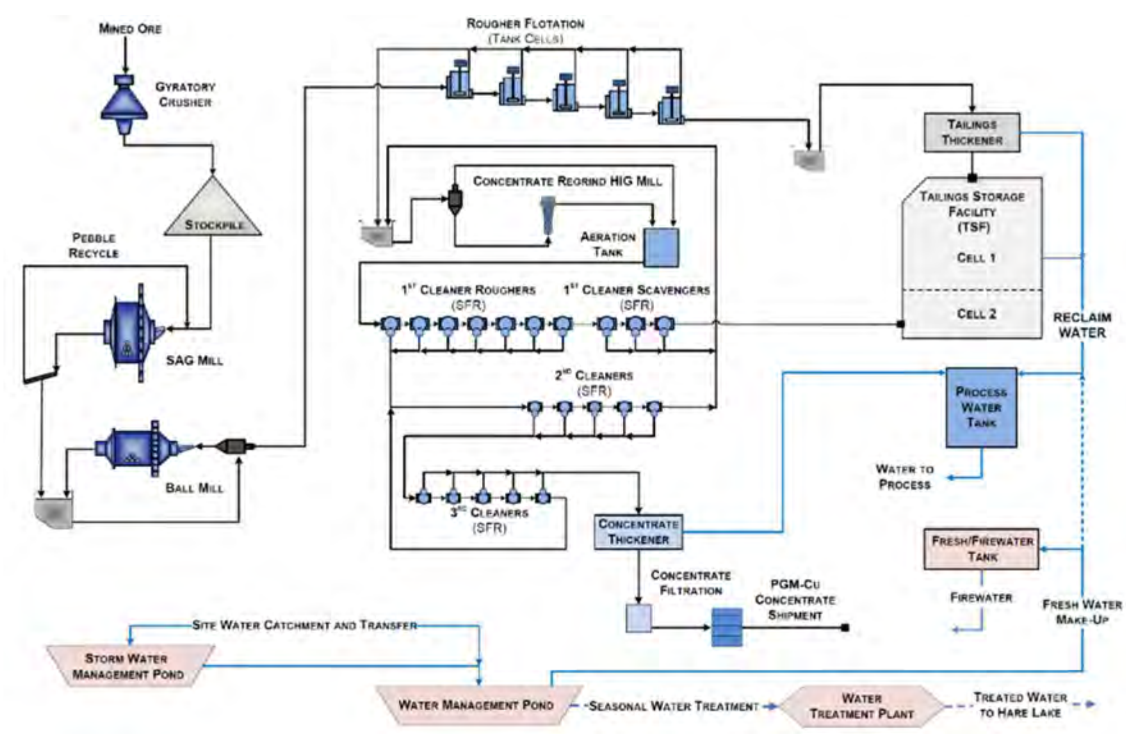
13. MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testing associated with the Project was first initiated in the 1960's and has been the subject of testing and study over the past sixty years.

Fluctuations in the value of PGM over time have influenced flowsheet development with a previous focus on copper recently shifting to PGM recovery and production as a priority. The evaluation of an optimized processing strategy and process flowsheet (Figure 13.1) by Gen Mining since 2020 has improved the ability to manage the influence of pyrrhotite in the cleaner circuit, simplified the process water balance, and yields maximum Pd, Pt, Au and Cu recovery to a single saleable concentrate.

The optimized process flowsheet involves a primary grind of 80% passing (P_{80}) of 106 µm grind to rougher flotation at natural pH, followed by rougher concentrate regrinding to a P_{80} of 18 µm at pH 11. The first, second and third cleaner flotation stages would be operated in closed circuit with rejection of iron sulphides and gangue to first cleaner scavenger tailings.

Figure 13.1: Optimized Process Flowsheet[1]



13.1 Recent Metallurgical Testwork and Studies

The focus of the metallurgical testwork completed by Gen Mining since 2020 has been to optimize the process flowsheet and associated criteria for PGM and copper recovery. Specifics and concepts determined as an outcome include:

- Mineralization: The metals of interest include Cu, Pd, Pt, Au, Ag. Less than 40% of the PGMs present are associated with sulphides with the remaining PGM mineralization as either free or silicate interlocked particles. Details with respect to mineralogy are outlined in Subsection 13.2.
- Material Characterization: Mineralized material is medium to hard with a ball mill bond work index (75^{th} percentile) of 17.45 KWh/t and a moderate abrasion index of 0.35. Additional ball mill work index testing completed in 2022 is summarized in Subsection 13.4. An SMC Axb value (75^{th} percentile) of 38 supports conventional SAG milling. Material characterization details are outlined in Subsection 13.3.
- Flotation Feed Grind Size: An initial grind size of P_{80} of 106 µm was defined in the 2020 Phase 1 test program based on optimization of Pd and Cu recovery to rougher concentrate over a range in Cu, Pd head grades. Additional Phase 3 testwork completed during 2022 studied the effects of grind size on Cu, Pd, Pt, Au, Ag recovery from a flotation feed size of P_{80} 75 µm to P_{80} 200 µm. A flotation feed size of P_{80} 106 µm remains the basis for GeoMet modeling to estimate metal recovery estimates, as well as the target for grinding circuit process design criteria. Details are outlined in Subsection 13.5.
- Flotation Reagents: An outcome of reagent optimization, PAX, Aero 3501, MIBC and lime were selected as a constant for baseline testing and yielded very acceptable results. Selectivity within the circuit was good. Pyrite and talc rejection was effective with silicate levels typically less than 4.0% Mg without mineral specific depressants. Details are outlined in Subsection 13.6.
- Rougher Flotation Rate Kinetics: A bench-scale rougher flotation retention time of 24 minutes and a target mass pull of 12-15% to rougher concentrate was defined as maximizing Pd and Cu recovery to rougher concentrate. Details are outlined in Subsection 13.7.
- Rougher Concentrate Regrind Size and Specific Energy: A rougher concentrate regrind size P_{80} of 18 µm was selected as the target for subsequent baseline conditions based on optimal concentration ratio and liberation from gangue materials. The specific energy required for the concentrate regrind duty is 11.9 kWh/t based on regrind circuit feed, as tested and evaluated by Metso Outotec. Details are outlined in Subsection 13.8.
- PGM Scavenger Circuit: The potential addition of a PGM scavenger circuit to reprocess the rougher tailings coarse fraction is excluded from initial project construction and considered as a potential future sustaining capital project subject to metal price. The incremental benefit of PGM scavenger

flotation considers PGM recovery loss to the rougher tailings coarse fraction. Subsequent regrinding of the coarse fraction followed by PGM scavenger flotation would yield an incremental gain in PGM recovery after upgrading through cleaner flotation. Following the 2022 metallurgical testwork program, the incremental gain associated with a PGM scavenger circuit has decreased and requires additional study for consideration as a sustaining capital project.

- Locked Cycle Testing: Phase 2 locked cycle test samples included: (i) 2012 Bulk Composite 3, (ii) 2020 Main Zone[2] Bulk Composite, (iii) 2020 W Horizon[3] Bulk Composite. Follow up testwork during 2022 considered the same bulk composites evaluated in 2020 in addition to five new composites from 2022 infill drilling. Locked cycle flotation test stability was achieved in all cases within the first few iterations. Platinum mineralization exhibits the slowest flotation rate kinetics requiring a particular focus on inter-stage mass pull, and the staged addition of reagents to maximize platinum recovery. Details are summarized in Subsection 13.9.
- PGM-Cu Concentrate Grade: PGM concentrate grades and recoveries achieved in 2020-2023 locked cycle testing varied from 6-22% Cu, 12-148 g/t Pd and 3-39 g/t Pt with metal recoveries of 87-96% Cu, 80-91% Pd and 63-84% Pt from starting head grades of 0.10-0.37% Cu, 0.48-1.25 g/t Pd and 0.13-0.42 g/y Pt. Aside from magnesium, which is expected to vary from an acceptable range of 2.0-6.0% Mg, there are no other deleterious elements of concern in the PGM-Cu concentrate. Details are outlined in Subsection 13.10.
- Geometallurgy: Metallurgical testwork completed by Gen Mining during 2020-2021 included bench-scale and locked cycle flotation testwork to develop an initial predictive geological-metallurgical model for the optimized process flowsheet. The GeoMet model has been expanded to include 2020-2023 testwork with separate equations for Cu, Pd, Pt, Au, Ag as a function of respective head grades. Details are outlined in Subsection 13.11.
- Direct Flotation Reactor ("DFR") Cells and Flotation Circuit Selectivity: In conjunction with 2020-2021 Phase 1 and Phase 2 bench-scale testing Gen Mining conducted a mini-pilot plant to evaluate the applicability of Woodgrove DFRs relative to conventional flotation cells. During detailed engineering, additional feedback, and references from other users of the relatively new DFR cell technology was obtained to validate equipment selection. As part of risk mitigation, Gen Mining opted to pursue conventional tank cells for rougher flotation and Woodgrove Staged Flotation Reactors ("SFR") cells for the cleaner circuit. Details are outlined in Subsection 13.12.
- Thickening and Filtration: In conjunction with 2020 Phase 2 bench-scale testing, Gen Mining involved SNF and Outotec Canada to complete flocculation, dewatering and pressure filtration

[2] The Main Zone refers to the general mineralogy associated with the North pit of the deposit.

[3] The W Horizon refers to the specific mineralogy associated with the higher-grade Pd domains in the South pit.

testing on samples generated from testwork at SGS Lakefield. Details are outlined in Subsection 13.13.

- Technical Risk and Future Testing: The Project's technical risk with respect to mineral processing is seen as minimal. Testwork completed by Gen Mining in 2020 provided an acceptable level of confidence for processing parameters and design criteria. Processing strategies and equipment considered for the Project are industrially proven with well recognized and capable equipment suppliers available. The requirement for additional testing is limited and would be related to any specific requirement for detailed engineering, or focused optimization efforts. Details are outlined in Subsection 13.14.
- Composite Samples: Specifics relating to composite samples associated with Phase 1, Phase 2 (2020) and Phase 3 (2022-2023) metallurgical test programs as well as previous composites (2010-2013) are summarized in Subsection 13.4

13.2 Mineralization Mineralogy (as applicable to Process Metallurgy)

Deposit mineralization is characterized by less than 40% of PGMs in association with sulphides. The majority of PGMs are present on grain boundaries of silicates, as opposed to finely disseminated or solid solution style deportment. Magnesium is associated with pyroxenes as well as a basic magnesium silicate within the host rock. Approximately 70% of PGM mineralization present was noted in mineralogical studies as being coarser than 20 µm in size (Cabri, 2014).

Palladium mineralization includes arsenides (arsenopalladinite $Pd_{11}Sb_2As_2$, palladoarsenide Pd_2As), bismuthides (sobolevskite PdBi, froodite $PdBi_2$), stannides (paolovite Pd_2Sn, atokite Pd/Pt_3Sn), tellurides (kotulskite $PdTe/Bi_2$, naldretteite Pd_2Sb), plumbides (zvyaginstevite Pd_3Pb), and sulphides (laflameitte $Pd_3Pb_2S_2$, coldwellite Pd_3Ag_2S).

Platinum content is typically associated with arsenic or iron as sperrylite ($PtAs_2$), isoferroplatinum (Pt_3Fe) or tetraferroplatinum (PtFe).

Dominant sulphides include chalcopyrite ($CuFeS_2$), cubanite ($CuFe_2S_3$), pyrite (FeS_2), pyrrhotite (FeS) and minor amounts of nickel (pendlandite $(Fe,Ni)_9S_8$, mackinawite [$(Fe,Ni)_{1+x}S$ (where x = 0 to 0.11)] Fe/Ni_9S_8). Gold and silver values are present as solid solutions within sulphides, as electrum, or in native form.

Copper mineralization is bimodal and present as both coarse- and fine-grained sulphides. The focus of studies by Gen Mining during 2020-2023 was to optimize PGM recovery. The same process conditions inherently yielded favorable performance for copper recovery.

13.3 Material Characterization

The Marathon Deposit is situated within a gabbro intrusion (coarse grained crystalline matrix associated with plagioclase, clinopyroxene, olivine, magnetite, apatite with minor amounts of biotite, chlorite, orthopyroxene, amphibole and feldspar). Emplacement of the gabbro involved multiple events resulting in a fine grained to pegmatic, brecciated metabasalt host rock. Material characterization details are listed in Table 13.1.

Based on breakage testwork, mineralized material of interest can be described as follows:

- Moderately high competency with a design (75^{th} percentile) SMC Axb value of 38 and SPI value of 100 minutes lab scale grinding time that supports the application of conventional SAG milling.
- Moderately high hardness with a design (75^{th} percentile) ball mill work index 17.5 KWh/t.
- Moderate abrasiveness with a design (average) abrasion index of 0.35 g.

An SMC Axb factor of 38 indicates reasonable material competency that supports the application of conventional SAG milling.

The natural pH of mineralized material tested was in the range of 8.0 to 9.0 which implies that the potential for corrosion or corrosion/abrasion within grinding and flotation circuits will be limited.

Table 13.1: Material Characterization

Parameter	Unit	Plant Feed
Specific Gravity	t/m^3	3.09
SMC Axb	--	38
Bond Crushing Index	kWh/t	18.6
Bond Ball Mill Work Index	kWh/t	17.5
Bond Abrasion Index	g	0.35

13.4 Composite Sample Details

Details on composites used in testwork undertaken by Stillwater Canada during the period 2010-2012 can be found in the 2021 FS Report (GMS, 2021). A 2.5 tonne remnant sample of 2012 Composite 3 for the Project, which was stored at site since 2012 in a wooden crate as ½ split HQ drill core, was shipped to SGS Lakefield and tested extensively during 2020 for comparison to past testwork.

Additional samples selected for bench-scale testing during 2020 included a total of eight 150 kg samples including five Main Zone samples identified as 2020 MZ-1 through 2020 MZ-5 and three W Horizon samples identified as 2020 WH-1 through 2020 WH-3 are summarized in Table 13.2.

Selection of the samples for 2020 bench-scale testing sought to provide samples that slightly varied in Pd and Cu grade. Another objective was to pursue contiguous discrete intervals, implying that aside from representing variable metal and mineral content, the composite sample also represented a spatial volume within the deposit that could conceptually be mined and fed to the process plant at some point in the time. This compares to a composite sample comprised of intervals from across the entire wireframe and surface of the Marathon Deposit which, while of interest, would realistically not represent a mix of lithology that might be considered as feed to the process plant.

Additional samples selected for Phase 2 bench-scale cleaner circuit optimization and pilot plant testing involved larger 1.5 tonne bulk samples selected from surface outcrops in the Main Zone and W Horizon. 2020 bulk sample locations and specifics are outlined in Table 13.2.

More recently, to support 2022 Phase 3 metallurgical test program and additional locked cycle flotation testing, five additional composite samples were defined, identified as Main Zone (MZ-22-1, MZ-22-2), Central Pit (CP-22-1) and W Horizon (WH-22-1, WH-22-2). Specifics for the composites are detailed in Table 13.4. All composites are show spatially in Figure 13.2.

Also included with 2022 Phase 3 metallurgical testing was the selection of 25 additional samples which were subject to standard ball mill work index testing. Samples specifics and material characterization data are summarized in Table 13.5 for reference.

Table 13.2: 2020 Gen Mining W Horizon and Main Zone Bulk Samples

Composite	Cu	Fe	S	Au	Pt	Pd
	(%)	**(%)**	**(%)**	**(g/t)**	**(g/t)**	**(g/t)**
2012 Composite 3 Bulk Sample	0.36	10.1	1.67	0.26	0.11	0.50
2020 North Pit (Main Zone) Composite Bulk Sample	0.23	9.65	0.44	0.07	0.15	0.63
2020 South Pit (W Horizon) Composite Bulk Sample	0.09	6.39	0.08	0.58	0.42	0.90

Table 13.3: 2022 Gen Mining Main Zone, Central Pit, W Horizon Composite Samples

Composite	Weight kg	Cu %	Pd g/t	Pt g/t	Au g/t	Ag g/t
WH-22-1	96.62	0.30	0.91	0.22	0.09	1.45
WH-22-2	95.27	0.23	0.66	0.19	0.80	1.13
CP-22-1	85.90	0.13	0.63	0.16	0.06	0.76
MZ-22-1	96.58	0.17	0.93	0.32	0.13	0.84
MZ-22-2	77.73	0.19	0.64	0.22	0.07	1.08

Figure 13.2: Location of Composite locations

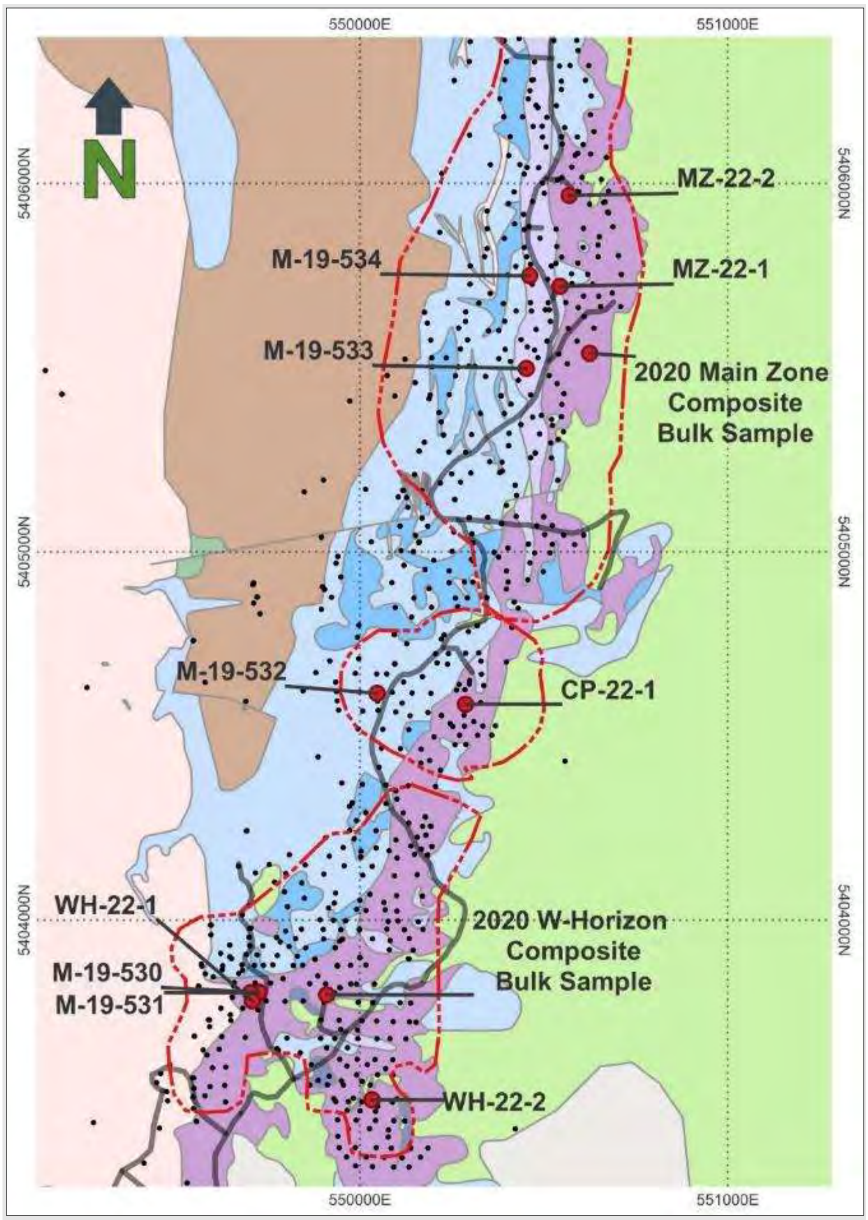


Table 13.4: 2022 Testwork Composite Intervals

Drill Hole	Composite	Interval (m)		Weight	Cu	Pd	Pt	Au	Ag
		From	To	kg	%	g/t	g/t	g/t	g/t
M-22-605	COMP-WH-22-1	7	8	1.77	0.23	1.72	0.48	0.12	0.90
M-22-605	COMP-WH-22-1	8	9	1.99	0.28	2.78	0.88	0.44	2.00
M-22-605	COMP-WH-22-1	9	11	4.09	0.11	5.72	1.52	1.14	0.40
M-22-605	COMP-WH-22-1	11	13	4.41	0.10	1.14	0.61	0.04	0.40
M-22-605	COMP-WH-22-1	13	15	3.90	0.02	0.63	0.19	0.07	0.20
M-22-605	COMP-WH-22-1	15	17	3.78	0.13	0.64	0.23	0.11	0.60
M-22-605	COMP-WH-22-1	17	19	3.90	0.08	0.26	0.12	0.05	0.40
M-22-605	COMP-WH-22-1	19	21	3.85	0.15	0.47	0.12	0.08	0.70
M-22-605	COMP-WH-22-1	21	23	3.79	0.24	0.31	0.12	0.06	1.00
M-22-605	COMP-WH-22-1	23	25	3.76	0.16	0.49	0.20	0.05	0.90
M-22-605	COMP-WH-22-1	25	27	4.07	0.13	0.48	0.21	0.06	0.60
M-22-605	COMP-WH-22-1	27	29	3.45	0.18	1.24	0.40	0.10	0.90
M-22-605	COMP-WH-22-1	29	31	3.78	0.03	0.09	0.06	0.01	0.10
M-22-605	COMP-WH-22-1	31	33	3.91	0.24	0.34	0.11	0.05	1.30
M-22-605	COMP-WH-22-1	33	35	3.82	0.11	0.12	0.04	0.02	0.50
M-22-605	COMP-WH-22-1	35	37	3.82	0.28	0.21	0.09	0.05	1.50
M-22-606	COMP-WH-22-2	4	6	3.17	0.08	1.15	0.63	0.13	0.60
M-22-606	COMP-WH-22-2	6	8	3.43	0.04	0.19	0.08	0.02	0.30
M-22-606	COMP-WH-22-2	8	10	3.34	0.05	0.28	0.10	0.05	0.40
M-22-606	COMP-WH-22-2	10	12	3.65	0.21	0.52	0.18	0.08	1.20
M-22-606	COMP-WH-22-2	12	13	1.70	0.62	1.91	0.70	0.33	3.90
M-22-606	COMP-WH-22-2	13	14	1.70	0.09	0.15	0.06	0.02	0.60
M-22-606	COMP-WH-22-2	14	16	3.44	0.19	0.53	0.21	0.06	1.10
M-22-606	COMP-WH-22-2	16	18	3.52	0.35	0.85	0.22	0.10	1.60
M-22-606	COMP-WH-22-2	18	20	3.97	0.18	0.34	0.12	0.04	1.00
M-22-606	COMP-WH-22-2	20	22	3.34	0.36	0.69	0.16	0.10	1.80
M-22-606	COMP-WH-22-2	22	23	1.78	0.56	0.72	0.19	0.10	2.70
M-22-606	COMP-WH-22-2	23	24	1.76	0.15	0.44	0.20	0.04	0.80
M-22-606	COMP-WH-22-2	24	26	3.27	0.13	0.59	0.08	0.04	0.70
M-22-606	COMP-WH-22-2	26	28	3.41	0.08	0.16	0.04	0.02	0.50
M-22-606	COMP-WH-22-2	28	30	3.49	0.05	0.10	0.03	0.01	0.30
M-22-606	COMP-WH-22-2	30	32	3.16	0.01	0.01	0.01	0.00	0.10
M-22-606	COMP-WH-22-2	32	33	1.79	0.02	0.04	0.02	0.01	0.20
M-22-606	COMP-WH-22-2	33	34	1.64	0.06	2.50	0.75	0.09	0.50
M-22-606	COMP-WH-22-2	34	35	1.91	0.05	0.31	0.05	0.02	0.30
M-22-607	COMP-CP-22-1	24	26	3.53	0.08	0.46	0.14	0.03	0.90
M-22-607	COMP-CP-22-1	26	27	1.86	0.20	0.51	0.17	0.07	0.70
M-22-607	COMP-CP-22-1	27	28	1.83	0.56	2.62	0.74	0.44	2.80
M-22-607	COMP-CP-22-1	28	30	3.11	0.39	1.94	0.32	0.18	1.80
M-22-607	COMP-CP-22-1	30	32	3.58	0.38	1.80	0.52	0.12	1.90
M-22-607	COMP-CP-22-1	32	34	3.50	0.06	0.25	0.05	0.02	0.40
M-22-607	COMP-CP-22-1	34	36	3.69	0.02	0.03	0.01	0.00	0.20

Drill Hole	Composite	Interval (m)		Weight	Cu	Pd	Pt	Au	Ag
		From	To	kg	%	g/t	g/t	g/t	g/t
M-22-607	COMP-CP-22-1	36	38	3.44	0.16	1.60	0.15	0.07	1.00
M-22-607	COMP-CP-22-1	38	40	3.72	0.03	0.32	0.05	0.01	0.30
M-22-607	COMP-CP-22-1	40	42	3.93	0.01	0.05	0.07	0.00	0.10
M-22-607	COMP-CP-22-1	42	44	3.70	0.01	0.44	0.26	0.00	0.20
M-22-607	COMP-CP-22-1	44	46	3.53	0.04	0.15	0.10	0.09	0.30
M-22-607	COMP-CP-22-1	46	48	3.14	0.08	0.16	0.13	0.06	0.60
M-22-607	COMP-CP-22-1	48	50	3.35	0.05	0.29	0.09	0.02	0.40
M-22-607	COMP-CP-22-1	50	52	3.27	0.06	0.35	0.10	0.04	0.40
M-22-607	COMP-CP-22-1	98	100	3.42	0.34	0.98	0.16	0.06	1.60
M-22-607	COMP-CP-22-1	100	102	3.76	0.52	1.07	0.18	0.08	2.40
M-22-608	COMP-MZ-22-1	96	98	3.72	0.42	0.84	0.26	0.08	2.10
M-22-608	COMP-MZ-22-1	98	100	3.48	0.31	1.14	0.28	0.12	1.60
M-22-608	COMP-MZ-22-1	100	102	3.31	0.63	2.01	0.45	0.18	3.00
M-22-608	COMP-MZ-22-1	102	104	3.80	0.25	0.77	0.17	0.08	1.20
M-22-608	COMP-MZ-22-1	104	106	3.46	0.11	0.24	0.05	0.02	0.60
M-22-608	COMP-MZ-22-1	106	108	3.30	0.45	1.11	0.32	0.10	2.00
M-22-608	COMP-MZ-22-1	108	110	3.33	0.41	1.15	0.25	0.11	2.00
M-22-608	COMP-MZ-22-1	110	112	3.60	0.62	2.37	0.54	0.25	3.00
M-22-608	COMP-MZ-22-1	112	114	3.04	0.45	1.68	0.42	0.15	2.40
M-22-608	COMP-MZ-22-1	114	116	3.47	0.64	2.15	0.50	0.20	3.10
M-22-608	COMP-MZ-22-1	116	118	3.46	0.19	0.66	0.15	0.06	0.90
M-22-608	COMP-MZ-22-1	118	120	3.46	0.04	0.18	0.02	0.02	0.30
M-22-608	COMP-MZ-22-1	120	122	3.46	0.14	0.44	0.12	0.04	0.70
M-22-608	COMP-MZ-22-1	122	124	3.54	0.02	0.01	0.01	0.00	0.10
M-22-608	COMP-MZ-22-1	124	126	3.20	0.02	0.04	0.03	0.00	0.10
M-22-609	COMP-MZ-22-2	69	71	3.24	0.16	0.32	0.10	0.03	0.90
M-22-609	COMP-MZ-22-2	71	73	3.38	1.04	2.54	0.45	0.28	5.20
M-22-609	COMP-MZ-22-2	73	75	3.58	0.62	1.64	0.51	0.26	3.10
M-22-609	COMP-MZ-22-2	75	77	3.45	0.30	1.34	0.37	0.26	1.50
M-22-609	COMP-MZ-22-2	77	79	3.78	0.47	1.56	0.37	0.09	2.20
M-22-609	COMP-MZ-22-2	79	81	3.41	0.17	0.59	0.18	0.04	0.70
M-22-609	COMP-MZ-22-2	81	83	3.76	0.24	0.64	0.17	0.07	1.20
M-22-609	COMP-MZ-22-2	83	85	3.64	0.08	0.46	0.13	0.03	0.40
M-22-609	COMP-MZ-22-2	85	87	3.66	0.02	0.08	0.03	0.02	0.20
M-22-609	COMP-MZ-22-2	87	89	3.70	0.44	0.70	0.28	0.13	2.20
M-22-609	COMP-MZ-22-2	89	91	3.49	0.10	0.71	0.26	0.05	0.50
M-22-609	COMP-MZ-22-2	91	93	3.65	0.12	0.32	0.11	0.07	0.60
M-22-609	COMP-MZ-22-2	93	95	3.82	0.07	0.01	0.01	0.01	0.30
M-22-609	COMP-MZ-22-2	95	97	3.69	0.09	0.07	0.05	0.01	0.40
M-22-609	COMP-MZ-22-2	97	99	4.50	0.08	0.11	0.09	0.03	0.50
M-22-609	COMP-MZ-22-2	99	101	3.93	0.09	0.04	0.04	0.01	0.40
M-22-609	COMP-MZ-22-2	101	103	4.05	0.04	0.05	0.02	0.01	0.10

Table 13.5: 2022 Bond Ball Mill Grindability Data

Sample Name	Work Index	Hardness Percentile
MZ-22-1	18.5	87
MZ-22-2	18.6	88
CP-22-1	17.5	82
WH-22-1	17.9	84
WH-22-2	19.1	90
BWI-CP-22-01	17.4	82
BWI-NP-22-01	15.7	66
BWI-NP-22-02	16.9	78
BWI-NP-22-03	17.3	81
BWI-NP-22-04	17.5	83
BWI-NP-22-05	17.6	83
BWI-NP-22-06	16.7	77
BWI-NP-22-07	17.6	83
BWI-NP-22-08	15.8	68
BWI-NP-22-09	17.5	82
BWI-NP-22-10	17.1	79
BWI-NP-22-11	16.2	72
BWI-NP-22-12	17.0	79
BWI-NP-22-13	14.9	58
BWI-NP-22-14	16.6	76
BWI-NP-22-15	15.9	69
BWI-SP-22-01	15.5	64
BWI-SP-22-02	18.4	87
BWI-SP-22-03	18.3	87
BWI-SPP-22-04	16.1	71

13.5 Grind Size Optimization

The determination of optimal grind size for flotation was a component of the 2020 metallurgical test program completed by Gen Mining and was validated during the 2022 test program.

Previous technical studies by Stillwater Canada during 2010-2013 considered a [Grind-Float] concept with initial Cu flotation at a feed size P_{80} of 212 µm, followed by regrinding of Cu rougher tails to a P_{80} of 110 µm as feed to a secondary PGM flotation circuit. The split flowsheet was intended to produce a high-grade Cu concentrate and a lower grade PGM-Cu concentrate.

The optimized process flowsheet (Figure 13.1) considered by Gen Mining involves rougher flotation at natural pH, followed by rougher concentrate regrinding at pH 11. The first, second and third cleaners would be operated in closed circuit with rejection of iron sulphides to first cleaner tailings.

A product grind of P_{80} of 106 µm (150 mesh) to rougher flotation yielded near optimal extraction of 93.6% Rec Pd and 97.7% Rec Cu (Rec = recovered) to a rougher concentrate at 15% mass pull from a flotation feed grade of 0.36% Cu and 0.53 g/t Pd (Figure 13.3).

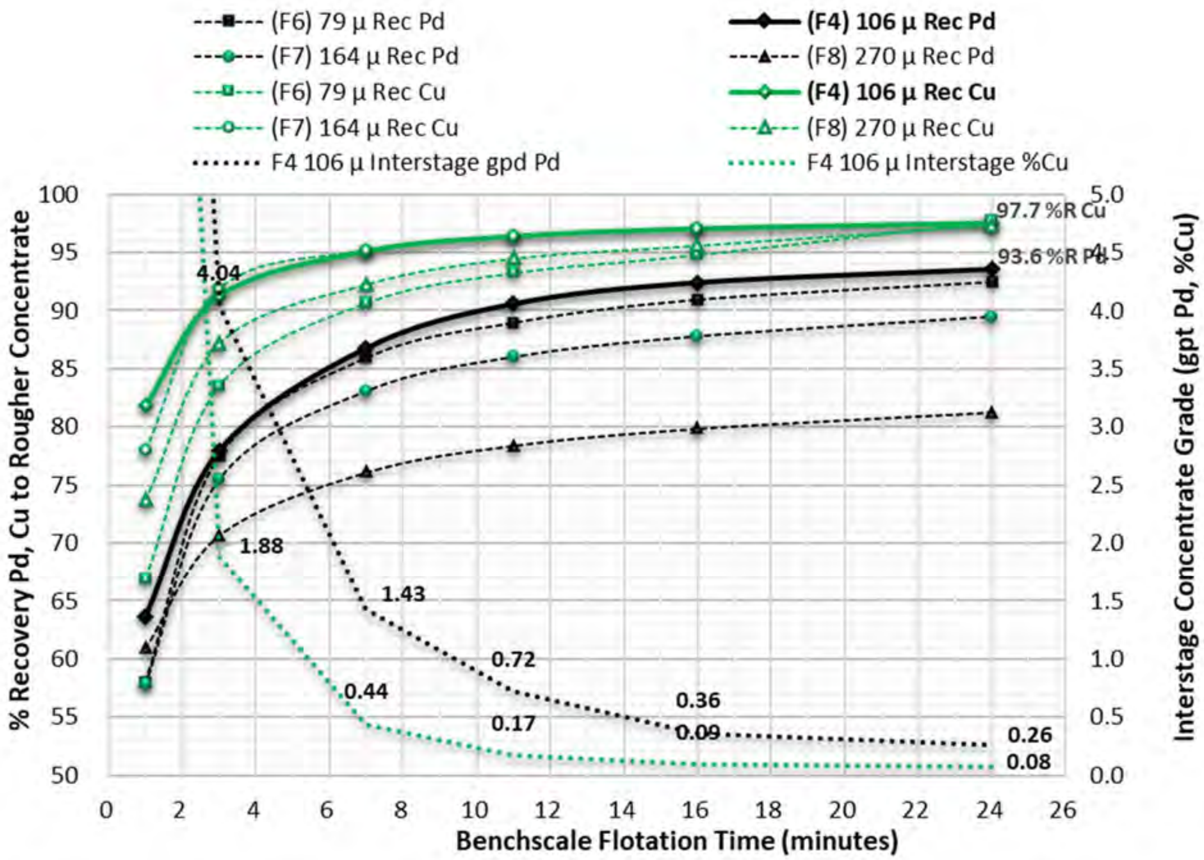
Additional Phase 3 testwork completed during 2022 studied the expected incremental difference in metal recovery to rougher concentrate at varying flotation feed grind size from a P_{80} 75 µm to a P_{80} 200 µm. Testwork was completed by subjecting eight separate samples including: (i) 2012 Pilot Plant Composite 3, (ii) 2020 Main Zone Pilot Plant Composite, (iii) 2020 W Horizon Pilot Plant Composite, (iv) 2022 Main Zone Composite MZ-22-1, (v) 2022 Main Zone Composite MZ-22-2, (vi) 2022 W Horizon Composite WH-22-1, (vii) 2022 W Horizon Composite WH-22-2 and (viii) 2022 Central Pit Composite CP-22-1 to rougher flotation with baseline flotation retention time and reagent addition rates. The variability in expected recovery at coarser grind size is of interest for future process plant throughput optimization efforts and is detailed in Table 13.6.

The incremental difference in metal recovery to final concentrate will be similar in magnitude when compared to metal recovery to rougher concentrate. Respective metals have a different, yet relatively constant recovery, or efficiency factor, from rougher concentrate to final concentrate which is influenced by concentrate regrind size, and cleaner circuit inter-stage mass pull. Section 13.11. A flotation feed grind size of P_{80} 106 µm was defined as a baseline parameter and remains the basis for present GeoMet modeling estimates and grinding circuit design.

Table 13.6: Flotation Feed Grind Size versus Metal Recovery to Rougher Concentrate

Metal	Regression Formula	P_{80} 75 µm	P_{80} 106 µm	P_{80} 125 µm	P_{80} 150 µm	P_{80} 175 µm	P_{80} 200 µm
		Metal Recovery As a Function of Grind Size					
Copper	%Rec Cu = (-0.021 x P_{80} µm) + 97.57	96	95.3	94.9	94.4	93.9	93.4
Palladium	%Rec Pd = (-0.067 x P_{80} µm) + 99.29	94.2	92.2	90.9	89.2	87.5	85.8
Platinum	%Rec Pt = (-0.057 x P_{80} µm) + 95.57	91.3	89.5	88.4	87	85.5	84.1
Gold	%Rec Au = (-0.030 x P_{80} µm) + 82.59	80.3	79.4	78.8	78	77.3	76.5

Figure 13.3: Flotation Feed Grind Size vs Recovery and Interstage Conc Grade (2012 Composite 3)



13.6 Flotation Reagent Selection

An evaluation of optimal flotation collectors and frother for the Project was a component of the 2020 metallurgical testing completed by Gen Mining.

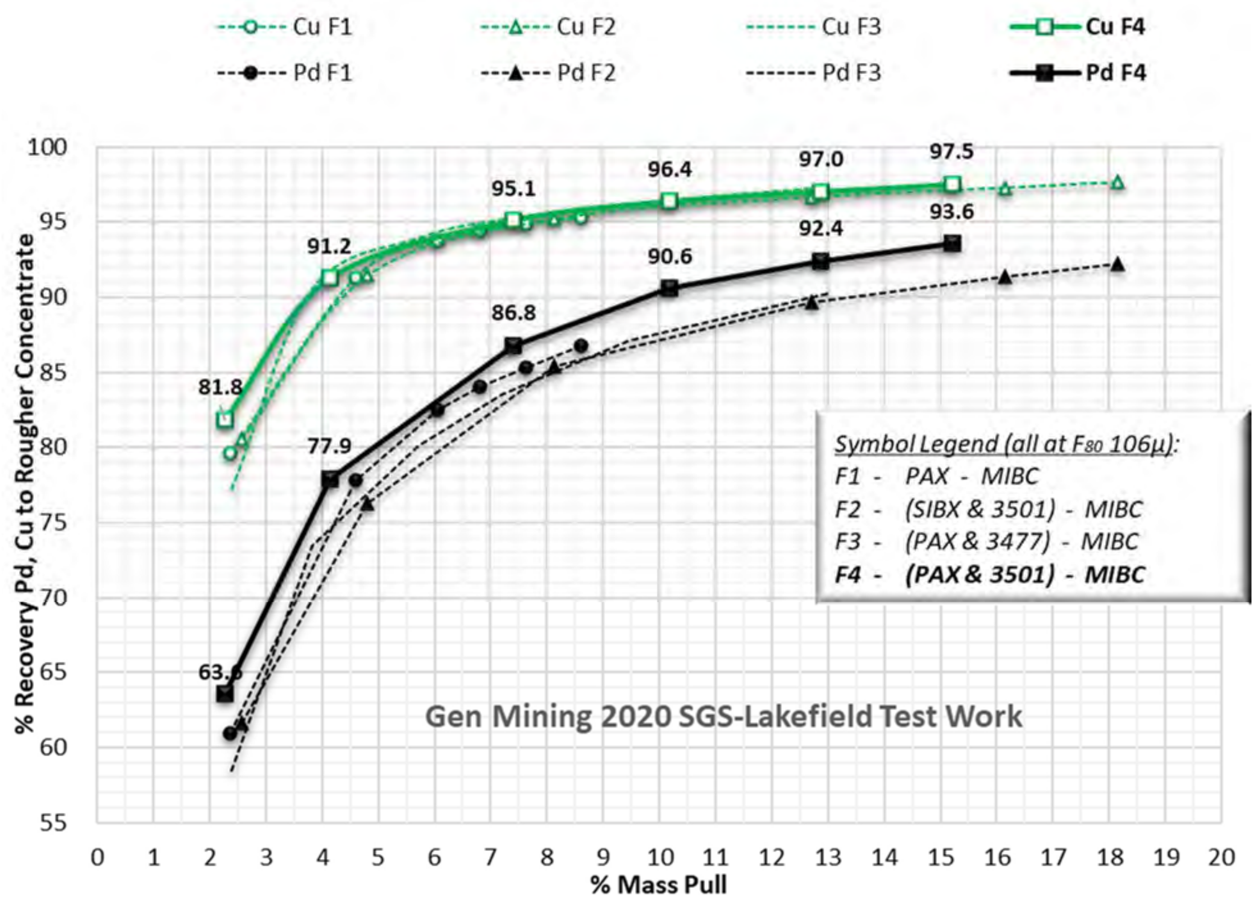
During previous 2012 metallurgical studies on the Project, potassium amyl xanthate (PAX) and isobutyl dithiophosphate (Aero 3477) were applied in the majority of testwork by RDi, ALS, and XPS. A mini-pilot plant completed by XPS during September 2009 considered a blend of 141 g/t Aero 3418A, 27 g/t PAX, and 24 g/t Aero 3477 as an ideal combination with 18 g/t MIBC and 34 g/t W34 as frother addition. Testwork completed with ALS in 2013 followed the same XPS reagent suite and addition rate. Previous testwork by RDi in 2009 considered PAX and MIBC as a reagent scheme with Aero 3477 used as a promoter on some materials.

A review of peer palladium producers suggests that PAX, sodium isobutyl xanthate (SIBX) and dodecyl mercaptan (DDM) are often applied as effective PGM promoters and collectors.

The intention of the 2020 testing was to re-evaluate the performance of collectors and frother on the 2012 Composite 3, which represents a blend of lithological types from across the entire Marathon Deposit.

For each of the discrete interval composites, 2020 MZ-1 to 2020 MZ-5 and 2020 WH-1 to 2020 WH-3, as well as 2012 Composite 3, collector combinations including PAX, (SIBX + Aero 3501), (PAX + Aero 3477) and (PAX + Aero 3501) yielded similar performance with respect to copper, and markedly different performance for slower floating PGM mineralization indicated in Figure 13.4 for 2012 Composite 3.

Figure 13.4: Flotation Circuit Reagent Selection (2012 Composite 3)



A key benefit associated with Aero 3501, an isoamyl di-thiophosphate, is that aside from being an effective PGM-Cu promoter, this collector also has a slight frothing characteristic.

For all samples tested, a rougher collector combination of (PAX + Aero 3501) or (SIBX + Aero 3501) exhibited an increase in metal recovery because of increased mass pull, surpassing the performance of PAX or (PAX + Aero 3477). The increased mass pull to rougher concentrate was not a function of frother addition rate, or the way the test was conducted, but from improved froth characteristics present with Aero 3501. PAX is a more aggressive collector than SIBX and in conjunction with Aero 3501 yielded a gain of 8% Pd recovery to rougher concentrate relative to PAX only, and 4% Pd recovery increase relative to (PAX + Aero 3477).

Another intention of reagent selection was to pursue moderately priced products that are readily available from reliable manufacturers.

As an outcome of reagent optimization, average reagent dosing of 35 g/t PAX, 35 g/t 3501, and 25 g/t MIBC were selected for rougher flotation baseline testing and remain the base case for the detailed design.

13.7 Rougher Flotation Performance and Rate Kinetics

Rougher flotation rate kinetic testing was a component of the 2020 metallurgical program completed by Gen Mining. Results of rougher kinetic testing are indicated on Figure 13.5 to Figure 13.7, as well as in Table 13.7. Flotation performance for both Pd and Cu were excellent for all samples tested.

Figure 13.5: Rougher Flotation Rate Kinetics and Interstage Grade

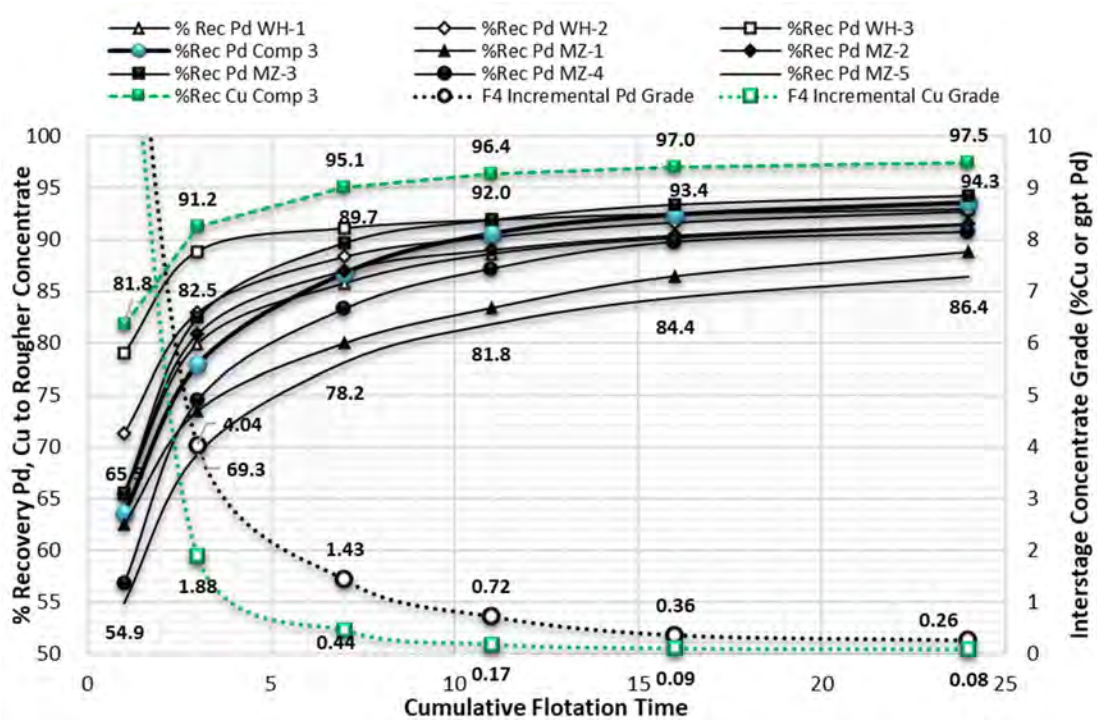


Figure 13.6: North Pit (Main Zone) Composites - Rougher Conc Rate Kinetics and Interstage Grade

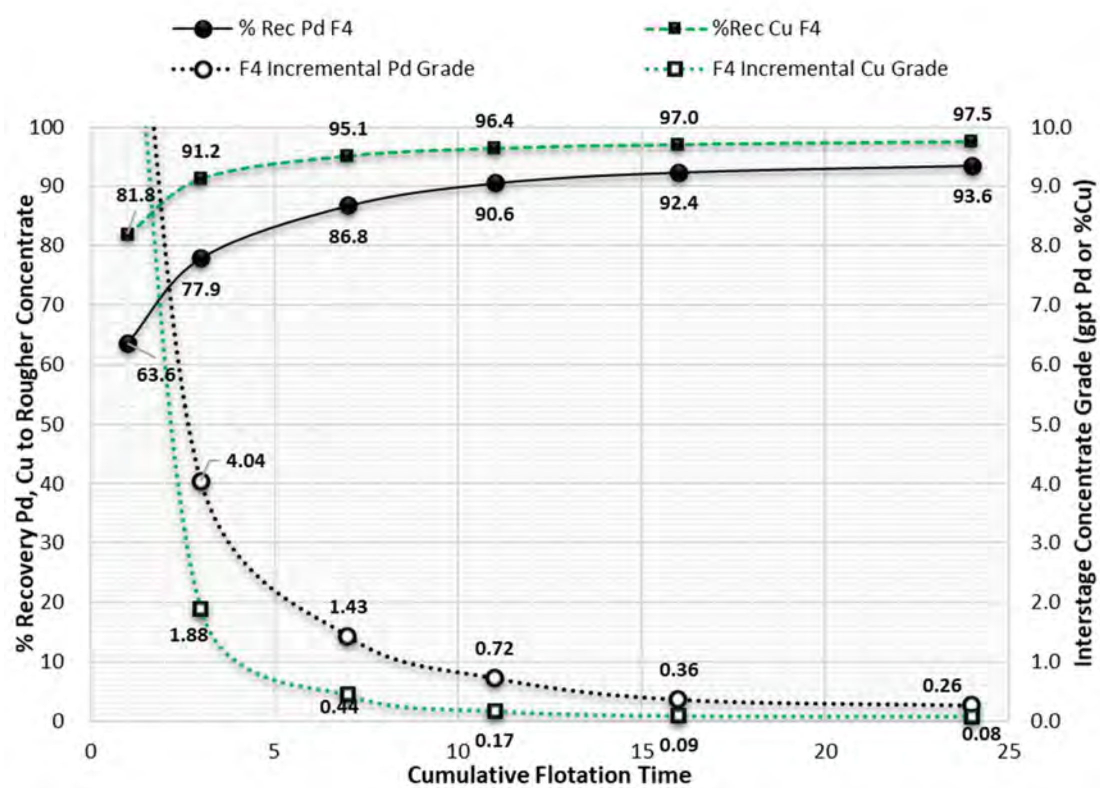
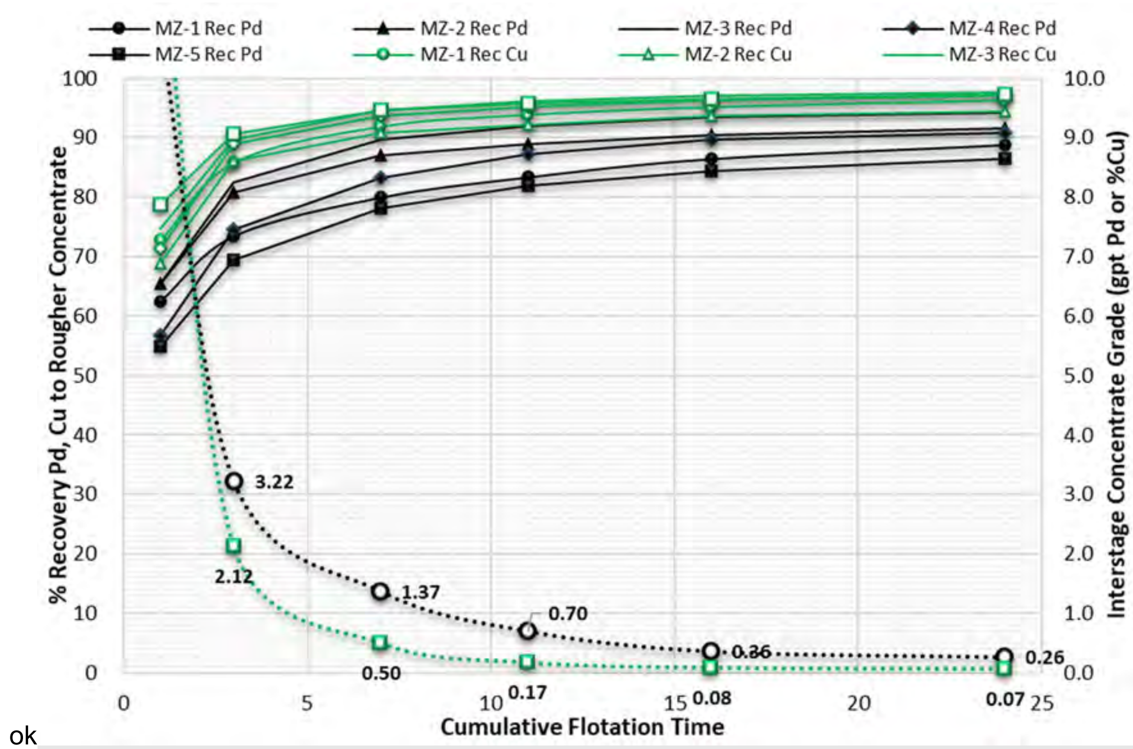


Figure 13.7: South Pit (W Horizon) Composites - Rougher Conc Rate Kinetics & Interstage Grade



Metallurgical testwork completed previously during 2010-2013 evaluated five separate bulk composites representing the entire cross section of the Marathon Deposit. The 2012 Composite 3, retested in 2020, is a blend of 890 separate intervals from the Main Zone, South Zone and W Horizon at varying grade. Testwork completed in 2020 confirmed that oxidation effects on the bulk composite were minimal, with the ½ split HQ core used for the testwork having been stored outdoors since 2012 in wooden crates. This information bodes well for the future processing of low-grade stockpiled material that is expected to remain exposed for 7 to 10 years as part of the LOM.

In contrast to previous testing, discrete interval composite samples selected for the 2020 testing were chosen from the Main Zone and W Horizon with a grade range from 0.05 to 0.47% Cu, 0.38 to 2.62 g/t Pd, and a Pd/Cu ratio from 1.2 to 51.4. The primary difference for sample selection in 2020 bench-scale testwork is that discrete interval samples were from specific drill holes and a continuous length that would be representative of a mineable bench. Composite sample specifics are detailed in Subsection 13.4.

Specific details and findings associated with rougher flotation rate kinetic testing are as follows:

- Baseline conditions were applied to all rougher flotation kinetic tests involving a flotation feed size of P_{80} 106 µm, natural pH in the order of 8.5, 30% slurry density, and the staged addition of collectors including 35 g/t PAX, 35 g/t 3501, and MIBC as a frother
- Mass pull to rougher concentrate between 5.0 to 21.0%
- Recovery to rougher concentrate varied from 90.1% to 97.8% for Cu, 81.9% to 94.9% for Pd, 76.2% to 95.3% for Pt and 66.7% to 93.3% for Au based over the range of feed grades analyzed
- Within the first seven minutes of bench-scale rougher flotation, 96.6% of final Cu recovery and 93.6% of final Pd recovery was achieved within 16 minutes, and 99.2% of final Cu recovery and 98.6% of final Pd recovery with final recovery of respective metals assumed after 24 minutes.
- For baseline conditions applied, Cu flotation rate kinetics were rapid with excellent recoveries. Pd flotation rate kinetics were slower in comparison with incremental recovery gains of 1.5% Pd recovery versus 0.5% Cu recovery with an extended bench-scale flotation time from 17 to 24 minutes.
- The scale up factor from bench-scale to full-scale design is typically in the order of 2.0 for copper sulphide only applications. The Project considers a scale up factor of 2.5 to compensate for the slower PGM flotation rate kinetics, and the relative value of PGM metals.
- Rougher flotation retention time of 24 minutes and a target mass pull of 12-15% feed weight to rougher concentrate were defined as baseline parameters and targets for subsequent testing and process design criteria (average of 13.3% used).

Results from metallurgical testing confirm an expected range in sulphide sulfur content in the rougher tailings from 0.01 to 0.10% S^{2-} and in first cleaner tailings from 0.10 to 10.0% S^{2-}. The recovery and separation of sulphides within the process flowsheet supports an intention of project design to produce separate low sulphide NAG tailings and sulfidic PAG first cleaner tailings for co-disposal in such a way as to prevent long-term ARD potential.

Table 13.7: Rougher Flotation Kinetic Testing

Sample	Pd/Cu	Flotation Feed				Flotation Tailings				Combined Rougher Concentrate #1 to 6						
	Ratio	Cu (%)	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Pd (g/t)	Pt (g/t)	Au (g/t)	%Wght	%Rec Cu	%Rec Pd	%Rec Pt	%Rec Au	%Rec Ni	%Rec Fe
2012 Composite #3 (entire deposit)	1.4	0.37	0.53	0.18	0.08	0.01	0.04	0.02	0.02	15.2	97.5	93.6	90.7	78.0	76.3	32.8
2020 W-Horizon Composite 1	24.5	0.08	2.01	0.71	0.13	0.00	0.20	0.06	0.02	14.0	96.8	91.4	92.8	86.4	31.1	17.5
2020 W-Horizon Composite 2	6.5	0.19	1.23	0.30	0.09	0.01	0.10	0.03	0.02	10.7	96.8	92.7	91.0	80.8	56.9	16.6
2020 W-Horizon Composite 3	51.4	0.05	2.62	0.79	0.14	0.00	0.11	0.02	0.02	11.5	94.8	96.3	97.7	86.9	23.5	12.9
2020 Main Zone Composite 1	2.3	0.31	0.71	0.18	0.07	0.01	0.09	0.03	0.02	11.8	96.2	88.8	85.2	76.2	48.7	16.7
2020 Main Zone Composite 2	1.9	0.59	1.12	0.30	0.13	0.04	0.11	0.03	0.03	14.2	94.4	91.6	91.4	80.8	76.6	24.1
2020 Main Zone Composite 3	1.9	0.46	0.88	0.24	0.11	0.01	0.06	0.02	0.02	16.3	97.6	94.3	93.0	84.3	79.4	27.9
2020 Main Zone Composite 4	1.6	0.47	0.73	0.20	0.08	0.02	0.08	0.02	0.02	15.9	97.1	90.7	91.8	79.3	84.8	35.1
2020 Main Zone Composite 5	1.2	0.31	0.38	0.12	0.06	0.01	0.06	0.02	0.02	13.8	97.2	86.4	85.5	72.6	79.9	27.8

Rate kinetic float tests were completed on 2 kg feed samples and baseline conditions: F80 106 micron grind size, 35 gpt PAX, 35 gpt, 3501, MIBC, natural pH 8.5

Details	COPPER (%)									PALLADIUM (g/t)								
	Comp 3	MZ-1	MZ-2	MZ-3	MZ-4	MZ-5	WH-1	WH-2	WH-3	Comp 3	MZ-1	MZ-2	MZ-3	MZ-4	MZ-5	WH-1	WH-2	WH-3
Flotation Feed	0.37	0.31	0.59	0.46	0.47	0.31	0.08	0.19	0.05	0.53	0.71	1.12	0.88	0.73	0.38	2.01	1.23	2.62
Flotation Tails	0.01	0.01	0.04	0.01	0.02	0.01	0.00	0.01	0.00	0.04	0.09	0.11	0.06	0.08	0.06	0.20	0.10	0.11
Ro Conc Mass Pull %	15.2	11.8	14.2	16.3	15.9	13.8	14.0	10.7	11.5	15.2	11.8	14.2	16.3	15.9	13.8	14.0	10.7	11.5
Ro Conc 1 min %Rec	81.8	72.7	68.9	74.7	71.3	78.8	71.8	70.8	79.7	63.6	62.5	65.4	65.5	56.7	54.9	65.4	71.3	82.4
Ro Conc 1 - 3min %Rec	91.2	85.7	86.1	89.8	89.0	90.7	90.4	86.2	88.9	77.9	73.5	80.9	82.5	74.5	69.3	79.9	83.0	90.7
Ro Conc 1 - 7 min %Rec	95.1	91.8	90.7	94.7	93.6	94.7	94.0	92.9	92.0	86.8	80.0	87.0	89.7	83.3	78.2	85.8	88.3	93.8
Ro Conc 1 - 11 min %Rec	96.4	93.8	92.1	96.2	95.2	95.8	95.5	94.7	93.0	90.6	83.4	89.0	92.0	87.2	81.8	88.6	90.2	94.9
Ro Conc 1 - 16 min %Rec	97.0	95.1	93.6	97.1	96.3	96.5	96.1	95.8	93.9	92.4	86.5	90.4	93.4	89.8	84.4	90.2	91.7	95.6
Ro Conc 1 - 24 min %Rec	97.5	96.2	94.4	97.6	97.1	97.2	96.8	96.8	94.8	93.6	88.8	91.6	94.3	90.7	86.4	91.4	92.7	96.3

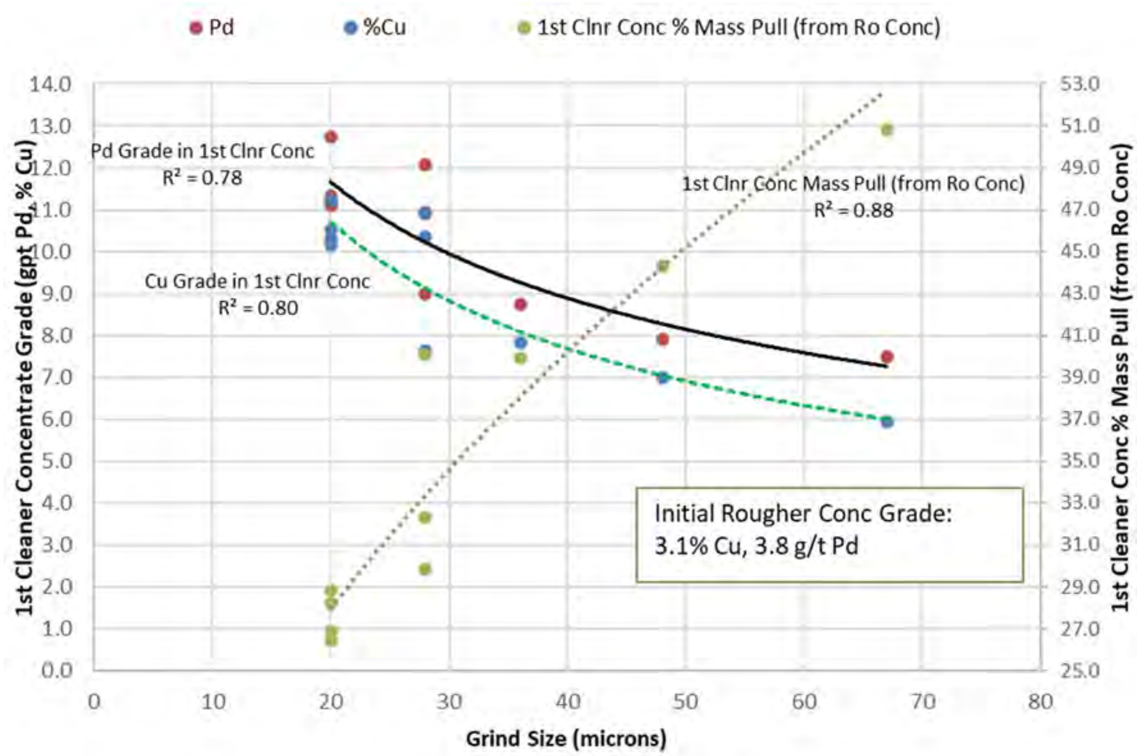
13.8 Rougher Concentrate Regrind Size and Specific Energy

Optimization of rougher concentrate regrind size was a component of the 2020 metallurgical program completed by Gen Mining. Results of rougher concentrate regrind testing are indicated on Figure 13.8. Data suggests a significant decrease in mass pull with maintained metal recovery and increasing Cu and Pd grade in 1st Cleaner concentrate grade as concentrate regrind size decreases towards and below a concentrate regrind particle size of P_{80} 20 µm. Comparative first cleaner flotation performance at varying rougher concentrate regrind size was completed on a rougher concentrate sample from the 2012 Composite 3.

Flotation conditions applied in the regrinding of rougher concentrate included the addition of lime at pH 10, followed by an adjustment to pH 11 with lime to first Cleaner Feed. Cleaner circuit collector addition included PAX (50 to 100 g/t) and Aero 3501 (50 g/t). No MIBC or frother addition was required. Reagent addition rates listed for first cleaner addition are relative to rougher concentrate tonnage and would be divided by a factor of 6.7 (assumes 15% mass pull to rougher concentrate) for equivalent addition rate per tonne flotation feed.

Irrespective of concentrate regrind size, the flotation response of Pd and Cu to final concentrate was excellent with a range from 94-99% Pd recovery and 94-99% Cu recovery. There was no indication of any detrimental effect on either metal with finer regrind sizes.

Figure 13.8: Rougher Concentrate Regrind Size Optimization



Testwork confirmed a decrease in mass pull to first cleaner concentrate from 51% to 27% with a decrease in rougher concentrate regrind size from P_{80} 68 µm to P_{80} 20 µm. Metal recovery was constant at varied regrind size resulting in an increase in concentrate grade from 7.0 to 11.5 g/t Pd and 6.0 to 10.5% Cu.

A rougher concentrate regrind size P_{80} 18 µm was selected as the target and included in process design criteria. Regrind tests to confirm specific energy requirements for the concentrate regrind HIG (high intensity grinding) mill were completed in 2022 and confirmed as 11.9 kWh/t based on regrind circuit feed. Previously completed 2014 mineralogy studies, are aligned with findings from testwork that sub-20 µm regrind sizes are preferable for mineral release to achieve optimal PGM-Cu concentrate grade. Magnesium silicate (talc) content in first cleaner concentrate at fine regrind size was noted as being less than 6% Mg confirming acceptable liberation of values from gangue materials without the use of depressants such as carboxyl methyl cellulose ("CMC") or Depramin.

Aside from rougher concentrate regrind size, first cleaner performance improved with the introduction of a 30 minute aeration step, which was effective in partially suppressing pyrrhotite in the first cleaners. While pyrite is effectively depressed with regrinding and lime addition to pH 11, pyrrhotite tends to be more persistent and exhibits natural hydrophobic tendencies. The effect of sulphide aeration, in conjunction with decreased PAX addition rates to the first cleaners, improved first cleaner selectivity favoring PGM and Cu flotation and iron sulphide rejection.

In 2022, Gen Mining pursued additional mineralogical studies on 1st cleaner scavenger tailings (2012 Composite 3 LCT Test A, and 2020 W Horizon LCT Test B) to determine whether platinum may be associated with pyrite mineralization. Since pyrite (FeS_2) is rejected as an impurity from the flotation circuit at a slurry pH of 11.0, the potential for Pt losses with pyrite rejection is of obvious importance. Mineralogical studies completed with SGS Lakefield confirmed that in samples tested, there was no association of Pt with pyrite which was determined through QEMSCAN SEM, and TIMA PGM SEM (Tescan Integrated Mineral Analyzer - PGM Scanning Electron Microscopy). The Pt content present in 1st cleaner tails is expected as free, or interlocked sperrylite ($PtAs_2$), which exhibits slower flotation rate kinetics when compared to other mineralization present. Requiring further study for optimization, the effective capture of Pt in rougher flotation of 85.5% to 92.2% Pt recovery, is followed by a slight loss to 1st cleaner scavenger tailings that is influenced by Pt flotation rate kinetics, slurry redox potential, slurry pH, as well as the potential requirement for a more specific Pt focused collector-promoter. Compared to a cleaner circuit efficiency, or metal transfer of 95.6% to 98.7% for Cu, Pd, Au from rougher concentrate to final concentrate (Table 13.13), the lower cleaner circuit efficiency of 84.8% for Pt signifies potential for future improvement opportunities.

13.9 Locked Cycle Flotation Testing

During the period 2020 to 2023, Gen Mining completed a series of locked cycle flotation tests, including cleaner circuit optimization, to expand the GeoMet model database and improve predictive recovery estimates for Cu, Pd, Pt, Au, Ag to final concentrate.

Samples subjected to locked cycle testing includes: (i) 2012 bulk composite 3, (ii) 2020 Main Zone plant composite, (iii) 2020 W Horizon pilot plant composite, (iv) 2022 Main Zone composite MZ-22-1, (v) 2022 Main Zone composite MZ-22-2, (vi) Central Pit composite CP-22-1, (vii) 2022 W-Horizon composite WH-22-1, and (viii) 2022 W Horizon Composite WH-22-2. Composite sample head grades are summarized in Table 13.8 with sample specifics detailed in Subsection 13.4.

Locked cycle testing involved six iterations (A to F) for each sample simulating grinding, rougher flotation, concentrate regrinding, aeration, and first to third cleaners with results detailed in Table 13.8. The schematic for locked cycle flotation testing is outlined in Table 13.9.

Locked cycle test results consider in all cases the metallurgical balance from cycles C-F when stability was confirmed as achieved for the iterative simulation of the flotation circuit. Recovery to final concentrate includes 50% of the minimal metal distribution associated with 3rd cleaner tailings since at the conclusion of the locked cycle test, values in this middling product at full scale would be expected as reporting to either final concentrate, or to 1st cleaner tailings.

Table 13.8: Locked Cycle Test Head Grades and Results

Locked Cycle Test Data		Grind Size	Flotation Feed					Rougher Concentrate Metal Recovery						Final Concentrate Metal Recovery					
		P_{80} µm	Cu %	Pd g/t	Pt g/t	Au g/t	Ag g/t	Wght %	Cu %	Pd %	Pt %	Au %	Ag %	Wght %	Cu %	Pd %	Pt %	Au %	Ag %
2012 Comp 3 (A)	Oct '20	99	0.37	0.57	0.14	0.08	--	11.0	97.0	88.8	87.2	76.3	--	2.3	94.5	80.1	66.8	67.1	--
2012 Comp 3 (B)	Jan '23	106	0.37	0.52	0.15	0.08	1.58	21.0	97.6	89.0	89.3	80.7	75.0	3.8	95.9	83.0	69.5	74.1	67.2
2020 MZ PP Comp	Dec '22	106	0.19	0.48	0.13	0.06	1.18	8.0	94.8	89.9	85.5	69.3	61.0	2.1	93.8	87.2	70.0	67.2	57.0
2022 MZ-22-1	Jan '23	103	0.29	0.94	0.24	0.10	1.65	11.1	95.1	93.2	91.7	82.9	73.1	1.5	91.3	87.7	73.0	76.1	63.3
2022 MZ-22-2	Jan '23	107	0.23	0.60	0.18	0.11	1.31	16.7	96.4	93.1	89.6	84.1	68.1	1.0	94.5	89.3	68.1	80.6	59.8
2020 WH PP Comp (A)	Oct '20	108	0.10	1.23	0.42	0.21	--	5.0	96.3	91.3	88.8	91.2	--	0.8	92.6	87.9	69.7	86.1	--
2020 WH PP Comp (B)	Jan '23	106	0.09	1.24	0.41	0.24	0.95	5.7	89.5	92.2	88.6	92.1	50.2	0.7	86.5	88.6	62.7	88.2	46.1
2022 WH-22-1	Jan '23	107	0.14	0.83	0.28	0.09	1.08	18.9	94.2	93.4	92.0	82.4	60.5	2.3	93.1	90.3	83.9	78.7	53.3
2022 WH-22-2	Jan '23	107	0.19	0.54	0.18	0.09	0.85	16.3	95.5	92.0	90.5	80.5	50.8	0.8	93.6	86.9	73.0	75.9	38.3
2022 CP-22-1	Jan '23	107	0.13	0.54	0.17	0.06	0.88	13.9	93.2	94.0	89.8	72.7	51.3	1.5	91.3	90.6	75.6	69.0	44.4
Locked Cycle Test Average		**106**	**0.21**	**0.75**	**0.23**	**0.11**	**1.19**	**12.7**	**95.0**	**91.7**	**89.3**	**81.2**	**61.3**	**1.7**	**92.7**	**87.1**	**71.2**	**76.3**	**53.7**
Project Design Parameters		106	0.20	0.60	0.20	0.07	1.40	13.3	95.1	91.0	88.8	75.1	67.5	1.1	93.9	80.4	75.3	71.8	62.4
								design values for metal recovery determined from GeoMet Model											

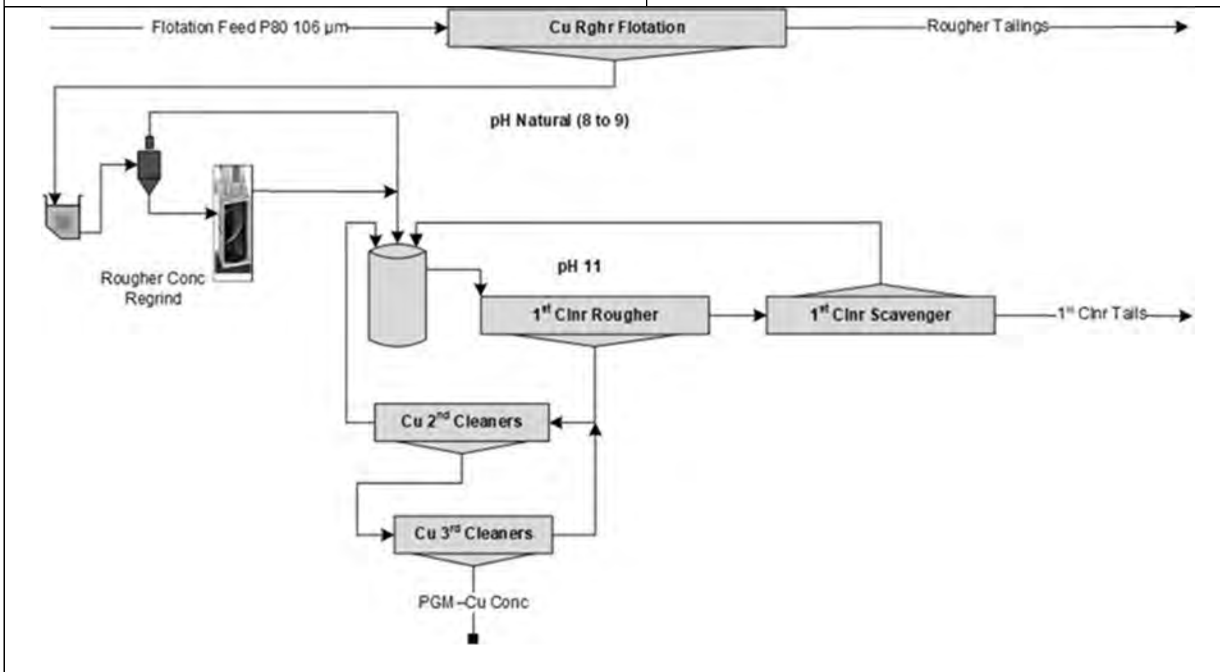
Baseline conditions were applied for all the locked cycle tests with a target flotation feed size P_{80} of 106 µm, natural pH in the roughers at 30% slurry density, collector addition including 35 g/t PAX, 35 g/t Aero 3501 and 22 g/t MIBC to the roughers. Rougher concentrate regrinding to a P_{80} 18 µm was followed by 30 minutes of aeration at pH 11. In the cleaner circuit, both PAX and Aero 3501 were applied at a rate of 7.5 g/t to the first half of 1st cleaner roughers, with an equivalent 7.5 g/t to the second half of 1st cleaner roughers, followed by 15 g/t to the 1st cleaner scavenger. An additional 5 g/t Aero 3501 was added to the 2nd cleaners, with MIBC addition at 10 g/t to 1st cleaner roughers.

For the more recent 2022 to 2023 locked cycle testing, a focus was placed on increasing the mass pull from 1st cleaners to transfer mineralization forward into the 2nd and 3rd cleaners for upgrading. Reagent addition rates for 2022 locked cycle testing were held constant. Aero 3501 addition to the 2nd cleaners was stage added over the 5-minute lab scale 2nd cleaner flotation cycle to promote PGM recovery.

Further optimization of reagent addition to the cleaner circuit would be expected at full scale. The transfer and recovery of mineral and metal from rougher concentrate to final concentrate is counter-current to the rejection of unwanted gangue. Striking the perfect balance between inter-stage mass pull, cleaner circuit density, optimal points for reagent addition, and potentially other PGM specific collector-promoters in the cleaner circuit is required to consistently achieve what would be considered as optimal PGM recovery.

Table 13.9: 2020 SGS Locked Cycle Test Schematic

Locked Cycle Test Details	
Six (6) iterations with 4 kg feed samples	Flotation Feed F80 106 µm grind
Reagents: PAX, Aero 3501, MIBC, Lime	24 minutes rougher flotation
Rougher conc regrind size P_{80} 15 to 20 µm	first to third cleaner pH 11



2012 Composite 3 Locked Cycle Testing (Test A Oct 2020)

Locked cycle testing on 2012 Composite 3 sample in Test A achieved recoveries of 94.5% Cu, 80.1% Pd and 66.8% Pt to a final Cu-PGM concentrate with a grade of 15.2% Cu, 19.8 g/t Pd and 3.97 g/t Pt at 2.3% mass pull to final concentrate (Table 13.8).

2012 Composite 3 Locked Cycle Testing (Test B Jan 2023)

Locked cycle testing on 2012 Composite 3 sample in test B achieved recoveries of 95.9% Cu, 83.0% Pd and 69.5% Pt to a final Cu-PGM concentrate with a grade of 9.4% Cu, 11.4 g/t Pd and 2.7 g/t Pt at 3.8% mass pull to final concentrate (Table 13.8).

2020 Main Zone Pilot Plant Composite Locked Cycle Testing (Dec 2022)

Locked cycle testing on the 2020 Main Zone composite sample achieved recoveries of 93.8% Cu, 87.2% Pd and 70.0% Pt to a final Cu-PGM concentrate with a grade of 8.4% Cu, 20 g/t Pd and 4 g/t Pt at 2.1% mass pull to final concentrate (Table 13.8).

<u>2020 W Horizon Pilot Plant Composite Locked Cycle Testing – Test A (Oct 2020)</u>

Locked cycle testing on the 2020 W-Horizon pilot plant composite sample in Test A achieved recoveries of 92.6% Cu, 87.9% Pd and 69.7% Pt to a final Cu-PGM concentrate with a grade of 12.2% Cu, 142.2 g/t Pd and 38.1 g/t Pt at 0.8% mass pull to final concentrate (Table 13.8).

<u>2020 W Horizon Pilot Plant Composite Locked Cycle Testing – Test B (Jan 2023)</u>

Locked cycle testing on the 2020 W-Horizon pilot plant composite sample in Test B achieved recoveries of 86.5% Cu, 88.6% Pd and 62.7% Pt to a final Cu-PGM concentrate with a grade of 10.9% Cu, 147.3 g/t Pd and 34.2 g/t Pt at 0.7% mass pull to final concentrate (Table 13.8).

<u>2022 Central Pit Composite CP-22-1 Locked Cycle Testing (Jan 2023)</u>

Locked cycle testing on the 2022 Central Pit CP-22-1 composite sample achieved recoveries of 91.3% Cu, 90.6% Pd and 75.6% Pt to a final Cu-PGM concentrate with a grade of 7.8% Cu, 33.4 g/t Pd and 8.6 g/t Pt at 1.5% mass pull to final concentrate (Table 13.8).

<u>2022 Main Zone Composite MZ-22-1 Locked Cycle Testing (Jan 2023)</u>

Locked cycle testing on the 2022 Main Zone MZ 22-1 composite sample achieved recoveries of 91.3% Cu, 87.7% Pd and 73.0% Pt to a final Cu-PGM concentrate with a grade of 17.8% Cu and 55.8 g/t Pd and 11.6 g/t Pt at 1.5% mass pull to final concentrate (Table 13.8).

<u>2022 Main Zone Composite MZ-22-2 Locked Cycle Testing (Jan 2023)</u>

Locked cycle testing on the 2022 Main Zone MZ-22-2 composite sample achieved recoveries of 94.5% Cu, 89.3% Pd and 68.1% Pt to a final Cu-PGM concentrate with a grade of 21.8% Cu and 53.9 g/t Pd and 12.0 g/t Pt at 1.0% mass pull to final concentrate (Table 13.8).

<u>2022 W Horizon Composite WH-22-1 Locked Cycle Testing (Jan 2023)</u>

Locked cycle testing on the 2022 W Horizon WH-22-1 composite sample achieved recoveries of 93.1% Cu, 90.3% Pd and 83.9% Pt to a final Cu-PGM concentrate with a grade of 5.7% Cu and 32.7 g/t Pd and 10.1 g/t Pt at 2.3% mass pull to final concentrate (Table 13.8).

<u>2022 W Horizon Composite WH-22-2 Locked Cycle Testing (Jan 2023)</u>

Locked cycle testing on the 2022 W Horizon WH-22-2 composite sample achieved recoveries of 93.6% Cu, 86.9% Pd and 73.0% Pt to a final Cu-PGM concentrate with a grade of 20.9% Cu and 55.8 g/t Pd and 15.2 g/t Pt at 0.8% mass pull to final concentrate (Table 13.8).

Locked cycle test results, including mass pull to rougher concentrate and final concentrate are well aligned with process design criteria. Decreased mass pull to final concentrate is expected with full-scale operation relative to bench-scale testing taking into account improved level control, improved air flow control, and froth wash water capabilities associated with the use of industrial Woodgrove SFR cells.

13.10 PGM-Cu Concentrate Quality

PGM-Cu concentrate grade is a variable, influenced by initial head grade, rougher concentrate regrind particle size, and cleaner circuit performance which ultimately define mass pull and metal recovery to a final Cu-PGM concentrate. Pilot Plant testwork during 2021 yielded sufficient concentrate to pursue multi-element analysis summarized in Table 13.10. Testwork confirmed an ability to achieve a range in Mg content in the final Cu-PGM concentrate from 2.0% to 6.3% Mg relative to an initial Mg head grade of 3.9-4.5% Mg. Aside from Mg, there are no other deleterious elements of concern in the final concentrate.

Table 13.10: Concentrate produced during 2021 Pilot Plant Testing

Element	Unit	South Pit	North Pit	Blended Historical Composite
		(W Horizon)	**(Main Zone)**	**(Composite 3)**
Cu	%	18.7	19.7	18.7
Ni	%	0.31	0.49	0.36
Zn	%	0.1	0.17	0.1
Fe	%	20.3	24.7	28.4
As	%	0.01	0.01	0.0
Au	g/t	17.6	3.3	2.7
Pt	g/t	43.5	7.6	4
Pd	g/t	171	39	19
Ag	g/t	50	68	42
S	%	17	24	26
F	%	0.07	0.07	0.04
Rh	g/t	2.4	0.58	0.22
Si	%	11.3	7	6.2
Mg	%	6.2	2.2	1.9
V	g/t	80	88	1000
Pb	%	0.02	0.02	0.01
Co	%	0.04	0.08	0.06
Al_2O_3	%	1.1	3.7	2.9
CaO	%	0.9	3.2	2.8
Mn	g/t	0.039	355	370
Cr	g/t	40	40	142
Ba	g/t	27	85	75
Se	g/t	174	87	70
Te	g/t	51	13	9
SG	g/ml	3.57	3.71	3.85

13.11 Geo-Metallurgical Modeling

The Geo-Metallurgical ("GeoMet") model has been updated with completion of the Phase 3 (2022-2023) metallurgical test program. An increase in the number of samples, a broader range in head grade, and an increased number of locked cycle tests, has allowed GeoMet recovery equations to be expressed as a function of head grade for respective metals summarized in Table 13.11.

Metallurgical studies completed by Gen Mining during 2020-2021 included rate kinetic bench-scale flotation testwork, which provided an indication of rougher, concentrate regrinding, and cleaner circuit performance. The initial 2020 Geo-Met model focused primarily on Cu and Pd recovery, with estimates for Pt and Au recovery being less quantitative due to the limited number of samples involved, and variable Au and Pt performance in the cleaner circuit.

Locked cycle testing in 2020 to confirm mineral and metal recovery to final concentrate was constrained to three samples requiring >25 kg per sample for the testwork, and involved 2012 Composite 3, the 2020 W Horizon pilot plant bulk sample, and the 2020 Main Zone pilot plant bulk sample.

Additional locked cycle testwork pursued by Gen Mining in 2022-2023 included mineralized intervals from 2022 drilling. The additional samples included Main Zone samples MZ-22-1 and MZ-22-2, W Horizon samples WH-22-1 and WH 22-2 and a Central Pit sample CP-22-1. Comparative head grade and respective metal recovery from locked cycle testing to final concentrate is summarized in Table 13.11.

The number of locked cycle tests associated with the updated 2023 GeoMet model has increased from three to 10, involving eight separate samples.

Table 13.11: Updated GeoMet Equations for metal recovery to final concentrate

Parameter	GeoMet Formula	Maximum Value
%Rec Cu to Final Conc	$= 97.55 \times (\%\text{ Cu head grade})^{0.0239}$	94% Rec Cu
%Rec Pd to Final Conc	$= 89.14 \times (\text{g/t Pd head grade})^{0.0203}$	90% Rec Pd
%Rec Pt to Final Conc	$= 104.51 \times (\text{g/t Pt head grade})^{0.2034}$	84% Rec Pt
%Rec Au to Final Conc	$= 116.51 \times (\text{g/t Au head grade})^{0.1822}$	86% Rec Au
%Rec Ag to Final Conc	$= 50.82 \times (\text{g/t Ag head grade})^{0.6090}$	68% Rec Ag
%Mass Pull to Final Conc	$= 0.625 \times e^{(2.899 \times \%\text{Cu head grade})}$	2.0% Mass Pull

An estimate of mass pull to final concentrate was determined as a function of Cu head grade considering locked cycle flotation test final concentrate grades, relative to modelled metal recovery, and the implied mass pull associated with those concentrate grades. The equation for mass pull is outlined in Table 13.11 and yields a calculated Cu concentrate grade that is close to observed locked cycle test results.

The recovery of metal (Cu, Pd, Pt, Au, Ag) to final concentrate is defined by a recovery equation for those respective values (Table 13.12). Mass pull to final concentrate is defined by a separate equation, as a function of copper head grade (Table 13.12). The calculated concentrate grade is an outcome of the two separate calculations determined as:

Concentrate Grade = metal head grade x metal recovery/% mass pull

For Cu at 0.20% Cu, Pd at 0.60 g/t Pd and Pt at 0.25 g/t Pt:

Cu Recovery to Final Conc = 97.55 x ($\mathbf{0.20}^{0.0239}$) = 93.9% Rec Cu (maximum 94%)

Pd Recovery to Final Conc = 89.14 x ($\mathbf{0.60}^{0.0203}$) = 88.2% Rec Pd (maximum 90%)

Pt Recovery to Final Conc = 89.14 x ($\mathbf{0.25}^{0.0203}$) = 78.8% Rec Pt (maximum 84%)

Mass Pull to Final Conc = 0.625 x exp (2.899 x **0.20**) = 1.12%

Concentration Ratio = 100 / 1.12 = 89

Concentrate Cu Grade = **0.20** x 93.9 / 1.12 = 16.82 % Cu (to a maximum 28% Cu)

Concentrate Pd Grade = **0.60** x 88.2 / 1.12 = 47.4 g/t Pd

Concentrate Pt Grade = **0.25** x 78.8 / 1.12 = 17.6 g/t Pt

Updated GeoMet equations to final concentrate are summarized in Table 13.12, with updated GeoMet recovery curves versus locked cycle test results for respective metal included in Figure 13.9 to Figure 13.13.

The updated GeoMet model considers a best fit recovery equation for each metal based on locked cycle test data to final concentrate. In conjunction with statistical analysis, consideration was given to whether mass pull to respective inter-stage concentrates for a given test was above or below target, and whether the flotation feed grind size relative to a target P_{80} 106 µm was close to, or coarser than target.

Concentrate regrind particle size for all cleaner circuit testwork was checked with a bench-scale Malvern Panalytical particle size monitor and determined as within a range of P_{80} 12 to 20 µm and not considered as a constraint in cleaner circuit performance.

In comparison to the modeled recovery to final concentrate in Table 13.12, metal recovery curves to rougher concentrate (Table 13.12) were defined to evaluate cleaner circuit efficiency summarized in Table 13.13.

Table 13.12: Updated GeoMet Equations for recovery to Rougher Concentrate

Parameter	GeoMet Formula	Maximum Value
%Rec Cu to Rougher Conc	$= 100.4 \times (\% \text{ Cu head grade})^{0.0335}$	97% Rec Cu
%Rec Pd to Rougher Conc	$= 91.93 \times (\text{g/t Pd head grade})^{0.0194}$	95% Rec Pd
%Rec Pt to Rougher Conc	$= 93.88 \times (\text{g/t Pt head grade})^{0.0344}$	94% Rec Pt
%Rec Au to Rougher Conc	$= 126.01 \times (\text{g/t Au head grade})^{0.1948}$	93% Rec Au
%Rec Ag to Rougher Conc	$= 55.29 \times (\text{g/t Ag head grade})^{0.5935}$	77% Rec Ag

Table 13.13: Cleaner Circuit Efficiency

Component	Cleaner Circuit Efficiency
Cu	98.7%
Pd	96.9%
Pt	84.8%
Au	95.6%
Ag	92.4%

For rougher circuit performance, data selected for analysis utilized metal recoveries from rate kinetic testwork between a range of 12-15% mass pull to rougher concentrate to avoid overstating metal recovery to a rougher concentrate at higher mass pull.

Based on the updated GeoMet metal, the calculated recovery to final concentrate versus the calculated recovery of metal to rougher concentrate, for the same head grade, suggests a cleaner circuit efficiency for the present flotation circuit configuration and design process criteria of 98.7% for Cu, 96.9% for Pd, 84.8% for Pt, 95.6% for Au, and 92.4% for Ag.

The cleaner circuit efficiency is an indicator of mineral or metal transfer from rougher concentrate to final concentrate, which occurs simultaneously with the counter-current rejection of gangue materials including silicate and pyrite to 1st cleaner scavenger tailings.

Recovery of values to rougher concentrate are excellent for all metals, with lower Ag recovery limited by low Ag head grade relative to a minimum analytical detection limit of 0.5 g/t Ag.

Cleaner circuit efficiency highlights the lower transfer rate of Pt to final concentrate with sperrylite ($PtAs_2$) continuing to exhibit slower flotation rate kinetics in 1st to 3rd Cleaners.

The PGM Scavenger circuit has been excluded from this analysis and is considered as a potential future sustaining capital project requiring additional evaluation.

13.11.1 GeoMet Model Recovery versus Locked Cycle Testing

The updated 2023 GeoMet model for recovery is plotted in comparison to the 2020 GeoMet model for reference in Figure 13.9 to Figure 13.13. In comparison to predicted metal recovery to final concentrate, locked cycle test results are included in the chart and reconcile well with the modeled curve.

The following maximum were applied to each element:

- The maximum Cu recovery to final concentrate is constrained to a recovery of 94% Cu, which corresponds to Cu head grades >0.21%.
- The maximum Pd recovery to final concentrate is constrained to 95% which corresponds to Pd head grades >1.60 g/t.
- The maximum Pt recovery to final concentrate is constrained to 84% Rec Pt.
- The maximum Au recovery to final concentrate is constrained to 86% which corresponds to Au head grades >0.19 g/t.
- The maximum Ag recovery to final concentrate is constrained to 68% which corresponds to Ag head grades >1.60 g/t.

Note that the previous 2020 GeoMet model considered Pt and Au recovery as a function of Pd head grade and Cu and Ag recovery at a fixed 93% and 71.5% respectively. The minimum analytical detection level for Ag of 0.50 g/t is a constraint that was not considered previously. For a head grade of 2.0 g/t Ag, 25% of contained value can't be determined as recovered or lost due to analytical detection limits.

The previous 2020 GeoMet model also included the incremental gain associated with the PGM Scavenger circuit, while the 2023 GeoMet model excludes the PGM scavenger circuit and is associated with the design flowsheet including rougher flotation, concentrate regrinding and 1st to 3rd Cleaners.

A slight overall improvement in recovery for the design flowsheet was achieved in 2022 metallurgical testwork with a focus on pursuing design mass pull in 1st to 3rd Cleaners, and most importantly pursuing a target 12-15% mass pull to rougher concentrate.

Figure 13.9: GeoMet Model for Copper Recovery to Final Concentrate

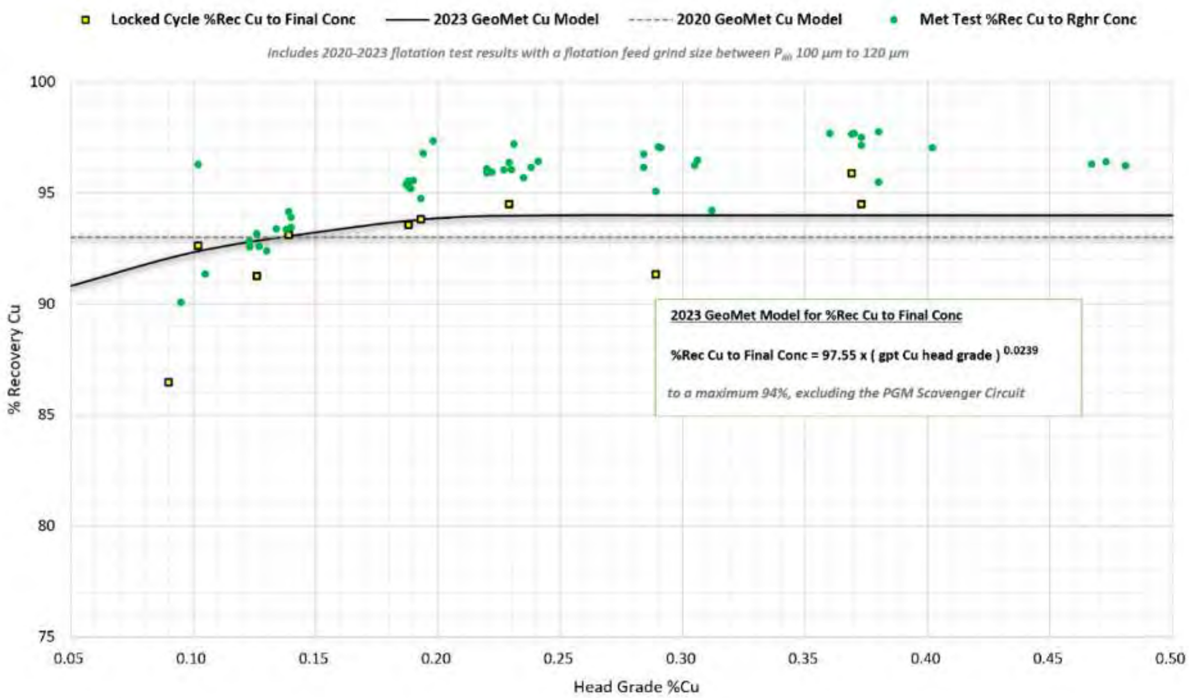


Figure 13.10: GeoMet Model for Palladium Recovery to Final Concentrate

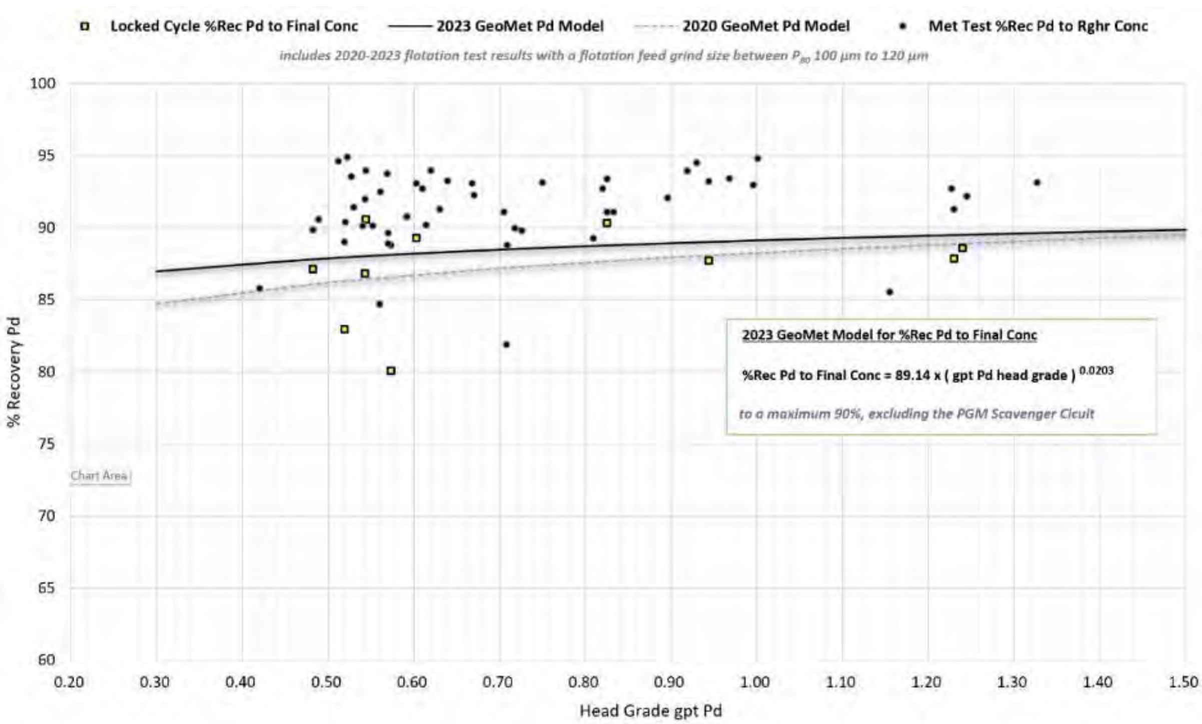


Figure 13.11: GeoMet Model for Platinum Recovery to Final Concentrate

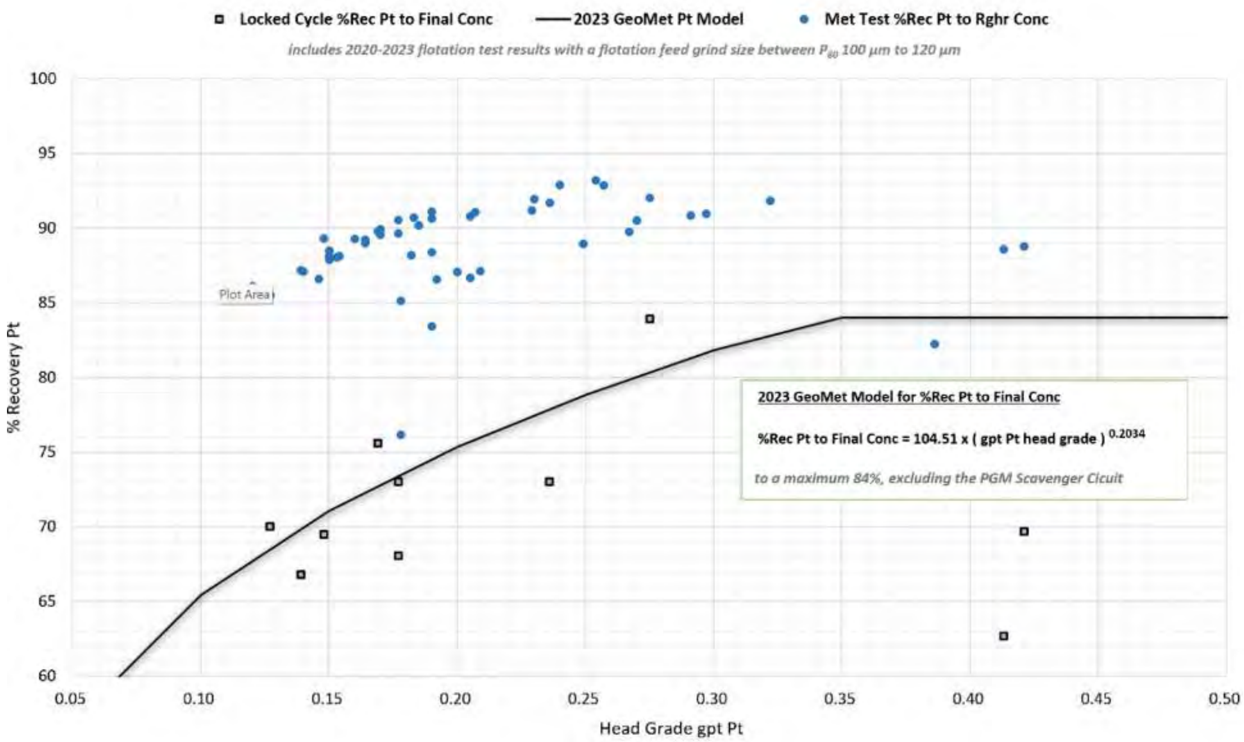


Figure 13.12: GeoMet Model for Gold Recovery to Final Concentrate

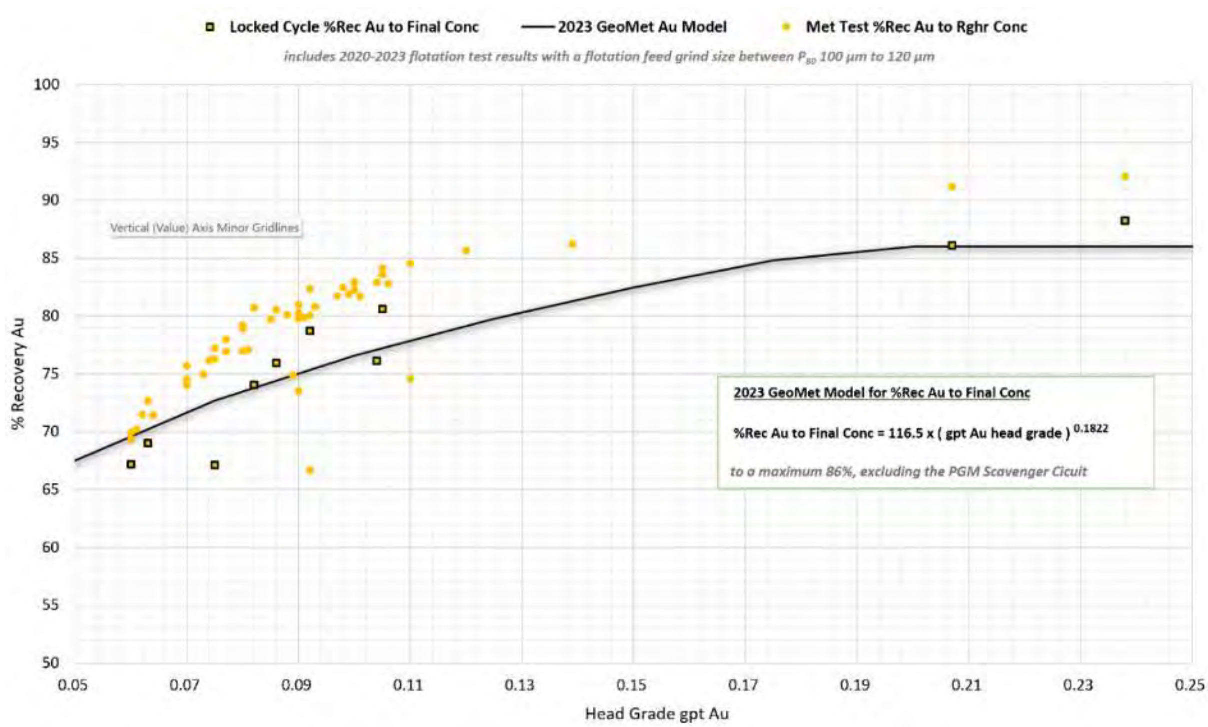
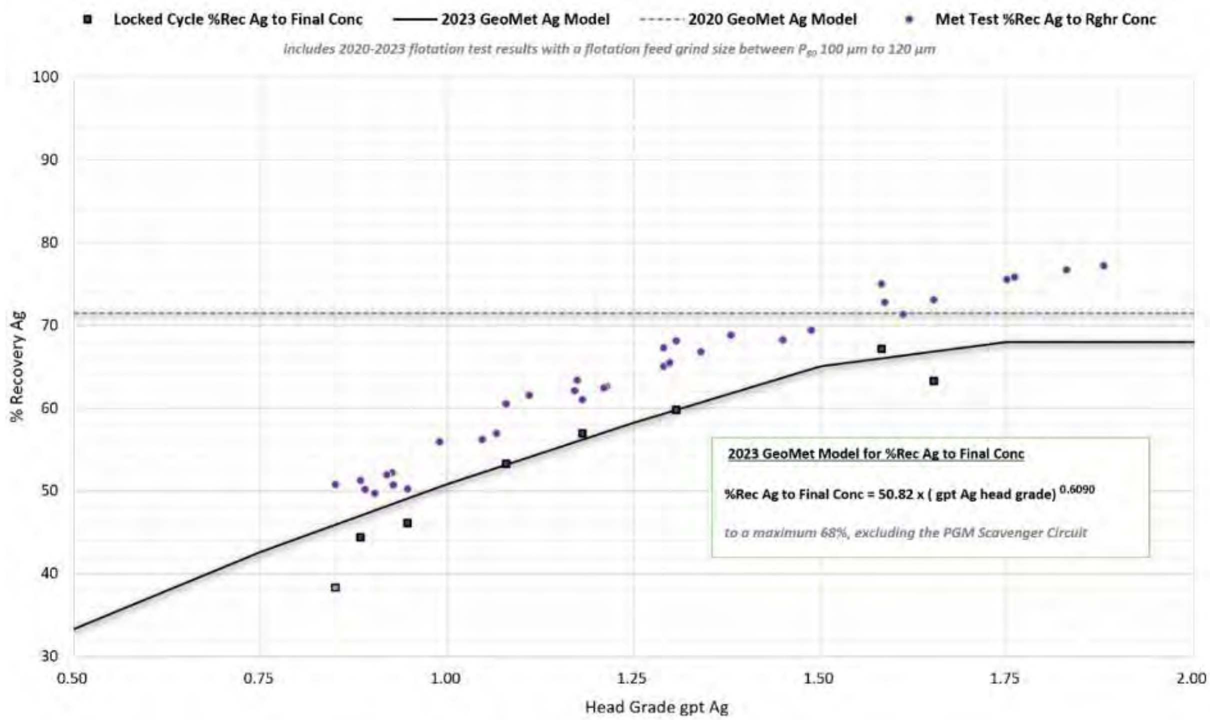


Figure 13.13: GeoMet Model for Silver Recovery to Final Concentrate



13.12 DFR (Direct Flotation Reactor) Cells and Flotation Circuit Selectivity

In conjunction with Phase 1 and Phase 2 bench-scale testing during 2020, Gen Mining conducted a mini-pilot plant to evaluate the applicability of Woodgrove DFRs relative to conventional flotation cells on both the roughers and first cleaner flotation.

The Project mineralogy includes chalcopyrite as the primary copper sulphide mineral, along with variable iron sulphide content including pyrite and pyrrhotite. A nominal 30-40% PGM are associated with recoverable copper with the remainder either free or associated with silicate prior to rougher concentrate regrinding. The relatively slow PGM flotation rate kinetics, compared to the performance of Cu mineralization, requires a relatively aggressive mass pull to rougher concentrate with the potential for froth collapse and cell to cell carry-over with conventional flotation.

The Woodgrove DFR cell design is intended to provide a distinct operating advantage since relative to conventional flotation cells the technology:

- Yields a mineral rich gas/slurry phase as opposed to a froth
- Incorporates wash water in the disengagement zone of each cell to promote silicate rejection
- Can provide increased concentration ratio (lower mass pull) with maintained metal recovery, which improves the consistency of final PGM-Cu concentrate grade.

- Requires a smaller footprint and lower power cost than conventional technology
- Minimizes the requirement for transfer pumps between stages since concentrate from respective DFR cells is moved pneumatically

During detailed engineering, additional feedback, and references from other users of the relatively new DFR cell technology was obtained to validate equipment selection. As part of risk mitigation, Gen Mining opted to pursue conventional tank cells for flotation and will apply the industrially proven Woodgrove SFR cells for the cleaner circuit.

13.13 Dewatering and Concentrate Filtration

The Project and optimized process flowsheet Figure 13.1 involves the dewatering of the final Cu-PGM concentrate in a concentrate thickener followed by pressure filtration to yield a nominal 12% w/w moisture concentrate for transport. Flotation tailings are dewatered in a tailings thickener prior to impoundment in the Cell 1 of the TSF.

Thickener overflow from both applications is expected to be clear and low in total suspended solids for recycle within the operation as process water.

In conjunction with Phase 2 bench-scale testing, Gen Mining involved SNF-Canada and Outotec-Canada in the completion of flocculation, dewatering and pressure filtration testing on samples generated from testwork at SGS Lakefield. The results indicated that concentrate and flotation tailings can be successfully treated with flocculant SNF AN934-SH.

Concentrate dewatering will require the addition of 35 g/t flocculant (relative to concentrate tonnage) to achieve acceptable overflow clarity (24 NTU or 29 ppm solids) with 55-60% solids underflow density. Concentrate thickener sizing from dynamic simulation tests was confirmed as 0.15 mt/m^2-h. Concentrate pressure filtration test specifics defined a filtration rate of 159 kg/m^2-h, yielding a filter cake of approximately 12% moisture. Project design and the FS considers the implementation of a larger vertically stacked pressure filter to minimize technical risk and support decreased transportation costs, while achieving target moisture levels for transport.

Tailings dewatering required a blend of coagulant and flocculant involving the addition of coagulant, 7 g/t SNF DB45-SH, with 13 g/t flocculant (relative to final tailings tonnage) to achieve acceptable overflow clarity (100 NTU or 83 ppm solids) with 55-60% underflow density. Tailings thickener sizing from dynamic simulation testing was confirmed as 0.60 mt/m^2-h.

13.14 Technical Risk and Future Testing

Metallurgical testwork completed by Gen Mining between 2020-2023 included an evaluation of deposit material hardness and competency; the influence of flotation feed grind size; the influence of independent

Cu, Pd, Pt, Au and Ag head grades; flotation rate kinetics; rougher concentrate regrind size; sulfide deportment; mineralogical analysis, reagent suite and cleaner circuit performance; thickening and filtration requirements, material environmental characterization studies, and water balance optimization. The design flowsheet has been validated and confirmed as well suited to the expected variability in Pd and Cu feed grades and iron sulphide content with a focus on maximizing PGM and Cu recovery.

Follow-up Phase 2 metallurgical testing was completed on three bulk samples including: (i) 2012 Composite 3, (ii) 2020 Main Zone Composite, and (iii) 2020 W Horizon Composite. Testwork included cleaner circuit optimization studies, locked cycle testing, an evaluation DFR performance, flocculant trials for dewatering, thickening and pressure filtration studies, concentrate regrind specific work index testing, and semi-continuous pilot plant trials for the validation of bench-scale and locked cycle testing.

Additional Phase 3 metallurgical testing included evaluating 25 additional samples from within the deposit to evaluate the variability in expected ball mill work index. Data is summarized in Table 13.5 and validates the design parameter of 17.45 KWh/t is appropriate. Phase 3 testing during 2022 also included eight additional locked cycle tests to improve the estimation of metal recovery to final concentrate. GeoMet formulas have been updated for respective metals relative to head grade. Details are outlined in Subsections 13.9 and 13.11.

During 2022, Gen Mining design optimization efforts included the consideration of a larger SAG mill and ball mill that were readily available as unused, second-hand equipment. Relative to an engineered, fit for purpose installation, the larger, higher horsepower grinding mills supported the elimination of the pebble crusher in the grinding circuit and was included as a positive design change. A peer review study pursued by Gen Mining with Orway Mineral Consulting (comminution engineering design specialists), confirmed the applicability of the two larger 18 MW grinding mills, and the potential to eliminate pebble crushing to a production rate of +15% relative to design. The availability of these unused mills also offered inherent advantages with decreased delivery time, price protection, and warranty coverage from the OEM provider with slight refurbishing prior to installation. Equipment specifications for the larger Hycroft grinding mills is included in Table 17.1.

Further consideration was given to the applicability of DFR cells within the flowsheet and is summarized in Subsection 13.12 with a switch to conventional tank cells in the roughers, and Woodgrove SFR cells in the cleaner circuit.

The scope of 2020-2023 metallurgical testing is thorough and supports the requirements for completion of a FS, basic engineering, and detailed engineering. Process QP has reviewed the metallurgical test results and the composite samples that were selected for metallurgical testing and considers it suitable for this level of study and support the process design in this report.

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource estimate presented herein has been prepared following the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101F1 and in conformity with generally accepted "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Mineral Resources have been classified in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition and Guidelines" as adopted by CIM Council on May 10, 2014:

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applied to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into Mineral Reserve. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

The Authors are not aware of any known permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate. All Mineral Resource estimation work reported herein was carried out by Mauro Bassotti, P.Geo. from Gen Mining, and reviewed by Fred Brown, P.Geo. and Eugene Puritch, P.Eng., FEC, CET, independent Qualified Persons as defined by National Instrument 43-101.

Wireframe modeling utilized Seequent Leapfrog Geo™ software. Mineral Resource estimation was carried out using Datamine Studio RM software. Variography was carried out using Snowden Supervisor™.

The effective date of the Mineral Resource estimates for Marathon is December 31, 2022 and for Geordie and Sally is June 30, 2020. The Geordie and Sally Mineral Resource estimates have remained unchanged and are summarized in Subsection 14.3.

14.2 Marathon Mineral Resource Estimate

14.2.1 Data Supplied

Sample data was provided in the form of Excel format files. Gen Mining supplied the database which contained 1,217 unique collar records (Table 14.1). Of these, 209 records fell outside the block model limits or had no reported assay data. Drill hole and surface channel sample records consist of collar, survey, lithology, bulk density and assay data. Assay data fields consist of the drill hole ID, down-hole interval distances, sample number, and g/t Ag, g/t Au, Cu %, g/t Pd, g/t Pt assay grades. All data are in metric units. Collar coordinates were provided in the NAD83 UTM Zone 16 coordinate system. The drilling covers an area of approximately 470 ha (Table 14.1)

Table 14.1: Drill Hole Database Summary

	Drill Holes	Channel Samples	Total
Count	1,107	110	**1,217**
Total Metres	191,673	9,851.27	**201,524**
Minimum Length (m)	0.005	0.07	**0.0375**
Maximum Length (m)	269.3	103.9	**186.6**
Average Length (m)	2.24	1.80	**2.02**

Gen Mining supplied the database which contained a total of 64,438 assays for Ag, Au, Cu, Pd, and Pt. For domain modeling a calculated NSR field was added to the assay table as follows:

NSR = (Cu % x 88.72) + (Ag g/t x 0.47) + (Au g/t x 44.69) + (Pd g/t x 58.63) + (Pt g/t x 28.54) -3.37

Industry standard validation checks were carried out on the databases and minor corrections were made where necessary. The Mineral Resource database was validated by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance, values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, missing interval and coordinate fields, and down-hole survey information beyond normal expected deviation.

No significant errors were noted with the provided databases. The drill hole database supplied is suitable for Mineral Resource estimation. The drill hole data were imported into Datamine Studio RM.

14.2.2 Economic Considerations

Based on knowledge of similar projects, review of available historical data, and consideration of potential mining scenarios for the Marathon Deposit, the economic parameters listed in Table 14.2 were deemed appropriate for the Mineral Resource estimate. Metal prices are based on the approximate three-year trailing average metal prices as of December 31, 2022. Process recovery factors are based on information from previous Technical Reports on the Property. Mining and processing costs are based on similar projects.

Table 14.2: Economic Parameters

Input Parameters					
Exchange Rate				C$:US$	1.30
Diesel Fuel Price Delivered				$/litre	1.17
Electricity Cost				$/kWh	0.07
Processing Inputs					
Nominal Milling Rate				Mt/yr	10.6
Concentrate Grade				% Cu	Variable (Refer to Section 13)
Concentrate Treatment Charge				US$/dmt	79
Concentrate Transport & Logistics				US$/dmt	145.16
Metal	**Copper**	**Palladium**	**Platinum**	**Gold**	**Silver**
Metal Prices (US$)	$3.50/lb	$1,800/oz	$1000/oz	$1,600/oz	$20/oz
Refining Charges (US$)	$0.079/lb	$24.50/oz	$24.50/oz	$5.00/oz	$0.50/oz
Payable Rates (%)	96.5	95	93	93.5	93.5
Concentrator Recovery (%)	Variable (Refer to recovery curves in Section 13)				
Mineralized Material -Based Costs					
Average Incremental Ore Mining Cost				$/t milled	0.20
Processing Cost (incl. power)				$/t milled	9.35
General and Administration				$/t milled	1.66
Rehabilitation and Closure				$/t milled	0.56
Sustaining Capital				$/t milled	2.65
Total Mineralized Material-Based Cost				**$/t milled**	**14.42**
Mining Inputs					
Mining Dilution				%	0%
Mining Loss				%	0%
Total Mining Reference Cost				$/t mined	2.8
Incr. Bench Cost ($ /10 m bench)				$/t mined	0.05
Overall Slope Angle in Fresh Rock				degrees	50

14.2.3 Geology Model

Based on fault interpretations developed by Gen Mining, the Marathon Deposit area was divided into 10 fault blocks (Figure 14.1). Within each fault block, the metabasalt, gabbro, troctolite, melagabbro, dyke and basement lithologies were modeled in Leapfrog based on drill hole lithological logging (Figure 14.2). The resulting lithological units were used for modeling bulk density.

Figure 14.1: Fault Blocks

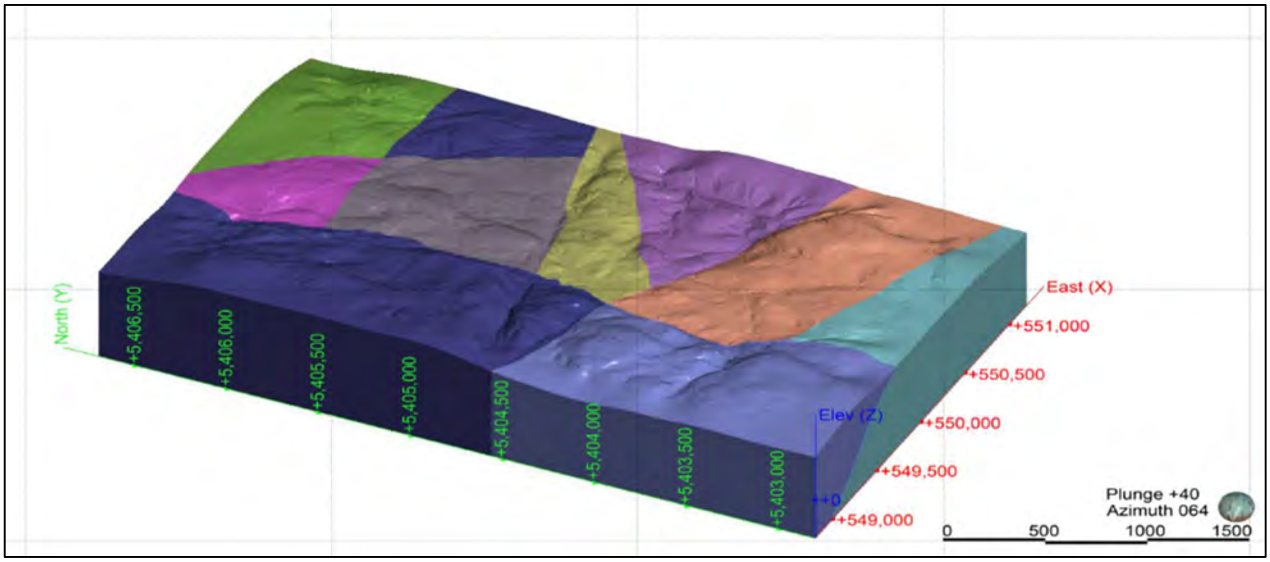
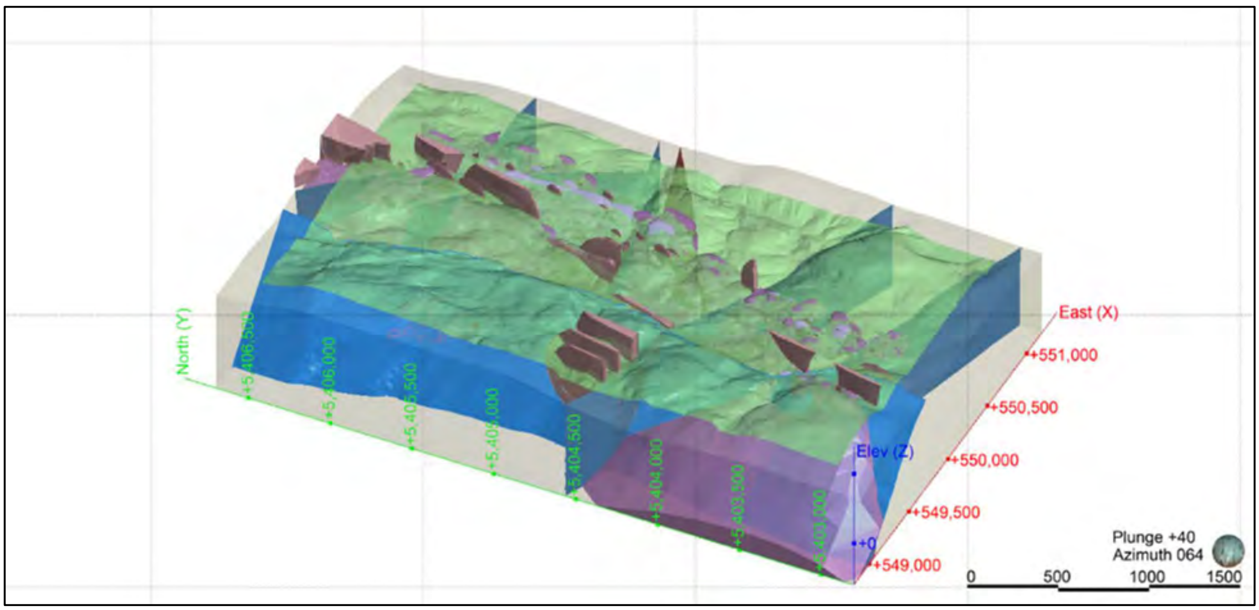


Figure 14.2: Lithology Model



14.2.4 Mineralization Domains

The updated Mineral Resource estimate is based on 19 mineralization domains with a total volume on the order of 96 Mm^3 (Figure 14.3). The mineralization domains have been based on zones developed by Dr. David Good, P.Geo., previously Vice President Exploration for Stillwater Canada. Mineralization domains are further broadly grouped into two areas: the northern domains where mineralization is dominated by paleo-topographic controls, and the remaining southern domains. The domains are further split into the identified fault blocks. Of the 19 domains modeled, the North Main (DOM 90) and Walford Zone (DOM 80) include approximately 66% of the total Mineral Resource by volume (Figure 14.3 and Figure 14.4).

The mineralization domains were based on NSR drill hole assay values equal to or greater than $13/t within the identified zones and with observed continuity downhole along strike and down dip. Drill hole intercepts were only used to define the mineralization domains and surface channel sample intervals were excluded from the process. The selected intervals include lower grade intervals or un-sampled intervals, where necessary, and were used to maintain continuity between drill holes. Three-dimensional wireframes linking drill hole sections were subsequently constructed using the Leapfrog Radial Basis Vein Function with hanging wall and footwall surfaces snapped directly to the selected drill hole intercepts within each fault block. The domain wireframes were used to back-tag the block model, as well as the assay, bulk density and composite tables with unique rock codes (Table 14.3).

Figure 14.3: Mineralization Domains

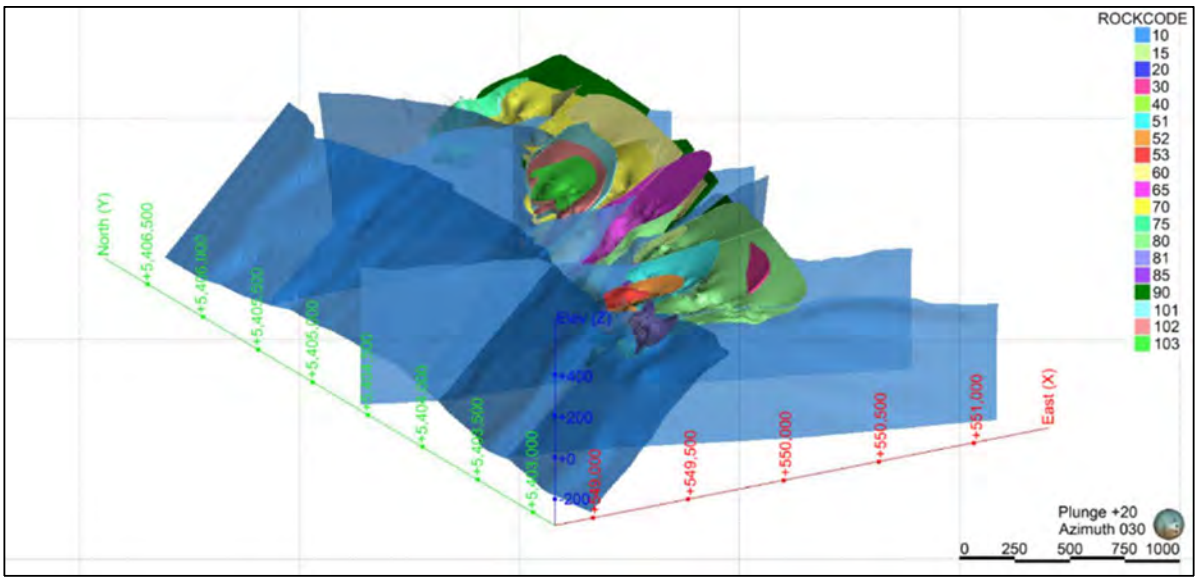


Figure 14.4: North Main and Walford Mineralization Domains

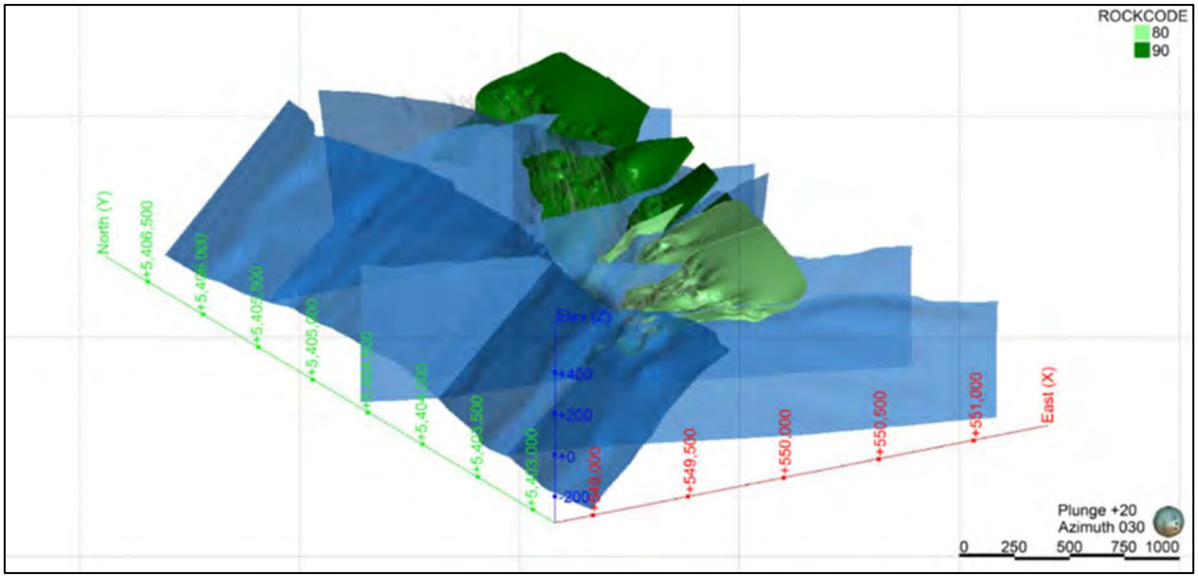


Table 14.3: Domain Rock Codes

Domain	Rock Code	Volume (k m³)
DOM 10	10	345
DOM 15	15	93
DOM 20	20	7,998
DOM 30	30	3,808
DOM 40	40	3,250
DOM 51	51	1,097
DOM 52	52	644
DOM 53	53	320
DOM 60	60	6,246
DOM 65	65	1,183
DOM 70	70	2,310
DOM 75	75	704
DOM 80	80	11,712
DOM 81	81	1,121
DOM 85	85	79
DOM 90	90	52,551
DOM 101	101	1,668
DOM 102	102	611
DOM 103	103	200
Total		**95,940**

14.2.5 Exploratory Data Analysis

Summary statistics for the domain coded assay data (drill hole and trench channel samples) are listed in Table 14.5.

A strong overall correlation between Pd and Pt as well as with Au, Pd and Pt was noted. A strong correlation between Cu with Pd and Pt was noted in the northern area (Table 14.4).

Table 14.4: Assay Correlation Table (Pearson Correlation Coefficient)

Total	Ag	Au	Cu	Pd	Pt
Ag	1	0.18	0.32	0.16	0.11
Au	0.18	1	0.32	0.5	0.37
Cu	0.41	0.32	1	0.18	0.18
Pd	0.16	0.5	0.32	1	0.85
Pt	0.16	0.37	0.18	0.85	1
WZONE 80	**Ag**	**Au**	**Cu**	**Pd**	**Pt**
Ag	1	0.1	0.21	0.08	0.06
Au	0.1	1	0.31	0.55	0.4
Cu	0.21	0.31	1	0.23	0.13
Pd	0.08	0.55	0.23	1	0.87
Pt	0.06	0.4	0.13	0.87	1
NMAIN 90	**Ag**	**Au**	**Cu**	**Pd**	**Pt**
Ag	1	0.37	0.62	0.44	0.36
Au	0.37	1	0.51	0.62	0.50
Cu	0.62	0.51	1	0.68	0.52
Pd	0.44	0.63	0.68	1	0.71
Pt	0.36	0.50	0.52	0.71	1

Gen Mining supplied the database that contained 9,475 bulk density measurements with values ranging from 1.075 to 4.307 t/m^3 (Table 14.6). The average bulk density measured is 3.01 t/m^3. Bulk density measurements were backtagged to the lithology model.

Table 14.5: Assay Summary Statistics

		Au g/t					Cu %					Pd g/t			
Min	Max	Mean	Std Dev	CoV	Min	Max	Mean	Std Dev	CoV	Min	Max	Mean	Std Dev	CoV	Min
0.2	6	0.07	0.121	1.73	0.003	0.7	0.118	0.101	0.86	0.009	0.5	0.39	0.413	1.06	0.015
0.1	5	0.049	0.072	1.47	0.001	0.4	0.042	0.113	2.69	0.004	0.7	0.512	0.762	1.49	0.003
0.1	44	0.057	0.083	1.46	0.001	1.2	0.25	0.215	0.86	0.001	3.3	0.461	0.625	1.36	0.001
0.1	19	0.085	0.266	3.13	0.001	8.1	0.113	0.123	1.09	0.000	1.0	0.572	1.283	2.24	0.001
0.1	33	0.055	0.056	1.02	0.001	0.5	0.123	0.105	0.85	0.001	0.9	0.27	0.349	1.29	0.001
0.1	24	0.055	0.098	1.78	0.001	0.8	0.068	0.073	1.07	0.002	0.5	0.328	0.923	2.81	0.005
0.1	25	0.051	0.048	0.94	0.002	0.5	0.108	0.065	0.6	0.003	0.3	0.225	0.261	1.16	0.005
0.1	5.6	0.061	0.06	0.98	0.003	0.3	0.105	0.085	0.81	0.006	0.3	0.282	0.350	1.24	0.005
0.1	38	0.056	0.068	1.21	0.001	0.8	0.13	0.138	1.06	0.002	1.4	0.436	0.594	1.36	0.001
0.1	9.1	0.073	0.099	1.36	0.003	0.7	0.121	0.133	1.10	0.005	1.0	0.356	0.684	1.92	0.005
0.1	73	0.044	0.05	1.14	0.001	0.5	0.089	0.077	0.87	0.003	0.5	0.279	0.266	0.95	0.004
0.1	29.3	0.068	0.073	1.07	0.003	0.4	0.191	0.169	0.88	0.004	0.8	0.578	0.686	1.19	0.005
0.1	68	0.078	0.196	2.51	0.001	7.2	0.103	0.123	1.19	0.000	1.5	0.800	2.752	3.44	0.001
0.1	12.9	0.057	0.08	1.40	0.001	0.8	0.167	0.169	1.01	0.003	1.0	0.426	0.82	1.92	0.001
0.1	3.5	0.032	0.029	0.91	0.002	0.1	0.066	0.087	1.32	0.004	0.4	0.251	0.173	0.69	0.031
0.1	17	0.07	0.096	1.37	0.001	2.6	0.244	0.2	0.82	0.000	2.4	0.607	0.76	1.25	0.001
0.1	9	0.047	0.045	0.96	0.001	0.3	0.074	0.073	0.99	0.001	0.3	0.383	0.388	1.01	0.001
0.1	6	0.072	0.126	1.75	0.001	0.9	0.091	0.064	0.70	0.003	0.3	0.312	0.336	1.08	0.001
0.1	8	0.051	0.051	1.00	0.001	0.2	0.108	0.112	1.04	0.001	0.7	0.16	0.266	1.66	0.001

Table 14.6: Bulk Density Summary Statistics (t/m3)

Lithology	Count	Average	Minimum	Maximum	Std Dev
Basement	4,349	3.065	2.9	3.4	0.149
Gabbro	2,694	2.974	1.175	4.37	0.178
Melagabbro	94	3.15	2.645	3.8	0.259
Metabasalt	1,099	2.876	2.074	4.37	0.205
Troctolite	25	3.102	2.9	3.4	0.149
Other	1,214	2.975	1.1	4	0.247
Total	9,475	3.006	1.057	4.37	0.236

14.2.6 Compositing

Constrained assay sample lengths range from 0.05 m to 269.3 m with an average sample length of 2.21 m and a sample length mode of 2.00 m. A total of 98% of the assay samples have a length of 2.00 m.

All constrained assay samples were therefore composited to the dominant sample length of 2.00 m. Length-weighted composites were calculated for all metals within the defined mineralization domains. Missing sample intervals in the data were assigned a nominal background grade of 0.001 g/t or 0.001%. The compositing process started at the first point of intersection between the drill hole and the domain intersected and halted upon exit from the domain wireframe. Channel samples that were intersected by the domain wireframes were also included in the compositing process. Residual composites that were less than half of the compositing length were discarded so as to not introduce a short sample bias into the estimation process. The wireframes that represent the interpreted mineralization domains were also used to back-tag a rock code identifier into the composite workspace. The composite data were visually validated against the domain wireframes and then exported for analysis and estimation. A summary of uncapped composite statistics is tabulated in Table 14.7.

14.2.7 Treatment of Extreme Values

Grade capping analyses were conducted on the domain-coded and composited grade sample data in order to evaluate the potential influence of extreme values during estimation. Capping thresholds were determined by the decomposition of the domain composite log-probability distributions (Figure 14.5). Where possible, the observed correlations between elements were also maintained when determining appropriate capping levels. Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Composites are capped to the defined threshold prior to estimation (Table 14.8).

Table 14.7: Composite Summary Statistics

		Au g/t					Cu %					Pd g/t			
Min	Max	Mean	Std Dev	CoV	Min	Max	Mean	Std Dev	CoV	Min	Max	Mean	Std Dev	CoV	Min
.001	6	0.037	0.079	2.14	0.001	0.7	0.07	0.092	1.31	0.001	0.4	0.224	0.351	1.57	0.001
0.1	5	0.053	0.075	1.42	0.001	0.4	0.044	0.119	2.7	0.004	0.7	0.507	0.772	1.52	0.003
.001	44	0.052	0.08	1.54	0.001	1.2	0.226	0.203	0.9	0.001	3.3	0.417	0.573	1.37	0.001
.001	19	0.082	0.272	3.32	0.001	8.1	0.108	0.124	1.15	0	1	0.537	1.205	2.24	0.001
.001	33	0.05	0.056	1.12	0.001	0.5	0.115	0.107	0.93	0.001	0.9	0.245	0.346	1.41	0.001
.001	24	0.043	0.083	1.93	0.001	0.8	0.055	0.07	1.27	0.001	0.4	0.276	0.895	3.24	0.001
.001	25	0.042	0.048	1.14	0.001	0.5	0.087	0.069	0.79	0.001	0.3	0.178	0.233	1.31	0.001
.001	5.6	0.062	0.064	1.03	0.001	0.3	0.104	0.09	0.87	0.001	0.3	0.281	0.381	1.36	0.001
.001	38	0.047	0.059	1.26	0.001	0.6	0.112	0.126	1.13	0.001	0.9	0.363	0.513	1.41	0.001
.001	9.1	0.062	0.095	1.53	0.001	0.7	0.109	0.134	1.23	0.001	1	0.27	0.53	1.96	0.001
.001	73	0.036	0.046	1.28	0.001	0.5	0.076	0.074	0.97	0.001	0.5	0.229	0.251	1.1	0.001
.001	29.3	0.068	0.071	1.04	0.001	0.4	0.192	0.169	0.88	0.004	0.8	0.579	0.667	1.15	0.001
.001	68	0.077	0.195	2.53	0.001	7.2	0.099	0.116	1.17	0	1.2	0.785	2.767	3.52	0.001
.001	12.9	0.054	0.074	1.37	0.001	0.8	0.149	0.169	1.13	0.001	1	0.406	0.772	1.9	0.001
.001	3.5	0.029	0.029	1	0.001	0.1	0.068	0.096	1.41	0.001	0.4	0.242	0.192	0.79	0.001
.001	17	0.067	0.091	1.36	0.001	2.6	0.228	0.194	0.85	0	2.2	0.581	0.731	1.26	0.001
.001	9	0.037	0.044	1.19	0.001	0.3	0.058	0.068	1.17	0.001	0.3	0.305	0.369	1.21	0.001
.001	6	0.057	0.095	1.67	0.001	0.9	0.078	0.063	0.81	0.001	0.3	0.281	0.329	1.17	0.001
.001	8	0.032	0.045	1.41	0.001	0.2	0.069	0.106	1.54	0.001	0.7	0.089	0.172	1.93	0.001

Figure 14.5: Grade Capping Selection for Domain 80 and 90 (Pd g/t and Cu %)

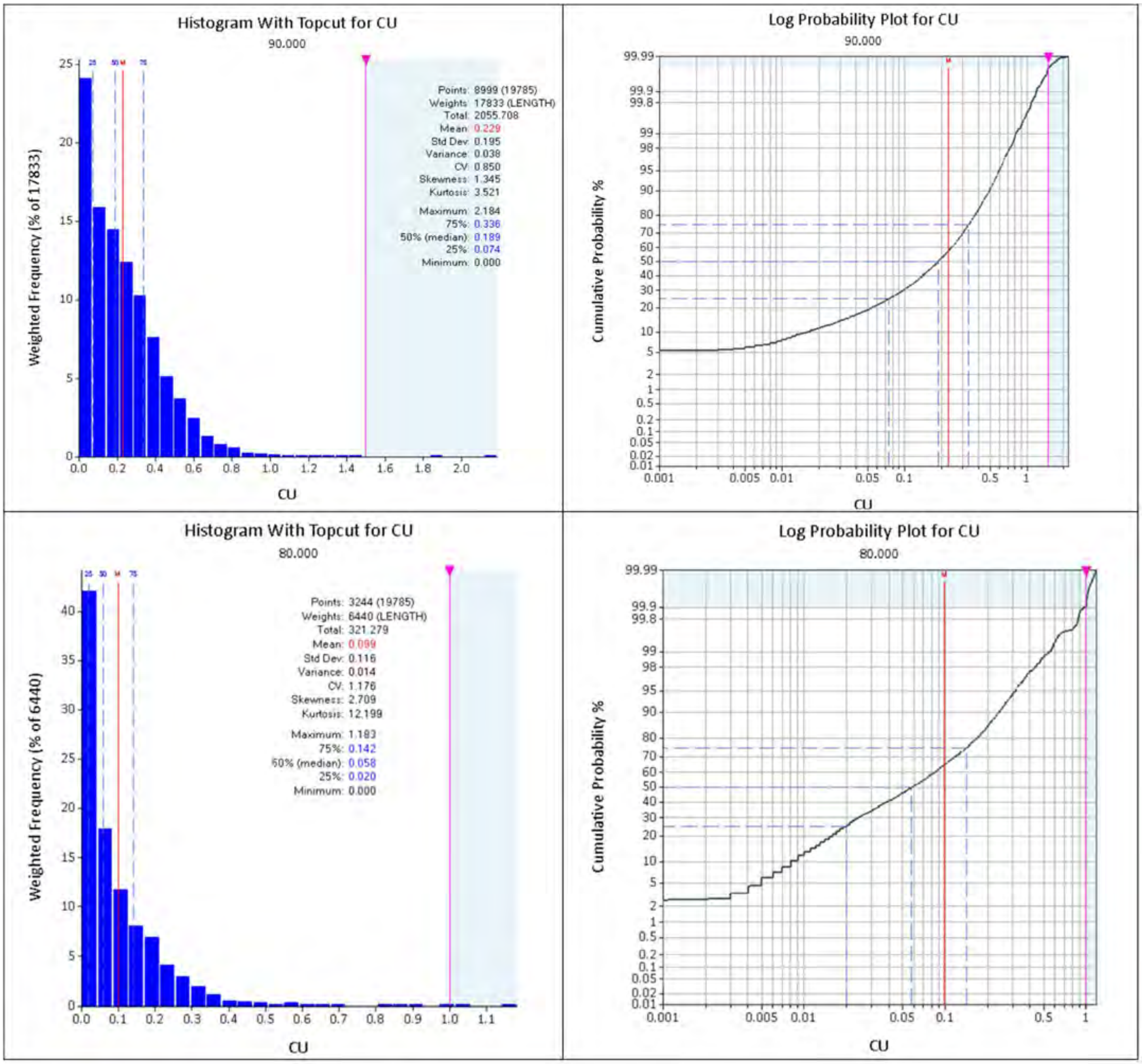


Table 14.8: Capping Thresholds

	Au g/t					Cu %					Pd g/t			
Change	Threshold	Avg Au g/t Uncapped	Number Capped	Avg Au g/t	Change	Threshold	Avg Cu % Uncapped	Number Capped	Avg Cu % Capped	Change	Threshold	Avg Pd g/t	Number Capped	Avg
-2.2%	0.2	0.04	2	0.03	-25.0%	0.26	0.07	8	0.07	0.0%	1.23	0.22	5	0
0.0%	0.3	0.05	1	0.05	0.0%	0.22	0.04	1	0.03	-25.0%	2	0.52	2	0
-1.9%	0.84	0.05	5	0.05	0.0%	1.3	0.23	3	0.22	-4.3%	2.5	0.42	15	
-0.6%	0.9	0.08	4	0.07	-12.5%	0.8	0.11	3	0.11	0.0%	6	0.54	8	
-1.9%	9,999	0.05	0	0.05	0.0%	0.6	0.12	3	0.11	-8.3%	2	0.25	5	0
-2.6%	0.38	0.04	4	0.04	0.0%	0.3	0.06	7	0.06	0.0%	1.5	0.28	3	0
-6.4%	0.15	0.04	2	0.04	0.0%	0.2	0.09	7	0.09	0.0%	0.6	0.18	4	0
0.0%	0.2	0.06	0	0.06	0.0%	0.3	0.1	3	0.1	0.0%	1.5	0.29	2	0
-7.7%	0.4	0.05	5	0.05	0.0%	0.7	0.11	7	0.11	0.0%	2.5	0.36	11	0
0.0%	0.4	0.06	4	0.06	0.0%	0.4	0.11	8	0.1	-9.1%	1.6	0.27	8	0
-12.8%	0.2	0.04	4	0.03	-25.0%	0.4	0.08	2	0.08	0.0%	1.2	0.23	6	0
-6.2%	0.3	0.07	3	0.07	0.0%	0.7	0.19	1	0.19	0.0%	2.6	0.59	3	0
-2.0%	2	0.08	3	0.07	-12.5%	1	0.1	4	0.1	0.0%	19	0.79	11	0
-1.8%	0.4	0.05	3	0.05	0.0%	0.8	0.15	5	0.15	0.0%	4	0.41	4	0
0.0%	9,999	0.03	0	0.03	0.0%	9,999	0.07	0	0.07	0.0%	9,999	0.25	0	0
-0.6%	1	0.07	7	0.07	0.0%	1.5	0.23	2	0.23	0.0%	7	0.58	7	0
-0.9%	0.2	0.04	2	0.04	0.0%	0.23	0.06	8	0.06	0.0%	1.7	0.31	3	0
-2.6%	0.14	0.06	2	0.05	-16.7%	0.25	0.08	1	0.08	0.0%	9,999	0.28	0	0
-3.6%	0.12	0.03	3	0.03	0.0%	0.3	0.07	1	0.06	-14.3%	0.55	0.09	3	0

14.2.8 Continuity Analysis

Variogram parameters have remained unchanged from the previous Mineral Resource estimate. Variography was reviewed taking into account the new drilling information and the below variograms were deemed appropriate to retain for the classification of the Mineral Resource.

Three-dimensional continuity analyses (variography) were conducted on the domain-coded uncapped composite data. The down-hole variogram was viewed at a 2.00 m lag spacing (equivalent to the composite length) to assess the nugget variance contribution. Standardized directional spherical models were used to model the experimental semi-variograms.

Back-transformed experimental semi-variograms were used to define appropriate ranges for Mineral Resource classification (Table 14.9). Based on the results of the variography, as well as the observed geological continuity and the existing drill hole pattern, a Measured classification range was defined as 70 m, and an Indicated classification range was defined as 120 m.

Table 14.9 : Experimental Semi-Variograms

Walford Zone 80							
Pd				Pt			
	Direction 1	Direction 2	Direction 3		Direction 1	Direction 2	Direction 3
C	-25 > 290	0 > 200	65 > 290	C	-25 > 290	0 > 200	65 > 290
0.47	0 m	0 m	0 m	0.53	0 m	0 m	0 m
0.38	48 m	60 m	8 m	0.32	34 m	45 m	8 m
0.15	100 m	75 m	70 m	0.16	100 m	75 m	50 m

North Main Zone 90							
Pd				Pt			
	Direction 1	Direction 2	Direction 3		Direction 1	Direction 2	Direction 3
C	-40 > 275	0 > 185	50 > 275	C	-40 > 275	0 > 185	50 > 275
0.11	0 m	0 m	0 m	0.21	0 m	0 m	0 m
0.55	40 m	26 m	17 m	0.5	38 m	11 m	11 m
0.34	100 m	70 m	60 m	0.29	70 m	30 m	80 m

14.2.9 Block Model

The modeled Marathon mineralization domains extend along a corridor of 2,000 m wide and 3,500 m in length. An orthogonal block model was established with the block model limits selected so as to cover the extent of the mineralized structures, the proposed open pit design, and to reflect the general nature of the mineralized domains (Table 14.10). The block model consists of separate variables for estimated grades, rock codes, percent, bulk density and classification attributes. A sub-cell model, with a minimum cell-size of 0.1 m x 0.1 m x 0.1 m, was used to accurately represent the volume and tonnage contained within the constraining mineralized domains. The sub-celled model was regularized to the parent-size of 5.0 m x 10.0 m x 5.0 m for the reporting of the Mineral Resource.

A dynamic anisotropy ("DA") model was also generated. This is a model of estimated dip and dip direction. Each block in the model has an estimated dip and dip direction value that is used to optimize the alignment of the search ellipse during the estimation process. Dip and dip directions were generated by digitizing strings that follow the orientation of the mineralization domains. The strings are then conditioned to 20 m points. The points are then tagged with the mineralization domains, and then estimation of dip and dip direction is performed in Datamine. Blocks that do not have an estimated dip and dip direction will use the default search parameters for grade estimation. The default search parameters have remained unchanged from the 2020 Mineral Resource estimate. The implementation of the DA model provides a better representation of the local grade estimate by honouring the variations of dip and dip direction of the mineralization.

Table 14.10: Block Model Setup

	Origin	Block Size (m)	Number of Blocks
Easting (X)	549,001.68	5.0	400
Northing (Y)	5,403,224.5	10.0	350
Elevation (max Z)	-340	5.0	168
Rotation	None		

14.2.10 Grade Estimation & Classification

Bulk density was modeled using inverse distance squared (ID2) linear weighting of between one and five bulk density samples with a maximum of one sample per drill hole. Bulk density estimates were constrained by lithological domains that form hard boundaries between the respective bulk density samples.

The Mineral Resource estimate was constrained by mineralization domains that form hard boundaries between the respective composite samples. Block grades were estimated in a single pass with inverse distance cubed (ID3) interpolation using a minimum of four and a maximum of 12 composites with a maximum of three samples per drill hole.

Composited samples were selected within a 200 m x 200 m x 50 m diameter search envelope oriented along the dip and strike of the mineralization domains. This was achieved by using the Datamine. For the DA process, the blocks that do not contain an estimated DA field used the default search parameters which are aligned to the overall orientation of the mineralization domains. For each grade element, a NN was also generated using the same search parameters. An NSR block model was subsequently calculated from the estimated block grades.

Blocks were classified algorithmically based on the local drill hole spacing within each domain. All blocks within 70 m of four or more drill holes were classified as Measured and blocks within 120 m of four or more drill holes were classified as Indicated. All additional estimated blocks were classified as Inferred. The average number of drill holes used and composite samples per block for grade estimation was as follows:

- Measured: 4.8 drill holes and 11.5 composite samples within 70 m.
- Indicated: 3.3 drill holes and 8.1 composite samples within 120 m.
- Inferred: 2.6 drill holes and 6.9 composite samples within 200 m.

The block model was then visually inspected to determine if a manual adjustment was required to remove any isolated (spotty dog effect) blocks. It was determined that this was not required.

14.2.11 Mineral Resource Estimate

Mineral Resources reported herein have been constrained within an optimized pit shell (Figure 14.6). The results within the constraining pit shell (Table 14.11) are used solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Pit-constrained Mineral Resources are reported using a NSR cut-off value of $15/t. A table summarizing the mineral resources at different cut-off grades is presented in Table 14.12.

Figure 14.6 : Isometric View of the Optimized Pit Shell

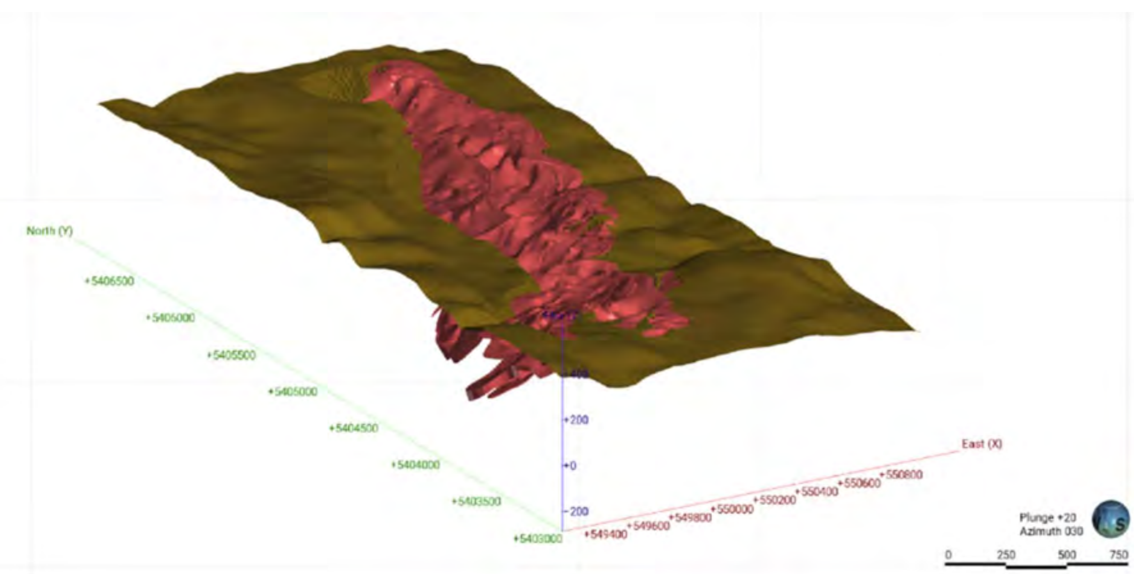


Table 14.11: Pit Constrained Mineral Resource Estimate for the Marathon Deposit(1-5)

Classification	Tonnes k	Pd g/t	Pt g/t	Cu %	Au g/t	Ag g/t	Pd koz	Pt koz	Cu Mlb	Au koz	Ag koz
Measured	158,682	0.6	0.19	0.20	0.07	1.75	3,077	995	712	359	8,939
Indicated	29,905	0.43	0.14	0.19	0.06	1.64	412	136	124	59	1,575
Total M+I	**188,587**	**0.58**	**0.19**	**0.20**	**0.07**	**1.73**	**3,489**	**1,131**	**836**	**418**	**10,514**
Inferred	1,662	0.37	0.14	0.16	0.07	1.25	20	7	6	4	67

Notes:

1. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
2. *Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*
3. *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
4. *Contained metal totals may differ due to rounding.*
5. *Mineral Resources are reported within a constraining pit shell at a NSR cut-off value of $15/t.*
6. *NSR = (Cu % x 88.72) + (Ag g/t x 0.47) + (Au g/t x 44.69) + (Pd g/t x 58.63) + (Pt g/t x 28.54) - 3.37.*
7. *Mineral Resource Estimate was based on metal prices of US$1,800/oz Pd, US$3.50/lb Cu, US$1,000/oz Pt, US$1,600/oz Au and US$20/oz Ag, and an exchange rate of 1.30 C$: 1 US$.*

The sensitivity of the Mineral Resource to NSR cut-off value was also calculated across a range of potentially economic NSR cut-off values for Measured and Indicated Mineral Resources (Table 14.17).

Table 14.12: Pit Constrained Measured and Indicated Mineral Resources Cut-off Sensitivities

NSR Cut-Off $/t	Tonnes k	Pd g/t	Pt g/t	Cu %	Au g/t	Ag g/t	Pd koz	Pt koz	Cu Mlb	Au koz	Ag koz
20	176,533	0.6	0.19	0.21	0.07	1.77	3,434	1,103	818	406	10,036
19	179,464	0.6	0.19	0.21	0.07	1.76	3,450	1,111	823	410	10,156
18	182,234	0.59	0.19	0.21	0.07	1.75	3,466	1,117	829	413	10,268
17	185,048	0.59	0.19	0.2	0.07	1.74	3,480	1,124	833	416	10,379
16	187,778	0.58	0.19	0.2	0.07	1.74	3,494	1,131	838	419	10,486
15	**190,250**	**0.57**	**0.19**	**0.2**	**0.07**	**1.73**	**3,506**	**1,136**	**842**	**422**	**10,581**
14	192,539	0.57	0.18	0.2	0.07	1.72	3,516	1,141	846	424	10,664
13	194,595	0.56	0.18	0.2	0.07	1.72	3,525	1,145	849	426	10,735
12	196,438	0.56	0.18	0.2	0.07	1.71	3,532	1,148	851	427	10,796
11	198,209	0.56	0.18	0.2	0.07	1.7	3,538	1,151	853	429	10,852
10	199,891	0.55	0.18	0.19	0.07	1.7	3,544	1,154	855	430	10,903

14.2.12 Validation

The block model was validated visually by the inspection of successive section lines in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values for Ag, Au, Cu, Pd and Pt.

The average estimated block grades were compared to the average NN block estimate at a zero cut-off grade (Table 14.14).

The results fall within acceptable limits for linear grade estimation.

An additional validation check was completed by comparing the average grade of the uncapped composites in a block to the associated model block grade estimate. The results fall within acceptable limits for linear grade estimation. The volume estimated was also checked against the reported volume of the individual mineralized domains (Table 14.13). Estimated volumes are based on the sub-blocked model.

Table 14.13: Volume Comparison

Domain	Volume Estimated (m^3)	Model Volume (m^3)
10	335	345
15	93	93
20	7,926	7,998
30	3,801	3,808
40	3,124	3,250
51	1,046	1,097
52	629	644
53	320	320
60	6,113	6,246
65	1,135	1,183
70	2,214	2,310
75	704	704
80	11,651	11,712
81	1,115	1,121
85	77	79
90	51,602	52,551
101	1,587	1,668
102	598	611
103	180	200
Total	**94,250**	**95,940**

A check for local estimation bias was completed by plotting vertical swath plots of the estimated block grade and the NN grade combining Measured and Indicated blocks. The swath plots demonstrated a reasonable level of smoothing for the block grade estimate and fall within acceptable limits for linear estimation.

Table 14.14: Comparison Between Block Estimated Grades and NN Grades

	Average Block Grades (NSR>0)					Average NN Grades (NSR>0)					Ratio of Estimated Block Grade and NN Grade (NSR>0)				
Domain	**Ag g/t**	**Au g/t**	**Cu%**	**Pd g/t**	**Pt g/t**	**Ag g/t**	**Au g/t**	**Cu%**	**Pd g/t**	**Pt g/t**	**Ag g/t**	**Au g/t**	**Cu%**	**Pd g/t**	**Pt g/t**
10	1.53	0.05	0.10	0.28	0.11	1.53	0.04	0.10	0.26	0.12	100%	112%	103%	109%	96%
15	1.67	0.05	0.03	0.50	0.28	1.68	0.04	0.01	0.37	0.30	99%	126%	231%	136%	95%
20	1.52	0.05	0.21	0.38	0.11	1.49	0.05	0.21	0.38	0.11	102%	103%	101%	102%	101%
30	1.54	0.08	0.10	0.47	0.21	1.55	0.07	0.09	0.43	0.21	99%	115%	105%	110%	103%
40	1.38	0.05	0.11	0.26	0.12	1.34	0.05	0.11	0.25	0.11	103%	103%	96%	101%	105%
51	1.94	0.04	0.06	0.22	0.09	1.87	0.04	0.06	0.22	0.09	104%	98%	102%	101%	101%
52	1.89	0.05	0.10	0.19	0.09	1.79	0.05	0.10	0.20	0.10	105%	113%	104%	96%	96%
53	2.01	0.06	0.11	0.26	0.11	1.95	0.06	0.09	0.24	0.10	103%	106%	116%	110%	109%
60	1.31	0.06	0.11	0.40	0.14	1.31	0.06	0.12	0.41	0.14	100%	95%	97%	98%	99%
65	1.64	0.06	0.10	0.25	0.14	1.69	0.06	0.09	0.21	0.12	97%	106%	108%	118%	112%
70	1.21	0.04	0.08	0.25	0.11	1.21	0.04	0.08	0.27	0.11	100%	95%	97%	96%	98%
75	2.14	0.07	0.18	0.57	0.19	2.01	0.06	0.17	0.56	0.19	107%	106%	110%	103%	102%
80	1.41	0.07	0.09	0.62	0.26	1.39	0.07	0.09	0.57	0.25	**102%**	**100%**	**102%**	**107%**	**102%**
81	1.15	0.05	0.14	0.34	0.14	1.18	0.05	0.14	0.34	0.14	97%	99%	103%	101%	101%
85	0.74	0.03	0.06	0.23	0.07	0.78	0.04	0.07	0.27	0.07	95%	77%	81%	86%	106%
90	1.60	0.06	0.21	0.49	0.15	1.58	0.06	0.21	0.48	0.15	**101%**	**100%**	**102%**	**103%**	**102%**
101	1.07	0.04	0.07	0.36	0.10	1.00	0.05	0.08	0.36	0.10	107%	95%	91%	99%	98%
102	1.54	0.06	0.10	0.27	0.09	1.53	0.06	0.10	0.25	0.09	101%	90%	93%	106%	101%
103	1.89	0.04	0.11	0.08	0.06	2.04	0.04	0.12	0.07	0.06	93%	99%	91%	112%	95%
Total	**1.53**	**0.06**	**0.17**	**0.46**	**0.16**	**1.51**	**0.06**	**0.17**	**0.45**	**0.16**	**101%**	**101%**	**102%**	**104%**	**102%**

14.2.13 Suitability of the Mineral Resource Estimate

Fred Brown, P.Geo., and Eugene Puritch, P.Eng., FEC, CET, have reviewed the Mineral Resource estimate provided by Gen Mining for the Marathon Deposit, and consider that the block model Mineral Resource estimates and Mineral Resource classification reported by Gen Mining represent a reasonable estimation of the global Mineral Resources for the Marathon Project with regard to compliance with generally accepted industry standards and guidelines, the methodology used for estimation, the classification criteria used and the actual implementation of the methodology in terms of Mineral Resource estimation and reporting. The Mineral Resources reported by Gen Mining have been estimated in conformity with the requirements of the CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (2019) as required by the Canadian Securities Administrators' National Instrument 43-101 (2014). Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

14.3 Geordie and Sally Mineral Resource Estimates

The Geordie and Sally Mineral Resource estimates have remained unchanged from the 2020 Mineral Resource statement.

Mineral Resource estimates were prepared by P&E for the Geordie and Sally Deposits. The methodologies to create the block models were similar to those used for the Marathon Deposit. All drilling and assay data were provided in the form of Excel data files by Gen Mining. The GEOVIA GEMS™ V6.8.2 database for the Geordie Deposit Mineral Resource estimate, compiled by P&E, consisted of 61 drill holes totalling 9,647 m, of which a total of 57 drill holes intersected the mineralization wireframes used for the Mineral Resource estimate. For the Sally Deposit, the database consisted of 82 drill holes totalling 16,975 m and 371 surface channels totalling 1,871 m, of which a total of 47 drill holes and 162 channels intersected the mineralization wireframes used for the Mineral Resource estimate.

The resulting pit constrained Mineral Resource estimates for the Geordie and Sally Deposits, at an NSR $13/t cut-off, as of the effective date of this Mineral Resourced estimate, are tabulated in Table 14.15 and Table 14.16, respectively. P&E considers the mineralization of Geordie and Sally to be potentially amenable to open pit economic extraction. Respective Geordie and Sally surface drill plans, 3D domains and constraining pit shells can be seen in Figure 14.7 to Figure 14.12.

Table 14.15: Geordie Pit Constrained Mineral Resource Estimate (Effective June 30, 2020)

Classification	Tonnes k	Pd g/t	Pt g/t	Cu %	Au g/t	Ag g/t	Pd koz	Pt koz	Cu Mlb	Au koz	Ag koz
Indicated	17,268	0.56	0.04	0.35	0.05	2.4	312	20	133	25	1,351
Inferred	12,899	0.51	0.03	0.28	0.03	2.4	212	12	80	14	982

Notes:

1. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.*
2. *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
3. *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
4. *The Mineral Resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
5. *The Mineral Resource Estimate was based on metal prices of US$1,600/oz Pd, US$3.00/lb Cu, US$900/oz Pt, US$1,500/oz Au and US$18/oz Ag and an NSR cut-off value of $13/t, and an exchange rate of 1.30 C$: 1 US$.*

Table 14.16: Sally Pit Constrained Mineral Resource Estimate (Effective June 30, 2020)

Classification	Tonnes k	Pd g/t	Pt g/t	Cu %	Au g/t	Ag g/t	Pd koz	Pt koz	Cu Mlb	Au koz	Ag koz
Indicated	24,801	0.35	0.2	0.17	0.07	0.7	278	160	93	56	567
Inferred	14,019	0.28	0.15	0.19	0.05	0.6	124	70	57	24	280

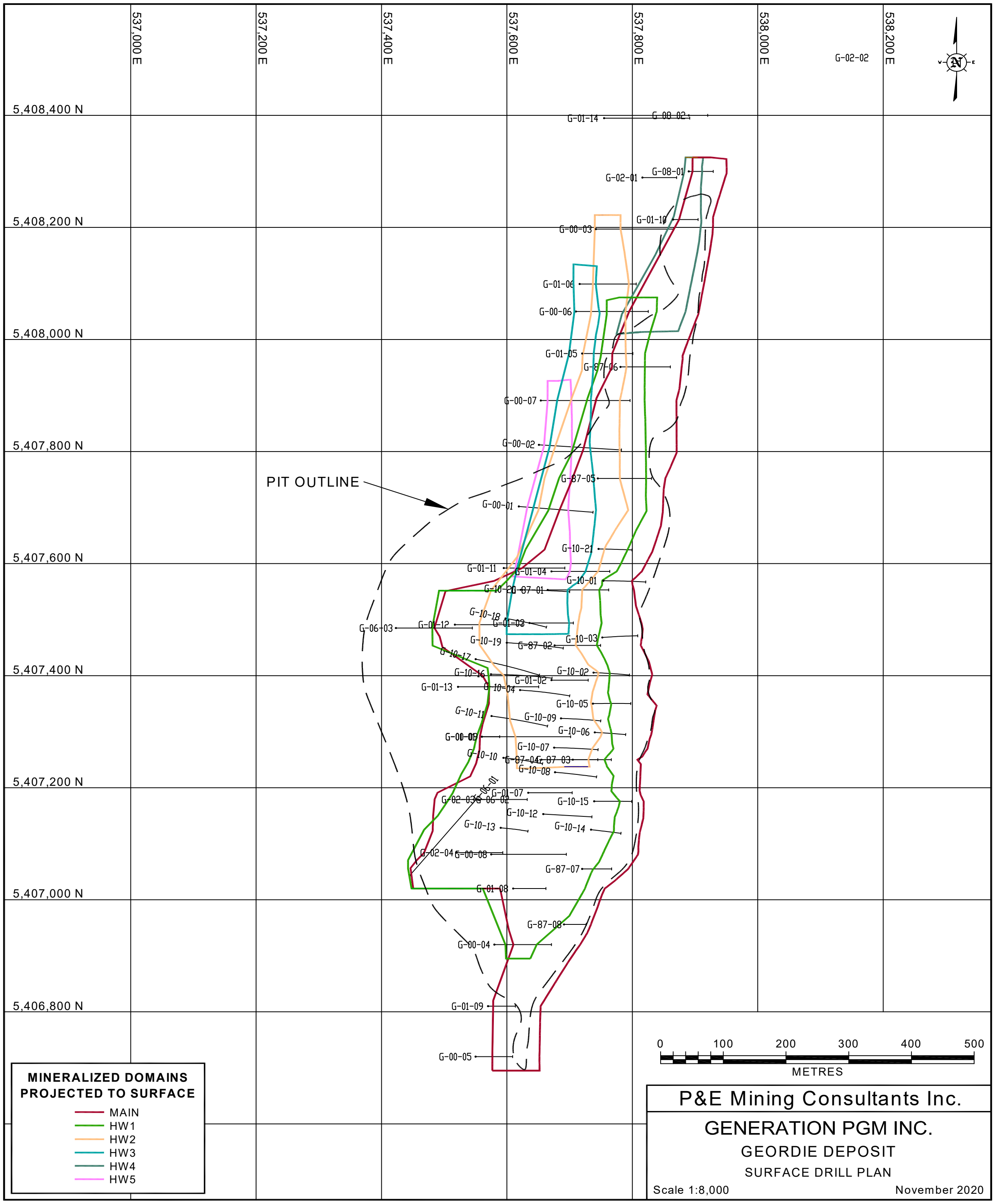
Notes:

1. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.*
2. *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
3. *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
4. *The Mineral Resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
5. *The Mineral Resource Estimate was based on metal prices of US$1,600/oz Pd, US$3.00/lb Cu, US$900/oz Pt, US$1,500/oz Au and US$18/oz Ag and an NSR cut-off value of $13/t., and an exchange rate of 1.30 C$: 1 US$.*

14.3.1 Grade Estimation and Classification

The Cu, Pd, Pt, Au and Ag grade blocks were interpolated with ID2. Multiple passes were executed for the grade interpolation to progressively capture the sample points to avoid over-smoothing and preserve local grade variability. Search ranges were based on the variograms and search directions which were aligned with the strike and dip directions of each domain accordingly. The block size assumed for the models is 5 m L x 5 m W x 6 m H.

Figure 14.7: Geordie Deposit Surface Drill Plan



Source: *P&E (2020).*

Figure 14.8: Geordie Deposit 3D Domains Isometric View

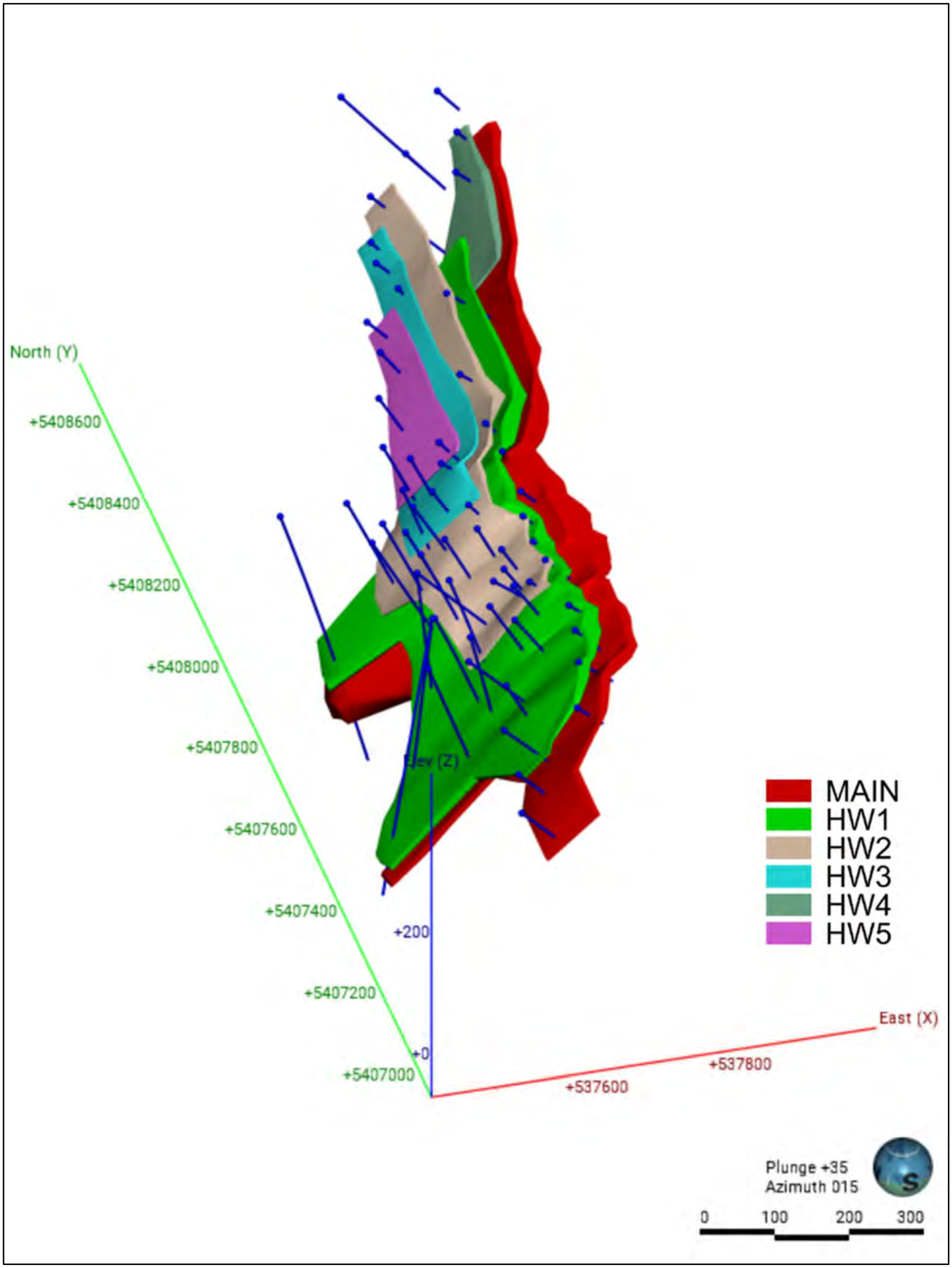


Figure 14.9: Geordie Deposit Constraining Pit Shell Isometric View

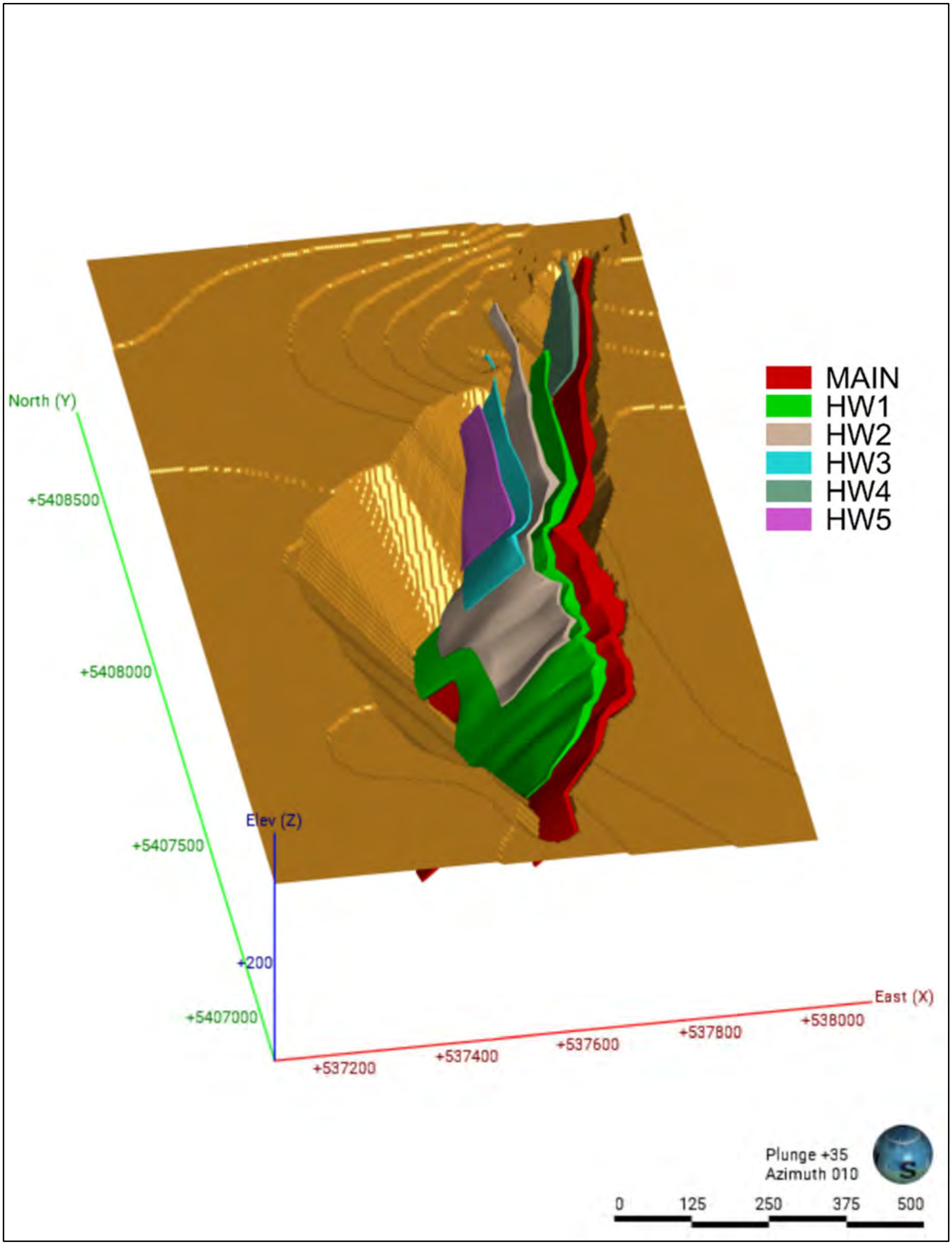
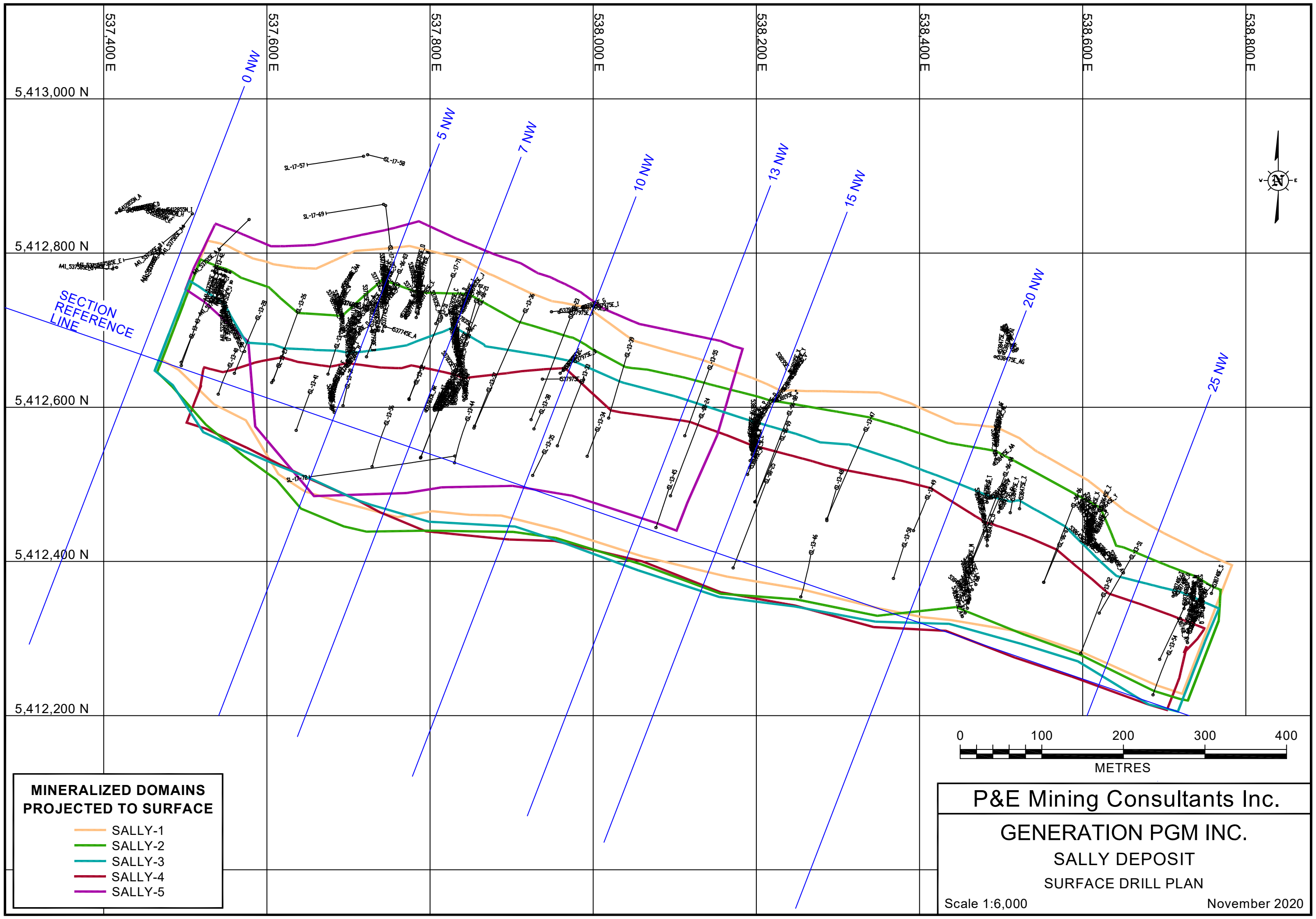


Figure 14.10: Sally Deposit Surface Drill Plan



Source: *P&E (2020).*

Figure 14.11: Sally Deposit 3D Domains Isometric View

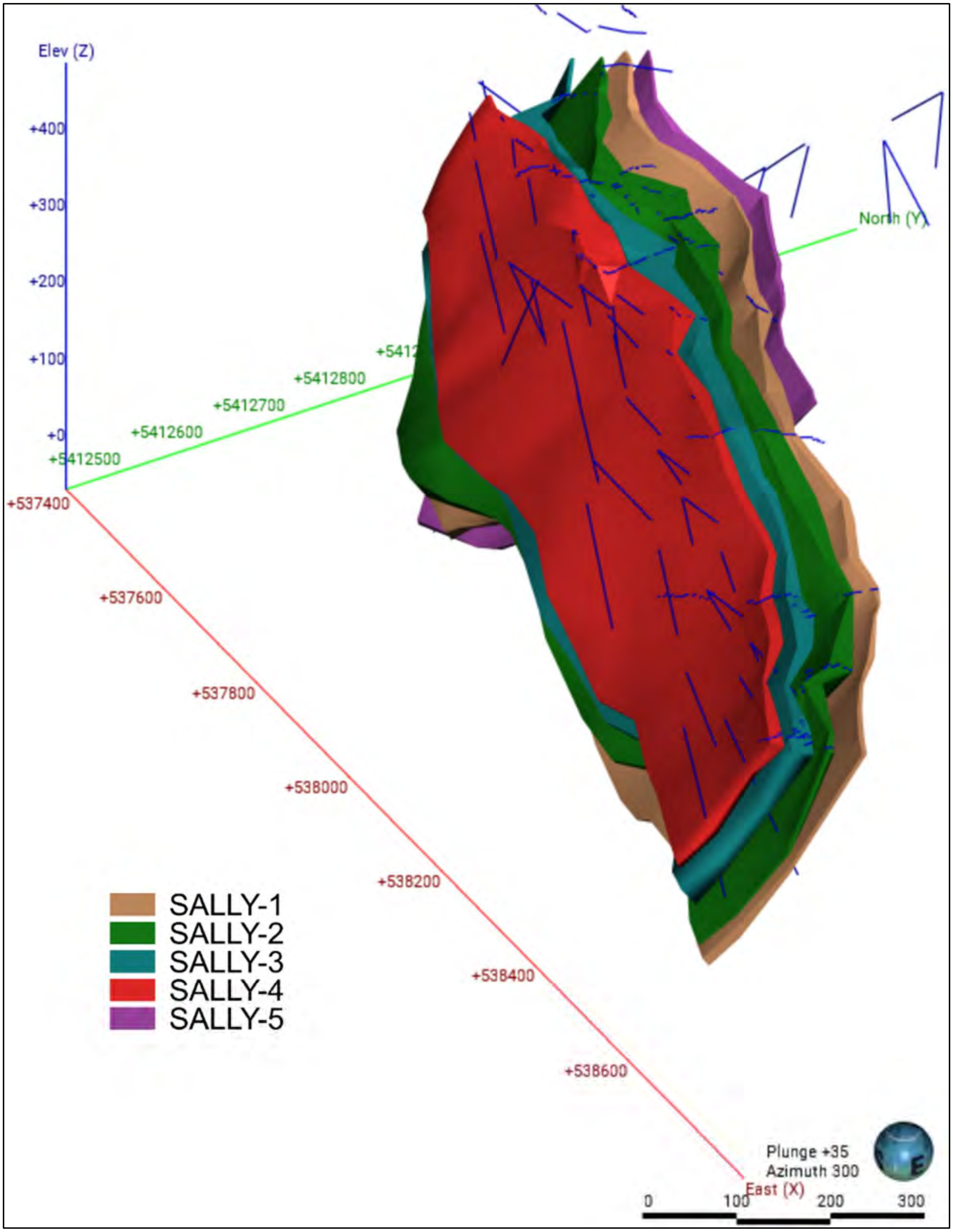
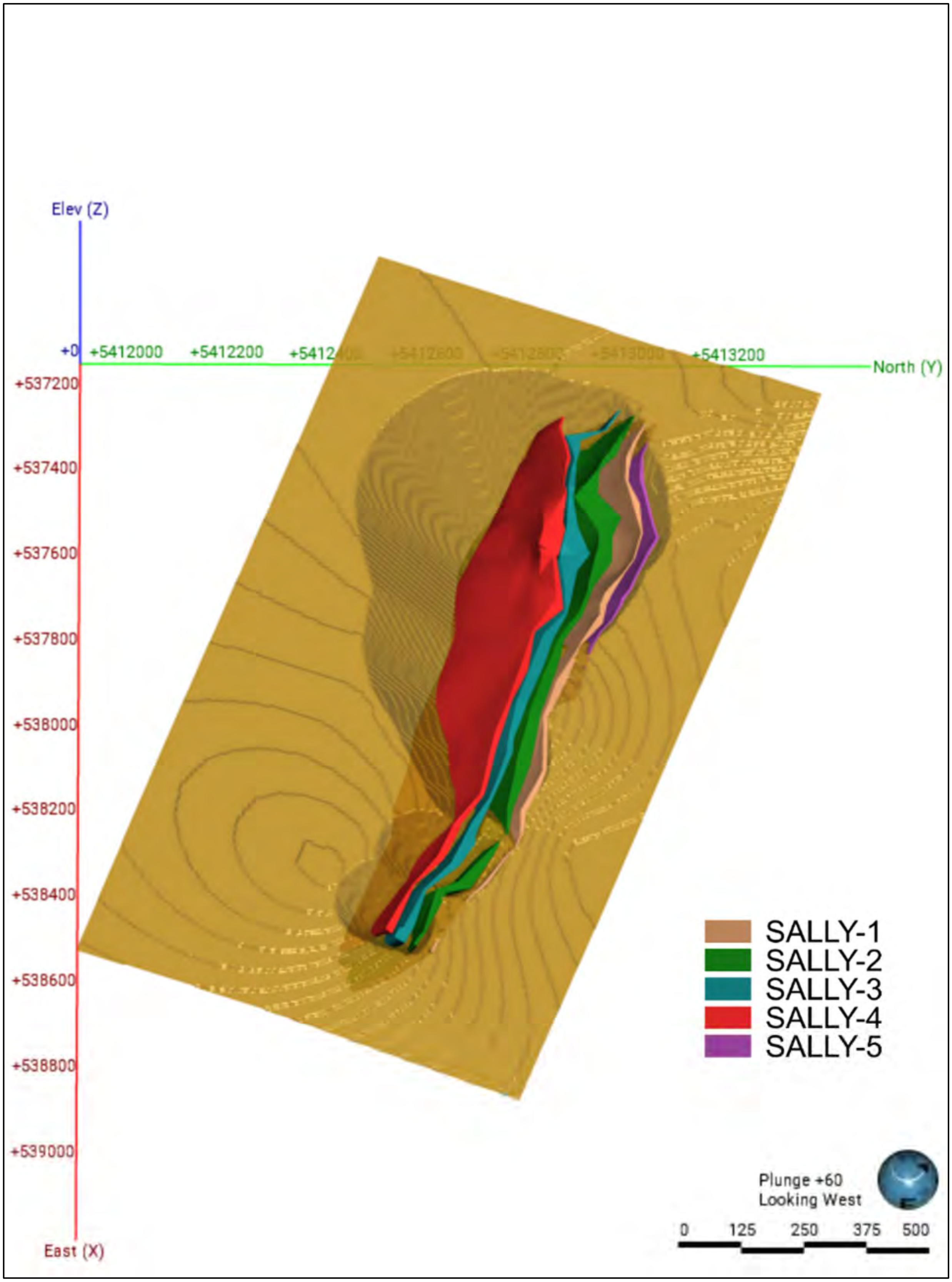


Figure 14.12: Sally Deposit Constraining Pit Shell Isometric View



14.4 Mineral Resource Estimates for the Property

Table 14.17: Pit-Constrained Mineral Resource Estimate for the Marathon, Geordie and Sally Deposits (Effective date December 31, 2022)

Mineral Resource Classification	Tonnes	Pd		Cu		Pt		Au		Ag	
	k	g/t	koz	%	M lbs	g/t	koz	g/t	koz	g/t	koz
Marathon Deposit											
Measured	158,682	0.60	3,077	0.20	712	0.19	995	0.07	359	1.75	8,939
Indicated	29,905	0.43	412	0.19	124	0.14	136	0.06	59	1.64	1,575
M+I	**188,587**	**0.58**	**3,489**	**0.20**	**836**	**0.19**	**1131**	**0.07**	**418**	**1.73**	**10,514**
Inferred	1,662	0.37	20	0.16	6	0.14	7	0.07	4	1.25	67
Geordie Deposit											
Indicated	17,268	0.56	312	0.35	133	0.04	20	0.05	25	2.4	1,351
Inferred	12,899	0.51	212	0.28	80	0.03	12	0.03	14	2.4	982
Sally Deposit											
Indicated	24,801	0.35	278	0.17	93	0.2	160	0.07	56	0.7	567
Inferred	14,019	0.28	124	0.19	57	0.15	70	0.05	24	0.6	280
Total Project											
Measured	158,682	0.60	3,077	0.20	712	0.19	995	0.07	359	1.75	8,939
Indicated	71,974	0.43	1,002	0.22	350	0.14	316	0.06	140	1.5	3,493
M+I	**230,656**	**0.55**	**4,079**	**0.21**	**1,062**	**0.18**	**1,311**	**0.07**	**499**	**1.67**	**12,432**
Inferred	28,580	0.39	356	0.23	143	0.1	89	0.04	42	1.45	1,329

***Notes**:*

1. *Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*
2. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
3. *The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
4. *The Marathon Mineral Resource is reported within a constrained pit shell at a NSR cut-off value of $15/t.*
5. *Marathon NSR (C$t) = (Cu % x 88.72) + (Ag g/t x 0.47) + (Au g/t x 44.69) + (Pd g/t x 58.63) + (Pt g/t x 28.54) - 3.37.*
6. *The Marathon Mineral Resource Estimate was based on metal prices of US$1,800/oz Pd, US$3.50/lb Cu, US$1,000/oz Pt, US$1,600/oz Au and US$20/oz Ag, and an exchange rate of 1.30 C$ to 1 US$.*
7. *The Sally and Geordie Mineral Resources are reported within a constraining pit shell at a NSR cut-off value of $13/t.*
8. *Sally and Geordie NSR (C$/t) = (Ag g/t x 0.48) + (Au g/t x 42.14) + (Cu % x 73.27) + (Pd g/t x 50.50) + (Pt g/t x 25.07) – 2.62.*
9. *The Sally and Geordie Mineral Resource estimate was based on metal prices ofUS$1,600/oz Pd, US$3.00/lb Cu, US$900/oz Pt, US$1,500/oz Au and US$18/oz Ag, and an exchange rate of 1.30 C$: 1 US$.*
10. *Contained metal totals may differ due to rounding.*

15. MINERAL RESERVE ESTIMATE

15.1 Summary

The Mineral Reserve estimate for the Marathon Project was prepared by GMS and is presented in Table 15.1.

Table 15.1: Marathon Project Open Pit Mineral Reserve Estimates (Effective Date of December 31, 2022)

Mineral Reserves	Tonnage	Pd		Cu		Pt		Au		Ag	
	kt	g/t	koz	%	M lbs	g/t	koz	g/t	koz	g/t	koz
Proven	114,798	0.65	2,382	0.21%	530	0.20	744	0.07	259	1.68	6,191
Probable	12,863	0.47	193	0.20%	55	0.15	61	0.06	26	1.53	635
P&P	**127,662**	**0.63**	**2,575**	**0.21%**	**586**	**0.20**	**806**	**0.07**	**285**	**1.66**	**6,825**

***Notes**:*

1. *Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) were used for Mineral Reserve classification.*
2. *Mineral Reserve Estimate completed by Alexandre Dorval, P.Eng., of GMS, an independent QP, as defined by NI 43-101.*
3. *Mineral Reserves were estimated at a cut-off value $16.90 NSR/t of ore.*
4. *Mineral Reserves were estimated using the following long-term metal prices: Pd = US$1,500/oz, Pt = US$1,000/oz, Cu = US$3.50/lb, Au = US$1,600/oz and Ag = US$20/oz, and an exchange rate of 1.30 C$ to 1 US$. The pit designs are based on a pit shell selected at a revenue factor of 0.74.*
5. *A minimum mining width of 5 m was used.*
6. *Bulk density of ore is variable and averages 3.1 t/m^3.*
7. *The average strip ratio is 2.6:1.*
8. *The average mining dilution factor is 9%.*
9. *Numbers may not add due to rounding.*

The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on the Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are classified as waste.

Factors that may affect the Mineral Reserve estimate include the following.

- Commodity price
- Changes in interpretations of mineralization geometry and continuity of mineralization zones
- Changes to geotechnical, hydrogeological, and metallurgical recovery assumptions
- Input factors used to assess dilution and recoveries

- Assumptions that the operation can obtain all required permits to operate
- Assumptions regarding social, permitting, and environmental conditions

The QP has reviewed the risks, opportunities, conclusions, and recommendation and is not aware of any conditions that would put the Mineral Reserve at a high risk level.

15.2 Resource Block Model

The resource model was produced as a sub-blocked model in Datamine™ Studio RM software. For mine planning purposes, the model was regularized to a standard SMU block size of 5 m x 10 m x 5 m and provided to GMS.

15.3 Pit Optimization

Open pit optimization was conducted in GEOVIA Whittle™ to determine the optimal economic shape of the open pit to guide the pit design process. This task was undertaken using the Whittle software which is based on the Pseudoflow algorithm. The method uses the values of the blocks to define a pit outline that has the highest possible total economic value, subject to the required pit slopes defined as structure arcs in the software. This section describes all the parameters used to calculate block values in Whittle™.

For the Mineral Reserve estimate, only Measured and Indicated Mineral Resource blocks were considered for optimization purposes and for mineable resource calculations.

15.3.1 Pit Slope Geotechnical Assessment

Knight Piésold Ltd. ("KP") produced a feasibility level pit slope design study to support the mine designs. The conclusions of this study have been used as an input to the pit optimization and design process.

The pit area was divided in sectors based on the data collected from the oriented core drill holes. In general, the pit area is controlled by bench geometry. The Central West (upper and lower), Central South and South Northwest sectors are controlled by bench scale failures and have different recommended slope geometry.

It has been assessed that the open pit will be developed in relatively consistent rock mass quality. The rock mass is generally of good quality with small zones of reduced rock mass quality associated with large-scale structures (faults, shears, lineaments, etc.). The rock mass characteristics for each domain as depicted by KP are:

- Hanging Wall: Average UCS value of 140 MPa and a Mi value of 11. It is classified as good quality rock with a RMR89 design value of 70.
- Ore Zone Gabbro: Average UCS value of 115 MPa and a Mi value of 9. It is classified as good quality rock with a RMR89 design value of 70.

- Footwall: Average UCS value of 180 MPa and a Mi value of 11. It is classified as good quality rock with a RMR89 design value of 65.

KP identified 18 design sectors based on the pit geometry and geomechanical domains. Slope analyses were performed on each sector to establish achievable slope configurations.

Based on the stability analyses and precedent practice, KP indicated that the recommended geometries were slightly aggressive but reasonable and appropriate when controlled blasting, proactive geotechnical monitoring and geomechanical analyses will be executed.

According to KP, the rock mass has a moderate to low permeability. The measured values suggest the pre-mining groundwater surface ranges from 4 to 18 m below the ground surface. Groundwater depressurization will not strongly influence overall slope stability. However, the phreatic surface water that develops behind the pit walls should be monitored and depressurized as needed.

A slope monitoring program is recommended for all stages of pit development. It should include geotechnical and tension crack mapping, surface displacement monitoring program using surface prisms.

The slope configuration options are presented in Table 15.2. Double benching will have to be done with pre-split, no sub-grade drilling, and well controlled blasting practices are required.

The final pit was designed using a double benching configuration to a final height of 20 m. The pit slope profile is based on recommendations by KP as presented in Table 15.2. A geotechnical catch-bench of 15 m was integrated in the latest ultimate pit designs at elevation 170 m.

KP did not consider the overburden in the domain definition process and analyses because it is expected to form only a minor part of the proposed pit slopes (0.3 to 1 m typical thickness). Where overburden thicknesses are more significant, adjustments to the pit geometry will be completed as part of ongoing operational risk assessments and mine planning.

Table 15.2: Marathon Final Wall Geotechnical Recommendations

Slope Parameters	
Final Bench Height (m)	20.0
Bench Face Angle (º)	65 to 75
Avg. Design Catch Bench Width (m)	10.4
Inter-ramp Angle (º)	48 to 55
Overall Slope Angle (º)	44 to 55
Geotechnical Benches (m)	15

15.3.2 Mining Dilution and Ore Loss

A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic cut-off grade. The block contacts are then used to estimate a dilution skin around ore blocks to estimate an expected dilution during mining. The dilution skin consists of 1.0 m of material in a N-S direction (across strike) and 1.0 m in an E-W direction (along strike). The dilution is therefore specific to the geometry of the ore body and the number of contacts between ore and waste. The ore body consists of two styles of mineralization. There are massive, mineralized envelopes such as for the main zone which incur relatively little dilution and other narrower mineralized envelopes that incur higher mining dilutions with this estimation technique. Orphaned blocks of ore are considered as ore loss

For each mineralized block in the resource model, diluted grades and a new density are calculated by considering the *in-situ* grades and *in-situ* density of the surrounding blocks.

15.3.3 Pit Optimization Parameters and Cut-Off Grade

Unit reference mining costs are used for a “reference mining block” usually located near the pit crest or surface and are incremented with depth, which corresponds to the additional cycle time and thus hauling cost. The reference mining cost is estimated at $2.80/t with an incremental depth factor of $0.05/t per 10 m bench.

The overall slope angles utilized in Whittle are based on the inter-ramp angles recommended in the KP pit slope study with provisions for ramps and geotechnical berms. The overall slope angle in competent rock is 40 to 52 degrees based on a designed inter-ramp angle of 48 to 55 degrees.

The total ore-based cost is estimated at $14.68/t (US$11.29/t), which includes processing, general and administration costs, sustaining capital and a closure cost provision (Table 15.3).

Table 15.3: Ore-Based Cost Assumption

Ore-Based Cost Assumptions	$/t
Avg. Incremental Ore Mining Cost	0.20
Processing (including power)	9.53
General & Administration	1.69
Sustaining Capital	2.70
Closure Cost Provision	0.56
Total Ore-Based Cost	**14.68**

For a polymetallic mine such as the Marathon Project, the cut-off grade is best expressed as a NSR value in $/t for the mineralized material. The marginal cut-off grade corresponds to the ore-based cost. However, an elevated cut-off grade was applied of $16.90/t (US$13.00/t) of ore. These elevated cut-off grades applied to select blocks prior to dilution will provide some operating margin and cover the impact of mining dilution.

A summary of the pit optimization parameters is presented in Table 15.4 for a nominal milling rate of 10.1 Mt per year based on long-term metal price assumptions and an exchange rate of 1.30 C$:1.00 US$. This copper concentrate will be sent to a smelter for smelting and refining to produce saleable metals. Indicative terms have been used to calculate the NSR for the concentrate and for the ore itself with the parameters summarized in Table 15.4. A concentrate transportation and logistics cost of US$145/t has been assumed.

Table 15.4: Marathon Project Optimization Parameters

Input Parameters					
Exchange Rate			C$	1.30	
Diesel Fuel Price Delivered			$/litre	1.17	
Electricity Cost			$/kWh	0.07	
Processing Inputs					
Nominal Milling Rate			Mt/yr	10.1	
Concentrate Grade			% Cu	Variable	
Concentrate Treatment Charge			US$/dmt	79	
Concentrate Transport & Logistics			US$/dmt	145.16	
Metal	**Copper**	**Palladium**	**Platinum**	**Gold**	**Silver**
Metal Prices (US$)	$3.50/lb	$1,500/oz	$1,000/oz	$1,600/oz	$20/oz
Refining Charges (US$)	$0.079/lb	$24.5/oz	$24.5/oz	$5/oz	$0.50/oz
Payable Rates (%)	96.5	95	93	93.5	93.5
Min. Deductions	1.00%	2.625 g/t	2.625 g/t	1 g/t	30 g/t
Concentrator Recovery (%)	Variable (Refer to recovery curves in Section 13)				
Ore-Based Costs					
Average Incremental Ore Mining Cost			$/t milled	0.2	
Processing Cost (incl. power)			$/t milled	9.53	
General and Administration			$/t milled	1.69	
Rehabilitation and Closure			$/t milled	0.56	
Sustaining Capital			$/t milled	2.7	
Total Ore-Based Cost			$/t milled	14.68	
Mining Inputs					
Mining Dilution			%	10%	
Mining Loss			%	3%	
Total Mining Reference Cost			$/t mined	2.8	
Incr. Bench Cost ($ /10 m bench)			$/t mined	0.05	
Overall Slope Angle in Fresh Rock			degrees	Variable	

15.3.4 Open Pit Optimization Results

The Whittle nested shell results are presented in Table 15.6 and Figure 15.1 using only the Measured and Indicated Mineral Resource. The nested shells are generated by using revenue factors to scale up and down from the base case selling price. The shell selection is presented in Table 15.5. Pit Shell 22 is selected as the optimum final pit shell which corresponds to a Revenue Factor of 0.74 (effective metal prices of approximately 1,110 US$/oz Pd, 2.59 US$/lb Cu, 740 US$/oz Pt, 1,184 US$/oz Au, 14.8 US$/oz Ag) . This shell has a total tonnage of 474.1 Mt, including 125.6 Mt of ore. This is the smallest shell that achieves close to maximum value using a practical phasing approach.

Figure 15.1: Pit by Pit Graph M&I Resource

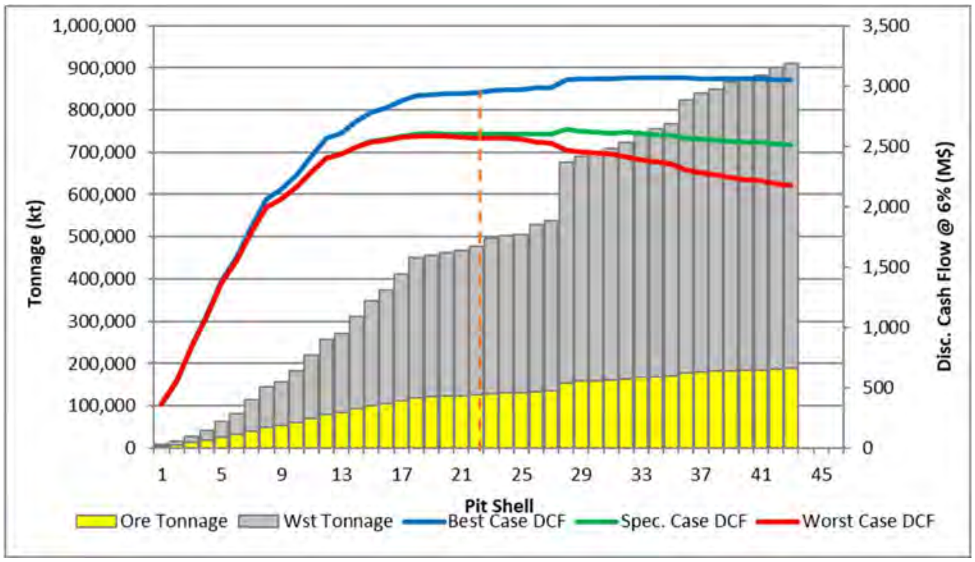


Table 15.5: Measured and Indicated Mineral Resource Pit Shell Selection

Shell Selection	Selection
Shell Number	22
Shell RF	0.74
Total Tonnage (kt)	476,140
Waste Tonnage (kt)	350,508
Strip Ratio (W:O)	2.79
Ore Tonnage (kt)	125,633
Cu Grade (%)	0.22
Ag Grade (g/t)	1.65
Au Grade (g/t)	0.07
Pt Grade (g/t)	0.21
Pd Grade (g/t)	0.68
NSR ($/t)	64.78
DCF @ 6 % ($ M)	2,597

Table 15.6: Measured and Indicated Mineral Resource Whittle Shell Results for Combined Diluted Model

Pit Shell	Best Case Disc. @ 8% ($ M)	Specified Disc. @ 8% ($ M)	Worst Case Disc. @ 8% ($ M)	Total Tonnage (kt)	Ore Tonnage (kt)	Waste Tonnage (kt)	Strip Ratio (W:O)	Cu Grade (%)	Ag Grade (g/t)	Au Grade (g/t)	Pt Grade (g/t)	Pd Grade (g/t)	Total NSR ($/t)
1	366	366	366	8,233	3,797	4,436	1.17	0.34	2.08	0.15	0.51	1.88	163.94
2	560	560	560	14,685	7,245	7,440	1.03	0.33	1.92	0.13	0.42	1.57	139.66
3	849	848	848	27,456	12,677	14,778	1.17	0.31	1.80	0.11	0.37	1.37	123.72
4	1098	1092	1092	41,687	18,278	23,409	1.28	0.30	1.76	0.11	0.34	1.24	113.94
5	1383	1368	1368	63,358	25,428	37,930	1.49	0.30	1.67	0.10	0.32	1.18	108.27
6	1578	1553	1553	82,231	31,182	51,049	1.64	0.29	1.63	0.10	0.31	1.12	103.28
7	1830	1785	1785	113,810	39,501	74,309	1.88	0.27	1.65	0.10	0.31	1.09	100.32
8	2058	1994	1994	144,152	48,875	95,277	1.95	0.27	1.64	0.09	0.29	1.01	94.07
9	2148	2068	2068	157,081	53,468	103,613	1.94	0.27	1.63	0.09	0.28	0.97	90.19
10	2263	2163	2163	182,032	59,307	122,726	2.07	0.26	1.62	0.09	0.27	0.94	87.53
11	2422	2292	2292	220,170	69,076	151,094	2.19	0.25	1.67	0.09	0.26	0.90	83.73
12	2,564	2,400	2,400	257,445	79,624	177,821	2.23	0.24	1.67	0.08	0.25	0.84	79.26
13	2,610	2,433	2,433	271,834	83,445	188,389	2.26	0.24	1.67	0.08	0.24	0.82	77.00
14	2,705	2,496	2,495	311,572	92,054	219,518	2.38	0.24	1.67	0.08	0.23	0.78	74.25
15	2,779	2,538	2,533	348,077	100,234	247,843	2.47	0.23	1.67	0.08	0.23	0.76	71.80
16	2,824	2,562	2,552	373,409	105,597	267,812	2.54	0.23	1.67	0.08	0.22	0.73	69.91
17	2,875	2,585	2,573	411,471	112,890	298,581	2.64	0.23	1.66	0.07	0.22	0.71	68.06
18	2,919	2,603	2,586	450,467	119,916	330,552	2.76	0.22	1.67	0.07	0.21	0.70	66.71
19	2,924	2,604	2,586	455,091	120,830	334,262	2.77	0.22	1.67	0.07	0.21	0.69	66.30
20	2,932	2,603	2,581	462,350	122,442	339,908	2.78	0.22	1.66	0.07	0.21	0.69	65.80
21	2,937	2,601	2,578	466,880	123,615	343,265	2.78	0.22	1.66	0.07	0.21	0.68	65.34
22	2,946	2,597	2,570	476,140	125,633	350,508	2.79	0.22	1.65	0.07	0.21	0.68	64.78
23	2,961	2,601	2,567	495,207	128,327	366,879	2.86	0.22	1.65	0.07	0.21	0.67	64.32
24	2,967	2,599	2,563	503,362	129,847	373,515	2.88	0.22	1.65	0.07	0.20	0.67	63.87
25	2,969	2,597	2,561	505,725	130,575	375,150	2.87	0.22	1.64	0.07	0.20	0.66	63.48
26	2,982	2,601	2,533	528,542	133,136	395,406	2.97	0.22	1.65	0.07	0.20	0.66	63.14
27	2,988	2,597	2,527	538,685	134,924	403,760	2.99	0.22	1.65	0.07	0.20	0.65	62.69
28	3,048	2,641	2,468	675,227	154,847	520,380	3.36	0.21	1.67	0.07	0.20	0.63	60.69
29	3,055	2,622	2,456	689,161	157,816	531,346	3.37	0.21	1.66	0.07	0.19	0.62	59.73
30	3,058	2,617	2,441	698,644	159,653	538,991	3.38	0.21	1.66	0.07	0.19	0.61	58.98
31	3,061	2,611	2,432	708,437	161,670	546,767	3.38	0.20	1.65	0.07	0.19	0.60	58.20
32	3,063	2,616	2,409	723,142	164,281	558,862	3.40	0.20	1.64	0.07	0.19	0.60	57.44
33	3,066	2,604	2,383	746,903	167,347	579,556	3.46	0.20	1.64	0.07	0.19	0.59	56.74
34	3,066	2,591	2,367	756,452	168,905	587,547	3.48	0.20	1.63	0.07	0.19	0.58	56.17
35	3,066	2,589	2,354	767,237	170,577	596,660	3.50	0.20	1.63	0.07	0.18	0.58	55.65
36	3,064	2,570	2,302	823,320	177,130	646,191	3.65	0.20	1.63	0.07	0.18	0.57	54.89
37	3,063	2,560	2,279	838,817	179,343	659,474	3.68	0.19	1.63	0.07	0.18	0.57	54.59
38	3,061	2,552	2,265	850,050	181,000	669,049	3.70	0.19	1.63	0.07	0.18	0.56	54.39
39	3,058	2,542	2241	866,696	183,083	683,614	3.73	0.19	1.63	0.07	0.18	0.56	54.17
40	3,056	2,535	2224	876,583	184,354	692,229	3.75	0.19	1.63	0.07	0.18	0.56	54.03
41	3,055	2,530	2216	882,881	185,186	697,694	3.77	0.19	1.63	0.07	0.18	0.56	53.93
42	3,051	2,519	2192	900,608	187,308	713,300	3.81	0.19	1.62	0.07	0.18	0.56	53.69
43	3,049	2,513	2177	909,684	188,361	721,323	3.83	0.19	1.62	0.07	0.18	0.56	53.58

15.4 Mine Design

15.4.1 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment being a 246 tonne haul truck (CAT793) with a canopy width of 8.30 m. For double lane traffic, industry best-practice is to design a travelling surface of at least three times the width of the largest vehicle. Ramp gradients are established at 10%.

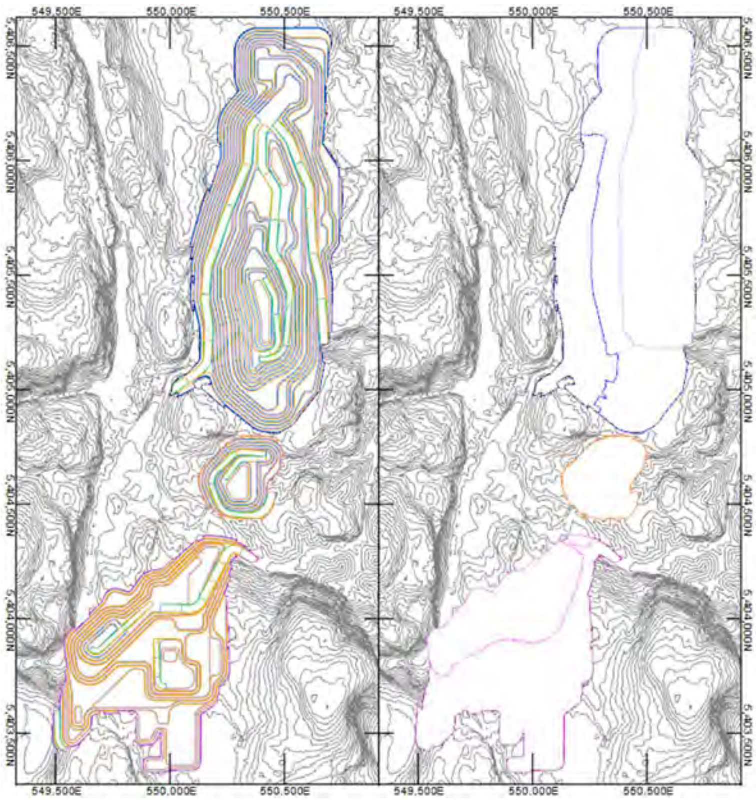
A shoulder barrier or safety berm on the outside edge will be constructed of crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the truck tire is 1.8 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.1H:1V. A water drainage ditch planned on the highwall will capture run-off from the pit wall surface and assure proper drainage of the running surface. The ditch will be 1.4 m wide. To facilitate drainage of the roadway a 2% cross slope on the ramp is planned.

The double lane ramp width is 35.0 m wide and the single lane ramp is 22 m wide. Single lane ramps are introduced in the pit bottom when the benches start narrowing and when the mining rates will be significantly reduced.

15.4.2 Open Pit Mine Design Results

The Marathon Deposit is mined with three (3) pits as presented in Figure 15.2. The pits are aligned along strike over 3,300 m. The North pit is 1,800 m long and 700 m wide and reaches a depth of 320 m. The Central pit is 450 m long and 350 m wide and reaches a depth of 135 m. Finally, the South pit is 1,100 m long and 700 m wide and reaches a depth of 180 m. The North pit design has two exits: one to the east and one to the west, providing access to the pushbacks and to shorten haul distances to the crusher and waste dumps. The west ramp system connects with the east ramp system at a plateau at elevation 170 m (north pit). The ramp system introduces several switchbacks in several instances to reduce the overall slope angle.

Figure 15.2: Final Pit Designs



Mining Services	Final Pit Design	Author: Peter Harquail Date: March 2023

15.5 Dilution and Ore Loss

The Mineral Reserve estimate is based on the final pit design presented above. The Proven and Probable Mineral Reserves are inclusive of mining dilution and ore loss. Isolated ore blocks are treated as an ore loss and represent 0.33% in terms of tonnage. The dilution tonnage represents 9.1% of the ore tonnage before dilution and the dilution grade is estimated from the block model and corresponds to the average grade of the dilution skin. Table 15.7 presents a Resource to Reserve reconciliation.

Table 15.7: Resource to Reserve Reconciliation

Resource to Reserve Reconciliation	Tonnage (kt)	Ag (g/t)	Au (g/t)	Cu (%)	Pd (g/t)	Pt (g/t)
Ore Before Dilution	116,455	1.75	0.07	0.22	0.68	0.21
Less: Ore Loss (Isolated Blocks)	383	1.39	0.06	0.1	0.34	0.14
Ore Before Mining Dilution	116,072	1.75	0.07	0.22	0.68	0.21
Add: Mining Dilution	11,587	0.99	0.04	0.18	0.36	0.12

16. MINING METHODS

16.1 Introduction

Mining of the Marathon Project is planned with six phases and three separate pits. The summary of each of the mining phases and pits is summarized in Table 16.1 and depicted in Figure 16.1. The objective of pit phasing is to improve the economics of the Project by feeding the mill with higher grade material during the earlier years and/or delaying waste stripping until later years. In addition, the intent is to bring palladium production early in the mine life to partially mitigate the potential impact of long-term Pd prices associated with the reduction in auto-catalyst use. Internal phases are designed to have a lower stripping ratio then the subsequent phases. The Project is split into the three separate pits: North Pit, Center Pit, and South Pit. The North Pit has three phases, the South Pit has two phases with the first phase being a quarry for construction waste rock, while the Center Pit has a single phase.

Over the mine life, the Project will produce 127.7 Mt of ore and 335.2 Mt of waste at an overall stripping ratio of 1 to 2.6 (ore to waste). Some 34.8 Mt of generated waste will be potentially acid generating (“PAG”) and must be stored separately and with different environmental considerations.

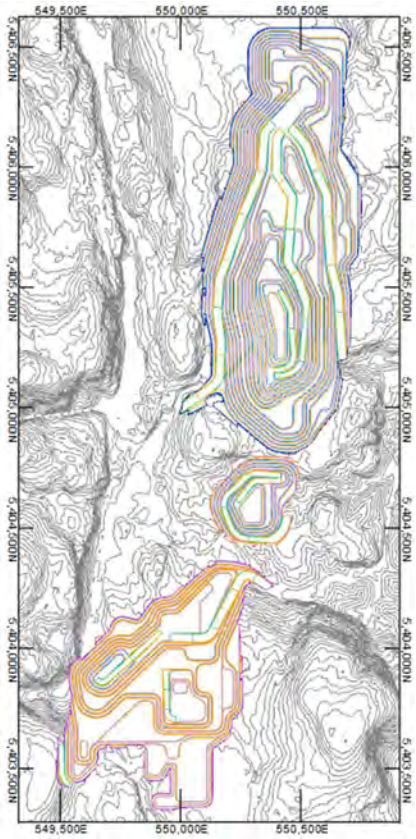
The pit designs are based on the optimized whittle shells described in Section 15 and created with the parameters outlined in Subsection 16.1.1.

Table 16.1: Pit Phase Design Summary

	Units	Grand Total	North Pit				Center Pit	South Pit		
			Phase 1	Phase 2	Phase 3	Total	Total	Quarry	Phase 2	Total
Total[1]	000 t	**462,881**	106,694	99,641	148,744	**355,079**	**18,056**	24,466	65,280	**89,746**
Waste Rock	000 t	**335,224**	65,235	77,231	113,234	**255,700**	**11,022**	23,417	45,085	**68,502**
Strip Ratio	W:O	**2.63**	1.57	3.45	3.19	**2.57**	**1.57**	22.32	2.23	**3.22**
Ore	000 t	**127,658**	41,460	22,410	35,510	**99,380**	**7,034**	1,049	20,195	**21,244**
Ag Grade	g/t	**1.66**	1.47	1.76	1.93	**1.7**	**1.55**	1.66	1.53	**1.54**
Au Grade	g/t	**0.07**	0.07	0.07	0.06	**0.07**	**0.06**	0.06	0.09	**0.08**
Cu Grade	%	**0.21**	0.25	0.22	0.22	**0.23**	**0.21**	0.07	0.11	**0.10**
Pd Grade	g/t	**0.63**	0.72	0.55	0.51	**0.6**	**0.5**	0.68	0.79	**0.78**
Pt Grade	g/t	**0.20**	0.2	0.17	0.16	**0.18**	**0.15**	0.24	0.31	**0.31**

[1] Total Material Moved.

Figure 16.1: End of LOM Pit Layout



Mining Services	**Pit Design: End of Mine Life**	**Author: Peter Harquail** **Date: March 2023**

16.1.1 Open Pit Optimization

All phases, except for the South Pit Phase 1 which acts as a quarry, use specific Whittle shells to optimize shape and layout to ensure the optimal ore and waste is mined to achieve maximum individual NPV. Table 16.2 depicts the nomenclature of each of the pits and the Whittle shell that guided their design. For more details on the individual pit shells, refer to Section 15.

Table 16.2: Pit Shell Hierarchy

Phasing	Whittle Shell
North Pit Phase 3	Shell #22
→ North Pit Phase 2	Shell #13
→ North Pit Phase 1	Shell #8
South Pit Phase 2	Shell #22
→ South Pit Phase 1	N/A
Center Pit Phase 1	Shell #22

Whittle shells are made without ramps, consideration of minimum mining width or ramp access. These details are included subsequently in the pit's designs. Whittle shells are designed with a lower overall slope angle ("OSA") to account for these changes. Even with these accommodations, it is typical for the inventory of the pits to fluctuate from the Whittle designs and the actual design. Ranges of fluctuation are typically within +/-5%. Table 16.3 depicts the comparison between the inventory calculated by Whittle and the inventory in the designed pits.

Table 16.3: Shell and Design Comparison

	Shell	Designs
Total Tonnage (kt)	471,182	**462,881**
Waste Tonnage (kt)	342,547	**335,224**
Strip Ratio (W:O)	2.66	**2.63**
Ore Tonnage (kt)	128,635	**127,658**
Ag Grade	1.68	**1.66**
Au Grade	0.07	**0.07**
Cu Grade	0.21	**0.21**
Pd Grade	0.64	**0.63**
Pt Grade	0.20	**0.20**

16.1.2 Open Pit Design Criteria

The open pit designs are established with the following design criteria.

16.1.2.1 Geotechnical Parameters

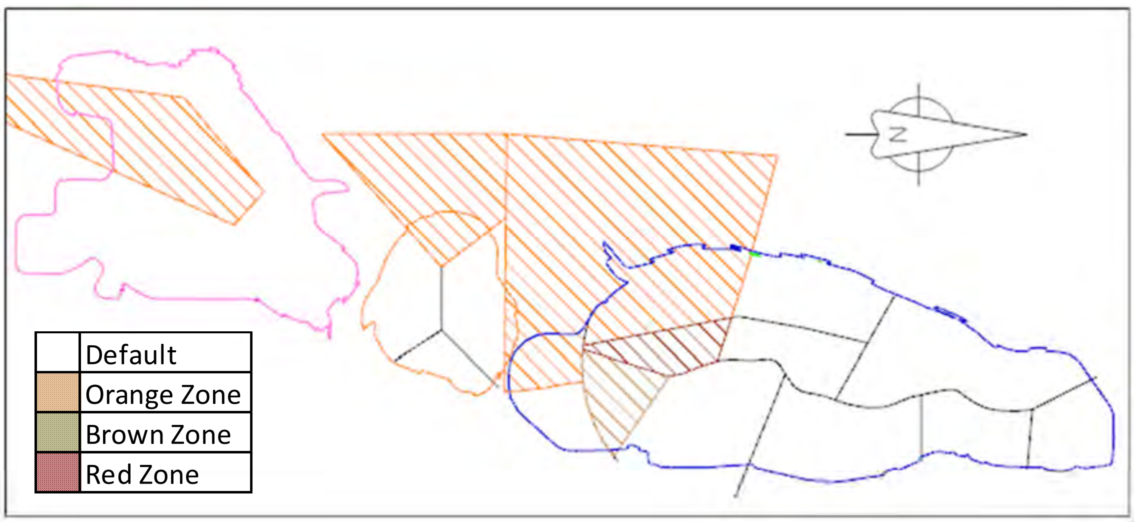
Table 16.4 summarizes the geotechnical parameters used in the design of the pit walls. Figure 16.2 outlines the geotechnical zones overlaying the mine designs with parameters summarized in a geotechnical assessment report (KP, 2020) in Figure 16.3.

Geotechnical berms were not used in the pits. Temporary walls between phases are assumed to have the same parameters as final walls as pre-split is planned for all mining.

Table 16.4: Geotechnical Design Parameters Summary

5 m Adjusted Blocks				
Design Sector	**Default**	**Orange Zone**	**Brown Zone**	**Red Zone**
Final Vertical Bench Height	20.0	20.0	20.0	20.0
Bench Face Angle	75.0	75.0	70.0	65.0
Avg. Catch Berm Width	8.70	10.00	8.70	8.70
Horizontal	14.06	15.36	15.98	18.03
Vertical	20.00	20.00	20.00	20.00
OSA (crest-to-crest)	54.89	52.48	51.38	47.97

Figure 16.2: Application of Geotech Zones



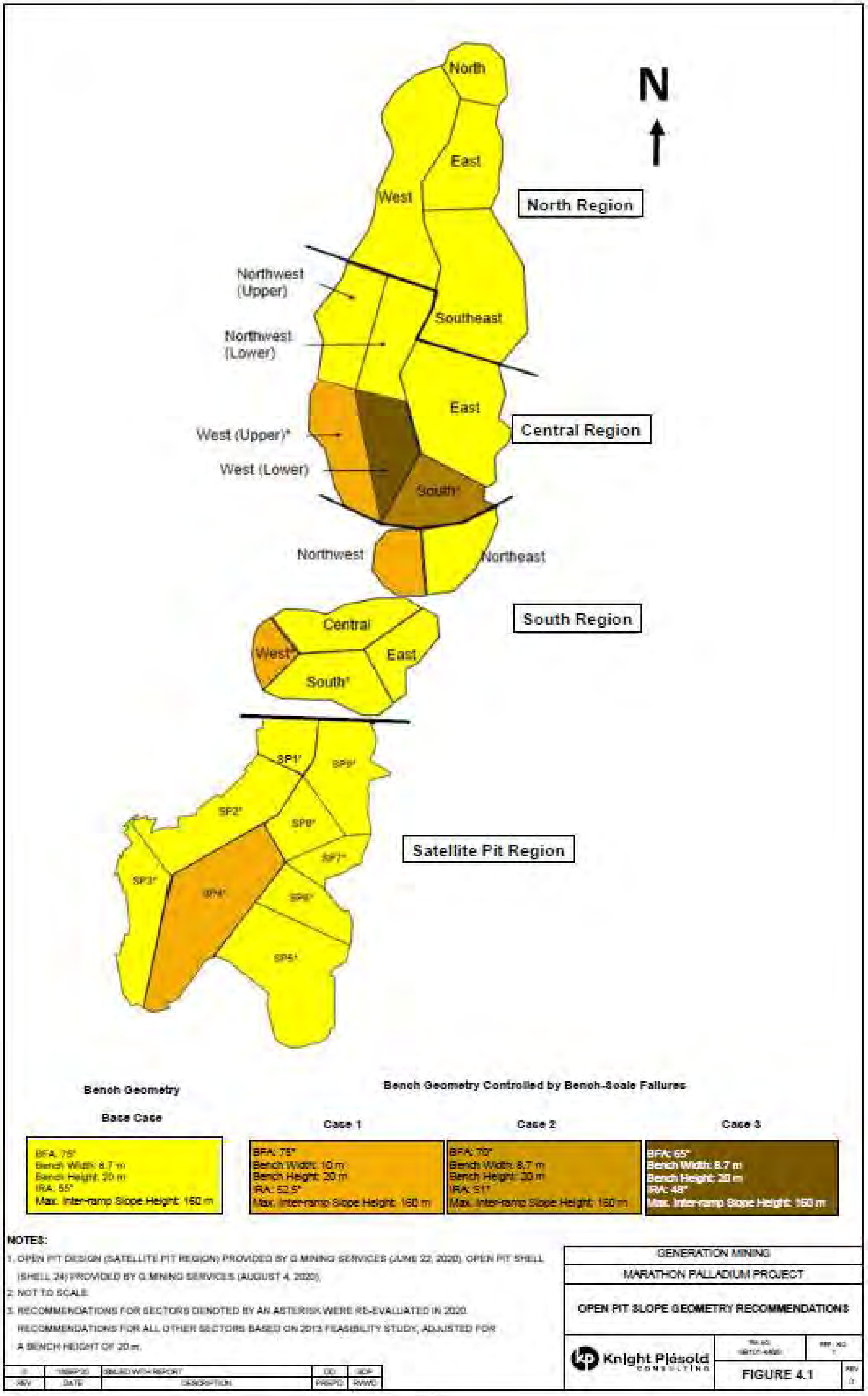
Mining Services

Geotech Zones

Author: Peter Harquail

Date: March 2023

Figure 16.3: Knight Piésold Geotechnical Recommendations



***Source:** KP, 2020*

16.1.2.2 Ramp and Road Design

Ramp designs are shown in Figure 16.4 and Figure 16.5 for the single lane and double lane ramps, respectively. The ramps are designed specifically for the primary hauler, the CAT 793. In accordance with SME Standard of 3.5 m x and 2.0 m x ramp width of the vehicle operating width. The operating width of the CAT 793 is 8.3 m. The ramp includes adequate distance for the vehicles to operate and includes a safety berm on the pit side and a drainage ditch on the wall side. The safety berm is designed to be at least half the height of the tallest tire to be used on site, in this case the tires of the CAT 793.

Figure 16.4: Single-Lane Ramp Haul Road Profile

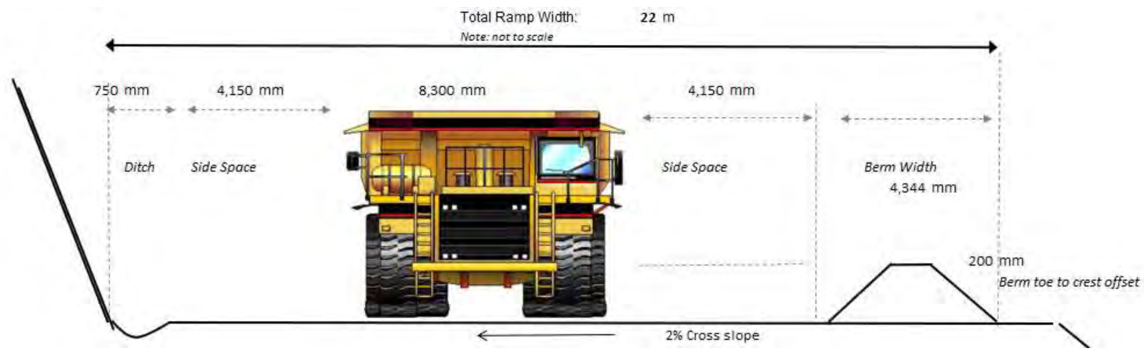
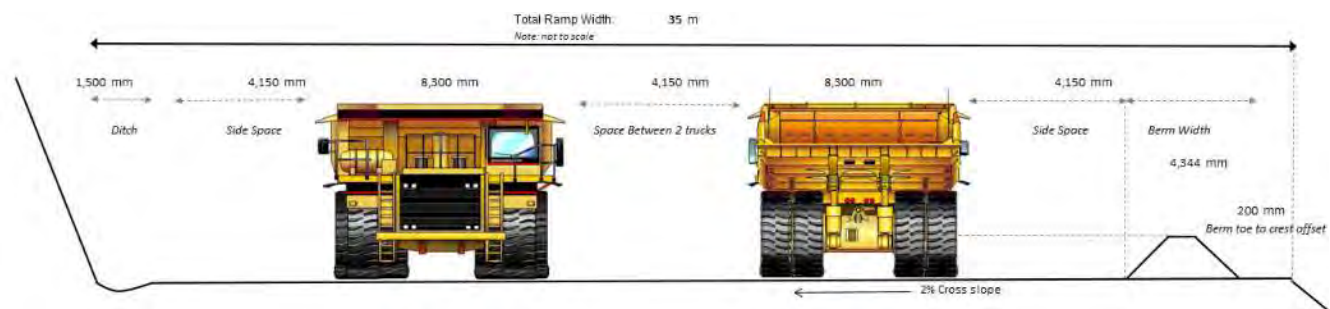


Figure 16.5: Double-Lane Haul Road Profile



16.1.2.3 Mine Design Parameters

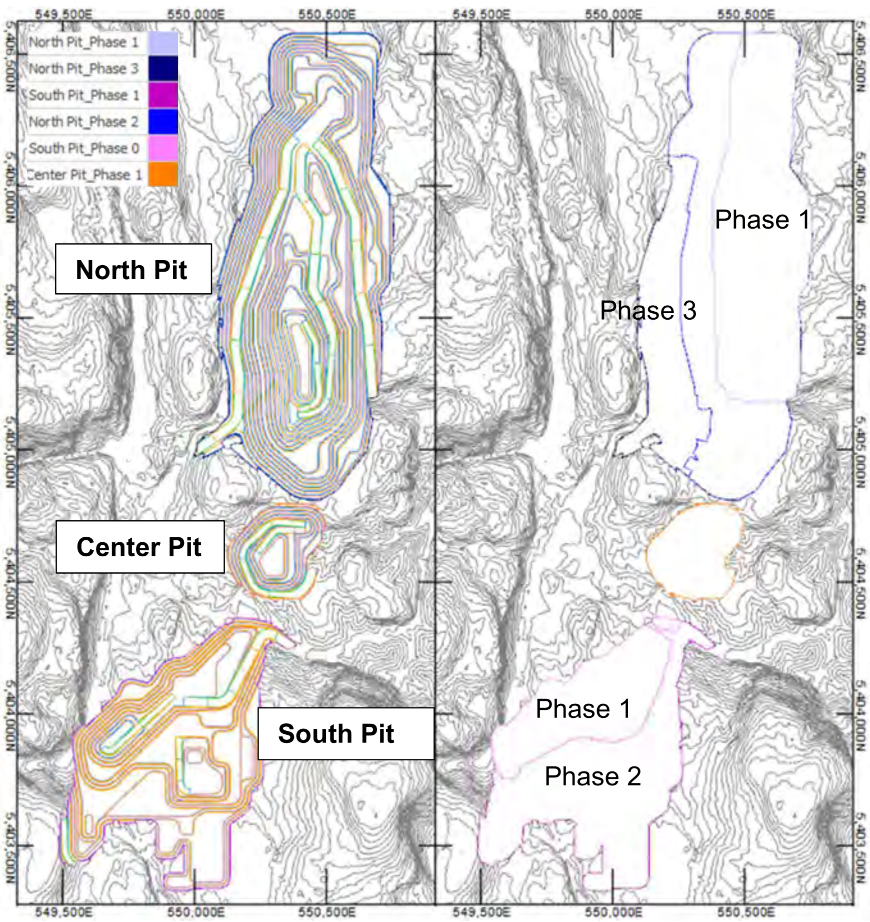
A minimum mining width of 30 m was used when controlling the minimum width that can be safely and optimally mined between phases or at the bottom of a pit. This value is determined by the operating width of the primary shovel, the width required for a double lane road with berm and the area required for the CAT 793 to safely complete a three-point turn.

Single lane ramps were used in the bottom approximately 50 m of the pits to reduce stripping and to capture more ore. Single lane ramps can cause bottle necks in the fleet productivity and this method is only used sparingly at the bottom of the pits. Reductions in productivity stemming from single lane ramping is captured in the mine ramp down in production or compensated by mining in other pits or phases. To attain additional ore at the bottom of the pit, 5-10 m box cuts are used.

16.1.3 Open Pit Designs

The final designs are depicted in Figure 16.6 along with the phase limits of each of the phases. Ramps for the pits are designed to exit either on the west or east of the pits to better access the primary waste dump and the ore crusher.

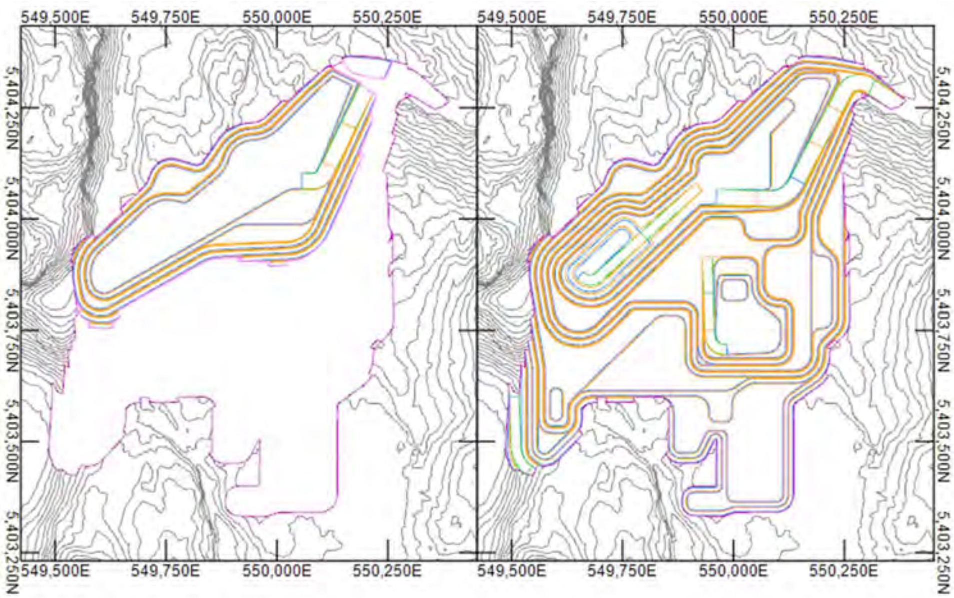
Figure 16.6: End of LOM Pit Layout and Phase Limits



Mining Services	**Pit Layout: End of Mine Life (Left), Phase Limits (Right)**	**Author: Peter Harquail** **Date: March 2023**

The South Pit consists of two phases both exiting to the north side of the pit to access the crusher and the East Dump (or Mine Rock Storage Area “MRSA”). Phase 1 is primarily a quarry to source construction rock. Phase 2 has a depth of 160 m and is approximately 1.0 km long at its longest and is 750 m wide.

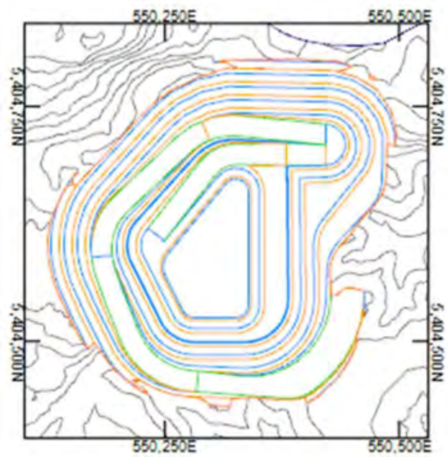
Figure 16.7: South Pit Phases 1 & 2



Mining Services	**South Pit: Phase 1 (left), Phase 2 (right)**	**Author: Peter Harquail** **Date: March 2023**

The Center Pit consists of one phase. The ore lays predominately on the east side dipping west. Ramping is planned to take advantage of the ore on the east wall and minimize waste mined on the west wall. The Center Pit is 130 m deep with a length of 500 m and width of 350 m.

Figure 16.8: Center Pit Phase 1



Mining Services	Center Pit: Phase 1	Author: Peter Harquail Date: March 2023

The North Pit is the largest of the pits and consists of three nested phases. The east wall is shared among all three phases as well as the majority of the east wall ramp.

Phase 1 defines the east wall of the North Pit and has a ramp that will be shared among all the phases.

Phase 2 maintains the east wall ramp and adds an additional ramp that runs along the west wall. There is a switchback at elevation 190 m where the two ramps meet before continuing down to the bottom of the phase.

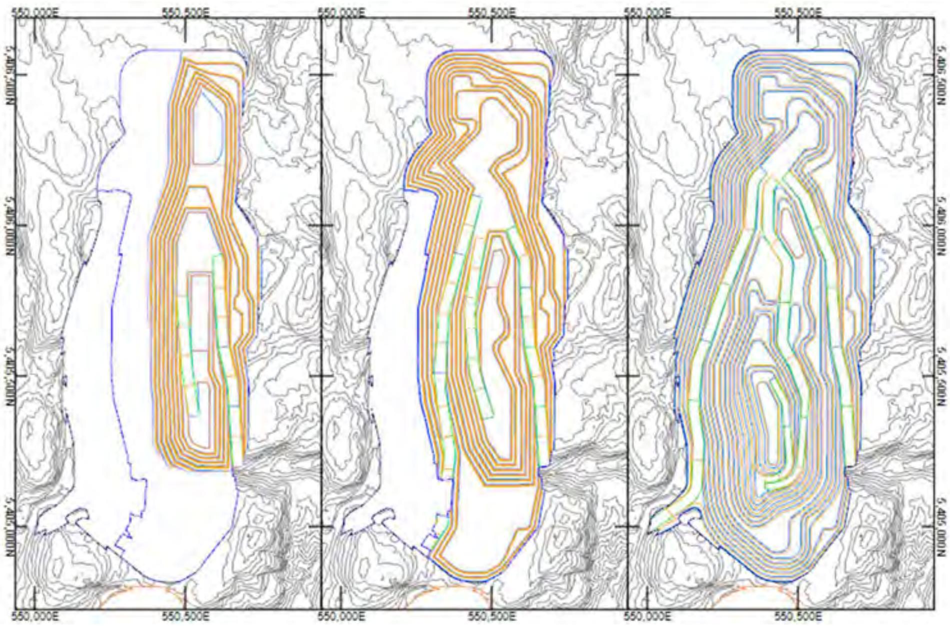
Phase 3 is the final phase and introduces the final west wall ramp. This ramp joins the east ramp wall to merge into the final ramp that will drive to the bottom of the pit.

North Pit Phase 1 has a depth of 170 m, length of 1.5 km and a width of 350 m.

North Pit Phase 2 has a depth of 180 m, length of 1.8 km and a width of 500 m.

North Pit Phase 3 has a depth of 350 m, length of 1.8 km and a width of 700 m.

Figure 16.9: North Pit Phases 1,2 & 3



Mining Services	North Pit: Phase 1 (left), Phase 2 (center), Phase 3 (right)	Author: Peter Harquail Date: March 2023

16.1.4 Overburden and Waste Rock Storage

A total of 343 Mt of waste rock is produced over the mine life. The waste is split into three categories of PAG, Non-Potentially Acid Generating ("NAG") Rock and Overburden. Each material has different dumping requirements and have unique dumps for storage. Table 16.5 depicts the design parameters of each of the dumps.

Table 16.5: Overburden and Dump Design Parameters

Waste Dump	Avg. Catch Bench Width (m)	Pile Face Angle (deg)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)
East Waste Dump (NAG)	10	28.6	2.3:1	350	200
South In-pit Dump (NAG+PAG)	N/A	N/A	N/A	300	160
South Dump Extension (NAG)	10	28.6	2.3:1	390	140
Center In-pit Dump (NAG+PAG)	N/A	N/A	N/A	310	160
North In-Pit Dump (NAG)	N/A	N/A	N/A	350	180
Overburden Pile (OVB)	10	26	2.6:1	410	90

Table 16.6 depicts the various dumps, their inventories and percentage filled. In parenthesis by the name of the dump is the type of material that the dump will accept. Figure 16.10 depicts a site view of the dumps and their nomenclature (Note: showing final dumps with max capacity). Refer to Subsection 16.2.3 - Surface Schedule for dumps at specific periods.

The East Dump (MRSA) is the largest NAG dump in the Project. The East Dump contains two accesses on the west side, one for the North Pit and one for the South / Center Pit. The ramp along the west side joins both entrances and will also act as the primary access route between the three pits on the east side. The explosive magazines are located further east of the East Dump. To access the magazines, the dump must be climbed, traveled across and descend on the opposite side. This method required less road building then alternative scenarios. The East Dump was designed to impact only two sub-watersheds. Diversion and / or containment structures will be constructed in the valley to the east to manage water run-off.

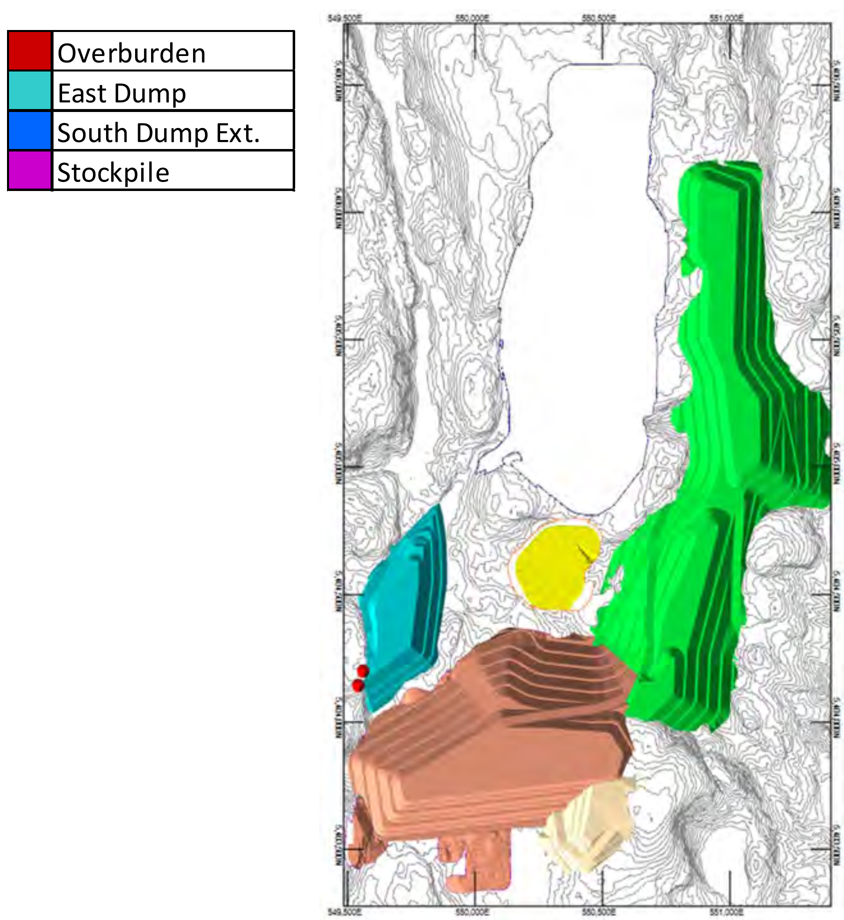
The South pit will be used as an in-pit dump (South Pit In-pit Dump) with both phases being back-filled after completion of the pit. This in-pit dump can be used as a PAG material storage with the material being fully covered with water prior to closure. The South Dump Extension is an extension of the East Dump that extends over the South Pit In-pit Dump. This dump would be usable exclusively for NAG material. The East ramp of the South Dump Extension will connect with the East Dump ramp to allow access from the North Pit and contain a ramp on the west side to allow access from the stockpile area.

The Central In-pit Dump is similar to the South In-pit Dump in which it can take NAG and PAG material, with the PAG material being covered with water at closure. In the current designs, only PAG material will be dumped at this location. The North In-pit Dump is primarily for NAG material with some minor quantities of PAG material placed at depth to be covered with water at closure. The Overburden pile is located south of the East Dump (Figure 16.10). The overburden material will be used for progressive reclamation and will be depleted during the final closure of the mine.

Table 16.6: Surface Storage Capacities

Waste Dump	Capacity (Mt)	Capacity (Mm^3)	Surface Area (ha)
East MRSA (NAG)	131.3	54	104.9
South In-pit Dump (NAG+PAG)	69.1	28.8	N/A
South MRSA Extension (NAG)	80.8	33.7	50.6
Center In-pit Dump (NAG+PAG)	13.9	5.8	N/A
North In-pit Dump (NAG)	15.0	6.7	N/A
Overburden Pile (Overburden)	3.7	2.7	17.4
TSF Construction (NAG)	64.8	N/A	N/A
TSF Internal Storage (PAG)	17.0	N/A	N/A
Total	**395.6**	**131.7**	**172.9**

Figure 16.10: Overburden and Waste Rock Storage Layout



Waste Storage: With South In-pit Dump (left), Without South In-pit Dump *shown for clarity* (right)

Author: Peter Harquail

Date: March 2023

16.1.5 Ore Stockpile

The ore stockpile in Figure 16.10 represents the maximum capacity required for the LOM. The stockpile will be used to store marginally lower grade material and to allow for the preferential plant feed of higher margin ore. The ore stockpile tonnage will fluctuate over the mine life. All ore placed on the stockpile is included in the LOM plan and will be milled over the LOM. The majority of the material stockpiled will be low-grade ore with some minor medium- and high-grade ore being stockpiled and depleted during the pre-production period. Different ore groupings will be stored separately within the stockpile to manage in-dump dilution. Table 16.7 depicts the design parameters for the stockpile.

The stockpile is accessible from the access road to the South Pit as well as along the crusher ramp. Material to be rehandled will require a loading unit, typically a front-end loader, and haul trucks to bring the material from the stockpile up to the crusher via the crusher ramp. Over the LOM, the peak inventory of stockpiled material will be approximately 10 Mt in Year 7 of mining and a total of 18 Mt of ore will be stored and reclaimed from the stockpile.

Table 16.7: Stockpile Design Parameters and Capacities

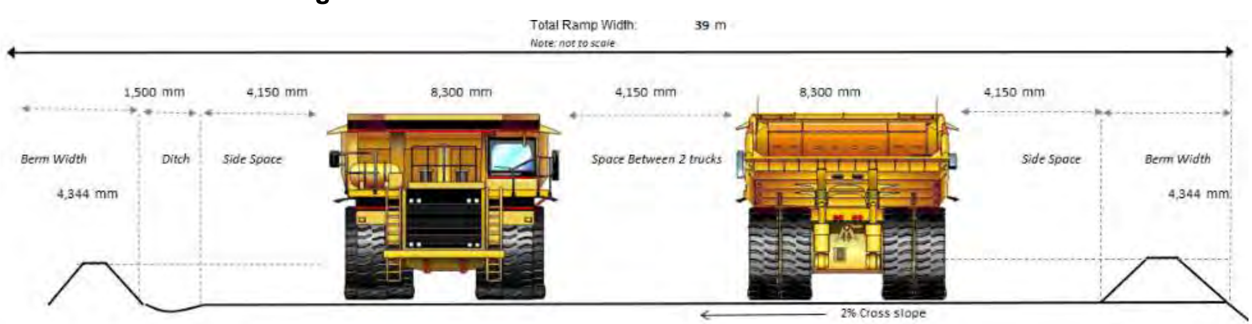
Ore Stockpiles	Catch Bench Width (m)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)	Max Capacity (Mt)
Stockpile Max Capacity	10	2.3:1	360	70	14.1

16.1.6 Surface Mine Haul Roads and Access

This section refers only to the haul and access roads accessible by haul trucks and heavy mine equipment.

Haul roads for the CAT 793 are designed with berms on both sides of the road and include a drainage ditch on one side. All surface roads are double lanes. The total width of a surface road is 39 m (Figure 16.11).

Figure 16.11: Double Lane Surface Haul Road Profile



The Mine Access Road is the road to the tailings area from the open pits and largely parallels the conveyor gallery. This section also includes the fill and cut required for the conveyor from the crusher pad to the processing plant.

The Crusher Ramp is the largest fill requirement in the Project and consists of a ramp up to the crusher located at a landing near the top of the hill west of the Center Pit. The ramp climbs approximately 65 m in elevation at a 10% grade. This road also connects the North and South Pits.

The pit access road allows access from the Center Pit to the crusher ramp road. This road runs between South Pit and Center Pit and will be the primary ore haulage route for material from the Center Pit.

16.2 Mine Production Schedule

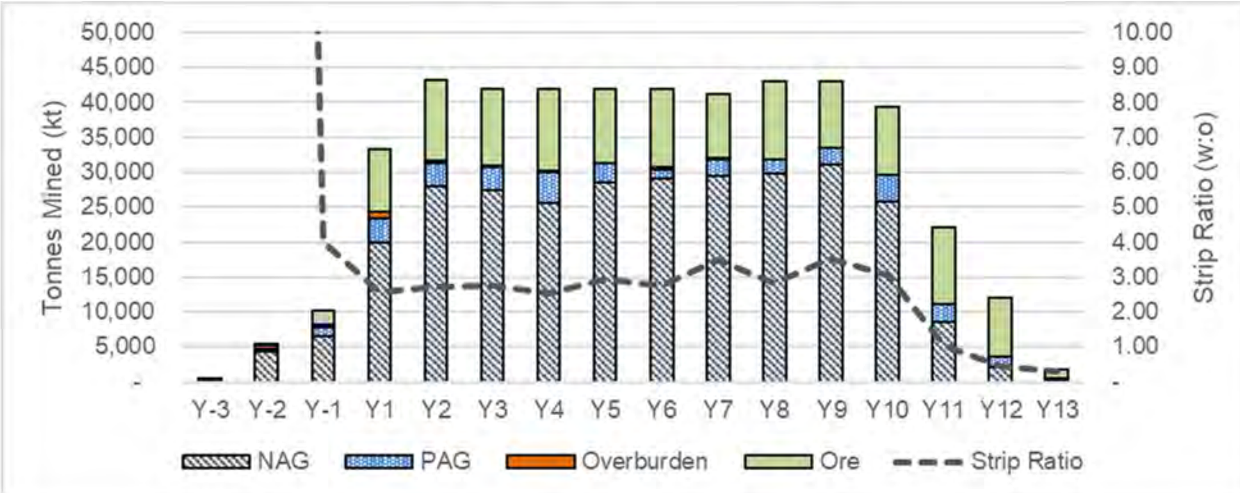
The mining and milling schedules were optimized by Minemax to maximize the Project NPV. The optimization includes mine sequencing and mining rate, stockpile usage and rehandling, and fleet usage. The results from Minemax were then further detailed, and a Deswik schedule was used to accurately track material movements, stockpile inventory, mill blending, block mining, waste movements and equipment usage/movements.

16.2.1 Mining Schedule

The total mining of the Project takes place over ~15 years of mining, including a construction and ramp up period of 2.5 years and 12.5 years of operations. Peak mining rate is 43 Mt/y (118,000 t/d) at an average stripping ratio (waste:ore) of 2.6 : 1. Figure 16.12 outlines the production schedule by material type and the stripping ratio. Ore feed is consistent through the mine life with no periods of significant stripping required to meet mill requirements. PAG material is mined throughout the mine life and is placed in specific locations, separately from the NAG material.

A detailed table that includes mined grades and materials can be found in Table 16.8.

Figure 16.12: Mine Production Schedule



Mining is split between large or wide ore domains and narrow ore domains. To lower dilution, narrow ore domain material is mined with a dedicated smaller fleet to better control dilution. Figure 16.13 depicts the material mined by the metallurgical groupings. Materials that are large ore domains are depicted in dark red while those that are narrow body ore domains are depicted in blue. Most mining will be done with the primary large-scale mining fleet in the large ore domains.

Figure 16.13: Material Mined by Domain Groupings

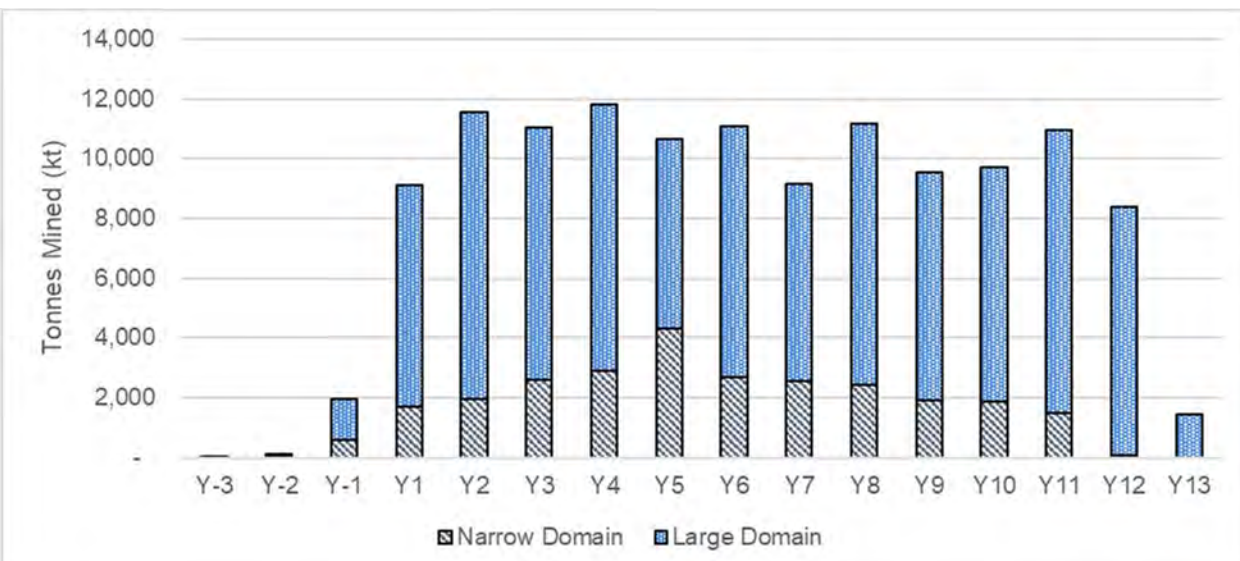
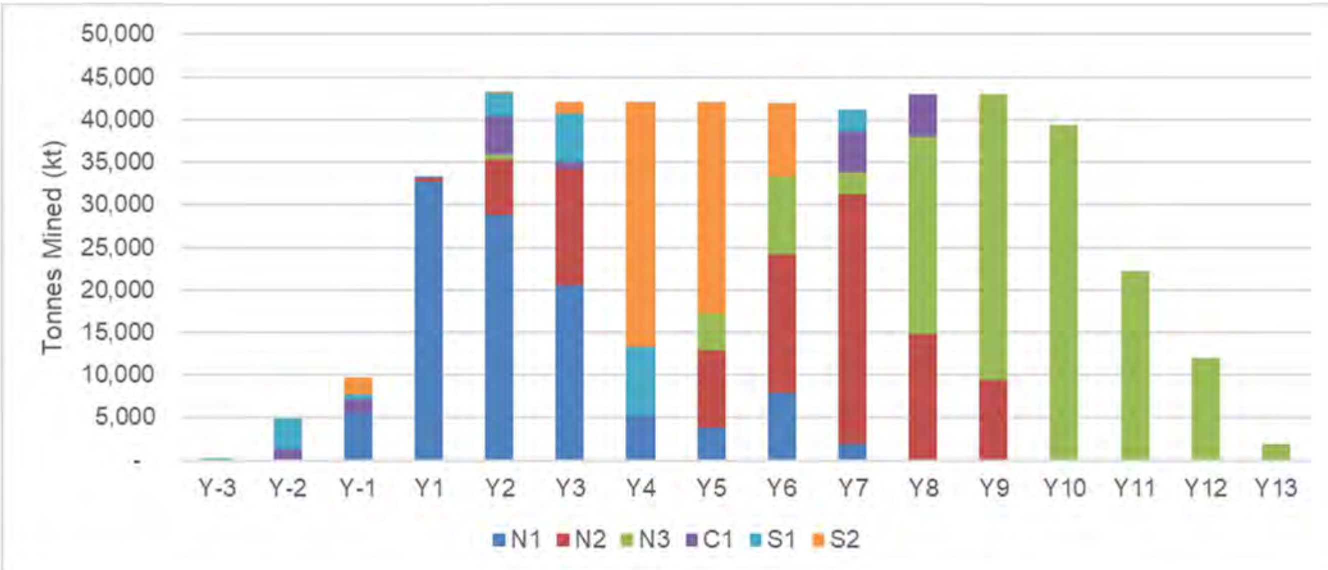


Figure 16.14 depicts the mining in each of the phases in the Project. The North Pit (N) is the largest pit in the Project and its three phases represent the bulk of mining, the Center Pit (C) and South Pit (S) are spread out over the mine life to fill in ore requirements during stripping periods. The South Pit and Center

Pit are available for in-pit dumping once completed. The South Pit is mined out by Year 6 and the Center Pit by Year 8.

Figure 16.14: Material Movements from Pit



16.2.2 Milling Schedule

The mill life for the Project is 12.5 years along with a 6 month pre-commercial production period. The peak milling capacity is 10.1 Mt per year (after year-2 of operation when increased electrical power capacity is available for the processing plant power) with a ramp up of 8.6 Mt in the first year of milling before reaching peak milling rate. Mill feed is kept consistent with ore direct from the pits and rehandled ore to cover reduction in ore generation from the pit. Milling is optimized to maximize NPV for the Project and to minimize rehandled. Medium and high-grade ore are prioritized to maximize revenue. The last year of milling consists mostly of low-grade material that was previously stockpiled. Figure 16.15 outlines the mill feed by source and the resulting NSR of material to the mill.

Figure 16.16 depicts the stockpile inventories by period and grade bin. Medium and high-grade ore is only stockpiled for the first two years of mining until it is rehandled to the mill as higher grade ore is prioritized. The peak stockpile capacity is approximately 10 Mt. All material is milled by the end of project life.

Table 16.9 and Figure 16.9 depicts the metals produced by the mill each year. Over the mine life, a total of 548 M lbs of copper, 4.5 Moz of silver, 204,000 oz of gold, 607,000 oz of platinum and 2.3 Moz of palladium will be produced (recovered to concentrate).

Table 16.8: Detailed Mine Production Schedule

		Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Total
Total Tonnage	**Mt**	**0.3**	**4.9**	**9.8**	**32.5**	**43.0**	**41.7**	**41.8**	**41.9**	**41.5**	**41.0**	**43.0**	**43.0**	**39.4**	**22.2**	**12.1**	**1.9**	**459.7**
Total Waste	Mt	0.3	4.8	7.8	23.4	31.4	30.7	30.0	31.2	30.4	31.8	31.8	33.5	29.6	11.2	3.7	0.4	**332.1**
Overburden	Mt	0.0	0.5	0.4	0.9	0.2	0.3	0.2	0.1	0.4	0.1	0.0	0.0	0.0	0.0	0.0	0.0	**3.2**
NAG	Mt	0.3	4.4	6.6	19.9	28.1	27.5	25.7	28.6	29.1	29.4	29.9	31.2	25.8	8.5	2.1	0.2	**297.3**
PAG	Mt	0.0	0.4	1.2	3.5	3.3	3.2	4.3	2.6	1.3	2.4	1.9	2.3	3.8	2.7	1.6	0.3	**34.8**
Strip Ratio	W:O	--	66.2	4.0	2.6	2.7	2.8	2.5	2.9	2.7	3.5	2.8	3.5	3.0	1.0	0.4	0.3	**2.60**
Ore Tonnage	Mt	0.0	0.1	1.9	9.1	11.6	11.1	11.8	10.6	11.1	9.2	11.2	9.5	9.7	10.9	8.4	1.5	**127.7**
Cu Grade	%	0.07	0.07	0.20	0.23	0.23	0.23	0.15	0.13	0.19	0.19	0.21	0.24	0.21	0.23	0.25	0.37	**0.21**
Ag Grade	g/t	1.33	1.33	1.70	1.27	1.29	1.48	1.52	1.68	1.54	1.60	1.84	1.71	1.93	2.04	2.00	2.57	**1.66**
Au Grade	g/t	0.04	0.04	0.07	0.07	0.06	0.07	0.08	0.08	0.08	0.07	0.06	0.07	0.06	0.06	0.07	0.08	**0.07**
Pt Grade	g/t	0.13	0.13	0.22	0.20	0.19	0.19	0.25	0.27	0.25	0.18	0.14	0.17	0.16	0.16	0.18	0.17	**0.20**
Pd Grade	g/t	0.41	0.41	0.66	0.75	0.70	0.67	0.69	0.75	0.75	0.57	0.48	0.57	0.52	0.52	0.52	0.55	**0.63**

Figure 16.15: Mill Production

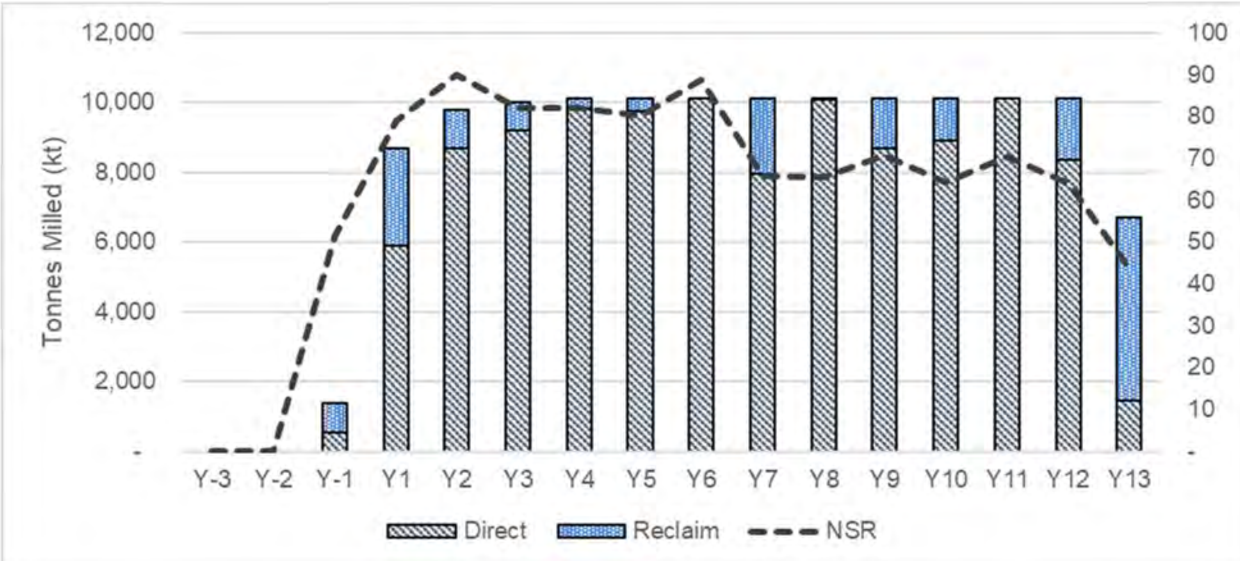


Figure 16.16: Stockpile Inventory

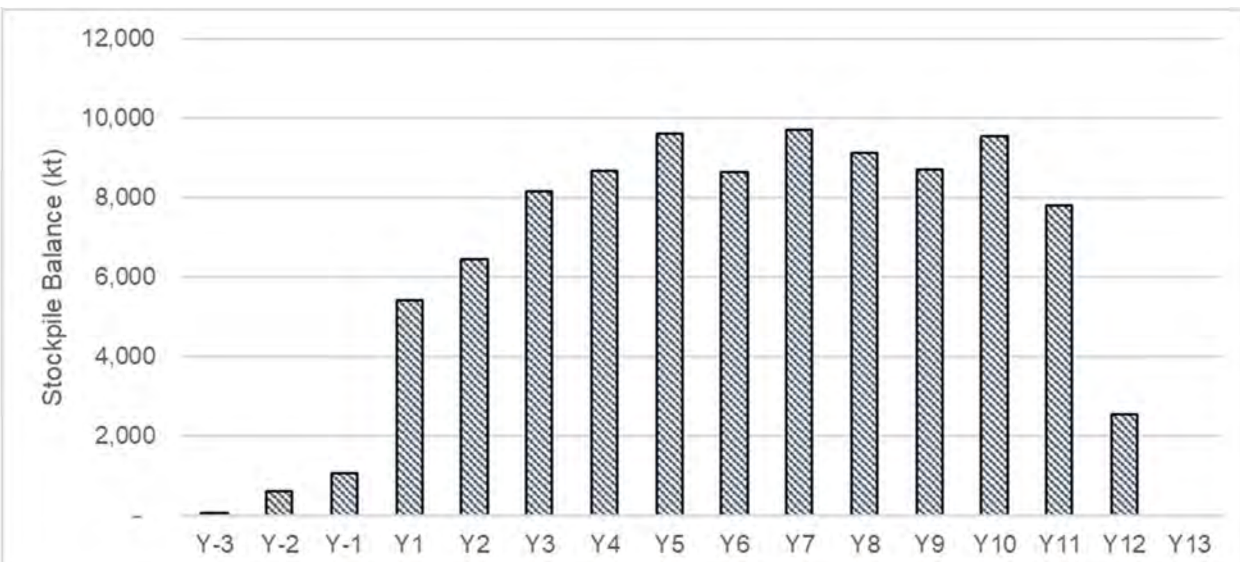


Figure 16.17:Metals Produced

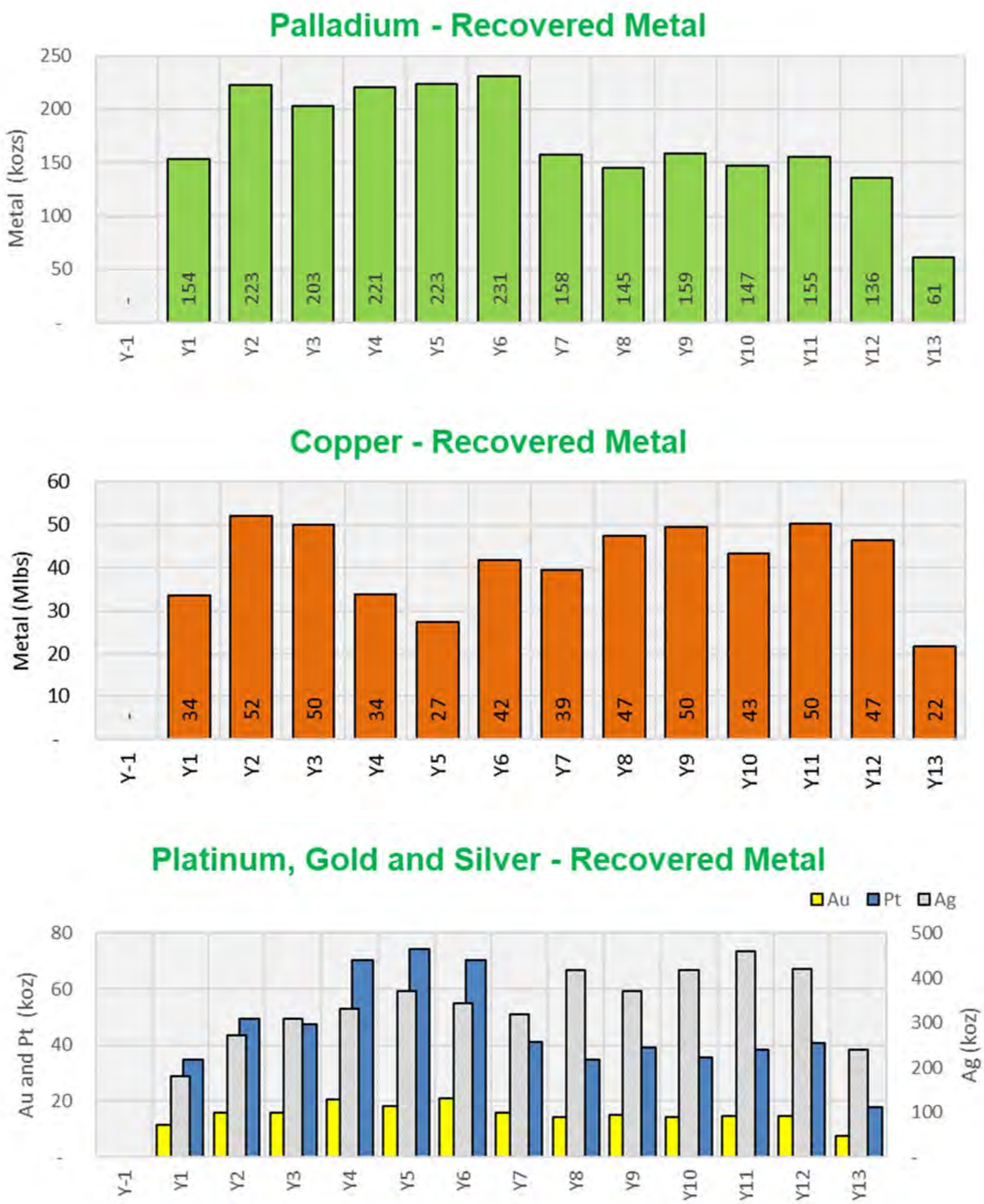


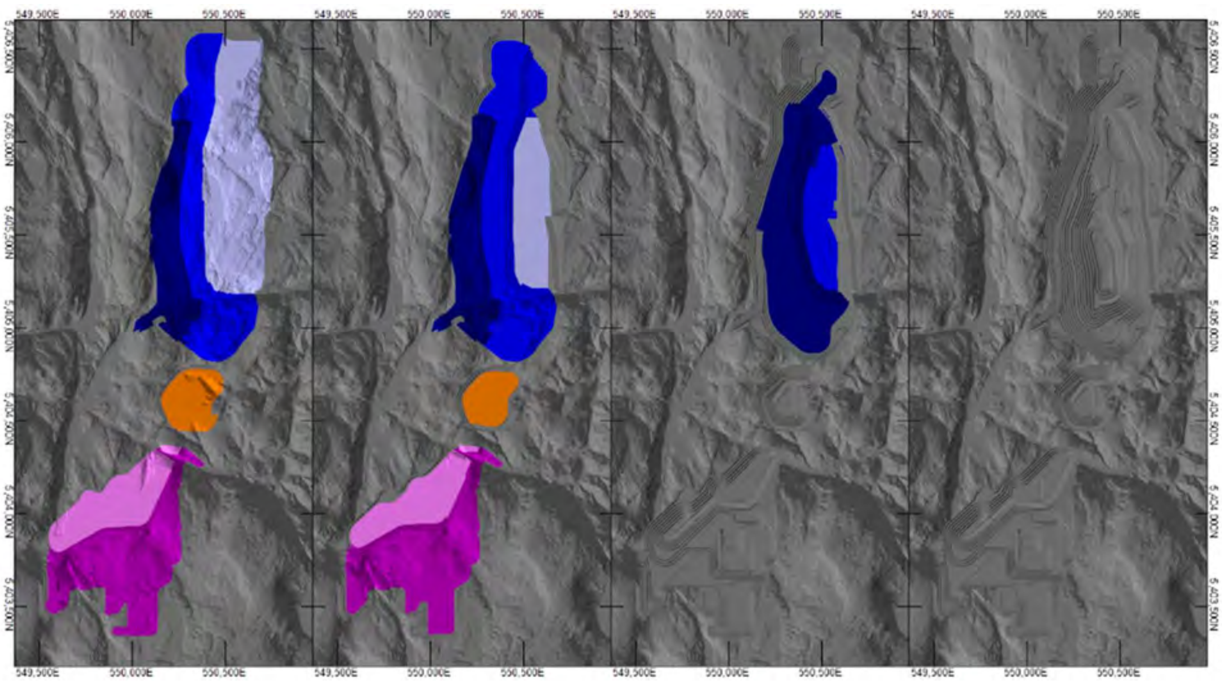
Table 16.9: Detailed Milling Schedule

Physicals Summary (Ops)		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Tonnage Milled	kt	125,047	7,457	9,775	10,028	10,120	10,120	10,120	10,120	10,120	10,120	10,120	10,120	10,120	6,708
Cu Con. Production	k dmt	1,368	86	128	125	101	92	105	102	113	117	107	118	112	61
Head Grades															
Cu		0.21	0.22	0.26	0.24	0.16	0.13	0.20	0.19	0.23	0.24	0.21	0.24	0.22	0.16
Ag	g/t	1.66	1.29	1.39	1.49	1.54	1.68	1.57	1.51	1.89	1.68	1.89	2.07	1.89	1.64
Au	g/t	0.07	0.07	0.07	0.07	0.09	0.08	0.09	0.07	0.06	0.06	0.06	0.06	0.06	0.05
Pt	g/t	0.20	0.20	0.21	0.20	0.27	0.28	0.27	0.17	0.15	0.17	0.15	0.16	0.17	0.12
Pd	g/t	0.62	0.74	0.80	0.71	0.77	0.77	0.80	0.55	0.51	0.55	0.51	0.54	0.48	0.32
Contained Metal															
Cu	M lbs	573	37	55	53	36	29	45	42	51	53	46	54	49	23
Ag	kozs	6,693	310	438	479	502	545	512	490	614	546	615	674	615	353
Au	kozs	279	17	22	22	28	25	28	22	20	21	20	21	21	11
Pt	kozs	786	48	65	63	88	92	88	56	49	54	50	53	56	26
Pd	kozs	2,511	178	252	229	249	252	260	179	164	180	167	176	155	70
Recovered Metal															
Cu	Mlbs	537	34	52	50	34	27	42	39	47	50	43	50	47	22
Ag	kozs	4,450	179	272	310	332	371	343	320	418	371	418	459	418	240
Au	kozs	200	11	16	16	21	18	21	16	14	15	14	15	15	8
Pt	kozs	593	35	49	47	70	74	70	41	35	39	35	38	41	18
Pd	kozs	2,213	154	223	203	221	223	231	158	145	159	147	155	136	61
Average Recoveries															
Cu		93.7	91.3	93.8	94.0	93.4	92.9	93.9	93.7	94.0	94.0	93.9	94.0	94.0	93.3
Ag		66.5	57.8	62.0	64.7	66.2	68.0	67.0	65.2	68.0	68.0	68.0	68.0	68.0	68.0
Au		71.6	69.5	71.8	71.5	74.4	73.1	74.6	71.4	70.1	70.8	70.4	70.7	70.6	67.8
Pt		75.4	73.1	75.7	75.0	80.0	80.8	80.0	73.1	71.0	72.5	71.3	72.2	72.9	68.0
Pd		88.1	86.0	88.5	88.5	88.7	88.7	88.7	88.1	87.9	88.1	87.9	88.0	87.8	87.1
Payable Metals															
Cu	Mlbs	507	31	49	47	31	25	39	37	45	47	41	48	44	20
Ag	kozs	3,106	93	147	187	233	280	240	220	306	257	313	343	308	179
Au	kozs	155	9	12	12	17	15	17	12	10	11	11	11	11	6
Pt	kozs	474	27	38	36	61	66	61	32	25	29	26	28	31	13
Pd	kozs	2,073	144	209	190	208	211	218	148	135	148	137	145	126	56
Average Payabilities															
Cu		94.5	93.1	94.2	94.1	92.9	92.3	94.3	94.7	95.3	95.4	95.1	95.4	95.3	94.1
Ag		69.8	51.8	54.0	60.4	70.2	75.5	70.0	68.8	73.3	69.2	74.8	74.7	73.7	74.8
Au		77.4	74.7	73.8	74.1	83.8	83.3	83.4	78.8	73.6	74.4	75.6	73.9	75.0	73.2
Pt		80.0	77.0	76.8	76.4	87.1	89.0	86.8	79.1	72.5	74.9	74.6	74.1	76.8	71.0
Pd		93.7	93.5	93.7	93.6	94.1	94.4	94.4	94.1	93.0	93.4	93.5	93.2	92.7	91.2
Gross Revenue															
Cu	M CAD	2,534	156	245	236	157	126	197	187	226	236	206	240	221	102
Ag	M CAD	94	3	4	6	7	8	7	7	9	8	9	10	9	5
Au	M CAD	375	21	29	29	42	37	42	30	25	27	26	27	27	13
Pt	M CAD	640	36	51	49	82	89	82	44	34	40	36	38	42	17
Pd	M CAD	5,037	349	507	461	505	513	530	361	327	360	333	351	306	135
Total	**M CAD**	**8,681**	**565**	**836**	**780**	**793**	**774**	**859**	**628**	**621**	**670**	**611**	**666**	**605**	**273**

16.2.3 Surface Schedule

Figure 16.18 depicts the working surfaces of the mine at the specified years. These images do not include dumps, stockpiles, surface works or construction.

Figure 16.18: End of Period (at Surface) (Year -1,3,8,11)



Mining Services	**EOP Surface: Left to Right (Year -1,3,8,11)**	**Author: Peter Harquail** **Date: March 2023**

16.3 Mine Operations and Equipment Selection

16.3.1 Mine Operations Approach

Mining is to be carried out using conventional open pit techniques with hydraulic shovels, wheel loaders and mining trucks in a bulk mining approach with 10 m benches. An owner mining open pit operation is planned with the outsourcing of certain support activities such as explosives manufacturing and blasting activities.

16.3.2 Production Drilling and Blasting

Drill and blast specifications are established to effectively single pass drill and blast a 10 m bench. For this bench height, a 229 mm blast hole size is proposed with a 6 m x 7 m pattern with 1.8 m of sub-drill. These drill parameters combined with a high energy bulk emulsion with a density of 1.2 kg/m^3 result in a powder factor of 0.28 kg/t. Blast holes are initiated with electronic detonators and primed with 450 g boosters. The bulk emulsion product is a gas sensitized pumped emulsion blend specifically designed for use in wet blasting applications.

Several rock types are present in the pit with the average rock hardness estimated at about 152 MPa.

The average drill productivity for the production rigs, using down-the-hole drill string, is estimated at 38 m/hr instantaneous with an overall penetration rate of 27.4 m/hr. This is based on a drilling trial at site conducted in 2022 using comparable drill rigs to those being considered. The overall drilling factor represents time lost in the cycle when the rig is not drilling such as move time between holes, moves between patterns, drill bit changes, etc.

The blast hole rig selected for production drilling will have a hole size range of 152 to 251 mm with a single pass drill depth of approximately 16 m. This rig will have both rotary and down-the-hole (“DTH”) drilling capability. It is expected that the DTH drilling mode will be most efficient.

Drill tele-remote and semi-autonomous operations are planned. Evaluation of fully autonomous drilling systems may be evaluated later during operations.

Blasting activities will be outsourced to an explosive’s provider for supply and delivery of explosives in the hole through a service contract. The mine engineering department will be responsible for designing blast patterns and relaying hole information to the drills via the wireless network.

Table 16.10 below summarizes the drill and blast parameters per rock type.

Table 16.10: Drill & Blast Parameters

Drill & Blast Parameters		Ore	Waste	Overburden
Drill Pattern				
Explosive Density	*g/cm^3*	1.20	1.20	1.20
Diameter (D)	*mm*	229	229	229
Burden (B)	*m*	6.0	6.0	12.0
Spacing (S)	*m*	7.0	7.0	12.0
Subdrill (J)	*m*	1.8	1.8	3.6
Stemming (T)	*m*	4.5	4.5	5.0
Bench Height (H)	*m*	10.0	10.0	10.0
Blasthole Length (L)	*m*	11.8	11.8	13.6
Pattern Yield				
Rock Density	*t/bcm*	3.05	3.05	1.80
BCM / Hole	*bcm/hole*	420	420	1,440
Yield per Hole	*t/hole*	1,281	1,281	2,592
Yield per Meter Drilled	*t/m drilled*	109	109	191
Explosive Column (LE)	*m*	7.3	7.3	8.6
Volume of Explosives/ Hole	*m^3*	0.30	0.30	0.35
Weight of Explosives/Hole	*kg*	360	360	424
Powder Factor	*kg/t*	0.28	0.28	0.16
Powder Factor	*kg/bcm*	0.86	0.86	0.29
Drill Productivity				
Re-drills	*%*	5.0%	5.0%	5.0%
Pure Penetration Rate	*m/hr*	38	38	50
Overall Drilling Factor (%)	*%*	72.0%	72.0%	80.0%
Overall Penetration Rate	*m/hr*	27.4	27.4	40.0
Drilling Efficiency	*t/hr*	2,970	2,970	7,624
Drilling Efficiency	*holes/hr*	2.32	2.32	2.94

16.3.3 Grade Control

The ore control program will consist of establishing dig limits for ore and waste in the field to guide loading unit operators. A high precision system combined with a stick and boom geometry system will allow shovels to target small dig blocks and perform selective mining. The system will give operators a real-time view of dig blocks, ore boundaries and other positioning information.

For optimal ore-waste boundaries identification, blasthole sampling will target 100% of all ore material and also capture 100% of the total waste in the pit. Reverse circulation drilling may be applied in some specific areas of the deposit where blasthole drilling may not adequately capture thinner high-grade zones (mainly in the narrower zones in the south pit).

The ore control boundaries will be established by the technical services department based on grade control information obtained through blast hole sampling with post-blast boundaries adjusted for blast movement measurements made using a BMM® system. A blast movement monitoring system has been included in the blasting cost.

The samples collected will be sent to a nearby, off-site laboratory for sample preparation and assaying for the LOM. Samples will be collected on the bench and tagged by grade control samplers on each shift.

16.3.4 Pre-Split

Pre-split drill and blast is planned to maximize stable bench faces and secure inter-ramp angles along pit walls as prescribed by the geotechnical pit slope study by Knight Piésold. The pre-split consists of a row of closely spaced holes along the design excavation limit of interim and final walls. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. Pre-split holes will be drilled with a smaller diameter drill of 165 mm (6.5 in.).

As presented in Table 16.11, the cost assumed for blasting of the pre-split holes is based on use of a special packaged pre-split explosive internally traced with 5 g/m detonating cord that ensures fast and complete detonation of the decoupled charge. For this specific application, a 410 mm long cartridge will be used, which corresponds to a complete case of 25 kg. This load factor of 1.47 kg/m allows for a targeted charge weight of 0.62 kg/m^2 of face. The specific pre-split drilling pattern and product type used will be refined as part of regular operations.

The drill selected for this application is more flexible type of rig capable of drilling angled holes for probe drilling and pit wall drain holes. The hole size range of this rig is between 110 mm and 229 mm with a maximum hole depth of 56 m.

Table 16.11: Pre-Split Parameters

Design Parameters		Pre-Split Holes
Drill Pattern		
Diameter (D)	*mm*	165
Spacing (S)	*m*	2.0
Stemming (T)	*m*	3
Bench Height (H)	*m*	20
Blasthole Length (L)	*m*	20
Face Area	*m^2*	40
Explosives Charge	*kg*	25
Charge Factor	*kg/m^2 face*	0.62
Cartridge Charge		
Nb Cartridges	*Qty*	41
Cartridge Length	*m*	0.41
Cartridge Loading Factor	*kg/m*	1.47
Decoupled Charge Length	*m*	17.00
Decoupled Charge	*kg*	25
Blasting		
Packaged Pre-Split Explosive	*kg*	24.99
Surface Delay NONEL	*unit*	0.20
Detonating Cord	*m*	5
Explosion Product Cost	*$/hole*	191.81
Drill Productivity		
Pure Penetration Rate	*m/hr*	41.2
Overall Drilling Factor (%)	*%*	0.58
Overall Penetration Rate	*m/hr*	23.9
Drilling Efficiency	*holes/hr*	1.2
Meters of Drilling per M Crest	*m/m of crest*	10.00

16.3.5 Loading

Most of the loading in the pit will be performed by two 29 m^3 face shovels. The shovels will be matched with a fleet of 246 t payload capacity mine trucks. The hydraulic shovels will be complemented by one production front-end wheel loader (“FEL”) with 19 m^3 bucket.

A fleet of two 90 t and two 45 t excavators will take care of the overburden tonnages, along with any narrow-thickness ore zones associated with the W-Horizon in the South Pit.

The loading productivity assumptions for both types of loading tools in ore, waste and overburden are presented in Table 16.12.

The two 29 m^3 face shovels are expected to achieve a productivity of 3,190 t/hr based on a 4-pass match with the mine trucks and an average load time of 2.7 minutes.

The wheel loaders are expected to achieve a productivity of 1,750t/hr based on a 7-pass match and an average load time of 4.9 minutes in ore and waste.

Table 16.12: Loading Specifications

		ORE	ORE	WASTE ROCK	WASTE ROCK
Loading Unit		**Diesel Hydraulic Shovel (29 m³) CAT 6060 FSD**	**Wheel Loader (19 m³) CAT 994K**	**Diesel Hydraulic Shovel (29 m³) CAT 6060 FSD**	**Wheel Loader (19 m³) CAT 994K**
Haulage Unit		**Mining Haul Truck (246 t) CAT 793**	**Mining Haul Truck (246 t) CAT 793**	**Mining Haul Truck (246 t) CAT 793**	**Mining Haul Truck (246t) CAT 793**
Rated Truck Payload	t	246	246	246	246
Heaped Tray Volume	m^3	152	152	152	152
Bucket Capacity	m^3	29	19	29	19
Bucket Fill Factor	%	90%	78%	90%	78%
In-Situ Dry Density	t/bcm	**3.10**	**3.10**	**3.09**	**3.09**
Moisture	%	3%	3%	3%	3%
Swell	%	35%	35%	35%	35%
Wet Loose Density	t/lcm	2.37	2.37	2.36	2.36
Actual Load Per Bucket	t	61.7	35.1	61.5	34.9
Passes (Decimal)	#	4.0	7.0	4.0	7.0
Passes (Whole)	#	4	7	4	7
Actual Truck Wet Payload	t	247	245	246	245
Actual Truck Dry Payload	t	240	238	239	237
Actual Heaped Volume	m^3	104	104	104	104
Payload Capacity	%	100%	100%	100%	99%
Heaped Capacity	%	69%	68%	69%	68%
Cycle Time					
Hauler Exchange	min	0.60	0.60	0.60	0.60
First Bucket Dump	min	0.10	0.10	0.10	0.10
Average Cycle Time	min	0.67	0.70	0.67	0.70
Load Time	min	2.70	4.90	2.70	4.90
Cycle Efficiency with Wait Time	%	60%	60%	60%	60%
No. of Trucks Loaded per Hr	#	13.3	7.3	13.3	7.3
Production / Productivity					
Productivity Dry Tonnes / Op. Hr	t/hr	3,196	1,750	3,186	1,745
Effective Hours per Year	hrs/y	5,578	5,317	5,547	5,547
Dry Annual Production Capacity	kt/y	17,830,462	9,305,759	17,674,751	9,677,579
Number of Units	#	2	1	2	1
Tonnes	t/y	35,660,923	9,305,759	35,349,502	9,677,579

16.3.6 Hauling

Haulage will be performed with a 246-t mine trucks. Several haulage profiles were digitized and simulated in Deswik LHS to optimize fleet usage and meet construction and site progression requirements.

16.3.6.1 Haulage Site Inputs and Assumptions

The assumptions and input factors for Deswik simulations are shown in Table 16.13, Table 16.14 and Table 16.15. Two speed limits were applied in the simulation. For all downhill ramps with an incline greater than 5%, the speed is limited to 30 km/hr otherwise, the maximum truck speed reaches 50 km/hr in the simulations.

Table 16.13: Site Speed Limits

Segment	Speed (km/hr)
Site Max	50
Down Hill	30

Table 16.14: Site Rolling Resistance Assumptions

Segment	Rolling Resistance (%)
Main Road	2.5
Pit Ramp	3.0
Dump	3.5
Pit Floor	3.5

Table 16.15: Fixed Cycle Time Components

Segment	Time (mins)
Queue Time	1.42
Spot Time	0.6
Loading Time	2.37
Total Loading	**4.39**
Queue Time	0
Spot Time	0.3
Dumping Time	0.2
Total Dump	**0.5**
Total Fixed	**4.89**

16.3.6.2 Haulage Simulation

A multiple waste dumps strategy will be used to help level the truck requirements for the Project. During the critical years of the Project, leveling will be achieved by sending waste rock to the closest dumps.

Figure 16.19 shows the trucks cycle times by pit. Figure 16.20 Typically, cycle time increases with the increase of the depth of the pit over the mine life. Cycle time is also dependent on the dumping schedule and the distance each dump is from the pit.

The overburden and narrow-width ore hauling will be done with a fleet of 90 t haul trucks. These trucks will also be used to supplement the ultra-class fleet as required.

The total haul hours required by period coupled to the truck mechanical availability were used to determine the number of trucks required throughout the LOM. That being a function of volume moved from source, the capacity of the truck and the cycle time associated. The truck fleet reaches a maximum of 11 units in Year 5 and remains at this level until Year 12 when truck requirements reduce with a reduced mining rate. Figure 16.21 below summarizes the truck requirements.

Over the LOM a total of 109.5 million liters of fuel will be consumed by the haulage fleet.

Figure 16.19: Truck Cycle Times by Pit

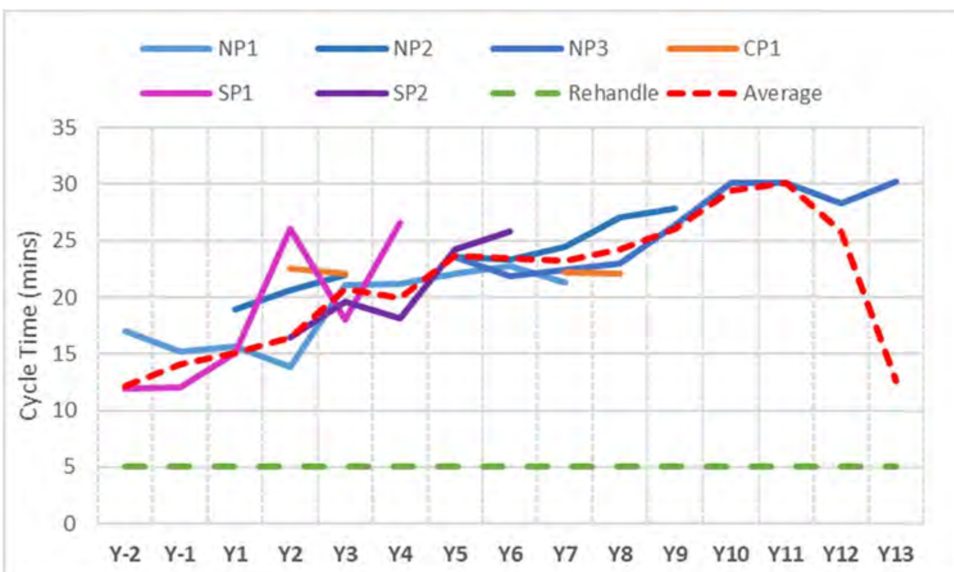


Figure 16.20: Haulage Hours by Source

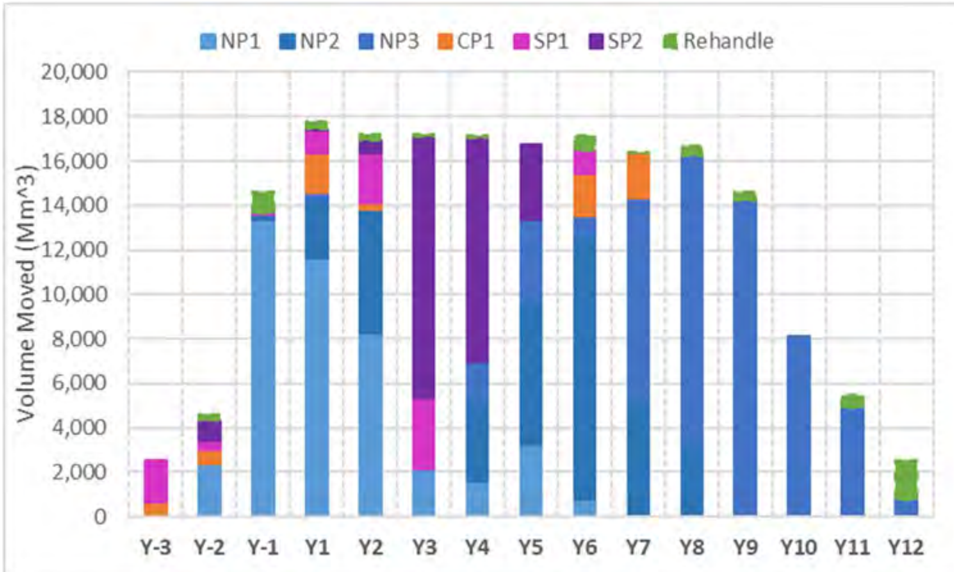
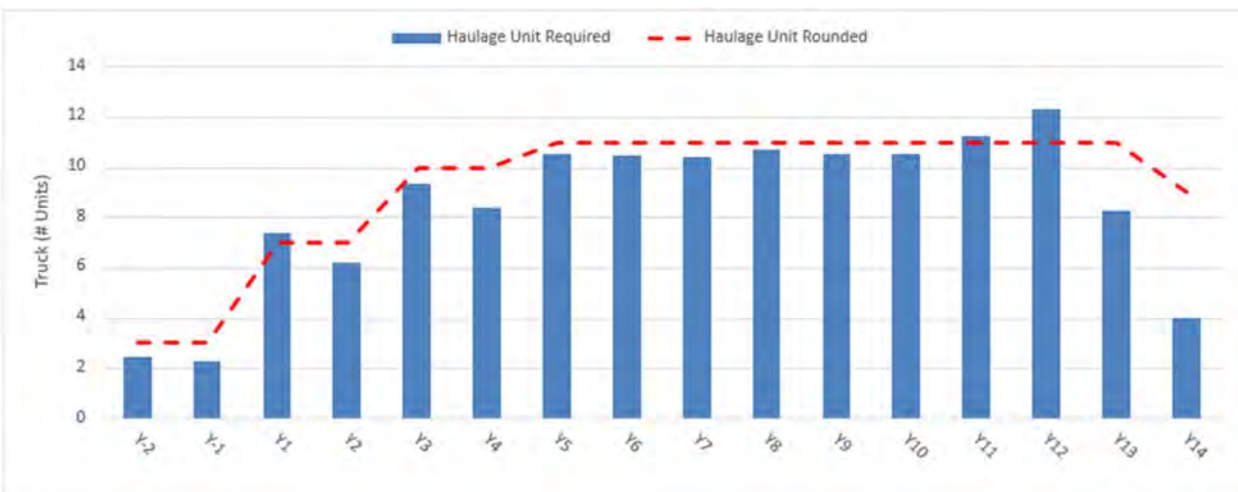


Figure 16.21: Truck Requirements



16.3.7 <u>Trolley System</u>

This Project has the potential to benefit from the addition of a trolley system. A trolley system could be implemented in long stretches of consistent uphill climb, typically in sections of the mine that are in place for many years and receive significant traffic. A potential location would be on the west and east ramps of the North Pit as these sections are in production for seven or more years and will see a significant percentage of the mine traffic.

Potential benefits of a trolley system:

- Reduction in fuel consumption and substitution with cheaper and cleaner electricity

- The reduction in fuel burn (when the truck is operating on trolley) will extend engine life
- A trolley system can deliver more power to the electric engines significantly increasing the uphill climbing speed (~20km/hr vs 12km/hr) and reducing cycle time
- Reduction in truck operating costs (due to decreased fuel and mechanical engine wear)
- Potential reduction in haulage fleet size from increased haulage productivity

Potential drawbacks of implementing a trolley system:

- Large initial CAPEX outlay for installation of trolley line, substations and other electrical infrastructure
- Additional OPEX cost to maintain and relocate trolley system as mining develops
- Increased CAPEX and OPEX for haulage units due to installation of pantographs and trolley package
- Potential increase to stripping ratio if wider ramps are utilized to increase trolley usage rate
- Increased road maintenance requirements under trolley lines to ensure good contact

Trolley systems typically include ramp widths capable of three lanes of traffic to maximize trolley usage rates by limiting impact of non-trolley trucks slowing trolley trucks down. Pit designs would need to be updated to reflect this. Typically an increase in ramp width leads to more waste mined and an increase in stripping ratio.

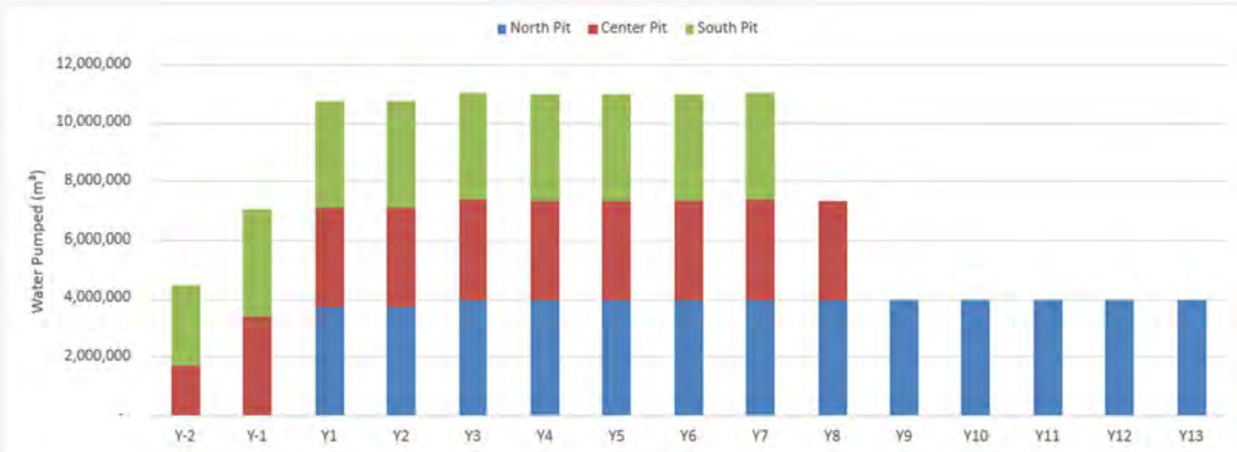
The implementation of a trolley system was evaluated in 2022 but did not show sufficient returns to offset the costs and operational complexity of a trolley network. Changes in commodity prices and recent developments by major equipment manufacturers towards trolley and/or battery assisted haul trucks may change the results of this evaluation in the future. Gen Mining will continue to evaluate the potential to implement a trolley system in the future.

16.3.8 Dewatering

It is assumed that each pit will receive 827 mm/y of rainwater and up to 457m^3/y per pit of ground water influx. Calculating from the production schedule, it is estimated that a total 115 Mm^3 of water will be pumped from all the pits over the mine life. To achieve this, two 10-inch pumps are required per pit. Due to the staggered mining, pumps can be moved to other pits when the pit is completely mined out. In-pit pumps are placed in sumps equal to the lowest mining level and, using 10-inch pipe segments, the water is pumped to surface settling ponds.

Figure 16.22 depicts the dewatering requirements by pit and period. Once the South Pit and Center Pit are mined out, they are used as in-pit dumps of PAG material. Pumping is planned to continue throughout the life of the mine. Also note, small variations in dewatering requirements by pit as new phases increase the surface area of the mine and increase the rainwater to be pumped from each of the sumps.

Figure 16.22: Dewatering Totals



16.3.9 Road and Dump Maintenance

Pit operating floors, waste and ore storage areas will be maintained by a fleet of six 630 HP track-type dozers. A 500 HP wheel loader will also be purchased and dedicated to mine roads and the loading areas.

Mine roads will be maintained by two 18 ft blade motor graders. Water/sand trucks will be used to spray roads to suppress dust or spread road aggregate during winter months.

16.3.10 Support Equipment

All construction related work, such as berm construction and water ditch cleaning will be done by two 49 t excavators (one of them will be equipped with a hydraulic hammer). The 90 t excavators dedicated to the overburden and narrow-width ore zone will also perform the pit wall scaling activities.

Pit buses will transport workers to their assigned workplace and pick-ups will be purchased for all the movements of operators, staff and supplies across the site.

Several other equipment purchases are included to support the mining activities. Also included are boom truck (28 t crane), 271 HP wheel loader and 100 t low-boy trailer and tractor for moving tracked equipment and other supplies.

16.3.11 Mine Maintenance

The Project has not included a maintenance and repair contract ("MARC") for its mobile equipment fleet. The maintenance department and personnel requirement has been structured to fully manage this function, performing maintenance planning and training of employees. However, reliance on dealer and manufacturer support will be key for the initial years of the Project, and major component rebuilds will be supported by the OEM's dealer throughout LOM. An evaluation of a MARC will be considered with the equipment selection process.

Tire monitoring, rotation and/or replacement will be carried out in-house, and a tire handler truck has been planned as part of the maintenance equipment fleet. Some other equipment will also be purchased to facilitate the maintenance activities and support the operation, such as two fuel and lube trucks, a dedicated lube truck, a forklift and some small equipment like tower lights, welding machines or portable air compressors.

A computerized maintenance management system will be used to manage maintenance and repair operations. This system will keep up-to-date status, service history and maintenance needs of each machine while being the source of data for key performance indicators ("KPIs") and cost tracking purposes.

16.3.12 Equipment Usage Model Assumptions

The typical equipment usage model assumptions are established by equipment groupings as presented in Table 16.16. The annual net operating hours ("NOH") varies approximately between 5,000 and 6,000 hours per year.

Table 16.16: Equipment Usage Model Assumptions

Parameter		Shovels	Loaders	Trucks	Drills	Ancillary	Support
Days in Period	*days*	365	365	365	365	365	365
Weather, Schedule Outages	*days*	5	5	5	5	5	5
Shifts per Day	*shift/day*	2	2	2	2	2	2
Hours per Shift	*hrs/shift*	12	12	12	12	12	12
Availability	*%*	82	80	83	80	85	85
Use of Availability	*%*	88	88	88	88	85	80
Utilization	*%*	72	72	73	70	72	68
Effectiveness	*%*	87	85	87	85	80	80
OEE	*%*	63	60	64	60	58	54
Total Hours	*hrs*	8,760	8,760	8,760	8,760	8,760	8,760
Scheduled Hours	*hrs*	8,640	8,640	8,640	8,640	8,640	8,640
Down Hours	*hrs*	1,555	1,728	1,469	1,728	1,296	1,296
Delay Hours	*hrs*	811	1,064	820	912	1,248	1,175
Standby Hours	*hrs*	850	677	861	829	1,102	1,469
Operating Hours	*hrs*	6,235	6,235	6,311	6,083	6,242	5,875
Ready Hours	*hrs*	5,424	5,170	5,490	5,170	4,994	4,700

16.3.13 Fleet Management

A fleet management system will be implemented to manage the operation, monitor machine health, and track KPIs. The system will be managed by a dispatcher on each crew who will control the system which will send operators onscreen instructions to work at peak efficiency. A dispatch system coordinator will be required to assure proper functioning of system hardware and software with ongoing annual vendor support.

A high-precision GPS for machine guidance is considered for grade control. Similarly, high precision drill navigation systems will be installed on the production drills and auxiliary drills to guide rigs into position and assure holes are drilled to the correct depth and location.

Teleremote drilling is considered as part of this study to maximize drilling productivity and utilization. Additional components of advanced technology and automation will be evaluated during final equipment selection and into operations.

16.3.14 Mine Equipment Requirements

The main factors which influenced the selection of the major mine equipment included the annual production requirements and optimization of the fleet size.

An analysis was performed to determine the optimal fleet size, equipment type and preferred suppliers. The requirements of major mining units purchased over the LOM are presented in Table 16.17.

Table 16.17: Equipment Purchase Schedule

Equipment Purchase Schedule	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Major Equipment																	
Mining Haul Truck (240t)	11	-	3	-	6	2	-	-	-	-	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (34 m³)	2	-	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Excavator (95t)	2	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-
Wheel Loader (30 m³)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Drill (6-10")	3	-	1	1	1	-	-	-	-	-	-	-	-	-	-	-	-
Track Dozer (600 HP)	10	2	2	-	2	-	3	-	-	-	-	1	-	-	-	-	-
Motor Grader (18ft)	5	1	-	1	-	-	1	-	-	1	-	-	-	1	-	-	-
Water/Sand Truck (76kL tank)	2	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Wheel Dozer (687 HP)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Auxiliary Pre-split Drill (4.5-8")	2	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Support Equipment																	
Mining Haul Truck (100t)	5	4	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Excavator (52t)	2	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Hammer for Excavator 52t	3	1	-	-	-	-	1	-	-	-	-	1	-	-	-	-	-
Skid Steer Loader	2	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Telehandler	2	-	1	-	-	-	-	-	-	-	1	-	-	-	-	-	-
Forklift Diesel 4t	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mechanic Service Truck	6	-	1	-	2	-	-	-	-	1	2	-	-	-	-	-	-
Shovel & Drill Repairs Trailer	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tire Handler Tooling & Equipment	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fuel & Lube truck 10Wheel	5	-	1	-	1	-	-	1	1	-	-	-	1	-	-	-	-
Lube Truck	2	-	-	-	1	-	-	-	-	-	1	-	-	-	-	-	-
Trailer Lowboy	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pick-up Truck	50	-	10	10	-	-	10	-	10	-	-	10	-	-	-	-	-
Pit Bus	12	-	4	-	-	-	-	4	-	-	-	-	4	-	-	-	-
Welding Machine Electric	4	-	1	-	1	-	-	-	-	-	-	1	-	1	-	-	-
Welding Machine Diesel 400A	4	-	1	-	1	-	-	-	-	1	1	-	-	-	-	-	-
Light Plant	30	-	10	-	-	-	-	4	6	-	-	-	4	6	-	-	-
Genset 6kW	6	-	3	-	-	-	-	-	-	3	-	-	-	-	-	-	-
Genset 60kW	3	-	1	-	-	-	-	1	-	-	-	1	-	-	-	-	-
Spare Box for Haul Trucks	2	-	-	-	1	1	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Shovels	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Excavator	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Loaders	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Small Excavator	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Equipment Simulator	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10" Pipe - 145psi	3008	900	240	-	310	190	333	197	100	100	310	228	100	-	250	300	300
Water Pump 10in - Diesel	36	-	4	4	2	6	6	-	6	-	4	-	2	2	2	2	-
Water Pump 6in - Diesel	8	-	2	-	2	-	-	-	-	2	2	-	-	-	-	-	-
Water pump 3" - Gasoline	16	-	4	-	-	4	-	-	4	-	-	4	-	-	4	-	-
Diesel Powered Air Heaters	6	-	2	-	-	-	2	-	-	-	2	-	-	-	2	-	-

16.3.15 Mine Workforce Requirements

Table 16.18 presents the mine workforce requirements over the LOM. The mine workforce peaks at 268 individuals in Year 2, then begins to decreases to 258 from Year 7 to Year 10 with a reduction occurring when the tonnage decreases starting in Year 11.

Table 16.18: Workforce Requirements

Department	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Operations	31	84	96	142	158	158	158	156	152	148	148	148	148	134	107	78
Mine Maintenance	24	33	45	80	80	80	80	80	80	80	80	80	80	73	55	55
Mine Geology	5	5	15	15	15	15	15	15	15	15	15	15	14	6	6	6
Mine Engineering	6	9	13	15	15	15	15	15	15	15	15	15	15	12	12	8
Total Workforce	**66**	**131**	**169**	**252**	**268**	**268**	**268**	**266**	**262**	**258**	**258**	**258**	**257**	**225**	**180**	**147**

17. RECOVERY METHODS

The Marathon Project FS process design is based on the information and metallurgical test results summarized in Section 13. The process plant will consist of a 25.2 kt/d with a subsequent ramp-up to 27.7 kt/d circuit with primary crushing, SAG and ball milling followed by flotation, concentrate dewatering and tailings impoundment. Cu-PGM flotation includes a rougher flotation circuit followed by regrinding rougher concentrate and cleaner circuit middlings with three stages of cleaning to yield a combined Cu-PGM concentrate. The PGM Scavenger circuit to reprocess the rougher tailings coarse fraction is excluded from Project design. Concentrate thickening, concentrate filtering, tailings thickening, water management, and a TSF complete the flowsheet.

This section describes the basis for process plant design and selection of major plant equipment for the recovery of copper and palladium, platinum, gold and silver to concentrate.

17.1 Process Design Criteria

The process plant is designed to process ore at a rate of 27,726 t/d (10.1 Mt/y) with an average LOM feed grade of 0.21% Cu and 0.63 g/t Pd to produce a combined Cu-PGM flotation concentrate.

The plant operating schedule and availability is based on two 12-hour shifts per day for 365 d/y. The operating schedule is based on 6,330 hr/y (72.26%) for the crushing circuit; 7,941 hr/y (90.65%) for the grinding and flotation circuits; and 7,446 hr/y (85.0%) for the concentrate filter.

Key design criteria used in the plant design, as well as the resulting sizing parameters of major equipment are listed in Table 17.1.

Table 17.1: Key Process Design Criteria

Description	Units	Value
Plant Capacity	Mt/y	10.1
LOM Average Feed Grade, Cu	%	0.21
LOM Average Feed Grade, Au	g/t	0.07
LOM Average Feed Grade, Ag	g/t	1.42
LOM Average Feed Grade, Pt	g/t	0.20
LOM average Feed Grade, Pd	g/t	0.63
Operating Schedule and Stockpile		
Crusher Operating Availability	%	72.26
Grinding and Flotation Operating Availability	%	90.65
Concentrate Filtration System	%	85.0
Primary Crushed Ore Stockpile Type	-	A-Frame
Stockpile Repose Angle	°	37
Stockpile Retention Time	hours	1,819.6
Ore Properties		
Specific Gravity	-	3.09
SPI Value	min	100
JK Axb (design, based on SPI value)	-	38
Bond Crushing Work Index (CWi)	kWh/t	18.6
Bond Rod Work Index (BRWi) (based on BWi)	kWh/t	16.5
Bond Ball Work Index (BBWi)	kWh/t	17.45
Bond Abrasion Index (Ai)	g	0.35
Primary Crushing		
Throughput, Nominal	t/h	1,915
Primary Crusher Type	-	Gyratory
Primary Crusher Model	-	Metso Superior™ MK-III 54-75
Primary Crusher Top Size, F100	mm	1,000
Primary Crusher Feed Size, F_{80}	mm	528
Primary Crushing Product, P_{80}	mm	150
Grinding and Pebble Crushing		
Throughput, Nominal (dry)	t/hr	1,274
Circuit product size, P_{80}	μm	106
SAG Mill Dimension (diameter x effective grinding length)	Ø x EGL (m)	7.32 m dia x 10.97 m
SAG Mill Installed Power	MW	18.0 (2 x 9.0 MW)
Pebble Crusher Type	-	N/A

Description	Units	Value
Pebble Crusher Model	-	N/A
Pebble Return Rate, Nominal	% new feed	22.3
Ball Mill Dimension (diameter x effective grinding length)	Ø x EGL (m)	7.92 m dia x 13.11 m
Ball Mill Installed Power	MW	17.25 (2 x 8.625 MW)
Circulating Load, Max for Design	%	300
Cyclone Overflow Solids	% solids (w/w)	35
Rougher Flotation		
Flotation Feed Density, Nominal	% solids (w/w)	35
Flotation Time Lab Testing	min	24
Residence Time Scale-Up Factor	-	2.5
No. of Cells	-	5
Cell Type	-	Outotec Tank Cell e630
Rougher Concentrate Regrind		
Regrind Circuit Feed, Nominal / Design	t/hr	185 / 219
Regrind Feed, Nominal / Design	t/hr	126 / 149
HIG Mill Specific Energy, Regrind Circuit Feed	kWh/t	11.9
Feed Size, F_{80}	µm	67
Product Size, P_{80}	µm	18
No. of Regrind Mills	-	1
Regrind Mill Type	-	HIG Mill (23,000 L)
Regrind Mill Installed Power	kW	3,500
Iron Sulfide Aeration		
Aeration Tank Residence Time, Nominal	min	30
Cleaner Flotation		
No. of Stages		3
Cell Type	-	SFR
Residence Time Scale-up Factor[4]		2.5
1st Cleaner		
No. of Cells	-	7
1st Cleaner Flotation Time Lab Testing	min	6
Cleaner Scavenger		
No. of Cells	-	3

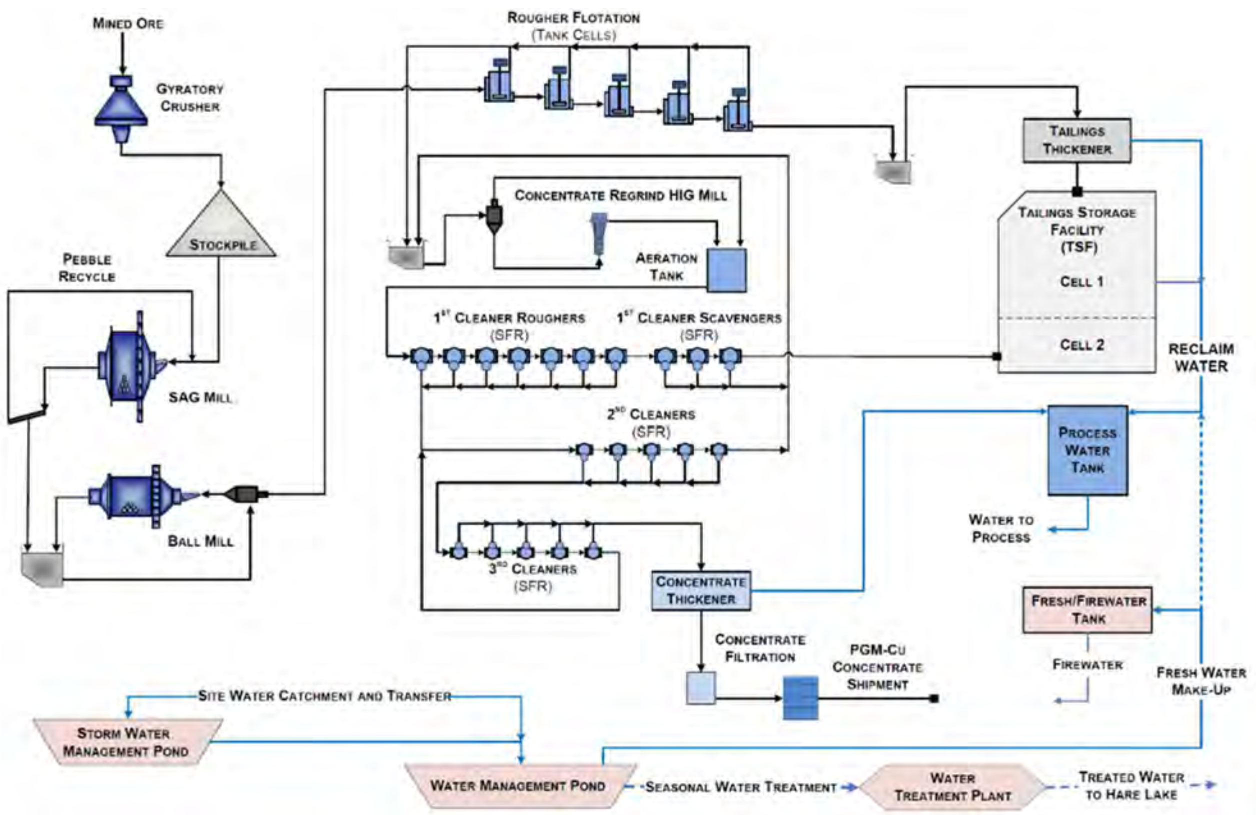
[4] The parameter as noted does not directly apply as SFR technology does not follow the scale-up factor as is used in conventional flot cell technology

Description	Units	Value
Cleaner Scavenger Flotation Time Lab Testing	min	9
Pd Concentrate Grade, Nominal	g/t	1.0
Cu Concentrate Grade, Nominal	%	0.5
2nd Cleaner		
No. of Cells	-	5
2nd Cleaner Flotation Time Lab Testing	min	5
3rd Cleaner		
No. of Cells	-	5
3rd Cleaner Flotation Time Lab Testing	min	5
Concentrate Thickening and Filtration		
No. of Thickeners	-	1
Thickener Type	-	High-Density
Unit Area Thickening Rate, Design	t/(m^2·hr)	0.15
Underflow Solids	% solids (w/w)	60
Filter Type	-	Vertical pressure filter
Filter Model	-	Outotec Larox PF 96/96
Total Filtration Area	m2	96
Filling and Filtration Time	min	2
Total Cycle Time	min	10
Number of Cycles per Hour		6
Cake Thickness	mm	25
Filtration Rate	kg/m^2/hr	283.1
Filter Cake Moisture	%	12
Tailings Thickening		
No. of Thickeners	-	1
Thickener Type	-	High Rate
Unit Area Thickening Rate, Design	t/m^2-hr)	0.60
Underflow Solids	% solids (w/w)	60

17.2 Process Flowsheet

The simplified overall flowsheet is shown in Figure 17.1.

Figure 17.1: Simplified Overall Process Flowsheet



17.3 Metallurgical Recovery

Determination of a predictive curve for metal recovery to a combined Cu-PGM concentrate was established from metallurgical testwork completed between 2020 and 2022 (Refer to Section 13 – Mineral Processing and Metallurgical Testing). Metal recovery estimates as a function of head grade have been refined in more recent testwork completed during Q4 2022 with separate GeoMet model equations for Cu, Pd, Pt, Au, Ag, excluding the PGM-Scavenger flotation circuit, and are outlined in Table 17.2.

Table 17.2: GeoMet Model Recovery and Mass Pull Formulas

Parameter	GeoMet Formula	Maximum Value
%Rec Cu to Final Conc	$= 97.55 \times (\%\ \text{Cu head grade})^{0.0239}$	94% Rec Cu
%Rec Pd to Final Conc	$= 89.14 \times (\text{g/t Pd head grade})^{0.0203}$	90% Rec Pd
%Rec Pt to Final Conc	$= 104.51 \times (\text{g/t Pt head grade})^{0.2034}$	84% Rec Pt
%Rec Au to Final Conc	$= 116.51 \times (\text{g/t Au head grade})^{0.1822}$	86% Rec Au
%Rec Ag to Final Conc	$= 50.82 \times (\text{g/t Ag head grade})^{0.6090}$	68% Rec Ag
%Mass Pull to Final Conc	$= 0.625 \times e^{(2.899 \times \%\text{Cu head grade})}$	2.0% Mass Pull

17.4 Overall Process Description

The Marathon Project process plant includes the following unit processes and facilities:

- Primary crushing
- Crushed ore stockpile and reclaim
- Crushed waste stockpile
- SAG mill and ball mill circuit operated in closed circuit with cyclones
- Flotation comprised of rougher flotation, concentrate regrind and three stages of cleaning
- Concentrate thickening and filtration
- Tailings thickening
- Reagents storage and distribution (including lime slaking, flotation reagents, water treatment and flocculant)
- Grinding media storage and addition
- Water services (including fresh water, fire water, cooling water and process water)
- Compressed air for SFR cells, plant air, and instrument air services
- Air blowers for conventional flotation cells
- Process plant system and control room

17.5 Primary Crushing

Run-of-mine ("ROM") ore will be delivered to the primary gyratory crusher dump pocket by 240-t haul trucks or alternatively for short periods by front-end loader ("FEL"). Single side dumping is included in the design with a dump pocket capacity of approximately 360 t.

ROM ore will be crushed in the gyratory crusher as it passes from the dump pocket to the crusher discharge vault. Crushed ore is withdrawn from the discharge vault by a variable speed belt feeder. The belt feeder transfers material onto a 0.7 km long overland conveyor. The overland conveyor is equipped with a weightometer to measure primary crusher tonnage.

A hydraulically operated rock breaker mounted on the perimeter of the dump pocket will be used to fracture any oversize ROM ore that is fed to the primary crusher, or to clear any build up within the pocket.

A metal detector will be located on the primary crusher discharge conveyor to detect tramp metal and protect the overland conveyor. A self-cleaning magnet situated above the discharge belt feeder and prior to the metal detector will assist in the removal of metallic objects.

The primary crusher area will be monitored and controlled from the crusher or process plant control room.

17.5.1 Coarse Ore Stockpile and Reclaim

Primary crushed ore from the overland conveyor will pass through a crushed ore/waste transfer tower with a diverter chute used to direct crushed ore to the coarse ore stockpile feed conveyor, and crushed waste to the crushed waste stockpile feed conveyor. The functionality of a crushed ore/waste split will accommodate a nominal 2.0 Mt per annum crushed waste rock requirement for TSF construction and site crushed rock.

Crushed ore will be fed via a crushed ore stockpile feed conveyor to a single covered stockpile with a total storage capacity of approximately 65,700 t. The nominal live capacity of the stockpile is 35% of total tonnes (approximately 19.6 hours), depending on material characteristics, weather, and ore moisture. Remaining capacity can be recovered using a bulldozer and/or excavator(s) when required.

Three apron feeders, two of them capable of supporting design throughput, will transfer material from the coarse ore stockpile to the SAG mill feed conveyor, which in turn feeds the SAG mill feed chute.

The SAG mill feed conveyor includes a weightometer to control and record feed rate to the SAG mill.

17.5.2 Waste Stockpile

On a campaign basis, the crushing circuit can crush, and convey waste material and dump it to the ground through the operation of the ore/waste diverter located at the overland conveyor transfer tower.

A FEL will reclaim the crushed waste material planned for construction purposes as required during the LOM. This area has the potential to install a radial stacking conveyor system to provide additional flexibility and reduce operating costs of the waste stockpile.

17.6 Grinding

The grinding circuit consists of a single SAG mill, followed by a single ball mill operating in closed circuit with a primary cyclopak. Primary cyclone overflow is the product from grinding with a target 80% passing size of 106 microns as flotation feed. The major equipment in the grinding circuit will include:

- SAG mill of 10.97 m diameter (inside shell) x 7.32 m effective grinding length (“EGL”) with two 9 MW twin pinion drives and a total installed power of 18 MW. SAG mill motors are controlled via a synchronous variable speed drive.
- Ball mill of 7.92 m diameter (inside shell) x 13.11 m EGL with two 8.625 MW twin pinion drives and a total installed power of 17.25 MW ball mill motors are controlled via a synchronous variable speed drive.

The SAG mill feed conveyor transfers ore (along with pebble recycle) to the SAG mill feed chute where it is combined with mill feed dilution water.

The SAG mill operates in closed circuit with a double deck screen. The double deck screen is fitted with spray water to wash slurry from coarse pebbles. The screen undersize flows by gravity to the cyclone feed pump box where it is combined with ball mill discharge. The pebbles reporting as screen oversize are transported by conveyor and combined with new incoming SAG mill feed.

Any tramp metal is removed using two self-cleaning belt magnets. A metal detector located before the transfer to the SAG mill feed conveyor but after the second self-cleaning magnet, will provide additional opportunity for metal removal. A weightometer located on the pebble transfer conveyor is used to monitor and record SAG mill pebble recycle rate.

The ball mill operates in closed circuit with primary cyclone underflow reporting to the ball mill feed chute. Ball mill discharge is combined with SAG mill screen undersize and feeds a cyclone feed pump box with a variable-speed cyclone feed pump and installed in-line spare. Process water is added to the launder of the grinding cyclone underflow, and to the cyclone feed pump box for dilution and level control prior to classification. This pump box also receives the addition of flotation collectors (potassium amyl xanthate, PAX) and promoter Aero 3501 (Sodium Isoamyl Di-thiophosphate) for an adequate mixing before feeding the product to the rougher flotation circuit.

Primary cyclone overflow will flow by gravity to the rougher flotation feed box. Cyclone overflow will be sampled and analyzed for copper using an on-stream analyser (“OSA”). Specific process streams within the flotation circuit will be sampled each shift, with composites analyzed for Cu and PGM metals. The OSA will provide copper analysis on a real time basis for flotation circuit operational control.

A couple of ball storage bunker and two daily bins are located on the process plant ground floor to manage SAG mill and ball mill grinding media. Ball charging buckets are filled from the grinding media day bins and transferred via grinding ball charging buckets to respective grinding media addition chutes to the SAG mill and ball mill via the mill area overhead crane.

A liner handler is included in design to support SAG mill and ball mill liner replacement activities.

17.6.1 Hycroft Mill Acquisition and Changes from 2021 Feasibility Study

During 2022, Gen Mining design optimization efforts included the consideration of a larger SAG mill and ball mill that were readily available as unused, *second-hand*[5] equipment. Relative to an engineered, fit for purpose installation, the larger, higher horsepower grinding mills supported the elimination of the pebble crusher in the grinding circuit and was included as a positive design change. A peer review study by Orway

[5] Previously purchased, never installed / never operated

Mineral Consulting ("OMC"), comminution engineering design specialists) confirmed the applicability of the two larger 18 MW grinding mills, and the potential to eliminate pebble crushing to a production rate of +15% relative to the 2021 FS design. The availability of these unused mills also offered inherent advantages with decreased delivery time, price protection, and warranty coverage from the OEM provider with slight refurbishing prior to installation. Equipment specifications for the larger Hycroft grinding mills is included in Table 17.1.

17.7 Rougher Concentrate Flotation and Cleaner Flotation

Primary cyclone overflow reports to the rougher flotation circuit feed tankbox. For rougher flotation, conventional tank cells have been selected, with SFRs selected for the cleaner circuit and summarized as follows:

- Rougher feed box: 2.0 m dia x 9.3 m high
- Rougher bank: five Outotec TCe630 tank cells (each 11 m dia x 8.2 m high)
- Cleaner 1/Cleaner scavenger feed tank: 2.5 m dia x 6.7 m high
- Cleaner 1 bank: seven (7) Woodgrove SFR cells (each 2.4 m dia x 5.7 m high)
- Cleaner scavenger bank: three (3) Woodgrove SFR cells (each 2.4 m dia x 5.5 m high)
- Cleaner 2 feed tank: 1.7 m dia x 5.6 m high
- Cleaner 2 bank: five (5) Woodgrove SFR cells (each 1.6 m dia x 4.6 m high)
- Cleaner 3 feed tank: 1.6 m dia x 5.1 m high
- Cleaner 3 bank: five (5) Woodgrove SFR cells (each cell 1.5 m dia x 4.1 m high)

The rougher flotation circuit produces a low-grade Cu-PGM rougher concentrate that requires regrinding to improve mineral liberation and upgrading within the cleaner circuit. Frequent measurement of mass flow and grade of the rougher concentrate is fundamental to optimize recovery and concentrate quality. This is accomplished by sampling it through an in-line launder sampler. A representative sample is sent to the on-stream analyzer ("OSA") and the mass pull will be calculated based on a simple mass balance. A shift composite sample will also be collected to account for the shift performance.

Rougher concentrate is combined in the regrind cyclone feed pump box with 1st Cleaner scavenger concentrate, and 2nd Cleaner tailings, and pumped to the regrind cyclones for classification. Regrind cyclone underflow reports to the concentrate regrind mill. The regrind cyclone overflow and regrind mill discharge are combined as feed to an iron sulfide aeration tank with a residence time of 30 minutes. The purpose of the iron sulfide aeration tank is to surficially oxidize and condition iron sulfides at pH 11 resulting in pyrite depression and partial rejection of pyrrhotite in the first cleaner circuit.

The Rougher tailing is the final tail of the process plant, and it is sampled through a vertical metallurgical sampler that takes a complete cut of the cross-section of the flow and a secondary cut that is sent to the

OSA for on-line analysis and the collection of a shift composite sample for metal accounting. The rejects from the vertical sampler are sent by gravity to the flotation tailing thickener feed tank before thickening.

First cleaner concentrate is upgraded in two additional cleaning stages, with third cleaner tailings reporting to second cleaners, and second cleaner tailings reporting to the rougher concentrate regrind cyclone feed pump box.

The final concentrate from the third flotation cleaner is collected in a pump box and then passes through a metallurgical sampler that periodically samples a cross-section of the slurry. The sample is sent to the OSA where it is analyzed and a shift composite sample is collected for metal accounting purpose.

A final Cu-PGM concentrate from the third cleaners is thickened to 60% solids and filtered to a nominal 12% moisture in preparation for transport and additional refining at off-site smelters.

First cleaner tailings are fed to first cleaner scavenger flotation with first cleaner scavenger concentrate recycled to the rougher concentrate regrind cyclone feed pump box. The first cleaner scavenger flotation tailings process stream is pumped to designated sub-aerial locations within the TSF to mitigate the potential for oxidation and acid generation from the sulfidic tailings fraction.

Benchscale metallurgical testing completed during 2020-2022 by Gen Mining applied conventional flotation cell technology. Predicted metal recovery assumes a baseline 12-15% mass pull to rougher concentrate and 0.8 to 1.5% mass pull to a final Cu-PGM concentrate to optimize PGM recovery.

17.7.1 Concentrate Regrind

Rougher concentrate, second cleaner tailings, first cleaner scavenger concentrate are combined in the regrind cyclone feed pump box. The combined product is fed to the regrind cyclopak for classification to a target P_{80} of 18 μm. Fine particles that do not require further regrinding report to regrind cyclone overflow. The coarse fraction which reports to cyclone underflow is pumped to the concentrate regrind mill at a slurry density of 40-45% solids. The concentrate regrind circuit will utilize Metso-Outotec HIG (High Intensity Grinding) mill technology.

Regrind mill discharge will be combined with regrind cyclone overflow and report to the iron sulfide aeration tank with a residence time of 30 minutes to surficially oxidize and condition iron sulfides at pH 11 prior to first cleaner flotation.

Ceramic grinding media will be sourced in bulk bags and added to the regrind mill as required.

17.7.2 Concentrate Thickening and Filtration

The final concentrate thickening and filtration circuit consists of a single 15.0 m diameter high-density thickener and a vertical pressure filter.

The concentrate from the third cleaner stage is pumped to the concentrate thickener via the cleaner concentrate pump. Flocculant is added to the thickener feed stream to enhance settling. The concentrate thickener overflow is recycled as process water. Concentrate thickener underflow density will be 55-60% w/w solids. The thickener underflow stream is pumped to an agitated filter feed tank by variable speed centrifugal pumps.

The filter feed tank provides 24 hrs surge capacity for filter maintenance without affecting mill throughput. The filter feed is pumped to a single vertical pressure filter to produce a filter cake of approximately 12% w/w moisture. The concentrate filter cake is discharged by gravity to a concrete storage bunker.

Fresh water is used for the filter cloth washing. Filtrate is returned to the concentrate thickener. High pressure compressed air is supplied from a dedicated bank of air compressors to the pressure filter for dewatering and drying of the filter cake.

17.7.3 Changes from 2021 Feasibility Study

The 2021 FS considered the use of Woodgrove DFRs cells as part of process plant design to minimize the installed footprint of the flotation circuit and decrease overall power requirements. During detailed engineering, additional feedback and references from other users of the relatively new DFR cell technology was obtained to validate equipment selection. As part of risk mitigation, Gen Mining opted to pursue conventional tank cells for rougher flotation and Woodgrove Staged Flotation Reactors (“SFR”) cells for the cleaner circuit.

SFR technology has been developed and improved by Woodgrove over the past 10 years and provides the unique combination of (i) a particle collection unit (“PCU”) where air is finely dispersed within the slurry, followed by (ii) a separate bubble disengagement unit (“BDU”), and (iii) a froth collection unit (“FCU”) where mineralized values are recovered and removed. The ability to inject wash water counter-current to the collection of mineralized froth in the FCU, along with effective cell to cell slurry level control, provides operating conditions which are considered as ideal to reject gangue (waste) materials while concentrating mineralized values in the cleaner circuit.

17.8 Concentrate Storage and Load Out

Concentrate filter cake is discharged by gravity to a concrete storage bunker. The storage bunker provides a storage capacity for up to seven days at nominal production rates, four days when operating at design conditions.

Concentrate is loaded by FEL and transferred into a load out chute feeding a cleated belt feeder to a concentrate truck loading conveyor capable of loading a transport truck within 10 minutes. The concentrate transport truck is weighed on a load-out scale at site prior to hauling concentrate to the nearby rail load out facility or directly to the smelter.

17.9 Tailings Thickening

The tailings thickening circuit consists of a single 50 m diameter high-rate thickener to dewater rougher tailings to 55-60% w/w solids.

Flocculant is added to the thickener feed stream to enhance settling. Thickener overflow is recycled to the process water tank. The thickened underflow is pumped to the TSF.

17.10 Tailings Storage Facility

Tailings thickener underflow will be pumped to Cell 1, Cell 2A and Cell 2B of the TSF as a NAG, low sulfide slurry. First cleaner scavenger tailings will be pumped separately to designated sub-aerial locations in Cell 2A of the TSF to mitigate the potential for oxidation and acid generation from the sulfidic tailings fraction. Reclaim water pumps located at the TSF will recycle reclaim water to the process water tank.

17.11 Reagents and Consumables

Flotation reagents including PAX (potassium-amyl-xanthate), Aero-3501 (isoamyl di-thiophosphate), MIBC (methyl-isobutyl-carbonol) and quicklime (CaO) will be added to the grinding and flotation circuits to facilitate the recovery of values to concentrate. Process reagents and consumables are received and stored on site as either dry product or bulk liquids. Reagents will be prepared and stored in dedicated areas and delivered by individual metering pumps or centrifugal pumps to the required addition points.

Mixed reagents will be prepared using either process water (PAX, coagulant) or fresh water (lime, flocculant). The remainder of reagents including Aero 3501 and MIBC will be supplied as a liquid with bulk tanker deliveries.

17.11.1 Quicklime

Lime will be trucked to the site as quicklime in 30 t lime transportation trucks. The lime will be transferred to the lime silo using a pneumatic blower mounted on the delivery trucks. From the lime silo, the quicklime will be conveyed by a screw feeder to a lime slaker. The slaked lime slurry will be pumped to an agitated storage tank. Distribution of the lime from the storage tank to the addition points will be accomplished using a lime slurry loop. The bulk of the lime slurry will be used in the concentrate regrind circuit prior to first cleaner flotation, with the option to be added at each of the feed tanks of the second and third cleaner flotation circuits. As indicated, the lime preparation will be performed with fresh water, but there is an option to do it by exception using process water in case of an emergency.

17.11.2 Collector 1 (PAX)

Potassium amyl xanthate (PAX) is used as the collector in the rougher flotation circuit and cleaner flotation circuit.

PAX is delivered to site as pellets in 1 t bulk bags which will be stored in a covered building. Bulk bags of PAX are unloaded into an agitated mixing tank where it is mixed with process water to produce a 10% w/w solution concentration. The mixing tank will be located on top of the 30 m^3 holding tank, therefore the solution will be transferred by gravity and is then delivered to the rougher and cleaner flotation circuits via dedicated dosing pumps. A forklift and reagent area overhead crane will be used to position the bulk bags containing PAX above the xanthate mixing tank.

PAX, which is classified as flammable in solid form and associated vapors combustible from the liquid form, will be appropriately controlled and ventilated in respective reagent mixing and distribution tanks.

17.11.3 Collector 2 (Cytec Aero 3501)

Solvay Aero 3501 (isoamyl di-thiophosphate) acts as PGM collector and is added to the rougher and cleaner flotation circuits.

Aero 3501 is delivered to site as a liquid in 20 m^3 trailer loads. Dedicated dosing pumps are used to deliver Aero 3501 from the Aero 3501 bulk storage tank to each dosing point.

Aero-3501 in its liquid form is neither flammable or combustible and will be stored in a vented, external insulated and heated storage tank at a temperature maintained above 5° C.

17.11.4 Frother (MIBC)

Methyl isobutyl carbinol (“MIBC”) is used to provide a stable froth in the flotation circuit.

MIBC is delivered to site as a liquid in 25 t trailer loads. Dedicated dosing pumps are used to deliver MIBC from the MIBC bulk storage tank to each dosing point.

MIBC, which is a flammable liquid, will be isolated in a vented, appropriately designed and controlled external un-insulated storage tank.

17.11.5 Depressant

Carboxymethyl cellulose (“CMC”) is often used in the cleaner flotation circuit to improve final concentrate grades by depressing talc. Based on metallurgical testwork, no benefit was derived from the use of CMC and therefore was excluded from project design concepts.

17.11.6 Flocculant

A single flocculant mixing system will be used to provide flocculant to the concentrate and tailings thickeners. The flocculant mixing system consists of a storage bin, screw feeder, auto jet wet mixer, mixing tank mounted on top of the storage tank and dosing progressive cavity pumps to respective thickeners.

Flocculant is delivered as dry powder in 750 kg bags and will be stored indoors to keep the non-flammable product dry. A forklift and reagent area overhead crane will be used to position the bulk bags of flocculant above the flocculant storage hopper.

Dry powder is removed from the feed hopper and transferred via a screw feeder to an air venturi which pneumatically transports the powdered flocculant to a wetting head, followed by intense agitation in the mixing tank for approximately one hour. Each batch of mixed flocculant is allowed to age for an additional three hours prior to being transferred to an unagitated storage tank for distribution by dedicated dosing pumps.

Dedicated dosing pumps deliver flocculant from the storage tank to the concentrate and tailings thickeners.

17.11.7 Coagulant

A single coagulant mixing system will be used to provide coagulant to the tailings thickener. The coagulant mixing system consists of a storage bin, screw feeder, and process water eductor which will deliver coagulant through an in-line mixer directly to the tailings thickener.

Coagulant is delivered as dry powder in 750 kg bags and will be stored indoors to keep the non-flammable product dry. A forklift and reagent area overhead crane will be used to position the bulk bags of coagulant above the coagulant storage hopper.

Dry powder is removed from the feed hopper and transferred via a screw feeder to a process water eductor which will transport the coagulant through an in-line mixer directly to the tailings thickener.

17.11.8 Miscellaneous Reagents

Additional miscellaneous reagents such as antiscalant or other potential promoter-collectors for PGM flotation in the cleaner circuit are expected as potentially being applied in relatively small quantities in the plant. Any other reagent distribution systems that are justified as required will be included at a later date.

17.12 Water Systems and Process Plant Services

17.12.1 Fresh Water

Fresh water make-up will be provided to the process plant from a submersible - booster pump in a transfer tower configuration located at the Water Management Pond ("WMP"). The freshwater tank serves as a combined fresh water / fire water tank with the lower section dedicated for fire water service and the remainder for general fresh water supply.

Fresh water is used to supply the following services:

- Potable water treatment plant (including safety showers)
- Filter cloth wash water
- Miscellaneous equipment (e.g., On-stream analyzer)
- Reagent mixing (where applicable)
- Make-up water for the process water system

17.12.2 Process Water

Process water is supplied from tailings thickener overflow, concentrate thickener overflow, return water from the TSF, and freshwater make-up as required.

Process water is stored in the process water tank. The tailings thickener overflow and concentrate thickener overflow streams report directly to the process water tank for immediate distribution and use. Process water pumps distribute process water to the grinding circuit, rougher flotation, regrind circuit, and cleaner flotation circuit.

The Process Plant is operated in closed circuit with the Tailings Storage Facility (TSF) with reclaim water from the TSF recycled back to the process plant. The water balance for the process facility is net negative over the entire LOM due to the entrainment of a nominal 22% weight/weight moisture within the impounded tailings solids.

Make-up water to support the process facility is provided from the WMP and estimated at 0.17 m3 solution per tonne processed (1.72 Mm3/yr). The recycle of TSF reclaim water accounts for 86.4% of process water needs, with the remainder sourced from the WMP. There are no dedicated fresh water pumping wells associated with the project.

17.12.3 Fire Water

The freshwater tank contains a dedicated firewater reserve with a design capacity of 910 m^3.

17.12.4 Potable Water

Potable water services are described in Section 20.

17.12.5 Gland Seal Water

The low and high-pressure pump gland seal water will be sourced from the process water tank, and it is distributed to all the pumps in the process plant.

17.12.6 Cooling Water

Cooling water is used in the SAG mill and ball mill to cool oil lubrication systems. In-line duplex filters will be used to remove fine particulates before distributing cooling water to respective mills.

17.12.7 Air Services

Compressed air supply includes: (i) a dedicated compressor for process plant air, SFR cell air and instrument air, (ii) a dedicated high-pressure compressor for concentrate filter operation, and (iii) a standby high-pressure compressor as a back-up for either process plant or concentrate filter operation. Rougher flotation air supply will be provided by low-pressure air blowers.

17.12.8 Instrumentation and Monitoring

Plant instrumentation includes an OSA that will be used to continuously monitor copper as a proxy to palladium in key flotation process streams to assist with optimising concentrate recovery and grade.

17.12.9 Process Control System

A process control system (“PCS”) will be included for the operation and monitoring of equipment including crushing, conveying, the process plant and water management systems. The PCS will include a distributed control system (“DCS”) which incorporates input signals from, and output signals to, field instrumentation, control devices and any Vendor supplied programmable logic controllers (“PLCs”). A central control room in the process plant will be linked to Human-Machine Interfaces and display stations (“HMIs”) which will provide the capability of local and/or remote operation and control of associated equipment.

The highly automated nature of the process facility requires the implementation of a well-defined and applied Lock Out-Tag Out-Try Out policy for all electrified or pressurized equipment prior to initiating any operational clean-up or maintenance activities.

17.12.10 Closed Circuit Television System

A closed-circuit television system will be included as part of the process control system to support control room operators with the remote monitoring of conveyor transfer points, stockpile levels and work areas.

18. PROJECT INFRASTRUCTURE

The required infrastructure to support the mining and processing operations include:

- Existing infrastructure
- Area 1100 – General Site Preparation
- Area 1200 – Mine Infrastructure
- Area 1300 – Support Infrastructure
- Area 1400 – Permanent Camp
- Area 1500 – Laboratory
- Area 1700 – Fuel/Oil Systems Storage
- Area 1800 – Transload Facility
- Area 2100 – Main Power Generation
- Area 2200 – Secondary Power Generation
- Area 2700 – MV Distribution O/H Line
- Area 2850 – Telecom & Automation Network
- Area 2900 – IT Network
- Area 3100 – Fresh Water / Wells
- Area 3200 – Surface Water Management
- Area 3300 – Potable / Domestic Water
- Area 3400 – Sewage Water
- Area 3500 – Fire Water
- Area 3600 – Water Treatment Plant
- Area 3700 – Tailings Storage Facility
- Area 3800 – Mine Rock Storage Area Catch Basins
- Area 3900 – Site Water Management Pond
- Area 6870 – Process Plant Power Distribution
- Area 6900 – Process Plant Infrastructure

18.1 Existing Infrastructure and Location

The Marathon Project, accessed by road from Highway 17, is located 10 km north of the Town of Marathon with a population of 3,138 (2021 census). There is no infrastructure or public services directly on site. The Town of Marathon administers the Marathon Airport located less than 3 km from the Project site. The Marathon region has access to the Canadian railroad network. The railway infrastructure in Marathon is well developed for exporting concentrate to North American ports or smelters in a timely manner.

HydroOne's grid network is well developed in the Project area. The 115 kV powerline from Marathon to Manitouwadge also known as M2W line is approximately 3.2 km from the planned main substation located beside the proposed mill site. There is an existing transformer substation, Marathon TS, 10.5 km from Marathon providing 115 kV power. The East-West Tie Transmission Line Project completed in 2022 ties Thunder Bay to Wawa with a 230 kV transmission line and 230 kV substation adjacent to the Marathon substation.

18.2 Area 1000 Infrastructure

Figure 18.1 and Figure 18.2 shows the Project site general layout and the local site infrastructure.

Figure 18.1: Marathon Project Overall Site Plan

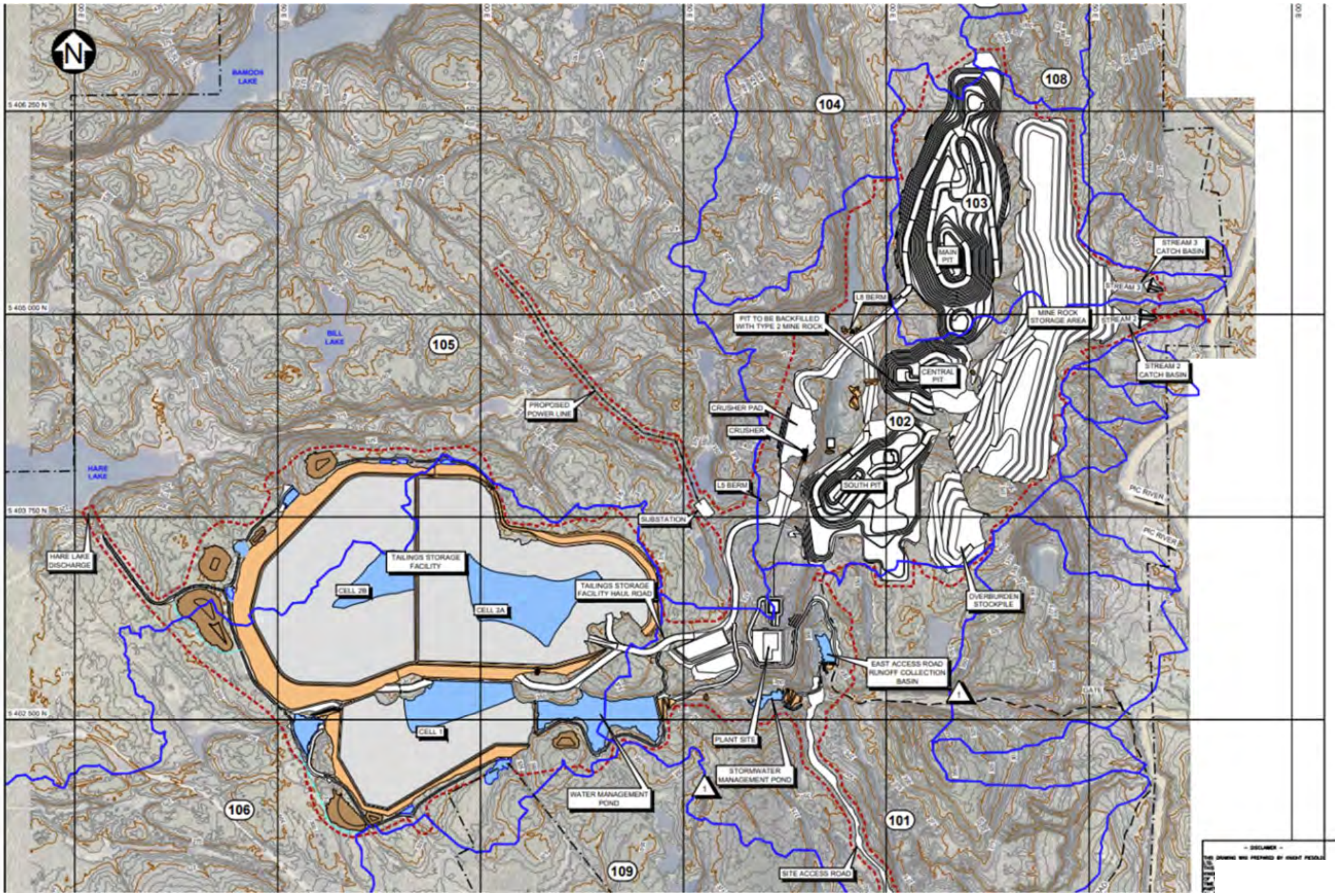
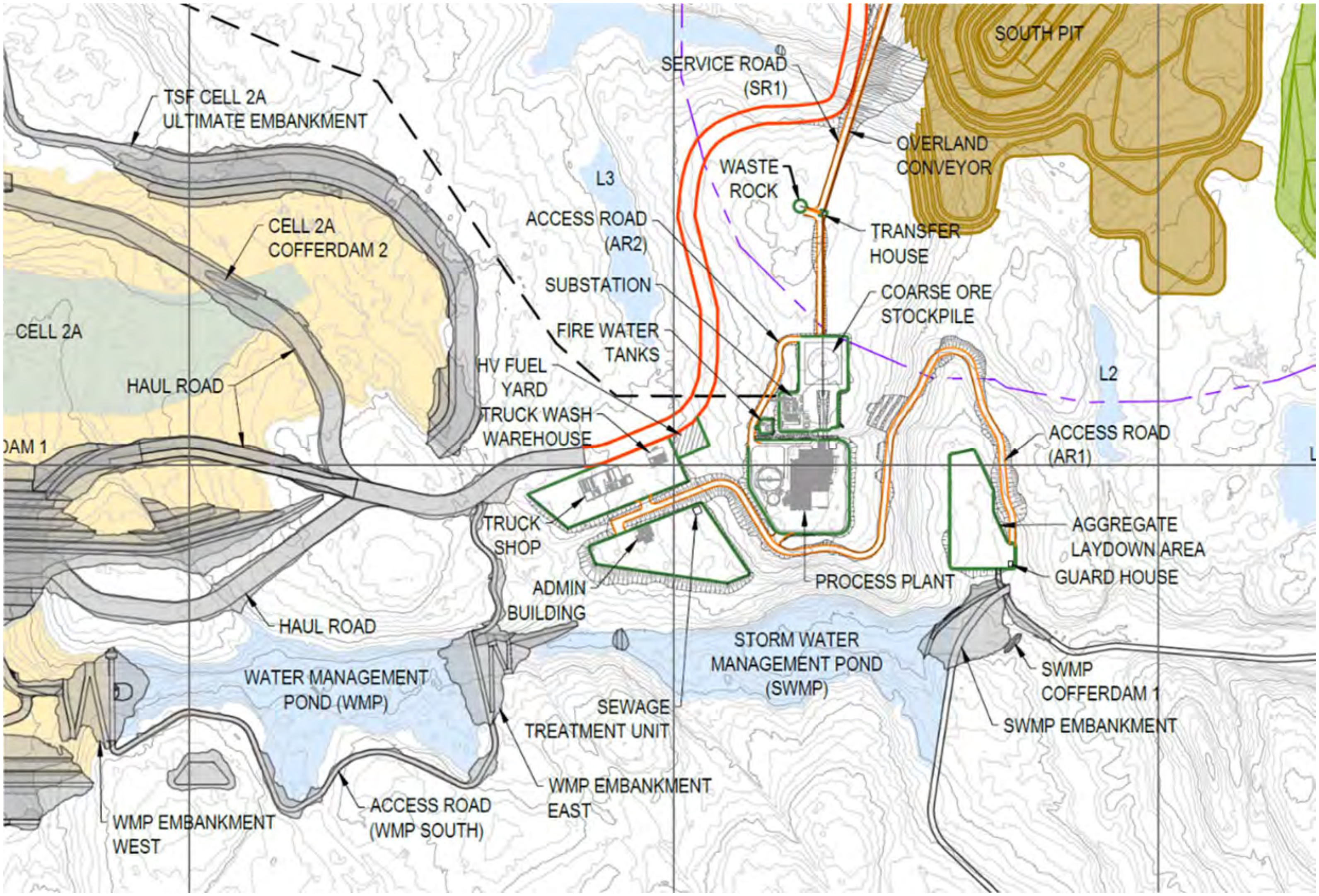


Figure 18.2: Marathon Project Site Infrastructure



18.2.1 Area 1100 General Site Preparation

This section presents the clearing for all areas: South and Central pits, process plant, mine services area, primary crusher, conveyors, ore stockpile area, laydown areas, aggregate production pad, explosives storage facility, site roads, TSF area, and related stockpile areas. It also includes the costs to load, haul, and stockpile stripped and grubbed organic materials to the designated areas on site.

Work related to site road construction accounts for minimized drill and blast excavation, while relying more heavily on the load, haul, place of rockfill material produced from the main borrow sources (pits, larger infrastructure pads). This assumption is due to shallow bedrock observed throughout site. Also included are the costs to deliver material to the aggregate plant to produce the transition and surfacing layers, then load, haul, and place these materials to the final destination at the road structures. These layers are to be 30 cm and 15 cm in thickness respectively.

All roads carry an allowance for drainage culverts, ditching, and safety berms sized appropriately to the largest vehicle size.

Roads are classified as follows:

- Site Access Road (Camp 19 & East Access Road)
  - 14 m total width
- Area Access Roads – To all be single lane utilizing pull outs
  - 240 t haul truck single lane – 23 m total width
    - 15.2 m running surface
  - 100 t haul truck single lane – 16.4 m total width
    - 10.4 m running surface
  - 40 t haul truck single lane – 12 m total width
    - 6.8 m running surface
- Haul Roads – To all be double lane for mining fleet
  - 240 t haul truck double lane – 35 m total width
    - 23 m running surface
    - Additional width for pipe bench (~5 m)

18.2.2 Area 1200 Mine Infrastructure

During the construction and initial mining phase, a simple truck shop/wash bay combination will be constructed inside an 18 m x 43 m pre-engineered tent structure. Once full-scale operations mining has commenced, a larger, fit for purpose truck shop will be installed. The construction phase truck shop will consist of one dedicated wash bay sized to accommodate a CAT 994 sized wheel loader, and one dedicated service bay sized to accommodate a CAT 793 sized haul truck.

Overhead doors will be located at either end of the building, sized to accommodate the widest equipment (12 m wide x 9 m tall overhead doors).

The wash bay will be designed to ensure wash water is captured and treated to remove hydrocarbons and then deposited into the Water Management Pond (“WMP”); a concrete slab with integral trench drain will be installed to capture and manage hydrocarbon contaminated water for the wash bay portion of the facility. An impermeable liner system will be deployed below the service bay area for the structure to provide secondary spill containment capability.

An externally installed modular building will house the heated pressure wash system, ancillary equipment, electrical room and mechanical room.

The truck shop will be connected to the mobile maintenance administration facility via a corridor. The admin facility will house the mobile maintenance personnel, planners, warehousing. It will have 3 to 5 closed door offices and 8 to 10 open workstations. In addition, the same corridor will access a washroom, lunchroom and a dry capable of supporting 60 mobile maintenance personnel.

Adjacent to the truck shop will be a heated and insulated warehouse facility servicing the balance of the site. Due to the site’s proximity to the town of Marathon, light vehicles and associated equipment will primarily be in available facilities in town.

18.2.3 Area 1300 Support Infrastructure

18.2.3.1 Area 1310 Administration Building

The administration building is located close to the mine service building facility on the mine services area pad. It is a single storey building that houses the human resources, general administration, mine management and operations, engineering and geology departments.

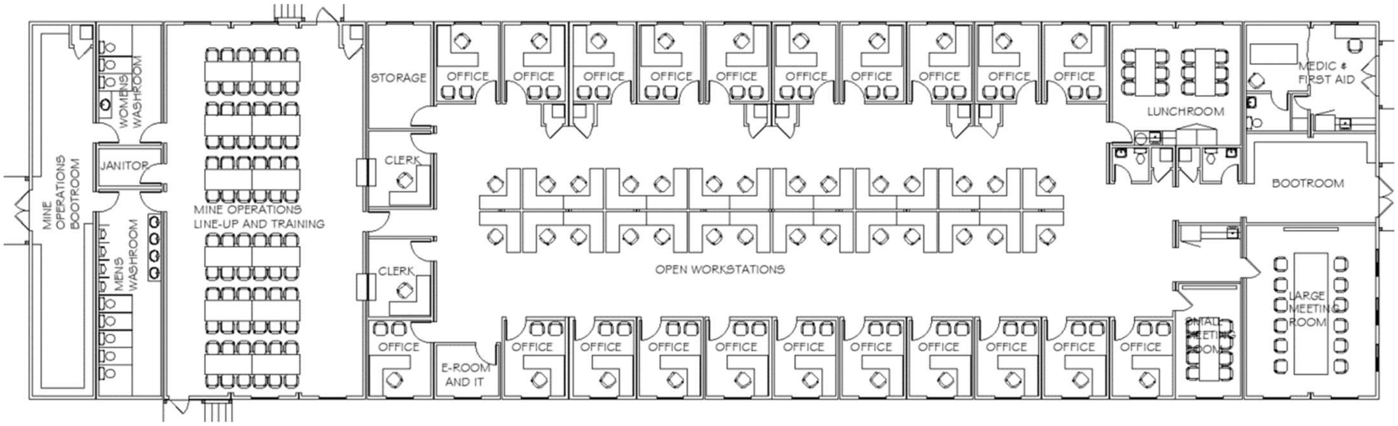
The administration building will feature between 15 and 20 office modules transported and connected to services on site.

To support these departments, the following infrastructure is incorporated into the administration building:

- Closed offices
- Open workstations
- Conference rooms
- Infirmary with private lavatory facility
- Boot room to support operations office and field personnel
- Kitchen/lunchroom
- Training/orientation room

- Copy/storage/filing room
- Mechanical/Electrical/Data rooms to support administration facility loading

Figure 18.3: Administration Building – Conceptual Floor Plan Layout



18.2.3.2 Area 1320 Site Guard Building

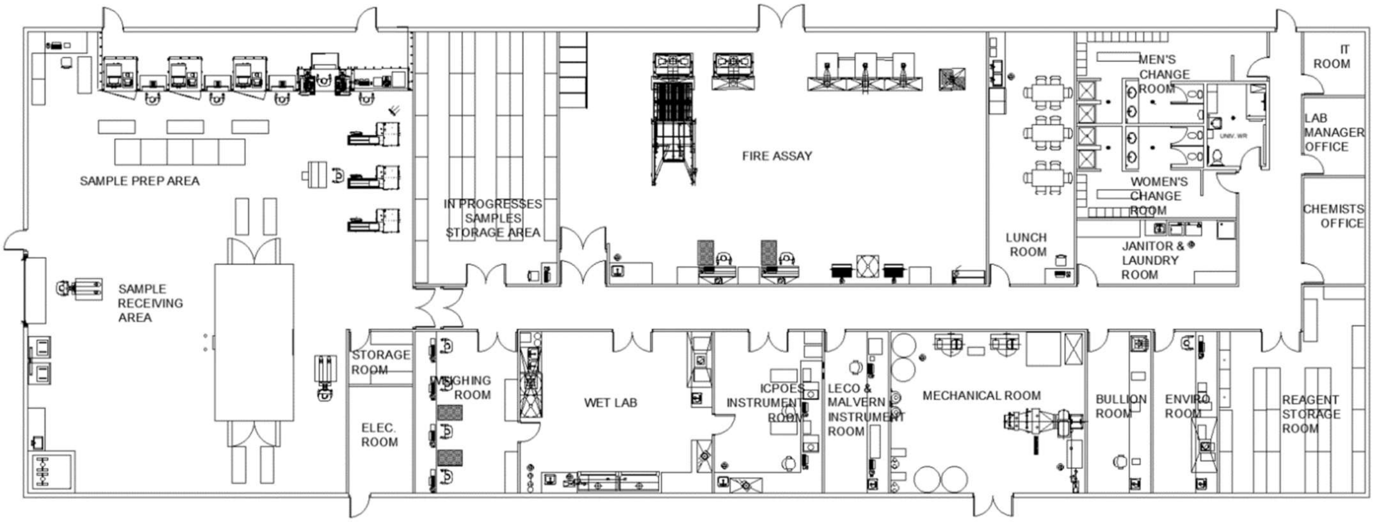
A modular guardhouse will be located close to the site entrance. The guard house will be sized to support security personnel offices, lockers, washroom facilities and smaller site orientations.

Site access will be controlled with an autonomous gate system and all access and egress from site will be monitored and controlled from the guardhouse.

18.2.4 Area 1500 Laboratory

The design, drawings, equipment requirements and costs of the laboratory are gathered in a document entitled, “Marathon PGM Mine Laboratory Proposal”, dated September 10, 2020. It shows the layout and list of equipment, as well as the planned testing program SGS intends to perform over the LOM. Figure 18.4 shows a plan view of the Laboratory.

Figure 18.4: Laboratory - Plan View



18.2.5 Area 1700 Fuel Storage

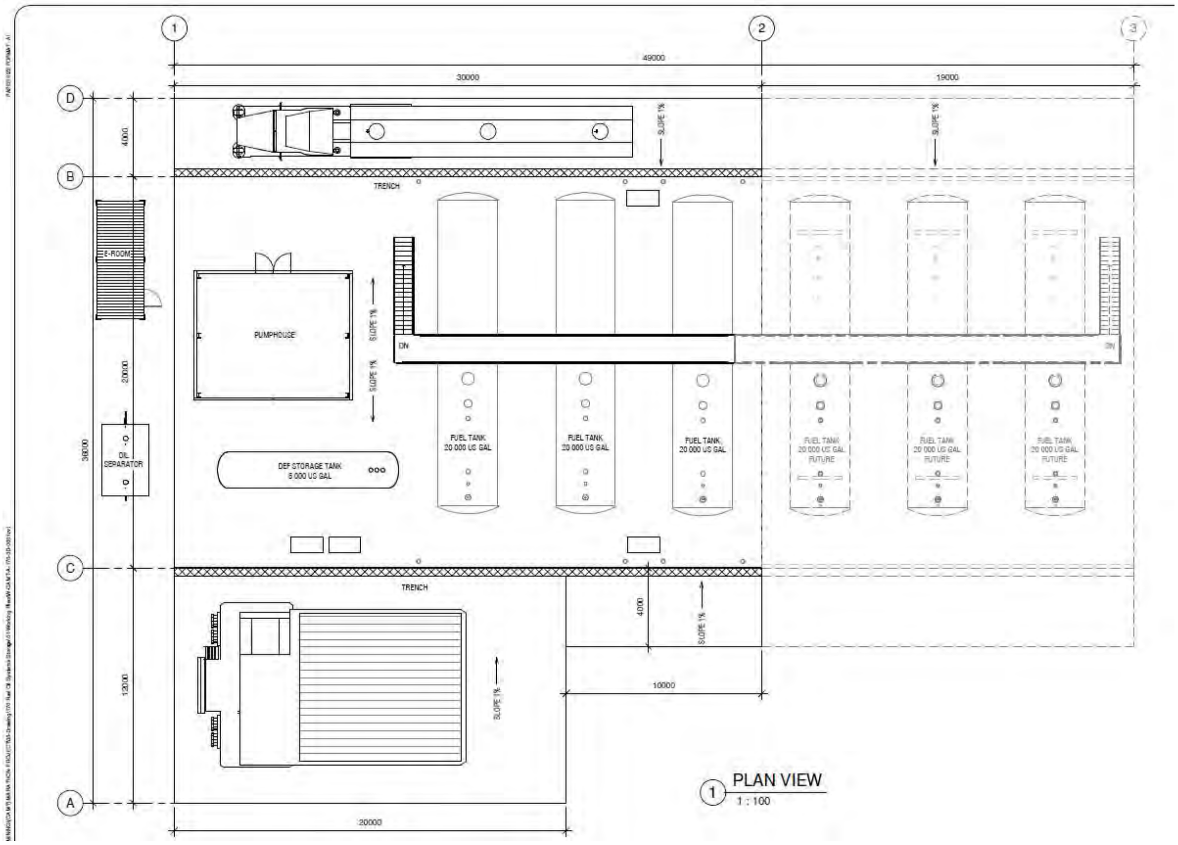
Fuel will be stored on site and located on the same pad as the truck shop and mine services office building with access possible from both the TMA Haul Road and site access road. The fuel storage facility will be used to fuel both light and heavy vehicles and equipment.

The fuel tanks will be staged in such a way that they can be easily refilled with super B sized tanker trucks.

The facility will be built in a manner that contains spills, routes precipitation to an oil-water separator and then pumps the water to the water management pond.

An expansion is planned in Year 3 to accommodate the fleet equipment increase.

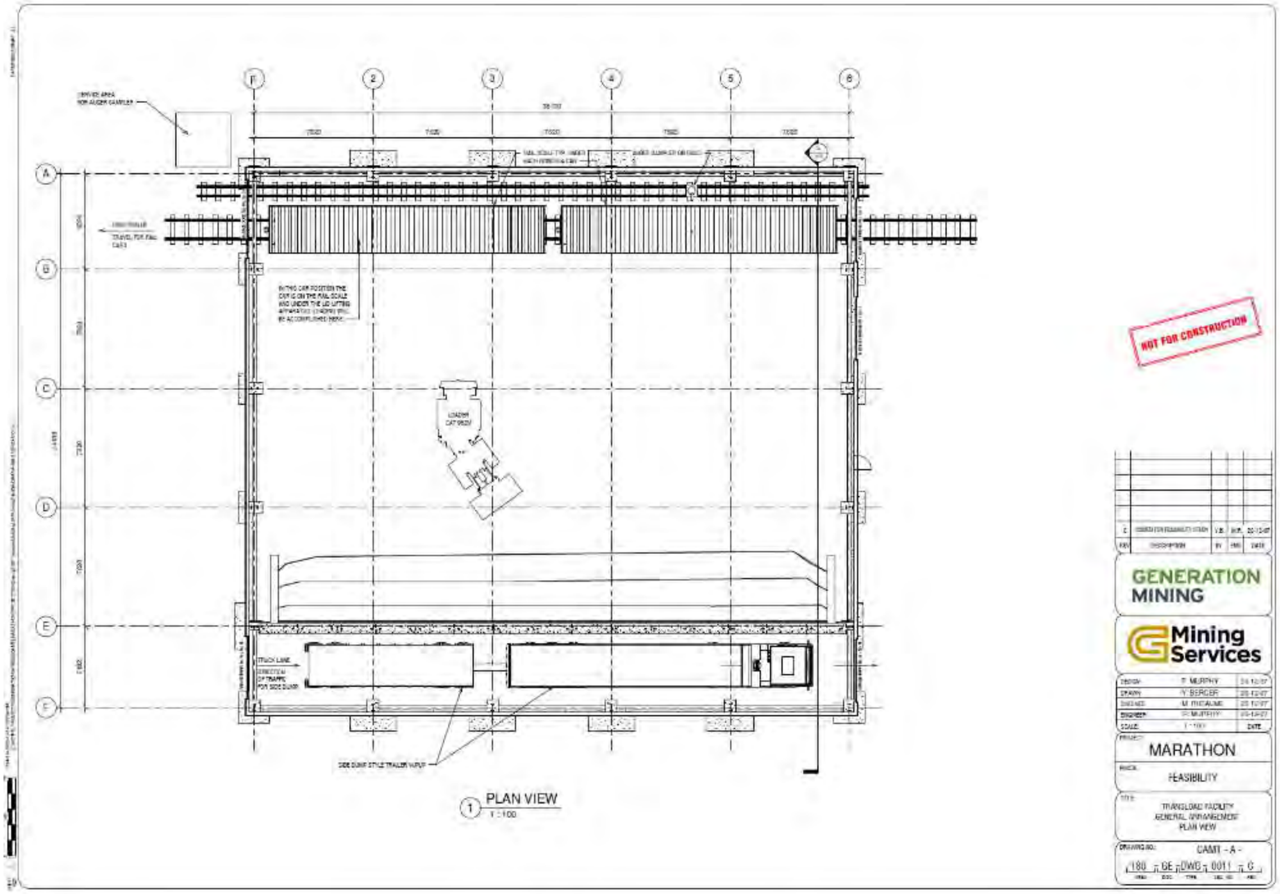
Figure 18.5: Conceptual Fuel Storage with Future Expansion



18.2.6 <u>Area 1800 Transload Facility</u>

Initial movement of concentrate will be trucked from site to the smelter. Construction of the transload facility will commence in Year 1 (following construction). The transload facility is planned to be located at a site in the Town of Marathon or in close proximity, and located on CPR line. The Canadian railway network has access to the potential Canadian smelter and ports to allow for sea-shipping to international smelter locations. The facility (Figure 18.6) will accept concentrate shipments from site via side-dump haul trucks. Haul trucks enter the building, dump the concentrate, and exit the building. The concentrate is loaded onto rail cars using a front-end loader. The building design will ensure adequate air quality control with sufficient air changes, as per the applicable codes and standards. Entrances and exits will have roll-up style doors to regulate airflow through the building.

Figure 18.6: (Conceptual) Transload Facility



The on-highway truck and trailers have a capacity of 36 to 40 t with a double tub (covered) single trailer pulled by a tractor truck (Figure 18.7 and Figure 18.8).

Figure 18.7: Example of Dual-trailer Side-dump Trailer with a Max 36-40 t Capacity

Figure 18.8: Example of Hard Cover Side-dump Trailer

18.3 Area 2000 Power and Electrical

18.3.1 Area 2100 Main Power Generation

Power demand of the Project is evaluated at 56.9 MW.

The main high voltage power source for the Project site is a tie-off connection to Hydro One 115 kV transmission circuit Marathon – Manitouwadge also known as M2W line (Figure 18.10).

The Project substation will be equipped with two 115–25 kV power transformers and a main 25 kV switchgear, the latter being the main power source of the entire project. Application of two identical transformers and double-ended configuration of the main 25 kV switchgear ("main-tie-main" scheme) will ensure full redundancy – each transformer will be sized to provide full power demand of the entire project.

The 115 kV substation will be of outdoor type with open air-insulated buswork. 25 kV switchgear will be installed in a pre-fabricated electrical room housing all necessary protection and communication equipment. The substation will allow for installation of Power Factor Correction Capacitors as required to comply with Hydro One power factor requirements. Refer to the Figure 18.9 for more details.

The point of connection is located about 10.5 km from Marathon transformer station. The line is approximately 3.2 km long and will be supported by wood poles. It has been designed for 85 MW load capacity. The transmission conductor is 267 kcmil ACSR Partridge and the overhead ground protection wire (OPGW) is AFL 48-fiber AC86/646. Preliminary design of the overhead line including line layout, structure geometries, transmission corridor, pole lengths, pole class and hardware assemblies have been completed. The design of the span supporting the point of interconnection with Hydro One line is expected to require two new additional structures with one carrying a disconnect switch.

Trees and vegetation shall be cleared within the right-of-way corridor and in 20 m from the edge of the right-of-way as identified in the preliminary design. Transport Canada and Marathon Airport Authority shall be engaged to determine the requirements for any additional lighting and marker balls on the overhead line.

Figure 18.9: Main Substation Layout

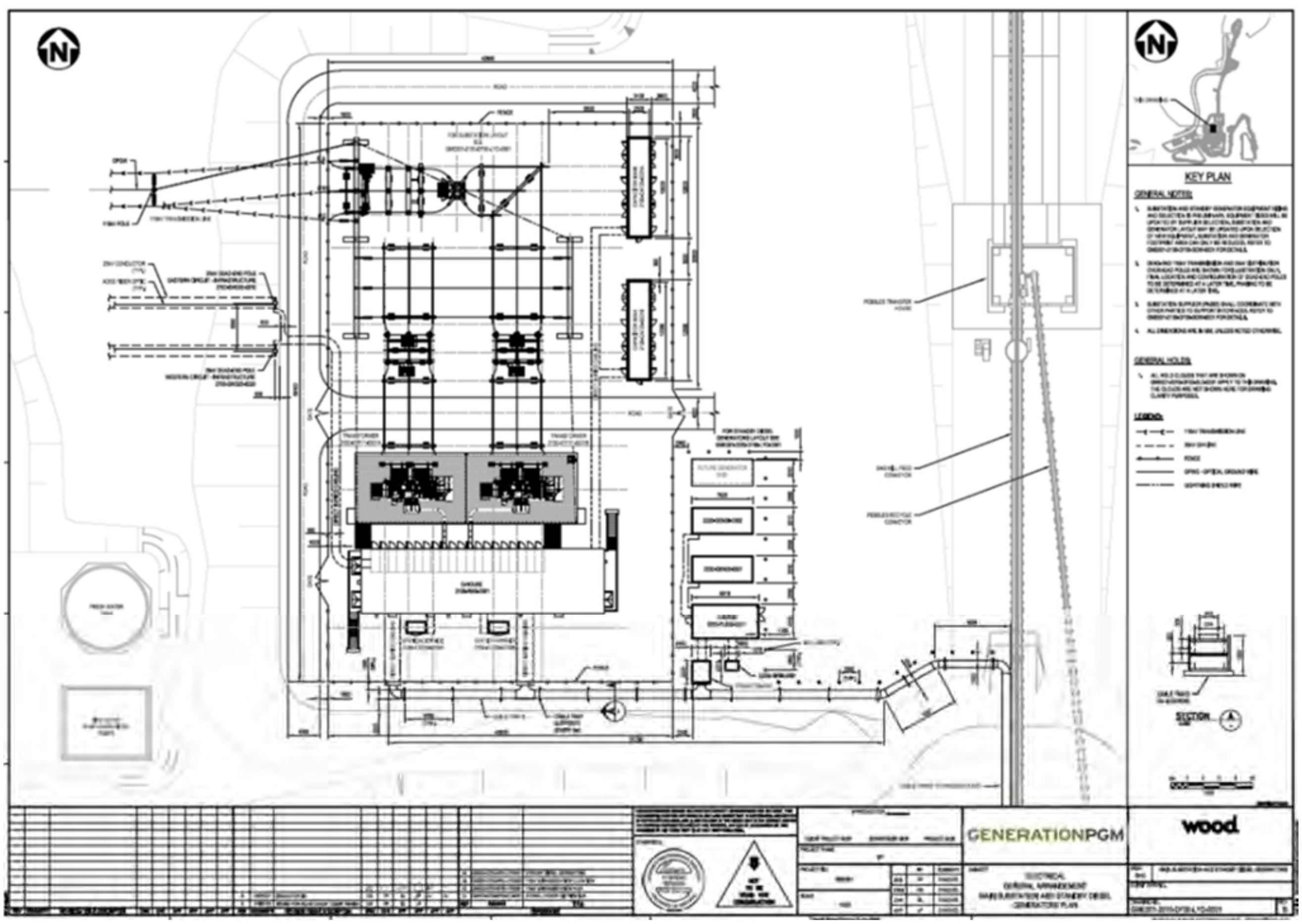
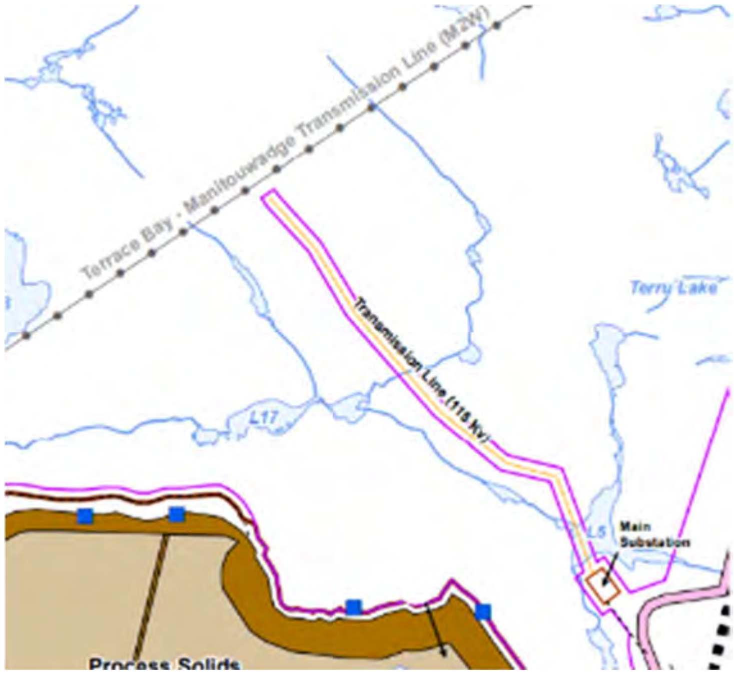


Figure 18.10: Overview of New 115kV Tap Line Layout



18.3.2 Area 2220 Secondary Power Generation

Diesel generators are considered as an alternative source of power in case of utility supply loss. Diesel generators will be sized to ensure power supply of all loads considered as critical (personnel safety and security, safe and controlled process shut-down, prevention of equipment damage) for at least eight hours.

Diesel generators will provide power to a dedicated 600 V emergency bus, for connection to the main 25 kV switchgear through 0.6-25 kV transformer. Diesel generators and their associated equipment will be located in immediate proximity of the main substation.

18.3.3 Area 2700 MV Distribution O/H Line

The Project site consists of two 25 kV overhead distribution systems. The first distribution line will be fed from the internal Marathon mine substation located in the process plant. The second distribution line will be interconnected to an existing Hydro One feeder located along Camp 19 road off highway 17.

The first 25 kV distribution line route is estimated to have a length of approximately 7.1 km in order to power the proposed loads and buildings. This line will be fed from the main Process Plant Main Substation 25 kV switchgear.

The second 25 kV distribution line route is estimated to have a length of approximately 8.2 km and will power auxiliary buildings (admin building, truck shop) and infrastructure pumps. This second line will be fed from the existing Hydro One line located along Camp 19 road off highway 17.

In addition to the transformers at the process plant, the electrical rooms listed in Table 18.1 are located near their service areas of the process plant.

18.3.4 Area 6870 Process Plant Power Distribution

The following power feeders will originate at the main 25 kV substation:

- Two feeders for 25 kV overhead lines supplying power to crushing and conveying equipment and remote site facilities;
- One cable feeder for SAG mill power supply via dedicated mill transformers;
- One cable feeder for ball mill power supply via dedicated mill transformers;
- Two cable feeders for 25-4.16 kV transformers powering 4.16 kV Motor Control Centre dedicated to large motors of the process plant; and
- Two cable feeders to the ring system of 25-0.6 kV transformers powering 600 V loads of the process plant.

Electrical equipment of the process plant will be free-issued by vendors and installed in a stick-built electrical building. The building footprint will be approximately 38 m x 18 m. The building will have three floors:

- Ground floor housing 25-0.6 kV transformers, 600 V MCC, 600 V VFD;
- Second floor – cable pull room; and
- Third floor housing 4.16 kV MCC, SAG and ball mill drives and excitation transformers, 4.16 kV VFD.

Electrical equipment for areas and facilities supplied via 25 kV OHL i.e. primary crusher, transfer tower, stockpile will be typically installed in prefabricated electrical rooms located near the supplied facilities.

Table 18.1: Process Plant Electrical Rooms

Process Plant Electrical Rooms	
Crusher Electrical Room	Prefabricated electrical room with free-issue electrical equipment installed – 14.0 m (L) x 5.5 m (W) x 4.0 m (H)
Conveyor Transfer House Electrical Room	Prefabricated electrical room with free-issue electrical equipment installed – 14.0 m (L) x 7.5 m (W) x 4.0 m (H)
Stockpile Electrical Room	Prefabricated electrical room with free-issue electrical equipment installed – 22.0 m (L) x 4.5 m (W) x 4.0 m (H)
Process Plant Electrical Room	Stick built electrical room with free-issue electrical equipment installed – 38.0 m (L) x 18.0 m (W) x 14.0m (H)

18.3.5 Area 2800 Automation Network

The infrastructure area will have wide area connectivity via an outside plant fibre optic distribution network. The fibre Optic backbone will provide services to the sites requiring inter-facility communication connectivity. The various networks will be segregated by utilizing separate fibre strands within the same fibre cable which will be underbuilt along the 25 kV power line as seen on Figure 18.11.

The automation network will utilize the fibre optic backbone with the objective to automate repetitive and physically tedious functions with an emphasis on the visual confirmation of operations to ensure personnel safety and equipment protection in the plant.

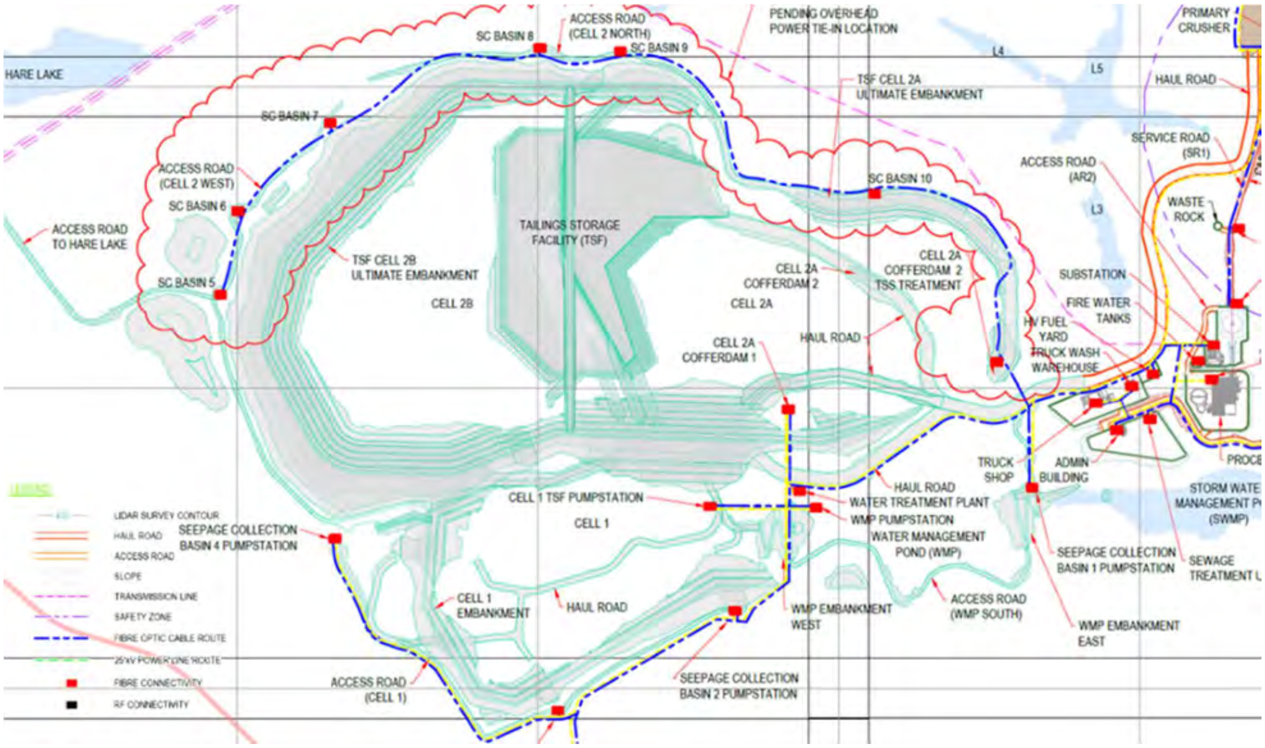
The operational control of the process plant will be performed from the Main Control Room by the Process Control System, and local operator workstations will be considered within process plant as required.

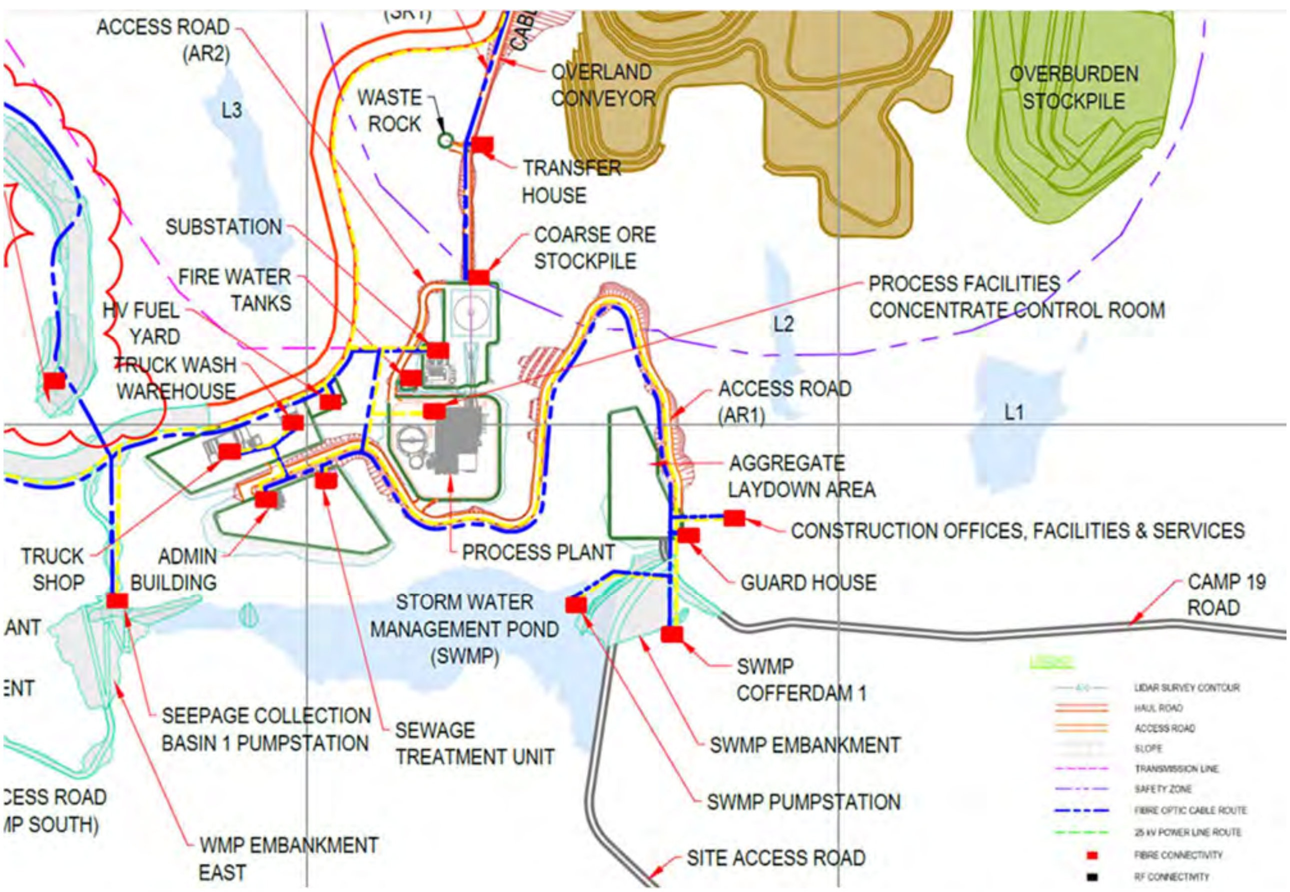
Field Operator Control Panels will be provided, where required, for process equipment for maintenance or safety reasons.

18.3.6 Area 2900 IT and Automation

As for the infrastructure automation system, schematics will be developed. However, the costs represent 2.7% of the process mechanical equipment costs. At the process plant, an allowance has been made for a control system interface and instrumentation. Piping and instrumentation diagram have not been developed at this stage of design. High level cable routing plans are presented in Figure 18.11.

Figure 18.11: Fibre Optic Network Architecture – Cable routing plan





18.4 Area 3000 Water Management

18.4.1 Area 3100 Fresh Water

During the initial construction, the TSF embankments will be developed for the WMP and Cell 1 storage cell. This infrastructure is required for commissioning and processing as the WMP is the source of fresh water for the Project. There are currently no plans for wells or other fresh water sources. The WMP pond have been sized to provide sufficient capacity for operational requirements.

Water will be pumped to the process plant area with a barge and pipeline system and other infrastructure as required.

The Reclaim Water Pipeline from the WMP to the process plant area will be heat-traced and insulated.

18.4.2 Area 3200 Surface Water Management

Figure 18.12 shows the site wide water balance that is supporting the infrastructure for water movements on site. Figure 18.13 shows the pumps and pipelines system for surface water management. Contact surface water from the Mine Rock Storage Area ("MRSA") Catch Basins, the mine surface, in pit ground water inflows, runoff from the mining activities and all other surface infrastructure drainage (haul road and access road ditches) will be collected and transferred to the WMP.

It should be noted that surface drainage at the process plant area, other buildings and access road will be collected in the SWMP and EARCB and transferred to the WMP. All collected surface water will be pumped to the WMP and either reclaimed to the process plant area for reuse in the process plant or treated and discharged to Hare Lake.

Figure 18.12: Site Wide Water Balance (KP)

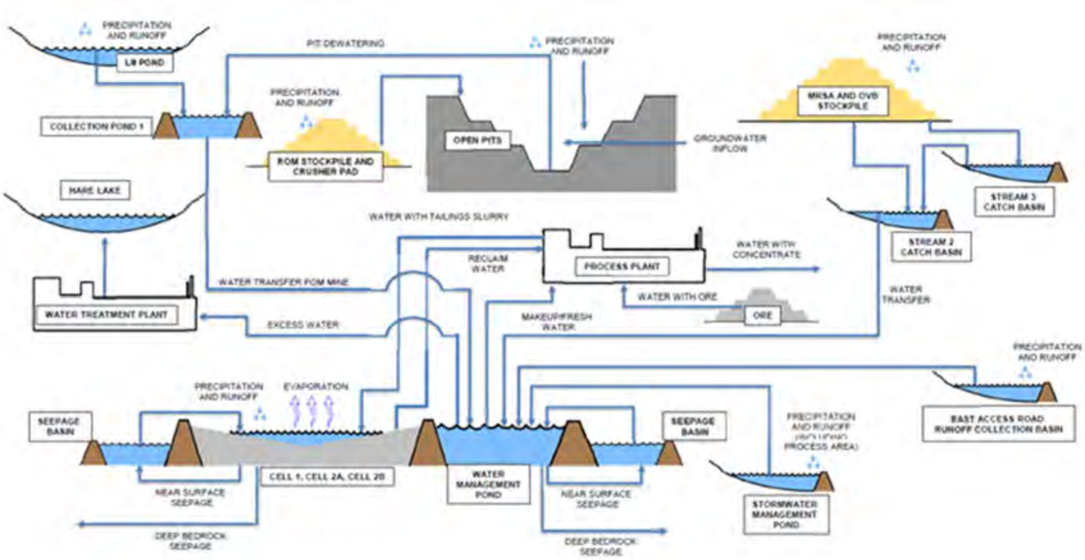
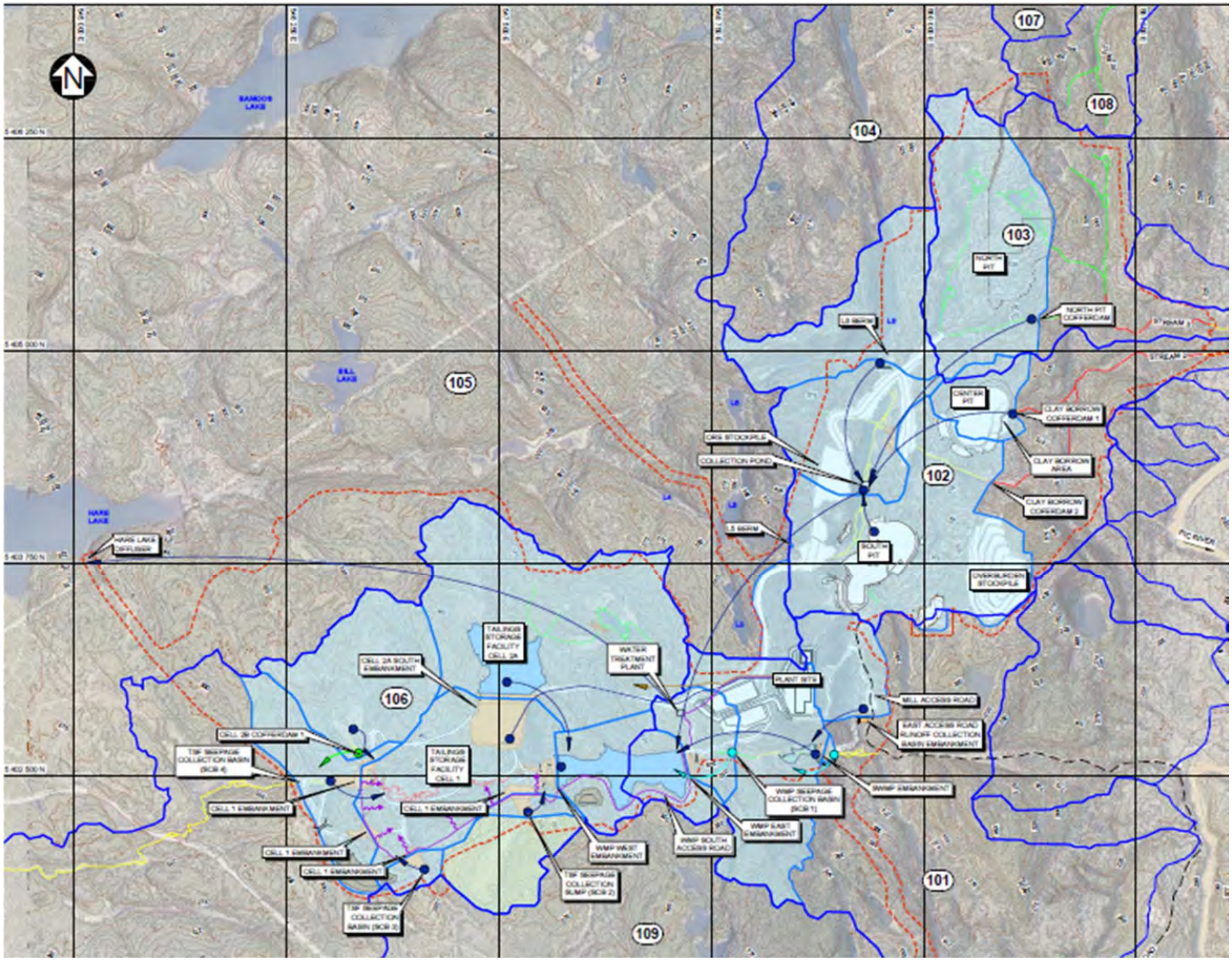


Figure 18.13: Surface Pipeline Network



18.4.3 Area 3300 Potable Water

Potable water will be trucked to site and stored in a potable water storage tank installed outdoors next to the process plant. The tank will be heated and insulated to prevent the water from freezing during the winter months. Water from the tank will be distributed to the offices area of the plant and to the safety showers water tempering skid by a containerized pumping module located outdoors close to the storage tank. The module will be provided with heating and ventilation.

18.4.4 Area 3400 Sewage Water

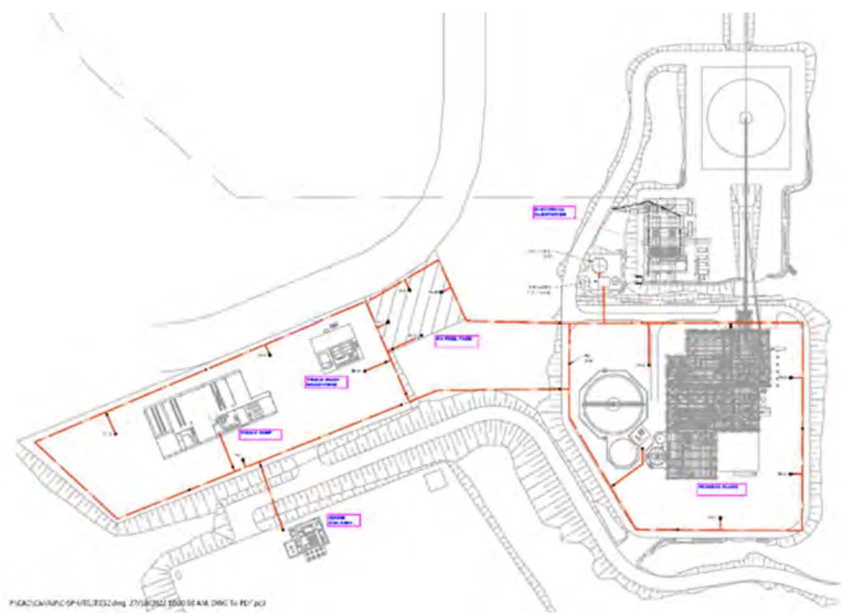
Sanitary waste from washrooms, sinks and floor drains will be collected by a buried sanitary drainage piping system and discharged into a sewage holding tank (400 bbl) used during construction.

The tank would be located outdoors above grade on the south side of the process plant at an elevation lower than the plant finished floor which allows the sanitary system to drain by gravity and utilize lift stations if necessary.

18.4.5 Area 3500 Fire Water

Figure 18.14 presents the fire water distribution main ring supplying outdoor fire hydrants and fire suppression (sprinkler and standpipe) systems inside the process plant, Administration, Truck Shop and Truck Wash buildings.

Figure 18.14: Fire Protection Piping Network



Fresh water will be supplied from WMP to an outdoor heated and insulated fire water storage tank. The supply of fire water from the fire water tank to the fire main ring will be provided by two fire pumps, one electric (main fire pump) and one diesel fire pump (backup pump) located inside a modular Fire Pump House as shown in Figure 18.14 above. A jockey pump installed together with the fire pumps will maintain the fire water system pressure at a design set point during all times. Based on current estimates the fire pumps are rated at 455 m3/h and the fire water storage volume, based on two hours of continuous operation of the fire pumps is 910 m3/h.

18.4.6 Area 3600 Water Treatment Plant

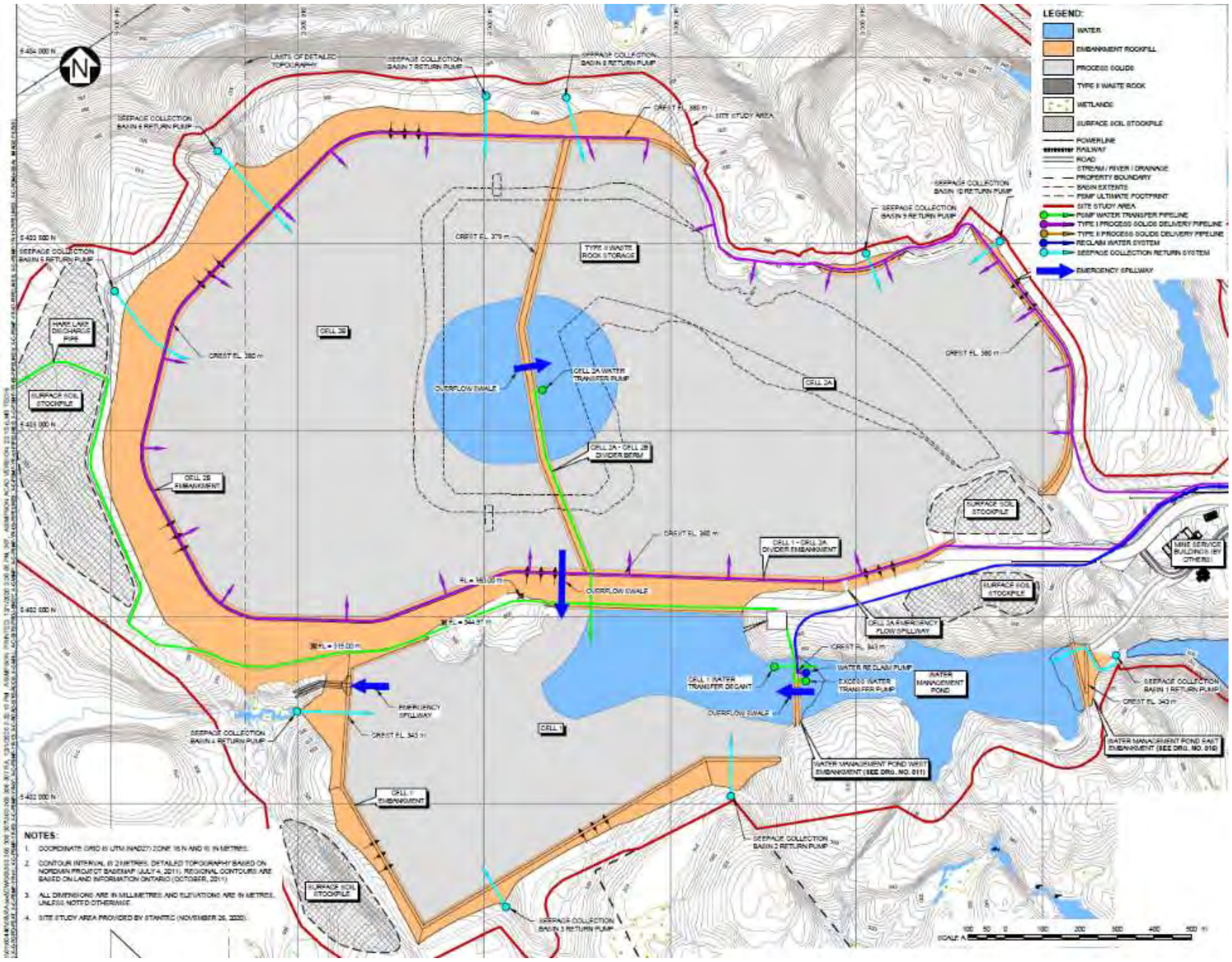
The WTP will be built as required in two phases. The first phase will occur in Year 1 after commercial production with an initial capacity of 520 m3/hr. A second phase, with equal capacity, will be constructed during Year 3 with completion in Year 4. The design parameters and criteria currently assumed are as outlined and validated in prior studies.

18.4.7 Area 3700 TSF

The process plant will produce two tailings' streams. The rougher tailings are NAG and are referred to as Type 1 material. The 1st Cleaner tailings are PAG and are referred to as Type 2 material. It is anticipated that approximately 85% of tailings will be Type 1 and approximately 15% of the tailings will be Type 2. The Type 1 tailings slurry will be thickened to about 55% solids by weight. The Type 2 tailings slurry will be about 22% solids by weight. The Type 1 and Type 2 tailings slurries will be conveyed from the process plant to the TSF via separate HDPE tailings delivery pipelines.

The TSF is located approximately 3 km west of the process plant as shown on Figure 18.15. The TSF will consist of a paddock style impoundment with three storage cells (Cell 1, Cell 2A and Cell 2B). A separate WMP will be constructed at the east side of Cell 1. The TSF perimeter embankments will be constructed using the downstream construction method with NAG mine rock sourced from the open pit. The TSF has been sized to store approximately 117 Mt of tailings and 31 Mt of PAG mine rock. During the last three years of operations, approximately 7 M m3 of Type 2 tailings will be deposited in the Central Pit.

Figure 18.15: Process Solids Management Facility



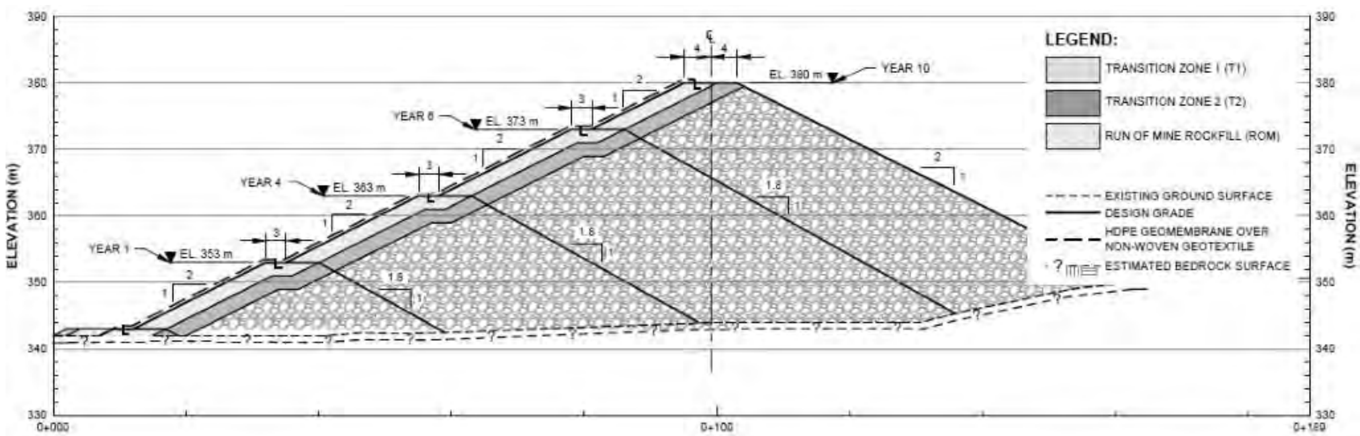
During the first three years of operations, Type 1 tailings will be deposited into Cell 1 and Type 2 tailings will be deposited towards the center of Cell 2A. Starting in Year 4, Type 1 tailings will be deposited into Cell 2A and Cell 2B, with Type 2 tailings continuing to be deposited towards the center of Cell 2A. After Year 10, Type 2 tailings will be stored in the Central Pit. PAG waste rock will also be placed with Cell 2A during the first 7 years of operations, Type 2 tailings and waste rock in Cell 2A will be covered with Type 1 tailings during the last 3 years to maintain Type 2 material in a saturated state to prevent the onset of acid generation.

The TSF embankments will be raised in stages to provide sufficient storage capacity for tailings and temporary water management. The final elevation of the dams ranges from 343 masl (Cell 1) to 380 masl (Cell 2A and 2B). The TSF embankments are constructed with upstream and downstream slopes of approximately 2H:1V and a minimum crest width of 8 m. The TSF arrangement utilizes site topography to reduce the size of the starter embankments. The final maximum embankment heights will range from about 43 m (Cell 1) to 80 m (Cell 2A) above the existing ground surface with foundation widths ranging from approximately 180 m to 330 m. The embankments will include specific rock fill zones with finer material towards the upstream portion of the embankment and coarser material towards the downstream portion of embankment. The embankment zones are filter-graded such that the embankments will not be susceptible to internal erosion or piping. The downstream rockfill zone consists of ROM rockfill and are resistant to downstream erosion.

The dams will include an HDPE liner on the upstream face of the embankments and the liner will be keyed into bedrock via a concrete plinth to minimize seepage from the TSF. Foundation preparation will include removal of overburden and unsuitable materials. Along the upstream toe of the embankments, below the concrete plinth, foundation preparation includes the removal of fractured bedrock, placement of slush grout on the prepared bedrock surface and / or injection grouting of deeper permeable bedrock zones to further reduce the potential for seepage from the TSF. Ten Seepage Collection Basins ("SCBs") will be constructed at select locations along the downstream toe of the embankments to intercept seepage. Collected seepage will be pumped back to the TSF. Monitoring locations downstream of the TSF will be established to confirm the effectiveness of the SCBs.

A typical cross section for the TSF perimeter embankments is shown in Figure 18.16.

Figure 18.16: Typical Cross Section for TSF Perimeter Embankments

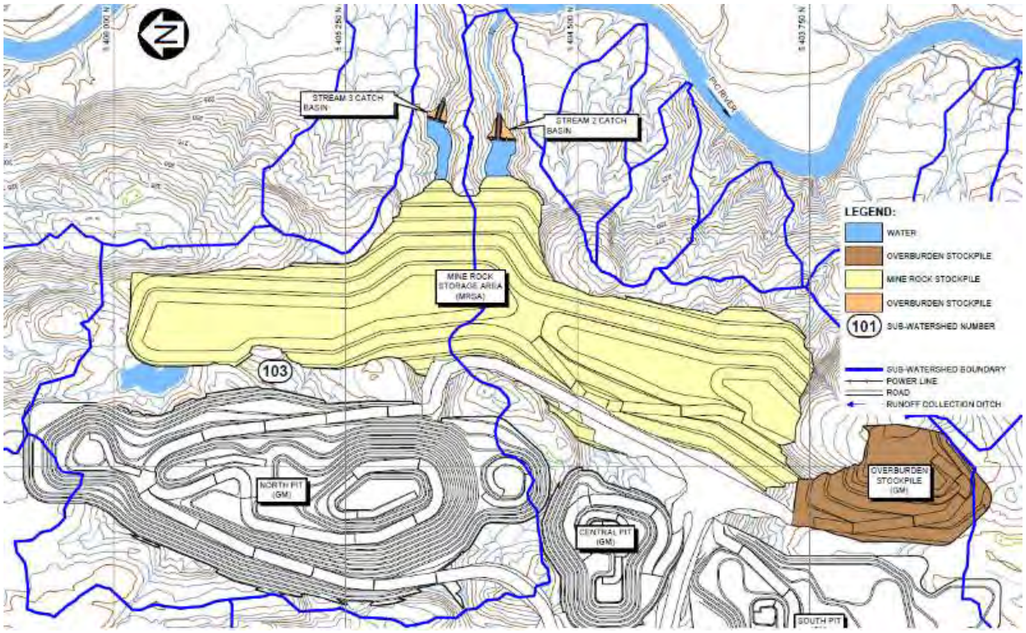


Supernatant water in the TSF will be reclaimed to the process plant for reuse in the process. Make-up water for the process will be drawn from the WMP and excess water in the WMP will be treated, as required, and then discharged to Hare Lake.

18.4.8 Area 3800 Mine Waste Rock Catch Basins

Runoff water and drainage from the MRSA will report to Sub-watersheds 102 and 103 (Stream 2 and Stream 3) which outlet to the Pic River. Two basins (Stream 2 Catch Basin and Stream 3 Catch Basin) will be established to collect contact water from the MRSA as shown in Figure 18.17. The Catch Basin embankments will be constructed as clay core rockfill dams that are designed to overtop during extreme meteorological events to prevent damage to the dams.

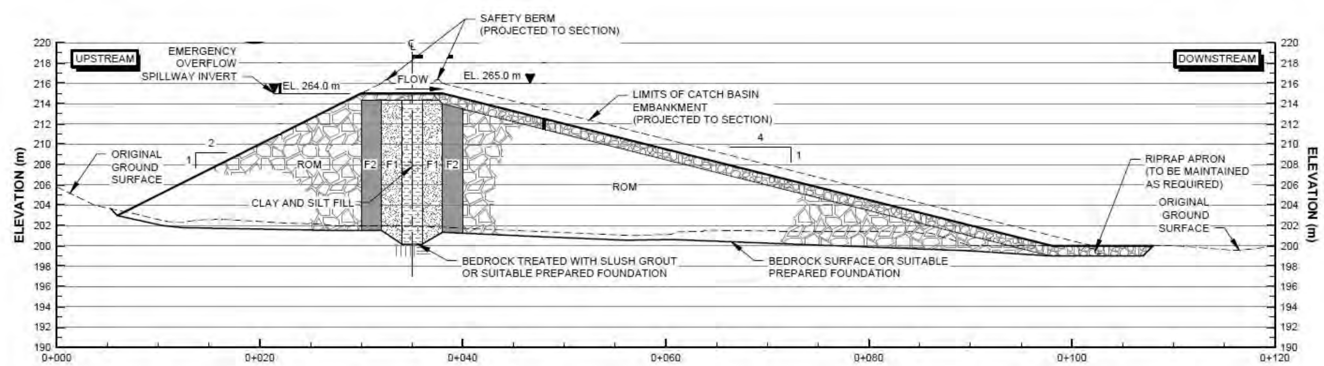
Figure 18.17: MRSA Catch Basin Locations



The Catch Basin embankments will be constructed using Type 1 mine rock from the open pits with a low permeability clay core and internal filter zones. The embankment materials will be sourced from the mine development, locally available borrow materials, and select processing where required. Material required for the filter zones and riprap will be produced on site by crushing and screening NAG mine rock. Silt and clay for the low permeability core will be excavated from local borrow areas within the MRSA and open pit footprint. The embankment core and shell will be founded on prepared subgrade. Foundation preparation will include the removal of organics and unsuitable materials. The embankment core will be keyed into the foundation to minimize seepage. Foundation preparation below the embankment core may include the removal of fractured bedrock and grouting to minimize seepage. Shear keys may be installed within the foundations to maintain embankment stability.

The embankments has 2H:1V upstream slopes and 4H:1V downstream slopes and a 6 m wide crest. The maximum embankment height will be about 21 m. An overflow spillway will be installed on the crest and downstream slope of each embankment. The spillway will be lined with riprap and will outlet to a riprap / boulder apron to dissipate energy. A typical cross section for the MRSA dams is shown in Figure 18.18.

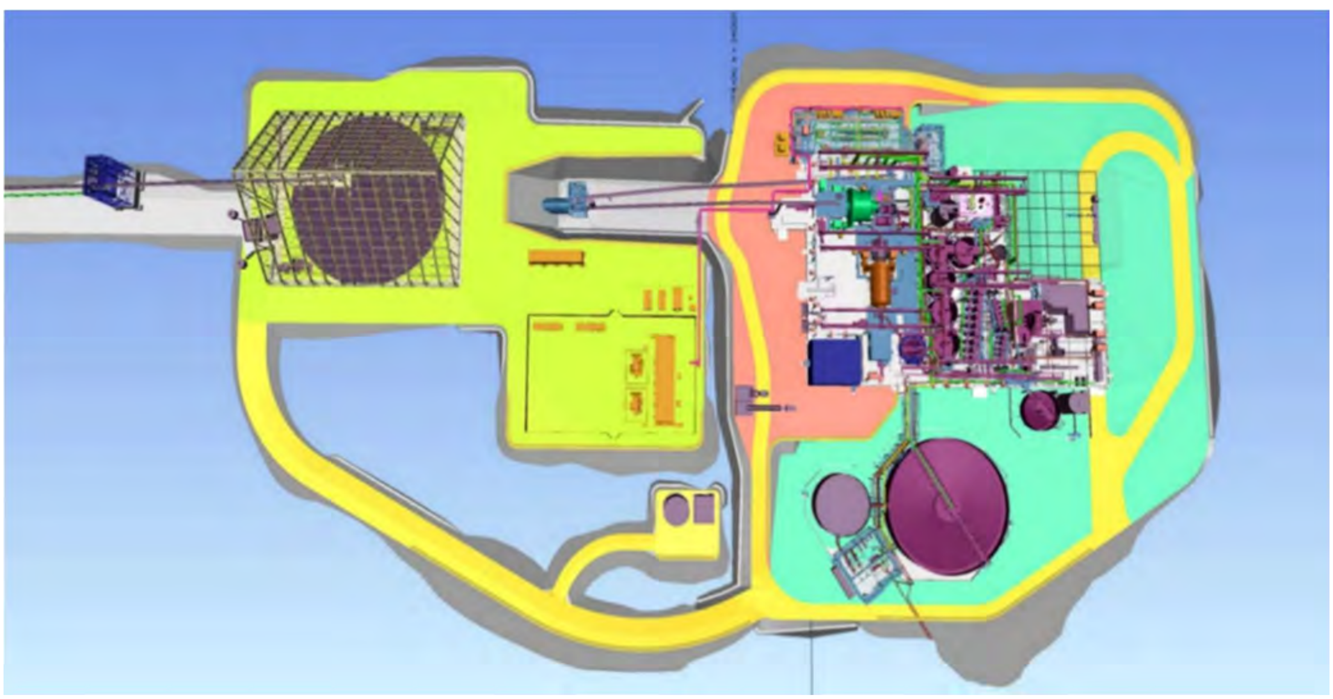
Figure 18.18: Typical Cross Section for MRSA Catch Basin Dams



18.5 Area 6900 Process Plant Infrastructure

The processing area consists of the main processing building and support infrastructure, as shown in Figure 18.19.

Figure 18.19: Process Plant Site



The process plant consists of the buildings shown in Table 18.2.

Table 18.2: Process Plant Building List

Building Description	Building Construction	Additional Description	L (m)	W (m)	H (m)	Area (m^2)	Volume (m^3)
Grinding Building	Pre-Engineered Building	Metal Cladded	69.2	47.0	31.0	3,252	100,824
Floc Building (Flotation Area)	Pre-Engineered Building	Metal Cladded	77.2	41.2	32.4	3,180	103,052
Stockpile A-Frame Building	Pre-Engineered Building	Fabric Cladded A-frame	65	78	31.5	5,070	159,705
Concentrate Loadout Building including storage, drive through and reagents	Pre-Engineered Building	Metal Cladded	41.6	14.70	28.7		

The grinding building is serviced by a 75 t overhead crane for the ball mill and SAG mill having 44.2 m span. Primary cyclone building serviced by 10 t overhead crane having 14.2 m span.

The flotation area building is serviced by a 35 t overhead crane having 39 m span which would serve HIG mill as well. Concentrate building pressure filter area serviced by a 5 t overhead crane having 11.5 m span.

18.5.1 Area 6960 Process Plant Reagents Storage Facilities

The reagent storage facilities are located both indoor and outdoor of the main process plant building. Indoor section is located within Concentrate Storage building which consists of a metal cladded pre-engineered building totalling 280 m^2 and the outdoor section is located west of Concentrate building. The indoor section, specifically the flocculant and coagulant tanks are serviced by a 5 t overheard crane which is primarily used for Pressure filter.

18.5.2 Area 6930 Process Offices and Control Room

The Service building (mill office, lunchroom, change room) consists of 12 prefabricated steel stackable modules placed on the east side of the Grinding building. This building has the following exterior dimensions: 25.6 m (L) x 16.5 m (W) x 8.1 m (H).

Plant Control room (modular) is located on the second level north of the Primary Cyclone platform inside Grinding building with the dimensions of 9.7 m (L) x 8.4 m (W) x 4.2 m (H).

The Met lab room (stick built) is located on the ground floor northwest section of the Grinding building (south of the Service building) with the dimension of 15.3 m (L) x 8.6 m (W) x 4.2 m (H).

Refer to Figure 18.20 to Figure 18.22 below for proposed Service building, on-site metallurgical laboratory, control room and offices.

Figure 18.20: Control Room Layout

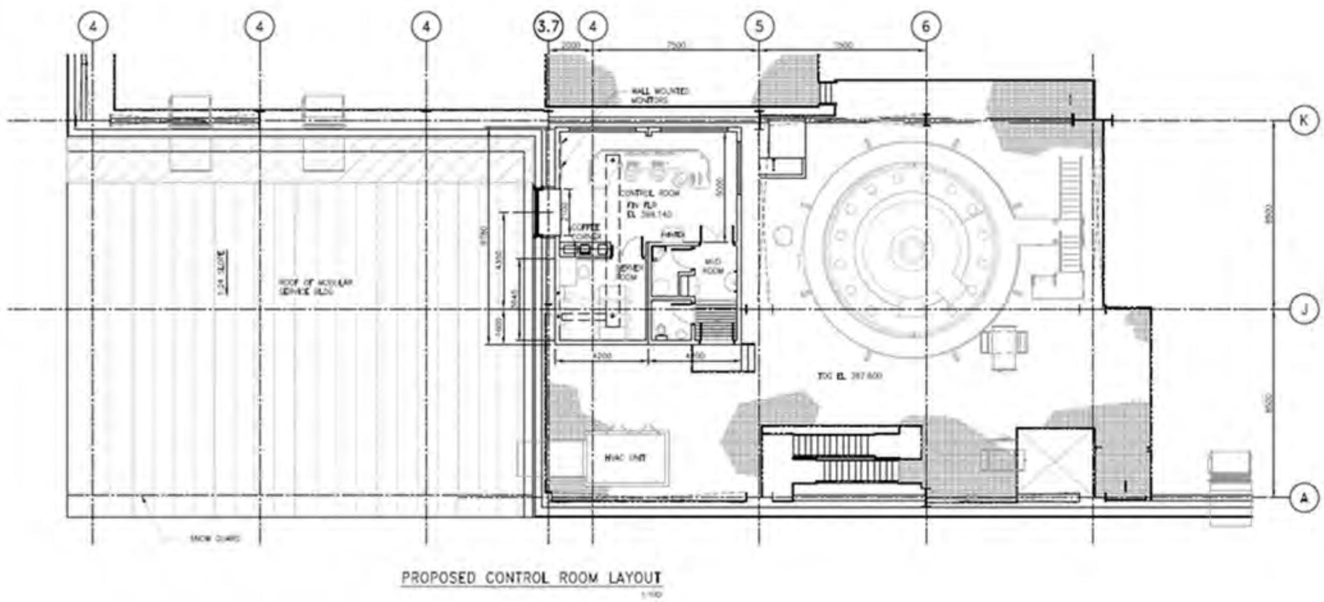


Figure 18.21: Service Building and Met Lab Layout

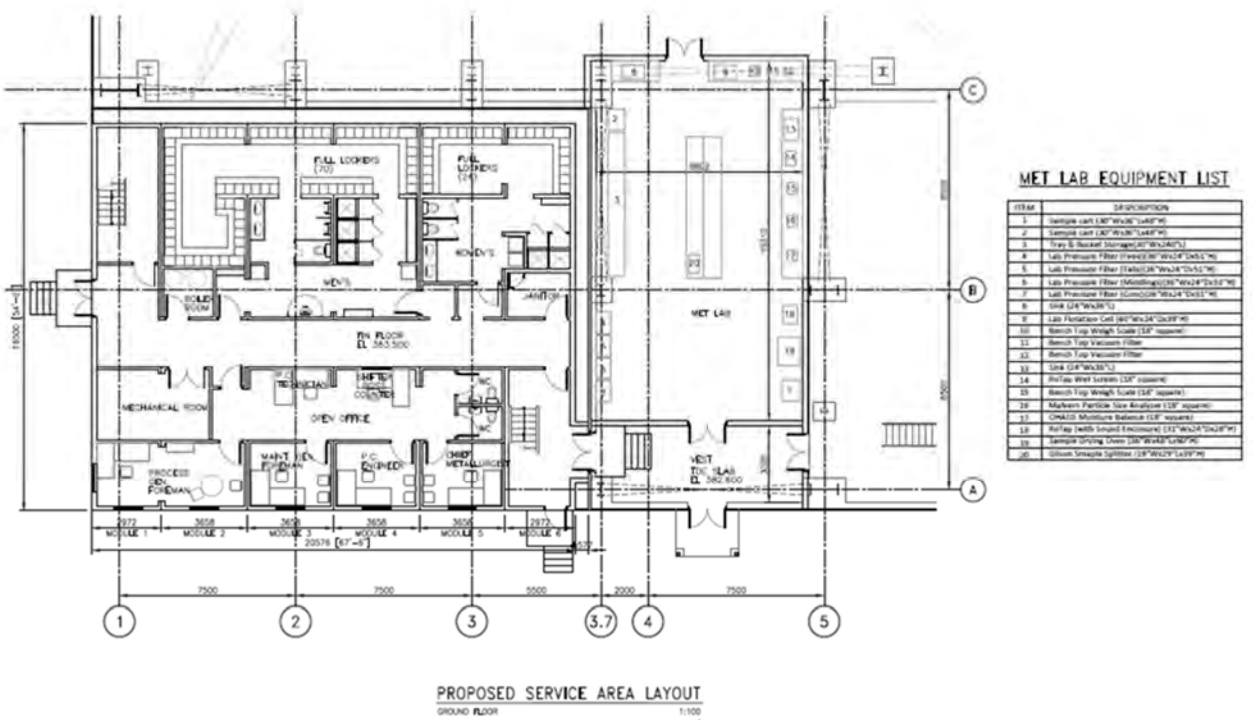
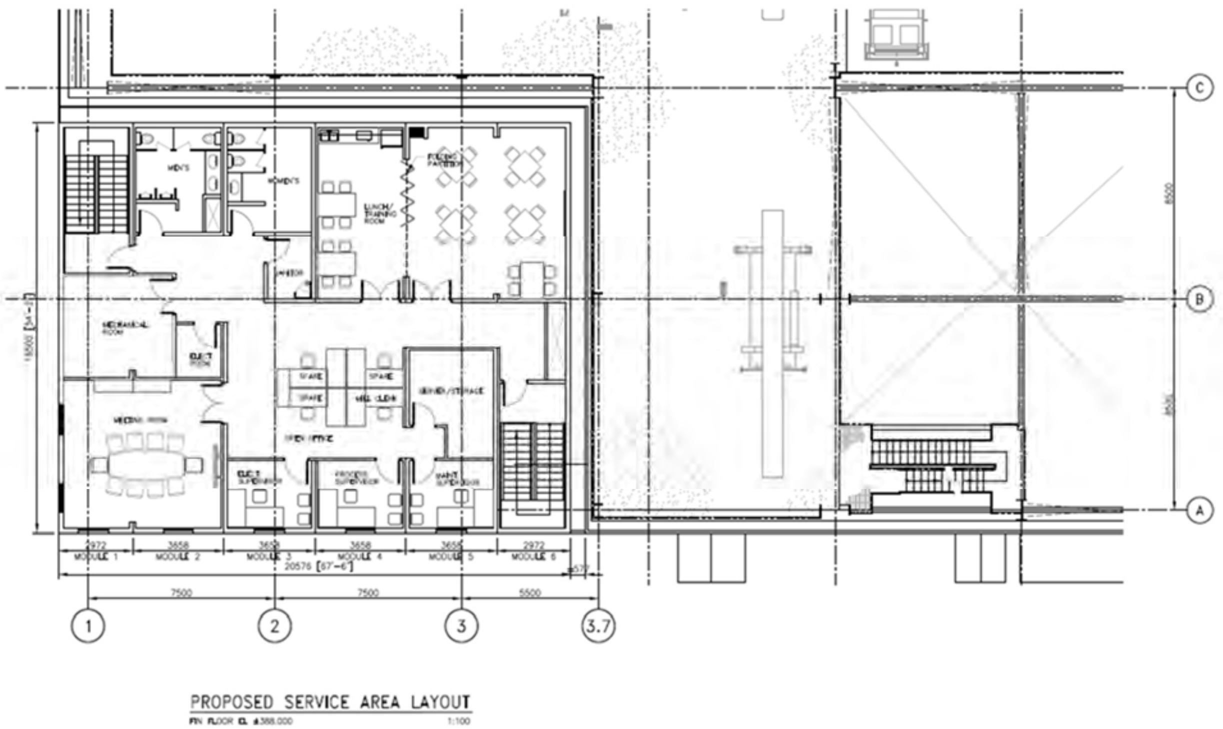


Figure 18.22: Service building layout - Offices



18.6 Construction Temporary Infrastructure

Permanent infrastructure, including pads and structures, will be utilized to support the construction phase where practical.

When permanent infrastructure is not practical, temporary modular construction offices, lunchrooms and washroom facilities will be utilized.

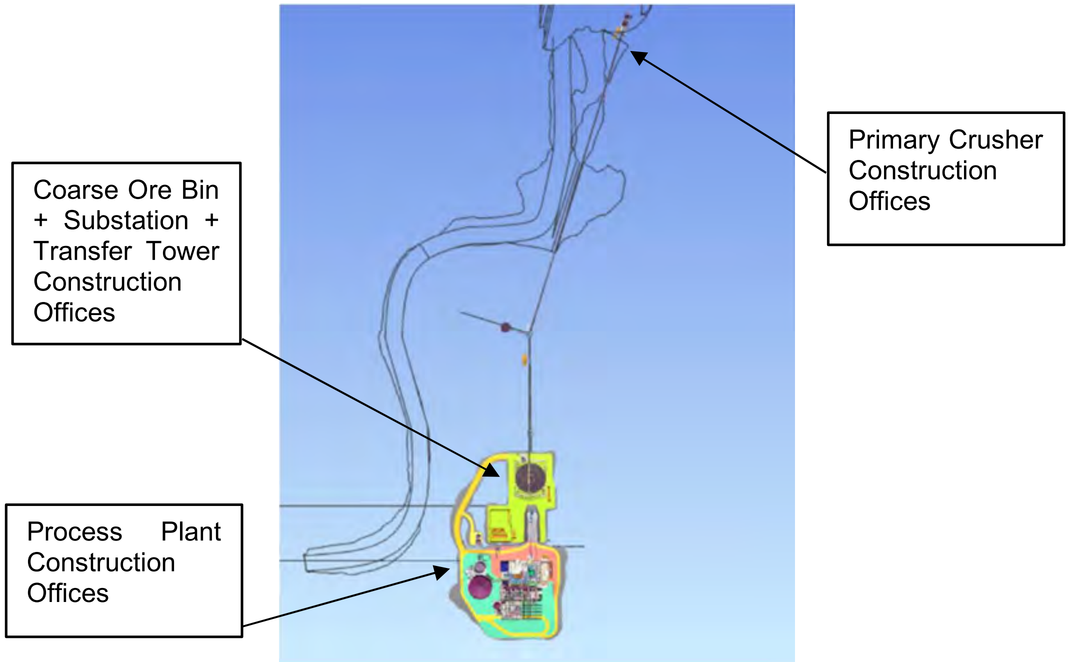
Areas requiring temporary infrastructure to support construction activities is noted below:

- Primary Crusher
- Coarse Ore Bin, Substation, Transfer Tower
- Process Plant
- Mine Services Area Pad
- Aggregate Production Pad (Crusher)
- Emulsion Plant Area (Drilling and Blasting Pad)

The temporary facilities for the process plant area are detailed below in Figure 18.23 and Figure 18.24.

The initial site laydown pad will be located on the Mine Services Area pad where the truck shop will be located. This pad will house construction offices, shop / warehouse and be the general receiving area for all construction materials (Figure 18.25).

Figure 18.23: Process Plant Area Overview



For reference, the temporary facilities for the process plant area are detailed below in Figure 18.24.

Figure 18.24: Process Plant Area Facilities

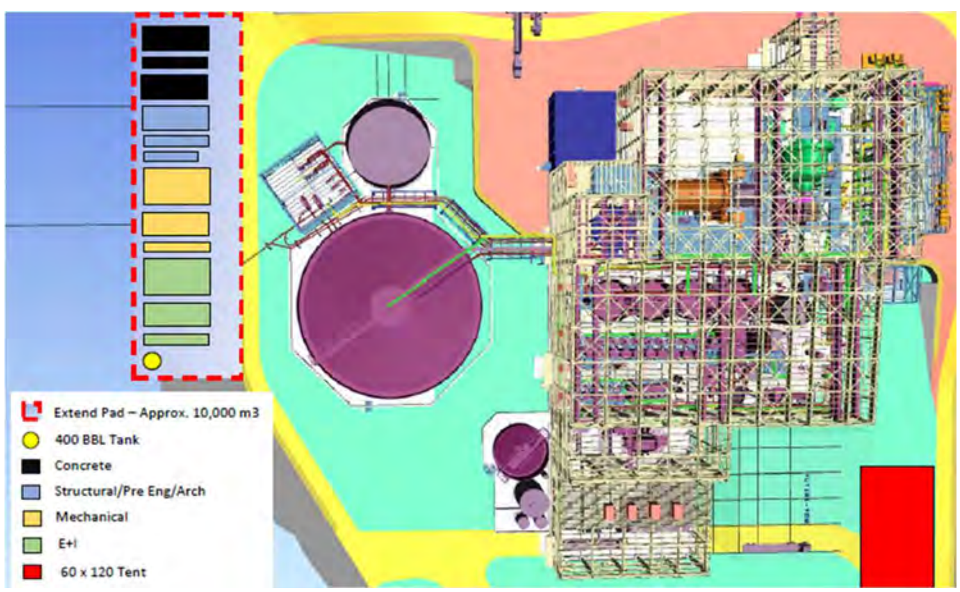
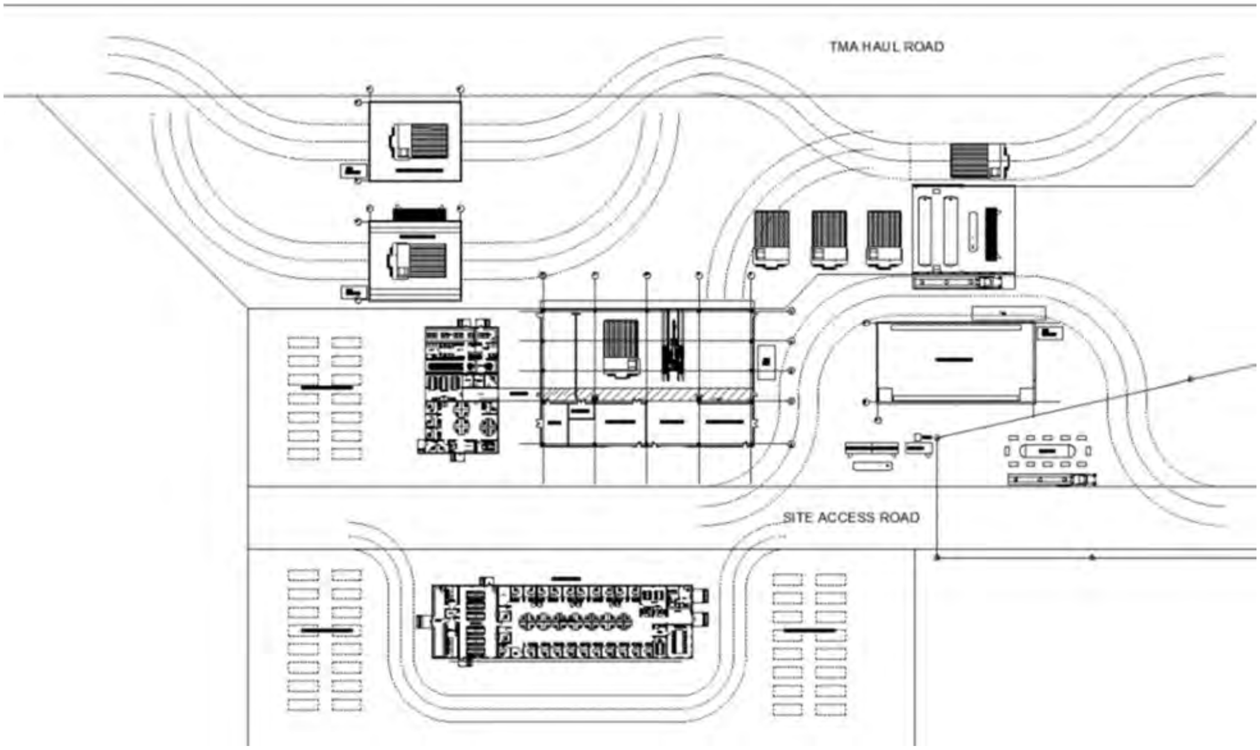


Figure 18.25: Mine Services Area Pad



18.6.1 Construction Offices

18.6.1.1 Mine Maintenance Offices

The offices for the mine maintenance department will be mobilized to site and installed near the truck shop on the mine services area pad. The offices will be constructed utilizing 5 to 7 modular office units that will be installed and connected on site. The mine maintenance office complex will feature a combination of closed and open offices, as well as a boot room/dry for maintenance personnel usage.

The offices will be installed early in the construction phase to support the construction management team and will then be utilized to support permanent mine maintenance operations.

18.6.1.2 Admin Offices

The administration building is located close to the mine service building facility on the mine services area pad. It is a single storey building that houses the human resources, general administration, mine management and operations, engineering and geology departments.

The administration building will feature between 15 and 20 office modules transported and connected on site.

18.6.1.3 Contractor Offices

Extra space for contractor or external engineering firms will be set aside on the various pads near work fronts as they are established.

Modular skidded offices (10 to 15), lunchrooms and wash cars will be staged on the mine services area pad to support the various earthworks contractors' operations.

18.6.2 Construction Water Management

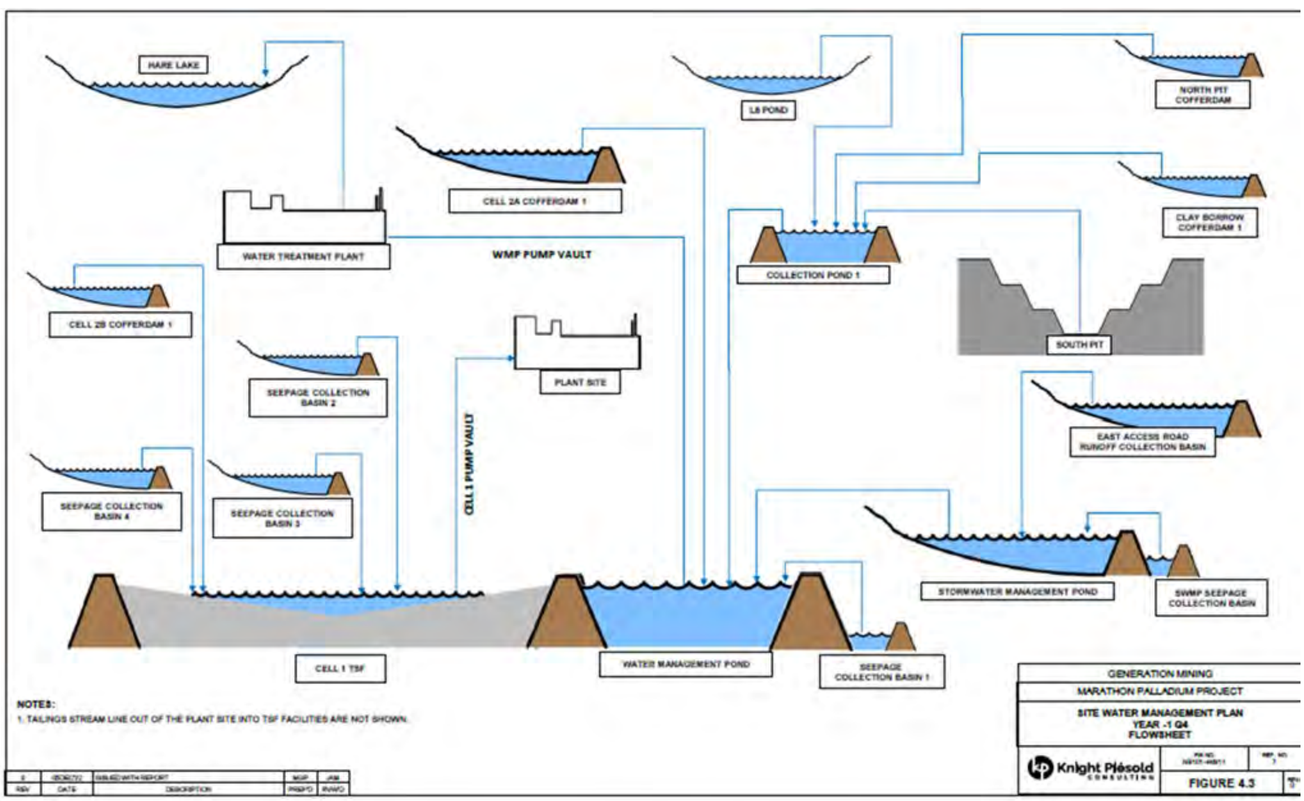
Contact surface water from the various construction areas for the water retention structures, access roads and haul roads, pads, borrow sources and laydowns, as well as the mining operations areas such as the South Pit and MRSA, will be collected, managed as required, and discharged or otherwise controlled.

A network of pipelines and pumps will be utilized to store, move, and otherwise manage the water to ensure all contact water is contained and managed as per the applicable regulations.

All collected surface water will be pumped to the WMP and either reclaimed to the process plant area for reuse in the process plant or treated and discharged to Hare Lake.

Figure 18.26 shows the pumps and pipelines system for construction surface water management.

Figure 18.26: Construction Water Management



18.6.3 Construction Camp

Gen Mining has an option to acquire the existing Valard construction camp located in the town of Marathon which will be utilized for the Project (Figure 18.27). The Valard camp was used to house construction workers for the recently completed East-West Transmission Line Project. Its capacity is 286 people and contains a kitchen and dining room (both will require upgrading to increased capacity), recreation facilities, laundry and sleeping quarters that share a bathroom with the adjacent room ("jack-and-jill"). An addition of approximately 500 beds of rental camp will be developed near the Project site in partnership with the BN to supply the balance of the construction beds required.

Figure 18.27: Marathon Construction Camp

19. MARKET STUDIES AND CONTRACTS

19.1 Copper-PGM Concentrate Sales

The proposed operation will produce a copper-PGM concentrate, which will then be transported and sold to domestic and/or international smelters.

The concentrate produced is expected to be low in deleterious elements commonly found in copper concentrates (i.e., lead, zinc, arsenic, antimony, mercury, bismuth) and is not expected to draw significant penalties. Fluorine and MgO penalties may occur in some conditions; however, this is not expected to be persistent, and ore-feed-blending is expected to meet smelter requirements.

In the 2021 metallurgical testing program a mini-pilot plant was built and operated at SGS-Lakefield (see Section 13 – Mineral Processing and Metallurgical Testing) to test representative samples from site mineralization. The mini-pilot plant produced a volume of concentrate. Samples of the concentrate were sent to international smelters and commodity traders that would be capable of receiving or processing a poly-metallic, copper concentrate and recovering the expected levels of PGMs. This allowed for the Project to tender the marketing (off-take) for the expected Cu-PGM concentrates from the operating plant.

Firm term sheets have been received from both domestic and international smelters with competitive treatment charges, refining charges (“TC/RC”) and payability terms are considerably higher than those found in typical copper concentrates (typically at trace levels). Copper smelters recover PGMs to copper anodes with subsequent electrorefining yielding by-product gold, silver, platinum, palladium and other metals from refining anode slimes.

The resulting terms are reflective of the high value per tonne of the concentrate and potential for higher margins than traditional clean copper concentrates.

Final payment terms will be based on prevailing metal prices from the London Metals Exchange (copper) and the London Bullion Market Association (palladium, platinum, gold and silver), subject to payabilities and minimum deductions. The economic model assumes a 50/50 blend of TC/RCs and payability terms between two smelters where the product is envisioned to be sold. A summary of the payment terms and costs is presented in Table 19.1 and Table 19.2.

Table 19.1: Payable Metals in Concentrates

Payable Element	Approximate Net Payable Rates (%)	Minimum Deductions
Palladium	95%	2.6 g/t
Copper	96.5%	1.1%
Gold	75%	1 g/t
Platinum	77%	2.6 g/t
Silver	75%	30 g/t

Table 19.2: Treatment and Refining Charges

Element	Treatment Charge	Refining Charge
Palladium	-	US$24.50/oz
Copper	US$79/dmt	US$0.079/lb
Gold	-	US$5.00/oz
Platinum	-	US$24.50/oz
Silver	-	US$0.50/oz

19.2 Precious Metal Purchase Agreement

In December 2021, the Company entered into a definitive PMPA with Wheaton. The PMPA became effective in January 2022, upon closing on the acquisition of the remaining interest in the Marathon Property from Sibanye-Stillwater to hold a 100% interest in the Marathon Project. The key terms of the PMPA are as follows:

- Wheaton will pay Gen Mining total cash consideration of $240 million, $40 million of which has been paid on an early deposit basis prior to construction to be used for development of the Marathon Project, with the remainder payable in four staged installments during construction, subject to various customary conditions being satisfied.
- Under the Marathon PMPA, Wheaton will purchase 100% of the payable gold production until 150,000 oz have been delivered, thereafter dropping to 67% of payable gold production for the LOM; and 22% of the payable platinum production until 120,000 oz have been delivered, thereafter dropping to 15% for the LOM.
- Wheaton will make ongoing payments for the gold and platinum ounces delivered equal to 18% of the spot prices (“Production Payment”) until the value of gold and platinum delivered less the Production Payment is equal to the consideration of $240 million, at which point the Production Payment will increase to 22% of the spot prices.

- Gen Mining and its subsidiary Gen PGM has provided Wheaton with corporate guarantees and other security over their assets, and will be subject to certain customary penalties and/or events of default if they fail to comply with the terms of the PMPA

The terms of this PMPA have been incorporated in the economic analysis in Section 22 .

19.3 Commodity Price Projections

As commodity prices are highly volatile, and based on what appears to be the beginning of a recessionary environment, Gen Mining does not believe spot prices at the time of this Technical Report are reflective of long-term fundamentals. When evaluating commodity price forecast, Gen Mining and its consultants have considered multiple sources of information including, but not limited to:

- Recent spot price trends, as well as the 2, 3-year and 5-year trailing averages
- Understanding of key macro level demand drivers for palladium and copper
- Understanding of existing and new mine and recycling supply sources
- Understanding of key geopolitical changes and sensitivities around source of supply
- Consensus forecasts where available

For this Technical Report, the metal prices presented in Table 19.3 were used for base case economic model, which are based on the lesser of spot price and 3-year trailing average ending December 31, 2022 of the respective metals (rounded):

Table 19.3: Metal Price and FX Assumptions for Economic Analysis

Metal	Price (US$)
Palladium (oz)	$1,800
Copper (lb)	$3.70
Gold (oz)	$1,800
Platinum (oz)	$1,000
Silver (oz)	$22.50
C$:US$ foreign exchange rate	1.35

Note: *Project economic sensitivities to changes in metal prices are evaluated in Section 22 – Economic Analysis.*

19.4 Material Contracts

As of the date of this Technical Report, Gen Mining has entered into the following material contracts related to the support of the development of the Project:

- Wheaton PMPA
- Agreement with Wood Canada Limited, an engineering consulting company, for the processing plant engineering and long lead-time procurement services ("EP Services")
- Agreement with Endeavour Financial to provide project financing advisory services
- Agreement with Hycroft Mining Holding Corporation for the purchase of an unused, surplus SAG mill and ball mill and the primary transformer and sub-station
- Agreement with Valard Equipment LP for the lease of an installed construction camp located in Marathon and an option, exercisable at Gen Mining's discretion, to purchase the camp on or before the end of the lease term

Additional major engineering, procurement and construction management contracts are envisaged as per the Project execution plan in Section 24 – Other Relevant Data and Information of this Technical Report.

Gen Mining has begun discussions with customers, suppliers and equipment manufacturers to support the operations. The following material contracts are expected to be executed to support the proposed operations:

- EPCM services for construction
- Power (electricity)
- Diesel, oil and lubricants
- Process reagents
- Explosives
- Camp services
- Mobile equipment
- Transportation
- Concentrate off-take

19.5 Comments on Market Studies and Contracts

The QP is of the opinion that the marketing and commodity price information is suitable to be used in the economic analysis of this Technical Report.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Regulatory Approvals

20.1.1 Regulatory Framework

The Environmental Assessment ("EA") for the Project was approved on November 30, 2022 in accordance with the Canadian Environmental Assessment Act ("CEAA", 2012) and Ontario's Environmental Assessment Act ("EA Act") through a Joint Review Panel ("JRP") pursuant to the Canada-Ontario Agreement on Environmental Assessment Cooperation (2004).

As of the effective date of this Technical Report, the Project is in the process of obtaining various Federal, Provincial and municipal permits, approvals and licenses as required to construct and operate the Project.

Environmental Assessment

The former Minister of the Environment referred the Project to a JRP on October 7, 2010 under the CEAA. The assessment continued under the CEAA, 2012. In 2014, the EA was placed on hold by the proponent at the time (Stillwater Canada) and the JRP was disbanded. In 2020, Gen Mining resumed the EA process and the Crown appointed new members to the JRP on November 16, 2020 to continue the assessment. The JRP conducted its review in a manner that met the requirements of the CEAA, 2012 and submitted its report to the Minister of Environment and Climate Change on August 2, 2022. Following the submission of the JRP report, the Federal and Provincial governments independently issued decision statements that outlined specific mitigation and monitoring conditions to protect the environment and specified follow-up program and reporting requirements.

These conditions include measures to address the effects of the Project on the current use of lands and resources for traditional purposes by Indigenous peoples, physical and cultural heritage and the health and socio-economic conditions of Indigenous peoples, as well as fish and fish habitat, migratory birds and species at risk, including woodland caribou. The decision statement also requires the proponent to develop and implement a reclamation plan for restoring the project site once operations have ended and the mine has been decommissioned.

Numerous conditions include the requirement to consult with Biigtigong Nishnaabeg and other Indigenous groups with respect to the development and implementation of mitigation and monitoring plans. In some cases, the conditions also require the Company to "seek consensus" with Biigtigong Nishnaabeg. This includes, for example, the content of the final reclamation plan and its implementation.

A total of seven Indigenous groups actively participated in the EA process, including the public hearing, and informed the JRP Report. Consultation undertaken by the Crown with these groups resulted in a number of accommodation measures to address potential impacts to established or asserted rights, as recognized and affirmed by Section 35 of *The Constitution Act, 1982*.

Information related to the Federal and Provincial processes is available as follows:

Federal

The Canadian Impact Assessment Registry file number for the Project is 54755 and the current internet address for related information is:

https://iaac-aeic.gc.ca/050/evaluations/proj/54755

Provincial

The provincial EA reference number is 11010 and the internet address for related information is:

https://www.ontario.ca/page/marathon-platinum-group-metals-and-copper-mine-project

20.1.2 Permits

A list of potential Federal, Provincial, and municipal approvals, permits, and/or authorizations required for the Project to move forward beyond the EA phase is provided in Table 20.1, Table 20.2, and Table 20.3, respectively.

Table 20.1: Potential Federal Approvals, Permits and/or Authorizations for the Project

Approval / Permit / Authorization	Rationale
Fisheries Act, Paragraph 35(2)(b) Authorization Legislation: *Fisheries Act* Responsible Agency: Fisheries and Oceans Canada	Project development will result in harm to fish and fish habitat for which offsetting measures are required.
Metal and Diamond Mining Effluent Regulations Legislation: *Fisheries Act* – Metal and Diamond Mining Effluent Regulations Responsible Agency: Environment Canada and Climate Change	Watercourses (or portions thereof) that are frequented by fish will be used for long-term storage of process solids and/or mine rock and or the management of contact water.
Navigation Protection Program (“NPP”) Approval Legislation: *The Canadian Navigable Waters Act* Responsible Agency: Transport Canada	The development of mine-related infrastructure including the open pits, mine rock storage area, process solids management facility and site road network may require approval under the NPP.
Licence for a Factory and Magazine for Explosives Legislation: *The Explosives Act* Responsible Agency: Natural Resources Canada	The proposed development includes facilities to store and supply nitrogen-based explosives that will be used for the purpose of excavating the ore body.

Table 20.2: Potential Provincial Approvals, Permits and/or Authorizations for the Project

Approval / Permit / Authorization	Rationale
Closure Plan approval in accordance with Schedule 2 of O. Reg. 240/00 Legislation: *Mining Act* Responsible Agency: Ministry of Mines	An approved Schedule 2 Closure Plan is required for the Project prior to starting construction.
Domestic Processing Exemption Legislation: *Mining Act* Responsible Agency: Ministry of Mines	An exemption under Section 91 of the Mining Act would be required in the event that ore was processed outside of Canada.
Environmental Compliance Approval (“ECA”) Legislation: *Environmental Protection Act* Responsible Agency: Ministry of the Environment, Conservation and Parks	An ECA is required for stationary source emissions, discharges and waste related to the Project, including air emissions, noise emissions, effluent discharges to water, stormwater management and waste disposal/transportation.
Permit to Take Water (“PTTW”) Legislation: *Ontario Water Resources Act* Responsible Agency: Ministry of the Environment, Conservation and Parks	A PTTW is required for instances where groundwater or surface water is taken at a rate of 50,000 L/d, or more. As it pertains to the Project a permit to take water will be needed for dewatering of the open pits and possibly for the development of groundwater well(s) for the supply of potable water.
Crown Land Work Permit Legislation: *Public Lands Act* Responsible Agency: Ministry of Natural Resources and Forestry	A work permit is required for Project related to construction on Crown Land, including dams, drainage channels, roads, culverts and bridges.
Lakes and Rivers Improvement Act Permit Legislation: *Lakes and Rivers Improvement Act* Responsible Agency: Ministry of Natural Resources and Forestry	A permit will be required for the construction of dams, water crossings, and diversion channels or enclosures.
Endangered Species Act Permit Legislation: *Endangered Species Act* Responsible Agency: Ministry of Environment, Conservation and Parks	A permit will be required for species at risk or its habitat that may be affected by the development of the Project. The potential effect of the Project on Woodland Caribou habitat has been assessed in this regard.
Aggregate Licence or Permit Legislation: *Aggregate Resources Act* Responsible Agency: Ministry of Natural Resources and Forestry	A licence may be required for the purposes of obtaining aggregate that is needed to develop Project infrastructure from borrow areas around the site study area (SSA) (Project footprint). The Company has a licensed aggregate quarry adjacent to the SSA.
Encroachment Permit Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	An encroachment permit would be required for construction of a transmission line over or under a Provincial Highway or within the highway right-of-way. A permit would also be required for any work within the highway right-of-way, including improvements to the highway itself required for the Project, specifically at the Highway 17 – site access road intersection.
Building and Land Use Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	Permits will be required for any development or construction within 45 m of the right-of-way limit of the highway and 395 m of the centre point of the intersection of a side road (such as the site access road) with Highway 17.
Sign Permit	A permit will be required for any sign erected within 400 m of the limit of the highway.

Approval / Permit / Authorization	Rationale
Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	
Licence to Operate a Bulk Storage Plant Legislation: *Technical Standards and Safety Act* Responsible Agency: Technical Standards and Safety Authority	A licence will be required for the purpose of operating a private bulk fuel storage and distribution system in the SSA.
Pre-Development Review and Approval Legislation: *Occupational Health and Safety Act* Responsible Agency: Ontario Ministry of Labour	The Ministry of Labour will subject the proponent to a safety and procedures review prior to the installation of portable crushing, screening or associated washing equipment.

Table 20.3 : Potential Provincial Approvals, Permits and/or Authorizations for the Project

Approval / Permit / Authorization	Rationale
Zoning By-law Amendment and Site Plan Agreement Legislation: *Planning Act* Responsible Agency: Town of Marathon	The Zoning By-law will need to be amended and a Site Plan agreement will need to be executed to permit mining operations.
Sewage Treatment System Permit Legislation: *Ontario Building Code* Responsible Agency: Thunder Bay District Health Unit/ Town of Marathon	A permit to construct an on-site private septic sewage system <10,000 L/d will be required.
Building Permit Legislation: *Ontario Building Code* Responsible Agency: Town of Marathon	A permit will be required for the construction of any Project buildings.

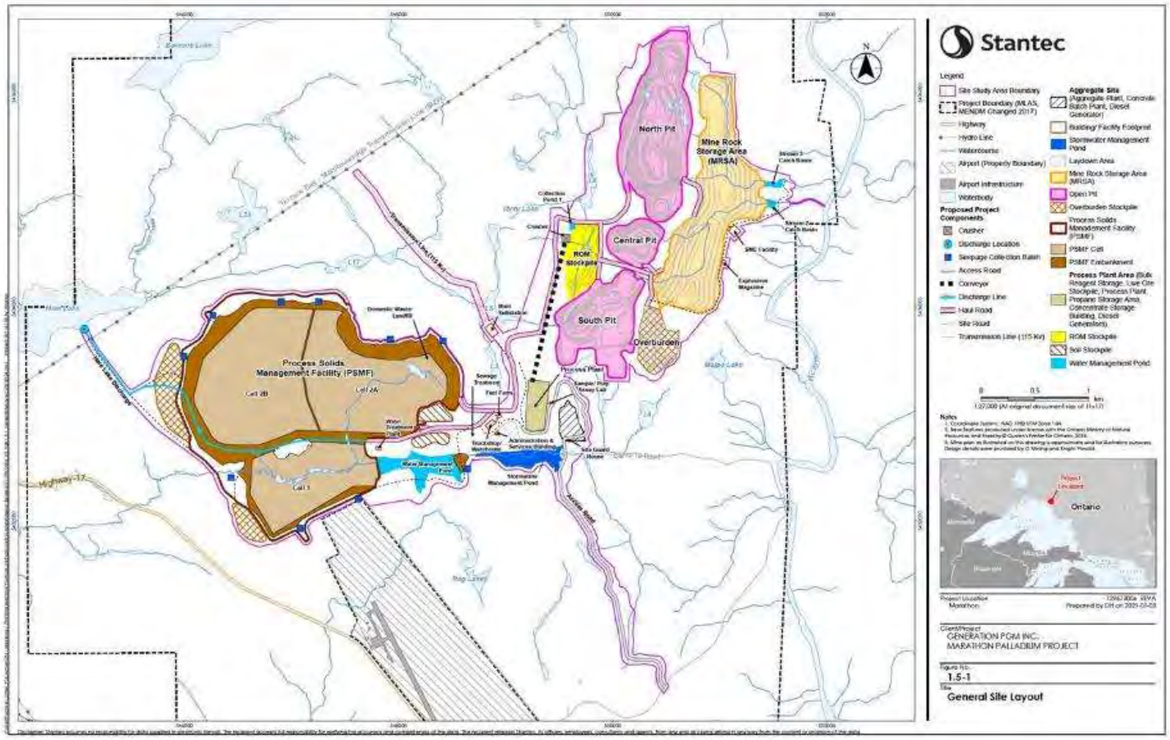
20.2 Environmental Studies

20.2.1 Background

The 2012 EA Report completed for the Project included the results of extensive environmental studies that were undertaken to characterize the Project site study area as shown in Figure 20.1. In 2021, an addendum was prepared to the 2012 EA Report to verify / and or update the studies in the 2012 report.

The results of the updated environmental studies presented in the 2021 EA Addendum Report are summarized in Subsections 20.2.2 to 20.2.12.

Figure 20.1: Marathon Project Site Study Area



Source: *Stantec (2021).*

20.2.2 Atmospheric Environment

20.2.2.1 Air Quality

Project activities may result in occasional short-term exceedances of some air quality guidelines and limits at the Property boundary though air quality meets applicable criteria at the nearest sensitive receptor locations. Fugitive dust emissions will mainly be generated from overburden and mine rock stockpiles, open pit mining activities and from operation of heavy and light vehicles on site roads. Proposed mitigation measures include applying water and/or other dust suppressants to active mining areas and roads. Air quality monitoring will assess the accuracy of water quality predictions and identify if any additional mitigation measures are required.

Light

Increased light levels from the site could potentially be visible to offset receptors. Proposed mitigation measures include using directional lighting and mounting lights as low as possible.

20.2.2.2 Greenhouse Gas

Greenhouse gas (GHG) emissions from site activities are predicated to have a negligible contribution to provincial and national CO_2 emissions and the associated phenomenon of climate change. The Project has been designed and will be operated to minimize GHG emissions to the extent possible.

20.2.3 Acoustic Environment

Noise levels, resulting from Project activities, are predicted to be below the applicable Provincial and Federal criteria at representative noise sensitive receptors. Noise emissions will be generated by Project activities such as drilling, blasting, material handling, haul trucks, light vehicles, pollution control equipment, building exhaust fans and rail traffic. Proposed mitigation measures include purchasing vehicles and equipment that meet applicable noise suppression regulations, scheduling concentrate shipments during certain periods of the day and implementing an overpressure and vibration monitoring program at the site upon commencement of blasting operations.

20.2.4 Water Quality and Quantity

Project activities are predicted to result in a permanent lowering of the groundwater table as a consequence of dewatering the open pits. Changes in groundwater levels and flow direction as well as in recharge / infiltration due to Project activities are predicted to result in a change in groundwater quality relative to background concentrations. Groundwater is not used as a resource on or immediately near the mine site. Groundwater monitoring will assess the accuracy of water quality predictions and identify if any mitigation measures are required.

Project activities will result in changes to local hydrology which are predicted to result in a reduction or increase in flows and / or water levels in lakes / streams in the project area. A change in surface water quality relative to background concentrations is expected due to the changes in subwatershed area associated with the Project and management of water at the site. Mitigation measures include sediment and erosion control measures, management of Type 2 (PAG) mine rock and tailings, collection and recycling of contact water for use in the process plant and treatment of contact water to meet applicable criteria prior to discharge. Surface water monitoring will assess the accuracy of water quality predictions and identify if any additional mitigation measures are required.

20.2.5 Fish and Fish Habitat

The development of the Project will have an effect on fish and fish habitat either as a result of the overprinting of approximately 10 ha of existing fish habitat or through a reduction in flow in streams in the Project area. Mitigation measures include implementation of fish habitat offsetting measures as specified by a *Fisheries Act* authorization, adhering to minimum setback distances to mitigate effects of blasting to fish, undertaking in-water works during time periods that are protective to fish and managing contact water to prevent the release of deleterious materials into lakes / streams. Fish and fish habitat monitoring will assess the effectiveness of the mitigation and offsetting measures.

20.2.6 Terrain and Soils

Project activities may result in the potential loss of stockpiled soil / overburden or a change in soil quality. Proposed mitigation measures include limiting the size of the SSA to the extent possible to minimize the need for soil/overburden excavation, stockpiling soil and overburden materials for later use in site rehabilitation activities, ensuring appropriate slopes for soil/overburden stockpiles to prevent erosion and slide hazards, and progressively rehabilitating disturbed areas as quickly as practical.

20.2.7 Vegetation

Project activities will result in the removal of approximately 1,081 ha of forest, 21 ha of open wetlands and an additional 10 ha of sparsely vegetated open water habitat within the SSA. Proposed mitigation measures for the SSA include using stockpiled soil and overburden for reclamation activities and re-seeding with non-invasive (and native, where practicable) plant species, isolating sensitive areas until native vegetation is established through reclamation activities and transplanting provincially or regionally rare plant species at suitable receiver sites. The effectiveness of the mitigation measures will be assessed periodically during the closure phase of the Project.

20.2.8 Wildlife

Project activities are predicted to result in the displacement of furbearer species, loss of actual and potential habitat for beavers, martens, moose, black bears, and forest dependent birds and the displacement of gray wolves. Wildlife habitat quality may also be affected due to dust fall deposition, spread of invasive species, increases or decreases in groundwater levels or changes to hydrology, and sensory disturbance from noise and vibration. Wildlife collisions with vehicles and wildlife collisions with Project infrastructure may also occur. Forest clearing for the Project will fragment wildlife habitat along the boundary of the SSA. Proposed mitigation measures include optimizing the location of Project components to reduce environmental impact including the area of vegetation clearing, incorporating existing disturbed areas into the SSA to accommodate Project components, using established best practices during site preparation and construction to reduce potential negative interactions with vegetation, undertaking progressive reclamation, using reflective markers on transmission lines over Canoe Lake and using directional lighting. Monitoring of wildlife habitat will be undertaken periodically during the closure phase of the Project.

20.2.9 Species at Risk

Project activities will result in the loss of caribou habitat within the SSA and may result in the loss of potential bat maternity roost habitat. Proposed mitigation measures for Species at Risk ("SAR") include conducting SAR awareness training, suspending construction activities if caribou are observed and notifying relevant regulatory agencies of the sighting, banning hunting within the SSA, using directional lighting, undertaking progressive reclamation, using native seed mixes during rehabilitation activities, providing bat boxes and rocket boxes as partial replacement for loss of potential roost trees and off-site mitigation for caribou elsewhere within the Lake Superior Coastal Range as authorized under the *Endangered Species Act*.

20.2.10 Socio-economics

Project activities are predicted to have both positive (employment, labour income, GDP and government revenue impacts, and business contracting potential) and adverse effects (loss of these positive benefits when the project transitions from operations to closure) with respect to impacts on the economy and employment. Loss of use of the SSA will also occur until reclamation activities have been completed and end-land use objectives have been achieved. Proposed mitigation measures include providing training opportunities to facilitate employment by residents within the Project area (including training of local youth and Indigenous community members), implementation of workforce transition strategies during decommissioning, use of an accommodations complex during construction and operation, engagement with municipal authorities to coordinate planning of infrastructure development or upgrades, providing funding support to key community services or organizations for fitness and recreational programs for workers, establishment of a Harvester Training Fund to support annual harvester and trapline training programs, and restricting of hunting, fishing, and harvesting of wildlife on the site. In addition to restoring the habitat within the SSA for use by wildlife (including SAR) and fish, and re-establishing access for hunting, fishing and trapping, end-land use objectives may also include other economic activities such as hydro-electric power generation.

20.2.11 Human Health

Project activities may result in occasional short-term exceedances of some air quality criteria for contaminants of potential concern ("CoPC") at the property boundary. With proposed mitigation and environmental protection measures such as the use of pollution control equipment (baghouses, scrubbers, etc.) and the application of amendments on stockpiles and gravel-surfaced roads to limit fugitive dust emissions, effects on human health from changes in air quality are not expected to be significant at any time during the Project.

Discharges to surface water during the Project are not expected to increase constituent concentrations in surface water in excess of water quality benchmarks for human health. Proposed mitigation measures include diversion of non-contact water around operational areas, recycling of contact water for use in the process plant and treatment of contact water to meet applicable criteria prior to discharge. As such, no adverse effects on human health are expected during any phase of the Project.

No adverse effects on human health are expected from groundwater affected by Project-related changes to groundwater quality because no existing or foreseeable groundwater users are located in the areas where groundwater quality is predicted to exceed Provincial and/or Federal drinking water standards. Proposed mitigation measures include minimizing the Project footprint and management of seepage from various Project components (TSF, Water Management Pond, Stormwater Management Pond, etc.).

With respect to country foods, there are minimal predicted Project-related effects on CoPC concentrations in the environment that would result in changes to CoPC concentrations in country foods in the Project area where country foods are likely to be harvested. Therefore, adverse effects on human health from country foods consumption are not expected from Project-related air and water emissions.

The electromagnetic fields (“EMFs”) from the proposed 2.2 km 115 kV overhead transmission line for the Project are not expected to adversely affect the health of people who visit or reside in the Project area. Power lines emit extremely low frequency EMFs (below 300 Hertz). The closest receptor to the proposed power line for the Project is a cottage on Hare Lake, which is located approximately 2 to 3 km from the proposed power line.

20.2.12 Physical and Cultural Resources

At this time no archaeological resources have been identified that would be affected by the Project. As such, no effects on archaeological resources are anticipated. Additional archaeological programs will be conducted in the Project area prior to site preparation activities, as required, to verify that no archaeological resources are present. A protocol will be implemented prior to initiating site preparation activities to protect archaeological resources in the event of a chance find.

There are no potential interactions between the Project and built or cultural heritage resources. Therefore, no effects on cultural heritage resources are anticipated.

20.3 Tailings, Mine Rock and Water Management

20.3.1 Tailings Management

The TSF is located approximately 3 km west of the process plant. An estimated 127 Mt (approx. 85 M m^3) of tailings will be generated over the LOM. The TSF is a paddock style impoundment with three storage cells (Cell 1, Cell 2A and Cell 2B). The TSF perimeter embankment will consist of lined rockfill embankments. Cell 2A and 2B are divided by an internal rockfill dyke to optimize tailings management and storage. Cell 1 and Cell 2 have been designed to store approximately 14 M m^3 and 64 M m^3 of tailings, respectively. Approximately 7 Mm^3 of Type 2 tailings will be stored in the Central Pit during the last few years of operation.

The process plant will produce two types of tailings, referred to as Type 1 or NAG and Type 2 or PAG. Type 1 tailings are anticipated to account for approximately 85% of the tailings from the process plant and have been determined to be NAG. Type 2 tailings are estimated to account for up to 15% of the tailings from the process plant and have been determined to be PAG. The Type 1 tailings slurry will be thickened to about 55-60% solids by weight. The Type 2 tailings slurry will be about 22% solids by weight. The Type 1 and Type 2 tailings slurries will be pumped from the process plant to the TSF via separate HDPE tailings delivery pipelines.

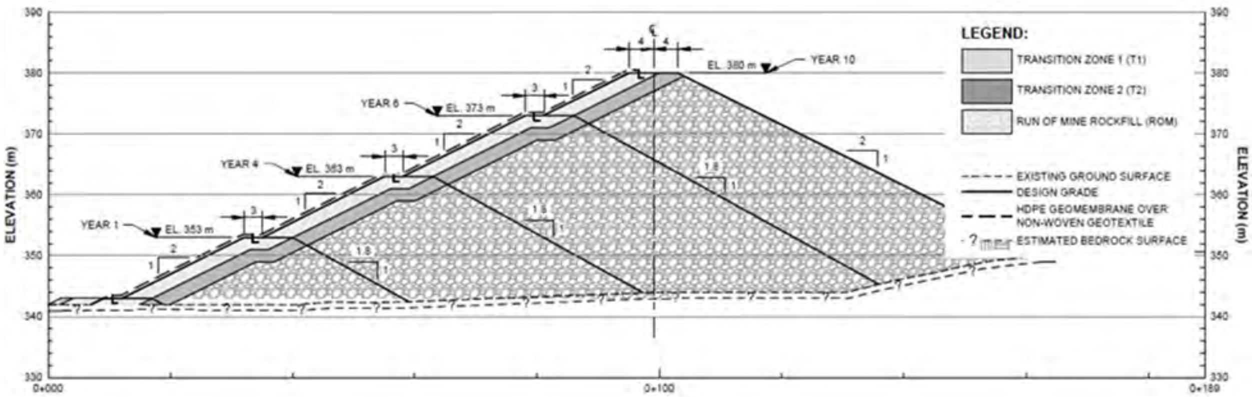
During the first three years of operation, NAG tailings will be deposited into Cell 1 and PAG tailings will be deposited into the centre of Cell 2A. Starting in Year 4, NAG tailings will be deposited into Cell 2A and Cell 2B, with PAG tailings continuing to be deposited into the centre of Cell 2A. After Year 10, PAG tailings will be stored in the Central Pit as the tailings management strategy envisages NAG tailings being used as cover material for PAG tailings and mine rock to prevent the onset of acid generation during both operations and following closure. PAG material will not be deposited in Cell 1 or Cell 2B.

The TSF rockfill embankments will be developed via the downstream construction method using NAG mine rock. The dams will be raised in stages to provide sufficient storage capacity for tailings and temporary water management. The final elevation of the dams will range from 343 masl (Cell 1) to 380 masl (Cell 2A and 2B). The TSF embankments will be constructed with upstream and downstream slopes of approximately 2H:1V and a minimum crest width of 8 m. The design utilizes site topography to minimize the size of the starter embankments.

The final maximum embankment heights will vary between about 43 m (Cell 1) and 80 m (Cell 2A above grade and foundation widths will be between approximately 180 and 330 m. The embankments will include specific rock fill zones with finer material towards the upstream portion of the embankment and coarser material towards the downstream portion of embankment. The embankment zones will be filter graded such that the embankment will not be susceptible to internal erosion or piping. The downstream rockfill zone will consist of run-of-mine rockfill and will be resistant to downstream erosion.

The dams will include a HDPE liner keyed into bedrock via a concrete plinth (or alternative) to minimize seepage from the facility. Removal of overburden and higher permeability bedrock, placement of slush grout on the prepared bedrock surface and/or injection grouting of deeper permeable bedrock zones will be completed as required by site conditions to further reduce the potential for seepage from the facility. Seepage collection basins will be constructed along the toe of the dams to intercept seepage and pump it back to the facility. Monitoring stations located downstream of the TSF will be used to verify the effectiveness of the collection basins. A typical cross section for the TSF dams is shown in Figure 20.2.

Figure 20.2: Typical Cross Section for TSF Dams – Cell 2 Ultimate



Source: *KP (2022).*

The TSF design will include requirements for instrumentation, monitoring, inspection and routine maintenance to ensure the facility performs as designed. These requirements will be documented in the Operation Maintenance and Surveillance Manual for the facility. A dam breach assessment and analysis of mitigating controls and design parameters has been completed to evaluate the magnitude of impacts of a hypothetical breach of the facility. The analysis includes an assessment of the dam breach characteristics, including breach outflow volumes and the downstream hydrology during sunny day and flood induced conditions.

The TSF will include capacity to manage storm water runoff inflows under normal operating conditions. The Environmental Design Storm ("EDS") consisting of the 1 in 100 year 24-hour precipitation event and 30-day spring snowmelt (408 mm) will be contained within the TSF without uncontrolled discharge to the environment. Emergency overflow spillways have been included in the TSF arrangement to manage storm events greater than the EDS. The TSF spillways will be sized to route the peak flow resulting from a 24-hour Probable Maximum Precipitation ("PMP") event (328 mm), which has been selected as the Inflow Design Flood ("IDF") for the TSF.

Ten seepage collection basins ("SCBs") will be located around the perimeter of the TSF. Near surface seepage and runoff collected in the SCBs will be pumped back to the TSF storage cells. Monitoring wells will be installed to monitor groundwater quality downgradient of the TSF.

Supernatant water in the TSF will be transferred to the water management pond ("WMP") for reuse in the process to reduce accumulation of water in the storage cells. The WMP will provide water to the process plant. Excess water in the WMP will be treated, as required, and then discharged to Hare Lake.

20.3.2 <u>Mine Rock Management</u>

An estimated 326 Mt of mine rock will be generated over the LOM. Mining operations will produce two types of mine rock, referred to as NAG and PAG. NAG mine rock is defined as rock with less than 0.18% sulphur (by weight), which has been predicted to be NAG. PAG mine rock is defined as mine rock with greater than 0.18% sulphur (by weight), which has been predicted to be PAG. NAG mine rock is anticipated to account for approximately 85-90% of the rock while PAG mine rock is anticipated to account for approximately 10-15% of the rock from the open pits.

NAG mine rock will primarily be stored in the MRSA, but also in the North, South and Central Pits. The MRSA is located to the east of the open pits as shown in Figure 20.1. The MRSA will be constructed with an overall slope of approximately 2.2H:1V, with 30 m tall benches with mid slopes at 2H:1V and 10 m wide mid-slope benches. The MRSA slopes will provide long-term stability and allow for progressive reclamation. Preliminary design criteria incorporated into the MRSA included the codes and standards of Ontario Mining Act, Regulation 240/00 (Advanced Exploration, Mine Development and Closure). NAG mine rock will also be used for construction of the TSF, WMP, and SWMP embankments as well as a source of aggregate to build site infrastructure and roads.

During operations, PAG mine rock will either be placed in the TSF during the first six years of operations and progressively covered by Type 1 (NAG) tailings or stored in the South Pit or the Central Pit. Following mine closure, all of the PAG mine rock stored in the TSF will remain below the groundwater table. The PAG mine rock stored in the pits will become submerged as the pits fill with water. In both cases, the storage of PAG mine rock under saturated conditions will effectively prevent the development of acidic drainage in the long-term.

Grade control will be undertaken to identify ore from mine rock in the open pits. Samples will be taken from blast holes and analyzed at the assay laboratory to determine ore and mine rock boundaries within blasted material prior to mining. Samples of mine rock will also be analyzed for total sulphur content to determine if the rock is NAG or PAG.

An estimated 3 Mt of overburden will be generated over the LOM. The overburden will be stored in the overburden stockpile located to the east of the South Pit. The overburden will be used for the progressive reclamation and final closure of the site.

20.3.3 Water Management

A detailed site water balance was developed for the Project using the GoldSim software package. The water balance considers all major components of the site, including the TSF, WMP, SWMP, Open Pits and the MRSA, as well as seasonal discharge requirements to Hare Lake.

The TSF will consist of three storage cells (Cell 1, Cell 2A and Cell 2B) and a separate WMP. The storage cells will provide permanent and secure storage for tailings from the process plant. Supernatant water (i.e., process water and precipitation) that accumulates in TSF storage cells will be reclaimed as the primary source of water for the process plant and routed to the WMP. The WMP will be established to the east of Cell 1 and will serve as the primary contact water pond for the site as well as the secondary water source for the process plant. The WMP will be constructed during the construction phase of the Project along with the SWMP and will initially be utilized as a storage pond for construction water management.

Runoff from the process plant area, truck shop, warehouse area, laydown area, fueling station and the aggregate plant areas will be collected in the SWMP. Water collected in the SWMP will be routed to the WMP or directly to the WTP via water transfer pipelines. The SMWP will also provide tertiary containment for the process plant area and associated pipelines (i.e., tailings slurry and reclaim water pipelines) and fuel farm, ensuring that Sub-watershed 101 and the Pic River will be protected in the case of an unplanned event.

Surface water runoff and groundwater inflow reporting to the open pits will be transferred to Collection Pond 1 (“CP1”) located adjacent to the ROM stockpile. Water collected in CP1 will be routed to the WMP via water transfer pipelines. Water levels in waterbody L-8 located to the northeast of the open pits will also be managed by pumping to CP1. Contact water from CP1 may be used for dust control on the mine haul roads.

Contact water from the MRSA located along the east side of the open pits will be collected in catch basins established in Sub-watershed 102 (Stream 2 Catch Basin) and Sub-watershed 103 (Stream 3 Catch Basin). The catch basins will be constructed prior to initial development of the open pits and the MRSA. Water collected in the catch basins will be collected and pumped to the SWMP and WMP via the MRSA Catch Basin pipelines. The collection system will be sized to manage the EDS, which is based on a 1 in 100-year rainfall event. In the event that the EDS is exceeded, water will be routed from the MRSA catch basins via the catch basin overflow spillways to the Pic River. The overflow spillways have been sized to convey the 1 in 1000-year rainfall event.

Under routine operating conditions contact water from the Project site will be transferred to the WMP. Water from the TSF will be reclaimed to the process plant on a continuous basis with make-up water drawn from the WMP. The recycling of water from the TSF and WMP to the process plant will be maximized. This arrangement was designed to limit the potential requirement for fresh water from other sources and keep TSF process water separate from the contact water from other sources. Overflow from the WMP can be managed within Cell 1 of the TSF to provide additional operational flexibility, as required. Excess water will be transferred from the WMP to the WTP, treated as required, and discharged to Hare Lake.

Water treatment will be undertaken to ensure applicable receiving water quality criteria are met in Hare Lake. Under average conditions, discharge rates to Hare Lake are anticipated to range between approximately 0.9 M m^3 to 2.4 M m^3 per year depending on the footprint of the site.

A network of surface and groundwater quality monitoring stations will be established prior to the start of construction to verify the effectiveness of the site water management system.

20.4 Community Relations

20.4.1 Indigenous Groups

Sixteen (16) Indigenous groups were identified by the Crown (Canada and Ontario) as having a potential interest in the Project. Table 20.4 provides a list of the Indigenous groups including their approximate distance from, and their stated interest, in the Project.

Table 20.4: Indigenous Groups

Indigenous Groups	Approximate Distance from Project (km)	Stated Interest
Animbiigoo Zaagi'igan Anishinaabek (Lake Nipigon)	150	Not Interested
Biinjitiwaabik Zaaging Anishinaabek (Rocky Bay FN)	150	Not Interested
Bingwi Neyaashi Anishinaabek (Sandpoint FN)	150	Not Interested
Fort William First Nation	225	Not Interested
Long Lake #58 First Nation	110	Not Interested
Kiashke Zaaging Anishinaabek (Gull Bay FN)	230	Not Interested
Red Rock Band	150	Not Interested
Whitesand First Nation	260	Not Interested
Pic Mobert First Nation	50	Not Interested
Biigtigong Nishnaabeg	20	Interested
Pays Plat First Nation	90	Interested
Mitchipicoten First Nation	145	Interested
Ginoogaming First Nation	100	Interested
Superior North Shore Métis - MNO	-	Interested
Jackfish Métis – Ontario Coalition of Indigenous Peoples (OCIP)	60	Interested
Red Sky Métis Independent Nation - RSMIN	300	Interested

Of the 16 Indigenous groups shown in Table 20.4, seven groups indicated that they were interested in participating in consultation processes related to the Project. As shown in Table 20.4, the seven groups are Biigtigong Nishnaabeg (BN), Pays Plat First Nation, Mitchipicoten First Nation, Ginoogaming First Nation, Superior North Shore Métis – MNO, Jackfish Métis – Ontario Coalition of Indigenous Peoples and Red Sky Métis Independent Nation. Accordingly, meaningful and informed engagement and consultation has been undertaken with these groups as part of the development of the Project. In conjunction with the consultation process opportunities for Indigenous groups to benefit from the Project have been identified and the Project design modified to ensure impacts to the environment and Indigenous rights are minimized.

The Project is situated within the geographic territory of the Robinson Superior Treaty area. It is also within lands claimed by BN as its asserted exclusive Aboriginal Title. In 2003, BN brought legal action (known as the Michano litigation) against Canada and Ontario seeking a declaration of unextinguished exclusive Aboriginal Title to an area north of Lake Superior, claiming they did not enter into the Robinson Superior Treaty in 1850 and did not adhere to the Robinson Superior Treaty subsequent to 1850. In 2016, the three parties began exploratory discussions to try to find a resolution outside of the court process. As a result of these discussions, the parties entered into formal negotiations in May 2019 and the Michano litigation was put into abeyance (on hold) in December 2019. At this time, negotiations between BN, Ontario and Canada are ongoing. In November 2022, a CBA was completed between BN and Gen Mining for the development of the Project.

20.4.2 Town of Marathon

The Town of Marathon is centrally located on TransCanada Highway (Hwy 17) between Thunder Bay and Sault Ste. Marie on the North Shore of Lake Superior in Northwestern Ontario. The Town is the closest population centre to the Project site, located 10 km south of the site. The current population of Marathon is approximately 3,200. Marathon is surrounded by the Towns of Terrace Bay and Schreiber to the west, the Town of Manitouwadge to the north northwest, the Town of White River to the east, and the First Nations groups of Biigtigong Nishnaabeg, Pic Mobert, and Pays Plat.

Historically, the region was supported economically by the forestry and pulp and paper sectors, as well as the mining industry. The significant downturn in forestry and pulp and paper in the last number of years has negatively impacted local and regional groups, including the Town of Marathon, whose pulp mill closed in 2009. Barrick Gold’s Hemlo Gold Camp, which includes one active mine, is the primary natural resource-based employer in the area. The Project plans to continue to work in partnership with the Town of Marathon to develop the Project. It is anticipated that the Project will provide a significant positive economic influence on the Town.

The Project site lies partially within the municipal boundaries of the Town of Marathon, as well as partially within the unorganized townships of Pic, O’Neil and McCoy. The primary zoning designation within the Project site is “rural”. Changes to the Town of Marathon Official Plan and Zoning By-law as it pertains to land-use zoning will be required so as to permit the development of the mine.

It is the intention of Project to work closely with the Town of Marathon to ensure that the economic benefits from the Project are realized and to determine how best to address issues such as increased demand for housing and community and healthcare services.

20.5 Closure and Reclamation Planning

The progressive reclamation and closure of the Project will be carried out in accordance with O. Reg. 240/00 and as described in a closure plan that will be accepted and filed by the Ministry of Mines prior to the start of construction. A closure cost of approximately $66.5 million (excluding the carrying cost of the closure bond) was estimated for the Project in 2022. This estimate is subject to change based on the results of the review conducted by the Ministry of Mines and Ministry of Northern Development prior to filing the closure plan.

The Company will be responsible for providing financial assurance (expected to be in the form of a closure bond or similar approved financially acceptable mechanism) to the Province of Ontario as specified in the approved closure plan. Financial assurance will be provided in phases consistent with the timing of the start of various Project activities as outlined in the closure plan.

Progressive reclamation will be undertaken during operations (as described in the closure plan) to achieve the end land use plan as soon as possible. Active closure is expected to be completed within five years following the completion of operations with monitoring of the site continuing for an estimated additional 45 years.

21. CAPITAL AND OPERATING COST

21.1 Summary of Operating and Capital Costs

A summary of the project capital costs ("CAPEX") and operating costs ("OPEX") are presented in Table 21.1 to Table 21.3.

Table 21.1: High-Level Capital Costs

Capital Costs	Value ($M)
Initial Capital	1,112
Pre-Production Revenue	(156)
Leased Equipment [1]	(58)
Initial Capital (Adjusted)	**898**
LOM Sustaining Capital	424
Total Capital Costs (Adjusted)	**1,322**
Closure & Reclamation	72

***Note**:*
Lease drawdowns net of lease payment during the construction and pre-production period

Table 21.2: Project Area Capital Cost

Capital Costs	Initial ($M)	Sustaining ($M)	Total ($M)
Mining	116.9	129.6	246.5
Process Plant (Excluding Site Works)	345.2	3.1	348.3
Infrastructure	72.1	94.3	166.2
Tailings Storage and Water Management	95.1	197.5	293.1
General and Owner's Cost	31.1		31.1
Construction Indirects	196.6		196.6
Preproduction, Startup, Commissioning	158.5		158.5
Contingency[1]	96.5		96.5
Subtotal	**1,112.1**	**424.4**	**1,536.5**
Equipment Financing	(58.4)		(58.4)
Pre-Commercial Production Revenue, net of related offsite costs (transport, smelter and royalties)	(155.8)		(155.8)
Total Capital Costs (Adjusted)	**897.9**	**424.4**	**1,322.3**
Closure & Reclamation[2]		72.4	72.4

***Notes**:*
[1] Contingency included at project sub-category basis and totals approximately 9.5%.
[2] Closure cost estimate is $66.5 M and additional cost included for carrying cost of closure bond.
Sums may not total due to rounding.

Table 21.3: Operating Costs

Operating Costs[1]	Units	
Mining[2]	$/t mined	**3.25**
Processing	$/t milled	**8.70**
General & Administration	$/t milled	**2.67**
Transport & Refining Charges	$/t milled	**4.13**
Royalties	$/t milled	**0.09**
Total Operating Costs	$/t milled	**27.04**
LOM Average Operating Cost	US$/oz PdEq	**709**
LOM Average AISC[2]	US$/oz PdEq	**813**

Note:

[1] Mining cost also noted as $9.23/tonne milled.

[2] AISC ("All-in Sustaining Cost") excludes the impact of the Wheaton PMPA

21.2 Basis of Estimate

21.2.1 Project Execution Strategy

The Project execution strategy is to employ an integrated Engineering Procurement and Construction Management and Commissioning ("EPCM"). Engineering and procurement will be performed by various parties given the specific area and scope. This will result in a Project management team with both the employees of the Company and the consulting firms throughout the execution and commissioning phases with experience in implementing similar sized projects.

21.2.2 Responsibility Matrix

Gen Mining is responsible for the overall coordination, compilation, documentation, and quality control of the CAPEX / OPEX / sustaining estimate as well as the financial model. Estimating responsibilities were assigned at the Work Breakdown Structure ("WBS") level to various contributors. Responsibilities for providing input were as follows:

- LQ – Assisted Gen Mining's in the overall coordination and compilation of the CAPEX / sustaining estimate; obtain budgetary contractor pricing for earthworks; joint build-up of indirect costs with Wood and JDS
- GMS – Responsible for the Mine Design, Open Pit Design Optimization, Mineral Reserves Estimate, Production Sequencing, LOM and Equipment Selection and Haulage Studies

- Wood – Quantities for the process plant and select infrastructure (except as noted in the subsequent sections): budgetary / firm quotations for major equipment, budgetary quotations / in-house historical data for minor equipment; build-up of installation costs for concrete, steel, mechanical and piping ("SMP") and Electrical and Instrumentation ("E&I"); budgetary quotations for material supply and fabrication of process piping, electrical, instrumentation and controls
- KP – Quantities and staging for the TSF. Pricing of earthworks was supplied by Gen Mining, coordinated by LQ. Water management requirements for the Project were developed by KP based on the site wide water balance
- JDS – Quantities for select infrastructure: Quantities for site bulk earthworks; estimate build-up for the mine services area and site water management pipelines; allowances for minor site buildings (e.g. admin building and guard house); allowance for the off-site laboratory; allowance for the off-site portion of the 25kV distribution line; indirect costs input for all other costs not estimated by Wood

21.2.3 CAPEX Base Date

The estimate's base date is Q4 2022.

The Project's CAPEX estimate start date is January 1, 2023. Project completion is achieved at the commercial production milestone, which is defined at the end of plant commissioning. During the commissioning period and prior to achieving commercial production, the operating costs and associated revenues will be capitalized . All non-operating costs required to complete Project handover and close out also form part of the CAPEX.

21.2.4 Estimate Accuracy

The accuracy of the capital cost estimate for the Project meets AACE Feasibility Study Class 3 guidelines and is within -15% to +20% (note: AACE classification: Low: -10% to -20% and High: +10% to +30%) of final project costs with contingency.

21.2.5 Currency and Commodity Rates

All costs are expressed in Q4 2022 Canadian dollars (C$). Currency exchange rate used for this estimate is established based on average Bank of Canada FX rate:

- C$ to US$ = 1.30
- Euro to C$ = 1.30

The estimate contains pricing information in native currencies and is converted into C$.

21.3 Estimate Methodology

21.3.1 Overview

Working with the listed consultants defined above, the development of the capital and operating costs followed a standard methodology to develop the estimate, as follows:

- Confirm the scope of work
- Define the estimate base date
- Define the estimate reporting currency
- Define the estimate by WBS
- Collect various data sets, including:
    - Discipline MTOs
    - Pricing from budgetary / firm price bids, budgetary / firm RFP, quotes, databases, and benchmarking
    - Direct labour wages
- Develop the labour rates
- Determine the productivity factors
- Determine the installed equipment and material costs
- Determine the indirect costs
- Determine foreign exchange content
- Determine the estimate contingency value through a quantitative risk assessment
- Complete internal reviews

Source data that was used in the development of the estimate included:

- Scopes of work
- Equipment lists
- Material Take-Off (“MTOs”)
- Design criteria
- Layouts and general arrangements
- Process flow diagrams
- Engineering calculations
- Geotechnical investigation
- Project execution plan
- Equipment pricing and budget quotes

- Material and labour rates, budgetary pricing
- Construction installation rates
- Mine plan
- Plant ramp-up plan
- Project schedule

The direct cost portion of the CAPEX estimate was reviewed for completeness and consistency against the Project description, and indirect costs were added to the direct cost estimates to complete the estimate.

21.3.2 Work Breakdown Structure

The CAPEX estimate has been structured on the WBS and the cost coding structure defined for the Project. The WBS was developed during the FS and has been updated in the detailed engineering phase as required. The first three levels of the WBS are shown in Table 21.4.

Table 21.4: Work Breakdown Structure

WBS L1	WBS L2	WBS L3	Item
		0010	All Site General
1000			**Infrastructure**
	1100		**General Site Preparation**
		1110	General Earthwork
		1120	Site Roads
		1130	Camp 19 Road Upgrade
		1140	Pad Construction
		1150	Material Sourcing
	1200		**Mine Infrastructure**
		1220	Mine Service Building
		1240	Emulsion/Explosive Magazine
	1300		**Support Infrastructure**
		1310	Administrative Building
		1320	Site Guard House
		1350	Laydown
	1400		**Permanent Camp**
	1500		**Laboratory**
	1700		**Fuel/Oil Systems Storage**
	1800		**Transload Facility**

WBS L1	WBS L2	WBS L3	Item
	1900		**Offsite Facilities**
2000			**Power & Electrical**
		2010	Generals And Single Lines
	2100		**Main Power Source**
		2120	Transmission Line
		2130	Main Substation
	2200		**Secondary Power Source**
		2210	Secondary Power Line
		2220	Onsite Generator
	2500		**Mine Electrical Room**
	2700		**MV Distribution O/H Line**
	2800		**Automation Network**
	2900		**IT Network & Fire Detection**
	2950		**Telecom**
3000			**Water**
	3100		**Fresh Water / Wells**
	3200		**Surface Water Management**
	3300		**Potable / Domestic Water**
	3400		**Sewage Water**
	3500		**Fire Water**
	3600		**Effluent Water Management**
	3700		**Tailings Storage Facility (“TSF”)**
		3710	Water Management Pond (“WMP”) East Embankment
		3720	WMP West Embankment
		3730	Cell 1
		3740	Cell 2A
		3750	Storm Water Management Pond
		3760	Instrumentation
		3780	Construction Water Management
	3800		**Mine Waste Rock Water Pond**
4000			**Surface Operations**
	4100		**Surface Operations Equipment**
		4140	Support Equipment
		4160	Operation And Maintenance
	4300		**Concrete Contractor Indirects**

WBS L1	WBS L2	WBS L3	Item
	4800		**Aggregate Plant**
5000			**Mining**
	5100		**Mine Development**
	5400		**Mine Infrastructure**
		5410	Haul Road
		5420	Cofferdams
		5430	L5 Berm
		5440	L8 Berm
	5500		**Mine Equipment**
		5510	Primary Mining Equipment
		5530	Support Equipment
		5540	Other Equipment
	5600		**Mine Dewatering**
6000			**Process Plant General**
	6050		**Process Industrial Facilities Site Development**
		6051	Earthworks
		6052	Site Development
	6100		**Crushing**
		6110	Primary Crushing
		6115	Overland Conveyor and Transfer House
		6120	Ore Stockpile and Reclaim
		6190	Crushing General
	6200		**Grinding**
		6210	Grinding
		6290	Grinding General
	6300		**Flotation & Regrind**
		6310	Rougher And Cleaner Flotation
		6320	Regrind
		6330	Flotation OSA
		6390	Flotation General
	6400		**PGM Circuit**
		6410	PGM Scavenger
		6420	PGM Regrind
		6490	PGM General
	6500		**Concentrate Dewatering and Handling**

WBS L1	WBS L2	WBS L3	Item
		6510	Concentrate Thickening
		6520	Concentrate Filtration
		6530	Concentrate Storage and Loadout
		6590	Concentrate General
	6600		**Tailings Thickening and Pumping**
		6610	Tailings Thickening and Pumping
		6620	Reclaim System (Barge, Barge Pumps, Pipelines)
		6630	Pumpback System
		6690	Tailings General
	6700		**Reagents**
		6710	Lime
		6720	Collector
		6730	Promoter
		6740	Frother
		6750	Test Reagent
		6790	Reagent General
	6800		**Process Plant Utilities and Services**
		6810	Potable water distribution
		6820	Fire protection
		6830	Process water distribution
		6840	Fresh water distribution
		6850	Process Control, Security and CCTV systems
		6860	Plant and Instrument Air
		6870	Electrical distribution
		6890	Process Utilities and Services General
	6900		**Process Facilities**
		6910	Crusher Shop and Control Room
		6920	Process Plant Building
		6930	Process Offices and Control Room
		6940	Process Shops
		6950	Process Mobile Equipment
		6960	Reagent Storage Facilities
		6990	Process Facilities General
7000			**Construction Indirect**
	7100		**Engineering, CM, PM**

WBS L1	WBS L2	WBS L3	Item
	7200		**Construction Offices, Facilities & Services**
	7300		**Contractor Mob/Demob and Indirects**
	7400		**Construction Camp Facilities & Operation**
	7500		**Freight & Logistics**
8000			**Owner's Cost**
	8100		**Departments**
	8200		**Insurance**
	8300		**Operations Accommodations**
	8400		**Others**
	8500		**Recruiting Expenses**
9000			**Pre-Prod, Start-Up, Commissioning**
	9100		**Mine Preprod /Commissioning**
	9200		**Mining Haul Roads**
	9400		**Spares**
	9500		**Process Plant Preprod / Commissioning**
		9510	Process Plant Management & Training
		9520	Process Plant Pre-Prod
		9530	Process Plant Commissioning
		9540	Vendor Reps
	9600		**Operational Readiness Support**
		9610	Spare Parts Capital
		9620	Spare Parts Commissioning
		9630	Spare Parts Pre-Prod
		9640	Spare Parts Mining
		9650	First Fill (reagents, grease & oil)
		9660	Consumables Pre-Prod
	9700		**Pre-production Revenue**
	9800		**Sunk Costs**
	9900		**Contingency**
		9910	General Capex Contingency
		9920	Design Growth / Develop Contingency
		9930	Schedule Contingency
		9940	Mining Eqpt Contingency
		9950	Owner's Cost 900 Series Contingency

21.3.3 Estimate Sorting Codes

The estimate contains information on various fields as shown in Table 21.5, the estimate can be sorted on any of those fields.

Table 21.5: Estimate Sorting Codes

Name	Description
Level 1	Project WBS Level 1
Level 2	Project WBS Level 2
Level 3	Project WBS Level 3
Level 4	Project WBS Level 4
Discipline	Estimate discipline codes
MTO #	MTO Number (as provided by Engineering)
MTO Rev #	MTO Revision Number (as provided by Engineering)
Tag #	Equipment Tag Number
Material Price Source	Material Price Source (e.g., Spent, Purchase Order/Contract, Firm Price, Budget Quote, In-House Data, Allowance)
Material Type	Material Type (e.g., Bulk Materials or Plant Equipment)
Quantity Source	Quantity Development Method (e.g., Design Drawing, Concept Drawing, Factored, Allowance)
Vendor	Vendor's Name
Consultant	Consultant's Name

21.4 Material Take-Off and Estimate Quantities

MTOs were provided in a structured and traceable manner in appropriate formats. The preparation and review of the MTOs followed standard engineering practices.

MTOs are based on neat quantities, with applied factors for waste and details. No design growth factor was applied on these quantities. There is a specific provision for design growth in the contingency provision. Before an MTO was issued to estimating, a review of the area being issued was undertaken to ensure all scope is captured.

- Wood – Developed MTOs for the process plant and select infrastructure (except as noted in the subsequent sections)
- KP – Developed MTOs for the TSF

- JDS – Jointly developed MTOs for site bulk earthworks with Gen Mining (i.e. access roads, haul roads, etc.), and developed MTOs for the mine services area and site water management pipelines

The final consolidated MTOs were reviewed by experienced construction personnel and validated against previous project experience. Based on the results of this review, certain additional allowances were included for electrical, instrumentation, piping and HVAC.

21.5 Basis of Estimate Processing Plant (by Wood)

21.5.1 Commodity Rates

The labour rates used are based on 2022 rates from the Oakbridges Industrial Relations Strategists ("Oakbridges") Labour Study provided by Gen Mining.

Equipment rates were updated based on firm price bids received by Q4 2022. Minor equipment prices were re-evaluated and adjusted against current project benchmarks.

21.5.2 Scope of the Estimate

21.5.2.1 Scope of Responsibility

Wood is responsible for the EP portion of the Process Plant and Infrastructure, however the estimate produced for these facilities will include all the direct costs, including supply and installation costs.

Area of responsibility by Wood includes:

- Process plant (crusher through concentrate, tailings, reagents)
- Outdoor process tanks
- Tailings pumping system and tailings/reclaim pipelines (tailings water transfer barge from Cell 1 to Cell 2 is also included as a mechanical supply package)
- Process plant buildings and other process support structures
- Process plant infrastructure, including fire water, potable water, fresh water, sewage
- Electrical infrastructure, including the main substation, 115 kV line, and on-site 25 kV line, distribution and telecommunications

Area of responsibility by Gen Mining and other subconsultants includes:

- Tailings ponds and site ponds
- Mine and haul roads
- Buildings including accommodation, truck shop services, guardhouse, explosives building, fuel storage

21.5.2.2 Estimate Support Documents

The baseline estimate has been developed in accordance with Class 3 Estimate criteria, philosophy, and preliminary project design, and is based on the following documents:

- Project scope of facilities
- Engineering discipline design criteria
- General arrangement drawings (“GA”)
- Process Flow Diagrams (“PFDs”)
- Piping and Instrumentation Diagrams
- Single Line Diagrams
- Mechanical Equipment List (“MEL”)
- Electrical Equipment and Load List
- Site layouts
- MTOs
- Firm and budget quotations from equipment vendors
- Budgetary and firm quotations for pre-engineered buildings/multiplate tunnels
- Budgetary quotes for some of the bulks
- Location specific and regional climatic data
- Project WBS
- In-house data (for the bulks/material, installation and logistics) when firm and budgetary quotes not available

21.5.3 Quantity Development

Quantities are organized by area and discipline codes.

Engineering material take-offs are based on the general arrangement drawings and sketches, P&IDs, 3D model, equipment lists and are “neat” quantities derived from basic/detail design. An allowance for quantities is made where drawing information is not available. Allowances in general, are quantities which are based on judgement made by engineering or estimating which are not supported with engineering data or calculations. Allowances MTO are minimized and are indicated in the estimate as such.

Metric units are used throughout the estimate, with the exception of piping diameter size which are indicated in imperial unit(s).

A MTO development table is shown below and summarized actual material take-off sources produced for this estimate.

Table 21.6: Material Take-Off Development

Discipline	2D layouts / 3D model	Equipment list /P&IDs	Sketches	Factored	Allowed	FS
Civil	X					
Civil Piping	X		X			
Concrete	X		X		X	
Steel	X			X		
Architectural	X		X	X		
Mechanical		X				
Bulk Mechanical	X	X			X	
Piping	X	X		X	X	
Electrical Equipment	X	X	X		X	
Electrical Bulks	X	X	X		X	
Instrumentation	X	X		X	X	
Telecommunication	X	X	X		X	

2D Layout/ 3D Model or Design Drawings	Quantities taken off from design drawings, 3D Models, and other engineered calculations specific for the project.
Equipment list /P&IDs	Mechanical and Electrical Equipment lists are basis for equipment definition/ size and counts. P&IDs are developed for all areas, Piping and Instrumentation MTOs are based on these documents.
Sketches	Preliminary sketches were created for the purpose of the estimate.
Factored	Calculated from similar sized projects and factored to adjust for plant size, capacity and site-specific requirements
Allowed	Quantities estimated based on engineering or estimating judgement and is unsupported with engineering data or calculations

21.5.3.1 Civil

Civil quantities have been developed by the Civil Discipline group using site layouts, and 3D models/sketches.

All earthworks' quantities are taken off neat in place without any allowance for swelling, compaction and wastage quantities captured through a design allowance applied on the neat quantities. Any allowance for swelling, compaction and wastage is captured in the Gen Mining provided earthworks unit rates.

The civil discipline group has provided mass excavation and backfill for the site facilities.

Earthwork quantities for buried electrical cables/duct banks have been quantified.

Aggregate supply costs are based on material being sourced on site and provided by Gen Mining at a stockpile. Base aggregate costs include crushing, screening, washing, loading and hauling to on-site stockpile(s).

Site Water Management Piping and Pumping of the Reclaim and Tailings water line have been estimated by Wood based on the latest site plan.

The Site Water Management Earth Structures, TSF and WMP earthwork quantities, overall site earthworks for roads, pads and laydowns have not been included in the Wood estimate and are included in the overall Gen Mining estimate. The only sitework included in the Wood estimate is for firewater, sanitary and potable water piping.

21.5.3.2 Concrete

Concrete quantities have been developed by the Structural Discipline group using General Arrangement drawings and sketches and 3D Model (Table 21.7).

Allowances for concrete finishing, sealing and concrete additives are included in the overall unit rate per m^3.

Quantities are calculated as “neat” without allowances for wastage, over-pour, and other variables. Allowances for these items are applied as a factor of the neat quantities.

Detailed excavation and backfill to support concrete installation are quantified on the same Structural MTO’s as concrete.

Table 21.7: Concrete Quantities

Commodity	Unit	Total Quantity (Including design growth)
Structural Excavation	m^3	72,530
Structural Backfill	m^3	45,244
Structural Concrete	m^3	22,799
Lean Concrete*	m^3	513
Anchor Rods	kg	39,031
Embedded Steel	kg	42,459

Note: the 513 m^3 for lean concrete is based on the minimum requirement for neat measurement of shown on the drawings. Allowances for over-break/additional lean concrete are not included in the design allowances.

21.5.3.3 Structural Steel

Structural Steel quantities are based on 3D model, general arrangement drawings and sketches, budgetary quotes for pre-engineered buildings and multiplate tunnels (Table 21.8). Steel quantities have been broken down per size/category. Quantities are calculated as “neat” without allowances.

Structural steel related to pre-assembled modules might be included with equipment cost if they are integral part of the equipment module.

Table 21.8: Steel Quantities

Commodity	Unit	Total Quantity (Including design growth, not including connections & base plates)
Light Steel [0 to 30kg/m]	t	415
Medium Steel [30 to 60kg/m]	t	361
Heavy Steel [60 to 90kg/m]	t	389
XH Heavy Steel [90 to 180kg/m]	t	639
XXH Heavy Steel [180 to 360kg/m]	t	387

21.5.3.4 Architectural

Architectural cost related to buildings are estimated as part of building cost for pre-engineered buildings and added on to in-house designed stick-built buildings. MTOs are based on the building size and spec, facility and functionality and insulation requirements. Allowances for building furnishings (fit-out) costs have been made, as appropriate, for each building and its intended usage. Door and cladding have been quantified.

Minor items, such as furniture and fit outs are not included in the MTO and are carried as allowance.

The pre-engineered steel buildings are:

- Grinding Building: (69.2 m L x 47 m W x 31 m H) Area: 3,252 m^2, Volume: 111,033 m^3
- Flotation Building: (77.2 m L x 41.2 m W x 31.6 m H) Area: 3,181 m^2, Volume: 100,508 m^3
- Concentrate Storage and Loadout/Reagent Building:
    - (41.6 m L x 29 m W x 14.3 m H) + (41.6 m L x 14.7 m W x 28.7 m H)
    - Area: 1,818 m^2, Volume: 34,549 m^3

21.5.3.5 Mechanical Equipment and Systems

The mechanical equipment has been identified on the equipment lit by tag number, complete with descriptions, quantities, size and capacity. Mechanical Equipment List reflects process configuration defined in PFDs and P&IDs.

21.5.3.6 Mechanical Bulks & Tanks

Mechanical Equipment List have been used as the basis for mechanical bulks and tanks sizes (Table 21.9). Platework weights related to these components have quantified on the Mechanical Equipment List.

Minor items, such as liners are not included in the MTO and are carried as allowance.

Table 21.9: Platework Quantities

Commodity	Unit	Total Quantity (Including design growth)
Chute (Steel Lined)	t	54
Hopper	t	7
Launder - Steel Plate	t	12
Pumpbox	t	36
Tank (Field Erected)	t	235
Tank (Shop Fabricated)	t	26
Bins - Carbon Steel	t	4
Chute (Rubber Lined)	t	8
Chute	t	4
Duct (Galvanized Steel)	t	60
Insulation (mineral wool with a jacket 50mm)	m2	546

21.5.3.7 Piping

Piping quantities have been developed based on P&ID and 3D models and layouts (Table 21.10). The piping quantified includes both small and large bore piping. Based on the level of design approximately 10% of small-bore piping is included in the MTO. Small bore piping contains lines between ½” to 3”.

MTOs include piping size, pipe material, valve type, fittings. Minor items, such as pipe supports, welds and flanges are not included in the MTO and are carried as allowance.

Infrastructure underground piping for fire water and underground services is covered under civil. Site water management piping is carried over from FS. Fire water piping was estimated based on the layout.

Table 21.10: Piping Quantities

Piping Size	Unit	Approximate Length
Carbon Steel		
4"	m	974
6"	m	1418
8"	m	568
10"	m	359
12"	m	101
14"	m	28
16"	m	27
18"	m	27
20"	m	66
24"	m	95
26"	m	22
28"	m	27
32"	m	5.5
36”	m	65
42”	m	14
HDPE		
16"	m	2162
18"	m	5346
24"	m	4490

Note: the lengths listed in the table are neat quantities from the MTO. Not in the above quantities included above are design allowances.
Small-bore pipe (up to 3”) was based on a combination of MTO’s and factoring the large bore pipe quantities.

21.5.3.8 Electrical

Cabling and cable trays are based on quantities prepared from process plant layout drawings (Table 21.11). Tray sizing are preliminary, based on the typical concentrator plant of comparable configuration. Cable trenching and buried PVC are allowed for where required. Cables and trays for electrical and instrumentation/controls are included as electrical MTOs.

Grounding is based on MTO quantities developed from the mechanical or civil layouts.

The electrical MTOs for Infrastructure are based on the current design. The 115 kV transmission line and 25 kV distribution lines are based on current site layout. 25 kV line to the mine facilities and the waste rock discharge pond on the east side is not included and is estimated by Gen Mining in the Infrastructure Scope.

Heat tracing, small receptacles, terminations, and other minor items are not included in the MTO and are carried as allowances.

Table 21.11: Electrical Cable Quantities

Cable	Unit	Approximate Length
1kV 1c-4/0AWG	m	524
1kV 2c#12AWG	m	7,880
1kV 2c#16AWG	m	1,612
1kV 3c#10AWG	m	4,549
1kV 3c#12AWG	m	59,969
1kV 3c#14AWG	m	89
1kV 3c#2AWG	m	3,281
1kV 3c#4AWG	m	11,023
1kV 3c#6AWG	m	3,676
1kV 3c#8AWG	m	4,467
1kV 3c-1/0AWG	m	6,536
1kV 3c-2/0AWG	m	231
1kV 3c-250kcmil	m	1,716
1kV 3c-350kcmil	m	2,498
1kV 3c-4/0AWG	m	5,242
1kV 3c-500kcmil	m	4,564
25kV 3c-2/0AWG	m	1,658
25kV 3c-350kcmil	m	502
300V 12Tr#16AWG	m	3,828
300V 1Pr#16AWG	m	17,667
300V 1Tr#16AWG	m	512
5kV 3c#2AWG	m	230
5kV 3c-2/0AWG	m	632
5kV 3c-250kcmil	m	733
5kV 3c-350kcmil	m	464
5kV 3c-4/0AWG	m	315
5kV 3c-500kcmil	m	4,222
600V 10c#14AWG	m	38,555
600V 3c#14AWG	m	425
600V 5c#14AWG	m	130

21.5.3.9 Instrumentation and Control System

The field instrumentation estimate is a combination of MTOs for areas where P&IDs are developed. Instrument supports, cabling and fittings are estimated based on field instrument counts. Minor items, such as junction boxes, instrumentation cabling, and supports are not included in the MTO and are carried as allowance.

A quote was received for the Process Plant control system.

21.5.3.10 Telecommunications

Aerial Fiber Optic Cables

- Fiber optic cable lengths estimated based on preliminary site plan and 25 kV overhead line routing. Cables lengths scaled up to include cable routing, service loops, and sparing
- Two fiber line cable management are allocated every 1 kilometer of cable length and on each end of cable line
- Fiber aerial splice enclosures, and cable management allocated for each drop location of cable line
- Splice enclosure interface to the infrastructure buildings, Admin, Truck Shop, Guardhouse, Fuel Yard, Sewage Treatment Plant, Water Treatment Plant, Primary Crusher, Main Substation, Process Plant Facilities and the water management sites, are included
- Final fiber cable run from splice enclosure into building and terminations is excluded

21.5.4 Bulk and Miscellaneous Unit Pricing

21.5.4.1 Civil

Aggregate supply costs will be based on material being sourced on site and provided by Gen Mining at a stockpile. Currently, the estimate uses Gen Mining supplied unit costs for Aggregate supply costs.

21.5.4.2 Concrete

Concrete supply rates are based on budget quotes received from a Contractor (Table 21.12). This data is from a Q2 2022 project located in Northern Ontario. The below table summarizes the supply rate of the major concrete types. Installation hours are based on recent contractor budget quotes received in Q2 2022 for a Northern Ontario project.

The current concrete supply strategy is to receive ready-mix concrete from a company in the town of Marathon. On-site batch plant costs are not included in the concrete supply rate.

Table 21.12: Concrete All-in-Rate

Description	Unit	All-in-Rate ($)
Concrete Placing - Lean Mix	m^3	779
Concrete Placing - Elevated Slab	m^3	4,290
Concrete Placing - Walls - >18" thick	m^3	2,563
Concrete Placing - Mat Foundations	m^3	1,912
Anchor Rods	kg	50

21.5.4.3 Structural Steel

Structural steel includes material supply, shop pre-fabrication, any paint, primer, and multi-layered coatings. Steel unit rates is based on a budgetary quote, received from an Asian steel supplier (Table 21.13). A budgetary bid was received for the pre-engineered buildings and firm bids were received for the multi plate tunnels. Installation hours are based on an informal budgetary quote received for the Project.

Table 21.13: Steel Supply Rate

Description (excluding delivery)	Unit	Rate ($)	Comment
Light Steel [0 to 30kg/m]	t	3,156	Supply only
Medium Steel [30 to 60kg/m]	t	3,074	Supply only
Heavy Steel [60 to 90kg/m]	t	2,987	Supply only
XH Heavy Steel [90 to 180kg/m]	t	2,987	Supply only
XXH Heavy Steel [180 to 360kg/m]	t	2,912	Supply only

21.5.4.4 Architectural

Table 21.14 below lists all the buildings and indicates the total cost. Supply and Install Bids were received for the Grinding and Flotation pre-engineered buildings. Costs for the Concentrate Storage/Loadout building were factored based on the m^2 bids for the Flotation building.

Table 21.14: Pre-Engineered Building All-in-Rate

Discipline	All-in-Rate ($)
Grinding Building	12,287,510
Flotation Building	9,396,891
Concentrate Storage / Loadout Building	3,134,301

21.5.4.5 Mechanical Equipment

Mechanical equipment list and PFDs is the basis for equipment counts/types/sizes. Firm and budget quotes are used when available. When a firm cost is not available, an in-house reference cost has been used, when reflective of current market conditions. Table 21.15 summarizes major equipment packages and their pricing source and package cost, excluding freight and spares. Installation hours are based on database hours, estimated based on weight and estimated based on crew sizes and number of days.

Table 21.15: Mechanical Package Costs

Package	Package Description	Price ($)	Pricing Source
PJ005	Mill Liner Handlers	2,696,538	Firm Price - Recommended Bidder
PJ010	Flotation Circuit – Roughers	8,115,688	Firm Price - Preferred Bidder
PJ011	Flotation Circuit – Cleaners	12,610,831	Firm Price - Preferred Bidder
PJ015	High-Rate Thickeners	3,538,572	Firm Price - Preferred Bidder
PJ020	Vibrating Screens	1,649,479	Firm Price - Preferred Bidder
PJ025	Hydrocyclones	668,768	Firm Price - Preferred Bidder
PJ031	Ball Mill and SAG Mill (Hycroft)	16,040,000	Gen Mining Firm Price (committed)
PJ031	Ball Mill and SAG Mill (Refurb & Wrap-around)	16,159,100	Budgetary
PJ035	Compressors	1,099,707	Bids received & under evaluation
PJ040	Pressure Filters	3,090,912	Firm Price - Preferred Bidder
PJ046	Agitators	324,907	Firm Price - Selected Bidder
PJ050	Slurry Pumps (incl. Heavy Duty)	1,752,163	Firm Price - Recommended Bidder
PJ060	Overhead Cranes	2,022,880	Bids received & under evaluation
PJ065	Flocculant System	364,010	Firm Price - Recommended Bidder
PJ070	Samplers	304,595	Firm Price - Recommended Bidder
PJ071	Sampler Pumps (CO to PJ155)	276,959	Bids received & under evaluation
PJ065	Flocculant System	364,010	Firm Price - Recommended Bidder
PJ080	High Intensity Grinding Mills	12,325,479	Firm Price - Preferred Bidder
PJ085	Platework	1,708,387	Historical
CC18	Field Assembled Tanks Supply	3,129,812	Historical
PJ100	Metering Pumps	171,053	Firm Price - Recommended Bidder
PJ110	Reclaim Barge	2,068,419	Bids received & under evaluation
PJ150	Water Pumps (in Process Plant)	667,475	Firm Price - Recommended Bidder

Package	Package Description	Price ($)	Pricing Source
PJ155	On-Stream Analyzers	822,036	Firm Price - Preferred Bidder
PV005	Apron Feeders	1,290,900	Firm Price - Preferred Bidder
PV006	Belt Feeder FLS	1,711,403	Firm Price - Recommended Bidder
PV010	Primary Crusher (Gyratory)	5,464,382	Preferred Bidder
PV030	Belt Conveyors	15,917,133	Bids received & under evaluation
PV035	Rock Breaker	838,566	Firm Price - Preferred Bidder
	Total Mechanical Equipment (excluding freight & spares)	**117,194,163**	

21.5.4.6 Mechanical Bulks & Infrastructure Tanks

Mechanical bulks and infrastructure tanks material pricing and installation are based on an informal, budgetary quote (Table 21.16).

Table 21.16: Platework Supply Rate

Description	Unit	Supply Rate ($)
Bins - Steel Plate, Chutes, Launders, Pump-boxes, Field Assembled Tanks	t	13,299
Hopper	t	13,407
Tank – Shop fabricated	t	20,000
Chute (Rubber Lined)	t	22,000
Duct (Galvanized Steel)	t	4,770
Insulation (Assumed Natural Rubber, 40-55 Duro Vulcanized)	m^2	228
Cover	t	15,428

21.5.4.7 Piping

Piping bulks material pricing is based of an informal quote for piping lengths and fittings. Table 21.17 is a sampling of major rates based on Q4 2021 escalated to 2022. A budget quote was received for manual valves. Piping that is less than 8” will be field fabricated (except rubber lined) and all piping larger than 8” will arrive spooled. The rubber lined piping quote includes for shop fabrication labour. The shop fabrication labour for carbon steel pipe, fittings and flanges for spooled piping was factored as 30% of the supply cost. Installation and hours are based on contractor quotes received in Q2 2022 for a Northern Ontario project and databased hours for spooled piping.

Heat tracing has been included for above ground and process plant pipelines. The price is based on historical rates, and has been applied to the following items:

- Sanitary HDPE pipeline
- Tailings and reclaim water pipelines

Table 21.17: Rubber-lined Piping Supply Rate

Description	Unit	Supply Rate ($)
A-53 Gr. B STEEL PIPE, RUBBER LINED 8”	m	1,100
A-53 Gr. B STEEL PIPE, RUBBER LINED 10”	m	1,500
A-53 Gr. B STEEL PIPE, RUBBER LINED 12”	m	1,750
A-53 Gr. B STEEL PIPE, RUBBER LINED 14”	m	1,900
A-53 Gr. B STEEL PIPE, RUBBER LINED 16”	m	2,150
A-53 Gr. B STEEL PIPE, RUBBER LINED 18”	m	2,490
A-53 Gr. B STEEL PIPE, RUBBER LINED 20”	m	2,790

21.5.4.8 Electrical

Electrical bulk materials including cabling, cable tray, connectors, and ancillary devices are based on in-house data or informal budget requests issued via email with brief, high-level scope of supply description on preliminary datasheets. Unit rates have also been sourced using recent quotations from major suppliers on a current project in the region, specifically grounding, cables and cable trays (Table 21.18).

Electrical equipment is based on firm and budgetary quotes. Firm bids for the transformers, LV and MV VFD’s, and MV MCC’s have been received. Budgetary quotes for the SAG and Ball Mill VFDs have been received. E-rooms pricing are based on current projects in the region. The main substation and 115 kV line cost has been estimated by Gen Mining. Installation hours are based on database hours.

Table 21.18: Electrical Supply Costs

Package Description	Supply Cost ($)	Basis
Cables	7,388,989	Recent project quotes
Cable Tray	809,654	Recent project quotes
LV and MV VFD	1,380,403	Bids received & under evaluation
MCC’s	5,021,709	MV MCC’s - Bids received & under evaluation. LV MCC’s - Budget Quote

Package Description	Supply Cost ($)	Basis
Distribution Transformers	4,239,963	Bids received & under evaluation
E-houses	2,126,250	Budget Quote
25KV Distribution Line	2,504,880	Estimated (Wood Portion)
115kV Transmission Line	1,953,000	Historical

Category	Cable Description	Supply ($/m)
25 kV 133% XLPE TECK cable	25kV 3c-2/0AWG	178
	25kV 3c-350kcmil	309
5 kV 133% XLPE TECK cable	5kV 3c#2AWG	125
	5kV 3c-2/0AWG	140
	5kV 3c-4/0AWG	165
	5kV 3c-350kcmil	243
	5kV 3c-500kcmil	274
	5kV 3c-250kcmil	243
1 kV XLPE TECK cable	1kV 3c-250kcmil	146
	1kV 3c-4/0AWG	117
	1kV 3c-1/0AWG	66
	1kV 3c#4AWG	31
	1kV 3c#8AWG	16
	1kV 3c#10AWG	12
	1kV 3c#12AWG	8
	1kV 2c#12AWG	7
	1kV 3c-2/0AWG	73
	1kV 3c#6AWG	25
	1kV 3c-500kcmil	209
	1kV 3c-350kcmil	171
	1kV 1c-4/0AWG	25
	1kV 3c#2AWG	49
	1kV 2c#16AWG	4
	1kV 3c#14AWG	6

Category	Cable Description	Supply ($/m)
	1kV 2c#12AWG	7
	1kV 3c-4/0AWG	117
	1kV 3c-250kcmil	146
	1kV 3c-500kcmil	209
600 V rated control TECK cable	600V 10c#14AWG	16
	600V 3c#14AWG	6
	600V 5c#14AWG	10
	600V 10c#14AWG	16
300V	300V 1Pr#16AWG	4
	300V 1Tr#16AWG	6
	300V 12Tr#16AWG	32
	300V 12Tr#16AWG	32
	300V 1Pr#16AWG	4

21.5.4.9 Instrumentation

Instrumentation and process controls pricing for bulk materials and devices was based on an quote received for instrumentation and from a Q2 2022 project located in Northern Ontario. The remaining instrumentation costs are based on other recent historical projects.

Cost of instruments supplied with mechanical package will be included as part of mechanical package cost.

Process Control System cost is based on a budget quote.

Miscellaneous components, such as CCTV circuit, local push button stations have been quantified in MTOs and priced based on current market prices.

Telecom pricing is based on the current design, using historical in-house rates.

Installation hours are based on contractor quotes received in Q2 2022 for a Northern Ontario project and database hours.

21.5.5 Design Development Growth and Waste Factors

Design growth and wastage factors will be applied to estimated quantities where 100% of design has not been reached. The percentages per discipline are shown below.

21.5.5.1 Wastage and Overbuy

Wastage and overbuy allowances are applied to cover waste of bulk materials during construction. The wastage and overbuy allowances are included with the material costs only. Typical wastage includes such items as; earthworks (over blast), concrete overpour and wastage, cladding overlap, pipe pile cut-offs, cabling left on reels and overpull.

An allowance line item of $1.67M is included for wastage and overbuy. Allocation is based on the distribution presented in Table 21.19.

Table 21.19: Waste and Overbuy Factors (for reference only)

Description	Percentage	Applied to	Notes
Overland Piping	1%	Material unit rate	Allowance for pre-weld preparation and trimming
Concrete	5%	Concrete supply unit rate	Overpour and spillage
Platework	10%	Material unit rate	For overlaps and site adjustment
Architectural	5%	Material unit rate	For sandwich panel overlaps and trimming
Measured Piping	10%	Material unit rate	Pipe trimming and loose of fittings
Structural Steelwork	0%	Material unit rate	Contractor Pricing – Fabricator responsible for wastage
Electrical Cables	5%	Material unit rate	Overbuy

21.5.5.2 Design Development Allowance

Design development allowances are made to cover not-yet-designed items of work within the defined scope (Table 21.20). The design allowance is intended to make allowances for the final quantities and sizes between the current known definition and the final design. The allowances are often referred to as “known unknowns”. The subject design allowances are currently included within the Project contingency.

Table 21.20: Design Development Allowances

Discipline	Calculation Method
Civil Earthworks	10% applied to Quantities
Civil Piping	15% applied to Culvert Quantities
Concrete	2% applied to Mill foundation Quantities

Discipline	Calculation Method
Concrete	5-15% applied to remaining Quantities
Structural Steel	5-15% applied to Quantities
Structural Steel	20% applied to Misc. Small Steel Fabrication Quantities
Ductwork	30% applied to Ductwork Quantities
Plateworks	15% applied to Chute/Hopper Quantities
Plateworks	10% applied to remaining Quantities
Mechanical	0-15% applied to Quantities (only on supply)
Mechanical	25% applied to Sump Pump Quantities (only on supply)
Process Piping	10% applied to Quantities from P&ID
Process Piping	8% applied to Quantities from Model
Electrical Equipment /Powerlines	0-10% applied to Quantities (only on supply)
Electrical Bulks (misc.)	10-15% applied to Quantities
Electrical Cables	0-15% applied to Quantities
Cable Trays	0% applied to Quantities
Instrumentation Bulks	10% applied to Quantities
Telecomm	15% applied to Quantities
Pre-Eng Buildings	3% applied to Building Quantities
Cladding	5-10% applied to Quantities

21.5.5.3 Labour Installation

Bare Labour rates are based on local unition rates provided by Oakbridges to Gen Mining (Table 21.21). Construction rates are based on in-house Wood information for rental rates from contractors in the area. The construction distributables have been estimated and are based on benchmarking Northern Ontario contractor indirects.

Table 21.21: Labour Rate

Discipline	Labour Rates ($)	Labour Rate % of Total	Constr. Equip. Rates ($)	Constr. Equip. Rates % of Total	Constr. Distribut. ($)	Constr. Distribut. % of Total	Total Labour Rate ($)
B-SiteWorks	110.30	57%	25.00	13%	59.53	31%	194.83
C-Concrete	110.00	80%	7.63	6%	20.08	15%	137.71
D-Structural Steel	130.05	69%	15.93	8%	43.79	23%	189.77
E-Platework	129.28	70%	13.72	7%	42.90	23%	185.90
F-Mechanical	129.28	70%	13.72	7%	42.90	23%	185.90
G-Piping	130.47	67%	19.75	10%	45.07	23%	195.29
H-Electrical	138.00	66%	15.75	8%	53.81	26%	207.56
I-Instrumentation	138.00	68%	11.63	6%	52.37	26%	202.00
J-Architecture	124.90	64%	19.85	10%	50.66	26%	195.41

***Note**: Concrete pump is not included in the Concrete Construction Equipment Rates shown above. The Concrete Pump is listed as a sub-contract cost at $100/m3 separate from the labour rates.*

21.5.5.3.1 Direct Labour Rates

Direct Labour Rates (shown in the table above) are from the 25-May-2022 Labour Market Report performed by Oakbridges. They are summarized as follows:

Contractors Work Force (to be confirmed as the execution plan is developed)

- Open-shop site is recommended
- Rotation: 14 x 7, 10-hour days
- Work week 7 x 10 week

The all-inclusive labour cost per hour will be based on the following criteria:

Direct Labour (from Gen Mining/Oakbridges)

- Base labour wage rate was calculated using Building Trades Union rates
- Overtime premiums over 40 hours per week, calculated at double-time rate
- Appropriate crew mixes
- Benefits and burdens (vacation, statutory, small tools, consumables, health, pension, etc.)
- Overhead and profit (profit includes Indigenous revenue sharing)
- Travel and Living Allowance of $160/day

Contractor Distributable Costs

Distributable costs are typically inclusive of but not limited to:

- Contractor mobilization and demobilization
- Contractor temporary facilities and services such as temporary construction buildings and shops, office supplies
- Field office overhead
- Home office overhead
- Construction supervision
- Material management
- General expense and profit
- First aid and safety supplies
- Administration and field construction support labour
- Non-productive time

21.5.5.3.2 Labour Productivity Adjustments

Productivity factors are used to capture the productivity loss due to conditions experienced in the Project's region and specifically the Project site. They are used to fill the gap between normal installation or construction periods and the actual time spent. The majority of rates are based on Contractor quotes received Q2 2022. Where quotes were not received, the unit workhours are based on ideal working conditions:

- Good source of craft labour with large experienced competent contractors, supervision and journeymen.
- A high majority of the workforce live in the area and go home after their shift.
- 60 hours per week or 6 days at 10 hours per day with no overtime, single shift.
- Lump sum contracts in a highly competitive market.
- Green fields work with no obstructions and good access to work fronts.
- Work constructed at ground level.
- Moderate weather conditions: 21ºC, very little rain, wind, snow or ice.

To account for less-than-ideal conditions at the site, productivity factors have been incorporated into the construction labour unit workhours as multipliers on the base manhours. When historical hours are used from other North Ontario projects no productivity adjustment has been made.

21.5.5.3.3 Construction Equipment

The cost of construction equipment, estimated as dollars per direct workhour by Discipline account, will include for rental of the equipment from the contractor. Each discipline account (except Bulk Earthworks, Civil Infrastructure and Detailed Earthworks) reflects the appropriate level of equipment required per workhour. Equipment operator labour costs are included in the composite crew labour mixes. Construction equipment costs for Bulk Earthworks, Civil Infrastructure and Detailed Earthworks are calculated on dollars per type of work and unit, not dollars per craft workhour.

Costs for Heavy Lift Cranes (90 tonne or greater lift capacity) has been included in the indirect cost. The rental rate is from a Q2 2022 Northern Ontario project.

21.5.6 Indirect Costs

Majority of construction indirect costs are provided by Gen Mining. The following sections describes the select indirects costs estimated by Wood.

21.5.6.1 Freight, Logistics, Taxes and Duties

Freight cost has been calculated from vendor quotations where available, this includes several mechanical equipment packages and freight costs per container for steel. For equipment coming from overseas where quoted freight numbers were not provided, a 20% freight cost is used. For equipment coming from in-land where quoted freight numbers were not provided, a 6% freight cost is used. Remainder of material and equipment has been calculated based on historical data as a percentage of the plant equipment and bulk material costs and a percentage of subcontractor's estimate cost. Duties have been included at 1.5% as for the packages originating outside of Canada.

21.5.6.2 Vendor Representatives

Vendor representative's costs have been calculated to capture costs for validation of manufacturer warranties, on-site provision of expertise, supervision in construction erection, pre-commissioning testing and commissioning work. Whenever received suggested vendor representative rates and durations were used. For remaining packages, the cost was calculated based on duration and estimated rate and the travel was estimated based on the location of vendor.

21.5.6.3 Spare Parts

Spare parts have been included based on vendor quotations and factored whenever a quote was not available. Spare parts are broken down into 3 categories: critical, commissioning, and 2-year operational spare parts. The two-year spares are not carried in the capital cost estimate.

21.5.7 Assumptions and Exclusions

21.5.7.1 Assumptions

Listed below are the assumptions:

- All equipment and materials will be new or the equivalent to new (i.e., Hycroft equipment).
- Gen Mining sourced pricing used for Hycroft SAG and ball mills is assumed to be a complete bid, including transformers and VFD's.
- Gen Mining sourced pricing used for the Hycroft Substation is assumed to be a complete bid including concrete and fencing.

21.5.7.2 Exclusions

The following general items are specifically excluded from the Wood capital cost estimate and are carried by Gen Mining as part of the overall Project cost:

- Construction Indirect, including:
  - Engineering and Procurement
  - Construction Management
  - Temporary camp and catering
  - Living out allowance
  - Temporary construction facilities - All temporary construction facilities, offices, sewage treatment plant, and construction telecommunications
  - Construction support and services
  - Construction utilities
  - Health, safety, security and environment ("HSSE")
- First fills
- Owner's Cost
- Permanent Camp
- Operation readiness
- Mining, Pre-Production, and Mine Infrastructure
- Site Clearing
- Site-wide earthworks, including:
  - Haul Roads
  - Camp 19 road upgrades
  - Transmission line corridor and access roads
  - Site water management pipeline corridor earthworks and access roads
- Bulk earthworks including drill and blast of process plant pad
- Effluent treatment & mine water management
- Tailings & site ponds
- Off-site facilities
- Support infrastructure
- Fresh water wells

- Upgrades to the substation yard beyond the Gen Mining's sourced substation
- Upgrades to the existing 115 kV and 25 kV overhead lines
- Geotechnical drilling programs & surveys (including surveys for transmission lines, roads and facilities)
- Cost of financing and interest during construction
- Cost due to currency fluctuations
- Sunk costs
- Costs of further studies
- Scope changes
- Modifications after hand-over
- Changes in Canadian law
- Global supply chain impacts
- Further Global Pandemic impacts
- Taxes and duties
- Reclamation and revegetation
- Sustaining capital
- Expansion costs
- Closure costs
- Any provision for force majeure events
- Cost recovery of construction buildings or equipment
- Schedule delays, not limited to but including:
    - Scope changes
    - Permit delays
    - Delay in notice to proceed/securing of funding
    - Other external influences
- Escalation
- Contingency

21.6 Capital Expenditures

The capital expenditure costs were estimated by the parties defined in section 1.2.2. The capital cost estimate is a detailed, bottoms-up, built-up effort by major facility and discipline. Each discipline executed a detailed cost build up by cost type, labor, material, equipment, consumables, construction materials and services costs.

This capital cost is estimated at $1,112M net of mining equipment financing and pre-production revenue or $898M after equipment financing and pre-commercial production revenue. This estimate has an accuracy within a range of -15% / +20%. A summary of the capital expenditures is presented in Table 21.22.

Labour and equipment costs for the Project were built up in a separate analysis to be included in each individual estimate. MTOs were also performed to generate the baseline quantities for the Project. Each discipline estimate cost, in complete cost type details and quantities and consistent with the Project's WBS, was then accumulated in a master estimate summary.

Most of the critical materials and components will be sourced in North America.

The estimate was developed by major group areas, which are then further subdivided in distinct areas, disciplines and activities and are included in each estimate line item per Gen Mining's standard WBS.

The approach allows for an efficient conversion of the estimate data, which is identical in WBS format to a control budget for project execution.

According to standards established at the outset of the Project, pricing of equipment, material and labor were estimated according to the following guidelines:

- Equipment proposals received specifically for the Project
- Where budgetary / firm quotes were not available, equipment prices were derived from recent project or from databases
- Material prices based on quotations received from suppliers
- The labour rates used are based on rates from the Labour Study provided by Oakbridges

Table 21.22: Capital Expenditures Summary

Capital Expenditures	$ (000)
1000 - Infrastructure	48,691
2000 - Power and Electrical	26,058
3000 - Water	71,354
4000 - Surface Operations	23,133
5000 - Mining	112,688
6000 - Process Plant	347,412
7000 - Construction Indirect	196,634
8000 - Owner's Cost	31,057
9000 - Pre-production, Start-up, Commissioning	158,542
9900 - Contingency	96,514
Total	**1,112,082**

Locally available material was used when possible for estimation purposes and prices were sourced from regional suppliers.

No escalation was built into the capital cost estimates. The estimates are as of Q42022.

The estimates include earthworks, construction material, equipment, and labor. Earthworks will be performed by regional contractors when possible.

21.6.1 Infrastructure

A capital expenditures summary for infrastructure is presented in Table 21.23.

Table 21.23: Infrastructure Capital Expenditures

Area 1000 Infrastructure	$ (000)
1100 – General Site Preparation	**38,323**
111 – General Earthwork	6,745
112 – Site Roads	19,905
1130 – Camp 19 Road	876
1140 – Pad Construction	1,941
1150 – Material Sourcing	8,856
1200 – Mine Infrastructure	**5,150**
1220 – Mine Service Building	5,000
1240 – Emulsion/Explosive Magazine	150
1300 – Support Infrastructure	**2,518**
1310 – Administrative Building	2,368
1320 – Site Guard House	150
1500 – Laboratory	**2,700**
1700 – Fuel Systems	**In OPEX**
Total	**48,691**

21.6.2 Power Supply and Communications

A summary of the capital expenditures for electrical and communications is presented in Table 21.24 This includes all equipment and installations for power supply and distribution. The power line and main site substation costs are negotiated with the power rates with the utility company and therefore are not shown in this table. The electrical infrastructures are detailed in Section 18 - Project Infrastructure.

Table 21.24: Power Supply and Communications Capital Expenditures

Area 2000 Power & Electrical	$ (000)
2100 – Main Power Generation	**18,764**
2120 – Power Line	1,860
2130 – Site Main Substation	16,904
2700 – MV Distribution O/H Line	**3,846**
2900 – IT Network & Fire Detection	**3,448**
Total	**26,058**

21.6.3 Water Management

Details and description of water management infrastructure including the TSF and others and installation and systems are provided in Section 18 - Project Infrastructure. The TSF is built in several phases in which Phase 1 costs are included in the initial CAPEX. All other phases are planned for construction and delivery as per Table 21.33 and therefore are included in sustaining expenditures. Capital costs include earthworks, concrete, structure steel, mechanical, electrical and instrumentation equipment and labor.

The surface water management system is constructed to gather all contact water generated on site. It includes ditches, pumping station and pipelines.

KP prepared the MTOs for the TSF, which have been based on Civil 3D Models, neat line estimates from feasibility level drawings, and geotechnical borehole and test pit information. Quantities have been based on the feasibility level embankment raising schedule which has based on the projected mine production. TSF embankment raise schedule is summarized in Table 21.25 below.

Table 21.25: TSF Embankment Raise Schedule

Year		Embankment Crest Elevation			
Operating	Cashflow Model	WMP (m)	Cell 1 (m)	Cell 2A (m)	Cell 2B (m)
-2	2023	**344**			
-1	2024	344	**315/316**	**326**	
1	2025	344	**332/333**	**334**	
2	2026	344	**343/344**	**343**	
3	2027	344	343/344	**353**	**336**
4	2028	344	343/344	353	**343**
5	2029	344	343/344	**363**	**353**
6	2030	344	343/344	363	**363**
7	2031	344	343/344	**373**	363
8	2032	344	343/344	373	**373**
9	2033	344	343/344	373	373
10	2034	344	343/344	**380**	**380**
11	2035	344	343/344	380	380
12	2036	344	343/344	380	380
13	2037	344	343/344	380	380
14	2038	344	343/344	380	380

***Notes**:*

1. *Embankment elevations based on end of year.*
2. ***Bold** values indicate stage completion during the year, grey indicates no change from previous year.*

As mentioned, budgetary contractor unit costs were used to produce the capital costs, initial and sustaining, mainly based on the mining and construction schedules.

A capital expenditures summary for water is presented in Table 21.26.

Table 21.26: Water Management Capital Expenditures

Area 3000 Water Management	$ (000)
3100 – Fresh Water / Wells	616
3200 – Surface Water Management	30,229
3300 – Potable / Domestic Water	91
3400 – Sewage Water	179
3500 – Fire Water	5,085
3700 – Tailings Storage Facility	35,154
Total	**71,354**

21.6.4 Surface Operations

A summary for the capital expenditures for surface operations equipment is presented in Table 21.27.

The Surface Operations CAPEX consist mainly of the capital expenditure for the acquisition of the mobile equipment (truck and side-dump trailer) required to transport the concentrate from the mill to the transload facility, along with a wheel loader at the transload facility for the re-handling of the concentrate into the railcars. It also includes another wheel loader at the process plant site to load the concentrate transport trucks, along with the required mobile equipment to support the operation and maintenance of the process plant.

A formal Request for Proposal ("RFP") process was completed for the surface operation equipment fleet. The equipment pricing includes, when applicable, tires, transport to the Project site, assembly, and commissioning.

The contractor aggregate crushing plant has been added to crush mine waste to build various infrastructure such as the haul and site roads and the TSF dams.

Table 21.27: Surface Operations Equipment Capital Expenditures

Area 4000 Surface Operations	$ (000)
4100 - Surface Operations Equipment	6,305
4300 - Concrete Batch Plant	965
4800 - Aggregate Plant	15,863
Total	**23,133**

21.6.5 Mining

The mining CAPEX consists of the capital expenditure for the development of the mining infrastructure (roads), the acquisition of the mine mobile equipment, and the mine dewatering equipment required to fully operate as of the first day of operations.

A formal RFP process was completed for the mine mobile equipment fleet. The equipment pricing includes tires, fire suppression, transport to the Project site, assembly and commissioning.

The capital costs estimate are presented in Table 21.28.

Table 21.28: Mining Equipment Capital Expenditures

Area 5000 Mining	$ (000)
5400 – Mine Infrastructure	2,102
5500/5600 – Mine Equipment & Dewatering	110,586
Total	**112,688**

21.6.6 Process Plant and Related Infrastructures

Wood developed equipment specifications, layouts and Basis of Estimate as shown in Section 21.5 of this Report. The capital costs estimate for the processing areas are presented in Table 21.29.

Table 21.29: Processing Capital Expenditures

Area 6000 Processing	$ (000)
6100 – Crushing	72,826
6200 – Grinding	114,958
6300 – Flotation & Regrind	81,544
6500 – Concentrate Dewatering and Handling	21,771
6600 – Tailings Thickening	17,728
6700 – Reagents	7,461
6800 – Process Plant Utilities and Services	26,262
6900 – Process Facilities	4,861
Total	**347,412**

21.6.7 Construction Indirect Costs

Construction indirect costs include all the engineering activities as well as site construction management. Temporary facilities are also included as are tools, operating and maintenance costs for construction equipment.

Construction Indirect Costs are presented in Table 21.30.

Table 21.30: Construction Indirect Capitals

Construction Indirects	$ (000)
7100 – Engineering, CM, PM	79,874
7200 – Construction Facilities & Services	36,352
7300 – Contractor Mob/Demob and Indirect Costs (Includes Plant Crane Rentals Only)	1,319
7400 – Construction Camp Facilities & Operation	44,198
7200 – Freight & Logistics	34,890
Total	**196,634**

21.6.8 Owner's Cost

Initial general services are defined as costs of the operations management that provides services over the construction and commissioning period. It allows operations personnel to increase their involvement in the mine's project activities and reduces the period of operation readiness critical for project success. It gathers the salaries and other personnel related costs for management, supply chain, human resources, environmental and sustainability, security, accounting and information technology. In addition to the above, the insurance costs and taxes are included. Cost estimates are presented in Table 21.31.

Table 21.31: Owner's Cost Expenditures

General Services	$ (000)
8100 – Departments	23,305
8200 – Insurance	3,833
8300/8400/8500 – Others	3,920
Total	**31,057**

21.6.9 Pre-production and Commissioning Expenditures

The pre-production costs are those of the process plant as mining pre-production costs as defined in the operations costs with ramp-up and the general services required to support these activities.

The process plant pre-production includes initial fills as well as salaries, reagents and fuel during the commissioning and ramp-up periods to commercial production. Staffing and training of mill personnel is planned progressively during the construction and commissioning periods.

Pre-production and commissioning expenditures are presented in Table 21.32.

Table 21.32: Pre-production and Commissioning Expenditures

Area	$ (000)
9100 – Mine Preprod / Commissioning	104,694
9400 – Spares	4,732
9500 – Process Plant Preprod / Commissioning / First Fills	49,116
9900 – Contingency	96,514
Subtotal	**255,056**
Equipment Lease Drawdowns and Payments	(58,403)
Pre-Production Revenue	(155,799)
Total	**40,854**

21.6.10 Sustaining Capital

Sustaining capital for the mine includes additional equipment purchases for a total of $130 M. Unlike the 2021 FS, major equipment repairs for the primary equipment fleet are not included in the table below and are instead reflected in overall mining operating costs.

Sustaining capital is presented in Table 21.33.

Table 21.33: Sustaining Capital Costs

Sustaining CAPEX	**Unit**	**Total**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**	**Y13**
Mine Equipment Capital Repairs	$ M	**130**	69	16	14	3	5	4	3	6	4	5	1	0	0
TSF and Water Management	$ M	**197**	11	13	29	19	30	19	20	20	1	34	1	1	0
Power Infrastructure and Upgrade	$ M	**42**	38		2		2								
Truck Shop and Other Infrastructure	$ M	**34**		7	20	6									
Off-Site Infrastructure	$ M	**10**	6	4											
Roads and Earthworks	$ M	**6**	4	2		0									
MRSA Catch Basin	$ M	**2**	2												
Misc. Plant Capital	$ M	**3**	3												
Total Sustaining Capital Cost	**$ M**	**424**	**133**	**42**	**65**	**28**	**37**	**23**	**23**	**26**	**5**	**39**	**2**	**1**	**0**

21.7 Closure Costs

The total closure cost for the project was estimated to be approximately $66.4 M in December 2022 by WSP E&I Canada Limited (not including bonding carrying costs). This cost included the decommissioning and reclamation of the Project site, site water management, and geotechnical and environmental monitoring programs. Closure of the site is proposed to occur in three stages as presented below.

Phase 1 – Active Closure (Years 1 to 5)

During Phase 1, the majority of the physical decommissioning, demolition and reclamation of the Project site will be undertaken. Filling of the open pits with water from the TSF, WMP, SWMP and the MRSA catch basins will be initiated. Closure phase geotechnical and environmental monitoring programs are implemented.

Phase 2 – Passive Closure (Years 6 to 30)

During Phase 2 the open pits continue to fill with water. Runoff collected in the MRSA catch basins is pumped to the pits. Natural drainage is restored from the TSF, WMP and SWMP. Closure phase geotechnical and environmental monitoring programs continue.

Phase 3 – Post-Closure (Years 31 to 50)

During Phase 3 filling of the open pits with water is completed and discharge of excess water to Hare Lake is initiated to maintain the target water level elevation. Closure phase geotechnical and environmental monitoring programs continue. Maintenance of infrastructure is undertaken as required.

21.8 Operating Costs

Operating expenditures (OPEX) are summarized in Table 21.34. The operating costs include mining, processing, General and Administration (“G&A”), royalties, concentrate transportation to smelters and smelting and refining charges. Details on treatment, refining and concentrate transport charges are summarized in Section 19 – Market Studies and Contracts. The transportation costs and smelter conversion charges (TC/RC) are deducted from gross smelter revenues to estimate the NSR.

A summary of the total operating costs including mining, milling, power, G&A and concentrate transportation as well as total cost per tonne milled is presented in Table 21.34.

Table 21.34: Total Operating Costs Summary

Cash Flow Summary	**Unit**	**Total**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**	**Y13**
Mining	$ M	**1,432**	86	125	139	135	140	126	131	135	128	122	81	59	24
Processing	$ M	**1,087**	67	86	87	88	88	88	88	88	88	88	88	88	55
General & Administration	$ M	**334**	13	18	18	18	30	35	26	25	29	26	31	48	18
Treatment & Refining Charges	$ M	**286**	15	23	22	21	22	24	23	24	26	23	26	24	11
Concentrate Transport & Insurance	$ M	**230**	20	20	20	16	15	17	17	19	19	18	20	19	9
Royalties	$ M	**12**	3	2	0		0	1	4	1		(0)			
Total Operation Cost	**$ M**	**3,381**	**205**	**274**	**287**	**278**	**296**	**291**	**289**	**293**	**290**	**276**	**246**	**237**	**118**
Unit Cost	**$/t**	**27.04**	**27.46**	**28.07**	**28.61**	**27.50**	**29.23**	**28.74**	**28.53**	**28.93**	**28.70**	**27.30**	**24.29**	**23.43**	**17.60**

21.8.1 Mining Costs

Table 21.35 presents the breakdown of mining costs, by department.

The mine operating costs are estimated from first principles for all mine activities. Equipment hours required to meet production needs of the LOM plan are based on productivity factors or equipment simulations. Each piece of equipment has an hourly operating cost which includes operating and maintenance labour, fuel and lube, maintenance parts, tires (if required) and ground engaging tools (if required). A budgetary RFP process has been completed for the mine equipment and associated operating costs, fuel, tires, explosives and accessories, etc.

The average mining cost during operations is estimated at $3.25/t mined including re-handling costs. The mining costs are lower than average during the early years and increase with increased haulage distances and pit deepening, in the later years. This operating cost estimate includes capital repairs for the major equipment. Note that in the 2021 FS, most of this amount was carried in sustaining capital.

Mining staffing was based on operating equipment numbers. Maintenance staffing was included to support the operating fleet. Technical staff was included in the estimates. Explosives loading and blasting labour was included as contract services (in operating costs) with no additional labour count.

Haulage is the major mining cost activity representing 30% of total costs followed by loading (14%) blasting (11%), and dump maintenance (9%). Loading and haulage for stockpile re-handling is also captured as a separate activity cost.

Maintenance parts is the dominant cost, by element, representing 28% of total costs, followed by salaries (26%), fuel (22%) and bulk explosives (7%).

Table 21.35: Mining Cost Summary

Mining OPEX		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine Operations	$ M	**34.2**	2.4	3.0	3.0	3.0	3.0	3.0	2.9	2.9	2.9	2.9	2.2	1.9	1.1
Mine Maintenance Admin.	$ M	**63.0**	5.6	5.5	5.7	5.7	5.7	5.0	4.7	4.7	4.7	4.7	4.7	3.7	2.7
Mine Engineering	$ M	**31.9**	2.4	2.6	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.2	2.2	1.0
Mine Geology	$ M	**23.5**	1.8	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.0	0.9	0.9	0.6
Grade Control	$ M	**13.8**	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.8
Drilling	$ M	**53.8**	4.0	5.6	4.9	4.9	4.9	4.9	4.9	5.0	5.0	4.7	2.6	1.7	0.6
Pre-Split Drilling and Blasting	$ M	**36.8**	1.8	2.8	2.8	3.0	4.3	3.9	3.1	4.4	3.3	3.2	2.4	1.3	0.4
Blasting	$ M	**167.0**	13.3	16.5	16.1	16.1	15.3	14.4	14.2	14.7	14.7	13.6	9.3	6.7	2.0
Loading	$ M	**204.1**	11.1	19.1	22.8	20.1	21.5	19.8	20.3	21.7	17.5	14.2	9.3	5.4	1.4
Hauling	$ M	**434.0**	18.6	29.5	42.6	45.6	45.1	38.0	43.9	46.8	48.3	41.8	19.8	12.3	1.7
Support Equipment	$ M	**102.9**	7.5	10.0	9.9	9.1	8.6	8.9	9.0	8.6	8.3	8.7	6.5	4.7	3.3
Dump Maintenance	$ M	**132.1**	8.1	15.3	11.8	11.3	15.7	8.9	13.0	10.1	8.2	12.9	7.3	6.0	3.4
Road Maintenance	$ M	**92.9**	5.7	8.2	10.4	7.4	7.9	10.0	6.3	5.6	6.3	6.8	6.8	6.9	4.6
Dewatering	$ M	**22.0**	1.8	2.1	2.1	2.1	2.1	2.1	2.1	1.7	1.2	1.2	1.2	1.2	1.0
Rehandling	$ M	**19.8**	1.1	1.8	0.6	0.7	0.4	1.1	0.5	3.1	1.5	1.0	5.0	2.9	
Total Mining Costs	**$ M**	**1,431.7**	**86.1**	**125.3**	**138.8**	**134.9**	**140.3**	**125.9**	**130.8**	**135.3**	**127.9**	**121.6**	**81.3**	**58.9**	**24.5**
Unit Cost	**$/t**	**3.26**	**3.16**	**2.92**	**3.32**	**3.23**	**3.35**	**3.04**	**3.19**	**3.15**	**2.98**	**3.09**	**3.67**	**4.88**	**13.01**

21.8.2 Processing Costs

The LOM process operating cost is estimated at $1,087 M over the 15-year LOM (2.5-years pre-production followed by 12.5-year operating life). A breakdown of this value and its unit costs is presented in Table 21.36. Note that the PGM scavenger circuit, previously in the 2021 FS, is no longer included in the flowsheet, and related operating costs removed in the costs below.

21.8.2.1 Labour

Plant staffing was estimated on a zero-based headcount for operating positions with additional benchmarking against similar projects. The labour costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay laboratory, and contractor allowance. The total operational labour averages 99 (19 staff and 80 operational) employees. Assay laboratory services are planned to be contracted.

Individual personnel were divided into their respective positions and classified as either staff or hourly employee. The rates were estimated as overall rates, including all burden costs.

21.8.2.2 Plant General Maintenance Parts, Tools and Supplies

General maintenance costs were 4.6% of the total operating cost at $ 0.43/t of plant feed. Annual maintenance consumable and supply costs were factored based on a total installed mechanical capital cost by area to account for repairs to pumps, mechanical equipment, platework, piping, etc.

21.8.2.3 Power

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services. Power will be supplied by the Ontario Power Generation grid to service the facilities at the site. The total process plant power cost is $2.91/t of plant feed, approximately 30.8% of the total process operating cost.

21.8.2.4 Reagents and Operating Consumables

Individual reagent consumption rates were estimated based on the metallurgical test work results, Wood's in-house database and experience, industry practice, and peer-reviewed literature. Each reagent cost was obtained either through vendor quotations.

Other consumables (e.g., liners for the primary crusher, SAG mill, ball mill, and ball media for the mills) were estimated using:

- Metallurgical testing results (abrasion index)

- Wood's in-house calculation methods, including simulations
- Forecast nominal power consumption

Grinding mills liner wear and grinding media consumptions were selected at the lower end of the range of values, based on abrasion characteristics. Wear rates were adjusted to align with operating results from Canadian operations with similar wear characteristics.

Reagents and consumables represent approximately 42.9% of the total process operating cost at $4.05/t of plant feed.

21.8.2.5 Process Plant Mobile Equipment Maintenance

Other mobile equipment costs were based on a scheduled number of light vehicles and mobile equipment, including fuel, maintenance, spares and tires, etc. These costs represent approximately 3.9% of the total process operating cost at $0.37/t of plant feed.

Table 21.36: Processing Operating Costs

Process OPEX		Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mill Operations Admin.	$ M	**101.4**	6.8	8.1	8.1	8.1	8.1	8.1	8.1	8.1	8.1	8.1	8.1	8.1	5.7
Mill Maintenance Admin.	$ M	**85.3**	5.6	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	6.8	4.8
Process Mobile Equipment	$ M	**42.7**	2.7	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	2.4
Crushing	$ M	**1.9**	0.1	0.1	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1
Grinding	$ M	**386.3**	23.0	30.2	31.0	31.3	31.3	31.3	31.3	31.3	31.3	31.3	31.3	31.3	20.8
Flotation	$ M	**90.1**	5.4	7.0	7.2	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	4.8
Tailings / Concentrate Handling	$ M	**19.4**	1.2	1.5	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.0
Power	$ M	**360.3**	22.5	28.7	29.2	29.4	29.4	29.4	29.4	29.4	29.4	29.4	29.4	29.4	15.6
Total Process OPEX Costs	$ M	**1,087.4**	**67.3**	**85.9**	**87.5**	**88.0**	**88.0**	**88.0**	**88.0**	**88.0**	**88.0**	**88.0**	**87.9**	**87.9**	**55.1**
Unit Cost	$/t	**8.70**	**9.02**	**8.79**	**8.72**	**8.69**	**8.69**	**8.69**	**8.70**	**8.69**	**8.69**	**8.69**	**8.69**	**8.69**	**8.21**

21.8.3 General and Administration

General administration and support services include general management, accounting and finance, IT, environmental and social management, human resources, supply chain, camp, surface support, health and safety, security, operating cost of the various supply chain equipment and costs to be incurred in connection with commitments to and agreements with Indigenous Communities.

In most cases, these services represent fixed costs for the site as a whole. The general administration costs exclude certain costs such as transport of concentrates and environmental rehabilitation costs.

General and Administration staff was estimated on a per position basis with consideration to required positions to support the operations. There was no consideration for head-office support or supervision staff costs associated with any head office employees.

Total G&A cost over the LOM is $334 M or 2.67 $/t milled.

22. ECONOMIC ANALYSIS

This section presents the economic analysis of the Marathon Project. The elements of the economic model principally consist of metal production and revenues, royalty agreements, operating costs, capital costs, sustaining capital, closure and reclamation costs, taxation and net Project cash flow.

The economic analysis is carried out in real terms (i.e., without inflation factors) in Q4 2022 Canadian dollars without any project financing but inclusive of equipment financing, PMPA and costs for closure bonding. The economic results are calculated as of the beginning of Q2 Year -3, which corresponds to the start of the pre-commercial production (or pre-production) CAPEX period (over 10 quarters), including engineering and procurement, with all prior costs treated as sunk costs but considered for the purposes of taxation calculations. The economic results such as the NPV and IRR are calculated on a quarterly basis.

22.1 Assumptions

The key assumptions influencing the economics of the Project include:

- Metal prices and smelter terms stated in US$.
- Canadian dollar to United States dollar exchange rate ("C$/US$")
- Diesel price in $/L and electricity price in $/kWh.

22.1.1 Metal Prices and Smelter Terms

The metal prices and smelter terms selected for the economic evaluation are summarized in Table 22.1. The basis of estimate for metal prices and smelter terms is described in Section 19. Smelter treatment charges are summarized in the table below.

Table 22.1: Metal Prices and Smelter Terms

Metal	Palladium	Copper	Platinum	Gold	Silver
Metal Price	US$1,800/oz	US$3.70/lb	US$1,000/oz	US$1,800/oz	US$20.50/oz
Refining Charges	US$24.50/oz	US$0.079/lb	US$24.50/oz	US$5.00/oz	US$0.50/oz
Payable Rate (%)	95%	96.5%	93%	93.5%	93.5%
Minimum Deduction	2.6 g/t	1.0%	2.6 g/t	1.0 g/t	30 g/t

22.1.2 Exchange Rate

The base case exchange rate for economic evaluation is 1.35 Canadian dollar equals 1 U.S. dollar. Most operating costs are estimated in Canadian dollars with the US dollar denominated metal revenue converted to Canadian dollars. The basis of this estimate is based on forward curve pricing.

22.1.3 Fuel and Electricity

The reference diesel fuel price used for estimating operating costs is $1.17/L, which is an estimated delivered price to site for coloured diesel destined for off-road vehicles. It is inclusive of provincial and federal excise tax. The reference price is based on a methodology shared by a Canadian bank, which correlates historical WTI to Thunder Bay rack diesel price for ultra-low sulfur diesel no. 1. The price assumption does not include a carbon tax cost assumption.

The reference electricity price used for estimating operating costs is $0.07/kWh.

22.2 Metal Production and Revenues

The process plant commissioning period is scheduled over six months. Commercial production is achieved with the plant processing of approximately 60% of nameplate capacity throughput over a period of 30 days. Ramp-up continues for an additional four months to reach 100% of initial nameplate capacity (9.2 Mt/y). Plant commissioning starts in the last quarter of Year -1 and ramp-up continues until the end of Q3 of Year 1. Beginning in Year 2, following the upgrade of the power line, the plant will continue to ramp-up to 10.1 Mtpa by Q4 of Year 2.

Payable metal over the Project life includes 2,122 k oz of palladium, 517 M lbs of copper, 485 k oz of platinum, 157 k oz of gold and 3,156 k oz of silver, as presented in Table 22.2. At the base case metal price assumptions this results in an estimated gross revenue of $8,876 M. Palladium represents 58.0% of gross revenue followed by copper (29.2%), platinum (7.4%), gold (4.3%) and then silver (1.1%). Of this $155.8 M, net of royalties, smelter transport and refining costs is generated during pre-production and is credited against pre-production costs.

The annual process plant schedule and metal production is summarized in Table 22.3. In the first 6 years of operation, the palladium head grade (at 0.76 g/t) is above the average head grade (at 0.63 g/t). Additional details on the production schedule are presented in Section 16.

Table 22.2: Life-of-Mine Metal Production & Revenue Summary

Metal Production		Pre-Prod.	Operations	Total
Tonnage Processed	kt	2,611	125,047	127,659
Concentrate Production	k dmt	27	1,374	1,401
Head Grades				
Cu	%	0.21	0.21	0.21
Ag	g/t	1.57	1.66	1.66
Au	g/t	0.08	0.07	0.07
Pt	g/t	0.23	0.20	0.20
Pd	g/t	0.76	0.62	0.63
Contained Metal				
Cu	M lbs	12	573	586
Ag	koz	132	6,693	6,825
Au	koz	7	279	285
Pt	koz	19	786	806
Pd	koz	64	2,511	2,575
Recovered Metal				
Cu	M lbs	11	537	548
Ag	koz	80	4,450	4,529
Au	koz	4	200	204
Pt	koz	14	593	607
Pd	koz	53	2,213	2,266
Average Recoveries				
Cu	%	87.4%	93.7%	93.5%
Ag	%	60.5%	66.5%	66.4%
Au	%	68.1%	71.6%	71.5%
Pt	%	72.1%	75.4%	75.3%
Pd	%	82.5%	88.1%	88.0%
Payable Metals				
Cu	M lbs	10	507	517
Ag	koz	50	3,106	3,156
Au	koz	3	155	158
Pt	koz	11	474	485
Pd	koz	49	2,073	2,122
Average Payability				
Cu	%	92.7%	94.5%	94.4%
Ag	%	62.8%	69.8%	69.7%
Au	%	77.5%	77.4%	77.4%
Pt	%	80.0%	80.0%	80.0%
Pd	%	93.5%	93.7%	93.7%
Gross Revenue				
Cu	$ M	50	2,534	2,584
Ag	$ M	2	94	96
Au	$ M	8	375	384
Pt	$ M	15	640	655
Pd	$ M	120	5,037	5,157
Total	**$ M**	**195**	**8,681**	**8,876**

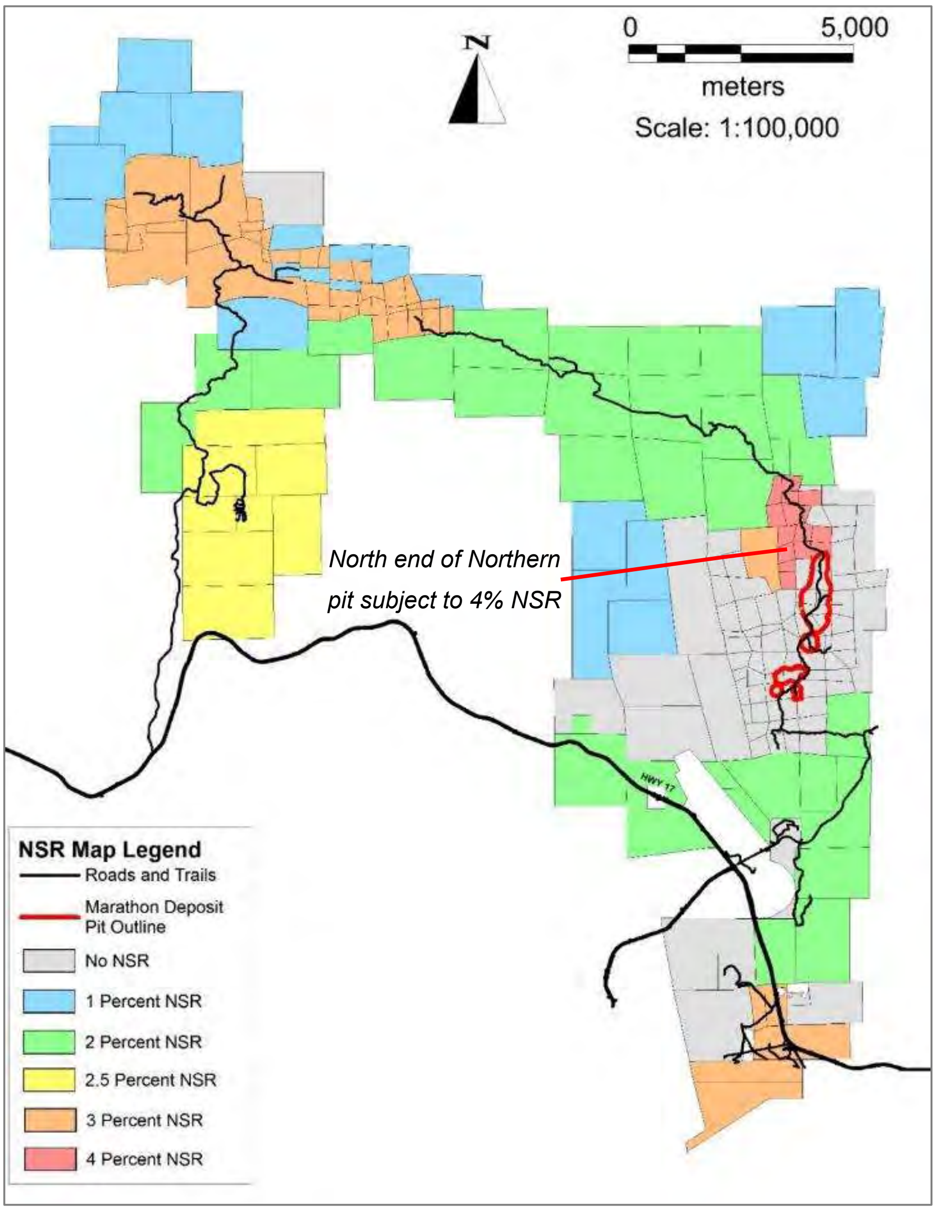
Table 22.3: Annual Metal Production – Operations Period

Physicals Summary (Ops)		Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Operating Years	years	12.5				0.8	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.7
Tonnage Milled	Kt	125,047	-	-	-	7,457	9,775	10,028	10,120	10,120	10,120	10,120	10,120	10,120	10,120	10,120	10,120	6,708
Cu Con. Production	k dmt	1,368				86	128	125	101	92	105	102	113	117	107	118	112	61
Head Grades																		
Cu	%	0.21	-	-	-	0.22	0.26	0.24	0.16	0.13	0.20	0.19	0.23	0.24	0.21	0.24	0.22	0.16
Ag	g/t	1.66	-	-	-	1.29	1.39	1.49	1.54	1.68	1.57	1.51	1.89	1.68	1.89	2.07	1.89	1.64
Au	g/t	0.07	-	-	-	0.07	0.07	0.07	0.09	0.08	0.09	0.07	0.06	0.06	0.06	0.06	0.06	0.05
Pt	g/t	0.20	-	-	-	0.20	0.21	0.20	0.27	0.28	0.27	0.17	0.15	0.17	0.15	0.16	0.17	0.12
Pd	g/t	0.62	-	-	-	0.74	0.80	0.71	0.77	0.77	0.80	0.55	0.51	0.55	0.51	0.54	0.48	0.32
Contained Metal																		
Cu	M lbs	573	-	-	-	37	55	53	36	29	45	42	51	53	46	54	49	23
Ag	koz	6,693	-	-	-	310	438	479	502	545	512	490	614	546	615	674	615	353
Au	koz	279	-	-	-	17	22	22	28	25	28	22	20	21	20	21	21	11
Pt	koz	786	-	-	-	48	65	63	88	92	88	56	49	54	50	53	56	26
Pd	koz	2,511	-	-	-	178	252	229	249	252	260	179	164	180	167	176	155	70
Recovered Metal																		
Cu	M lbs	537	-	-	-	34	52	50	34	27	42	39	47	50	43	50	47	22
Ag	koz	4,450	-	-	-	179	272	310	332	371	343	320	418	371	418	459	418	240
Au	koz	200	-	-	-	11	16	16	21	18	21	16	14	15	14	15	15	8
Pt	koz	593	-	-	-	35	49	47	70	74	70	41	35	39	35	38	41	18
Pd	koz	2,213	-	-	-	154	223	203	221	223	231	158	145	159	147	155	136	61
Average Recoveries																		
Cu	%	93.7%	-	-	-	91.3%	93.8%	94.0%	93.4%	92.9%	93.9%	93.7%	94.0%	94.0%	93.9%	94.0%	94.0%	93.3%
Ag	%	66.5%	-	-	-	57.8%	62.0%	64.7%	66.2%	68.0%	67.0%	65.2%	68.0%	68.0%	68.0%	68.0%	68.0%	68.0%
Au	%	71.6%	-	-	-	69.5%	71.8%	71.5%	74.4%	73.1%	74.6%	71.4%	70.1%	70.8%	70.4%	70.7%	70.6%	67.8%
Pt	%	75.4%	-	-	-	73.1%	75.7%	75.0%	80.0%	80.8%	80.0%	73.1%	71.0%	72.5%	71.3%	72.2%	72.9%	68.0%
Pd	%	88.1%	-	-	-	86.0%	88.5%	88.5%	88.7%	88.7%	88.7%	88.1%	87.9%	88.1%	87.9%	88.0%	87.8%	87.1%
Payable Metals																		
Cu	M lbs	507	-	-	-	31	49	47	31	25	39	37	45	47	41	48	44	20
Ag	koz	3,106	-	-	-	93	147	187	233	280	240	220	306	257	313	343	308	179
Au	koz	155	-	-	-	9	12	12	17	15	17	12	10	11	11	11	11	6
Pt	koz	474	-	-	-	27	38	36	61	66	61	32	25	29	26	28	31	13
Pd	koz	2,073	-	-	-	144	209	190	208	211	218	148	135	148	137	145	126	56
Average Payabilities																		
Cu	%	94.5%	-	-	-	93.1%	94.2%	94.1%	92.9%	92.3%	94.3%	94.7%	95.3%	95.4%	95.1%	95.4%	95.3%	94.1%
Ag	%	69.8%	-	-	-	51.8%	54.0%	60.4%	70.2%	75.5%	70.0%	68.8%	73.3%	69.2%	74.8%	74.7%	73.7%	74.8%
Au	%	77.4%	-	-	-	74.7%	73.8%	74.1%	83.8%	83.3%	83.4%	78.8%	73.6%	74.4%	75.6%	73.9%	75.0%	73.2%
Pt	%	80.0%	-	-	-	77.0%	76.8%	76.4%	87.1%	89.0%	86.8%	79.1%	72.5%	74.9%	74.6%	74.1%	76.8%	71.0%
Pd	%	93.7%	-	-	-	93.5%	93.7%	93.6%	94.1%	94.4%	94.4%	94.1%	93.0%	93.4%	93.5%	93.2%	92.7%	91.2%
Gross Revenue																		
Cu	$ M	2,534	-	-	-	156	245	236	157	126	197	187	226	236	206	240	221	102
Ag	$ M	94	-	-	-	3	4	6	7	8	7	7	9	8	9	10	9	5
Au	$ M	375	-	-	-	21	29	29	42	37	42	30	25	27	26	27	27	13
Pt	$ M	640	-	-	-	36	51	49	82	89	82	44	34	40	36	38	42	17
Pd	$ M	5,037	-	-	-	349	507	461	505	513	530	361	327	360	333	351	306	135
Total	**$ M**	8,681	-	-	-	565	836	780	793	774	859	628	621	670	611	666	605	273

22.3 Royalties

Mining lease 109766 covering the Marathon Project at the north end of the North Pit (Figure 22.1) is subject to a 4% NSR royalty (pink block in Figure 22.1), payable to Teck Resources (2%) and Benton Resources (2%). The ore tonnage mined from these claims is estimated at 6.3 Mt with an estimated NSR value of $432 M, resulting in royalty payments of $16.8 M of which $5.3 M is associated with pre-commercial production ore.

Figure 22.1: Claims Subject to Royalties



22.4 Operating Cost Summary

Operating costs include mining, processing, G&A (including estimated payments to Indigenous communities), concentrate treatment charges, refining charges, concentrate transportation charges and royalties. The operating cost summary is presented in Table 22.4.

Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivities. Additional details on operating costs are presented in Section 21 – Capital and Operating Costs.

Table 22.4: Operating Cost

Category	Total Costs ($ M)	Unit Cost ($/t milled)
Mining	1,432	11.45
Processing	1,087	8.70
G&A	334	2.67
Concentrate Transport Costs	230	1.84
Treatment & Refining Charges	286	2.29
Royalties	12	0.09
Total Operating Cost	**3,381**	**27.04**

The average operating cost for the LOM is $27.04/t of ore processed.

22.5 All-In Sustaining Cost Summary

The AISC, which includes closure, reclamation and sustaining capital costs but excludes the impact of the Wheaton PMPA is presented in Table 22.5 and averages US$813/oz PdEq over the LOM.

Table 22.5: AISC Cost Summary

Category	Total Costs
Total Operating Cost	$3,381 M
Closure & Reclamation	$72 M
Sustaining Capital	$424M
All-in Sustaining Cost (AISC) [2]	**$3,878 M**
All-in Sustaining Cost (AISC) [2]	**US$813/oz PdEq**

22.6 Wheaton PMPA

The terms of the Wheaton PMPA have been described in Section 19. The economic analysis includes the deposits to be received under the PMPA during the development period, and the metal delivered, net of production payments to Wheaton under the terms of the PMPA. A comparison of project economics after-tax with and without the PMPA is presented in Table 22.6. For the purposes of the Technical Report, the PMPA is included in the economic calculations.

Table 22.6: Wheaton PMPA Impact Analysis

ECONOMIC ANALYSIS BASE CASE	UNITS	Including PMPA	Excluding PMPA
After-tax Undiscounted Cash Flow	$M	**2,285**	2,562
After-tax NPV6%	$M	**1,164**	1,285
After-tax IRR	%	**25.8**	24.2
After-tax Payback	years	**2.3**	2.5

22.7 Capital Cost Summary

The capital expenditures include initial capital as well as sustaining capital to be spent after commencement of commercial production. Commercial production has been defined as achieving 60% of nameplate capacity throughput of the process plant for a period of 30 days.

22.7.1 Initial Capital

Initial Capital of $1,112 M includes all costs expected to be incurred as of the Effective Date of the technical report (excluding historical sunk costs) until the point where commercial production is achieved, including costs related to engineering, equipment purchase and installation, process plant and mine infrastructure construction, pre-production operating costs and any other costs associated with putting the Project into operation.

The Initial Capital includes a contingency of $97 M, which is 9.5% of the total CAPEX before contingency as presented in Table 22.7.

The economic analysis also assumes equipment financing for the mine fleet, which will reduce total capital by $58 M, which includes equipment lease drawdowns of $101 M, net of lease payments of $43 M incurred during the construction period.

Additionally, the pre-production ramp up activities generate $156 M of revenues during the development period, which will be credited to total capital cost of the Project.

Table 22.7: Initial Capital Cost Summary

Capital Costs	Initial ($M)
Mining and Surface Equipment	117
Processing Plant	345
Infrastructure	72
TSF, Water Management and Earthworks	95
General and Owner's Costs	31
Construction Indirects	197
Pre-production, Start-up and Commissioning	159
Contingency	97
Sub-Total Initial Capital	**1,112**
Equipment Financing adjustment	(58)
Pre-Production Revenue	(156)
Initial Capital (Adjusted)	**898**

22.7.2 Sustaining Capital Expenditures

Sustaining capital is required during operations principally for additional equipment purchases, mine civil works, TSF expansion and power infrastructure. Capital rebuilds for the mine fleet have been included in operating costs. The sustaining capital is estimated at $424.4 M (Table 22.8).

Table 22.8: Sustaining Capital Summary

Sustaining Capital Costs	$ M
Mining Equipment	129.6
TSF and Water Management	197.5
Power Infrastructure and Power Upgrade	42.2
Truck Shop and Other On-Site Infrastructure	33.6
Off-site Infrastructure	10.5
Roads and Earthworks	6.0
MRSA Catch Basin	2.1
Misc. Plant Capital	3.1
Total Sustaining Capital	**424.4**

22.7.3 Salvage Value

No salvage value is estimated as part of the FS update. Some salvage value could be expected for some mining equipment purchased during operations that will not have been utilized to its useful life. A residual value is probable for some of the major process plant equipment such as grinding mills, crushers and tank agitators.

22.8 Working Capital

Working capital is required to finance supplies in inventory. Given the accessibility of the site, the working capital requirements are considered low compared to remote operations. Additionally, working capital incorporates the benefit of early payment option presented by likely off-takers, which allow for cash receipt of concentrate produced on site, effectively reducing receivables to 15 days for concentrate sales

22.9 Reclamation and Closure Costs

The total closure cost for the project was estimated to be approximately $66.5 M in December 2022 by WSP E&I Canada Limited. This cost included the decommissioning and reclamation of the Project site, site water management, and geotechnical and environmental monitoring programs. As part of the Project obligations with the province, financial assurance in an amount equal to the closure estimate must be posted with the Ministry of Mines. The model assumes this financial assurance will be posted through typical surety bond. The total closure cost including carrying interest is $72M.

22.10 Taxation

The after-tax results assume of a taxable Canadian entity with tax calculated based on the tax rules in Ontario and Canada. The calculations reflect the benefit of any historical tax positions held by Gen PGM as of December 31, 2022. The Ontario mining tax, federal income tax and provincial income tax during the LOM is estimated at $1,102 M.

22.10.1 Ontario Mining Tax

Ontario mining tax is levied at a rate of 10% on taxable profit in excess of $0.5 M derived from a mining operation in Ontario. There are specific guidelines for the calculation of profit and depreciation for the purpose of the Ontario mining tax. The total Ontario mining tax is estimated at $302 M over the LOM.

22.10.2 Income Taxes

The federal and provincial income taxes have both been estimated from an identical taxable income which is arrived at by deducting the Ontario mining tax and various tax depreciation allowances. The federal income tax rate is 15% while the Ontario income tax rate is 10%. The total federal income tax is estimated at $480 M and the provincial income tax at $320 M.

22.11 Economic Results

The main economic metrics used to evaluate the Project consist of net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. The base case discount rate used to evaluate the present value of the Project is 6%. Sensitivities have been presented at various discount rates ranging from 0 to 10% (Table 22.11).

A summary of the Project economic results is presented in Table 22.9 and the annual Project cash flows are presented in Table 22.10. The total after-tax cash flow over the LOM is $2,285 M and after-tax $NPV_{6\%}$ is $1,164 M. The after-tax Project cash flow results in a 2.3-year payback period from the commencement of commercial operations with an after-tax IRR of 25.8%.

Table 22.9: Project Economic Results Summary

Project Economics - Base Case Results					
Production Summary (LOM)					
Tonnage Mined (Mt)					459.7
Ore Processed (Mt)					127.7
Strip Ratio (W:O)					2.60
Cu Concentrate (k dmt)					1,401
Metal	**Cu**	**Ag**	**Au**	**Pt**	**Pd**
Head Grade (% for Cu and g/t for others)	0.21	1.66	0.07	0.20	0.63
Cont. Metal (Mlbs / kozs)	586	6,825	285	806	2,575
Rec. Metal (Mlbs / kozs)	548	4,529	204	607	2,266
Pay. Metal (Mlbs / kozs)	517	3,156	158	485	2,122
Cash Flow Summary ($ M)					
Gross Revenue					8,681
Mining Costs (incl. rehandle)					(1,432)
Processing Costs					(1,087)
Concentrate Transportation					(230)
Treatment & Refining charges					(286)
G&A Costs (incl. CBA payments)					(334)
Royalty Costs					(12)
Total Operating Costs					(3,381)
Operating Cash Flow Before Taxes					5,301
Initial CAPEX					(959)
Sustaining CAPEX					(424)
Total CAPEX					(1,384)
Working Capital Adjustment					22
Closure Costs					(72)
Stream Adjustment (net of production payments and prepayments)					(420)
Interest and Financing Expenses					(59)
Taxes (mining, prov. & fed.)					(1,102)
Before-Tax Results					
Before-Tax Undiscounted Cash Flow (M $)					3,387
NPV 6% Before-Tax					1,798
Project Before-Tax Payback Period					2.0
Project Before-Tax IRR					31.9%
After-Tax Results					
After-Tax Undiscounted Cash Flow					2,285
NPV 6% After-Tax					1,164
Project After-Tax Payback Period					2.3
Project After-Tax IRR					25.8%

Table 22.10: Project Cash Flow Summary

Cash Flow Summary	Total[3]	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15
Gross Revenue	**8,681**				565	836	780	793	774	859	628	621	670	611	666	605	273		
Mining	**(1,432)**				(86)	(125)	(139)	(135)	(140)	(126)	(131)	(135)	(128)	(122)	(81)	(59)	(24)		
Processing	**(1,087)**				(67)	(86)	(87)	(88)	(88)	(88)	(88)	(88)	(88)	(88)	(88)	(88)	(55)		
General & Administration	**(334)**				(13)	(18)	(18)	(18)	(30)	(35)	(26)	(25)	(29)	(26)	(31)	(48)	(18)		
Treatment & Refining Charges	**(286)**				(15)	(23)	(22)	(21)	(22)	(24)	(23)	(24)	(26)	(23)	(26)	(24)	(11)		
Concentrate Transport & Insurance	**(230)**				(20)	(20)	(20)	(16)	(15)	(17)	(17)	(19)	(19)	(18)	(20)	(19)	(9)		
Royalties	**(12)**			(1)	(3)	(2)	(0)		(0)	(1)	(4)	(1)		0					
Total operating costs	**(3,381)**			(1)	(205)	(274)	(287)	(278)	(296)	(291)	(289)	(293)	(290)	(276)	(246)	(237)	(118)		
EBITDA	**5,301**			(1)	361	562	493	515	478	568	339	329	380	334	420	368	155		
Initial Capital (excl. Contingency)[1]	**(860)**	(167)	(542)	(197)	47														
Contingency	**(97)**	(10)	(65)	(22)															
Sustaining Capital	**(424)**				(133)	(42)	(65)	(28)	(37)	(23)	(23)	(26)	(5)	(39)	(2)	(1)	(0)		
Change in Working Capital	**19**	1	1	(24)	(19)	(6)	3	(1)	1	(4)	10	1	(2)	2	(5)	1	63	(3)	
Closure Costs	**(72)**	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(66)										
Stream Adjustments (Net of Deposits)		**(420)**		140	60	(40)	(47)	(46)	(71)	(68)	(74)	(49)	(40)	(44)	(42)	(43)	(41)	(16)	
Equip. Financing Drawdowns	**204**	19	65	8	70	13	11	1	2	2	2	4	3	4	0				
Equip. Financing Payments (Capital + Int)	**(263)**	(0)	(16)	(20)	(30)	(37)	(41)	(40)	(23)	(22)	(12)	(6)	(3)	(3)	(3)	(3)	(2)	(1)	(1)
Pre-Tax Cash Flow	**3,387**	(158)	(417)	(196)	254	441	354	376	286	447	268	261	329	256	368	324	200	(4)	(1)
Taxes (Mining, Prov., Fed.)	**(1,102)**					(85)	(102)	(125)	(104)	(133)	(71)	(73)	(93)	(79)	(110)	(99)	(29)		
After-Tax Cash Flow	**2,285**	(158)	(417)	(196)	254	356	252	251	182	313	197	188	236	177	258	226	171	(4)	(1)

Notes:
1. *Pre-production metal revenue treated as credit against pre-production costs in construction capital.*
2. *Numbers may not add due to rounding.*

22.12 Sensitivity Analysis

A sensitivity analysis was performed for variations for metal price, exchange rate, OPEX and CAPEX. Each parameter was calculated independent of any correlations that may exist between variables such as for gold price and exchange rate, which tend to be negatively correlated.

The Project is most sensitive to metal prices and exchange rate followed by operating costs and initial capital costs. The exchange rate sensitivity is similar to metal prices.

The results of the sensitivity analysis on the NPV 6% and IRR is presented in Table 22.11 and in Figure 22.2 and Figure 22.3 respectively.

Figure 22.2: After-Tax NPV 6% Sensitivity

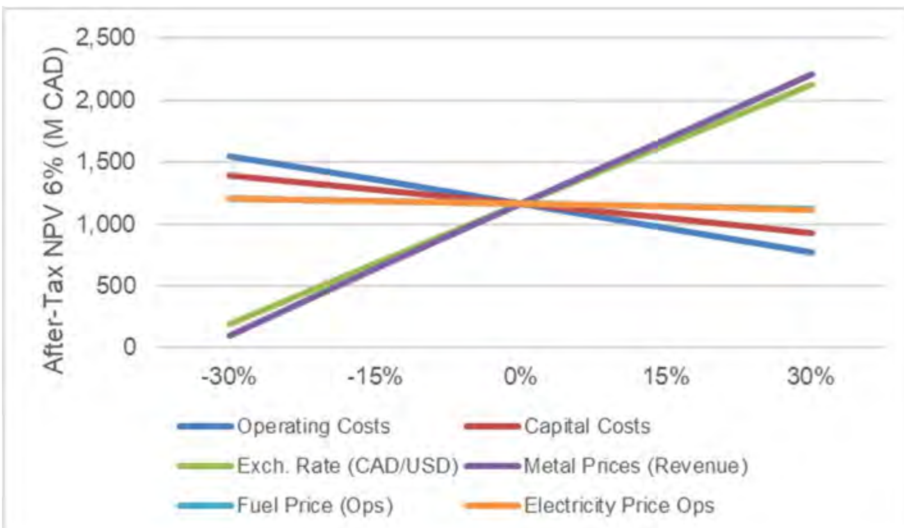
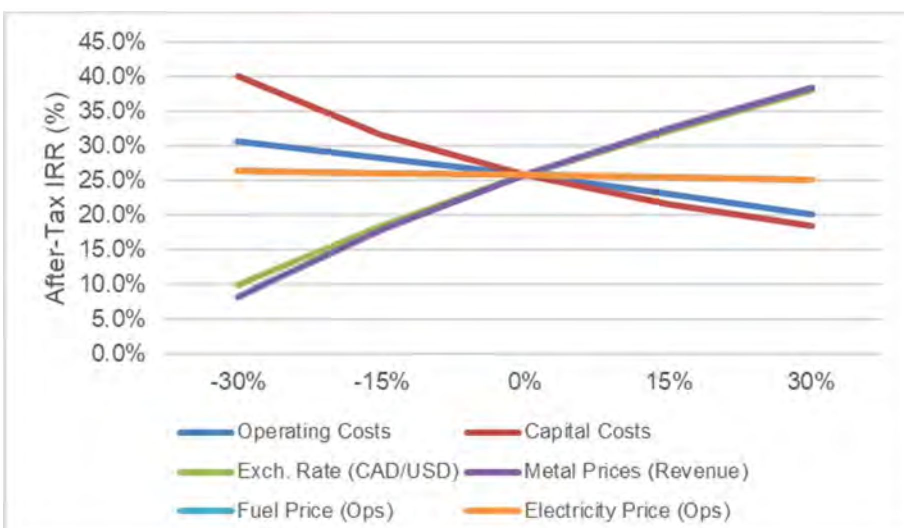


Figure 22.3: After-Tax IRR Sensitivity



***Note**: Exchange Rate and Metal Price Sensitivity are very similar with overlapping lines.*

Table 22.11: Project After-Tax Sensitivities

Palladium Price US$/oz	1,400	1,600	1,700	1,800	1,900	2,000	2,200
NPV6% ($M)	696	930	1,047	**1,164**	1,282	1,400	1,634
Payback (yrs)	3.3	2.9	2.5	**2.3**	2.2	2.0	1.9
IRR (%)	18.5%	22.3%	24.0%	**25.8%**	27.5%	29.1%	32.3%

Copper Price US$/lb	2.5	3.0	3.5	3.7	3.9	4.5	5.0
NPV6% ($M)	836	972	1,109	**1,164**	1,219	1,386	1,522
Payback (yrs)	3.0	2.6	2.4	**2.3**	2.2	2.0	1.9
IRR (%)	21.1%	23.1%	25.0%	**25.8%**	26.5%	28.7%	30.4%

After-Tax Results	OPEX Sensitivity				
	+30%	+15%	0%	-15%	-30%
NPV 6% ($M)	1,031	1,085	**1,164**	1,274	1,411
Payback (yrs)	2.7	2.5	**2.3**	2.1	2.0
IRR (%)	23.4%	24.4%	**25.8%**	27.4%	29.2%

After-Tax Results	CAPEX Sensitivity				
	+30%	+15%	0%	-15%	-30%
NPV 6% ($M)	932	1,048	**1,164**	1,281	1,397
Payback (yrs)	3.3	3.0	**2.3**	1.9	1.3
IRR (%)	18.4%	21.6%	**25.8%**	31.6%	40.1%

Discount Rate Sensitivity	NPV (After-Tax) ($M)
0%	2,285
5%	1,303
6%	**1,164**
8%	925
10%	731

Foreign Exchange Rate C$:US$	NPV (After-Tax) ($M)
1.25	928
1.30	1,046
1.35	**1,164**
1.40	1,284
1.45	1,403

Fuel Price Sensitivity	NPV (After-Tax) ($M)
0.90	1,197
1.00	1,185
1.10	1,173
1.17	**1,164**
1.30	1,148
1.40	1,136

Power Price Sensitivity ($/kWhr)	NPV (After-Tax) ($M)
0.05	1,207
0.06	1,186
0.07	**1,164**
0.08	1,143
0.09	1,121
0.10	1,100

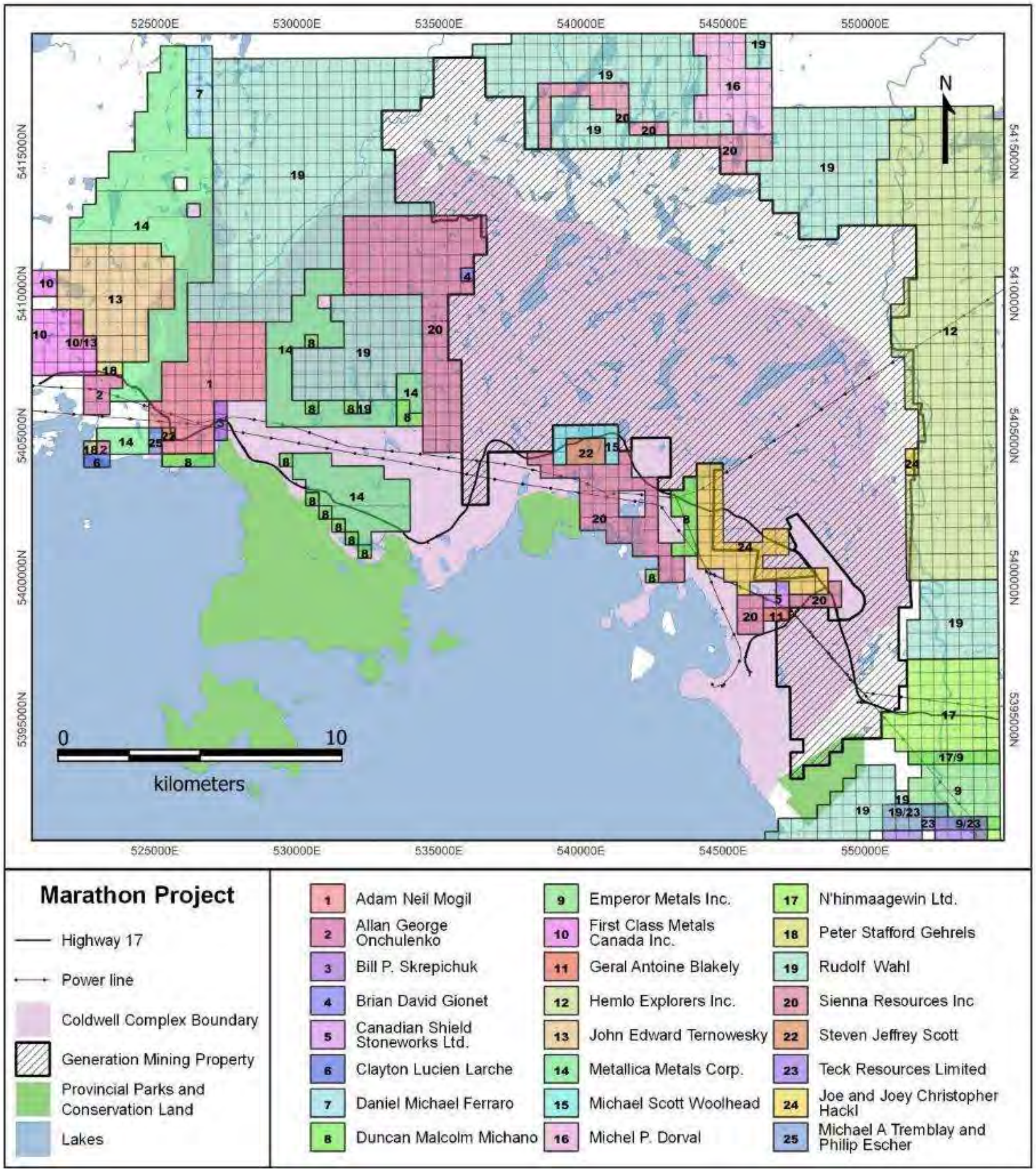
23. ADJACENT PROPERTIES

There are 24 land holders adjacent to Gen Mining's land holding or covering a portion of the Coldwell Complex (Figure 23.1). The land holders include exploration companies, but are predominantly individual prospectors. There are no adjacent properties that have any significant information relating to the Project. Generation Mining maintains the largest land position in the CC, and most of the historical mineral deposits areas are located within the boundaries of the Company's property boundary.

There is a historical showing on the southwest boundary of the CC, called the Middleton Occurrence that is separately staked by Duncan Michano. The Middleton Occurrence is located approximately 0.5 km south of the mouth of Dead Horse Creek and is accessed by trail from Highway 17 east of Dead Horse Creek. The property was explored from 1988 to present with the initial discovery of copper mineralization in the form of malachite precipitate coating fractures, joints, and foliation surfaces. Initial grab samples from 1989 - 1991 by prospects and Resident Geologist staff returned assays results with varied from nil to 0.064 oz/t Au, nil to 4.34 oz/t Ag, 0.055 to 13.925% Cu and 0.008 to 0.019% Zn (Mckay and Pettigrew, 1997).

The land north of the Property, and the western portion of the Coldwell Complex has recently been staked by various companies focused on PGM exploration. These projects are not related to the Eastern Gabbro which host the Marathon and Sally Deposits, but rather to the Western Gabbro or Archean footwall adjacent to the Coldwell Complex. To the east, Hemlo Explorers Inc. has staked within the footwall margin of the Coldwell Complex covering the Pic Project to explore for Archean gold. A recent publication by Good et. al (2021) has suggested that the footprint of the Coldwell Complex could be much more extensive than previously thought, with potentially PGM-Cu bearing CC related intrusions extending as far as 700 km outside the rim of the Coldwell Complex. This has led to renewed focus on PGM-Cu exploration by several adjacent landowners, however, no significant discoveries have been reported.

Figure 23.1: Claims Adjacent to Gen Mining's Marathon Property



Source: *Gen Mining (2023).*

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

The Project execution strategy is to employ integrated EPCM. Engineering and procurement will be performed by various parties given the specific area and scope. This will result in a Project management team with both the employees of the Company and the consulting firms throughout the execution and commissioning phases with experience in implementing similar sized projects.

The Project team will manage and execute the engineering, procurement and construction, provide Project control, staff for start-up and operation, and commission the mine and process areas. Certain operation departments (mining, environment and training) will be integrated in the Project team early in the process to allow their parallel development and will focus on the ORP.

The site’s location and proximity to well qualified contracting companies allows for the on-site labour services in the construction phase to be provided by contracting groups who are both local-regional and/or from across Ontario and Canada.

Reputable third-party consultants and engineers will be hired to complete the detailed design engineering and QA/QC work to reduce the risks to Gen Mining’s critical and specialized components, such as the powerline engineering, long-lead items for the process plant, TSF, water treatment, and environmental issues. Specifications established by these firms will be approved by the team in charge of each task.

The overall result of this style of management will be the placement of experienced and skilled personnel in their respective positions, which will result in overseeing a qualified workforce and taking advantage of the developed industries near the site.

The ultimate Project authority lies with Gen Mining, with the Project Director having responsibility for all site Project personnel until the completion of the Project.

24.2 Project Development Organization

The Project team will comprise the following departments: Engineering, Project Services (Project Controls, Contracts, Supply Chain, and Site Services), Construction, Environment and H&S all reporting to the Project Manager. The operations group consists of a mining group, mill group and general services group, which are all created in advance to support execution activities or to begin pre-production planning activities. These teams will work together with the objective of safely executing the project and managing the project capital costs through planning of equipment use, staff and employees, construction activities, project commissioning, and project start-up.

Detailed engineering and a portion of the procurement will be outsourced. The Project construction period is estimated at 24 months. Estimated construction labour is to average approximately 520 full-time equivalents over the construction period and a peak of approximately 800 full-time equivalent contractors and employees on the Project.

In parallel to the construction phase, the Company's General Manager will progress and develop the ORP. The ORP will establish all critical operating systems and operating procedures to allow for efficient start-up and ramp-up into Commercial Production.

Once the Project has been commissioned, the Project Manger will hand-over responsibility for the site to the General Manager. The Construction Group will be phased out as the remaining construction activities are completed. The Project Manager will remain on the Project assisting the Operations team as needed to achieve commercial production.

24.3 Project Engineering

The basic and detailed engineering phases of the Project will be managed by the Project Management Team ("PMT") who will appoint an Engineering Manager to supervise, direct and control the engineering work for the Project and will report directly to the Project Manager. The Engineering Manager will manage consultants and contractors who are responsible for individual engineering work packages.

Basic engineering is finalized with the completion of principal trade-offs, plant site alternatives, TSF options, and process specifications. In addition, the process flowsheets have been finalized and all of the design criteria and equipment specifications are complete.

Engineering work will be scheduled to meet critical dates for placing purchase orders and construction activities.

Some areas of engineering will be contracted to specialized firms, such as:

- Tailings Storage and permanent Water Management Facilities by Knight Piésold.
- Process Plant by Wood
- General Infrastructure Facilities by JDS
- Water Management Pump Systems by Technosub

24.4 Early Works

This update has identified activities to be developed as early as possible to meet the Project schedule.

The Owner's mining group (including personnel for supervision, mining operations and equipment maintenance) and the development mining fleet will be mobilized to support the waste development from the open pit footprint during the Project phase and will provide waste material for construction and infrastructure. Additionally, the mining fleet will stockpile ore during the construction phase of the Project.

This work will be done in parallel with the contractor scope of work (including plant and infrastructure earthworks, roads, TSF and water management structures).

Detailed engineering and procurement activities for the process plant have advanced to 42% (as of the effective date of this Technical Report). Some plant equipment packages will be awarded as a 2-phase PO following Gen Mining's approval of partial or full financing for the Project. The first phase will provide critical engineering data to support timely engineering development while the second phase will be full release for manufacture.

Detailed design for TSF, MRSA, SWMP, Water Management Structures and Site Water Balance are advanced to 75%. Construction Water Management and Construction Quality Management Plans have been developed to ensure environmental and regulatory compliance. Dam breach assessment has been completed and reviewed with relevant stakeholders.

Arrangements have been made to procure the SAG mill and ball mill from Hycroft, as well as two WEG Transformers from ABB for the main substation. This equipment was previously delivered (Hycroft's predecessor company) but never installed. Purchase of this equipment helps mitigate potential schedule impacts caused by manufacturing and fabrication delays, in addition to cost savings from purchasing existing equipment versus new equipment at current market pricing. Updating the ABB drives and cosmetic improvements to the mills will be completed prior to delivery to site for the Project; the equipment will be provided with OEM warranties typical for this sort of equipment.

Early construction works will involve clearing and grubbing of key locations (such as the process plant pad and select water management areas), upgrading the Camp 19 road, establishing pioneer access roads, and select temporary installations for the mining equipment, and temporary power access.

Engineering and coordination of the main powerline with Hydro One is also part of the early works. A schedule will be established to meet the Project requirements. IESO has developed a System Impact Assessment which will be the basis of Hydro One's Connection Impact Assessment Report.

24.5 Basis of Detailed Engineering/Procurement

The Marathon site has been mapped using LIDAR. The general arrangement uses the topographic information from the LIDAR in the form of UTM coordinates in addition to other specific detailed surveys to locate infrastructure and estimate earthworks.

Significant site geotechnical studies were conducted as part of basic and detailed engineering on the Marathon site to inform the designs of the various buildings, dams, and other infrastructure. Additional geotechnical studies (drilling, test pitting, etc.) are planned to occur on an ongoing basis to minimize both technical and cost risks during construction. In general, consistent values for permeability, soil type, strata thickness, density, and bearing capacity are present throughout the site.

In addition, overburden coverage is minimal over most of the site area and the underlying rock is generally competent. Most of the major building foundations will be constructed on bedrock or competent fill overlying bedrock. Geo-mechanical studies have been conducted as part of the metallurgical/grinding design process, and to inform the design basis for building and infrastructure foundations and excavations.

24.6 Procurement and Contracts

Project procurement and contracts takes a cross-functional approach which aims to coordinate, include, integrate, and align for an effective construction and Project execution strategy. This will consider and include the Gen Mining operations team, engineering firms, contractors, subcontractors, equipment and material suppliers, raw material suppliers, the Town of Marathon and the First Nation community of BN and applicable Indigenous communities.

BN and Gen Mining have executed a CBA. As a component of the CBA, there are contracting *set-asides* and defined contracting processes which the Project procurement team will follow and work closely with the BN leadership.

The most effective and applicable methods and approaches to procure the Project components are incorporated into the Project procurement and contracts process from the start of basic and detailed engineering and continues throughout construction. On Project completion, the procurement and contracts systems and procedures will be transitioned to operations.

The Project procurement and contracting strategy includes, to the extent practical and where available, that goods and services will be procured locally to support and develop local businesses. As part of the community engagement strategy, local groups are considered as one of the key services and labour providers for Project and the PMT is committed, in the case of similar capacities and capabilities, that priority is given to local labour resources. Whilst preference shall be given to local procurement, the quotes received for sourcing services or commodities in all instances shall be evaluated against those in the global market. If there are competing and qualified bids of the same value from local businesses, the priority is to engage vendors and contractors significantly impacted by the mine’s development.

As part of tender process, preferably, at least three qualified and competent bidders will be invited to submit a tender, where only bidders with technical competency and suitability of the proposed service will be shortlisted.

The PMT will ensure a rigorous bidder pre-qualification process and a thorough tender assessment process is employed, such that chosen contractors and suppliers are able to meet safety, schedule, technical and quality obligations and not just the bidding price will be factored in a contractor’s selection, but the best overall offer.

This will be done by way of developing bidder lists consistent with requirements and specifications. The bidder lists will, as need be, compiled by way of a pre-qualification process. In addition, the Procurement and Contracts team shall ensure all contractors and suppliers are cognizant of the prevailing conditions on-site to the extent that a site visit will be mandatory for all bidders intending to work on site.

The contents of the contracts shall define the necessary scope, quality, cost, and timely contract results required, along with the necessary level of warranties and guarantees required by the Project.

24.7 Project Controls

The strategy, approach and outline of Project controls is to provide sufficient and timely information to ensure that the Project is completed to meet the goals, dates and costs as outlined in the master schedule and approved budget.

To ensure that the Project is executed successfully, the Project monitoring and control approach will convey both accurate and timely information to assist management staff in making informed decisions. The goal of the Project monitoring system is to allow management staff to clearly understand the following:

- Historical performance of the Project on a daily, weekly, monthly, yearly and Project to date basis, as applicable, depending on the type of information being tracked, and how it compares to the planned performance as set out in the master schedule and approved budget
- Future performance of the Project in order to meet the Project goals, which shall be both realistic and achievable when compared to the historical data

To maximize efficiency and transparency, the Project controls approach will also be structured to guide management staff to any critical areas to allow focused discussions and decision-making, while minimizing the possibility of critical variances.

24.8 Quality and Design Standards

The Project's infrastructure and equipment will be designed based on the relevant Ontario / Canadian design codes and standards using qualified and proven manufacturers.

The process plant and Project infrastructure will be designed with a minimum 20-year design life.

Health and safety standards will comply with all relevant regional regulations, industry best practices and Gen Mining's requirements.

Process solids and water management dams and dikes will be designed following Best Available Practices and Technologies in addition to guidelines published by the Canadian Dam Association's Dam Safety Guidelines ("CDA").

24.9 Quality Management

QA/QC of all construction activities will be performed by a suitably accredited third-party engineering firm under the direct supervision of the Resident Engineer(s) for the various areas of the Project. All QA/QC documentation will be stored in the document control system for archival and review purposes.

QA/QC of welding for critical structures (e.g., fuel tanks) will also be performed by a suitably accredited inspection firm. All QA/QC documentation is posted to the document control system for archival and review purposes.

The process equipment will be subject to vendor verifications and factory acceptance testing programs included in the Project procurement plan with consideration to an overall process equipment risk analysis.

24.10 Commissioning

As Project areas are mechanically completed, commissioning activities begin immediately. There are three basic stages of commissioning checks: dry, wet and ore commissioning. Dry commissioning checks verify the correct installation of equipment and proper connections to all interfaces: electrical, instrumentation, and piping. Wet commissioning verifies the integrity of tanks and piping connections as well as proper equipment functionality. Ore commissioning is the final stage of the process plant. As ore moves through-out each processing area, the operating parameters, checklists and procedures are monitored, adjusted and documented.

Commissioning checklists are continually updated and uploaded to the document control system by the site commissioning team as commissioning progresses. Commissioning of high value or complex process equipment will be supported by vendor representatives who will also provide specialized operations and maintenance training to the operations staff.

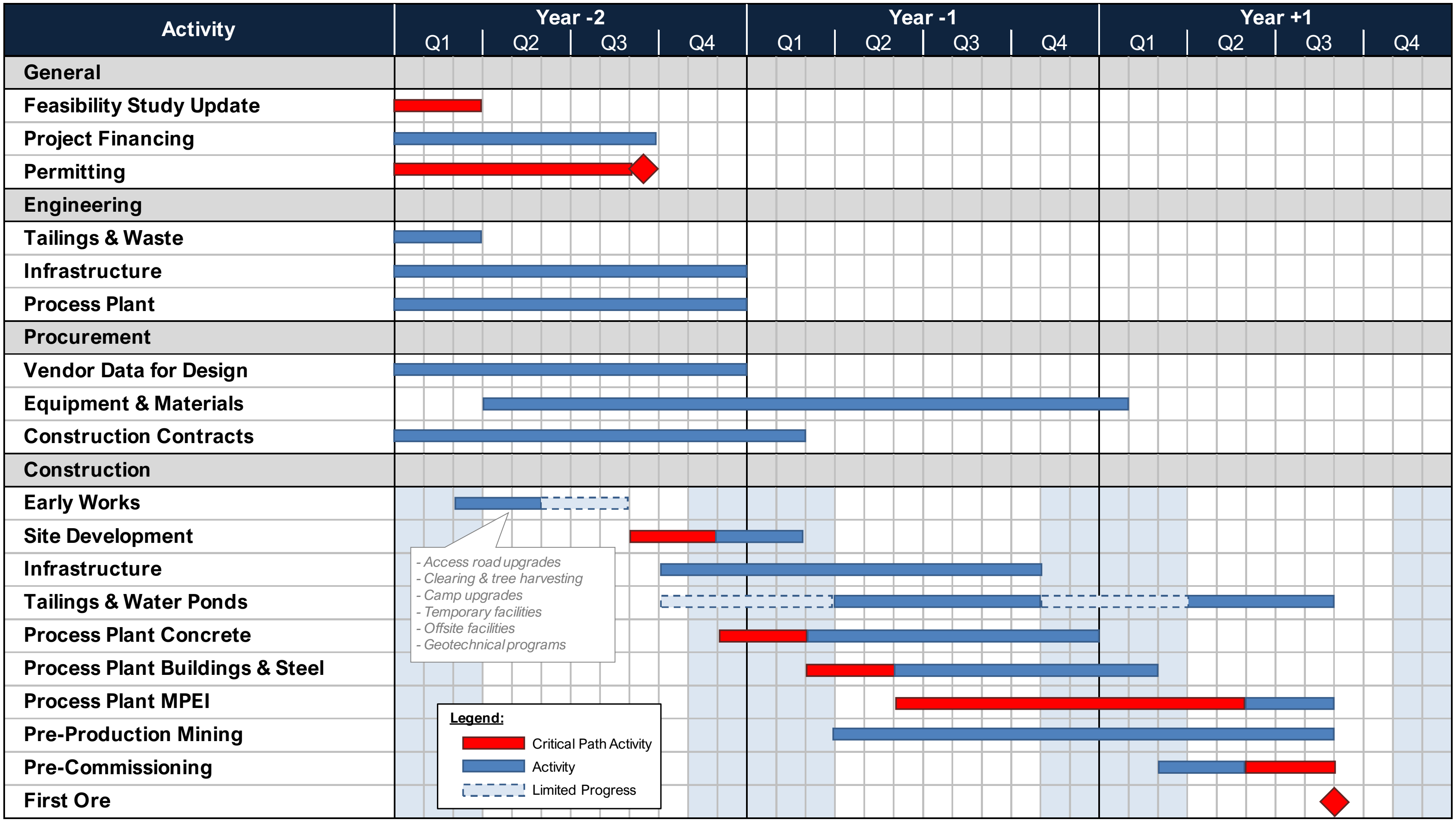
The automation team will be on-site as process equipment installation begins to ensure that pre-assembled and bench-tested automation systems are functional to reduce the commissioning time. As equipment is installed, input / output interfaces are verified, controls are tested, and automation drawings are updated to as-built-drawings.

Equipment technical documentation and checklists will be stored on the document control system and continually updated and reviewed for the entire plant. This will ensure that by the end of commissioning, the operation team has all necessary information to ensure a smooth transition from construction to operations.

24.11 Project Schedule

The construction and pre-production development schedule is 24 months, consisting of three months for initial mobilization of key personnel and equipment and 21 months of on-site construction activities from the start of site development. The high-level Project schedule is summarized in Figure 24.1.

Figure 24.1: High-Level Project Schedule



25. INTERPRETATION AND CONCLUSIONS

The completion of this Technical Report has confirmed the technical and economic viability of the Marathon Project, based on an open pit mining operation with a production rate of approximately 40 Mtpy and an SAB / flotation plant operating at up to 10.1 Mtpy.

The main conclusions are detailed below:

25.1 Geology and Mineral Resources

- The understanding of the Project geology, structure and mineralization, together with the deposit type, and origin of the mineralization is sufficiently well established to support Mineral Resource and Mineral Reserve estimation.
- Mineral Resources using a NSR cut-off value of $15/t within a constraining pit shell is appropriate for reporting Mineral Resources for the Project.
- The Mineral Resource model is suitable and fit for the FS.

25.2 Mineral Reserves

- The Mineral Reserve estimate has been completed to a level appropriate to support a project commencement decision.
- Mineral Reserves are estimated at a cut-off grade of $16.90 NSR/t of ore resulting in a total of Proven and Probable tonnage of 127 Mt of ore.
- The methodology and adjustments for ore loss and dilution are appropriate for the current level of study and appropriate for the equipment and operating conditions that are expected at the future operation. As with any operation, ongoing refinement will continue during operations to achieve operational reconciliation in-line with Mineral Reserve estimates.
- The Mineral Reserves are based on Measured and Indicated Mineral Resources and do not include any Inferred Mineral Resources. Mineral Reserves were estimated only for the Marathon deposit.

25.3 Mining

- The mine design has been completed to a level appropriate to support a project commencement decision. On-going refinement should continue to further optimize and de-risk the mine plan.
- The mine design, equipment and operating conditions that are expected at the future operation is appropriate and aligned with the estimated ore loss and dilution modeled in the study. As with any

operation, ongoing refinement will continue during operations to achieve operational reconciliation in-line with Mineral Reserve estimates.

25.4 Metallurgical Testing and Mineral Processing

- The optimized process flowsheet and process design criteria were established from operational considerations and metallurgical test programs completed by Gen Mining during 2020-2022 and included historical rock hardness test data where appropriate.
- Industrial benchmarks along with Owner, Vendor and Wood design engineering subject matter experts have been considered and included within the Project design concepts.
- The process plant flowsheet includes a conventional comminution circuit consisting of a SAG mill, followed by a ball mill (SAB).
- The flotation portion of the process plant includes rougher flotation, concentrate regrind and three stages of cleaning.
- The latest testwork completed during Q4 2022 has resulted in the removal of the PGM-Scavenger circuit from the current LOM plan. Further evaluation may be considered during operations as metal prices allow.
- The flotation circuit design incorporates conventional tanks cells for the roughers and Woodgrove SFRs for the cleaning circuit.
- The process plant is designed to operate at an average throughput of 27,700 tpd.
- Metallurgical recovery curves were established for each element in the 2022 metallurgical test program and are representative of the expected geo-met recoveries.

25.5 Infrastructure

- The infrastructure considered for the Project is appropriate to support the operation.
- The power connection to the existing M2W line is suitable for the initial site power requirements up to 9.2 Mtpa. Additional power will be required to increase plant capacity to 10.1 Mtpa and/or for other future electrification projects (for example, the Trolley Assist).
- The consideration of the location and placement of roads, infrastructure, crusher and processing plant with respect to the sub-watersheds is appropriate and is suitably protective of the environment.

25.6 Tailing Storage Facility and Water Management

- The foundation conditions and TSF design make the construction designs and construction methodology appropriate for the Project.
- The water management structures have been designed for the modeled conditions and anticipated variability in the weather patterns with appropriate environment and operational risks considered.

25.7 Market Studies and Concentrate Marketing

- The Cu-PGM concentrate that will be produced by the operation is highly marketable and low in deleterious elements and is not expected to incur any significant penalties.
- The Cu-PGM concentrate is marketable to Cu smelters with Cu-PGM recovery capacity.
- The supply and demand conditions of the payable metals are adequately reflected (based on the current and future market conditions) with the consideration of consensus long-term metal price for the key elements.

25.8 Environmental Studies, Permitting and Social or Community Impact

- The Technical Report presents designs and operating conditions that comply with the requirements of the Federal and Provincial EA decision statement conditions issued for the Project.
- The Technical Report considers the anticipated requirements of other approvals, permits and authorizations that will be issued for the Project.

25.9 Project Execution

- The Project execution strategy that will incorporate an integrated EPCM style construction project.
- The mining fleet will be procured early in the Project and operated by the Owner's Team as part of the EPCM supervision.

25.10 Risks and Opportunities

Table 25.1 outlines the significant risks and uncertainties that could reasonably be expected to affect the reliability of confidence in the projected economic outcome for the FS update.

Table 25.2 outlines the significant opportunities that could reasonably be expected to have a positive impact on improving the Project economics in the future.

Table 25.1: Risks

Risk Category	Description	Potential Impact[1]
Mineral Resource Estimate	Until the operation commences and operational grade reconciliation is undertaken, there is some level of uncertainty related to the predictability of the Mineral Resource estimate	▪ Reduction in Mineral Resources available for conversion to Mineral Reserves
Environment Assessment Conditions and Permitting	There is uncertainty associated with the precise timing for the approval of permits required to build, and operate the Project as designed and there are EA conditions which are required to be completed prior to construction commencing	▪ A delay to the start date for project construction ▪ A delay to the start of operations or future operations
Project Financing	There is uncertainty with the Company securing timely and/or adequate Project financing	▪ Delay (short-term or long-term) in the start date of the Project
COVID-19	The resurgence, or unexpected impact of the COVID-19 pandemic is uncertain	▪ Reduced efficiency of the construction workforce or delayed construction schedule
Construction Costs	Construction costs are based on the current designs; final designs and construction methodology may change	▪ Increased construction costs
Operating Costs	Operating efficiency, operating time, productivity and consumables are assumed based on provisional budgetary quotations along with similar benchmark operations; any reduction in operating efficiency or increased consumables will increase operating costs.	▪ Increased operating costs
Processing Plant Metallurgical Recovery	The plant metallurgical recovery models are based on laboratory scale testing. Actual metallurgical recovery and mass pull of the operating plant may be different to the predicted model	▪ Less payable metal or increase in plant operating costs
Labour and Skilled Resources	There is a national and international shortage of unskilled, skilled and technical expertise in mining.	▪ Increased labour costs ▪ Increase in remote employees with an increase in camp requirements
Metal Prices and Exchange Rates	For each payable element and the exchange rate, the economic assumptions are sensitive (both positively and negatively impacted) by metal prices and changes in C$/US$ exchange rates	▪ Variability in economic results with changing metal prices. ▪ Strengthening of the C$ as compared to the US$ will negatively impact economic results

***Note*:**

[1] This is not intended to outline all potential impacts, simply the impacts that could reasonably be expected to occur in the event the risk item results in an impact.

Table 25.2: Opportunities

Opportunity	Description	Potential Impact[1]
Mineral Resource Estimate	Unrealized local variability due grade interpolation smoothing may lead to opportunities to extract somewhat more metal from fewer tonnes	▪ Higher value per tonne of ore
Plant Throughput	2022 metallurgical tests indicated variability in material hardness; the process design criteria has allowed for the higher than the average material hardness	▪ Decreased material hardness would increased the plant throughput rate and would imply increased value and cash flow
Exploration Success on the Property	With the conversion of the Property resources to reserves or new exploration success, it would be expected to increase material feed to the plant and increase either mine life beyond the 13 years or allow for increased throughput over the same operating life.	▪ Increased reserves would increase production which would imply increased value and cash flow. ▪ Increased mine life would extend employment opportunities and increase operating cash flow
Trolley Assist ("TA") or the 'next generation' powered mining fleet	The concept of TA was evaluated with equipment suppliers / dealers but was not included in the Base Case operating design. TA would conceptually increase up-ramp truck speed and allow for additional tonnage (with a reduced cycle time) or reduce capital requirements. Mining fleet manufactures are testing battery and fuel cell mining equipment with viable options being marketed within the life of mine of the operation.	▪ Improved operating efficiency and lower mine operating costs ▪ Reduction in the generation of GHG from operations (reduced diesel consumption)
Automation of the mining fleet	With the truck fleet being relatively small, autonomous haulage is not expected to be viable; however, the automation of drills and dozers would improve operating efficiency or reduce operating.	▪ Reduced operating costs on a $/t basis

***Note**: [1] This is not intended to outline all potential benefits but those that could reasonably be expected to occur or possibly realized.*

26. RECOMMENDATIONS

Following the completion of this Report, the Authors and the QPs recommend progressing work required to allow for funding and subsequently to construct the operation as defined in the document. The total cost of the next phase of the project up to commercial production is estimated at $1,112M. The following are the recommendations and initiatives as generally outlined by the QPs:

26.1 Production Decision

- With the demonstrated and positive economic analysis, progress to the next phase of project development including project financing, advancing required permits to allow for the property to be developed through construction and into production. Estimate for the cost to construct the Project as defined are included in the Report.

26.2 Environmental Permitting

- Advance on the EA conditions as outlined by the federal and provincial agencies per the positive EA decision report.
- Progress the permitting activities to allow for construction to start as soon as financing is available.
- Advance the permitting activities to allow for operations to commence following the Project construction.

26.3 Detailed Engineering Design

- Continue to progress detailed engineering for the process plant and associated site infrastructure.

26.4 Mine Methods

- Proceed to vendor selection for mobile equipment fleet.
- Determine if a mining fleet Maintenance and Repair Contract (MARC) and a maintenance agreement for the drills are viable for the construction phase and initial years of operation.
- Continue to evaluate and where financially viable, implement advanced operating technology including: electrification or battery/fuel cell propulsion of open pit mining equipment, trolley assist for the future haulage fleet, autonomous haulage fleet, autonomous or tele-remote operation of drilling fleet, autonomous dozer operation and other applications and technology that may become available.
- Evaluate fleet management system and advanced analytics for operational management.

- Pursue further optimizations of open-pit stage sequencing to maximize financial returns of the asset.

26.5 Mineral Reserves

- Evaluate local areas in the mineral reserve that are within the first 3 years of production to determine if advanced grade control may be beneficial or necessary prior to operational start-up; this grade control drilling would provide additional resolution for tonnes and grade thereby increasing the confidence in metal production in the pay-back period.

26.6 Infrastructure

- Progress the IESO study for the power line and connection requirements for the increased plant throughput and the future electrification of the operation (where financially viable options are available).
- Progress efforts to secure a long-term power contract for the operation.
- Advance designs for infrastructure facilities that are to be constructed on and off-site.
- Advance designs to allow for construction and operation of camp facilities (for construction and operations phases), an assay lab and transload facility with third-party partners for this off-site infrastructure.

26.7 Tailings Storage Facility

- Where necessary, complete geotechnical site investigations for the TSF in specific locations that have less definition to reduce construction uncertainty.
- Progress the TSF design to "*for construction*" details.
- Design and procure the appropriate water treatment plant for discharge.
- Continue with implementation of an Independent Tailing Review Panel for the oversight during the TSF life cycle including a review of forecasting and scheduling for required TSF expansions.

26.8 Concentrate Marketing

- Execute off-take agreements with smelters.
- Continue to optimize metallurgical recovery and mass pull while balancing the resulting impacts to concentrate transport costs, concentrate marketability and NSR terms.

26.9 Indigenous Affairs

- Ensure agreements with impacted Indigenous communities are implemented.
- Continue to inform and consult with Indigenous communities on applicable matters related to the Project.

26.10 Process Plant

- During the detailed engineering phase, final equipment selection and first fills will include consideration of commissioning and operational wear parts, consumables, and long lead time capital spares involving respective OEMs and Vendors.
- Continued optimization of process plant equipment layout is being considered to minimize footprint, improve operability and maintenance access, while further decreasing associated capital cost.
- Operational variability studies have considered +10 to 15% increases in throughput above a nominal 9.2 Mtpa. Continued review will identify potential process bottlenecks as part of detailed engineering.

26.11 Metallurgical Test Work

- Continue to expand and develop the GeoMet model and database as additional intercepts and composite samples become available from drilling and the onset of mining activity.
- Pursue potential improvements in cleaner circuit platinum recovery:
- Adjust the cleaner circuit slurry pH from 11.0 to 9.5 may help to improve sperrylite (platinum) recovery. It will decrease overall lime consumption while maintaining pyrite rejection to 1st cleaner scavenger tailings.
- Evaluate the potential benefit of an additional platinum specific promoter-collector. An additional collector-promoter, Aero 5100, (dialkyl thionocarbamate), at low dosage rates could be advantageous for platinum recovery in the cleaner circuit based on similar industrial trials.

26.12 Project Execution

- Advance the Project detailed engineering, including construction planning and scheduling.
- ORP is to progress in parallel with Project execution; this will require the staffing of necessary Company employees early in the Project execution phase.
- Support the involvement as appropriate of local Contractors, businesses, and community in the Project advancement and development.

- Include specific considerations to businesses associated with the BN community per the set-aside agreement and other contracts (where possible).
- Finalize the location of the extended construction camp and operations camp locations.
- Define and develop training programs for local hires as appropriate to infill suitable positions at the site.

26.13 Reducing the Carbon Impact

- Develop and implement an anti-idling policy for all vehicles and motorized equipment operating within the Designated Project area.
- Monitor fuel usage of Designated Project-related vehicles and mobile motorized equipment through fuel tracking policies.
- Consider the employment of trolley assist (electrical assistance for haul trucks), carbon dioxide capture in construction concrete and processed solids stream, utilization of low carbon fuels, and utilization of electric off-road vehicles.
- Monitor developing technologies that may be applicable and financially viable for reducing the carbon impact of the operation.

26.14 Exploration

- Explore in the area immediately west of the main deposit with the goal of discovering an underground resource that could be exploited to add high-grade material to the LOM production profile.
- Explore and further define the known resources of Geordie and Sally to determine if additional material could supplement the LOM plan production profile.
- Explore and further define the Biiwobik Prospect to the northwest of the currently envisaged open pit to determine if additional material could be added to the LOM.
- Infill drilling with the goal of defining a maiden mineral resource at the Four Dams Prospect as well as down dip exploration with the goal of discovering high-grade material that could supplement the LOM production profile.
- Explore on the Project claim blocks to define the potential for additional mineralized resources that could supplement the present LOM production profile.

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Generation Mining, 2019i: Generation Mining Initiates PEA on Marathon Palladium-Copper Project, October 15, 2019.

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Generation Mining, 2022d: Generation Mining Announces Completion of Public Hearings for the Environmental Assessment, May 19, 2022.

Generation Mining, 2022e: Generation Mining Acquires Option On Construction Camp In Marathon, July 12, 2022.

Generation Mining, 2022f: Joint Review Panel Delivers Environmental Assessment Report for the Marathon Project to Federal and Provincial Ministers, August 3, 2022.

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28. CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF QUALIFIED PERSON

EUGENE J. PURITCH, P. ENG., FEC, CET

I, Eugene J. Puritch, P. Eng., FEC, CET, residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled “Amended Feasibility Study Update Marathon Palladium & Copper Project Ontario, Canada”, (The “Technical Report”) with an effective date of December 31, 2022.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen’s University. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee’s Examination requirement for Bachelor’s Degree in Engineering Equivalency. I am a mining consultant currently licensed by the Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); and Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912). I am also a member of the National Canadian Institute of Mining and Metallurgy.

   I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

   I have practiced my profession continuously since 1978. My summarized career experience is as follows:

Mining Technologist - H.B.M.& S. and Inco Ltd.,	1978-1980
Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd.,	1981-1983
Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine,	1984-1986
Self-Employed Mining Consultant – Timmins Area,	1987-1988
Mine Designer/Resource Estimator – Dynatec/CMD/Bharti,	1989-1995
Self-Employed Mining Consultant/Resource-Reserve Estimator,	1995-2004
President – P&E Mining Consultants Inc,	2004-Present

4. I have visited the Property that is the subject of this Technical Report numerous times between 2005 and 2010.
5. I am responsible for co-authoring Sections 1, 14, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a “Qualified Person” for Technical Reports titled “Feasibility Study – Marathon Palladium and Copper Project, Northwestern Ontario, Canada”, with an effective date of March 3, 2021, “(Amended) Technical Report, Updated Mineral Resource Estimate and Preliminary Economic Assessment of The Marathon Deposit, Thunder Bay Mining District, Northwestern Ontario, Canada” with an effective date of January 6, 2020, “Technical Report, Updated Mineral Resource Estimate of the Marathon Deposit, Thunder Bay Mining District Northwestern Ontario, Canada”, with an effective date of September 9, 2019 and “Updated Technical Report and Preliminary Economic Assessment on the Marathon PGM-Cu Property Marathon Area, Thunder Bay Mining District, Northwestern Ontario, Canada”, with an effective date of April 5, 2007.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: December 31, 2022
Signed Date: May 31, 2024

{SIGNED AND SEALED}
[Eugene J. Puritch]

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Eugene J. Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON

JARITA BARRY, P.GEO.

I, Jarita Barry, P.Geo., residing at 4 Creek View Close, Mount Clear, Victoria, Australia, 3350, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Amended Feasibility Study Update Marathon Palladium & Copper Project Ontario, Canada", (The "Technical Report") with an effective date of December 31, 2022.
3. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for a total of 14 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875), Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L3874). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397);

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

Geologist, Foran Mining Corp.	2004
Geologist, Aurelian Resources Inc.	2004
Geologist, Linear Gold Corp.	2005-2006
Geologist, Búscore Consulting	2006-2007
Consulting Geologist (AusIMM)	2008-2014
Consulting Geologist, P.Geo. (EGBC/AusIMM)	2014-Present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 11, and co-authoring Sections 1, 12, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for Technical Reports titled "Feasibility Study – Marathon Palladium and Copper Project, Northwestern Ontario, Canada", with an effective date of March 3, 2021, "(Amended) Technical Report, Updated Mineral Resource Estimate and Preliminary Economic Assessment of The Marathon Deposit, Thunder Bay Mining District, Northwestern Ontario, Canada" with an effective date of January 6, 2020 and "Technical Report, Updated Mineral Resource Estimate of the Marathon Deposit, Thunder Bay Mining District Northwestern Ontario, Canada", with an effective date of September 9, 2019.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: December 31, 2022
Signed Date: May 31, 2024

{SIGNED AND SEALED}
[Jarita Barry]

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Jarita Barry, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

FRED H. BROWN, P.GEO.

I, Fred H. Brown, of PO Box 332, Lynden, WA, USA, do hereby certify that:

1. I am an independent geological consultant and have worked as a geologist continuously since my graduation from university in 1987.
2. This certificate applies to the Technical Report titled “Amended Feasibility Study Update Marathon Palladium & Copper Project Ontario, Canada”, (The “Technical Report”) with an effective date of December 31, 2022.
3. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987. I obtained a Graduate Diploma in Engineering (Mining) in 1997 from the University of the Witwatersrand and a Master of Science in Engineering (Civil) from the University of the Witwatersrand in 2005. I am registered with the South African Council for Natural Scientific Professions as a Professional Geological Scientist (registration number 400008/04), the Association of Professional Engineers and Geoscientists of British Columbia as a Professional Geoscientist (171602) and the Society for Mining, Metallurgy and Exploration as a Registered Member (#4152172).

   I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

Underground Mine Geologist, Freegold Mine, AAC	1987-1995
Mineral Resource Manager, Vaal Reefs Mine, Anglogold	1995-1997
Resident Geologist, Venetia Mine, De Beers	1997-2000
Chief Geologist, De Beers Consolidated Mines	2000-2004
Consulting Geologist	2004-2008
P&E Mining Consultants Inc. – Sr. Associate Geologist	2008-Present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 1, 14, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a “Qualified Person” for Technical Reports titled “Feasibility Study – Marathon Palladium and Copper Project, Northwestern Ontario, Canada”, with an effective date of March 3, 2021, “(Amended) Technical Report, Updated Mineral Resource Estimate and Preliminary Economic Assessment of The Marathon Deposit, Thunder Bay Mining District, Northwestern Ontario, Canada” with an effective date of January 6, 2020 and “Technical Report, Updated Mineral Resource Estimate of the Marathon Deposit, Thunder Bay Mining District Northwestern Ontario, Canada”, with an effective date of September 9, 2019.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: December 31, 2022

Signed Date: May 31, 2024

{SIGNED AND SEALED}
[Fred Brown]

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Fred H. Brown, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

DAVID BURGA, P.GEO.

I, David Burga, P. Geo., residing at 3884 Freeman Terrace, Mississauga, Ontario, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Amended Feasibility Study Update Marathon Palladium & Copper Project Ontario, Canada", (The "Technical Report") with an effective date of December 31, 2022.
3. I am a graduate of the University of Toronto with a Bachelor of Science degree in Geological Sciences (1997). I have worked as a geologist for over 20 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No 1836).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:

Exploration Geologist, Cameco Gold	1997-1998
Field Geophysicist, Quantec Geoscience	1998-1999
Geological Consultant, Andeburg Consulting Ltd.	1999-2003
Geologist, Aeon Egmond Ltd.	2003-2005
Project Manager, Jacques Whitford	2005-2008
Exploration Manager – Chile, Red Metal Resources	2008-2009
Consulting Geologist	2009-Present

4. I have visited the Property that is the subject of this Technical Report on April 4, 2012.
5. I am responsible for co-authoring Sections 1, 9, 10, 12, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for Technical Reports titled "Feasibility Study – Marathon Palladium and Copper Project, Northwestern Ontario, Canada", with an effective date of March 3, 2021, "(Amended) Technical Report, Updated Mineral Resource Estimate and Preliminary Economic Assessment of The Marathon Deposit, Thunder Bay Mining District, Northwestern Ontario, Canada" with an effective date of January 6, 2020 and "Technical Report, Updated Mineral Resource Estimate of the Marathon Deposit, Thunder Bay Mining District Northwestern Ontario, Canada", with an effective date of September 9, 2019.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: December 31, 2022
Signed Date: May 31, 2024

{SIGNED AND SEALED}
[David Burga]

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David Burga, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

WILLIAM STONE, PH.D., P.GEO.

I, William Stone, Ph.D., P.Geo, residing at 4361 Latimer Crescent, Burlington, ON, do hereby certify that:

1. I am an independent geological consultant working for P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Amended Feasibility Study Update Marathon Palladium & Copper Project Ontario, Canada", (The "Technical Report") with an effective date of December 31, 2022.
3. I am a graduate of Dalhousie University with a Bachelor of Science (Honours) degree in Geology (1983). In addition, I have a Master of Science in Geology (1985) and a Ph.D. in Geology (1988) from the University of Western Ontario. I have worked as a geologist for a total of 35 years since obtaining my M.Sc. degree. I am a geological consultant currently licensed by the Professional Geoscientists of Ontario (License No 1569).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Contract Senior Geologist, LAC Minerals Exploration Ltd. 1985-1988
   - Post-Doctoral Fellow, McMaster University 1988-1992
   - Contract Senior Geologist, Outokumpu Mines and Metals Ltd. 1993-1996
   - Senior Research Geologist, WMC Resources Ltd. 1996-2001
   - Senior Lecturer, University of Western Australia 2001-2003
   - Principal Geologist, Geoinformatics Exploration Ltd. 2003-2004
   - Vice President Exploration, Nevada Star Resources Inc. 2005-2006
   - Vice President Exploration, Goldbrook Ventures Inc. 2006-2008
   - Vice President Exploration, North American Palladium Ltd. 2008-2009
   - Vice President Exploration, Magma Metals Ltd. 2010-2011
   - President & COO, Pacific North West Capital Corp. 2011-2014
   - Consulting Geologist 2013-2017
   - Senior Project Geologist, Anglo American 2017-2019
   - Consulting Geoscientist 2020-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 4 to 8 and 23 and co-authoring Sections 1, 9, 10, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: December 31, 2022
Signed Date: May 31, 2024

{SIGNED AND SEALED}
[William Stone]

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William E. Stone, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “Amended Feasibility Study Update – Marathon Palladium & Copper Project” with an effective date of December 31st,2022 and a report date of May 31st, 2024 (the “Technical Report”).

I, **Alexandre Dorval,** ing., P.Eng., do hereby certify that:

1) I am currently employed as Open Pit Mining Engineering Coordinator with G Mining Services Inc. (“GMS”). with an office located at 5025 Boul. Lapinière, Suite 1010, Brossard, Québec, J4Z 0N5 .
2) I graduated from Laval University, Canada with a B.Sc. in Mining Engineering in 2012.
3) I am a professional engineer in good standing with the L’Ordre des Ingénieurs du Québec (“#5027189”), Professional Engineers of Ontario (“#100214598”), and Professional Engineers and Geoscientists of Newfoundland & Labrador (“#11042”) in Canada.
4) I have practiced my profession in the mining industry since graduation. I have over 8 years experience as an open pit mining engineer in diverse roles and 3 years in underground mining engineering in planning. I have relevant experience having worked on projects ranging from pre-economic assessments (O3 Mining – Marban), and feasibility studies (Quebec Iron Ore – Kamistiatusset) to project implementation related to mining engineering. I was part of the start-up team for Quarry McInnis (Ciment McInnis) in 2016. I have had no prior involvement with the property which is the subject of this Report.
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation and review of the Technical Report and I am co-author for the following sections and subsections 15, 16, 21, 22, 25, 26.
7) I have not visited the site.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have not had prior involvement with the property that is the subject of the Technical Report. I have read and understand NI 43 101 and I am considered independent of the issuer as defined in Section 1.5 of NI 43 101 Rules and Policies.

Dated this 31st day of May 2024,

/signed and sealed/

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Alexandre Dorval, ing., P. Eng.
Open Pit Mining Engineering Coordinator
G Mining Services inc.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “Amended Feasibility Study Update – Marathon Palladium & Copper Project” with an effective date of December 31st, 2022 and a report date of May 31st, 2024 (the “Technical Report”).

I, **Craig Hall**, P. Eng., do hereby certify that:

1) I am currently employed as Managing Principal with Knight Piésold Ltd. in an office located 200-1164 Devonshire Avenue, North Bay, ON P1B 6X7, Canada;
2) I graduated from the University of Waterloo, Ontario, Canada with a Bachelor of Applied Science in 2003 in Geological Engineering;
3) I am a Professional Engineer registered with Professional Engineers Ontario, (Licence: 100075047);
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 19 years, including tailings, mine waste, water management facilities and other mining related surface infrastructure.
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I have participated in the preparation and review of Sections 18, 20, 21;
7) I have visited the site property that is the subject of this report in April 2011 and March 2012;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 31st day of May 2024,

/signed and sealed/

Craig N. Hall, P.Eng.,
Managing Principal
Knight Piésold Ltd.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “Amended Feasibility Study Update – Marathon Palladium & Copper Project” with an effective date of December 31st,2022 and a report date of May 31st, 2024 (the “Technical Report”).

I, **Carl Michaud P. Eng**, do hereby certify that:

1) I am currently employed as Open Pit Mining Engineering Coordinator with G Mining Services Inc. (“GMS”). with an office located at 5025 Boul. Lapinière, Suite 1010, Brossard, Québec, J4Z 0N5
2) I have graduated from Université Laval, Canada with a B.Sc. in Mining Engineering in 1996, and from Université du Québec à Chicoutimi, Canada with an M.B.A. in 2012.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 117090).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 26 years. I have occupied different positions, both technical and operational, related to mining engineering, in Underground and Open pit operation. This experience includes Kiena and Sigma Gold mine (Placer Dome), Éléonore Mine (Goldcorp) and Mont Wright Mine (Arcelor Mittal).
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and I am author and/or co-author for sections 1, 2, 3, 19, 21, 22, 25, 26 of the Technical Report.
7) I have visited the site property that is the subject of this Technical Report from January 16 to January 17, 2023.
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I have not had prior involvement with the property that is the subject of the Technical Report. I have read and understand NI 43 101 and I am considered independent of the issuer as defined in Section 1.5 of NI 43 101 Rules and Policies.

Dated this 31st day of May 2024,

/signed and sealed/

Carl Michaud, P.Eng.,

Vice President, Mining Engineering

G Mining Services

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “Amended Feasibility Study Update – Marathon Palladium & Copper Project” with an effective date of December 31st, 2022 and a report date of May 31st, 2024 (the “Technical Report”).

I, **Benoit Bissonnette**, P. Eng., do hereby certify that:

1) I am employed as a Principal Engineer, Metallurgical Process with Wood Canada Limited, 2020 Winston Park Drive, Oakville, Ontario, Canada L6H 6X7;
2) I am a member of the Professional Engineers of Ontario. I graduated with a B.Sc. degree in Mining Engineering from Ecole Polytechnique in 1987 and with a M.Sc. degree in Mining Engineering/Mineral Processing from Laval University in Quebec City in 1989. I have practiced my profession for 34 years. I have been directly involved in the operations, management and technical services of several large mining companies and operations. I have joined Wood in 2018 and I have conducted several engineering studies, including precious metals projects.
3) 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), for those sections of the Technical Report that I am responsible for preparing;
4) I have not visited the Marathon property site;
5) I am responsible for sections 1.11, 1.12, 13, 17, 21.8.2, 25.3, 26.10 and 26.11, 27
6) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible–for preparing, contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading;
7) I have read NI 43-101 and the content in the Technical Report that I am responsible for have been prepared in compliance with NI 43-101;
8) I am independent as that term is defined in Section 1.5 of NI 43-101. I have had no previous involvement with the Marathon property.

Dated this 31st day of May 2024,

/signed and sealed/

Benoit Bissonnette, P.Eng.
Principal Engineer, Metallurgical Process
Wood Canada Limited

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “Amended Feasibility Study Update – Marathon Palladium & Copper Project” with an effective date of December 31st,2022 and a report date of May 31st, 2024 (the “Technical Report”).

I, **Jean-Francois Maille**, P.Eng., do hereby certify that:

1) I am a Project Manager for JDS Energy & Mining with an office at 900, 999 West Hastings, St. Vancouver, BC, V6C 2W2 ;
2) I am a graduate of Ecole de Technologie Superieure with a bachelor (Mechanical Engineering) in 2007;
3) I am a Professional Engineer registered with the “Ordre des Ingénieurs du Québec” (OIQ-Licence: 143426);
4) I have practiced my profession continuously since 2007 and have been involved in project and team management, construction coordination and engineering with projects located in North and South America;
5) I have read the definition of “qualified person” set out in the National Instrument 43101 (“NI 43101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43101;
6) I have participated in the preparation and review of sections: 18, 21 and 24;
7) I visited the Project from October 18th to October 20th; 2022
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43101;
10) I have read and understand NI 43101 and I am considered independent of the issuer as defined in section 1.5 of NI 43101 Rules and Policies.

Dated this 31st day of May 2024,

/signed and sealed/

Jean-Francois Maille, P.Eng.,
Project Manager
JDS Energy & Mining inc.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “Amended Feasibility Study Update – Marathon Palladium & Copper Project” with an effective date of December 31st, 2022 and a report date of May 31st, 2024 (the “Technical Report”).

I, **Sumit Rajendran Nair**, P. Eng., do hereby certify that:

1) I am employed as a Project Engineer, with Wood Canada Limited, 2020 Winston Park Drive, Oakville, Ontario, Canada L6H 6X7;
2) I am a member of the Professional Engineers of Ontario. I graduated with a B.Sc. degree in Mechanical Engineering from American University of Sharjah, UAE in 2007. I have practiced my profession for 14 years. I have worked as a project engineering management professional involved in numerous projects from scoping study to detailed engineering and construction coordination. For this project, I have overseen portions of the infrastructure design.
3) 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), for those sections of the Technical Report that I am responsible for preparing;
4) I have not visited the Marathon property site;
5) I am responsible for sections 1.22, 18.1, 18.3.1, 18.3.2, 18.3.3, 18.3.4, 18.3.5, 18.4.3, 18.4.4, 18.4.5, 18.5, 21.1, 21.5, 21.6.6 and 26,10;
6) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible–for preparing, contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading;
7) I have read NI 43-101 and the content in the Technical Report that I am responsible for have been prepared in compliance with NI 43-101;
8) I am independent as that term is defined in Section 1.5 of NI 43-101. I have had no previous involvement with the Marathon property.

Dated this 31st day of May 2024,

/signed and sealed/

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Sumit Rajendran Nair, P.Eng.
Project Engineer
Wood Canada Limited