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MARATHON PGM CORPORATION

TECHNICAL REPORT ON THE UPDATED FEASIBILITY STUDY FOR THE MARATHON PGM-Cu PROJECT, MARATHON, ONTARIO, CANADA

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Charley Murahwi, P.Geo., Micon International Limited
Sam Shoemaker, MAusIMM, Micon International Limited
Richard Gowans, P.Eng., Micon International Limited
John Lemieux, ing., AMEC Earth & Environmental
Christopher Jacobs, C.Eng., MIMMM, Micon International Limited

SUITE 900 - 390 BAY STREET, TORONTO ONTARIO, CANADA M5H 2Y2
Telephone (1) (416) 362-5135 Fax (1) (416) 362 5763

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IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Marathon PGM Corporation (Marathon PGM) by Micon International Limited (Micon). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in Micon's, services, and based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended to be used by Marathon PGM, subject to the terms and conditions of its contracts with Micon. This contract permits Marathon to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Any other use of this report by any third party is at that party's sole risk.

1.0 EXECUTIVE SUMMARY

1.1 INTRODUCTION

Micon International Limited (Micon) has been retained by Marathon PGM Corporation (Marathon PGM) to update a Feasibility Study on the Marathon PGM-Cu property near Marathon, Ontario. The previous Feasibility Study was completed in December, 2008.

As part of this update of the study Micon prepared an updated mineral resource estimate, a new open pit mine design and new mine schedule, and a new mineral reserve estimate. Met-Chem Canada Inc. (Met-Chem) was retained through, and under the supervision of, Micon to update the process plant design and process and infrastructure capital and operating cost estimates. AMEC Earth & Environmental (AMEC) and EcoMetrix Incorporated (EcoMetrix) were retained directly by Marathon PGM to review process solids disposal methods and costs, and environmental and permitting issues, respectively. Additional metallurgical testwork was undertaken by Xstrata Process Support (XPS) under the supervision of Micon. Andrew Falls of Exen Consulting Services (Exen) was retained directly by Marathon PGM to provide an updated analysis of the markets for the metals to be produced from the Marathon PGM-Cu deposit.

This Technical Report presents the updated mineral resource and reserve estimates and discusses the results of the updated Feasibility Study for the Marathon PGM-Cu deposit.

The effective date of the updated Feasibility Study is 24 November, 2009.

The Qualified Persons responsible for this report are the following:

Charley Murahwi, P.Geo., Micon International Limited
Sam Shoemaker, MAusIMM, Micon International Limited
Richard Gowans, P.Eng., Micon International Limited
John Lemieux, ing., AMEC Earth & Environmental
Christopher Jacobs, C.Eng., MIMMM, Micon International Limited

The Marathon PGM-Cu project is located approximately 10 km north of the Town of Marathon, Ontario. The Hemlo Mining Camp is located 30 km to the southeast. The population of Marathon is approximately 5,000, and the town is situated adjacent to the Trans-Canada Highway No. 17 on the northeast shore of Lake Superior. The centre of the property sits at approximately 48° 45’ N latitude, 86° 19’ W longitude. Marathon is approximately 300 km east of Thunder Bay by highway and 400 km northwest of Sault Ste-Marie by highway. Primary industries supporting the Town of Marathon are pulp-and-paper and mining.

The climate is typical of northern areas within the Canadian Shield with long winters and short, warm summers. The Marathon PGM-Cu property is located in an area characterized by moderate to steep hilly terrain with a series of creeks and lakes and dense vegetation. The project area is bounded to the east by the Pic River and Lake Superior to the south and west.

1.2 GEOLOGY AND MINERALIZATION

The Marathon PGM-Cu deposit is hosted within the Eastern Gabbro Series of the Proterozoic Coldwell Complex which intrudes and bisects the much older Archean Schreiber-Hemlo Greenstone Belt. The sub-circular 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. The Coldwell Complex was emplaced as three nested intrusive centres that were active during cauldron subsidence near where the northern end of the Thiel Fault intersected Archean rocks, on the north shore of Lake Superior.

Mineralization at the Marathon PGM-Cu deposit is part of a very large magmatic system that consists of at least three cross-cutting intrusive olivine gabbro units that comprise the Eastern Gabbro Series of the Coldwell Complex. In order of intrusion, the three gabbroic units consist of Layered Gabbro Series, Layered Magnetite Olivine Cumulate (LMOC) and Two Duck Gabbro (TD Gabbro). The relative size and abundance of the gabbroic units decrease in the order Layered Gabbro Series>TD Gabbro>>LMOC. Late quartz syenite and augite syenite dikes cut all of the gabbros but form a minor component of the intrusive assemblage. The TD Gabbro is the dominant host rock for copper-PGM mineralization and is the focus of exploration. 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. Additional accumulations of copper-PGM mineralization are associated with LMOC and occur in the hanging wall of the deposit.

1.3 EXPLORATION, SAMPLING AND ASSAYING

Since acquiring the Marathon PGM-Cu deposit from Polymet Mining Corp. (Polymet) in December, 2003 Marathon PGM has funded continuous programs of advanced exploration and diamond drilling commencing with its surface exploration program in June, 2004.

A total of 705 drill holes totaling 130,560 m of drill core were used to delineate the mineral resource estimate described in this Technical Report. In 2007, 36,779 m were drilled including 176 holes drilled for a total of 35,057 m as infill and step out holes within the Marathon PGM-Cu deposit, and 1,722 m drilled in 13 holes outside the pit area. A total of 19,538 m in 92 holes were drilled in 2008 as infill and step out holes within the Main Zone. An additional 842 m in five holes were drilled for exploration outside the pit area, and four holes for a total of 858 m were drilled as condemnation holes at the process solids management facility (PSMF), crusher and mill sites. A total of 2,334 m in 21 holes were drilled in 2009 as step out holes and were primarily intended to expand the resource. Drilling in 2008 on the Benton JV portion of the project area included 23 holes for a total of 6,862 m.

For the 2007, 2008 and 2009 drilling programs, the NQ core holes were sawn in half and sampled on regular 2-m intervals through the mineralized zone. Samples were delivered to Accurassay Laboratories (Accurassay) in Thunder Bay, Ontario. All samples were analyzed for Cu, Ni, Au, Pt and Pd. Rhodium analysis was requested on samples within an intersection of two or more consecutive samples with an NSR value greater than $8, as well as the two samples on either side of the intersection, even though the values are likely to be below detection limit.

Independent verification sampling was carried out by Charley Murahwi who made an independent selection of sample pulps in October, 2009. Independent repeat analyses on the pulps showed a good degree of reproducibility by the Accurassay laboratory.

A QA/QC program, initially instituted in 2006, was maintained throughout 2007, 2008 and 2009. Uncertified property standards named APG1 and APP7, as well as the Canmet certified standard WMG-1, were used as reference materials. In mid-2007, the supply of APP7 was exhausted and was replaced by another property standard, APG6. The QC program was monitored on a real-time basis by Marathon PGM throughout 2007, 2008 and 2009.

1.4 MINERAL PROCESSING AND METALLURGICAL TESTWORK

The updated feasibility metallurgical flowsheet and process design criteria are based on a program of flotation circuit optimization testing, including a mini pilot plant (MPP) run in April, 2009, at XPS, Sudbury, Ontario and a detailed program of metallurgical testwork undertaken by SGS Lakefield Research (SGS-L) at Lakefield between March, 2007 and March, 2008. This work is complemented by a substantial amount of historical work ranging from the 1960s.

Other testwork completed for the feasibility includes two pilot scale programs to test the suitability and gather scale-up data for high pressure grinding roll technology. This work was undertaken at the testing facilities of KHD Humboldt Wedag GmbH (KHD) located near Cologne, Germany.

A number of general and PGM specific mineralogical investigations have been conducted on samples of Marathon PGM-Cu mineralization.

1.5 MINERAL RESOURCE ESTIMATES

The revised mineral resource estimate for the Marathon PGM-Cu property was undertaken by Sam Shoemaker, MAusIMM, and Charley Murahwi, P.Geo., of Micon with the assistance of David Good, Ph.D., P.Geo., V.P. Exploration of Marathon PGM.

A review of the basis for the previous mineral resource estimate (geologic cross-sections) was completed by Micon using both the previously used drill holes along with the additional 21 new drill holes. The new in-fill drilling indicated minor changes from the previously interpreted geologic model which required that an updated cross-sectional interpretation be completed before a new mineral resource estimate could be established. In order to better represent the geology of the Marathon PGM-Cu deposit, a new block model was constructed which used an unfolding technique on the sample search ellipsoid. This approach allowed a search ellipsoid to better reflect the actual trend of the mineralization. In addition, smaller block sizes were used in the mineralized zones to further help delineate the overall potential resource.

The diluted block model was exported to Whittle where the model was prepared for optimization. A number of pit optimization runs were completed at along with extensive sensitivity analysis. Table 1.1 shows the estimated pit shell mineral resource contained within the selected optimized pit shell.

Table 1.1
Marathon PGM-Cu Pit Shell Mineral Resource (Diluted Block Model)

Total Resource (Lower and Higher Grade) above $10.50/t NSR Cut-off

Category	Pit Shell 46 Mineral Resource						Contained Metal				
	Tonnes millions	**Pd (g/t)**	**Pt (g/t)**	**Au (g/t)**	**Cu (%)**	**Ag (g/t)**	**Pd (oz 000)**	**Pt (oz 000)**	**Au (oz 000)**	**Cu (lb million)**	**Ag (oz 000)**
Measured	94.3	0.846	0.243	0.088	0.262	1.599	2,564	736	266	545	4,847
Indicated	20.5	0.451	0.160	0.062	0.140	1.421	386	133	50	73	976
Measured + Indicated	**114.8**	**0.775**	**0.228**	**0.083**	**0.241**	**1.567**	**2,950**	**869**	**316**	**618**	**5,823**
Inferred	6.2	0.306	0.104	0.047	0.151	1.459	61	21	9	21	290

1. The mineral resources presented above are the subject of the Feasibility Study discussed in the present Technical Report.
2. The quantity and grade of reported inferred resources in this estimate are conceptual in nature and there has been insufficient exploration to define them as indicated mineral resources. It is uncertain if further exploration will result in their conversion to indicated or measured mineral resources.

Following the completion of 21 additional exploration drill holes in September, 2009, the block model was updated. The mineral resource estimate provided in Table 1.1 is effective as of 24 November, 2009.

1.6 MINING AND MINERAL RESERVE ESTIMATES

The Marathon PGM-Cu project comprises open pit mining and processing at an average rate of 22,000 t/d of ore to produce a saleable flotation concentrate containing Cu, Pd, Pt, Au, Ag and Rh. The life of the operation is estimated at approximately 11.5 years.

The proposed Marathon PGM-Cu open pit will be a conventional open pit mining operation that will be developed by the Owner using its own equipment and workforce.

The mineral resource model used for the pit optimization, pit design, and production scheduling is the diluted block model developed by Micon in 2009 and used to estimate the mineral resources. Only material in the block model with the resource classification of 'measured' or 'indicated' were considered as potential mill feed. In addition to the estimated grade values for Cu, Pd, Pt, Au, Ag, and Rh contained within the diluted block model, other variables were calculated or input into the diluted block model. These included the net smelter return, geotechnical parameters, block economic net value, haulage simulation results, block material type, and Whittle rock types.

In order to complete an open pit design on the Marathon PGM-Cu deposit, Micon used:

- The available geotechnical data describing the inter ramp slope angle, slope sectors, and berm widths that are required to develop a geotechnically stable pit design.

- Economic and metallurgical criteria such as estimated metal pricing, metal recoveries, downstream operating costs (smelting, refining, and shipping), currency conversion rates, and projected annual mill feed requirements.

- Pit optimization based on the economic, metallurgical, geotechnical and production requirements for the project.

Pit optimization was completed using a Lerchs-Grossmann algorithm (LG) on the block model. GEMCOM's LG software, the Whittle optimizer was selected. Once a pit optimization was completed, the selected pit shell was used as a design basis for the open pit. For the Marathon PGM-Cu deposit, three major mining areas are present, the North pit, South pit, and Malachite pit. Once these three pit areas were designed, a production schedule was prepared, followed by equipment selection and estimation of operating costs, capital costs and personnel requirements.

Mineral reserves have been estimated for the North, South and Malachite pits from the diluted block model, pit optimization and pit design. The mineral reserves are summarized in Table 1.2.

Table 1.2
Mineral Reserves for the Marathon PGM-Cu Deposit

Classification	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	Cu (Mlb)	Pd (oz 000)	Pt (oz 000)	Au (oz 000)	Ag (oz 000)
Proven	76,461,000	0.910	0.254	0.090	0.268	1.464	452	2,237	625	222	3,600
Probable	14,986,000	0.435	0.147	0.060	0.138	1.318	46	209	71	29	635
Total	91,447,000	0.832	0.237	0.085	0.247	1.440	497	2,447	696	251	4,235

1. The mineral reserves presented in Table 1.2 are included in the mineral resources presented in Table 1.1.

The mineral reserve estimate presented in Table 1.2 is effective as of 24 November, 2009.

The scheduled life of mine tonnes of ore and mine (waste) rock are 91.4 Mt and 263.5 Mt, respectively. The average ore to mine rock ratio is 2.88.

Two mine rock (waste) storage areas (MRSA) are envisioned for the Marathon PGM-Cu project. The first is the west mine rock storage area (MRSA) with a capacity of 151.3 Mm^3 or 227 Mt. Total surface area impacted by this MRSA is 270 ha. The second storage area is the east MRSA area with a capacity of 40.7 Mm^3 or 61 Mt. Total surface area impacted by this facility is 106 ha. Total mine rock storage capacity is 192.0 Mm^3 or 288 Mt.

1.7 MINERAL PROCESSING

The design of the 22,000 t/d concentrator comprises primary crushing, secondary crushing, high pressure grinding rolls (HGPR), ball milling, flotation, concentrate dewatering and process solids (tailings) disposal. The concentrator is designed to produce a copper sulphide flotation concentrate containing PGMs and gold.

Mined ore-grade material is hauled by mine trucks to the primary crusher situated on the eastern side of the main open pit. Primary crushed ore is conveyed onto a coarse ore stockpile from which it is reclaimed to the secondary crushing and screening plant. Product from the secondary crushing plant is fed to the HPGR feed storage bins situated at the main plant facility. Material from the HPGR product storage bins feeds the grinding circuit located in the process plant. Ground material feeds the flotation circuit.

The flotation circuit comprises two conditioners, a primary rougher stage, a primary cleaner stage, a secondary rougher stage, a secondary cleaner stage and a cleaner scavenger stage. The primary cleaning circuit comprises one stages of cleaning and two stages of secondary cleaning. The flotation circuit is based on the metallurgical flowsheet developed by SGS-L and XPS.

The final concentrate is thickened, filtered in a continuous vertical plate type pressure filter and stored in a stockpile located on the ground floor of the mill building. The concentrate is periodically loaded into trucks and transported to the concentrate storage and rail load-out area, which is situated in Marathon

1.8 INFRASTRUCTURE

The access road to the site will be routed in a northeast direction from the extension of Peninsula road branching north from the Trans Canada Highway No. 17 at the Marathon Town intersection.

Infrastructure facilities for the operation comprise:

- Site roads.
- Construction camp.
- Plant buildings and facilities.

- Mine equipment and maintenance building.
- Site water systems and potable water treatment.
- Heating, ventilation and air conditioning.
- Fuel storage and delivery systems.
- Fire protection equipment.
- Plant mobile equipment.
- Mine rock disposal.
- Explosives plant and storage.
- Concentrate load-out facility.
- Process solids thickening plant.
- Electrical power supply and distribution.
- Automation and control systems.
- Communications.

1.9 DISPOSAL OF PROCESS SOLIDS

AMEC of Pointe Claire, Quebec, Canada, was retained by Marathon PGM in 2009 to carry out a new study for the disposal of process solids (tailings). AMEC's report presented three options based on criteria related to: the production objectives proposed by Marathon PGM; process data obtained by other consultants; the sulphur content of the process solids; the available meteorological data for the region; and the environmental criteria in effect. AMEC's conclusions were based on basic design elements and criteria, preliminary analysis of potential sites, water assessments, evaluation of typical sections of dykes and dams, fill plans, material borrow areas and capital costs estimates.

The three options designed and costed by AMEC were:

1. Base case – Sub-aqueous storage of process solids in Bamoos Lake.

2. Option 1A – Land-based separated low and high sulphur process solids management facility with excess treated water discharge to the environment through the operational/emergency spillway of the high sulphur PSMA into Stream 6.

3. Option 1B - Modified version of Option 1A to release water to the environment directly to Hare Lake via Hare Creek.

1.10 ENVIRONMENTAL ISSUES

Environmental baseline studies have been ongoing since 2005. In 2009, Marathon retained EcoMetrix and True Grit Consulting Ltd., (True Grit) to provide the environmental research relevant through 2009 and into 2010 and beyond. The overarching objective of this research is to provide the necessary information to develop an EIA and ultimately deliver the EIS for the Marathon PGM-Cu project to the government. The detailed results from these field studies will form part of the EIS.

1.11 PROJECT SCHEDULE

A list of the key project development milestones is provided below:

• Complete updated Feasibility Study	November, 2009
• Project Description for EA issued	December, 2009
• EA Report issued to authorities	April, 2010
• Process optimization and basic engineering start	May, 2010
• Detailed engineering start	July, 2010
• Long lead equipment purchased	September, 2010
• Process optimization and basic engineering complete	February, 2011
• Mobilization on site	February, 2011
• Environmental assessment approved and all permits granted	December, 2011
• Detailed engineering complete	December, 2011
• Ball mill delivery to site	February, 2012
• Construction complete	January, 2013
• Wet commissioning start	January, 2013
• Production start-up completed	May, 2013

Assuming that basic construction access is granted prior to the final approval of the EA the estimated production start-up date is May, 2013. If access is only acceptable after all permits are in place and the EA has been approved, which is the scenario currently assumed in the Project Description document, then the estimated start-up date is December, 2013.

1.12 CONCENTRATE MARKETING

Andrew Falls of Exen was retained by Marathon PGM to prepare an updated analysis of the market for concentrate to be produced from the Marathon PGM-Cu project. The concentrate is considered a copper concentrate from a marketing perspective, notwithstanding the relatively high PGM content. In this respect, the concentrate is relatively unusual but the copper content, at about 22% Cu, is low compared to the majority of copper concentrates, and will have to be blended in order to meet the requirements of almost all smelters.

Mr. Falls' analysis has resulted in the identification of a small number of potential buyers which are able to handle copper-PGM materials in their smelting/refining facilities and which, because of the high grade of precious metals, may be anticipated to provide reasonable credit for precious metals in the Marathon PGM-Cu concentrate.

1.13 CAPITAL AND OPERATING COST ESTIMATES

The estimated pre-production project capital costs are summarized in Table 1.3.

Table 1.3
Summary of Estimated Pre-Production Project Capital Costs

Area	Cost ($ thousand)
Mining pre strip	5,762
Mine equipment[1]	18,536
Process plant and infrastructure	261,695
PSMF and water treatment	8,396
Owners Costs	7,202
Contingency	49,531
Pre-production total	**351,122**

[1]Assumes a 10% down payment on the cost of mining equipment and financing of the balance over 5 years at 9%/y interest rate.

The life-of-mine capital cost estimate is $495 million comprising $351 million of pre-production capital and $144 million of sustaining and closure capital. The sustaining capital consists of mainly $103 million for mining, which includes a credit for mine equipment salvage.

The total average life-of-mine unit operating costs are presented in Table 1.4.

Table 1.4
Estimated LOM Unit Operating Cost

Component Cost	$/t milled
Mining	5.67
Processing	6.79
Water treatment	0.05
General and administration - site	0.58
General and administration – mine equipment financing	0.29
Total on-site cost	**13.39**
Concentrate transportation, smelting and refining	3.25
Total operating cost	**16.64**

1.14 ECONOMIC ANALYSIS

The overall level of accuracy of the cost estimates in the Feasibility Study is ±15%.

Micon has prepared its assessment of the project on the basis of a discounted cash flow model, from which net present value (NPV), internal rate of return (IRR), payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to evaluate the potential for establishing a viable open pit mine and concentrator to exploit the Marathon PGM-Cu deposit. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

For the purposes of the Feasibility Study evaluation, the three-year trailing average prices were selected to provide a base case against which each of the other scenarios could be compared (see Table 1.5). As part of its sensitivity analysis, Micon also tested a range of prices 30% above and below these base case values.

Table 1.5
Metal Price Forecasts
(LOM Averages)

Item	Units	3-y trailing	Bank forecast	5-y trailing
Copper	US$/lb	2.91	2.03[1]	2.63
Platinum	US$/oz	1,346.65	1,750.00	1205.73
Palladium	US$/oz	321.44	400.00	293.23
Gold	US$/oz	819.22	900.00	695.11
Silver	US$/oz	14.10	13.00	12.04
Exchange rate	$/US$	1.099	1.10	1.131

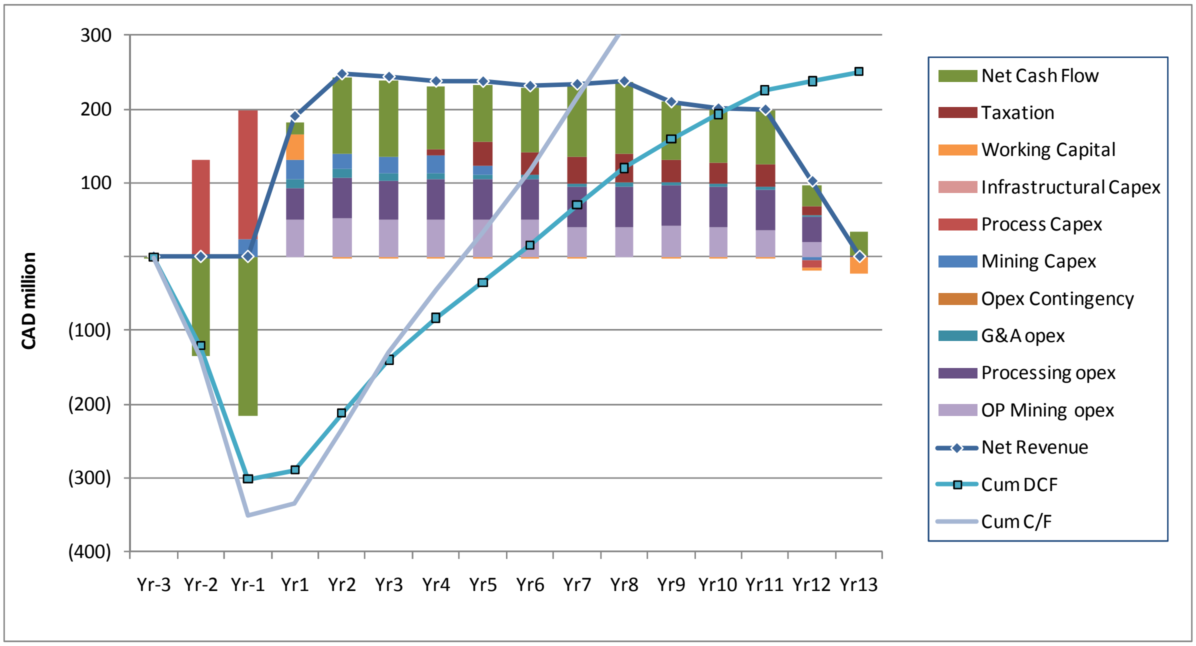
[1] US$2.50/lb Cu in 2013 (Yr 1), US$2.00/lb Cu long term.

Using the parameters outlined in the undated feasibility study, a cash flow and net present value projection was prepared for the base case. This projection is summarised in Table 1.6 and Figure 1.1, based on a discount rate of 6%/y (NPV_6).

Table 1.6
Cash Flow Projection

	LOM Total ($ thousand)	$/t treated	US$/lb Cu	NPV_6 ($ thousand)
NSR copper only	1,222,847	13.37	2.58	723,170
NSR co-products	1,347,385	14.73	2.84	812,451
less Royalty	4,928	0.05	0.01	3,715
Sub-total net revenue	2,565,304	28.05	5.41	1,531,906
Operating costs				
Mining costs - open pit	518,591	5.67	1.09	314,610
Processing costs	625,962	6.85	1.32	368,129
General & Administrative costs	79,524	0.87	0.17	50,683
Contingency	-	-	-	-
Total cash operating cost	1,224,078	13.39	2.58	733,422
Net operating margin	1,341,226	14.67	2.83	798,484
Capital expenditure	494,645	5.41	1.04	415,104
Pre-tax cash flow	846,581	9.26	1.79	383,380
Taxation	249,768	2.73	0.53	132,663
Net cash flow after tax	596,813	6.53	1.26	250,718

Figure 1.1
LOM Cash Flow Projection



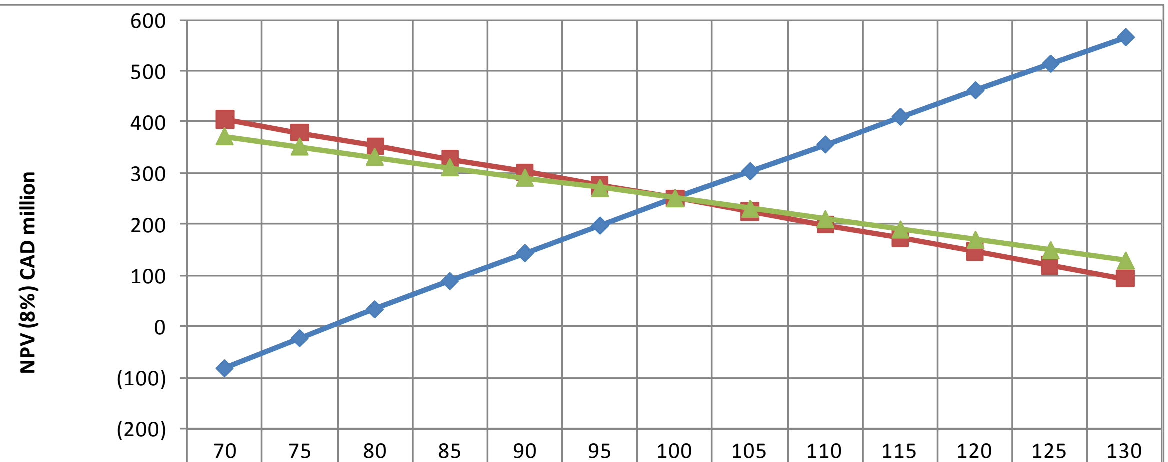
The results show that the project generates an IRR of 21.2% before tax and 17.4% after tax. The undiscounted payback period is 4.4 years, and the discounted cash flow is positive after six years. The NPV_6 is $250.7 million after tax.

The results of sensitivity analysis of product price and capital and operating costs are shown in Figure 1.2.

Given the sensitivity to price assumptions, and the volatility in metal prices observed in the market, Micon tested the cash flow using several other price scenarios.

It is apparent that the project provides an attractive return when using the base case '3-year trailing' average prices obtaining during the 36 months to October 31, 2009. Similar returns are seen when using the independent forecast of a leading Canadian commercial bank, published in October, 2009. Returns using the 5-year trailing average are also positive, but less attractive.

Figure 1.2
NPV Sensitivity Diagram



	70	75	80	85	90	95	100	105	110	115	120	125	130
Product Price	(81)	(23)	34	89	143	197	251	303	356	409	461	513	565
Opcosts	405	379	353	327	302	276	251	224	199	173	146	120	93
Capex	372	351	331	311	291	271	251	231	210	190	170	150	130

The base case cash flow considers the leasing of the mining fleet. Micon also considered the outright cash purchase of the equipment as an alternative strategy. The NPV_6 for the all-equity fleet purchase option is \$261 million, an increase of \$10 million versus the leasing scenario, though, at the same time, the project IRR falls from 17.4% to 17.0%.

The base case cash flow provides for the sub-aqueous deposition of process solids within Bamoos Lake. In this case, no thickening of the process slurry before pumping to storage is required, and minimal capital costs are associated with establishing the impoundment.

Using the alternative process solids disposal option, Option A1, (sub-aerial deposition of tailings) the impact of this on project economics is a reduction in NPV from \$251 million to \$211 million, and a reduction in project IRR from 17.4% to 15.3%.

1.15 CONCLUSIONS AND RECOMMENDATIONS

The updated Feasibility Study completed on the Marathon PGM-Cu project demonstrates the potential to generate strong cash flow under appropriate metal price assumptions. The base case results show that the project generates an IRR of 21.2% before tax and 17.4% after tax. The undiscounted payback period is 4.4 years, and the discounted cash flow is positive after 6 years. The NPV_6 is \$250.7 million after tax. The sensitivity studies demonstrate that the project is quite sensitive to adverse changes in price assumptions and moderately sensitive to changes in operating cost or capital expenditure.

The project schedule suggests that production of copper/PGM/Au concentrate could commence at the end of 2013. The present critical path item is the environmental assessment approval process and associated receipt of the required construction and operating permits.

The immediate efforts of Marathon PGM will be concentrated on securing the required funding to proceed with the development of the deposit. Throughout the process, the company will undoubtedly be restructuring toward a producing mining company, with exploration geared toward reserve and resource sustainability.

1.15.1 Project Development

As a result of its Feasibility Study on the Marathon PGM-Cu Project, Micon recommends that Marathon PGM proceeds with the development of the project.

The life-of-mine capital cost for the Marathon PGM-Cu project is $495 million, including estimated initial capital costs of $351 million, as noted above. The estimated annual expenditures over the first three years of project development (Years -3 through -1) are detailed in Table 18.27, which provides the base case annual cash flows for the project.

The metallurgical testwork programs completed to date were used to design the process used in the updated Feasibility Study. This work includes a pilot plant run in 1986, bench scale tests including locked cycle tests (LCT) at SGS-L in 2004, 2007 and 2008, and LCT and a 6-day continuous mini pilot plant run completed by XPS in 2008 and 2009. Although Micon believes that the metallurgical testwork completed to date on the Marathon PGM-Cu deposit provides ample proof that good metallurgical performance can be achieved using conventional flotation, it is suggested that additional work may be worthwhile in order to try and reduce the reagent costs. This could entail reducing reagent dosage rates or substituting the existing reagent suite with less expensive chemicals.

Three feasible process solids (tailings) management areas (PSMA) for the Marathon PGM-Cu project were evaluated by AMEC. AMEC concluded that the sub-aquatic option (Bamoos Lake) seems to be the best PSMA since capital investment will be the lowest, no separation process between high/low sulphur process solids will be required and the risks associated to this option are low. However, this option utilizes an existing lake for containment which may be difficult to permit. AMEC commented that Option 1A represented the best on-land PSMA and should continue as an alternative during the advanced development and permitting process. AMEC further recommends the following:

- Detailed operational water management will need to be evaluated to take into account the detailed mining schedule.

- An extended geotechnical investigation is required for detailed design of the PSMA infrastructure. Furthermore, detailed evaluation of available clay deposits is required to determine dam design and cost.

The Marathon PGM-Cu project will likely be subject to both federal and provincial Environmental Assessment processes, and Marathon PGM intends to work in a coordinated way with both governments in order to drive the process forward with regard to achieving the necessary approvals in a timely manner.

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 TERMS OF REFERENCE

Micon International Limited (Micon) has been retained by Marathon PGM Corporation (Marathon PGM) to update a Feasibility Study on the Marathon PGM-Cu property near Marathon, Ontario. The previous Feasibility Study was completed in December, 2008.

As part of this update of the study, Micon prepared an updated mineral resource estimate, a new open pit mine design and new mine schedule, and a new mineral reserve estimate. Met-Chem Canada Inc. (Met-Chem) was retained through and under the supervision of Micon to update the process plant design and process and infrastructure capital and operating cost estimates.

AMEC Earth & Environmental (AMEC), EcoMetrix Incorporated (EcoMetrix) and True Grit Consulting Ltd. (True Grit) were retained directly by Marathon to review process solids disposal methods and costs, and environmental and permitting issues, respectively. Additional metallurgical testwork was undertaken by Xstrata Process Support (XPS) under the supervision of Micon.

Andrew Falls of Exen Consulting Services (Exen) was retained directly by Marathon PGM to provide an updated analysis of the markets for the metals to be produced from the Marathon PGM-Cu deposit.

Table 2.1
List of Feasibility Study Participants

Updated mineral resource estimate	Micon International Limited.
Revised mine design, schedule, mine equipment selection, mine facilities, mining cost estimates	Micon International Limited.
Metallurgical testing and flowsheet development	SGS Lakefield Research Xstrata Process Research Micon International Limited
Process engineering	Met-Chem Canada Inc.
Process solids management system	AMEC Earth & Environmental
Infrastructure and plant design, capital expenditures and operating costs	Met-Chem Canada Inc.
Review of environmental baseline studies and geochemical characterization studies	EcoMetrix Incorporated True Grit Consulting Ltd.
Economic evaluation	Micon International Limited
Overall study management	Micon International Limited
Market analysis	Exen Consulting Services

The effective date of the Feasibility Study is 24 November, 2009.

The results of the updated Feasibility Study for the Marathon PGM-Cu deposit are discussed in this Technical Report. The Technical Report also presents the updated mineral resource estimate.

The Qualified Persons responsible for this report are the following:

Charley Murahwi, P.Geo., Micon International Limited
Sam Shoemaker, MAusIMM, Micon International Limited
Richard Gowans, P.Eng., Micon International Limited
John Lemieux, ing., AMEC Earth & Environmental
Christopher Jacobs, C.Eng., MIMMM, Micon International Limited

Site visits were conducted by the Qualified Persons as follows:

Richard Gowans, P.Eng.	7 August, 2007
Sam Shoemaker, MAusIMM	24 August, 2009
Charley Murahwi, P.Geo.	16 and 17 August, 2009

The present Technical Report is prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101) of the Ontario Securities Commission (OSC) and the Canadian Securities Administrators (CSA). The resource estimate is prepared in compliance with the CIM Definitions and Standards on Mineral Resources and Mineral Reserves, as adopted by the CIM council on December 11, 2005.

Previous Technical Reports issued on the Marathon PGM-Cu deposit include the following:

- 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.

- 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.

- 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.

2.2 UNITS AND CURRENCY

In this report, all currency amounts are stated in Canadian dollars ($), with commodity prices typically expressed in US dollars (US$). Quantities are generally stated in Système International d'Unités (SI) metric units, the standard Canadian and international practice, including metric tons (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, grams (g) and grams per tonne (g/t) for gold grades (g/t Au). Platinum group metal (PGM) and gold grades may also be reported in parts per million (ppm) or parts per billion (ppb). Quantities of PGM and gold may also be reported in troy ounces (oz) and quantities of copper in avoirdupois pounds (lb). Copper metal assays are reported in percent (%) while gold and PGM assay values are reported in grams of metal per tonne (g/t) unless ounces per short ton (oz/T) are specifically stated.

Table 2.2
List of Abbreviations

Term	Abbreviation
Acceleration due to gravity	*g*
Acid base accounting	ABA
Acid rock drainage	ARD
Aluminum	Al
Alternating current	AC
Ampere(s)	A
Atomic absorption spectrometry	AAS
Billion years old	Ga
Canadian dollar	$
Canadian National Instrument 43-101	NI 43-101
Centimetres per second	cm/s
Chromium	Cr
Carboxymethylcellulose	CMC
Cobalt	Co
Cubic metre(s)	m^3
Cubic metres per day	m^3/d
Cubic metres per minute	m^3/min
Cubic metres per second	m^3/s
Cubic metres per second per metre	$m^3/s/m$
Cubic metres per year	m^3/y
Degree(s)	^{o}
Degrees Celsius	^oC
Environmental assessment	EA
Environmental impact assessment	EIA
Environmental impact statement	EIS
Factor of safety	FS
Foot(feet)	ft
Gallons per minute	gpm
Gram(s)	g
Grams per cubic centimetre	g/cm^3
Grams per litre	g/L
Gold	Au
High density polyethylene	HDPE
High pressure grinding roll	HPGR
Horsepower	HP
Hour(s)	h
Hour(s) per day	h/d
Inch(es)	in
Internal rate of return	IRR
Inverse distance to the power of 2	ID^2
Inverse distance to the power of 5	ID^5
Iridium	Ir
Iron	Fe

Term	Abbreviation
Kilogram(s)	kg
Kilograms per cubic metre	kg/m^3
Kilograms per day	kg/d
Kilometre(s)	km
Kilopascal(s)	kPa
Kilovolt(s)	kV
Kilovolt ampere	kVA
Kilowatt(s)	kW
Kilowatthours per tonne	kWh/t
Lead	Pb
Litre(s)	L
Lerchs-Grossmann	LG
Life-of-mine	LOM
Litres per second	L/s
London Metal Exchange	LME
Loss on ignition	LOI
Low voltage	LV
Megavolt ampere	MVA
Megawatt(s)	MW
Metre(s)	m
Metres above sea level	masl
Metres per second	m/s
Micron(s)	µm
Milliampere(s)	mA
Milligrams	mg
Milligrams per litre	mg/L
Millimetre(s)	mm
Millimetres per year	mm/y
Million	M
Million pounds	Mlb
Million cubic metres	Mm^3
Million tonnes	Mt
Million tonnes per year	Mt/y
Million years old	Ma
Mine rock storage area(s)	MRSA
Minute(s)	min
Molybdenum	Mo
Motor control centre	MCC
Net acid generating	NAG
Net present value	NPV
Net present value at 6%/y discount rate	NPV_6
Neutralization potential	NP
Neutralization potential ratio	NPR
Nickel	Ni
Newtons per square millimetre	Nmm^2
Osmium	Os
Ounce(s) (troy ounce)	oz
Ounces per tonne	oz/t
Ounces per short ton	oz/T
Palladium	Pd
Parts per billion	ppb

Term	Abbreviation
Parts per million	ppm
Platinum	Pt
Platinum group metals	PGM
Potentially acid generating	PAG
Potassium amyl xanthate	PAX
Pound(s)	lb
Process solids management facility	PSMF
Provincial water quality objectives	PWQO
Quality assurance	QA
Quality assurance/quality control	QA/QC
Quality control	QC
Rhodium	Rh
Rock quality designation	RQD
Programmable logic controller	PLC
Ruthenium	Ru
Second	s
Short ton (2,000 pounds)	T
Specific gravity	SG
Square metre(s)	m^2
Square metres per tonne	m^2/t
Square kilometre(s)	km^2
Standard deviation	Std Dev
Sulphur	S
Supervisory control and data acquisition	SCADA
Thousand tonnes	kt
Three dimensional	3D
Tonne(s)	t
Tonnes per cubic metre	t/m^3
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/y
Tonne-seconds per hour-cubic metre	ts/hm^3
Treatment charges/refining charges	TC/RC
United States dollars	US$
Vanadium	V
Volt(s)	V
Weight	Wt.
Year	Yr or yr
Zinc	Zn

3.0 RELIANCE ON OTHER EXPERTS

The authors wish to make clear that they are qualified persons only in respect of the areas in this report identified in their "Certificates of Qualified Persons" submitted with this report to the Canadian Securities Administrators.

Although copies of the licenses, permits and work contracts were reviewed, an independent verification of land title and tenure was not performed. Micon has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties.

Andrew Falls of Exen Consulting Services was retained directly by Marathon PGM to provide an analysis of the markets for the metals to be produced from the Marathon PGM-Cu deposit. His report is summarized for the purpose of the present Technical Report in Section 18.7.

EcoMetrix Incorporated and True Grit Consulting Ltd. were retained to review the environmental baseline assessment of the aquatic and terrestrial environments associated with the Marathon PGM-Cu project in support of the updated Feasibility Study and the Environemntal Assessment. A summary of the environmental baseline activities, and the environmental assessment and permitting processes are included in Section 18.5.

A draft copy of the report has been reviewed for factual errors by Marathon PGM. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this report.

4.0 PROPERTY LOCATION AND DESCRIPTION

4.1 LOCATION AND ACCESS

The Marathon PGM-Cu project is located approximately 10 km north of the Town of Marathon, Ontario (Figure 4.1 and Figure 4.2). The Hemlo Mining Camp is located 30 km to the southeast. The population of Marathon is approximately 5,000, and the town is situated adjacent to the Trans-Canada Highway No. 17 on the northeast shore of Lake Superior. The centre of the property sits approximately at 48° 45’ N latitude, 86° 19’ W longitude. Marathon is approximately 300 km east of Thunder Bay by highway and 400 km northwest of Sault Ste-Marie by highway. Primary industries supporting the Town of Marathon are pulp-and-paper and mining.

Local access to the property is by gravel from highway 17, which lies just north of Marathon.

4.2 DESCRIPTION AND TENURE

Upon incorporation, Marathon PGM issued 150,000 Common Shares to JDF Consulting LLC in exchange for the option agreement to purchase 100% of the Marathon PGM-Cu property. This option was exercised in January, 2004 and a 100% interest in the 1,654 ha Marathon PGM-Cu property was purchased from a subsidiary of Polymet Mining Corp. (Polymet) in December 2003. There are no remaining royalties or other interests in the property. At the time of purchase, the Marathon PGM-Cu property consisted of two Crown leases and three additional unpatented mining claims. The property was originally acquired as mineral claims by The Anaconda Company (Anaconda) in 1963.

The original property has since been enlarged by Marathon PGM through the periodic staking of unpatented mining claims and by the acquisition of existing claims.

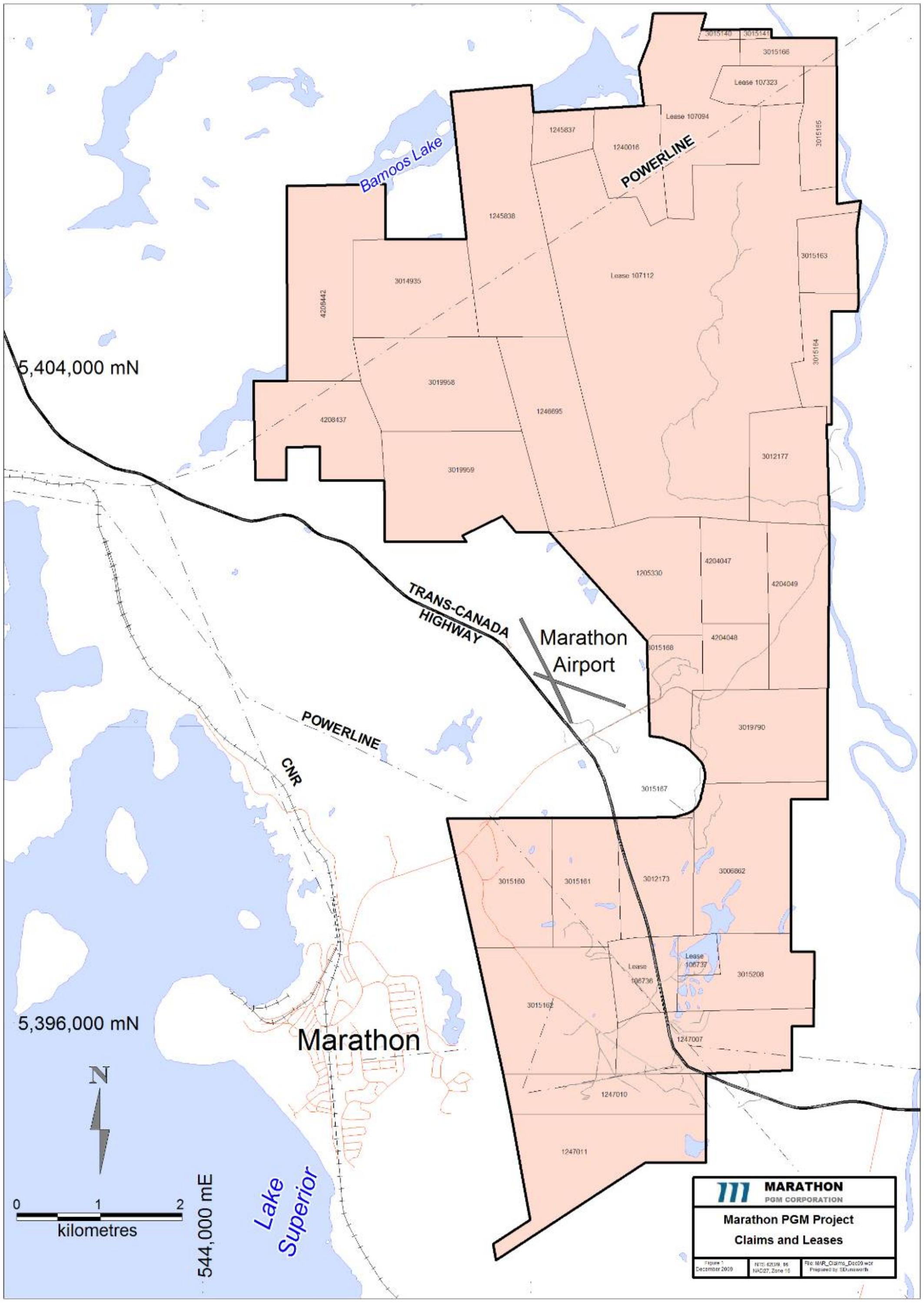
As of the date of this report, the property consists of five Crown Leases and 32 unpatented mining claim blocks for a total of 5,740 ha, the details of which are shown in Table 4.1.

4.3 BAMOOS-CLAW LAKE FOUR DAM PROPERTY (BENTON JV)

On March 11, 2008, Marathon PGM entered into an Option and Joint Venture Agreement (OJVA) with Benton Resources Corp. (Benton) on the Bamoos-Claw Lake-Four Dam Property (the BCF property), adjacent to the Marathon PGM-Cu property.

Figure 4.1
Regional Location Map

Figure 4.2
Detailed Location Map and Claims Map



In March, 2009, Marathon PGM announced the agreement to purchase from Benton 100% of a land package consisting of one mining lease and one claim block covering a total of 329 ha adjoining the northern border of the Marathon PGM-Cu project area. As a result, the original OJVA with Benton has been concluded. The purchased property is represented by Lease 107094 and claim 1240016 as shown in Table 4.1 and Figure 4.2.

Under the terms of the March, 2009 purchase agreement:

- Marathon PGM will issue 1,500,000 shares to Benton over a two-year period and make a cash payment of $300,000.

- Benton will retain a 2% NSR royalty, 1% of which can be purchased for $1 million.

- Marathon PGM will assume existing NSRs on the acquired property of 2% applied to the lease and 1% on the claim.

4.4 GEORDIE LAKE PROPERTY ACQUISITION

On 19 February, 2008, Marathon PGM announced its intention to acquire 100% of the outstanding securities of Discovery PGM Exploration Ltd. (Discovery) by means of a share exchange, subject to regulatory approval and acceptance by a minimum of 66⅔% of Discovery's shareholders.

Discovery owns 100% of the Geordie Lake deposit (GLD), located 14 km northwest of Marathon, Ontario, which is within the Coldwell Complex, hosting the Marathon PGM-Cu deposit. The property is subject to a 2.5% NSR royalty that reduces to 1.5% after the first $1 million in payments has been made. The property is also subject to a 12½% back-in working interest upon Discovery presenting a feasibility study and the right holder paying 31.25% of all the costs to the point of the study. The property covers 1,538 ha. In June, 2008, Marathon PGM completed a NI 43-101 report on Geordie Lake copies of which can be found on Sedar and Marathon PGM's website.

It should be noted that the resources presented herein do not include the GLD and do not form part of the project.

Table 4.1
Description of the Marathon PGM-Cu Property

Lease	Claim*	Ha	Units	Area	Recorded	Expire / Assess. Date
106736	G4040128	72		Pic	01-Jun-93	31-May-14
106737		19		Pic	01-Jun-93	31-May-14
107112	G4040009	1,111		Seeley Lk	01-Nov-97	31-Oct-18
107323		65		Seeley Lk	01-Aug-00	31-Jul-21
107094	G4040129	217		Seeley Lk	01-Mar-97	28-Feb-18
Subtotal		**1,484**				
1	4208437	160	10	McCoy	28-Sep-07	11-Apr-11
2	3012177	96	6	O'Neill	26-May-04	26-May-12
3	3015164	64	4	O'Neill	24-Aug-06	24-Aug-12
4	3015168	80	5	O'Neill	13-Oct-06	13-Oct-13
5	1247007	128	8	Pic	21-Sep-00	17-Jan-12
6	1247010	192	12	Pic	21-Sep-00	17-Jan-12
7	1247011	192	12	Pic	21-Sep-00	17-Jan-12
8	3006862	240	15	Pic	13-Apr-04	13-Apr-11
9	3012173	176	11	Pic	31-Mar-04	31-Mar-11
10	3015160	176	11	Pic	14-Jul-05	14-Jul-11
11	3015161	128	8	Pic	14-Jul-05	14-Jul-12
12	3015162	240	15	Pic	14-Jul-05	14-Jul-11
13	3015167	32	2	Pic	13-Oct-06	13-Oct-12
14	3015208	128	8	Pic	23-Mar-06	23-Mar-12
15	1205330	32	2	Seeley Lk	15-Nov-00	15-Nov-12
16	1240016	112	7	Seeley Lk	11-Apr-00	11-Apr-15
17	1245837	32	2	Seeley Lk	02-Mar-01	02-Mar-11
18	1245838	224	14	Seeley Lk	02-Mar-01	02-Mar-13
19	1246695	224	14	Seeley Lk	30-Mar-01	30-Mar-11
20	3014935	192	12	Seeley Lk	26-Oct-05	26-Oct-12
21	3015140	16	1	Seeley Lk	25-Jul-07	25-Jul-14
22	3015141	16	1	Seeley Lk	25-Jul-07	25-Jul-14
23	3015163	80	5	Seeley Lk	24-Aug-06	24-Aug-12
24	3015165	64	4	Seeley Lk	24-Aug-06	24-Aug-12
25	3015166	48	3	Seeley Lk	24-Aug-06	24-Aug-12
26	3019790	192	12	Seeley Lk	03-May-04	03-May-11
27	3019958	224	14	Seeley Lk	10-Nov-04	10-Nov-11
28	3019959	208	13	Seeley Lk	10-Nov-04	10-Nov-11
29	4204047	112	7	Seeley Lk	10-Feb-05	10-Feb-12
30	4204048	64	4	Seeley Lk	10-Feb-05	10-Feb-11
31	4204049	160	10	Seeley Lk	10-Feb-05	10-Feb-11
32	4208442	224	14	Seeley Lk	06-Feb-07	06-Feb-11
Subtotal		**4,256**	**266**			
Total		**5,740**	**266**			

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPY

The Marathon PGM-Cu property is located latitude of 48.77º North and 86.30º West, approximately 10 km north of the Town of Marathon, Ontario. See Figure 4.2, above. Local access to the property is by gravel road from highway 17, which lies just north of Marathon.

Mining equipment and personnel are available in Marathon and in Thunder Bay, which is located approximately 300 km to the west of the property.

Exploration and drilling may be carried out throughout the year except during the spring break up when most gravel roads are not suitable for vehicles and weight restrictions are placed on Highways.

5.1 CLIMATE AND PHYSIOGRAPHY

The climate is typical of northern areas within the Canadian Shield with long winters and short, warm summers.

Average annual precipitation in the area of Marathon was 826 mm for the period 1952-1983, of which 240 mm fell as snow. Average annual surface runoff is approximately 390 mm. 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.

The Marathon PGM-Cu property is located in an area characterized by moderate to steep hilly terrain with a series of creeks and lakes and dense vegetation. The vegetation consists of northern hardwood and conifer trees as well as muskeg areas, 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 general elevation around the mine site is slightly higher than the overall regional topography. Ground surface elevations in the area of the proposed mine range from about 260 m to over 400 m above sea level with a gradual decrease in elevation from north to south.

5.2 INFRASTRUCTURE

Logistical support, including power and telephone lines, is available at the property and at Marathon, which is linked to the Ontario power grid. Water is available from the Pic River as well as from many creeks which drain the area.

A high voltage power line transects the property. A rail line runs close to it and deep water dock facilities are available at Marathon.

The Marathon airport is located near the southwest corner of the Marathon property. The airport is used by private plane owners and several small commercial helicopter companies.

5.3 LAND USE

Land-use activities in the project area include hunting, fishing, trapping and snowmobiling. The existing access road is likely used by anglers to access the Pic River and by snowmobile users in the winter. Sport fishing activity is likely focused on the Pic River which contains a variety of warm water fish species and in Hare and Bamoos Lakes, located northwest of the project. Hare Lake is road accessible from its southwest corner. There are two permanent cottages on the shoreline of Hare Lake. Bamoos Lake is accessible by air and portage from Hare Lake/Creek and may be accessible during the winter using snow machines.

Pukaskwa National Park is located near the mouth of the Pic River approximately 20 km downstream of the project.

6.0 HISTORY

The reader is referred to the technical report titled, "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 that was filed on www.sedar.com on March 28, 2006 (P&E Mining Consultants Inc., 2006a).

7.0 GEOLOGICAL SETTING

For a detailed account, the reader is referred to (a) the technical report titled, "Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated February 2, 2009 that was filed on www.sedar.com on February 2, 2009 (Micon International Limited, 2009) and (b) an in-house report entitled "Geology of the Marathon Deposit" dated October 24, 2009, written by Dr. David Good, VP Exploration for marathon PGM. The following summary has been compiled from extracts from Dr. Good's report and is presented here for completeness of the report.

7.1 REGIONAL SETTING

The Marathon PGM-Cu property currently comprises the Marathon deposit only but the Geordie Lake deposit remains a property for further evaluation.

The Marathon PGM-Cu deposit is hosted within the Eastern Gabbro of the Proterozoic Coldwell Complex which intrudes the much older Archean Schreiber-Hemlo greenstone belt. The sub-circular 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 Geordie Lake deposit is located in gabbro similar to that which hosts the Marathon PGM-Cu deposit but is located near the centre of the Complex and is presumably related to the Western Gabbro series.

The Coldwell Complex was emplaced as three nested intrusive centres (Centres I, II and III) (Mitchell and Platt, 1982) that were active during cauldron subsidence near where the northern end of the Thiel Fault intersected Archean rocks, on the north shore of Lake Superior (Figure 7.1). It is considered to be related to other intrusive complexes associated with the Mid Continental rift system such as the Duluth Complex, Logan sills, and Crystal Lake Gabbro which were emplaced at around 1,108 Ma (Heaman and Machado 1992).

7.1.1 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 2 km thick and strikes for 33 km around the eastern margin of the Coldwell Complex (Figure 7.2). It is considered the oldest intrusive phase of the Complex and is interpreted to have formed by at least three discrete intrusions of magma into restricted dilatant zones within a ring dyke possibly associated with ongoing caldera collapse (Walker et al., 1993; Shaw, 1997).

7.1.2 Geordie Lake Gabbro

The Geordie Lake Gabbro is located 12 km west of the Marathon PGM-Cu deposit (Figure 7.2) and is presumably related to the Western Gabbro series in Centre II of the Coldwell

Complex. The gabbro occurs as an elongated north south striking sheet like body approximately 3 km long and up to 700 m wide and dips moderately to the west. The body is bounded to the east and south by fine grained amphibole quartz syenite and to the west by alkali feldspar porphyritic amphibole-syenite (MacTavish, 1988). The Geordie Lake Gabbro is similar to the Two Duck Gabbro (TD Gabbro) in terms of crystallization history, mineral compositions and trace element geochemistry (Good, 1992).

Figure 7.1
Regional Geology of the Lake Superior Area

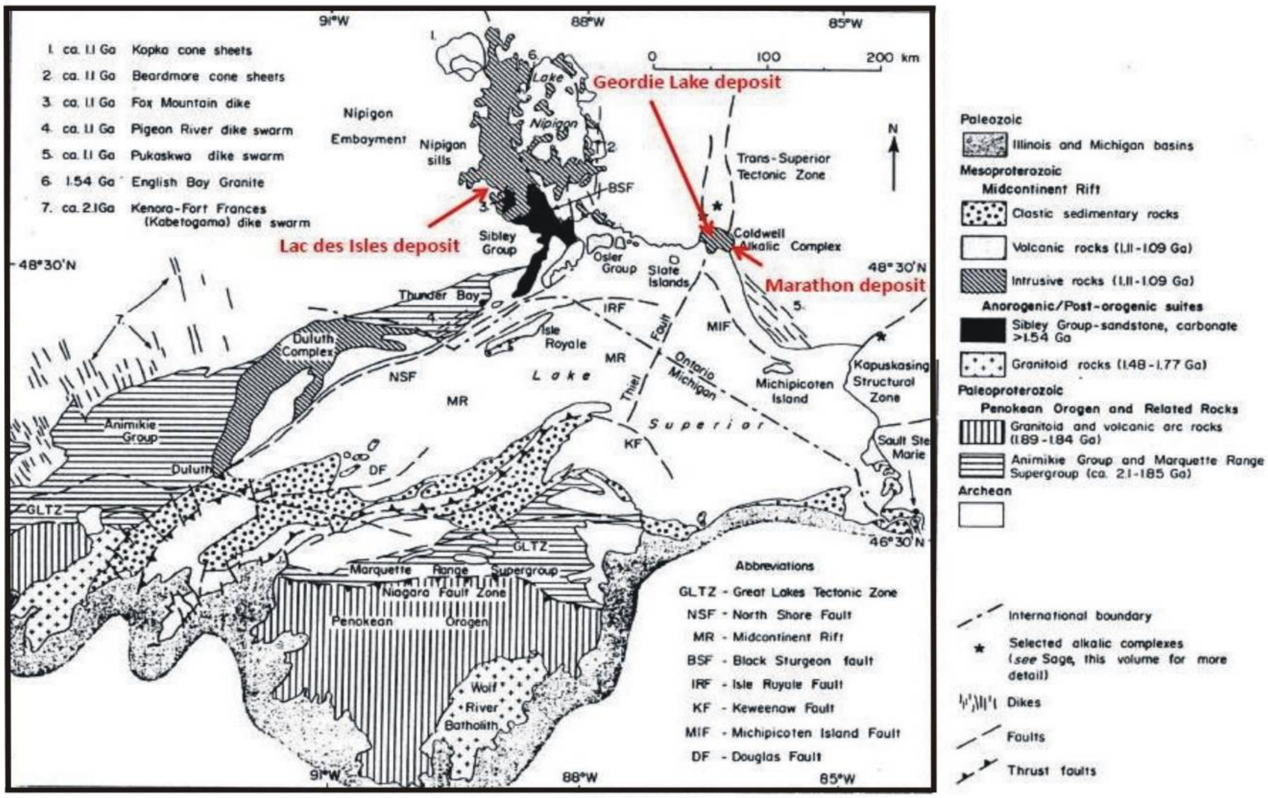
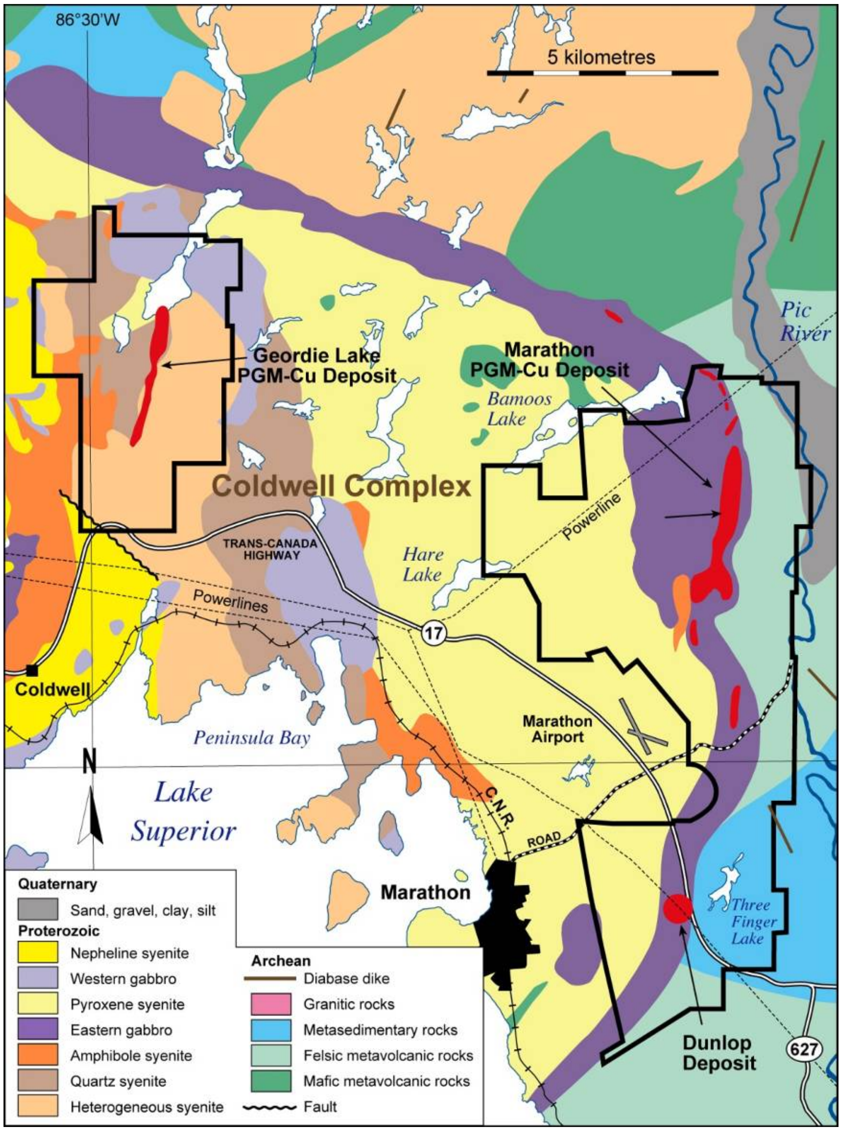


Figure 7.2
Geology of the Eastern Half of the Coldwell Complex Showing the Locations of the Marathon Deposit and the Geordie Lake Deposit and the Outlines of the Marathon PGM Property Boundaries



7.2 OUTLINE OF PROPERTY GEOLOGY

Mineralization at the Marathon PGM-Cu property is part of a very large magmatic system that consists of at least two major intrusive events of predominantly olivine gabbroic units that form the Eastern Gabbro of the Coldwell Complex (Figure 7.3). The earlier of the two events is termed the Layered Gabbro Series (LGS) and is made up of alternating layers of gabbro, olivine gabbro and troctolite. The grain size for units within the LGS varies considerably, with some units, on the order of 100 m in thickness, being comprised of numerous 1- to 5-m thick layers of fine grained gabbro. The LGS was intruded by the Two

Duck Intrusion (TDI) in multiple horizons within the stratigraphic package that makes up the LGS. The TDI is composed of coarse grained to pegmatitic relatively homogeneous gabbro and olivine gabbro or troctolite. Late quartz syenite and augite syenite dykes cut all of the gabbros but form a minor component of the intrusive assemblage.

The TDI is the host rock for Cu-PGM mineralization and is the focus of exploration.

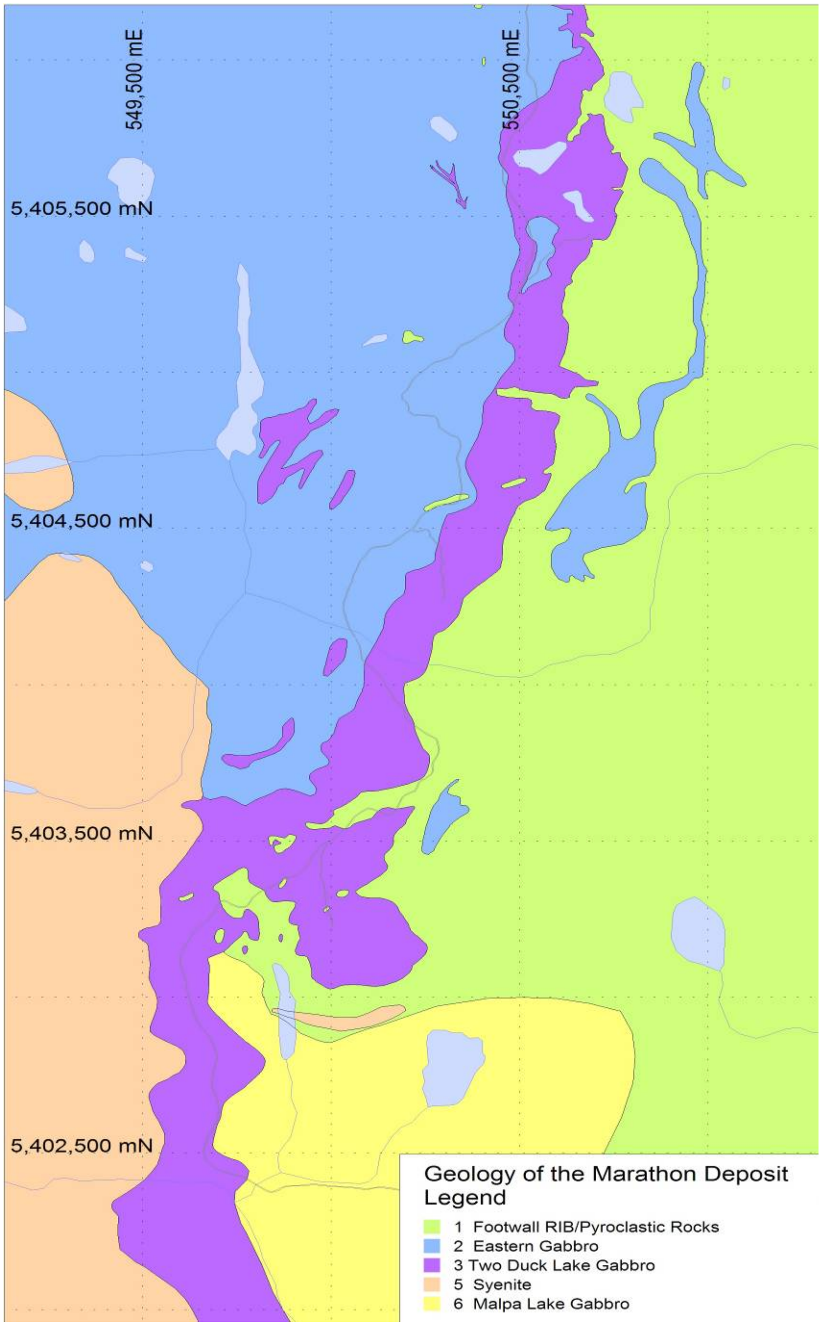
Previous workers have suggested the LGS and TD Gabbro are part of a single large layered intrusive complex with upper, lower and basal zones and TDI is the basal or contact phase of the Eastern Gabbro Layered Intrusion (Mainwaring et al., 1982, Good, 1993, Dahl et al., 2001, and Barrie, 2001). However, the gabbro units clearly do not co-exist as an orderly assemblage similar to other layered intrusions. Recent detailed mapping of numerous exploration trenches shows the TDI cross-cuts the LGS at multiple horizons with most of the TDI occurring at the base of the LGS, but a significant amount of TDI occurs higher up in the stratigraphy of the LGS as anastomozing or bifurcating series of dykes or sills that cut the pre-existing gabbros.

The TDI that occurs higher up in the LGS is a very distinct unit of the TDI because it locally contains significant amounts of Cu-PGM mineralization within layers of up to 90% cumulus magnetite that are tens of metres thick and so this unit is termed the layered magnetite olivine cumulate (LMOC). These magnetite rich layers were previously believed to form an oxide reef within the LGS (Mainwaring et al., 1982, and Barrie, 2004) but recent mapping clearly shows the magnetite rich layers occur within zoned or layered dikes and pods that cut the LGS and does not form a continuous reef. The LMOC occur as complicated assemblages of cumulus olivine, magnetite and plagioclase and interstitial clinopyroxene.

Only the TDI occurs as a continuous and uninterrupted body and can be traced over a strike length of at least 7 km. All workers agree the cross cutting relationships complicate the geology. Whether the gabbros of the LGS and TDI intruded sequentially in a single event or there was a hiatus between intrusions is the focus of an ongoing U-Pb isotopic study.

There are many striking similarities between the TDI and the Partridge River intrusion within the Duluth Complex (Figure 7.1) which is host to major Cu-Ni-PGM deposits (for example, the Northmet deposit). The relevant features described in both locales as discussed by Good and Crocket (1994) includes similar ages (about 1,100 Ma) and tectonic origin (mid-continent rifting event), and composition and textures of gabbro and nature of sulphide mineralization.

Figure 7.3
Geological Map of the Marathon Deposit



8.0 DEPOSIT MODEL

The following summary descriptions are updates taken from an internal report by Dr. David Good titled "Geology of the Marathon Deposit" dated 24 October, 2009.

8.1 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, 2001). A hydrothermal mechanism at low or intermediate temperatures (<600^{o}C) is not likely given the near total absence of hydrous minerals in the W Horizon and the significant correlation between Pd and Ir which could not occur in the light of the virtually immobile behavior of Ir in hydrothermal fluids. The high temperature, zone refining mechanism suggested by Barrie (2001) 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. There is just too much PGM and too little gabbro.

It seems most likely that more than one process operated at high temperatures (>700^{o}C) to concentrate metals in the Marathon deposit. Three possible mechanisms include:

1. Accumulation of sulphide liquid in fluid dynamic traps in the magma conduit.

2. Ongoing interaction of sulphides with magma that is flowing through the conduit (N-factor).

3. Removal of S, Cu, and Au from the sulphide assemblage.

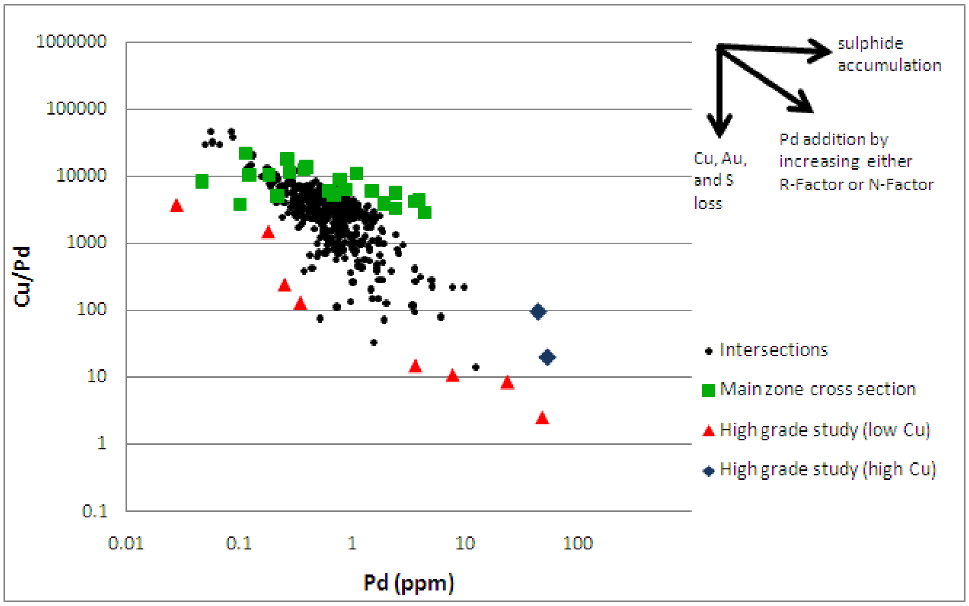
The effects of the three mechanisms on the abundance of Cu and Pd are shown in Figure 8.1. The effect of accumulating sulphides is shown by the trend for the Main Zone samples (green squares). The effect of the N-factor is the rapid increase in Pd relative to Cu and pulls samples toward the lower right corner of the figure. The intersection data (black dots) represent the average affects of both sulphide accumulation and N-factor enrichment. Finally, the removal of Cu from PGM enriched zones (W Horizon) is shown by the downward displacement of the samples from the high grade sample study (low Cu type) (red triangles).

8.2 FLOW THROUGH MODEL FOR MARATHON MINERALIZATION

In the current exploration model, the present exposure of the Two Duck 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.

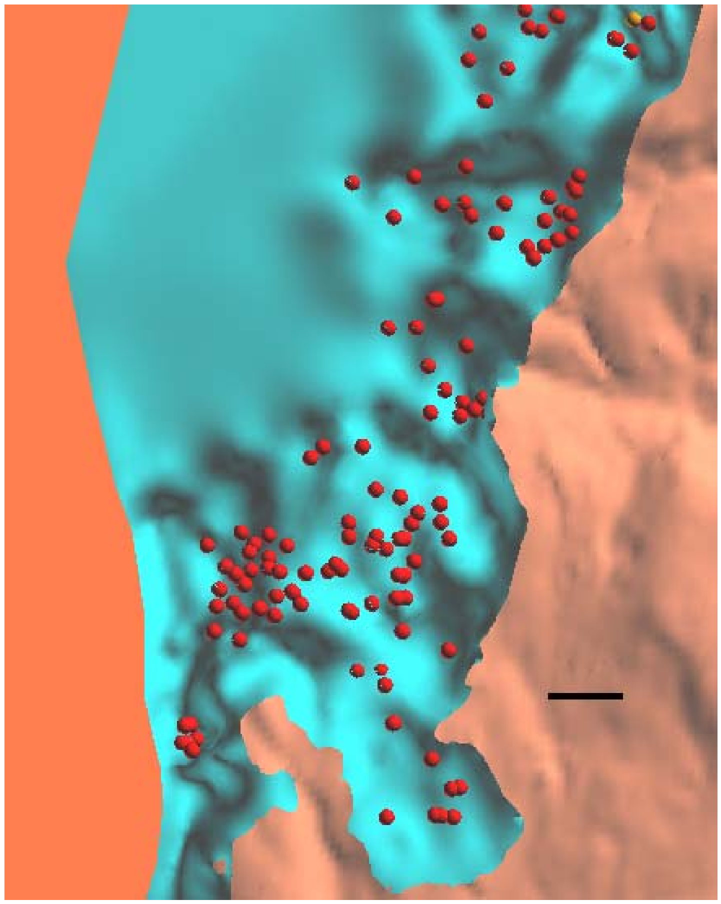
Figure 8.1
Diagram Illustrating the Effects on Metal Values of the Three Dominant Mechanisms Proposed to Explain the Concentration of Cu and PGM in the Marathon PGM-Cu Deposit



It is envisaged that a very large volume of magma, perhaps greater than 10,000 times the current volume of gabbro, flowed through the conduit and formed the TDI. On the basis of mass balance calculations, and considering the TDI is less than 250 m thick, only a very large magmatic system such as this can explain the extreme enrichments of platinum metals such as 45 g/t of combined platinum, palladium and gold over 10 m or the accumulations of disseminated sulphide layers that are up to 160 m thick. Similarly, in the case of the LMOC, very large volumes of magma are required to deposit the very thick layers (tens of metres) with >75% magnetite.

The relationship between the shape of the footwall and the abundance of sulphides is best shown at the south end of the Marathon PGM-Cu deposit in Figure 8.2 where the best intersections, identified by filtering out those intersections worth less than $75/t (calculated NSR value), occur within valleys, troughs or basins in the footwall. In the flow through model, fluid dynamic factors that affected magma flow are relevant to exploration. Features such as cooling of TDI magma in basins within the footwall or brecciation of gabbro in the LGS by TDI magma as it stopes its way upward during ascent are important examples of how the magma was either slowed down or interfered with enabling the precipitation of the more dense sulphide liquid from the magma. Conversely, above ridges or crests in the footwall, where TDI 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 PGM-Cu deposit and the observed metal trends such as in diamond drill hole MB-08-10.

Figure 8.2
3D View of the Footwall of the Two Duck Intrusion (Light Blue) Showing the Distribution of the Highest Value Intersections (Red Dots) With Respect to Valleys and Troughs

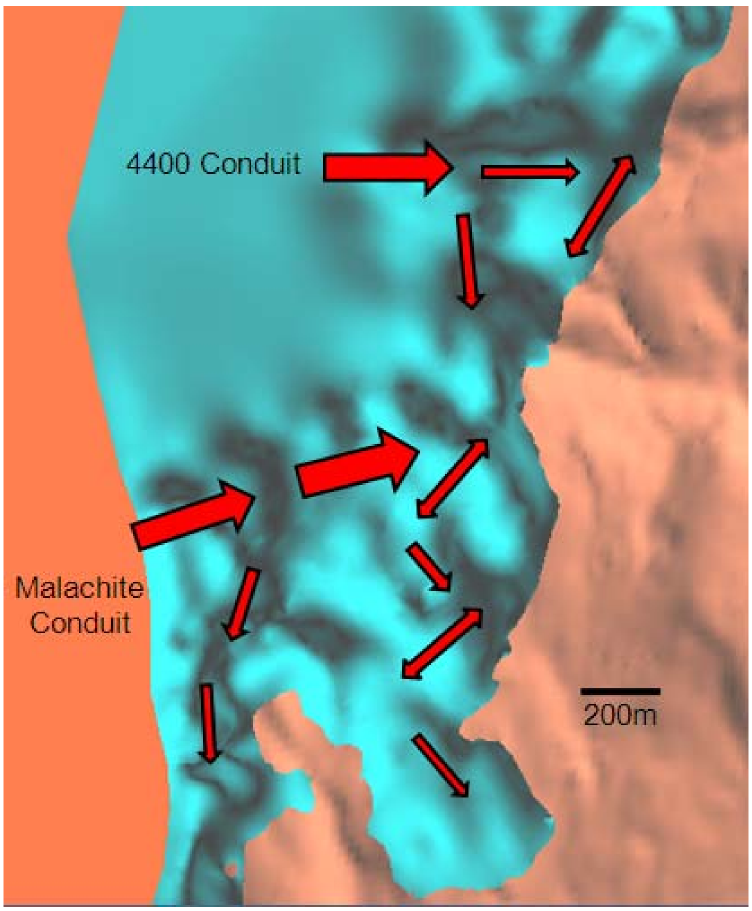


Area shown is located between lines 4700N and 3100N. Black bar represents 200 m scale.

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. The upgrading occurred as magma passed through the conduit and interacted with sulphides in the crystal pile possibly by stirring up early formed sulphides or by diffusion of metals out of the magma and down into the crystal pile. It also seems possible that sulphides were picked up and transported in the magma during flow through. This process of sulphide upgrading was used to describe the extreme enrichments of PGM relative to copper in disseminated sulphides at the Norilsk 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 N-factor. Under conditions where the N-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 107 g/t over 2 m (hole M07-237), or 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 a result of continuous upgrading.

The envisaged magma conduits that describe variations in Cu/Pd ratios at the south end of the Marathon PGM-Cu deposit are shown in Figure 8.3. In the flow through model, very low Cu/Pd ratios for the sulphides correspond to very high volumes of magma interacting with the sulphides. A contour map of Cu/Pd ratios in samples from the W Horizon (not shown) is thus a remnant artifact of the magma flow.

Figure 8.3
Proposed Magma Conduits and Flow Model for Two Duck Intrusion



The 4400 conduit and malachite conduit represent troughs that have the lowest Cu/(Pt+Pd) ratios. The direction arrows represent contour lines for increasing Cu/(Pd+Pt) ratios. Data used for the contour map were taken from the W Horizon and have a range of Cu/(Pd+Pt) ratios between 2 and 400.

It is evident that S, Cu and Au were removed in some areas of the W Horizon presumably during interaction between pre-existing sulphides and sulphur undersaturated magma. This process would explain the few instances where up to 75 g/t Pd occurs in samples with only

0.01 to 0.02% Cu. These levels of Pd when recalculated to abundances in 100% sulphides correspond to untenable concentrations of between 2 and 4% Pd in 100% sulphide. This process would also explain the unusual feature shown in the detailed study of the PGM enriched samples where there is relative depletion of Au as compared to Pd. High temperature removal of Au, Cu and S (de-sulphurization) is proposed to account for the extreme ratios of Pd/Cu and Pd/S and the relative depletion of Au compared to PGM.

9.0 MINERALIZATION

The following concise descriptions are complementary to the descriptions as detailed in the previous report entitled "Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated February 2, 2009 that was filed on www.sedar.com on February 2, 2009 (Micon International Limited, 2009).

9.1 SULPHIDE MINERALIZATION IN THE TWO DUCK INTRUSION

Sulphides in the TDI consist predominantly of chalcopyrite, pyrrhotite and minor amounts of pentlandite, cobaltite, bornite 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 in replacement zones at the margins of plagioclase crystals (Good and Crocket, 1994a).

The sulphide assemblage changes gradually up section from the base to the top of mineralized zones. Sulphides at the base of the TDI 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.

There is a relationship between mineralization and the paleo-topography of the footwall contact. For example, mineralization is best developed within basins of the footwall and thins or pinches out above prominent footwall ridges (Figure 9.1)

9.2 MINERALIZED ZONES

The Marathon PGM-Cu deposit consists of several large, thick and continuous zones of disseminated sulphide mineralization hosted within the TDI. The mineralized zones occur as shallow dipping sub-parallel lenses that follow the basal gabbro contact and are shown as footwall, main, hanging wall zones and the W Horizon (Figure 9.1). The Main Zone is the thickest and most continuous zone.

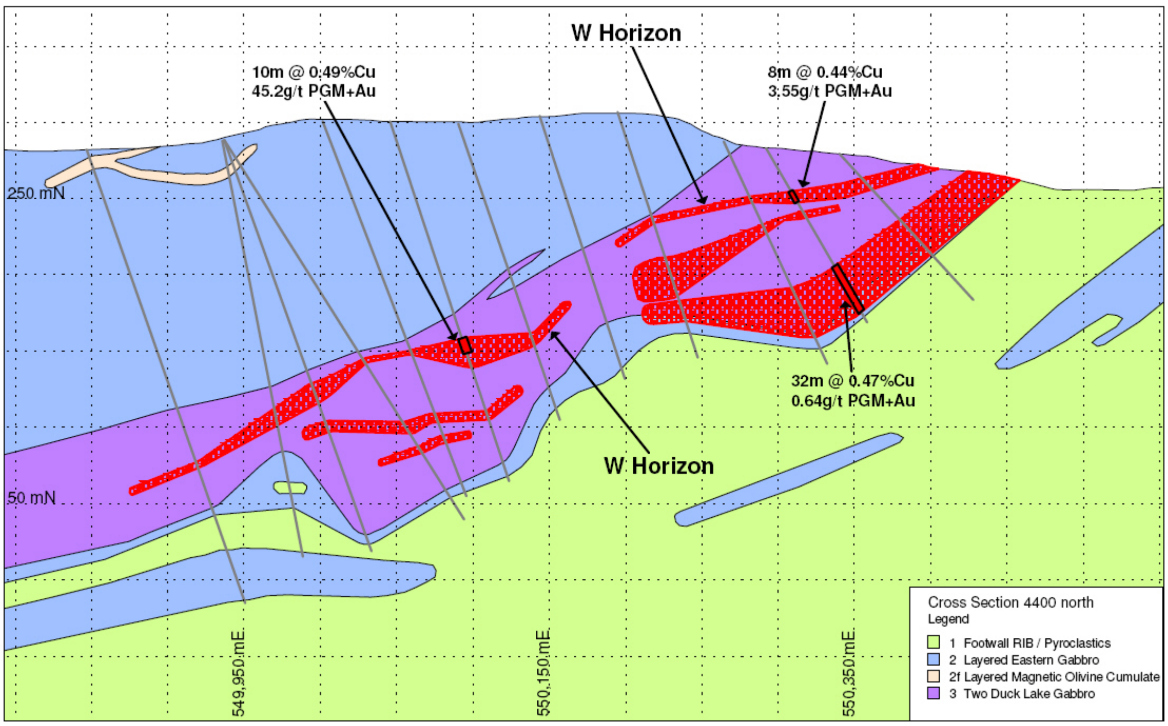
For 516 intersections greater than 4 m thick, the average thickness is approximately 35 m and the maximum thickness is 183 m.

9.3 THE W HORIZON

The W Horizon forms a nearly continuous sheet of mineralization that strikes north south for over 1 km from section 3450N to section 4500N and continues down dip for over 300 m. The zone is open at depth. It ranges in thickness from 2 m (minimum sample width) to 30 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, but if sulphides are present, they consist of 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 in size from 30 m to 200 m occur within the W horizon. The highest intersection to date contains 107 g/t PGM+Au, 1.04 g/t Rh and 0.02% Cu over 2 m (hole M07-239), but the best intersection contains 45.2 g/t PGM+Au and 0.49 % Cu over 10m (hole M07-306 in Figure 9.1).

Figure 9.1
Cross-section of the Marathon PGM-Cu Deposit at 4400 North



In general the mineralized zones shown as mottled red thicken in basins of the footwall and thin or pinch out over crests where the TDI unit becomes thinner. The W Horizon is the upper red layer at both the east and west side of the section.

9.4 DISTRIBUTION OF PGMS, COPPER AND NICKEL, AND PGM/METAL RATIOS

The reader is referred to the previous technical report entitled “Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada”, dated February 2, 2009 that was filed on www.sedar.com on February 2, 2009 (Micon International Limited, 2009).

10.0 EXPLORATION

10.1 INTRODUCTION

Since acquiring the Marathon PGM-Cu deposit from Polymet in December, 2003, Marathon PGM has funded continuous programs of advanced exploration and diamond drilling commencing with its surface exploration program in June, 2004. The early field work of follow-up surface sampling and trenching of prospective mineralized zones focused on the southern part of the property where only a slight amount of past work had uncovered good surface showings and mineralized intercepts in very widely spaced drilling.

The summer 2004 program of prospecting and mechanical stripping enabled several previously known zones, such as the RD, BR and Malachite Zones, to be significantly enlarged. As a result of this program, the BR Zone was traced by stripping over an area of approximately 100 m by 100 m. The RD Zone was traced by prospecting and trenching over an area 400 m by 150 m and the southern resource area was enlarged to a defined strike length of 600 m and a width of about 200 m. The Malachite Zone was enlarged by mechanical stripping, from an initial area approximately 10 m by 50 m to a zone extending over an area at least 100 m wide and 300 m long (Marathon PGM press release dated 12 October, 2004).

The 2004 summer field work was followed by a 4,000-m drilling program consisting of 32 NQ core holes. The areas tested included the Malachite Zone (10 holes), BR Zone (6 holes), RD Zone (8 holes) and the southern part of the south resource area (8 holes). In addition, further trenching and channel cutting proceeded in the RD and BR Zones as the drill program advanced.

The 2005 exploration program encompassed detailed geophysical surveying, airborne geophysical re-interpretation, geological mapping and approximately 14,000 m of drilling in 102 holes, and was very successful in expanding the extent of mineralization in the Main Zone as well as in the Malachite, BR and RD Zones.

Marathon PGM undertook an extensive diamond drilling program on the property in 2006, comprised of 108 holes for 21,800 m (excluding the holes drilled on the Dunlop Zone). These drill results were added to the prior drilling on the property for a total of 83,104 m.

Prospecting was conducted during the 2006 season on magnetite layers approximately 200 m west of the Main Zone. Trenching was done in the SG and WD Zones. The SG Zone was sampled along five trenches, for a total of 233 m of sampling. The WD Zone was sampled along eight trenches for a total of 921 m of sampling.

In 2007, trenching focused on exploration in the hanging wall of the Main Zone. The objective was to evaluate the potential of sulphides associated with the LMOC. Work included clearing and washing 22 trenches totaling 2,670 m in length, and cutting a total of 1,812 m of channels from which 1,277 samples were collected.

In 2008, trenching focused on exploration of the Benton-Marathon JV Option Property. The objective was to extend the Main Zone mineralization and host rock (TD Gabbro) to the north. Work included clearing and washing 17 trenches totaling 3,300 m in length, and cutting a total of 1,960 m of channels from which 1,494 samples were collected.

For a complete account of the 2004, 2005, 2006, 2007 and 2008 exploration programs, including diamond drilling, the reader is referred to the report titled "Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated February 2, 2009 that was filed on www.sedar.com on February 2, 2009 (Micon International Limited, 2009).

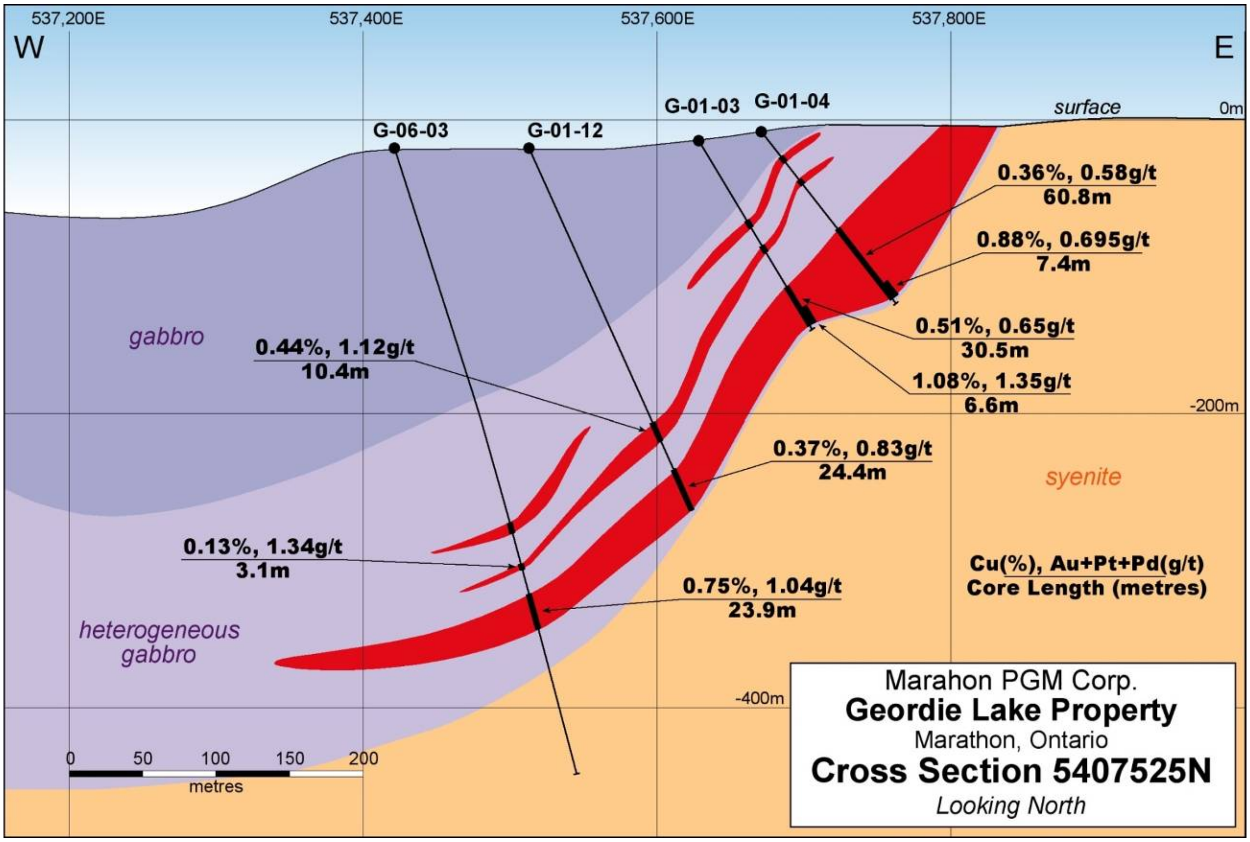
10.2 2009 EXPLORATION AND CURRENT PROGRAMS

Cu-PGM occurrences in the Coldwell Complex are reported along the entire 33-km length of the Eastern Gabbro and in the vicinity of the Geordie Lake Gabbro associated with the Western Gabbro in the centre of the Complex. As of the date of this report, Marathon PGM has explored approximately 11 km of the Eastern Gabbro and the 2009 program culminated in 21 holes being drilled as detailed in Section 11. Current exploration efforts are focused to the north of the Geordie Lake Gabbro. The Marathon PGM exploration strategy is to apply systematic prospecting, trenching, geological mapping, 3D modeling and finally diamond drilling within the framework of the geological deposit model. The strategy has shown excellent results in recent years with the development of the RD and Malachite Zones to the south and more recently, the Bamoos Zone to the north.

The Geordie Lake Gabbro is a promising target located 12 km west of the Marathon PGM-Cu deposit (Figure 7.2) and is presumably related to the Western Gabbro series in Centre II of the Coldwell Complex. Mineralization occurs within the Geordie Lake Gabbro as several sub-parallel zones (Figure 10.1). The thickest and best mineralization occurs within gabbro along the eastern contact with syenite (Giroux and Stanley, 2002). The sulphides consist predominantly of chalcopyrite, bornite and pyrite with minor amounts of galena and rare siegenite, millerite, sphalerite, cobaltite and niccolite (Mulja and Mitchell, 1991). The sulphides are associated with locally intense albite and actinolite alteration within heterogeneous textured gabbro (Good and Crocket, 1994b).

The Geordie Lake mineralization is interpreted to have formed at temperatures below about 600°C by hydrothermal processes related to volatiles that were driven out of the Geordie Lake crystal pile by fractional crystallization (Good and Crocket, 1994b).

Figure 10.1
Cross-section Through the Geordie Lake Gabbro at 5407525N



11.0 DRILLING

The number and depth of holes used by Micon in the mineral resource estimate described in this Technical Report is summarized in Table 11.1 which also includes historical drilling. Details on the earlier drilling, i.e., prior to 2007, are given in the technical report entitled "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, that was filed on SEDAR on March 28, 2006 (P&E Mining Consultants Inc., 2006a). The most recent 2009 drilling comprises 21 holes and is shown on Figure 11.1.

Table 11.1
Summary of Diamond Drill Holes Used for the Marathon PGM Resource Estimate

Drill Hole Series	Total Holes	Date Drilled	Total Depth (m)	Company
M-09-470 to 490	21	2009	2,334	Marathon PGM
M08-418 to 469	52	2008	11,537	Marathon PGM
MB-08-1 to 42	39	2008	7,607	Marathon PGM
M-07-237 to 417	181	2007	35,720	Marathon PGM
M-06-135 to 236	102	2006	20,495	Marathon PGM
M-05-33 to 134	102	2005	14,602	Marathon PGM
M-04-1 to 32	32	2004	4,080	Marathon PGM
GD-06-1 to 6	6	2006	1,304	Marathon PGM
G1 to G15	15	2001	3,158	Geomaque
F1 to F37	37	mid 1980s	3,627	Fleck
DH1-DH169	106	1964-1967	23,869	Anaconda
BO-05-1 to BO-07-55	12	2005 to 2007	2,227	Benton Resources
Total	**705**		**130,560**	

11.1 MARATHON PGM 2007 DRILLING

A total of 36,779 m was drilled in 2007 including 176 holes totaling of 35,057 m as infill and step out holes within the Marathon PGM-Cu deposit intended to upgrade or expand the resource. An additional 1,722 m were drilled in 13 holes outside the pit area. The results of the 2007 drilling campaign are presented in the report titled "Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated February 2, 2009 that was filed on www.sedar.com on February 2, 2009 (Micon International Limited, 2009). The drilling shown in Table 11.1 excludes 8 barren holes not used in the resource estimate.

11.2 MARATHON PGM 2008 DRILLING

A total of 19,538 m in 92 holes were drilled in 2008 as infill and step out holes within the Main Zone. An additional 842 m in five holes were drilled as exploration outside the pit area, and four holes for a total of 858 m were drilled as condemnation holes at the process solids (tailings) management facility (PSMF), crusher and mill sites. The results of the 2008 drilling campaign are presented in the report titled "Technical Report on the Updated Mineral

Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated February 2, 2009 that was filed on www.sedar.com on February 2, 2009 (Micon International Limited, 2009). All barren holes not included in the resource estimate are excluded from Table 11.1.

11.3 BENTON RESOURCES CORP. JV 2008 DRILLING

The mineral resources presented in this Technical Report include a portion that belongs to the Benton JV and which accounts for an approximate 150-m northerly extension to the main Marathon PGM-Cu deposit. This extension includes approximately 5 Mt of resources.

Drilling in 2008 on the Benton JV portion included 23 holes for a total of 6,862 m. Holes were named MB-08-01 through MB-08-42. The results of this drilling campaign are presented in the report titled "Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated February 2, 2009 that was filed on www.sedar.com on February 2, 2009 (Micon International Limited, 2009).

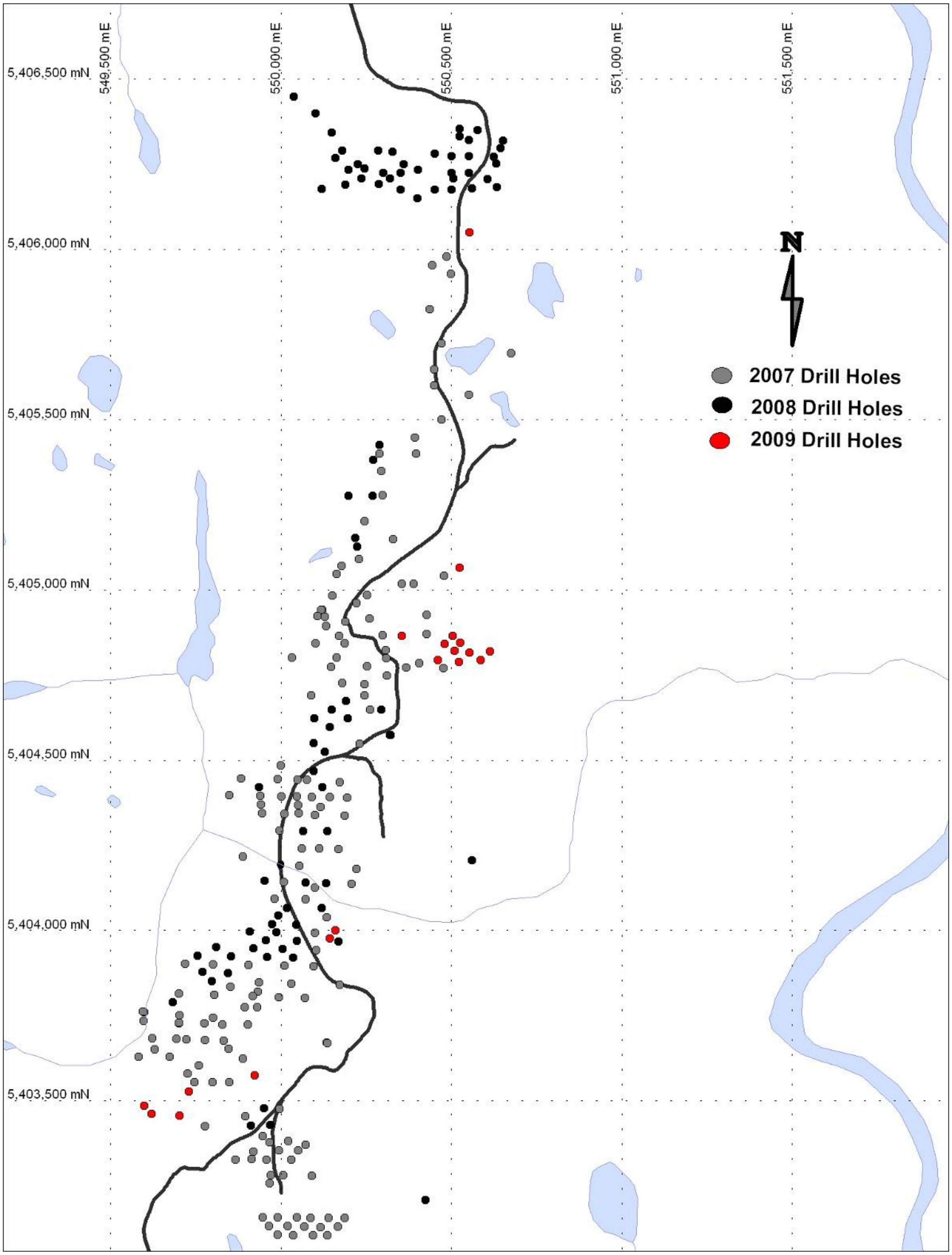
As noted in Section 4.3, in March, 2009, Marathon PGM concluded a purchase agreement with Benton in respect of one mining lease and one claim from the Benton JV property and, as a result, the original joint venture agreement was concluded.

11.4 MARATHON PGM 2009 DRILLING

A total of 2,334 m in 21 holes were drilled in 2009 as step out holes and were primarily intended to expand the resource. The results of this drilling significantly increased the resource as detailed in Section 17 of this report.

The location of the 2009 drill holes together with those for the 2007 and 2009 drill campaigns is shown in Figure 11.1.

Figure 11.1
Diamond Drill Hole Location Map for the 2007 Through 2009 Programs



12.0 SAMPLING METHOD AND APPROACH

12.1 METHODOLOGY

The 2007 and 2008 sampling protocols were maintained throughout 2009.

All core samples comprise 2-m intervals and were selected by a geologist. Samples were then tagged by a technician and divided into alternating batches of 23 or 22 samples. The batches were named by the hole number and sub-batch (a, b, c etc.).

The significance of this numbering system, with respect to the sub-batches, is due to the furnace runs completed at Accurassay Laboratories (Accurassay) (furnace runs of 23 or 22 samples).

All core was split on site (Marathon core facilities) using a diamond saw. While half of the core was retained for later reference or later sampling, the other was ready for processing.

Each sub-batch was cut and bagged together in a rice bag, sealed and dispatched twice a week to the Accurassay laboratories located at 1046 Gorham Street, Thunder Bay, Ontario, via Courtesy Transportation.

All core samples were recorded in the geological drill logs and in a sample chain of custody spreadsheet. While samples were en-route, the chain of custody spreadsheet was e-mailed to Accurassay.

For quality control purposes, each sub-batch consisted of a duplicate, blank and standard which were always positioned on the same sample location of each sub-batch. Repetitive QC positioning eliminates the chance of a duplicate from the laboratory (quality control procedure for the laboratory) being run on a submitted duplicate, blank or standard and also reduces the chances of mistakes at the sampling stage.

12.2 APPROACH

Due to the disseminated nature of the deposit and thickness of the mineralized zone of from 4 to 100 m, the sample interval of 2 m was justified. The potential mining method of large tonnage open pit was also considered when selecting sample intervals and smaller intervals would only be taken if there was a particular geological reason to do so.

12.3 COMMENTS

Core recovery was excellent and was not an issue during the program.

Micon considers that the sampling method and approach was logical and satisfactory.

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

13.1 PROTOCOLS BEFORE DISPATCH OF SAMPLES

A tag with a sample identification (ID) number was placed in each sample bag 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 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 before being dispatched. Other than the insertion of control samples as described in Section 12 above, there was no other action taken at site.

As for the 2007 and 2008 drilling campaigns, samples were delivered either by Marathon PGM personnel, shipped via Courtesy Courier, or rarely, and if samples were deemed to be high priority, shipped via Greyhound Bus Lines out of Marathon, to Accurassay's facilities in Thunder Bay, Ontario. Upon receipt of the samples, Accurassay personnel would ensure that the seals on rice bags and individual samples had not been tempered with.

Accurassay provides analytical services to the mining and mineral exploration industry and is registered under ISO 9001:2000 quality standard.

13.2 LABORATORY PROTOCOLS

At the time of delivery, the laboratory would acknowledge receipt of the sample shipment in good order. Samples were both prepared and analyzed at the Accurassay laboratory.

All samples were analyzed for Cu, Ni, Ag, Au, Pt and Pd. Rhodium was requested on samples within an intersection of two or more consecutive samples with an NSR value greater than $8, 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.

The following details have been extracted from the Accurassay's established procedures on the Marathon PGM samples.

13.2.1 Sample Preparation

The samples provided to Accurassay by Marathon PGM were core samples, rock samples and pulp samples. The samples were dried, if necessary, crushed to approximately minus 10 mesh and split into 250-g to 450-g sub-samples using a Jones Riffler. The sub-samples were then pulverized to 90% passing150 mesh using a ring and puck pulverizer and homogenized prior to analysis. Silica sand cleaning between each sample was performed to prevent cross-contamination between samples.

13.2.2 Fire Assay

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

Weighing: A 30.2-g sample mass was used for the Marathon PGM's samples. Note: sample masses may have been altered to accommodate sample chemistry, if required.

A furnace load consists of 23 or 24 samples with a check done every 10th sample (by client ID), along with a blank and a Quality Control Standard. Note: duplicate checks were done on pulverized samples.

Fluxing: Samples provided to Accurassay by Marathon PGM, did not require preliminary treatment and were mixed directly with the assay flux and fused. Currently, Accurassay uses 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.

Fusion: Samples are typically fused for 1¼ h at 1,800 to 2,000 degrees Fahrenheit. The fusion time may be increased if needed.

Cupellation: Samples are typically cupelled for 50 minutes at 1,900 degrees Fahrenheit. The cupellation time may be increased if needed.

13.2.3 Base Metals

For flame AAS determinations of Cu, Co, Ni, Pb, and Ag, an acid digestion consisting of aqua regia (1 part nitric to 3 parts hydrochloric acid) was the preferred method. A sample mass of 0.25 g and a final volume of 10 mL is used for the analysis. For samples requiring a full assay digestion (ore grade); a sample mass of 2.5 g and a final volume of 250 mL is used. A full assay is required whenever the concentration of any given element is greater than 1% for any of the above noted elements.

13.2.4 Digestion – Precious Metals

Precious metal beads were digested using a nitric/hydrochloric acid digestion and bulked up with a 1% 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 the analysis.

13.2.5 Flame Atomic Absorption Spectrometric Measurement

Accurassay uses 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 is used for the determination of base metals.

Calibration standards are made up from 1,000 ppm certified stock solutions. Quality assurance (QA) solutions are made up from separately purchased 1,000 ppm certified stock solutions. All stock solutions are prepared commercially by ISO certified suppliers.

13.2.6 Reporting

Laboratory reports are currently produced using Accurassay' local information management system (LIMS) program. All duplicate assays are reported on the certificate of analysis. Quality control (QC) standards and blanks are not reported unless requested by the client.

13.2.7 Control Charts for Quality Control Standards

All data generated for quality control standards, blanks and duplicates are retained with the client's file and are used in the validation of results. For each quality control standard, control charts are produced to monitor the performance of the laboratory. Warning limits are set at $\pm$2 standard deviations, and control limits are set at $\pm$3 standard deviations. Any data points for the quality control standards that fall outside the warning limits, but within the control limits require 10% of the samples in that batch to be reassayed. If the results from the reassays match the original assays the data are validated, if the reassay results do not match the original data the entire batch is rejected and new reassays are performed. Any quality control standard that falls outside the control limits is automatically reassayed and all of the initial test results are rejected. Any result that appears to be outside these criteria on the control charts provided below have already been reassayed as part of our internal quality control system.

13.2.8 Standards

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

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 standards. The quality control standards were used to monitor the processes involved in analyzing the samples. The quality assurance 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 standard or quality assurance standard may not be listed by job number on the control charts, a standard and quality assurance sample was run with each job.

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.

14.0 DATA VERIFICATION

14.1 2008 DATA VERIFICATION

A discussion of quality control measures and data verification procedures applied to the 2007 and 2008 drilling campaigns and the resultant databases is well documented in the previous Micon technical report entitled 'Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada' dated February 2, 2009. (Micon International Limited, 2009).

14.2 2009 DATA VERIFICATION

The data verification completed by Micon in 2009 was carried out in two stages, i.e., the site visit to the project area and repeat analyses on independently selected pulps.

14.2.1 Site Visit

Micon (represented by Charley Murahwi, P.Geo.) conducted a site visit to the Marathon PGM-Cu project area in Marathon from 16 to 17 October, 2009, and accomplished the following tasks.

- Verification of topography and all the 2009 drill hole collar positions in the company of Rod Swire, Field Manager for Marathon PGM.
- Review of property geology and mineralization on outcrops.
- Review of the drill core logging and sampling procedures.
- Review of facilities and security arrangements in place for samples and drill cores.
- Visual verification of mineralized intercepts in drill hole cores.
- Verification of lithological units of the complex as revealed in drill cores.

The main observations arising from Micon's site visit are listed below:

- The landscape is of a rugged nature and thus, a digital terrain or elevation model (DTM) is critical for resource estimation.
- Mineralization in the form of chalcopyrite and other sulphides not identifiable from hand specimens is, in some cases, located at surface as observed on rock exposures. This bodes well for an open pit type operation as the stripping ratio would be minimal.

- Standard logging and sampling procedures are in place and this ensures consistency in the construction of the resource database.

Under the guidance of Tracy Armstrong, P.Geo., (of P&E Consultants), a QA/QC program, initially instituted in 2006 was maintained throughout 2007, 2008 and 2009. Uncertified property standards named APG1 and APP7, as well as the CANMET certified standard WMG-1, were used as reference materials. In mid-2007, the supply of APP7 was exhausted and was replaced by another property standard, APG6. One standard sample, one blank sample and one field duplicate sample (¼ split core) were inserted into each batch of samples. In addition, Accurassay inserted its own internal QC samples which included standards, blanks and both coarse reject and pulp duplicates. Follow-up on the performance of control samples (standards and blanks) has been achieved through the use of control charts and, when necessary, re-submission of failed batches. Marathon PGM conducted the monitoring on a real time basis.

Marathon PGM maintains adequate security measures at its core storage and sampling facilities by keeping them under lock and key and restricting access to authorized personnel only

Based on observations made at site, Micon is satisfied that the database used in this resource estimate was generated in a credible manner.

14.2.2 Repeat Analyses on Sample Pulps

Micon selected 11 sample pulps (assay splits) and re-numbered them in a different sequence using a new set of sample numbers. The samples were re-submitted to the Accurassay laboratory in Thunder Bay for repeat analyses. The original assays and repeat analyses are compared in Figures 14.1 through 14.3.

Figure 14.1
Comparison of Original Cu Assays Versus Repeat Assays on Sample Pulps

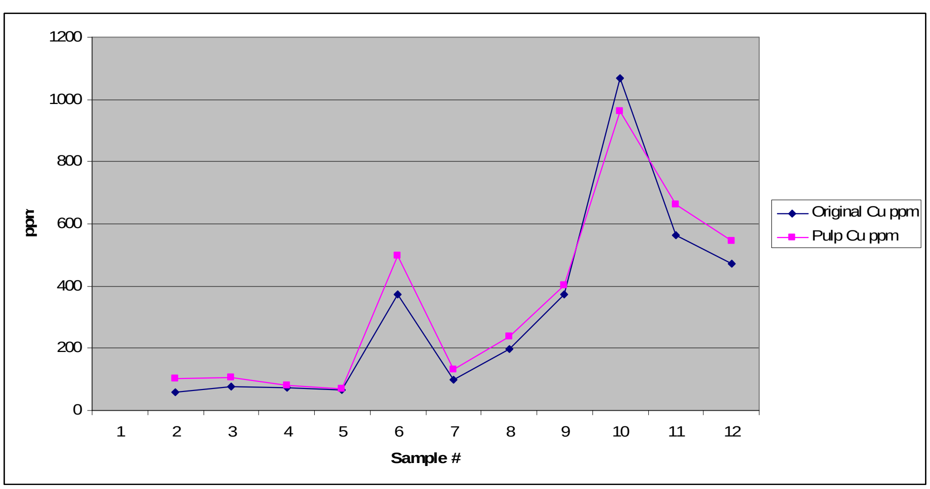


Figure 14.2
Comparison of Original Pt Assays Versus Repeat Assays on Sample Pulps

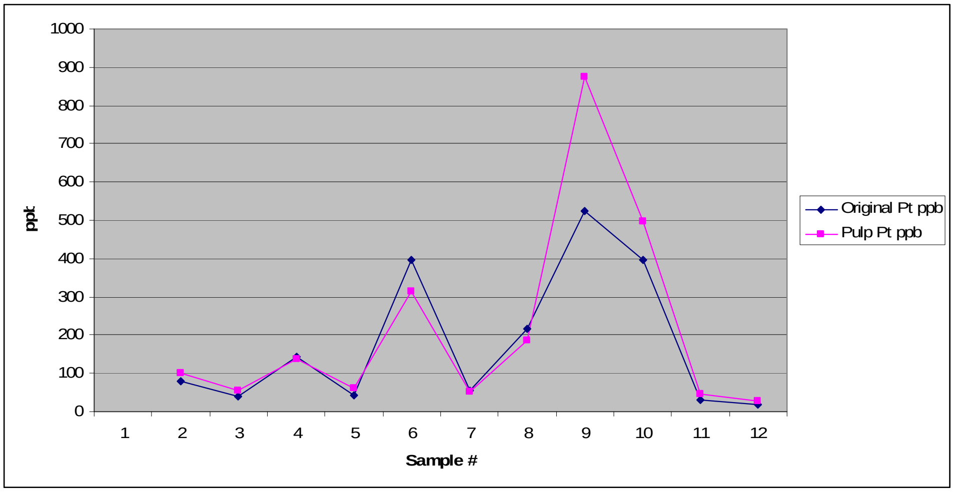
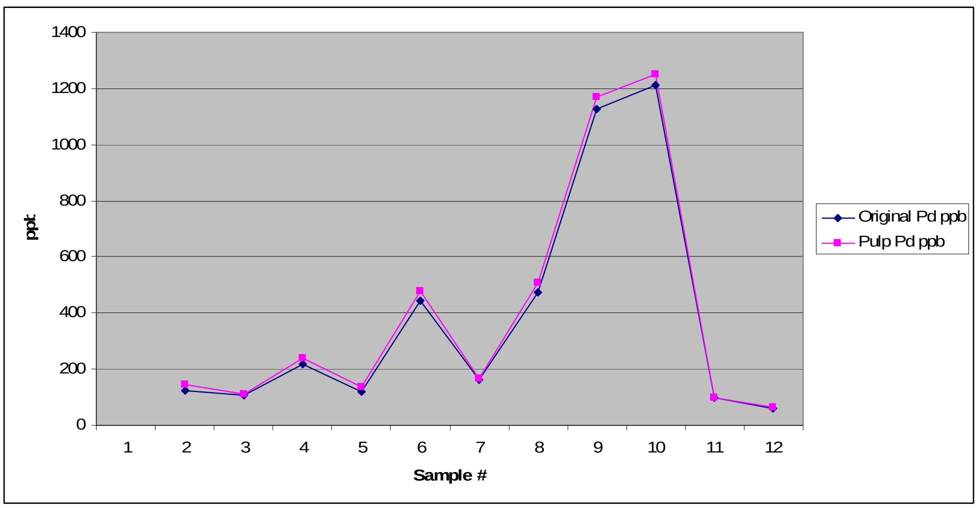


Figure 14.3
Comparison of Original Pd Assays Versus Repeat Assays on Sample Pulps



The results show a good degree of accuracy and reproducibility by Accurassay and this gives credibility to the assay database.

15.0 ADJACENT PROPERTIES

15.1 INTRODUCTION

Since its acquisition of the Marathon PGM-Cu property in January, 2004, Marathon PGM has systematically added to its land position through the periodic optioning, purchasing and staking of adjacent lands. The PGM-Cu mineralization appeared to extend onto some of these lands and had been the subject of drilling by Anaconda in the 1960s. A 12-km strike length of the mineralized trend that runs along the contact between the intrusive gabbros of the Coldwell Complex and the older volcanic and sedimentary rock is now covered by the land package controlled by Marathon PGM. From the initial 1,600 ha, consisting of two Crown leases and three unpatented mining claims, the property has been expanded to five Crown leases and 32 unpatented mining claim blocks for a total of 5,740 ha.

15.2 REGIONAL PROPERTIES

The Marathon PGM-Cu deposit is one of two contact-type PGM-Cu deposits in the Coldwell Complex that have been described in the literature (Good and Crocket, 1994). The second is the Geordie Lake palladium-copper property which Marathon acquired in 2008. Another similar deposit, also located in the Hemlo-Schreiber greenstone belt but not within the Coldwell Complex, is the Nickel Royale group of properties controlled by NovaWest Resources Inc. (NovaWest). The Thunder Bay North property of Magma Metals Limited (Magma Metals) is located within the Current Lake intrusive complex.

Although the Lac des Isles deposit, owned and operated by North American Palladium Ltd., is geographically related to and has some similarities with the Marathon PGM-Cu deposit, there are many dissimilarities, including age of formation (2.69 Ga for Lac des Isles compared with 1.1 Ga for Marathon PGM-Cu), dominant ore textures, and overall style of mineralization and metal ratios.

The Marathon PGM-Cu deposit contains mineralization textures that are considered fairly typical of contact style mineralization, while textures of the Lac des Iles deposit display some fundamental differences to that type of deposit. The Marathon PGM-Cu deposit is very fresh and coarse grained when compared with Lac des Isles. The Lac des Isles deposit is metamorphosed and hydrothermally altered, which translates to a significant difference in metallurgy. Despite the lower palladium grade in the Marathon PGM-Cu deposit, recoveries are similar to Lac des Isles due to the differences in alteration and texture.

The Lac des Iles deposit is not localized near the contact between the host intrusion and the country rocks and evidence of the assimilation of the host rocks is entirely lacking. Instead, the mineralization at Lac des Iles has many features in common with layered intrusion-hosted deposits, in which pulses of primitive magma introduced the PGM. However, unlike the quiescent magma chambers of most layered deposits, the magmas at Lac des Iles were intruded energetically, forming breccias and magma mingling textures.

The mineralization at Lac des Iles has less Pt with respect to Pd, compared to the Marathon PGM-Cu deposit and most other PGM deposits. With Pd:Pt ratios of 10:1, Lac des Iles stands in marked contrast to other deposits in the general vicinity (e.g., the Marathon PGM-Cu and Nickel Royale deposits) where Pd:Pt ratios average approximately 4:1.

15.3 GEORDIE LAKE DEPOSIT

The Geordie Lake property is located about 14 km west-northwest of the town of Marathon. Geologically, the property is underlain by syenite and gabbro phases of the Proterozoic-age Coldwell Complex. Drilling and surface mapping has defined a small, locally well mineralized, gabbro/troctolite body trending north-south named the Geordie Lake Intrusion. The intrusion, as mapped to date, is about 3 km long, from 30 to 700 m wide and dips from 30 to 60 degrees west. 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 platinum-group minerals and precious-metal tellurides, bismuthinides and alloys. In 2001, a series of metallurgical tests indicated average recoveries of 87% for Cu and 76% for Pd in mineralized zones.

Four main mineralized zones within the Geordie Lake Gabbro are recognized with each zone separated from the next by low grade mineralization. Utilizing a cut-off NSR value of $10, Giroux and Stanley (2002) estimated a NI 43-101 compliant Indicated Resource of 24.4 Mt averaging 0.326% Cu, 0.537 g/t Pd, 0.007% Co, 0.011% Ni, 0.030 g/t Pt, 2.52 g/t Ag and 0.04 g/t Au, and an additional 5.4 Mt of Inferred material at an average grade of 0.36% Cu, 0.626 g/t Pd, 0.007 % Co, 0.012% Ni, 0.04 g/t Pt, 3.04 g/t Ag and 0.05 g/t Au.

15.4 NICKEL ROYALE PROPERTIES

The Nickel Royale group of properties, controlled by NovaWest Resources Inc., encompasses 203 claim units, identified as the Nickel Royale #1, Nickel Royale #2, Alibaba, Four Sox, Solano and Solurus properties. They are situated east of Thunder Bay, Ontario, in the western portion of the Hemlo-Schreiber greenstone belt, which also contains the Marathon PGM-Cu deposit to the east. The property is easily accessible by road and lies 14.5 km west of the town of Schreiber and 10 km from the Winston Lake polymetallic operation owned by Inmet Mining Corporation.

The Nickel Royale properties straddle the northern contact of the Hemlo-Schreiber greenstone belt with the Crossman granitoids to the north. At present the known mineralization appears to be related to a presumed, younger, south dipping (35-40^{o}), layered intrusive complex involving layered ultramafic rocks and various phases of gabbro.

The last significant drilling was conducted by Nicohal Mines in 1970 and proved the continuity of one Ni-PGM-Cu zone to at least 400 ft (122 m) down dip with a surface width of 7 ft (2.1 m) and a strike length of 450 ft (137 m). It was concluded that the grade of about 1.0% Ni and 0.03% Cu, over 5 to 15 ft (1.5-4.5 m) showed indications of improving at depth.

The base metal sulphide mineralization, which occurs as massive to net-textured and vein-like accumulations, is comprised of pyrrhotite-chalcopyrite-pyrite-pentlandite. The sulphide textures are reminiscent of ore bodies that have undergone extensive remobilization and relocation of the sulphides (NovaWest, 2005). The remobilization of the precursor sulphides probably explains the origin of the mineralized granite footwall rocks.

As reported by NovaWest (2005) the massive sulphides exhibit an average Ni:Cu ratio of approximately 3:1. The dominant Pd:Pt ratio is 4:1 with facies ranging up to 9:1.

Emplacement of the sulphides is believed to have occurred in association with the development of the more extensive, rift-related structures related to the mid-continental rifting. In overview, the Nickel Royale mineralization may have age and/or compositional similarities to deposits in the Duluth and Coldwell Complexes.

15.5 THUNDER BAY NORTH

The Thunder Bay North property of Magma Metals is located approximately 50 km north-northeast of Thunder Bay and covers an area of approximately 700 km^2.

Diamond drilling on the northwestern part of the Current Lake intrusive complex formed the basis for an initial mineral resource estimate announced in September, 2009 comprising indicated mineral resources of 4.6 Mt at 1.35 g/t Pt, 1.27 g/t Pd, 0.32% Cu and 0.22% Ni and inferred mineral resources of 3.6 Mt at 1.06 g/t pt, 1.00 g/t Pd, 0.26% Cu and 0.19% Ni. Magma Metals has initiated a preliminary assessment of the project.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 METALLURGICAL TESTWORK

The updated feasibility metallurgical flowsheet and process design criteria are based on a program of flotation circuit optimization testing, including a mini pilot plant (MPP) run in April, 2009, at Xstrata Process Research (XPS), Sudbury, Ontario and a detailed program of metallurgical testwork undertaken by SGS Lakefield Research (SGS-L) at Lakefield between March, 2007 and March, 2008. This work is complemented by a substantial amount of historical work ranging from the 1960s.

Other testwork completed for the feasibility includes two pilot scale programs to test the suitability and gather scale-up data for high pressure grinding roll technology. This work was undertaken at the testing facilities of KHD Humboldt Wedag GmbH (KHD) located near Cologne, Germany.

16.1.1 Historical Testwork

Since the 1960s, there have been a number of metallurgical test programs carried out on Marathon PGM-Cu mineralized samples. This work includes investigations by Anaconda's Extractive Metallurgy Research Division (EMRD) between 1965 and 1972, Inco in the 1960s, Lakefield Research (now SGS-L) for Fleck Resources (Fleck) in 1985, and Bacon Donaldson & Associates (Bacon Donaldson) for Fleck in 1987.

16.1.1.1 EMRD – 1960s

During the mid 1960s, EMRD completed a series of metallurgical batch and pilot flotation plant tests. These tests included around 70 batch tests using drill core samples, 21 pilot plant tests using four bulk samples from surface trenches and eight pilot plant tests using a 23-T sample, reported by Anaconda as "taken from a single pit on line 262N in the main ore zone".

The results from the eight EMRD pilot plant tests indicated a recovery of over 90% for copper, around 80% for precious metals and approximately 60% for nickel into a concentrate grading 22% Cu. The average feed grade of the bulk samples was 0.72% Cu, 0.063% Ni, 1.13 g/t Pd, 0.24 g/t Pt, 5.1 g/t Ag and 0.10 g/t Au.

In the latter half of 1966, five duplicate samples prepared by Anaconda from drill core were tested by EMRD and Inco's testing facility at Copper Cliff, Ontario. Sample grades varied from 0.44% Cu to 0.64% Cu. The objective of this exercise was to determine recoveries to a relatively low grade cleaned concentrate. Test results were reported as follows:

- EMRD: Recoveries for copper ranged from 75% to 88% for concentrates grading from 10% Cu to 14% Cu. Palladium recoveries ranged from 71% to 83%.

- Inco: Higher copper recoveries were obtained, ranging from 92% to 96% for Cu and 75% to 84% for Pd. The reported concentrate grades produced were from 8% to 12% Cu.

It was noted by Anaconda at the time that the results from EMRD were from open circuit cleaner batch tests and the recycle of cleaner tailings (process solids) would significantly improve the recoveries. This point was postulated as the main reason for the differences in flotation performance.

In 1972, EMRD prepared a summary report which included projected recoveries of ≥90% for Cu, 72% for Pt and 80% for Pd, in a concentrate containing 15% Cu.

16.1.1.2 Historical Lakefield Research Testwork – 1980s

In 1985, Fleck provided four samples of fresh drill core to Lakefield for metallurgical investigations. Bench scale tests on one of the samples showed a significant Pd recovery to concentrate grade relationship. The Pd recoveries ranged from 88% in a concentrate grading 10% Cu, 82% recovery in a concentrate grading 15% Cu and 72% in a concentrate grading 21.6% Cu.

A closed circuit Bond ball mill test on a sample of Marathon PGM-Cu material gave a Bond ball mill index of 16.2 kWh/t (metric).

In early 1986, Lakefield also carried out pilot plant testing on two bulk samples with an average grade of 0.47% Cu, 1.85 g/t Pd, 0.40 g/t Pt, 0.037% Ni, 2.90 g/t Ag and 0.27 g/t Au. The weight of the two bulk samples were 16 t and 24 t, respectively. The combined composite was prepared by blending equal amounts of the crushed samples.

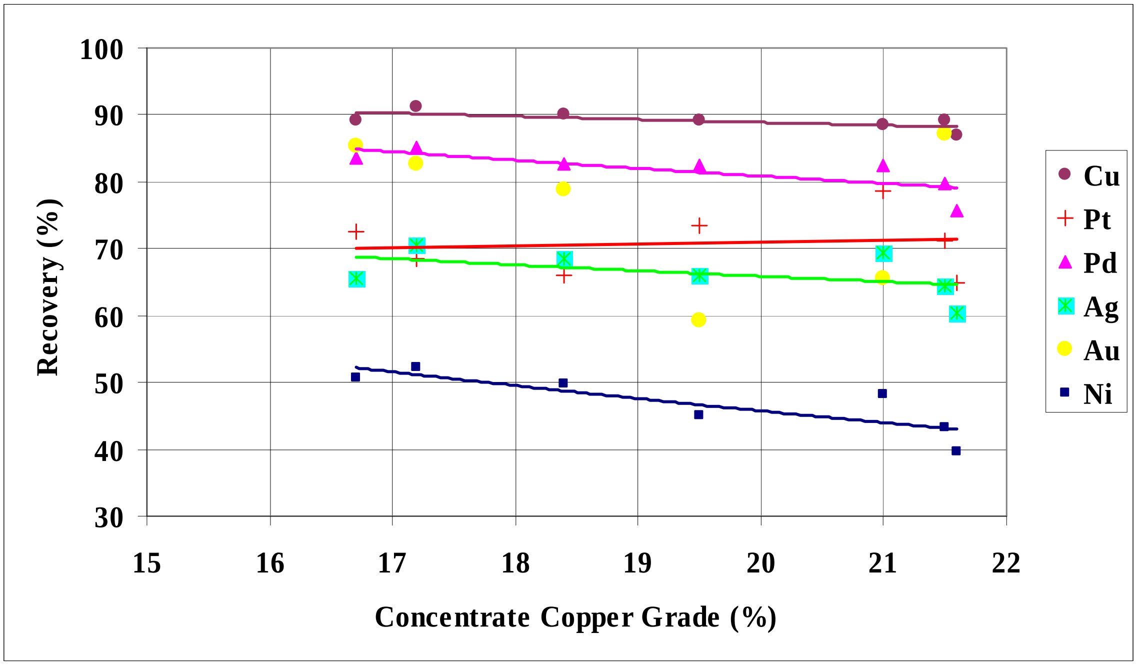
A total of 16 pilot plant runs were undertaken using 11 different operating conditions. Variables considered included primary grind size, regrinding of scavenger concentrates and reagents (lime, copper sulphate, sodium silicate and CMC). Lakefield reported that the results from these pilot plant tests indicated that the following results (Table 16.1) can be obtained using a relatively simple flowsheet involving a regrind and three cleaning stages.

Table 16.1
Lakefield Research 1986 Pilot Plant Results

Element	Feed Assay (g/t, %)	Concentrate Assay (%)	Recovery (%)
Copper	0.47	21	89
Nickel	0.037	0.9	43
Palladium	1.85	79	80
Platinum	0.4	15	71
Silver	2.90	94	65
Gold	0.27	7	80 (estimate)
Rhodium	-	1.23	-

A results for pilot plant test runs 5A, 5B, 6A, 6B, 7, 8A and 8B are presented in Figure 16.1.

Figure 16.1
Lakefield Research Pilot Plant Flotation Test Results



The reagents used for the tests that achieved the best results included potassium amyl xanthate (PAX) (59 g/t), Dowfroth 250 (23 g/t) and carboxy-methyl-cellulose (CMC) (196 g/t) for the rougher and scavengers and nothing added to the cleaners. The primary grind used for most of the tests was 73% passing 200 mesh and scavenger regrind 95% passing 400 mesh.

Lakefield carried out additional bench-scale tests using the pilot plant feed sample, and conducted concentrate thickening and vacuum filtration tests to determine design parameters for dewatering equipment.

Standard thickening tests showed that with a 10g/t addition of a non-ionic flocculant (Percol 351/Magnafloc 351) the concentrate had a settling rate of 3.47 m/h and a theoretical thickening area requirement of 0.04 m^2/t of dry solids per 24 hours. The estimated thickener underflow density was 64.1% solids by weight.

16.1.1.3 Bacon Donaldson Metallurgical Program – 1980s

Bacon Donaldson conducted pyrometallurgical (roasting) and hydrometallurgical (chemical leaching) tests on a Marathon PGM-Cu flotation concentrate provided by Fleck in 1987. The objective of this program of tests was to determine if the PGM grade of the concentrate could be increased.

The roasting test was unsuccessful but two ferric/cupric chloride-leaching processes were successful in reducing quantity of concentrate with little loss of PGM. This hydrometallurgical process removes copper and iron from the concentrate and leaves the PGM in the residue. Bacon Donaldson noted, however, that further research was required to prove out the validity of this process.

16.1.2 Mineralogical Investigations

A number of general and PGM specific mineralogical investigations have been conducted on samples of Marathon PGM-Cu mineralization.

During the initial testwork program undertaken by SGS-L in 2004, two investigations on the samples were completed. One investigation looked into the petrography of the Marathon PGM-Cu mineralization. The second investigation included an Automatic Digital Imaging System (ADIS) study on a sample of the overall composite, ground to 80% passing (k_{80}) of 100 µm.

As part of a metallurgical confirmatory study undertaken by XPS in 2008, a mineralogical study was completed on typical mineralization to characterize the modal mineralogy and to investigate the liberation characteristics of sulphide and potential diluent phases.

A number of detailed mineralogical investigations have been conducted by Dr. Ruslan Liferovich of Lakehead University Mineralogy and Experimental Laboratory (LUMINX), Thunder Bay, Ontario. (See Liferovich, 2006, 2007a and 2007b, and LUMINX, 2006. These studies involved the identification and mineralogical distribution of PGM contained within a series of samples collected from the Marathon PGM-Cu deposit.

16.1.2.1 SGS-L Mineralogical Studies – 2004

The SGS-L petrographic study and AIDS investigation, which used an automated microscope and image analysis software to recognize and measure particle size, composition and degree of liberation investigations, suggested that the Marathon PGM-Cu mineralization shows a bimodal copper distribution, in which most of the chalcopyrite (the main copper mineral) is relatively coarse which, being softer than the silicates, tends to grind finer than the average size distribution. The secondary occurrence of chalcopyrite is as very fine blebs, locked with other sulphides and silicates. Liberation of this fine fraction will require fine regrinding. Fine regrinding will also probably be required for liberation of the PGM minerals.

16.1.2.2 XPS QEMSCAN Analysis – 2008

As part of a confirmatory testwork program, XPS conducted a mineralogical analysis of a sample of Marathon PGM-Cu mineralized material using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscope).

The sample submitted to XPS comprised a composite of 34 coarse assay rejects. The grade of the composite sample was close to the average mineral resource grade.

The objectives of this work were to measure modal mineralogy, liberation characteristics of sulphide and potential diluent phases, quantify Mg deportment and to characterize PGM mineralogy within a sized sample of sample. The observations and conclusions derived from this study are summarized below:

- Copper mineralogy is dominated by chalcopyrite. Trace amounts of cubanite were identified, generally associated with chalcopyrite mineralization.
- Cu/Fe sulphide particles (mainly chalcopyrite) are moderately well liberated at the primary grind size of 80% passing 110 µm. A total of 77.1% of all Cu/Fe sulphide occurs as liberated particles.
- Cu/Fe sulphides in locked and middling particles occur most often in size fractions greater than 53 µm. Grain size averages of Cu/Fe sulphides within the locked and middlings particles vary between 11 and 47 µm. It is suggested that regrind should target a size of between 15 to 25 µm in order to liberate the majority of these locked and middlings particles.
- A total of 88.7% of pyrrhotite in the feed is liberated at the primary grind size of 80% passing 110 µm and has a fairly coarse size distribution.
- Minor pentlandite (0.09%) occurs in the sample. It is associated with both pyrrhotite and Cu/Fe sulphide.
- MgO content within the sample is 6.9% (4.2% Mg). The Mg deportment is dominated by clinopyroxene (augite) and amphibole (actinolite), with lesser amounts of Fe/Mg olivine, orthopyroxene, chlorite, serpentine, biotite and talc.
- Mg silicates are well liberated (91.5%) at the primary grind size of 80% passing 110 µm, which suggests that normal gangue depressants should be effective in minimizing MgO contamination in the flotation concentrate. Cu/Fe sulphide association data suggest that about 5% of the Cu/Fe sulphides are associated with Mg silicates, therefore some Cu losses may occur by depressing these binary particles. These losses may be mitigated with regrinding.
- Several PGM minerals were identified using bright phase searches on QEMSCAM. These included froodite ($PdBi_2$), sperrylite ($PtAs_2$) and at least one other Pd bearing species too small to identify by name. Manual searches also identified a Pd-Bi-Te mineral, possibly michenerite and trace amounts of Bi sulphide, Bi telluride and Pb telluride.

16.1.2.3 LUMINX PGM Mineralogical Investigations

Three detailed mineralogical reports produced by LUMINX, comprise the following:

- Interim Mineralogical Report on the Occurrence of Platinum Group Minerals from the Marathon PGM-Cu project, LUMINX, Dr. Ruslan Liferovich, March 18, 2006.

- Analysis of Size and Volume Fractions Distribution (In-Situ Granulometric Analysis) of PGM in Gabbroic Rocks of the Main Ore Zone, Two Duck Lake Deposit, Marathon, NW Ontario, Dr. Ruslan Liferovich, October 15, 2006.

- Mineralogy of PGE, Gold and Silver in the Malachite Ore Zone, Two Duck Lake Deposit, Coldwell Complex, NW Ontario, Dr. Ruslan Liferovich, March 12, 2007.

The mineralogical work undertaken by LUMINIX suggests that typically, the PGMs occurring in the Marathon PGM-Cu mineralization are less than 30 µm in size, around 80% being less than 10 µm. Approximately 30 to 50% of these minerals occur at the sulphide mineral- (or altered sulphide mineral)-silicate boundary. About 12 to 20% are hosted by sulphides or hematite, about 4 to 9% occur as liberated PGM particles or PGM aggregates and around 20 to 57% are associated with silicates, mainly chlorite and serpentine, but also plagioclase.

16.1.3 Feasibility Study Metallurgical Testwork

As part of the Marathon PGM-Cu Feasibility Study, a detailed metallurgical test program was completed by SGS-L in March, 2008. Further work was undertaken by XPS in late 2008 and early 2009 in order to improve the flotation circuit performance and to operate a mini pilot plant (MPP). The MPP results are used as a basis for the metallurgical parameters used in the updated Feasibility Study economic evaluation. XPS also conducted a mineralogical analysis of the Marathon PGM-Cu feed sample using QEMSCAN.

In addition to the metallurgical work undertaken by SGS-L, two HPGR pilot scale programs were completed by KHD at its testing facilities located near Cologne, Germany. This work was undertaken to test the suitability and gather scale-up data for high pressure grinding roll technology.

16.1.4 Metallurgical Samples

Samples for the Feasibility Study metallurgical testwork programs were selected by Marathon PGM. Samples used for the development program, which provided quantitative design data, were drill core composites selected to represent typical Marathon PGM-Cu mineralization.

In addition to the metallurgical samples used for quantitative metallurgical development and design, two bulk samples were prepared for HPGR pilot plant testing.

16.1.4.1 SGS-L 2007/08 Test Program Samples

Drill core for the Feasibility Study metallurgical program was selected, packaged and transported to SGS-L in March, 2007. The drill core, from which the master composite test sample and variability samples were prepared, included material from holes M06-207, M06-210, M05-95 and M05-112. Large diameter core from hole number M07-300 was also included in this batch of samples. This material was used for grindability investigations.

The small drill core shipped in March, 2007, was used to produce six grindability composites and six grade composites for metallurgical testing, including a master composite, which represented the average resource in terms of lithology and grade.

The six grade composites were based on one overall master composite that would have a grade close to the deposit average and two higher grades and two lower grades. In addition, a separate set of drill core was provided by Marathon PGM from a separate area, in which the PGM values were significantly higher, but contained much less copper. None of these Hi-PGM cores were used for any of the other composites. The two drill cores used for the Hi-PGM grade sample were M-06-186 and M-06-183.

In addition to these grade composites prepared from core delivered to SGS-L in March, 2007, an additional two composites, prepared to supplement the original main composite, (main composites #2 and #3) were blended from crushed drill core in October, 2007. Table 16.2 presents the grades of the Feasibility Study metallurgical sample composite head assays.

There are 21 lithologies identified at the Marathon PGM-Cu deposit. The samples used for the preparation of the main grade composite comprised six lithologies combined in the same proportions as the deposit. These proportions were:

- 3b (Coarse grained gabbro [>5mm]) 60%
- 3d (Coarse grained to pegmatitic gabbro) 25%
- 2d (Fine grained gabbro with coarse grained gabbro dikelets) 8%
- 1a (Footwall breccia [RIB]) 3%
- 2a (Fine grained gabbro) 3%
- 3c (Coarse grained gabbro with leuco pods) 1%

The grindability samples were selected based on the lithology. The six grindability samples prepared from the small drill core shipped in March, 2007, represented the six lithologies listed above.

In June, 2007, 36 test composite samples were prepared at the project site and shipped to SGS-L for the grinding variability testwork program.

Table 16.2
2007/08 SGS-L Metallurgical Composite Grades

Element/ Compound	Main Comp	Variability Composites					Main Comp#2/#3
		Lo	Lo-Lo	Hi	Hi-Hi	Hi-PGM	
Cu (%)	0.28	0.22	0.17	0.31	0.49	0.12	0.25
Ni(T) (%)	0.021	0.018	0.021	0.020	0.036	0.018	0.024
Ni(S) (%)	0.021	0.016	0.014	0.018	0.030	0.008	0.019
S (%)	0.59	0.46	0.63	0.86	0.80	0.090	0.51
Au (g/t)	0.090	0.080	0.060	0.090	0.15	0.72	0.120
Ag (g/t)	1.60	0.90	1.10	1.80	2.35	< 0.5	1.10
Pd (g/t)	0.80	0.70	0.51	0.97	1.76	12.9	0.79
Pt (g/t)	0.19	0.18	0.100	0.56	0.40	3.79	0.17
Rh (g/t)	0.02	< 0.02	< 0.02	< 0.02	0.04	0.41	0.02
SiO_2 (%)	46.8	47.2	47.4	46.4	46.4	47.5	46.6
Al_2O_3 (%)	14.7	15.0	14.4	14.3	15.7	15.6	15.2
Fe_2O_3 (%)	12.8	12.7	13.2	15.0	12.5	10.9	12.9
MgO (%)	7.21	6.90	6.76	6.27	6.56	7.84	7.47
CaO (%)	12.0	12.0	11.9	11.9	12.3	13.6	12.1
Na_2O (%)	2.41	2.50	2.53	2.47	2.45	2.32	2.44
K_2O (%)	0.44	0.50	0.51	0.52	0.48	0.35	0.43
TiO_2 (%)	0.94	0.99	1.04	1.28	0.88	0.75	0.83
P_2O_5 (%)	0.28	0.38	0.44	0.47	0.21	0.35	0.25
MnO (%)	0.18	0.18	0.19	0.19	0.17	0.18	0.18
Cr_2O_3 (%)	0.040	0.030	0.030	0.020	0.020	0.030	0.040
V_2O_5 (%)	0.060	0.060	0.060	0.080	0.050	0.050	0.050
LOI (%)	1.21	1.08	1.03	1.07	1.38	1.34	1.42
Weight kg	150	21	26	35	21	21	

16.1.4.2 XPS 2008/09 Test Program Sample

The sample used for the optimization study work at XPS was selected by Marathon PGM and comprised approximately 3 t of relatively fresh drill core. The sample was crushed, blended, split, sampled and stored at XPS. The analysis of this sample is shown in Table 16.3.

Table 16.3
2008/09 XPS Composite Metallurgical Sample

Analyses	Cu (%)	Ni (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	MgO (%)	S (%)
Mean	0.322	0.031	0.092	0.225	0.809	0.023	6.733	1.072
Std. Dev.	0.041	0.003	0.016	0.041	0.106	0.008	0.094	0.066

A total of 46 drill holes were used for this sample. These drill holes were selected to represent the mineral resources spatially and in terms of average grade for Cu and PGMs.

16.1.4.3 HPGR Pilot Plant Test Samples

Two composite samples were prepared in at the project site for HPGR testing at KHD's factory in Cologne, Germany. The first sample comprised approximately 3.5 t of lump material, nominally minus 40 mm in size. This material originated from the main pit area and was excavated from surface trenches. The first program of HPGR pilot testwork was undertaken in November, 2007.

A second sample, used to confirm the results of the November, 2007, pilot test program, comprised 1.3 t of relatively fresh split drill core. This sample was selected by Marathon PGM to approximate the average grade of the economic mineralization with the proviso that the crushed product could be used for additional metallurgical testing. This second confirmatory test program was completed in May, 2008.

16.1.5 SGS-L 2004 Test Program

A preliminary test program was completed in 2004 on a composite sample selected from four drill holes. This testwork program was a pre-cursor to the more detailed study undertaken by SGS-L in 2007 and 2008.

The scope of this testwork program comprised mineralogical studies with batch and locked cycle flotation tests for preliminary flowsheet development. The program also included variability testing of selected samples using the developed flotation flowsheet and conditions. This phase of testwork did not included grindability tests.

Two locked cycle tests were undertaken during the 2004 test program and the results are summarized in Table 16.4.

Table 16.4
2004 Metallurgical Locked Cycle Test Results

Test F017	Wt.%	Assays, %, g/t				% Distribution/Recovery		
		Cu	Pd	Pt	Insol.	Cu	Pd	Pt
Cu Prim Cl con	0.97	29.8	49.6	12.7	4.89	75.7	47.6	52.7
Cu Sec Cl con	0.48	14.4	68.6	11.2	14.4	18	32.3	22.8
Cu Sec Cl tail	12.4	0.11	0.53	0.27		3.72	6.6	14.5
Rough tail	85.4	0.008	0.09	0.015		1.79	7.6	5.5
Head calc	99.3	0.38	0.96	0.23		99.3	94	95.5
Combined Con[1]	**1.45**	**24.7**	**55.8**	**12.2**	**8.01**	**93.8**	**79.9**	**75.5**
Test F021	**Wt.%**	**Assays (%, g/t)**				**% Distribution/Recovery**		
		Cu	**Pd**	**Pt**	**Insol.**	**Cu**	**Pd**	**Pt**
Cu Prim Cl con	1.06	28.6	53.1	12.6	4.77	82.8	49.2	38.9
Cu Sec Cl con	0.34	10.5	38.1	10.2	14.2	9.9	11.5	10.2
Cu Sec Cl tail	12.9	0.16	0.8	0.36		5.72	9.1	13.5
Rough tail	85.1	0.006	0.085	0.03		1.4	6.3	7.4
Head calc	99.4	0.37	0.87	0.24		99.8	76.1	70
Combined Con[1]	**1.4**	**24.2**	**49.4**	**12**	**7.07**	**92.7**	**60.7**	**49.1**

[1] The combined concentrate represents the final concentrate grades and recoveries.

The interpretation by SGS-L suggested the following average recoveries for the relatively high grade metallurgical sample and the lower grade mineral resource estimate of June, 2006 for the P&E scoping study. (See P&E Mining Consultants Inc., 2006b). These estimated recoveries assume a concentrate product containing 25% copper.

Table 16.5
Estimated Recoveries from the 2004 Testwork Program

Metal	Metallurgical Composite		Scoping Study	
	Head Grade (%, g/t)	Recovery (%)	Head Grade (%, g/t)	Recovery (%)
Copper	0.37	92	0.31	86.6
Platinum	0.31	76	0.25	75.5
Palladium	1.22	80	0.91	79.9
Gold	-	-	0.09	60.0

16.1.6 SGS-L 2007/08 Test Program – Flotation

Using the 2004 test program as a base, SGS-L was engaged in February, 2007, by Marathon PGM, to undertake a detailed metallurgical development program. The objectives of this program of work were to develop a final optimized process flowsheet and to obtain metallurgical design data suitable for use in a Feasibility Study. This test program was completed in February, 2008. (See SGS Lakefield Research Limited, 2008).

The main focus of the test program was the optimization of the flotation process using batch rougher and cleaner flotation tests and to simulate this process using a series of locked cycle flotation tests. Batch variability flotation tests were also scheduled in order to examine the sensitivity of the flowsheet to changes of ore-type and grade.

Additional work included in the detailed testwork scope included a comprehensive grindability program, simulation and recommendations for comminution circuit sizing and equipment. A batch scoping scale amenability test was also included to assess the potential of the PLATSOL™ process for the recovery of valuable metals from flotation concentrates using hydrometallurgy.

The test program in 2004 exhibited good metallurgical results. However, SGS-L deemed that the occurrence of tight flotation froth was not suitable for full scale operation. Much of the development flotation work undertaken in the 2007/08 test program was geared to a successful resolution of this issue.

The SGS-L 2007/08 testwork program included a total of 95 flotation tests. The main areas investigated during flotation development were:

- The effects of the primary grind size.
- Collector selection, dosage and addition points. This was investigated in the initial rougher tests and then revisited when the cleaner tests gave conflicting results.

- Cleaner conditions. Collector, depressants and activator testing
- Regrinding requirements.
- Grade variability testing. Both rougher variability and cleaner variability were tested.
- Comparative process response using different samples. This included testing a sample from the previous test program in 2004.
- Comparative testing, using final protocol developed during the previous test program in 2004.
- Locked cycle tests (LCT) to provide Feasibility Study design data.

Based on the LCT results, SGS-L estimated the metal recoveries for Marathon PGM-Cu mineralization assuming metal grades approximating the mineral resource estimate at that time. These estimates are shown in Table 16.6.

Table 16.6
SGS-L Estimate of Metal Recoveries

Metal	Unit	Feed Grade	Concentrate Grade	Recovery (%)
Copper	%	0.28	22.0	91.0
Gold	g/t	0.11	6.53	73.0
Platinum	g/t	0.23	13.0	63.0
Palladium	g/t	0.87	57.0	77.0

In addition to the recovery estimates presented in Table 16.6, approximate recoveries for rhodium and silver were estimated using the average composite feed assays and final concentrate grades. These estimated recoveries are 46% and 77% for Rh and Ag, respectively.

16.1.6.1 Flotation Variability Tests

The flotation process developed by SGS-L was tested using a number of variability composite samples (see Table 16.2). These variability flotation test results suggested that Cu recovery is relatively insensitive to feed grade while Pt, Au and Pd recoveries tend to improve with respective higher head grades.

16.1.7 XPS Optimization Program

The XPS metallurgical testing program was initiated to optimize the metallurgical flowsheet and complement the extensive metallurgical work completed by SGS-L in 2008 that supported the Feasibility Study that was issued in December 2008. The XPS metallurgical test program comprised a number of bench scale open circuit and locked cycle tests, and a continuous 6-day MPP campaign which processed about 2,000 kg of crushed sample. The total composite sample supplied to XPS by Marathon PGM weighed approximately 3,000 kg.

The open circuit testwork was primarily focused on the optimization of the reagent suite. This series of tests used factorial design (also called Design of Experiment, or DOE) in order to optimize the reagents and their addition rate. One set of DOE tests developed a three-component collector mix that aimed to improve metal recoveries. Another limited factorial DOE series was performed to optimize frother selection and dosage.

An optimized collector suite of Aerofloat 3418A, PAX and Aerofloat 3477 was developed from this testwork. The conclusion from the collector optimization was increased copper and PGE recoveries. The frother DOE concluded that mixture of MIBC and Polyfroth W34 at similar dosage rates was the best arrangement for froth stability and metallurgical performance.

The open circuit testing also reviewed the need for regrinding. This work concluded that a target regrind of 30 µm of the primary cleaner tailings and the secondary rougher concentrate significantly improved the concentrate quality in terms of both the Cu and PGM grade.

Open circuit testing also included a review of the depressants Depramin C (a type of carboxy methyl cellulose, CMC) and sodium metabisulphite (SMBS). These tests suggested that these reagents will not benefit metallurgical performance at the primary cleaner stage. XPS opined that use of SMBS (pyrrhotite depressant) in the secondary cleaner stages would reduce PGM recoveries due to the depressing of PGEs that were associated with pyrrhotite.

During the locked cycle flotation testwork that followed the open circuit work, Hercules 7M, a gangue depressant was introduced in the place of Depramin C. A total of 4 locked cycle tests were undertaken by XPS. A problem with gangue recovery in the secondary cleaning stages that was only noted during LCT and not open circuit tests, was alleviated with the use of Hercules 7M. The summary of average results from the fourth LCT which used Hercules 7M is presented in Table 16.7.

Table 16.7
Summary of the XPS LCT 4 Average Results

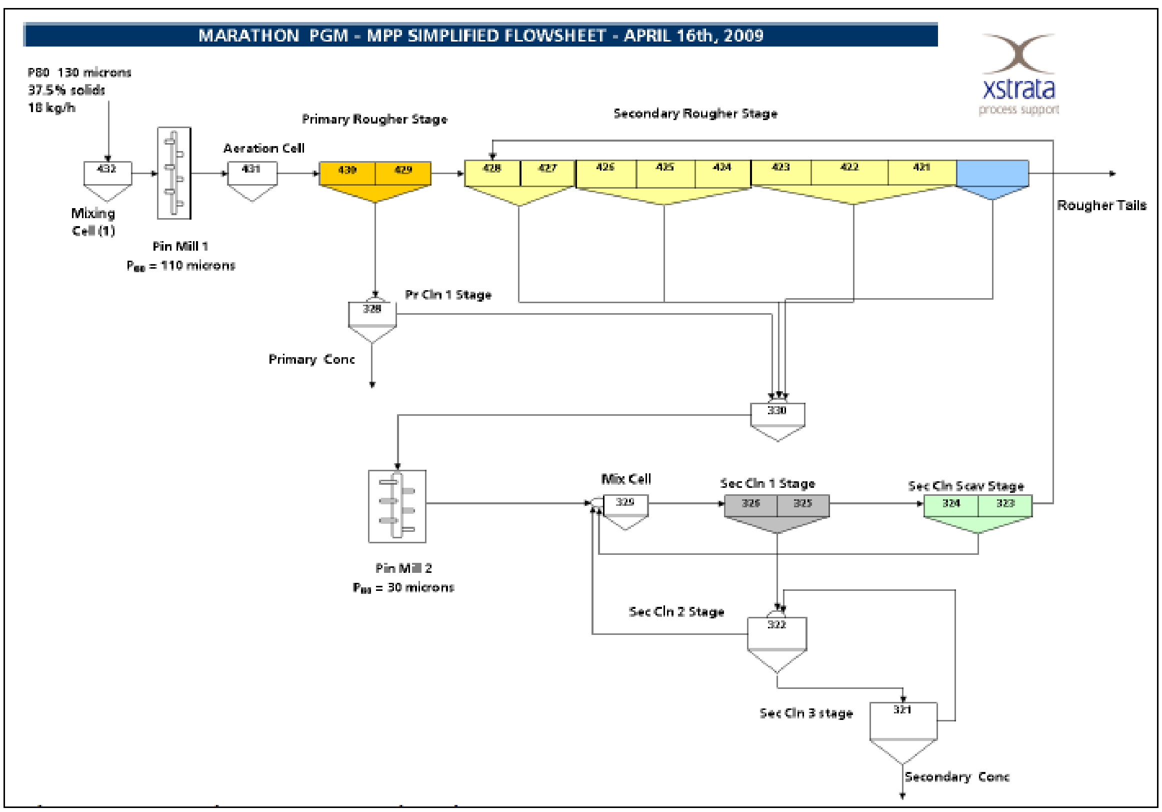
Metal	Unit	Calculated Feed Grade	Concentrate Grade	Recovery (%)
Copper	%	0.28	21.65	90.49
Gold	g/t	0.09	6.00	83.07
Platinum	g/t	0.24	15.46	80.99
Palladium	g/t	0.84	56.76	77.33
Sulphur	%	0.90	34.26	51.62
MgO	%	6.25	2.24	0.44

16.1.7.1 XPS Mini Pilot Plant

The MPP for Marathon PGM-Cu material was run by XPS on a continuous basis (24 h/d) at a rate of 18 dry kg/h, starting on Sunday, April 26 and finishing on Friday, May 1, 2009. No major downtime or mechanical related stoppages were experienced during this MPP run. Final product samples of concentrates and tailings were collected periodically throughout the test period and hourly composite samples were also collected for approximately 13 continuous hours. In addition, a single set of flowsheet internal stream samples were collected for internal mass balance purposes.

The flowsheet used for the MPP is presented in Figure 16.2. This flowsheet was based on the locked cycle testwork and used a target primary grind of 80% passing 110 μm and a regrind of 80% passing 30 μm.

Figure 16.2
XPS MPP Test Flowsheet



From Xstrata Process Support Phase II Bench Scale and Mini Pilot Plant Final Report.

The average metallurgical performance obtained during the MPP test is presented in Table 16.8.

Table 16.8
XPS MPP Average Metallurgical Results

Metal	Wt. (%)	Cu (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Ag (g/t)	Ni (%)	MgO (%)	S (%)
Rougher Feed	100.0	0.32	0.07	0.19	0.84	0.02	1.33	0.04	5.4	0.90
Primary Con. Grade	-	22.94	3.51	8.3	42.6	0.50	65.9	0.39	3.2	26.4
Secondary Con Grade	-	15.57	3.43	9.0	41.9	0.69	64.6	0.78	4.8	25.7
Final Concentrate Grade	-	18.75	3.47	8.7	42.2	0.61	65.1	0.61	4.1	26.0
Primary Con. Recovery	0.69	48.8	33.8	29.1	35.0	17.7	34.0	6.7	0.3	20.1
Secondary Con. Recovery	0.91	43.7	43.5	41.9	45.4	32.5	43.9	17.9	0.7	25.8
Total Recovery (%)	1.59	92.5	77.3	71.0	80.4	50.3	77.9	24.6	1.0	45.9

Average Cu recovery was 92.5% with tailings assays consistently between 0.02% and 0.03% Cu. The Pd recovery averaged 80.4% with tailings assays between 0.15 g/t and 0.18 g/t Pd. The Pt recovery averaged 71.0% with tailings assays between 0.05 g/t and 0.06 g/t Pt. The Au assays in the tailings were found to be around 0.016 g/t.

The average primary cleaner concentrate grade was 22.9% Cu, 3.2% MgO, 8.3 g/t Pt, 3.5 g/t Au, 42.6 g/t Pd, 0.5 g/t Rh and 65.9 g/t Ag. This concentrate was about 43% of the global concentrate mass and about 53% of the total copper recovered. The average secondary cleaner concentrate grade averaged 15.6% Cu, 4.8% MgO, 9.0 g/t Pt, 3.4 g/t Au, 41.9 g/t Pd, 0.7 g/t Rh and 64.6 g/t Ag. This concentrate is made up about 57% of the global final concentrate mass and approximately 47% of the total recovered copper. Pyrrhotite and MgO were the two main diluent minerals in this concentrate product.

The upper quartile of the primary and secondary cleaner concentrate grades along with upper quartile tailings grades were used to calculate the upper quartile metallurgical performance. The results are shown in Table 16.9. The results indicate that the recovery remains relatively constant when higher grade concentrate is produced. These results were similar to those obtained during the locked cycle tests.

Table 16.9
XPS MPP Upper Quartile Metallurgical Results

Metal	Wt. (%)	Cu (%)	Au (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Ag (g/t)	Ni (%)	MgO (%)	S (%)
Rougher Feed	100.0	0.32	0.07	0.19	0.86	0.02	1.33	0.04	6.3	0.87
Primary Con. Grade	-	24.84	0.01	15.4	65.2	0.63	74.0	0.32	2.6	29.4
Secondary Con Grade	-	22.01	6.92	9.3	50.4	0.69	71.7	0.66	3.5	28.8
Final Concentrate Grade	-	22.61	3.66	10.6	53.5	0.68	72.2	0.60	3.3	29.0
Primary Con. Recovery	0.27	21.0	26.8	21.7	20.6	8.9	15.4	2.2	0.1	9.2
Secondary Con. Recovery	1.01	69.7	53.1	49.2	59.5	36.4	56.0	17.6	0.6	33.6
Total Recovery (%)	1.29	90.8	79.9	71.0	80.1	45.3	71.4	19.8	0.7	42.8

These upper quartile MPP results are used as the basis for the metallurgical parameters used in the updated Feasibility Study financial evaluation.

16.1.7.2 Reagents

The flotation reagents used during the MPP test and that expected to be used based on the LCT work are compared in Table 16.10.

Table 16.10
Flotation Test Reagent Usage

Reagent	MPP Usage (g/t)	Expected Dosage (g/t)
MIBC-Polyfroth W34 blend	35.4	46.2
PAX	76.8	42.5
Aero 3477–Aero 3418A blend	34.7	30.0
Hercules 7M	273.3	194.4
Aero 3418A	11.6	10.2

For the Feasibility Study, the reagent consumption rates were estimated by applying a factor of 75% to the MPP usage rates. For the Aero 3477-Aero 3418A blend, Aerophine 3406 Promoter was used in the Study. This reagent is a commercially available blend of these two collectors.

16.1.7.3 Concentrate Quality

The final concentrate analyses, including minor elements that may incur penalties from smelting and refining companies, are presented in Table 16.11. This table presents analyses for the LCT concentrates produced in the SGS-L 2004 program and the 2007/08 program as well as the concentrate from the XPS MPP test. The two concentrate analyses from the 2007/08 program comprise an initial sample composited from a number of the LCT and a second sample composited from concentrate products. The second sample was required in order to provide sufficient sample to undertake a complete suite of assays.

The multi-element concentrate analyses suggest that there are no elements of concern in the final product that will affect its saleability.

Table 16.11
Final Concentrate Analyses

Element/ Compound	SGS-L 2004		SGS-L 2007/08			XPS MPP April, 2009	
	Head	Conc.	Head	Conc. 1	Conc. 2	Head	Conc.
Cu (%)	0.39	24.2	0.28	21.9	21.1	0.322	22.4
Ni (%)	0.037	0.89	0.021	0.52	0.49	0.031	0.53
Co (%)		0.14		0.060	0.059		0.076
Fe (%)	9.86	33.2	8.95	29.0	28.0		28.3
S (%)	1.12	30.4	0.59	24.1	24.8	1.072	29.0
Au (g/t)	0.08	5.84	0.090	6.63	7.6	0.092	4.7
Ag (g/t)		142	1.60	127	68	1	69
Pd (g/t)	0.99	55.9	0.80	67.9	57.8	0.809	50.2
Pt (g/t)	0.22	12.3	0.19	16.7	12.5	0.225	9.9
Rh (g/t)		0.76	0.02	0.95	0.8	0.023	1.0
Ru (g/t)		0.05		0.1	0.06		
Ir (g/t)		0.04		0.06	0.04		
SiO_2 (%)	45.5	4.61	46.8		13.9		11.2
Al_2O_3 (%)	14.8	1.69	14.7	2.84	2.65		1.17
MgO (%)	7.04	0.76	7.21	3.64	2.60	6.73	3.89
CaO (%)	12.7	1.12	12.0	1.96	1.96		0.8
Na_2O (%)	2.15	0.20	2.41	0.39	0.36		0.14
K_2O (%)	0.40	0.03	0.44	0.08	0.07		0.03
TiO_2 (%)	0.91	0.075	0.94	0.11	0.11		0.07
Cl (g/t)		61		84	70		
F (%)		0.014		0.025	0.028		
As (g/t)		<30		38	<30		26
Ba (g/t)		40		60	49		<50
Be (g/t)		<0.03		< 0.2	<0.08		<0.5
Cd (g/t)		8		10	<15		16.5
Cr (g/t)	400	40	270	44	18		40
Li (g/t)		<8		< 5	<5		4.0
Mn (g/t)	1,900	180	1,800	350	360		250
Mo (g/t)		<5		33.0	32		29.6
P (g/t)	3,300	<200	1200	< 200	<200		100
Pb (g/t)		<350		610	540		216
Sb (g/t)		<10		< 10	<10		5.6
Se (g/t)		<50		84	65		110
Sn (g/t)		<20		< 20	<20		4.0
Sr (g/t)		76		110	99		63
Tl (g/t)		<30		< 30	<30		14.2
U (g/t)				< 60	<40		1.0
V (g/t)	450	25	340	40	36		23
Y (g/t)		1.2		1.9	1.7		2.0
Zn (g/t)		860		1,200	920		1,240

16.1.8 Grinding Testwork

As part of the 2007/08 SGS-L metallurgical program, SGS-L performed extensive grindability testwork and used the data gathered from the tests to design conventional grinding systems capable of milling 22,000 t/d. SGS-L used the CEET2® and JKSimMet technologies, as well as Bond's third theory of comminution for the grinding circuit design. Seven 'large-core' and thirty-six variability samples, representing seven lithologies, footwall/waste (F/W) and 1a, 2a, 2d, 3b, 3d, and 4a, from the Marathon PGM-Cu deposit, were prepared and submitted by Marathon PGM for SPI and ModBond tests at SGS-L. Sample 3B was also submitted to JKTech drop-weight (DWT), MacPherson autogenous grindability and Bond abrasion tests. The results from these tests are summarized in Table 16.12.

Table 16.12
Summary of Grindability Test Results

Sample	% in	S.G.	DWT Parameters		SPI	18" Mill		Work Index (kWh/t)			AI
Name	Mine	g/cm^3	A x b	t_a	Min.	kg/h	kWh/t	AWI	ModBond	BWI	g
F/W	-	-	-	-	99.5	-	-	-	14.8	-	-
2A	1	-	-	-	97.1	-	-	-	16.0	-	-
2D	18	-	-	-	123	-	-	-	16.3	-	0.396
3B	57	3.03	46.0	0.36	62.3	10.2	8.3	14.5	16.4	-	0.347
3B#1	57	-	-	-	88.4	-	-	-	14.2	-	-
3B#2	57	-	-	-	83.0	-	-	-	16.4	-	-
3B#3	57	-	-	-	84.6	-	-	-	15.9	-	-
4a (1), 703924	17	-	-	-	40.6	-	-	-	15.6	-	-
4a (2), 703925	17	-	-	-	51.0	-	-	-	15.6	15.4	-
4a (3), 703926	17	-	-	-	65.8	-	-	-	15.3	-	-
2d (1), 703916	18	-	-	-	97.7	-	-	-	16.0	15.7	-
2d (2), 703918	18	-	-	-	110	-	-	-	16.6	-	-
2d (3), 703919	18	-	-	-	140	-	-	-	17.2	-	-
1a (1), 703920	1	-	-	-	135	-	-	-	14.8	15.1	-
1a (2), 703928	1	-	-	-	115	-	-	-	15.2	-	-
1a (3), 703931	1	-	-	-	79.0	-	-	-	13.8	-	-
3d (1), 703923	5	-	-	-	68.3	-	-	-	16.5	-	-
3d (2), 703927	5	-	-	-	47.8	-	-	-	16.1	-	-
3d (3), 703929	5	-	-	-	57.3	-	-	-	16.5	-	-
3d (4), 703930	5	-	-	-	73.7	-	-	-	16.9	-	-
3d (5), 703932	5	-	-	-	67.3	-	-	-	16.4	-	-
3d (6), 703933	5	-	-	-	62.0	-	-	-	15.9	-	-
3d (7), 703934	5	-	-	-	73.1	-	-	-	16.4	-	-
3d (8), 703935	5	-	-	-	64.1	-	-	-	15.3	-	-
3d (9), 703936	5	-	-	-	72.7	-	-	-	16.5	-	-
3b (1), 703901	57	-	-	-	83.4	-	-	-	15.5	-	-
3b (2), 703902	57	-	-	-	87.2	-	-	-	15.8	-	-
3b (3), 703903	57	-	-	-	102	-	-	-	15.7	-	-
3b (4), 703904	57	-	-	-	93.7	-	-	-	15.8	-	-
3b (5), 703905	57	-	-	-	81.1	-	-	-	15.7	-	-
3b (6), 703906	57	-	-	-	75.9	-	-	-	15.5	-	-
3b (7), 703907	57	-	-	-	87.2	-	-	-	15.7	-	-
3b (8), 703908	57	-	-	-	99.8	-	-	-	15.7	-	-
3b (9), 703909	57	-	-	-	82.7	-	-	-	15.6	-	-
3b (10), 703910	57	-	-	-	95.6	-	-	-	16.6	-	-
3b (11), 703911	57	-	-	-	87.5	-	-	-	16.5	-	-
3b (12), 703912	57	-	-	-	75.3	-	-	-	15.8	-	-
3b (13), 703913	57	-	-	-	92.8	-	-	-	16.7	17.2	-
3b (14), 703914	57	-	-	-	100	-	-	-	16.0	-	-
3b (15), 703915	57	-	-	-	102	-	-	-	16.4	16.2	-
3b (16), 703917	57	-	-	-	106	-	-	-	15.8	-	-
3b (17), 703921	57	-	-	-	94.4	-	-	-	16.6	-	-
3b (18), 703922	57	-	-	-	97.7	-	-	-	16.3	-	-

Table from 2008 SGS-L Grind Report March 28.

The grindability data were used by SGS-L to develop preliminary SAG and ball mill circuit studies using CEET2® and JKSimMet software. Several designs were completed by SGS-L using SABC (semi-autogenous mill/ball mill/crusher) and SAB (semi-autogenous mill / ball mill) circuit configurations. The recommended grinding equipment from this design exercise is shown in Table 16.13. The design considered two product sizes 80% passing product size, P_{80} of 120 and 85 µm.

The recommended grinding circuit comprised a SABC flowsheet and all of the recommended circuits shown in Table 16.13 included a 5-ft diameter pebble crusher.

Table 16.13
Summary of Recommended Grinding Circuit Mill Sizes

P_{80} = 125 µm	**JK SimMet (SABC)**		**CEET2 (SABC)**	
Nominal size (ft)	34 x 15	21 x 36	34 x 15	22 x 36
Design ball charge (%)	10	33	10	33
Design power (kW)	8,435	7,933	8,653	9,040
Installed power (kW)	10,220	8,877	10,444	10,444
P_{80} = 85 µm	**JK SimMet (SABC)**		**CEET2 (SABC)**	
Nominal size (ft)	34 x 15	23 x 39	34 x 15	24 x 38
Design ball charge (%)	10	33	10	33
Design power (kW)	8,435	10,636	8,653	11,738
Installed power (kW)	10,220	12,085	10,444	12,682

16.1.8.1 High Pressure Roll Crusher Pilot Plant Tests

Two pilot plant scale test runs were undertaken by KHD at the factory and testing facility in Germany. The first series of tests were completed in November, 2007 and used approximately 3.5 t of near-surface lump material extracted from the Main Zone area of the Marathon PGM-Cu deposit. The second series of tests were designed as confirmatory tests to assess any variability in operating parameters by treating deeper mineralization. The feed sample used for these tests comprised about 1.3 t of drill core samples, selected from deeper drilling of the main ore body. The confirmatory tests were undertaken in May, 2008.

The November, 2007 test program designed to obtain these design parameters consisted of a series of single pass tests, and a subsequent series of closed circuit (locked cycle) grinding tests in combination with dry screening or product recycle. Additional tests to determine flake strength, wear rate and ball mill grindability (Bond work index) were also included in the program. These tests included three different configurations; namely open circuit, closed circuit with a screen and closed circuit by edge recycle.

For the closed circuit confirmatory grinding tests undertaken in May, 2008, the pilot HPGR was run at conditions close to those established in November, 2007.

Table 16.14 summarizes the recommended HPGR size and type, and the capacity of 22,000 t/d (dry) would be processed, at an overall assumed availability of 90 %. Given a moisture content of 4 %, the fresh feed would be 1,059 t/h (wet). At a circulating load of 154 %, the

HPGR feed rate then would calculate as 1,631 t/h (wet). This recommendation was confirmed by KHD during the May, 2008 testwork program.

KHD stated that the wear rate tests provided a basis for a wear life prediction of stud-lined rolls of 9,000 operating hours.

Table 16.14
Summary of HPGR Design Criteria and Sizing Recommendations

Criterion	Units	Value
Plant feed, dry	t/d	22,000
Overall availability	t/h	90
Plant feed, dry	t/h	1,019
Moisture	%	4
HPGR section feed	t/h	1,059
Circulating load	%	154
HPGR effective feed	t/h	1,631
No. of HPGR units	1	
HPGR Feed per Unit	t/h	1,631
Recommended HPGR Size		
Roll diameter	m	1.70
Roll width	m	1.80
Specific throughput	ts/hm³	300
Roll peripheral speed	m/s	1.78
Roll rotational speed	RPM	20.0
Specific energy	kWh/t	1.80
Total power	kWh/t	2,936
Motor safety factor	1.15	
Motor size (2 per HPGR unit)	kW	1,688
Specific pressure	N/mm²	4.6
Pressure	kN	14,076
RP pressure system	16	

During the November, 2007 test program, a number of comparative standard Bond ball mill grindability tests (product of <125 µm) were carried out. These tests suggest that a reduction of the Bond ball mill index may be expected as a result of the generation of micro-cracks in the mineral grains, which causes an inherent weakness of the grains and which, in the subsequent ball mill grinding stage, will lead to a reduced energy requirement for finer grinding.

Met-Chem completed a comparison in 2008 between the use of a SAG mill and a HPGR unit for the Marathon PGM-Cu project comminution circuit. The SAG mill circuit option considered in this engineering and cost study comprised a primary crusher, SAG mill with pebble crusher and a ball mill. The HPGR option included a primary crusher, secondary crusher, HPGR and a ball mill. Two HPGR options were considered, a closed circuit using a 4 mm screen and a closed circuit with edge recycle. A comparison of the costs for the three options is shown in Table 16.15. These costs are considered to be an order of magnitude level of accuracy.

Table 16.15
Comparison of SAG and HPGR Circuit Costs

Item	SAG Mill	HPGR (Screen)	HPGR (Edge Recycle)
Capital cost estimate ($M)	143	128	131
Unit operating cost estimate ($/t)	6.22	4.10	4.41

Note: Costs for grinding circuit only.

The results from this comparative study suggest that the HPGR option is better than the SAG mill alternative with regard to both capital and operating costs. The HPGR option with a screen was the selected comminution circuit for the Marathon PGM-Cu Feasibility Study.

16.1.9 Miscellaneous Metallurgical Tests

16.1.9.1 Magnetic Separation

Magnetic separation was performed by SGS-L during the 2007/08 testing program using a Davis tube. The purpose of these tests was to determine the recovery of magnetite from the flotation tailings. The tests were undertaken using a sample of rougher flotation tailings from Test F024. One test used the tailing 'as-is' (k_{80} = 84 µm) and the second used reground tailings to a product size of k_{80} = 42 µm. The 'as-is' test resulted in a magnetite yield of around 28% grading 79.3% Fe_2O_3 and 5.5% SiO_2. The re-ground sample yielded a 29% by weight magnetic fraction grading 83.2% Fe_2O_3 and 3.8% SiO_2.

16.1.9.2 PLATSOL™

In order to assess the potential of the hydrometallurgical treatment of the flotation concentrate, SGS-L performed a single PLASTOL™ test. The PLASTOL™ process is a high pressure leach process developed to recover the platinum group metals (PGMs) from their ores and concentrates.

The test comprised standard leaching conditions for 2 hours at a temperature of 225°C using a reground sample of the second bulk cleaner concentrate from Test F-075. As shown in Table 16.16 the Cu and Pt were virtually fully dissolved. About 80% Pd and approximately 50% of the Au and Ag were leached.

Table 16.16
Summary of the PLATSOL™ Test Results

Product	Analyses (mg/L, %, g/t)					Distribution (%)				
	Au	Pt	Pd	Ag	Cu	Au	Pt	Pd	Ag	Cu
Pregnant Soln.	0.16	0.64	2.00	4.00	12,000	45.6	95.0	80.0	52.2	99.0
Residue	1.80	0.32	4.72	34.6	0.12	54.4	5.04	5.04	47.8	1.05
Head grade (calc)	2.36	4.53	16.8	51.5	8.15	100.0	100.0	100.0	100.0	100.0

16.2 ONGOING METALLURGICAL TESTWORK

The testwork program completed by SGS-L in 2008 and the HPGR testwork undertaken by KHD were used to design the process used in the Marathon PGM-Cu Feasibility Study. In order to optimize the process developed and improve the estimated metallurgical performance in terms of valuable metal recoveries, an additional program of work was completed in 2009 by XPS that included a 6-day mini pilot plant test run. Micon believes that the metallurgical testwork completed to date on the Marathon PGM-Cu deposit provides ample proof that good metallurgical performance can be achieved using conventional flotation. However, Micon suggests that additional work may be worthwhile in order to try and reduce the reagent costs. This could entail reducing reagent dosage rates or substituting the existing reagent suite with less expensive chemicals.

17.0 MINERAL RESOURCE ESTIMATE AND MINERAL RESERVE ESTIMATES

17.1 INTRODUCTION

Micon was requested by Marathon PGM to provide an updated mineral resource estimate for its Marathon PGM-Cu deposit. The previous mineral resource estimate was completed in October, 2008. Since that time, Marathon has completed drilling 21 additional holes extending and infilling the mineral resource domains used to develop the previous mineral resource estimate. Further, the previous mineral resource estimate used a simple search ellipsoid which did not fully capture the continuity of the mineralization. As a result, Marathon PGM requested that Micon complete a new mineral resource estimate for the Marathon deposit.

17.2 PREVIOUS MINERAL RESOURCE ESTIMATE

The previous mineral resource estimate was completed in October, 2008 by Eugene Puritch, P.Eng., and Antoine Yassa, P.Geo., of P&E Mining Consultants Inc. (P&E) of Brampton, Ontario, with the assistance of David Good, Ph.D., P.Geo., V.P. Exploration of Marathon PGM. This mineral resource estimate was described in the February 2, 2009 Technical Report prepared by Micon. This estimate is presented below in Table 17.1.

Table 17.1
October, 2008 Marathon Mineral Resource Estimate

Resource within Pit Shell								Contained Metal					
Category	**Tonnes millions**	**Pd (g/t)**	**Pt (g/t)**	**Au (g/t)**	**Cu (%)**	**Ag (g/t)**	**Rh (g/t)**	**Pd oz (000)**	**Pt oz (000)**	**Au oz (000)**	**Cu lb million**	**Ag oz (000)**	**Rh oz (000)**
Measured	69.5	0.79	0.23	0.09	0.30	1.7	0.006	1761	513	194	455	3763	12.6
Indicated	27.9	0.66	0.24	0.08	0.21	1.6	0.007	588	215	72	128	1442	6.6
Meas & Ind	97.4	0.75	0.23	0.09	0.27	1.7	0.006	2349	728	266	583	5205	19.2
Inferred	2.7	0.50	0.16	0.07	0.20	2.1	0.004	43	14	6	12	181	0.4

1. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. The quantity and grade of reported inferred resources in this estimation are conceptual in nature and there has been insufficient exploration to define an indicated mineral resource on the property and it is uncertain if further exploration will result in discovery of an indicated or measured mineral resource on the property.

The October 2008 mineral resource estimate is superseded by the updated mineral resource estimate prepared by Micon and described in this report.

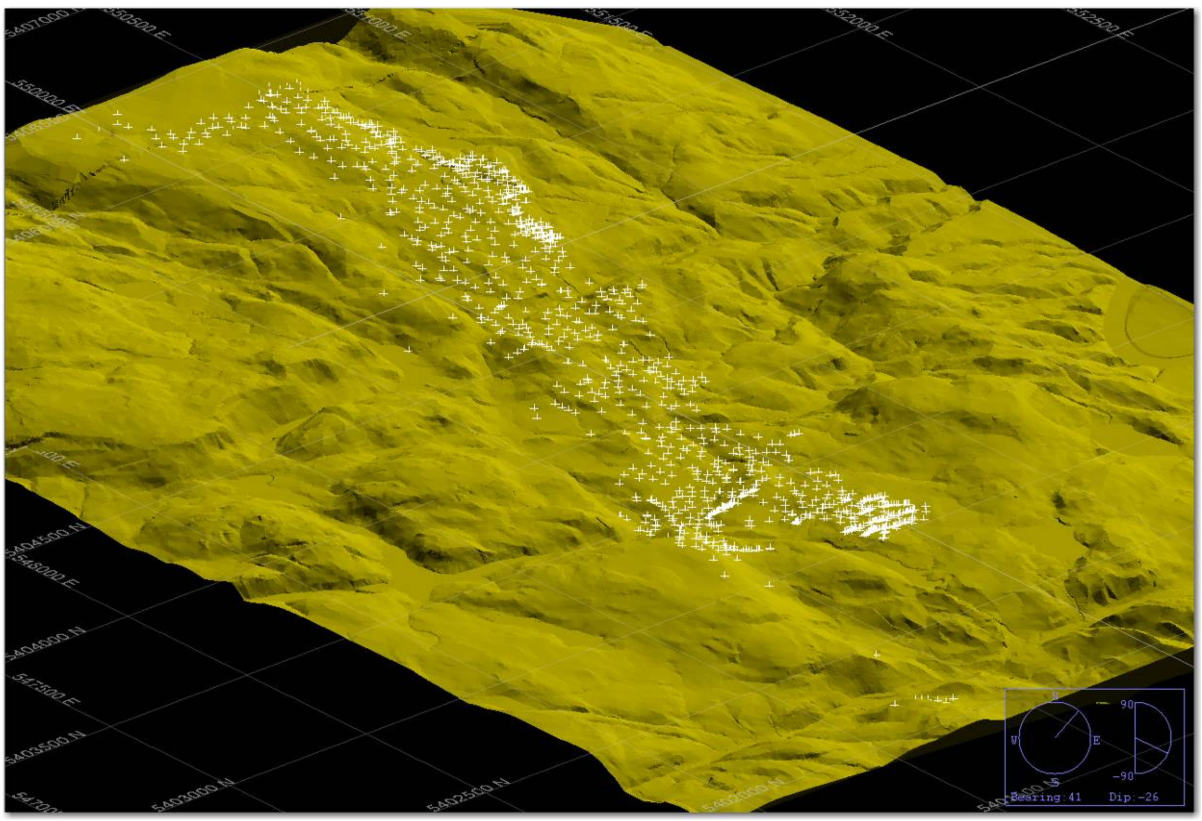
17.3 MINERAL RESOURCE ESTIMATION PROCEDURE

A review of the basis for the previous mineral resource estimate (geologic cross-sections) was completed using both the previously used drill holes along with the new, additional 21 drill holes. The new in-fill drilling indicated minor changes from the previously interpreted geologic model which required that an updated cross-sectional interpretation be completed before a new mineral resource estimate could be established.

17.3.1 Topography

Topography for the property was provided by Marathon PGM and is the same as was described in the February, 2009 Micon Technical Report. This topography was clipped to include a surface area slightly larger than the block model extents. The topographic surface is shown below in Figure 17.1.

Figure 17.1
Isometric View Showing the Topography of the Marathon PGM-Cu Deposit and Drill Hole Collar Locations
(Looking northeast)



17.3.2 Database

All drilling data on the Marathon property was provided to Micon in the form of a Microsoft Excel spreadsheet file. A total of 818 drill holes and 456 surface channel samples are contained within this database. This data was used to develop 259 drill cross-sections on a UTM grid looking north on an azimuth of 360° on a nominal 25-m spacing named 5,403100N to 5,406,337.5N. These drill cross sections were used to validate the previous domain interpretations used in the previous mineral resource estimate (see Section 17.2 above). A surface and drill hole collar map is shown below in Figure 17.2. Using the drill hole and channel information, a Vulcan ISIS database was constructed for use in statistics, composting, and grade estimation.

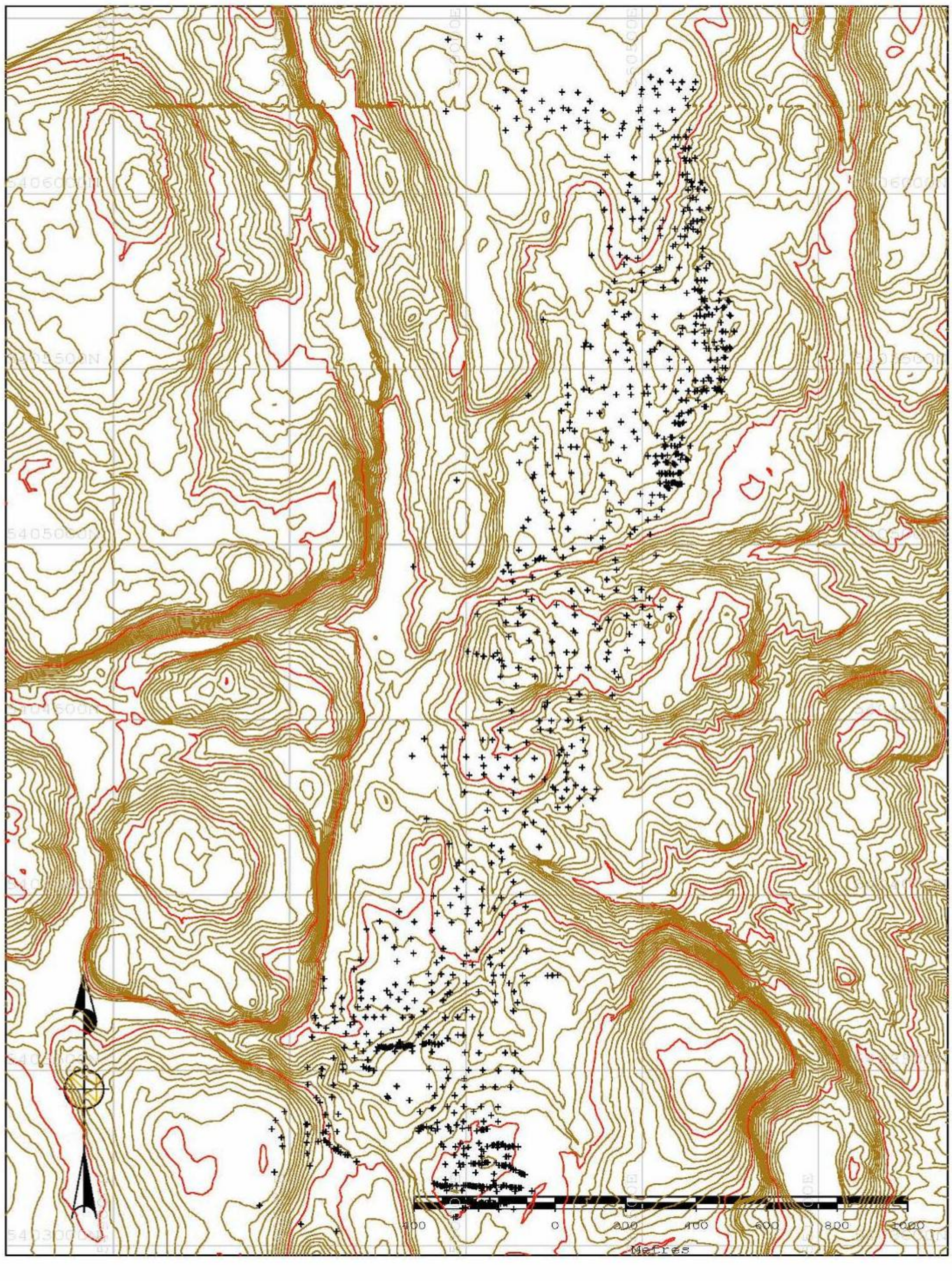
The Vulcan ISIS database was validated and minor corrections applied. The assay table of the database contains 45,858 assay intervals for Cu, Pd, Pt, Au, Ag and Rh. All location data are expressed in metric units and grid coordinates are in a NAD27 UTM system. The survey table of the database contains 27,808 records, while the geology table contains 9,331 records.

17.3.3 Mineralized Domain Interpretation

The mineralized domains from the previous Technical Report were used as the basis flagging both the block model and drill hole database with geologic codes. A description from the previous Technical Report (Puritch and Yassa, 2008) describes how these domains were created:

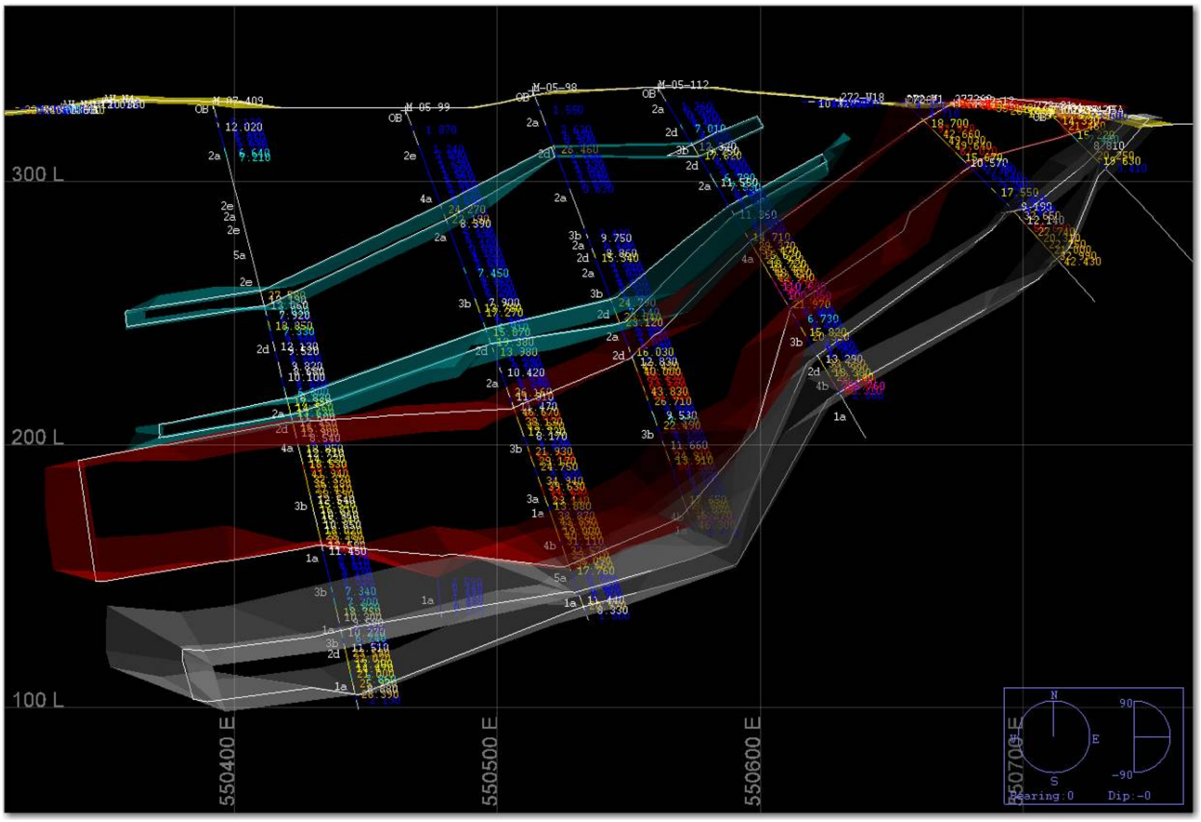
> *Domain boundaries were determined from lithology; structure and net smelter return (NSR) boundary interpretation from visual inspection of drillhole sections. Eleven domains were developed and are named in Section 17.5. These domains were created with computer screen digitizing on drillhole sections in Gemcom by the authors of this report. The outlines were influenced by the selection of mineralized material with an NSR value above $6.63/t that demonstrated zonal continuity along strike and down dip. In some cases, mineralization below $6.63/t was included for the purpose of maintaining zonal continuity. Smoothing was utilized to remove obvious jogs and dips in the domains and incorporated a minor addition of inferred mineralization. This exercise allowed for easier domain creation without triangulation errors from solids validation.*
>
> *On each section, polyline interpretations were digitized from drill hole to drill hole but not extended more than 50 m into untested territory. Minimum constrained true width for interpretation was 5.0 m. The interpreted polylines from each section were "wireframed" in Gemcom into 3-dimensional (3D) domains. The resulting solids (domains) were used for statistical analysis, grade interpolation, rock coding and resource reporting purposes.*

Figure 17.2
Marathon PGM-Cu Drill Hole Collar Location Map



The mineralized domain solids created were checked on every drill hole cross-section to ensure that the solids were accurate to the drilling and had been correctly imported into the Vulcan mine planning software system. Several minor corrections and additions were made to encompass the additional 21 drill holes completed during 2009. A typical cross-section is shown below in Figure 17.3.

Figure 17.3
Geologic Cross-Section at 5,405,450N
(View Looking North)



17.3.4 Vulcan Block Model Domain Code Determination

The Vulcan block model domain codes used for the resource model were derived from the mineralized domain solids. The list of Vulcan block model domain codes used is shown in Table 17.2 below.

Table 17.2
Vulcan Block Model Domain Codes

Vulcan Model Code	Domain
air	Air
n_main	North Main
n_s_hw	North Hanging Wall
n_fw	North Footwall
mbr_main	Malachite Main
mbr_hw	Malachite Hanging Wall
mbr_fw	Malachite Footwall
s_main	South Main
s_fw	South Footwall
w_zone	W
w_hg	HG
mag	Mag Zones
waste	Waste (mine) Rock

These codes were flagged in the block model during construction as well as into the composite database.

17.3.5 Composites

Composting of the exploration drill hole database was completed using the same criteria as was used in the previous Technical Report (Puritch and Yassa, 2008). Compositing was completed using Vulcan software and a composite database was constructed as a Vulcan ISIS file. The approach is the same as was previously used and was described as follows:

> *Length-weighted composites were generated for the drill hole data that fell within the constraints of the above-mentioned domains. These composites were calculated for Cu, Au, Pt, Pd, Ag and Rh over 2.0 m lengths starting at the first point of intersection between assay data hole and hanging wall of the 3D zonal constraint. The compositing process was halted upon exit from the footwall of the aforementioned constraint. Un-assayed intervals were treated as null data. Any composites calculated that were less than 0.5 m in length, were discarded so as to not introduce a short sample bias in the interpolation process. The composite data were transferred to extraction files for the grade interpolation as X, Y, Z, Cu, Au, Pt, Pd, Ag and Rh files for each domain.*

17.3.6 Vulcan Tetra Modeling

The previous mineral resource estimate (October, 2008) used a fixed search ellipsoid which represented the average strike and dip of the overall Marathon deposit. In reviewing that work, it became apparent that because of changing dips and rolling stratigraphy along strike, higher and lower grade zones within the mineralized domains were being incorrectly spatially represented. In order to correct this, an unfolding method needed to be applied to the search ellipsoid during statistics, variography, and resource estimation. A tool within the Vulcan mine planning software called Tetra Modeling was used to accomplish this.

According Maptek (vendor of the Vulcan software) Tetra Modeling is described as:

> *Tetra modeling is used in the grade estimation and variography of deformed strata bound deposits. Tetra modeling can be applied to deposits where mineralization is controlled by a structural surface that can be modeled. In Tetra modeling the grade estimation search ellipse or variography search ellipse is distorted from the usual "football" shaped ellipse to follow nominated surfaces.*
>
> *The great benefit of using distorted search ellipses is that the block model stays in the position that it was created and the samples stay in their true position. The difference between a normal estimation and tetra estimation is that the search ellipse is molded to follow the surfaces used to bound the deposit.*
>
> *A tetra model is created from two triangulated surfaces (the hanging and floor surfaces). These surfaces are the two "nearest" surfaces to the block cell. A line is calculated that passes through the centroid of the block cell with one end point touching the hanging surface and the other end point touching the floor surface. The line of minimum distance is then used to define a "mid-surface" between the hanging surface and the floor surface.*
>
> *A line of minimum distance is calculated for each block cell. Tetrahedra are then constructed from the end points of the lines, alternating in direction. A tetra model is made up of these tetrahedra that are used to calculate the minimum distance between the two surfaces at any given point in the model.*

The Marathon PGM-Cu deposit, although igneous, behaves very much like a stratigraphic type deposit and thus a Tetra model can be constructed and used to unfold the search ellipsoid. To accomplish this, a line was digitized at the footwall and hanging wall contacts of the mineral domains on every cross-section. These lines were then used to create a grid model (both upper and lower surfaces) that would act as boundaries for the Tetra model. The resulting Tetra model was used to unfold the ellipsoid and better approximate the stratigraphic structure of the deposit. Figures 17.4 and 17.5 show the bounding Tetra model surfaces.

17.3.7 Grade Capping

Grade capping was investigated on the raw assay values in each mineralized domain to ensure that the possible influence of erratic high grade values did not bias the database. The capping values are shown in Table 17.3 below. Statistics were run on Cu, Pd, Pt, Au, Ag and Rh assays within each mineralized domain. Assays above a cumulative population of 99.5% were capped at that value (99.5%).

Table 17.3
Grade Capping Values

North Main (N_MAIN) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	6,216	0.291	0.219	0.000	3.550	1.150
Pd	6,216	0.776	0.846	0.001	15.716	4.610
Pt	6,216	0.224	0.295	0.000	8.200	1.504
Au	6,216	0.083	0.107	0.001	2.610	0.584
Ag	6,216	1.636	1.576	0.000	29.300	7.130
Rh	6,216	0.0055	0.0135	0.0000	0.2690	0.0820

North Footwall (N_FW) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	1,124	0.256	0.236	0.000	4.270	1.156
Pd	1,124	0.523	0.789	0.001	14.906	4.610
Pt	1,124	0.146	0.177	0.000	1.770	1.258
Au	1,124	0.061	0.093	0.001	1.165	0.590
Ag	1,124	1.227	1.380	0.000	11.840	8.000
Rh	1,124	0.0023	0.0079	0.0000	0.1640	0.0450

North Hanging Wall (N_S_HW) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	798	0.149	0.141	0.000	1.010	0.845
Pd	798	0.462	0.597	0.001	6.630	3.625
Pt	798	0.174	0.189	0.000	1.637	1.271
Au	798	0.059	0.074	0.001	0.790	0.489
Ag	798	1.447	1.259	0.000	15.000	6.000
Rh	798	0.0036	0.0104	0.0000	0.1500	0.0690

Malachite Main (MBR_MAIN) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	582	0.137	0.129	0.000	0.997	0.692
Pd	582	0.603	0.881	0.005	7.246	4.915
Pt	582	0.229	0.300	0.007	2.990	1.760
Au	582	0.087	0.108	0.001	0.727	0.675
Ag	582	1.498	1.182	0.450	9.000	6.000
Rh	582	0.0072	0.0162	0.0000	0.1950	0.0935

Malachite Footwall (MBR_FW) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	505	0.177	0.128	0.000	0.904	0.851
Pd	505	0.344	0.486	0.001	5.840	2.983
Pt	505	0.132	0.147	0.001	1.190	0.883
Au	505	0.053	0.048	0.001	0.439	0.260
Ag	505	1.700	1.928	0.450	33.000	8.000
Rh	505	0.0037	0.0064	0.0000	0.0800	0.0390

Malachite Hanging Wall (MBR_HW) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	996	0.172	0.157	0.000	0.982	0.814
Pd	996	0.538	0.737	0.001	9.468	5.156
Pt	996	0.222	0.260	0.001	2.683	1.550
Au	996	0.082	0.103	0.001	1.587	0.591
Ag	996	1.723	1.384	0.450	12.000	7.000
Rh	996	0.0075	0.0208	0.0000	0.3379	0.1130

South Main (S_MAIN) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	89	0.139	0.141	0.004	0.645	No Cap
Pd	89	0.491	0.509	0.005	3.125	No Cap
Pt	89	0.237	0.183	0.007	0.944	No Cap
Au	89	0.077	0.062	0.002	0.320	No Cap
Ag	89	1.649	2.719	0.450	14.000	No Cap
Rh	89	0.0079	0.0149	0.0000	0.1210	No Cap

South Footwall (S_FW) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	340	0.325	0.366	0.001	4.910	2.920
Pd	340	0.426	0.576	0.005	8.970	3.163
Pt	340	0.124	0.128	0.007	1.830	0.659
Au	340	0.060	0.082	0.001	0.983	0.653
Ag	340	1.493	1.313	0.450	8.000	6.333
Rh	340	0.0018	0.0038	0.0000	0.0340	0.0260

Walford Zone (W_ZONE) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	1,325	0.124	0.175	0.001	1.220	0.980
Pd	1,325	1.728	4.333	0.001	69.976	23.995
Pt	1,325	1.867	3.484	0.001	39.102	8.758
Au	1,325	0.132	0.305	0.001	7.229	1.542
Ag	1,325	1.776	1.553	0.450	27.000	7.710
Rh	1,325	0.0255	0.0822	0.0000	1.0390	0.6550

Walford High Grade Zone (W_HG) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	371	0.195	0.238	0.003	1.220	1.183
Pd	371	3.847	7.200	0.005	69.976	61.163
Pt	371	1.425	3.095	0.001	39.102	25.641
Au	371	0.254	0.506	0.001	7.229	3.163
Ag	371	1.932	1.551	0.450	8.000	7.000
Rh	371	0.0607	0.1409	0.0000	1.0390	0.9570

Magnetite Zone (MAG) Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	244	0.124	0.128	0.001	1.110	0.915
Pd	244	0.474	0.529	0.001	2.910	2.808
Pt	244	0.151	0.158	0.007	1.494	1.066
Au	244	0.075	0.113	0.001	1.097	0.945
Ag	244	2.174	1.389	0.450	6.570	6.368
Rh	244	0.0030	0.0003	0.0000	0.0140	0.0140

Waste Domain						
Element	**Samples**	**Mean**	**Std Dev**	**Minimum**	**Maximum**	**Cum 99.5%**
Cu	34,887	0.042	0.004	0.000	2.220	0.168
Pd	34,887	0.097	0.056	0.000	14.557	0.560
Pt	34,887	0.050	0.006	0.000	3.094	0.200
Au	34,887	0.020	0.002	0.000	1.900	0.099
Ag	34,887	1.134	1.916	0.000	72.982	3.848
Rh	34,887	0.0016	0.0000	0.0000	0.3160	0.0100

17.3.8 Variography

Variography was run on the samples contained within the individual mineralized domains within the deposit. Anisotropic variograms were determined for Cu and Pd in all domains (with the exception of the South Main domain where all variograms are onmi directional) as well as Pt, Au, Ag and Rh in the North Main domain. Omni variograms were determined for Pt, Au, Ag and Rh within all of the remaining domains. Variography was run to determine the anisotropic dimensions for the search ellipsoid. The results of the variography are shown below in Table 17.4.

Table 17.4
Variography Results for the Updated Marathon PGM-Cu Deposit Dataset

North Main (N_MAIN) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	87	160	135
Pd	---	313	190	100
Pt	---	30	220	180
Au	---	129	190	80
Ag	---	180	300	90
Rh	---	209	90	60

North Footwall (N_FW) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	138	300	300
Pd	---	12	295	40
Pt	117	---	---	---
Au	75	---	---	---
Ag	200	---	---	---
Rh	90	---	---	---

North Hanging Wall (N_S_HW) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	162	195	110
Pd	---	14	295	140
Pt	230	---	---	---
Au	120	---	---	---
Ag	75	---	---	---
Rh	150	---	---	---

Malachite Main (MBR_MAIN) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	257	85	60
Pd	---	8	60	35
Pt	160	---	---	---
Au	125	---	---	---
Ag	50	---	---	---
Rh	100	---	---	---

Malachite Footwall (MBR_FW) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	60	280	95
Pd	---	25	300	135
Pt	80	---	---	---
Au	95	---	---	---
Ag	50	---	---	---
Rh	50	---	---	---

Malachite Hanging Wall (MBR_HW) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	85	300	285
Pd	---	146	70	55
Pt	35	---	---	---
Au	100	---	---	---
Ag	120	---	---	---
Rh	200	---	---	---

South Main (S_MAIN) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	60	---	---	---
Pd	60	---	---	---
Pt	75	---	---	---
Au	80	---	---	---
Ag	90	---	---	---
Rh	70	---	---	---

South Footwall (S_FW) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	43	135	70
Pd	---	43	150	85
Pt	40	---	---	---
Au	135	---	---	---
Ag	50	---	---	---
Rh	10	---	---	---

Walford Zone (W_ZONE) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	249	300	130
Pd	---	270	170	100
Pt	140	---	---	---
Au	50	---	---	---
Ag	65	---	---	---
Rh	80	---	---	---

Walford High Grade Zone (W_HG) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	0	295	120
Pd	---	155	295	80
Pt	75	---	---	---
Au	60	---	---	---
Ag	75	---	---	---
Rh	60	---	---	---

Magnetite Zone (MAG) Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	27	200	185
Pd	---	188	200	140
Pt	50	---	---	---
Au	25	---	---	---
Ag	30	---	---	---
Rh	50	---	---	---

Waste Domain				
Element	**Omni**	**Azimuth**	**Major**	**Semi-Major**
Cu	---	0	150	40
Pd	---	90	50	15
Pt	25	---	---	---
Au	50	---	---	---
Ag	50	---	---	---
Rh	10	---	---	---

17.3.9 Bulk Density

In the previous Technical Report (Puritch and Yassa, 2008), the average bulk density was calculated to be 3.08 t/m^3. This density was updated by David Good of Marathon PGM in July, 2009 and his procedure to determine bulk densities are described below:

1. *Define the proportion of rock types in 109,645 m of drill core by (a) subdividing drill core into mineralized and mine (waste) rock and (b) subdividing each group into specific rock types. It is assumed that the distribution of rock types in the drill core will be representative of rock units in the pit (except felsic footwall material, see below).*

2. *Define the average specific gravity values for each rock type. A total of 303 analyses were determined. Note the average specific gravity for the samples is 3.08 (the value used in 2008 calculation).*

3. *Calculate weighted average for mineralized and mine (waste) rocks.*

Table 17.5
Proportion of Rock Types in Drill Core

Mine Rocks	Thickness (m)	%
Rock unit		
Eastern Gabbro	40,457	57.16
Footwall (felsic)	16,484	23.29
TD Gabbro	6,830	9.65
Syenite	6,001	8.48
Gabbro breccia	1,011	1.43
Total	**70,784**	
Host rocks		
Rock unit		
TD Gabbro	25,493	71.03
Eastern + TD Gabbro	5,108	14.23
Gabbro breccia	4,892	13.63
Footwall (felsic)	400	1.11
Total	**35,893**	
Magnetite-rich gabbro	2,969	

Figure 17.4
Porportion of Mine Rock in Drill Core

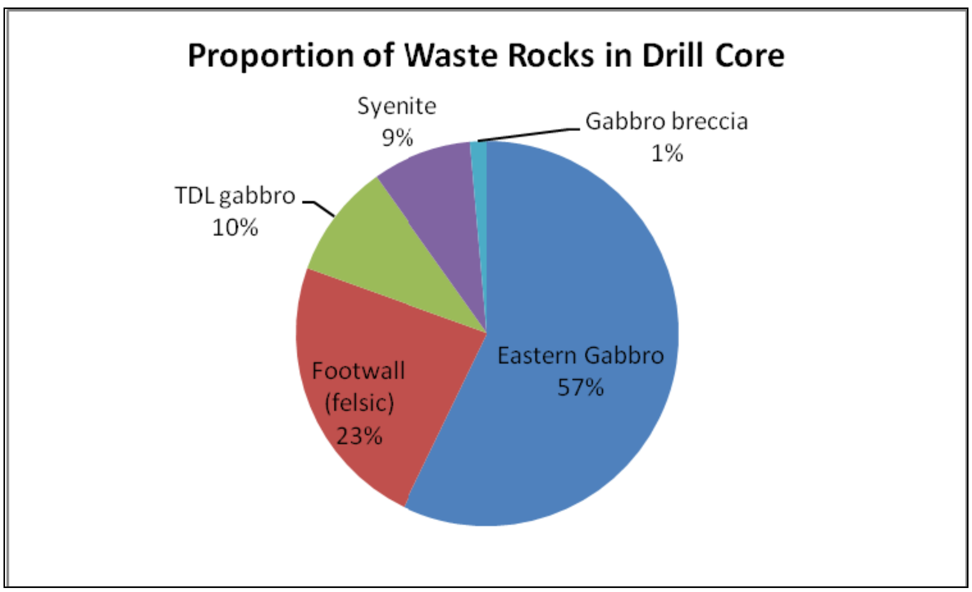


Figure 17.5
Proportion of Host Rocks in Drill Core

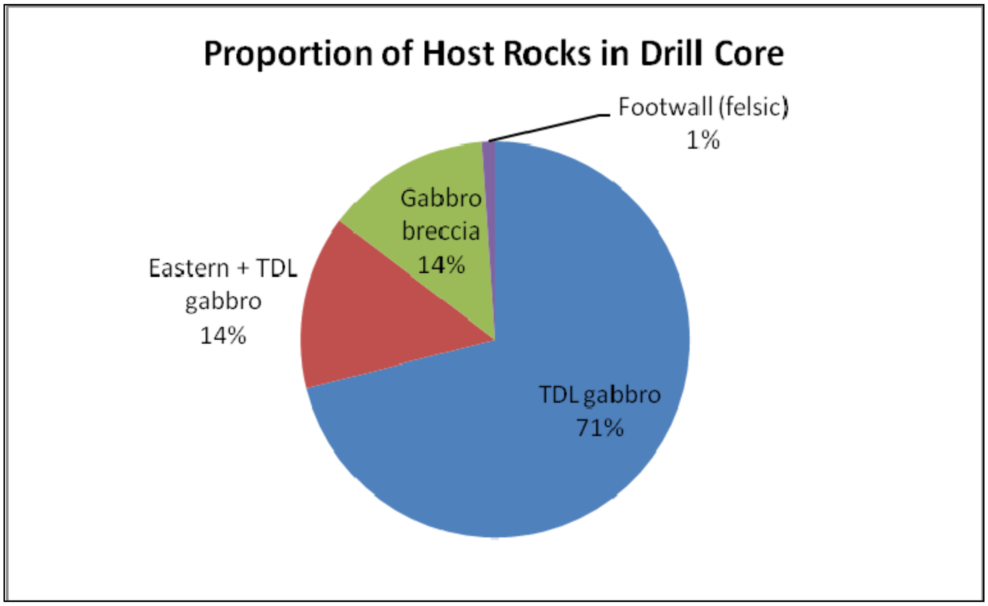


Table 17.6
Average Specific Gravity of Rock Types

Code	Rock Type	Number of Samples	Average SG
Mineralization			
3b	Two Duck Gabbro	216	3.08
2d	Eastern plus Two Duck Gabbro	10	3.19
4a	Gabbro breccia	9	3.24
1a	Felsic basement	1	2.94
2f	Magnetite rich gabbro	5	3.29
Mine Rock			
2a	Eastern Gabbro	7	3.03
3b	Two Duck Gabbro	50	3.06
1a	Felsic basement	3	2.98
5a	Syenite dikes	2	2.88
4a	Gabbro breccia	1	3.17

Adjustment to proportions in mine rock was carried out by inspection. The amount of footwall and syenite in the drill core is believed to be high relative to the amount that will be encountered in the pit shell. The proportions were therefore reduced from 23% and 9% in drill core to 10% and 5%, respectively. The weightings of the Eastern Gabbro were increased by similar amounts to make up the difference. The net effect of this adjustment is an increase in the average mine rock density from 3.01 to 3.02 as shown in Table 17.7.

Table 17.7
Weighted Average SG Values for Mineralized and Mine Rocks

Code	Rock Type	Average SG	Proportion (%)
Mineralization			
3b	Two Duck Gabbro	3.08	71.03
2d	Eastern plus Two Duck Gabbro	3.19	14.23
4a	Gabbro breccia	3.24	13.63
1a	Felsic basement	2.94	1.11
Weighted Average		**3.12**	
2f	Magnetite rich gabbro	3.29	
Mine Rock			
2a	Eastern Gabbro	3.03	70.45
3b	Two Duck Gabbro	3.06	13.13
1a	Felsic basement	2.98	10
5a	Syenite dikes	2.88	5
4a	Gabbro breccia	3.17	1.42
Weighted Average		**3.02**	

Table 17.8
Bulk Densities Used in the 2009 Mineral Resource Estimate

Material Type	Bulk Density (t/m^3)
Mineralization	3.12
Mine Rock	3.02
Magnetite Gabbro	3.29

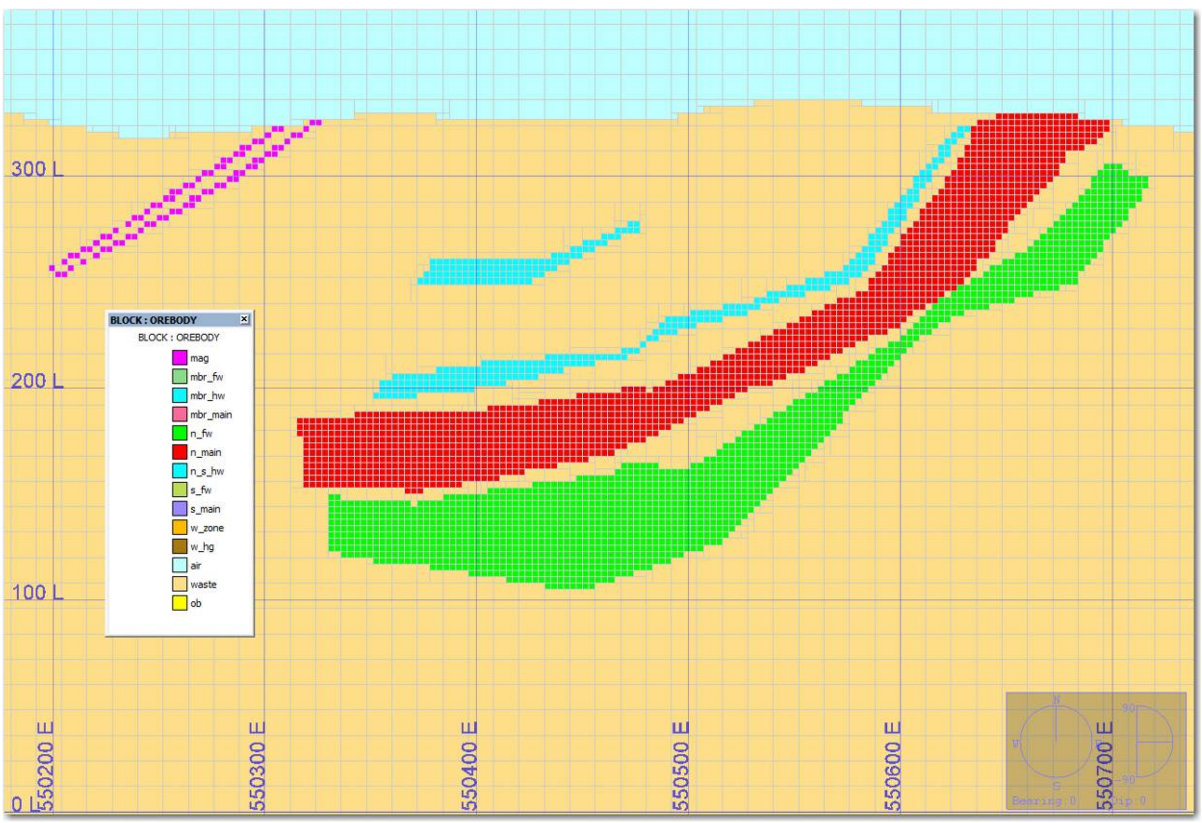
17.3.10 Block Model

A 3D block model was constructed in the Vulcan mine planning software that was constrained by the various mineralizing domain solids. The block model is sub-blocked with the minimum block size being 3 m by 3.125 m by 3 m (X, Y, Z) to a maximum block size of 12 m by 12.5 m by 12 m (X, Y, Z). Within mineralized domains the maximum block size is the same as the minimum block size (3 m by 3.125 m by 3 m). The block model was not rotated. Table 17.9 describes the block model setup parameters. A typical cross-section through the block model is shown in Figure 17.8 below.

Table 17.9
Marathon Block Model Parameters

Item	X (m)	Y (m)	Z (m)
Minimum Coordinates	549,100.0	5,402,975.0	-264.0
Maximum Coordinates	551,188.0	5,406,600.0	480.0
Minimum Block Size	3.000	3.125	3.000
Maximum Block Size	12.000	12.500	12.000
Rotation	0.00	0.00	0.00

Figure 17.6
Typical Vertical Block Model Cross-Section at 5405425N
(Looking North)



Bulk densities were assigned to each block depending on the domain. For air blocks, density was set to zero; for waste blocks (everything outside of the mineralized domains and air), density was set to 3.02 t/m^3; for mineralized domains (the exception being magnetite blocks), density was set to 3.12 t/m^3; while for the magnetite domain, density was set to 3.29 t/m^3. Once the density was flagged, grade estimation could be completed.

No attempt was made to apply a block percentage (percent of the block that is ore and waste), instead sub-blocking along the domain boundaries was used. This creates a cleaner model for later resource and reserve runs. Grade interpolation runs were set-up for each domain and each element. Additionally, mine rock (waste) material outside of the mineralized domains was estimated as well to provide a dilution grade during later reblocking to create the diluted block model.

17.3.11 Grade Estimation

Using the Vulcan ISIS composite file (described above), interpolations were run in each domain for Cu, Pd, Pt, Au, Ag and Rh. For Pd, Pt and Au, two runs were completed in the North Main and Footwall domains. The first run used a smaller search ellipsoid in order to limit the influence of higher value samples. The second pass populated the more distance blocks beyond the higher value samples. Runs were completed in all domains for all metals using inverse distance squared (ID^2). Additional runs were completed on Cu only using inverse distance to the fifth power (ID^5, roughly a polygonal estimate) and a nearest neighbour estimates. These two additional estimates are used as checks on the ID^2 estimate. The block model interpolation parameters are shown in Table 17.10 below.

Table 17.10
Block Model Interpolation Parameters

North Main Domain										
Element	**Pass**	**Bearing**	**Plunge**	**Dip**	**Min Samples**	**Max Samples**	**Max Per DDH**	**Major (m)**	**Semi-Major (m)**	**Minor (% Tetra)[1]**
Cu	1	87	0	0	2	15	3	300	300	0.04
Pd	1	313	0	0	2	15	3	100	100	0.03
Pd	2	313	0	0	2	15	3	300	300	0.04
Pt	1	30	0	0	2	15	3	100	100	0.03
Pt	2	30	0	0	2	15	3	300	300	0.04
Au	1	129	0	0	2	15	3	100	100	0.03
Au	2	129	0	0	2	15	3	300	300	0.04
Ag	1	180	0	0	2	15	3	300	300	0.04
Rh	1	209	0	0	2	15	3	300	300	0.04

North Footwall Domain										
Element	**Pass**	**Bearing**	**Plunge**	**Dip**	**Min Samples**	**Max Samples**	**Max Per DDH**	**Major (m)**	**Semi-Major (m)**	**Minor (% Tetra)[1]**
Cu	1	138	0	0	2	15	3	300	300	0.04
Pd	1	12	0	0	2	15	3	100	100	0.03
Pd	2	12	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	100	100	0.03
Pt	2	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	100	100	0.03
Au	2	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

North Hanging Wall Domain										
Element	**Pass**	**Bearing**	**Plunge**	**Dip**	**Min Samples**	**Max Samples**	**Max Per DDH**	**Major (m)**	**Semi-Major (m)**	**Minor (% Tetra)[1]**
Cu	1	162	0	0	2	15	3	300	300	0.04
Pd	1	14	0	0	2	15	3	100	100	0.03
Pd	2	14	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

Malachite Main Domain										
Element	**Pass**	**Bearing**	**Plunge**	**Dip**	**Min Samples**	**Max Samples**	**Max Per DDH**	**Major (m)**	**Semi-Major (m)**	**Minor (% Tetra)[1]**
Cu	1	257	0	0	2	15	3	300	300	0.04
Pd	1	8	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

Malachite Footwall Domain										
Element	**Pass**	**Bearing**	**Plunge**	**Dip**	**Min Samples**	**Max Samples**	**Max Per DDH**	**Major (m)**	**Semi-Major (m)**	**Minor (% Tetra)[1]**
Cu	1	60	0	0	2	15	3	300	300	0.04
Pd	1	25	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

Malachite Hanging Wall Domain										
Element	**Pass**	**Bearing**	**Plunge**	**Dip**	**Min Samples**	**Max Samples**	**Max Per DDH**	**Major (m)**	**Semi-Major (m)**	**Minor (% Tetra)[1]**
Cu	1	85	0	0	2	15	3	300	300	0.04
Pd	1	146	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

South Main Domain										
Element	**Pass**	**Bearing**	**Plunge**	**Dip**	**Min Samples**	**Max Samples**	**Max Per DDH**	**Major (m)**	**Semi-Major (m)**	**Minor (% Tetra)[1]**
Cu	1	0	0	0	2	15	3	300	300	0.04
Pd	1	0	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

South Footwall Domain										
Element	Pass	Bearing	Plunge	Dip	Min Samples	Max Samples	Max Per DDH	Major (m)	Semi-Major (m)	Minor (% Tetra)[1]
Cu	1	43	0	0	2	15	3	300	300	0.04
Pd	1	43	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

Magnetite Domain										
Element	Pass	Bearing	Plunge	Dip	Min Samples	Max Samples	Max Per DDH	Major (m)	Semi-Major (m)	Minor (m)
Cu	1	27	0	40	2	15	3	300	300	50
Pd	1	188	0	40	2	15	3	300	300	50
Pt	1	0	0	40	2	15	3	300	300	50
Au	1	0	0	40	2	15	3	300	300	50
Ag	1	0	0	40	2	15	3	300	300	50
Rh	1	0	0	40	2	15	3	300	300	50

Waste Domain										
Element	Pass	Bearing	Plunge	Dip	Min Samples	Max Samples	Max Per DDH	Major (m)	Semi-Major (m)	Minor (% Tetra)[1]
Cu	1	0	0	0	2	15	3	300	300	0.04
Pd	1	90	0	0	2	15	3	300	300	0.04
Pt	1	0	0	0	2	15	3	300	300	0.04
Au	1	0	0	0	2	15	3	300	300	0.04
Ag	1	0	0	0	2	15	3	300	300	0.04
Rh	1	0	0	0	2	15	3	300	300	0.04

[1] The minor search axis in Tetra modeling uses a maximum search distance that is a percentage of the distance in that direction between the upper and lower Tetra surfaces. If that distance were 100 m, then a 0.04 search distance would be 4 m on either side of the point being estimated.

17.3.12 Mineral Resource Classification

For the purposes of this mineral resource estimate, classifications of all interpolated grade blocks were determined from the Cu interpolations for Measured, Indicated and Inferred due to Cu being the dominant revenue producing element in the NSR calculation. The mineral resource classification logic is shown below in Table 17.11.

Table 17.11
Marathon Resource Classification Logic

Domain	Class	Average Distance (m)	Min. No. of Samples	Max. No. of Samples
North Main	Measured	74	7	15
North Main	Indicated	148	4	15
North Main	Inferred	200	2	15
North Footwall	Measured	100	7	15
North Footwall	Indicated	200	4	15
North Footwall	Inferred	200	2	15
North Hanging Wall	Measured	76	7	15
North Hanging Wall	Indicated	152	4	15
North Hanging Wall	Inferred	200	2	15
Malachite Main	Measured	36	7	15
Malachite Main	Indicated	72	4	15
Malachite Main	Inferred	200	2	15
Malachite Footwall	Measured	94	7	15
Malachite Footwall	Indicated	188	4	15
Malachite Footwall	Inferred	200	2	15
Malachite Hanging Wall	Measured	100	7	15
Malachite Hanging Wall	Indicated	200	4	15
Malachite Hanging Wall	Inferred	200	2	15
South Main	Measured	30	7	15
South Main	Indicated	60	4	15
South Main	Inferred	200	2	15
South Footwall	Measured	51	7	15
South Footwall	Indicated	102	4	15
South Footwall	Inferred	200	2	15
Walford Zone	Measured	100	7	15
Walford Zone	Indicated	200	4	15
Walford Zone	Inferred	200	2	15
Walford High Grade Zone	Measured	100	7	15
Walford High Grade Zone	Indicated	200	4	15
Walford High Grade Zone	Inferred	200	2	15
Magnetite Zone	Measured	96	7	15
Magnetite Zone	Indicated	192	4	15
Magnetite Zone	Inferred	200	2	15

17.3.13 Diluted Block Model Construction

For later pit optimization, mineral resource and reserve estimates, and mine planning activities, a diluted block model was required. This block model has the same extents as the undiluted block model except that block size is now constant with no sub-blocks. The reblocked block size selected was 6 m by 6.25 m by 12 m (X, Y, Z). Individual block grade values were calculated using a weighted average of the source blocks (undiluted). Since mine rock grades were estimated during the interpolation runs, this material could be used in this process. Resource classification was determined by a weighted majority code of the source blocks. This was also applied to the domain codes as well. Lastly, density was calculated using a volume weighted average.

17.4 MINERAL RESOURCE ESTIMATE

The mineral resource estimates in this report used the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by CIM Standing Committee on Reserve Definitions and adopted by CIM Council on December 11, 2005. The mineral resource estimates provided in this report are classified as “measured”, “indicated”, or “inferred” as defined by CIM.

According to the CIM definitions, a Mineral Resource must be potentially economic in that it must be “in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction”. For the Marathon PGM-Cu deposit, a net smelter return (NSR) was calculated to determine the various economic cut-off valuess used in resource and reserve calculations. The NSR was calculated for each block in both the undiluted and diluted block models. Table 17.12 shows the economic parameters used in the NSR calculation.

The anticipated Marathon PGM-Cu open pit operation, mill processing, smelting, refining, shipping, G&A and mining costs combine for a total of \$6.47 + \$0.58 + \$1.51 + \$3.44 = \$12.00/t milled which became the NSR cut-off value for higher grade resource reporting. The lower grade NSR cut-off was derived from mill processing, smelting, refining, shipping, and G&A costs only for total of \$6.47 + \$0.58 + \$3.44 = \$10.50/t ore milled.

Contribution of the various metals in the NSR calculation is as follows:

Cu = (90.8% Rec. x 95.5% Payable x 22.05/t x US\$2.50/lb)/0.87 = \$54.94/%
Au = (79.9% Rec. x 90% Payable x US\$700/oz)/31.1/0.87 = \$18.60/g/t
Pt = (71.0% Rec. x 88% Payable x US\$1,100/oz)/31.10.87 = \$25.40/g/t
Pd = (80.1% Rec. x 92.5% Payable x US\$300/oz)/31.1/0.87 = \$8.22/g/t
Ag = (74.5% Rec. x 90% Payable x US\$12/oz)/31.1/0.87 = \$0.30/g/t

Table 17.12
Parameters for NSR Cut-off Grade Calculation
(Currency in Canadian dollars unless otherwise stated)

Description	Value
$/US$ exchange rate	0.87
Copper price	US$2.50/lb
Gold price	US$700.00/oz
Platinum price	US$1,100.00/oz
Palladium price	US$300.00/oz
Silver price	US$12.00/oz
Ore mining cost (all material)	$1.51/t mined
Processing cost	$6.47/t milled
Copper flotation recovery	90.8%
Gold flotation recovery	79.9%
Platinum flotation recovery	71.0%
Palladium flotation recovery	80.1%
Silver flotation recovery	74.5%
Concentration ratio	88.5 to 1
Copper smelter payable	95.5%
Gold smelter payable	90.0%
Platinum smelter payable	88.0%
Palladium smelter payable	92.5%
Silver smelter payable	90.0%
Smelting, Refining, Shipping charges	$3.44/t milled
General and administration	$0.58/t milled

17.4.1 Whittle Pit Optimization

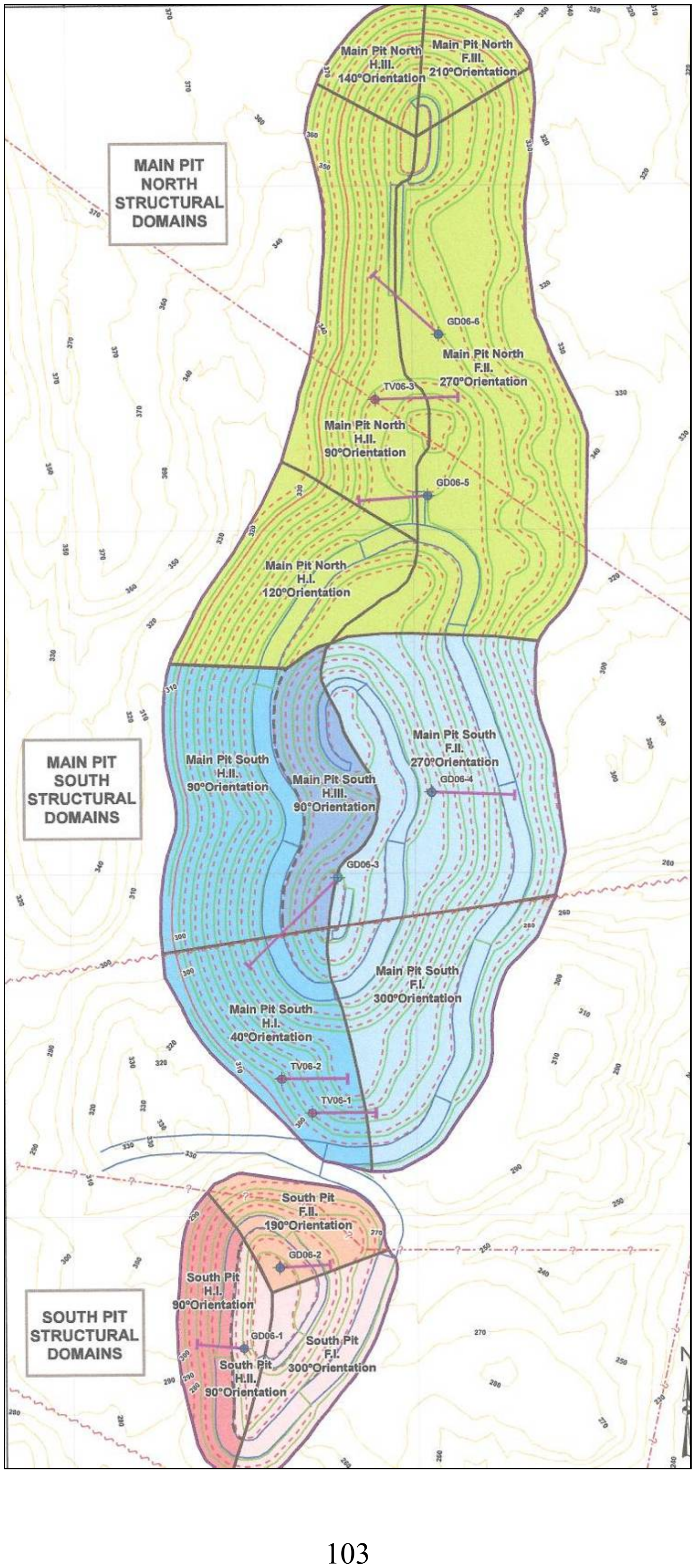
In order for the constrained mineralization in the Marathon PGM-Cu model to be considered as a mineral resource which is potentially economic, a Whittle 4X pit optimization was carried out utilizing the criteria shown below in Table 17.13.

Table 17.13
Criteria for Whittle 4X Pit Optimization

Item	Value
Mine rock mining cost	$1.51/t
Ore mining cost	$1.51/t
Ore processing cost	$6.47/t
Smelting, refining, & shipping cost	$3.44/t
General and administration cost	$0.58/t
Production rate	8,030,000 t/y

Pit slope sectors were provided by Golder Associates Ltd. (Golder). The slope sectors for the Marathon PGM-Cu deposit are shown in Figure 17.9 below. These slope sectors were coded into the model and used for both Whittle pit optimization and pit designs.

Figure 17.7
Marathon Slope Sectors



The diluted block model was exported to Whittle where the model was prepared for optimization. A number of pit optimization runs were completed at the base numbers listed above along with extensive sensitivity analysis. The pit by pit results are shown below in Figure 17.8. Table 17.14 shows the estimated pit shell mineral resource contained within the optimized pit shell (revenue factor of, pit shell 46).

Figure 17.8
Marathon Whittle Pit by Pit Analysis

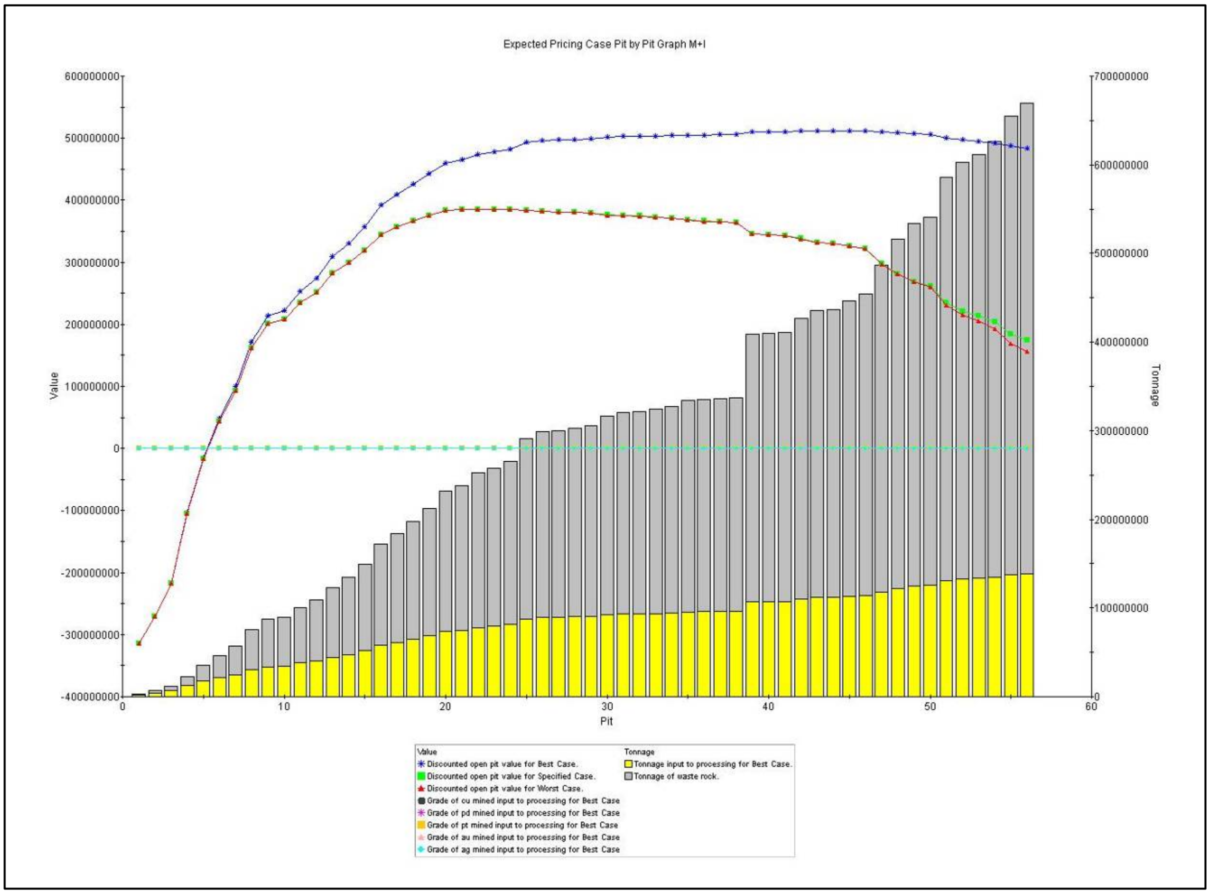


Table 17.14
Marathon Pit Shell Resource (Diluted Block Model)

Higher Grade Resource above $12.00/tonne NSR Cut-off (excluding lower grade)

Category	Pit Shell 46 Mineral Resource						Contained Metal				
	Tonnes millions	Pd g/t	Pt g/t	Au g/t	Cu %	Ag g/t	Pd oz (000)	Pt oz (000)	Au oz (000)	Cu lb million	Ag oz (000)
Measured	91.7	0.862	0.247	0.089	0.267	1.609	2,544	728	263	540	4,747
Indicated	15.4	0.696	0.236	0.087	0.181	1.527	346	117	43	62	758
Measured + Indicated	***107.2***	***0.813***	***0.237***	***0.086***	***0.250***	***1.586***	***2,889***	***845***	***307***	***601***	***5,505***
Inferred	4.4	0.340	0.114	0.051	0.167	1.506	48	16	7	16	211

Lower Grade Resource between $10.50 and $12.00/tonne NSR Cut-off

Category	Pit Shell 46 Mineral Resource						Contained Metal				
	Tonnes millions	Pd g/t	Pt g/t	Au g/t	Cu %	Ag g/t	Pd oz (000)	Pt oz (000)	Au oz (000)	Cu lb million	Ag oz (000)
Measured	2.5	0.247	0.099	0.039	0.103	1.228	20	8	3	6	100
Indicated	5.1	0.247	0.096	0.041	0.101	1.338	40	16	7	11	218
Measured + Indicated	***7.6***	***0.247***	***0.097***	***0.040***	***0.101***	***1.301***	***60***	***24***	***10***	***17***	***318***
Inferred	1.8	0.223	0.081	0.039	0.112	1.347	13	5	2	4	79

Total Resource (Lower and Higher Grade) above $10.50/tonne NSR Cut-off

Category	Pit Shell 46 Mineral Resource						Contained Metal				
	Tonnes millions	Pd g/t	Pt g/t	Au g/t	Cu %	Ag g/t	Pd oz (000)	Pt oz (000)	Au oz (000)	Cu lb million	Ag oz (000)
Measured	94.3	0.846	0.243	0.088	0.262	1.599	2,564	736	266	545	4,847
Indicated	20.5	0.451	0.160	0.062	0.140	1.421	386	133	50	73	976
Measured + Indicated	***114.8***	***0.775***	***0.228***	***0.083***	***0.241***	***1.567***	***2,950***	***869***	***316***	***618***	***5,823***
Inferred	6.2	0.306	0.104	0.047	0.151	1.459	61	21	9	21	290

1. The mineral resources presented above are the subject of the Feasibility Study discussed in the present Technical Report.
2. The quantity and grade of reported inferred resources in this estimate are conceptual in nature and there has been insufficient exploration to define them as indicated mineral resources. It is uncertain if further exploration will result in their conversion to indicated or measured mineral resources.

Following the completion of 21 additional exploration drill holes in September, 2009, the block model was updated. The mineral resource estimate presented in Table 17.14 is effective as of 24 November, 2009.

The mineral resources listed in Table 17.14 were estimated by Sam J. Shoemaker, Jr., M.AusIMM. Mr. Shoemaker is a QP as defined in NI 43-101 and is independent of Marathon PGM.

Table 17.15 shows the pit shell mineral resources at various NSR cut-off values.

Table 17.15
Marathon PGM-Cu Deposit NSR Sensitivity in Pit Shell 46

Diluted Resource Model December 16, 2009 Estimated Mineral Inventory Pit Shell 46												
NSR Cutoff ($)	Measured + Indicated						Inferred					
	Tonnes	Cu %	Pd g/t	Pt g/t	Au g/t	Ag g/t	Tonnes	Cu %	Pd g/t	Pt g/t	Au g/t	Ag g/t
0.01	272,970,000	0.123	0.376	0.122	0.045	1.334	117,953,000	0.037	0.080	0.041	0.017	1.142
5.00	171,464,000	0.182	0.566	0.174	0.065	1.461	30,297,000	0.081	0.162	0.065	0.030	1.236
7.50	135,482,000	0.216	0.685	0.205	0.076	1.524	12,574,000	0.117	0.237	0.086	0.040	1.343
10.50 [1]	***114,782,000***	***0.241***	***0.775***	***0.228***	***0.083***	***1.567***	***6,173,000***	***0.151***	***0.306***	***0.104***	***0.047***	***1.459***
12.00 [2]	***107,179,000***	***0.250***	***0.813***	***0.237***	***0.086***	***1.586***	***4,357,000***	***0.167***	***0.340***	***0.114***	***0.051***	***1.506***
13.00	102,707,000	0.256	0.836	0.243	0.088	1.598	3,424,000	0.178	0.366	0.121	0.053	1.533
14.00	98,534,000	0.262	0.859	0.248	0.090	1.609	2,713,000	0.189	0.386	0.127	0.056	1.555
15.00	94,659,000	0.267	0.881	0.254	0.092	1.618	2,109,000	0.200	0.414	0.136	0.060	1.564
16.00	90,616,000	0.273	0.904	0.259	0.093	1.627	1,630,000	0.211	0.441	0.146	0.063	1.589
17.00	86,489,000	0.278	0.930	0.266	0.095	1.634	1,286,000	0.223	0.464	0.154	0.067	1.608
18.00	82,278,000	0.284	0.957	0.272	0.097	1.643	1,057,000	0.231	0.486	0.162	0.070	1.615
19.00	78,214,000	0.289	0.985	0.279	0.099	1.648	848,000	0.242	0.507	0.171	0.073	1.670
20.00	74,358,000	0.295	1.013	0.285	0.101	1.653	692,000	0.254	0.519	0.176	0.075	1.734
22.00	66,952,000	0.305	1.071	0.299	0.105	1.654	433,000	0.272	0.583	0.197	0.083	1.936
24.00	59,629,000	0.315	1.135	0.315	0.109	1.648	277,000	0.299	0.641	0.211	0.085	1.809
26.00	53,082,000	0.323	1.201	0.331	0.113	1.633	184,000	0.329	0.634	0.226	0.092	2.041
28.00	47,043,000	0.332	1.267	0.347	0.118	1.614	123,000	0.361	0.656	0.237	0.094	2.094
30.00	41,850,000	0.339	1.330	0.363	0.122	1.604	77,000	0.402	0.664	0.252	0.097	2.029
35.00	29,987,000	0.357	1.490	0.406	0.134	1.593	41,000	0.442	0.693	0.287	0.106	2.510
40.00	19,212,000	0.377	1.685	0.463	0.147	1.619	19,000	0.416	1.010	0.360	0.120	2.386

[1] Incremental mill cut-off (i.e., mill cost + G&A + TC/RC).
[2] Full cut-off (i.e., mining cost + milling cost + G&A + TC/RC).

17.5 MINERAL RESERVE ESTIMATE

The mineral reserves for the Marathon PGM-Cu deposit have been estimated by Micon as summarized in Table 17.16.

Mineral reserves have been estimated for the North, South and Malachite pits from the diluted block model and are the result of work described in Section 18 of this report.

Table 17.16
Mineral Reserves for the Marathon PGM-Cu Deposit

Classification	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	Cu (M lb)	Pd Oz (000's)	Pt Ozs (000's)	Au Ozs (000's)	Ag Ozs (000's)
Proven	76,461,000	0.910	0.254	0.090	0.268	1.464	452	2,237	625	222	3,600
Probable	14,986,000	0.435	0.147	0.060	0.138	1.318	46	209	71	29	635
Total	91,447,000	0.832	0.237	0.085	0.247	1.440	497	2,447	696	251	4,235

1. The mineral reserves for the Marathon PGM-Cu deposit, as shown in Table 17.16 are included within the mineral resources shown in Table 17.14.

CIM Standards on Mineral Resources and Reserves Definitions and Guidelines (1) define 'Proven Mineral Reserve' as "the economically mineable part of a Measured Mineral Resource" and 'Probable Mineral Reserve' as "the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource." Economics shall be "demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified."

The mineral reserve estimate presented in Table 17.16 is effective as of 24 November, 2009.

The mineral reserves presented in Table 17.16 were estimated by Sam Shoemaker, Jr., MAusIMM. Mr. Shoemaker is a QP as defined in NI 43-101 and is independent of Marathon PGM.

The known potential effects of any environmental, permitting, taxation, socio-economic, marketing, political or other relevant issues are discussed in Section 18.0 of this report. Legal and title issues are discussed in Section 4.2.

The ultimate pit limit is based on the economic Lerchs-Grossmann algorithm design on all sides, with no external factors constraining the pit. The final pit plans are presented in Figures 18.6, 18.7 and 18.8.

The North pit is subdivided into four phases to reduce the front end stripping requirements for an even ore tonnage release in the production schedule and allow a higher grade ore delivery to the mill in the first four years of the production schedule. The phase designs are based on slope design parameters provided by Golder (Golder Associates, 2007).

The mine plan developed in this report is based on Measured and Indicated resources only. There is opportunity to upgrade some minor amounts of the inferred resource mineralization to ore classification with additional infill drilling.

18.0 OTHER RELEVANT DATA AND INFORMATION

The purpose of this Technical Report is to report the results of an updated Feasibility Study with an effective date of 24 November, 2008.

The Marathon PGM-Cu project comprises open pit mining and processing at an average rate of 22,000 t/d of ore to produce a saleable flotation concentrate containing Cu, Pd, Pt, Au, Ag and Rh. The life of the operation is estimated at approximately 11.5 years.

18.1 MINING AND MINERAL RESERVES

The proposed Marathon PGM-Cu open pit will be a conventional open pit mining operation that will be developed by the Owner using its own equipment and workforce.

The Owner will have responsibility for site preparation; haul road construction; production drilling and blasting; the excavation and haulage of ore to the primary crusher and mine rock to the MRSA; oversize breakage; pit dewatering; haul road maintenance; and equipment maintenance. The Owner will provide the open pit equipment, operator training, supervision, pit technical support services, mine consumables, and the pit operations and maintenance facilities. The Owner will also utilize specialized contractors for initial site clearing and overburden stripping, and will source explosives, blasting agents, fuel and other consumables from established suppliers.

18.1.1 Mineral Resource Model

The mineral resource model used for the pit optimization, pit design, and production scheduling is the diluted block model described above in Section 17. Only material with the resource classification of ‘measured’ or ‘indicated’ could be considered as potential mill feed. In addition to the estimated grade values for Cu, Pd, Pt, Au, Ag and Rh contained within the diluted block model, other variables were calculated or input into the diluted block model. These included the net smelter return (see Section 18.1.2 below), geotechnical parameters (see Section 18.1.3.2 below), block economic net value, haulage simulation results, block material type, and Whittle rock types. All of this additional information was used in the pit optimization, design and scheduling described below.

The final diluted block model was completed on October 1, 2009 and is current as of December 16, 2009. Construction of this block model commenced in July, 2009 and the first pass completed in August, 2009. Following the completion of 21 additional exploration drill holes in September, 2009, the block model was updated with the final model being finished on 1 October, 2009. All of the work completed on the diluted (and undiluted) resource block models was completed using the Maptek Vulcan Mine Planning software version 8.0.2.

18.1.2 NSR Calculations

All blocks containing mineralization within the diluted block model had NSR values determined. This not only included blocks with the resource classification of 'measured' 'indicated' or 'inferred', but also included 'waste' blocks as well. The NSR value is one of several variables used to determine if a block is potentially mill feed. For the Marathon PGM-Cu deposit, a NSR was calculated to determine the various economic cut-off values used in resource and reserve calculations. The NSR was calculated for each block in both the undiluted and diluted block models. Table 17.12, in Section 17.4 shows the economic parameters used in the NSR calculation.

The estimated operating costs are based on the Micon Technical Report dated 2 February. 2009, updated as part of the present Feasibility Study. Revenue and exchange rate values are based on three- and five-year trailing averages. Flotation recoveries are based on the current numbers used in this report (see Section 18.2 below) as are smelting, refining, and shipping charges. For smelting, refining, and shipping charges; a cost per tonne of crude ore was calculated ($3.44/t crude ore) and used in the NSR and economic calculations for pit optimization, pit design, and production scheduling.

18.1.3 Open Pit Design

In order to complete an open pit design on the Marathon PGM-Cu deposit, several items are required:

- Geotechnical information – Geotechnical data describing the inter ramp slope angle, slope sectors, and berm widths are required in order to develop a geotechnically stable pit design.

- Economic and metallurgical criteria – Estimated operating and capital costs associated with the project. Estimated metal pricing, metal recoveries, downstream operating costs (smelting, refining, and shipping), currency conversion rates, and projected annual mill feed requirements. This information may change as a result of the pit optimization, design, and production scheduling which could require additional passes with the more refined data.

- Pit optimization – A pit optimization that is based on the economic, metallurgical, geotechnical and production requirements for the project.

Once a pit optimization has been completed, the selected pit shell can be used as the basis for design of the open pit. For the Marathon PGM-Cu deposit, three major mining areas are present, the North pit, South pit, and Malachite pit. Once these three pit areas had been designed, a production schedule was then prepared. This was followed by equipment selection, estimation of operating and capital costs, and personnel requirements.

18.1.3.1 Geotechnical Pit Slope Design Criteria

The pit slope design criteria for the Marathon PGM-Cu project were developed by Golder and provided in a report titled "Recommendations for Open Pit Rock Slope Design Marathon PGM-Cu Project" dated March, 2007 (Golder Associates Ltd., 2007). These recommendations were followed closely during the pit design portion of the updated Feasibility Study.

18.1.3.2 Optimization Parameters

Pit optimization is typically completed using a Lerchs-Grossmann algorithm (LG) on a block model. For the Marathon PGM-Cu deposit, GEMCOM's LG software, the Whittle optimizer was selected. The diluted block model was prepared by creating a Whittle rock code variable (wht_rx) in the Vulcan block model. The variable was populated with rock codes that were broken into waste, ore1 ('measured'), ore2 ('indicated'), and ore3 ('inferred'). For the pit shells developed for the Marathon pit design, only the Whittle rock codes of ore1 and ore2 could be considered as mill feed. Ore3 and mine rock were always considered as going to the MRSA.

Once the diluted block model had been prepared, it was exported into a format suitable for reading by the Whittle software. Variables exported out of the diluted block model included the Whittle rock code; Cu, Pd, Pt, Au and Ag assays; and slope sector numbers.

18.1.3.3 LG Mining and Processing Costs

The model exported from the diluted Vulcan block model was read into the Whittle pit optimization software. The input data were checked to ensure that the imported values matched the source diluted block model. Within the Whittle software, an economic and metallurgical model was created and populated with the values shown in Table 18.1 below. Metallurgical data in the Whittle model is the same as is listed in Table 17.12 above.

Table 18.1
Marathon PGM-Cu Deposit Whittle Economic and Production Criteria

Item	Value
Mine rock mining cost	$1.51/t
Ore mining cost	$1.51/t
Ore processing cost	$6.47/t
Smelting, refining, & shipping cost	$3.44/t
General and administration cost	$0.58/t
Production rate	8,030,000 t/y

Once the Whittle model was finished, it was run for various economic scenarios.

18.1.3.4 LG Sensitivity Cases, Pit Shell Selection, and Pit Phase Selection

Sensitivity runs were completed on various combinations of pricing and operating cost parameters. Additionally, seven different maximum pit shells at the base economics were selected for determining the optimal economic pit shell for final design. Pit shells were selected at pit numbers 21, 24, 29, 38, 41, 44, and 46 (revenue factor = 1). Of these maximum pit shells in the economic base case, pit shell 29 was selected as the design basis for the ultimate pit. Pit shells 21 and 24 were selected as potential limits for phased pit designs. The criteria for selection were to balance the mine life and the Whittle predicted NPV. Table 18.2 shows economic assumptions used in the sensitivity analysis while Table 18.3 below shows the results.

None of the scenarios listed in Table 18.3 included inferred mineral resources as plant feed. Inferred resource material has been allocated as waste.

18.1.3.5 Pit Design Parameters

For a pit design to be completed, the face angle and berm width are required for every block within the block model. This information was coded into the block model previously in preparation for export to the Whittle pit optimization. Slope sectors for the Marathon PGM-Cu deposit are shown below in Figure 18.1. Table 18.4 shows the slope design parameters provided by Golder.

Table 18.2
Marathon LG Sensitivity Assumptions

Item	Units	Worst	3-Yr	5-Yr	Expected	Expected $1.35	Expected $2.00	As at 7 Oct 2009	Best
Mining	$/t all material	1.51	1.51	1.51	1.51	1.35	2.00	1.51	1.51
Processing	$/t milled	6.47	6.47	6.47	6.47	6.47	6.47	6.47	6.47
G&A	$/t milled	0.58	0.58	0.58	0.58	0.58	0.58	0.58	0.58
Smelter	$/t milled	3.44	3.30	3.41	3.44	3.44	3.44	3.44	3.44
Ag	US$/oz	8.00	13.81	13.50	12.00	12.00	12.00	17.52	15.00
Au	US$/oz	700.00	795.50	675.90	700.00	700.00	700.00	1,044.00	1,000.00
Cu	US$/oz	1.75	2.83	2.73	2.50	2.50	2.50	2.75	3.09
Pd	US$/oz	200.00	322.00	290.10	300.00	300.00	300.00	312.00	400.00
Pt	US$/oz	900.00	1,337.00	1,190.00	1,100.00	1,100.00	1,100.00	1,330.00	1,700.00
$/US$	---	0.87	0.91	0.88	0.87	0.87	0.87	0.87	0.87
Concentration Ratio	---	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50
CAPEX (millions)	---	385.0	385.0	385.0	385.0	385.0	385.0	385.0	385.0
Ag	Flotation Recovery	74.5	74.5	74.5	74.5	74.5	74.5	74.5	74.5
Au	Flotation Recovery	79.9	79.9	79.9	79.9	79.9	79.9	79.9	79.9
Cu	Flotation Recovery	90.8	90.8	90.8	90.8	90.8	90.8	90.8	90.8
Pd	Flotation Recovery	80.1	80.1	80.1	80.1	80.1	80.1	80.1	80.1
Pt	Flotation Recovery	71.0	71.0	71.0	71.0	71.0	71.0	71.0	71.0
Ag	% Payable at Smelter	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0
Au	% Payable at Smelter	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0
Cu	% Payable at Smelter	95.5	95.5	95.5	95.5	95.5	95.5	95.5	95.5
Pd	% Payable at Smelter	92.5	92.5	92.5	92.5	92.5	92.5	92.5	92.5
Pt	% Payable at Smelter	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0

Table 18.3
Marathon LG Sensitivity Results

Scenario	Pit Number	Class	Ore Tonnes	Mine Rock Tonnes	Total Tonnes	Strip Ratio	Life (Yr)	Ag (g/t)	Au (g/t)	Cu (%)	Pd (g/t)	Pt (g/t)
Worst	45	M+I	69,321,000	185,116,000	254,437,000	2.67	8.6	1.462	0.098	0.276	0.997	0.277
3-Yr Trailing	46	M+I	128,477,000	388,423,000	516,900,000	3.02	16.0	1.587	0.080	0.232	0.736	0.220
5-Yr Trailing	46	M+I	120,904,000	365,041,000	485,945,000	3.02	15.1	1.578	0.082	0.237	0.761	0.225
Expected P46	46	M+I	114,416,000	339,437,000	453,853,000	2.97	14.2	1.571	0.084	0.242	0.780	0.229
Expected P44	44	M+I	111,995,000	324,754,000	436,749,000	2.90	13.9	1.566	0.084	0.243	0.785	0.230
Expected P41	41	M+I	107,440,000	303,551,000	410,991,000	2.83	13.4	1.551	0.085	0.245	0.799	0.233
Expected P38	38	M+I	96,546,000	240,276,000	336,822,000	2.49	12.0	1.493	0.086	0.248	0.836	0.241
Expected P29	***29***	***M+I***	***90,815,000***	***214,971,000***	***305,786,000***	***2.37***	***11.3***	***1.478***	***0.087***	***0.252***	***0.853***	***0.243***
Expected P24	24	M+I	81,322,000	183,678,000	265,000,000	2.26	10.1	1.441	0.090	0.256	0.899	0.253
Expected P21	21	M+I	75,157,000	162,402,000	237,559,000	2.16	9.4	1.432	0.091	0.261	0.921	0.256
Expected $1.35	46	M+I	119,258,000	380,373,000	499,631,000	3.19	14.9	1.593	0.083	0.241	0.773	0.228
Expected $2.00	46	M+I	94,598,000	224,691,000	319,289,000	2.38	11.8	1.489	0.086	0.248	0.835	0.239
As at 7 October, 2009	46	M+I	133,946,000	399,656,000	533,602,000	2.98	16.7	1.595	0.079	0.227	0.718	0.215
Best	46	M+I	154,559,000	452,198,000	606,757,000	2.93	19.2	1.580	0.075	0.212	0.660	0.201

Table 18.4
Slope Sectors for Pit Design

Whittle Sector	Slope Domain	Sector	IRA[1] (°)	Bench Height (m)	Per Berm (m)	Batter	Berm
1	Main Pit North	H.III	55	12	2	75	10.4
2	Main Pit North	F.II	52	12	2	75	12.3
3	Main Pit North	H.II	55	12	2	75	10.4
4	Main Pit North	F.I	55	12	2	75	10.4
5	Main Pit North	H.I	51	12	2	75	13
6	Main Pit South	H.II	55	12	2	75	10.4
7	Main Pit South	H.III	55	12	2	75	10.4
8	Main Pit South	F.II	55	12	2	75	10.4
9	Main Pit South	H.I	50	12	2	70	11.4
10	Main Pit South	F.I	51.4	12	2	70	10.4
11	South Pit	F.II	51.4	12	2	70	10.4
12	South Pit	H.I	55	12	2	75	10.4
13	South Pit	H.II	48	12	2	65	10.4
14	South Pit	F.I	51.4	12	2	70	10.4

[1] Inter ramp angle.

Catch benches or safety berms are double benched. Ramps are designed with a maximum grade of 10% and a haul road width of 30 m. The haul road width selected allows for enough space to allow 2.5 times the typical truck width plus room for a safety berm and drainage ditch. Minimum mining width used was 70 m.

18.1.3.6 Pit Designs

Pit designs were completed on the Marathon PGM-Cu deposit using the Whittle pit shell number 29. Pit shells 21 and 24 were used as guidelines in preparing internal production phases. The majority of the mineral resource at the Marathon PGM-Cu deposit is contained in the North pit area. Because of this, this area was broken into four designs phases to allow future scheduling to better balance required stripping of mine rock over the life of the mining operation.

The South pit is a small pit just south of the North pit and it was designed as a single phase pit. The Malachite pit complex is composed of three different independent pit designs or phases. Although the three Malachite phases are referred to as phases, they are actually stand-alone pit designs that can be exploited at any time during scheduling. The four phases of the North pit are required to be exploited one after the other, that is, the same bench in North pit phase 1 must be mined out before the same bench in North pit phase 2. This rule applies to all of the North pit phases. Figures 18.1 through 18.7 show the individual phased pit designs.

On the last several benches of each phase, the access ramps were narrowed from 30 m to 20 m widths. This allows a slightly deeper pit while maintaining the geotechnical slope requirements. On these last few benches in each phase, it is envisioned that there will be traffic control to limit only one vehicle on the narrow portions of the access ramp at a time.

Figure 18.1
Overall Marathon Pit Layout
(Post Mining)

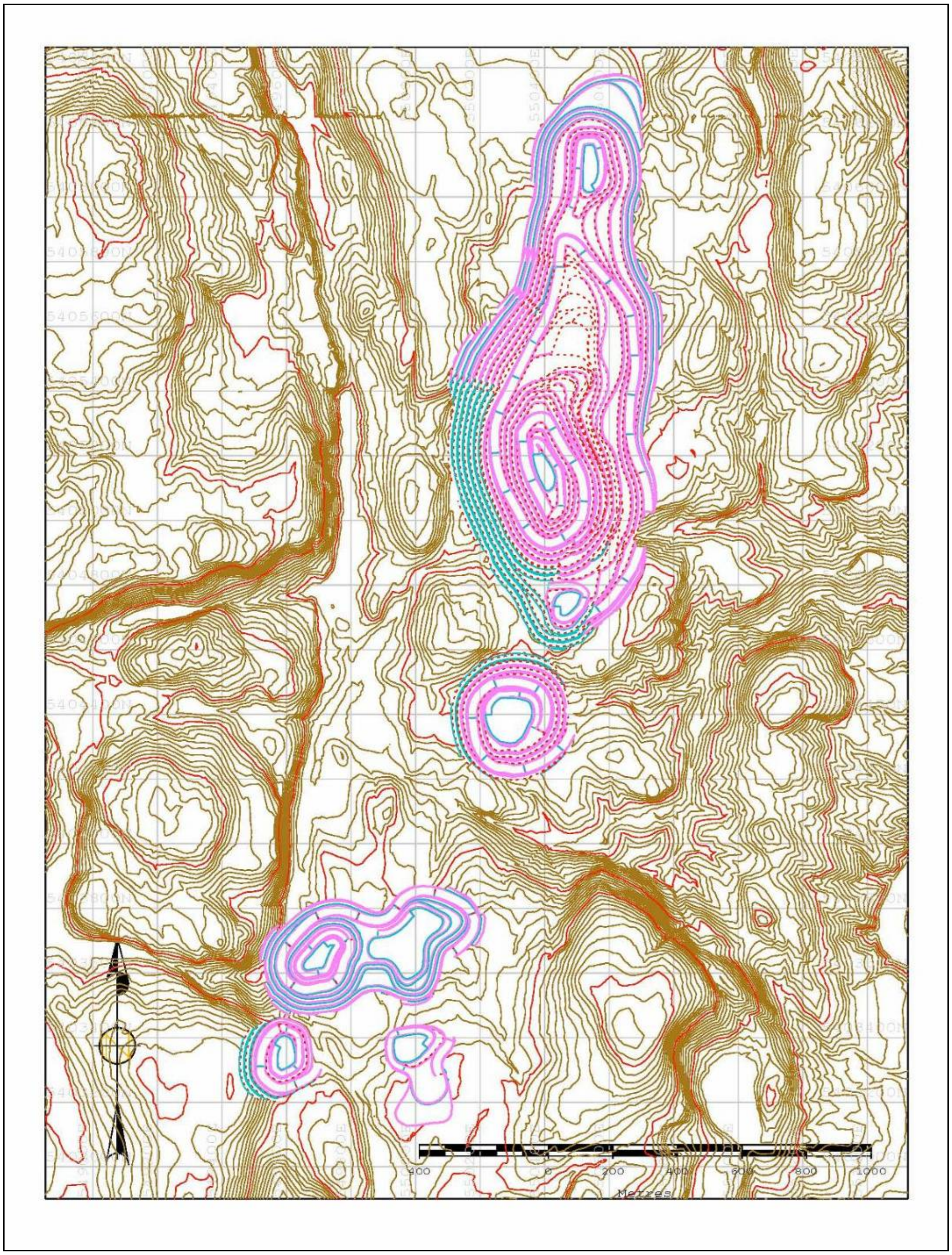


Figure 18.2
Marathon North Pit Phase 1 Design (Post Mining)

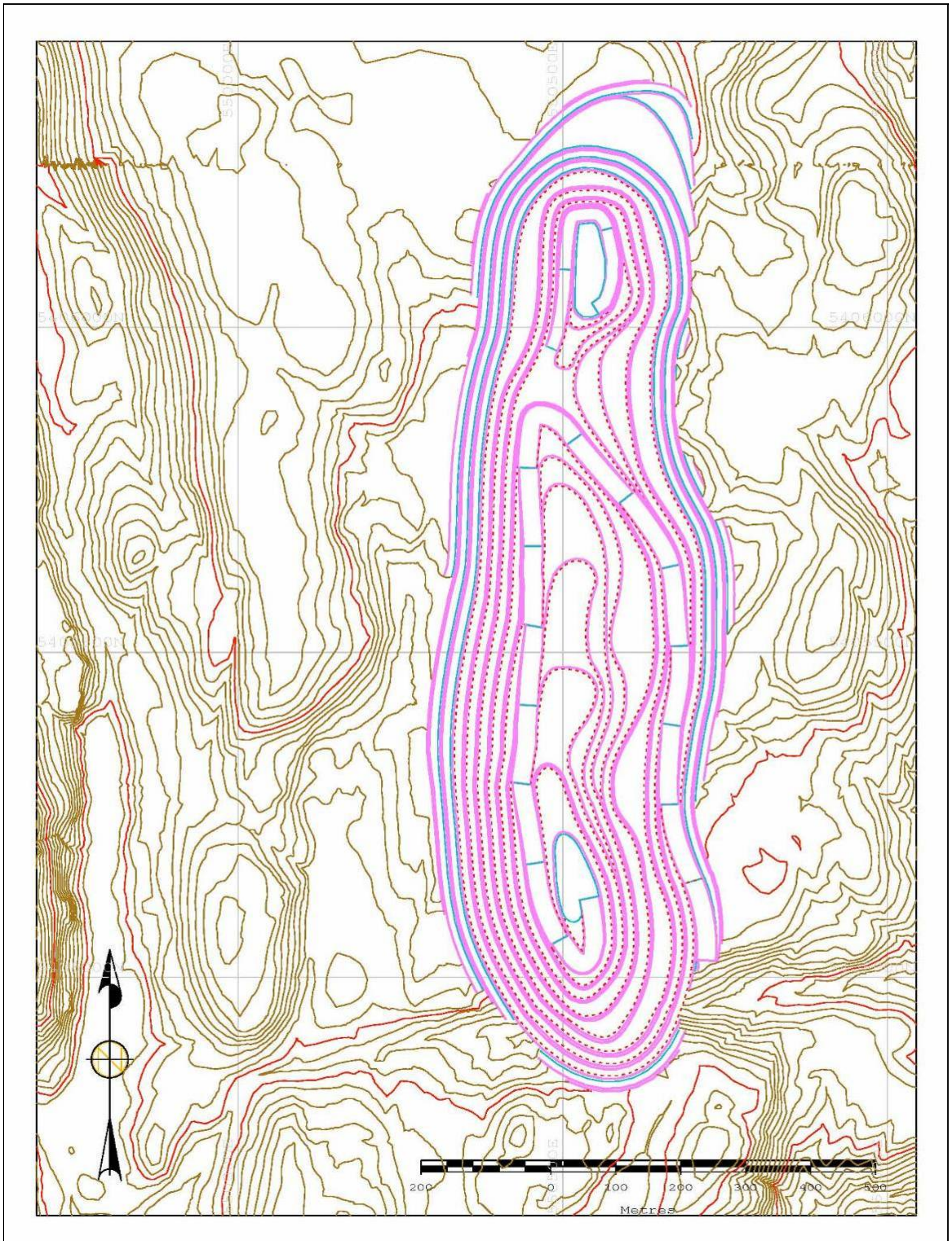


Figure 18.3
Marathon North Pit Phase 2 Design
(Post Mining)

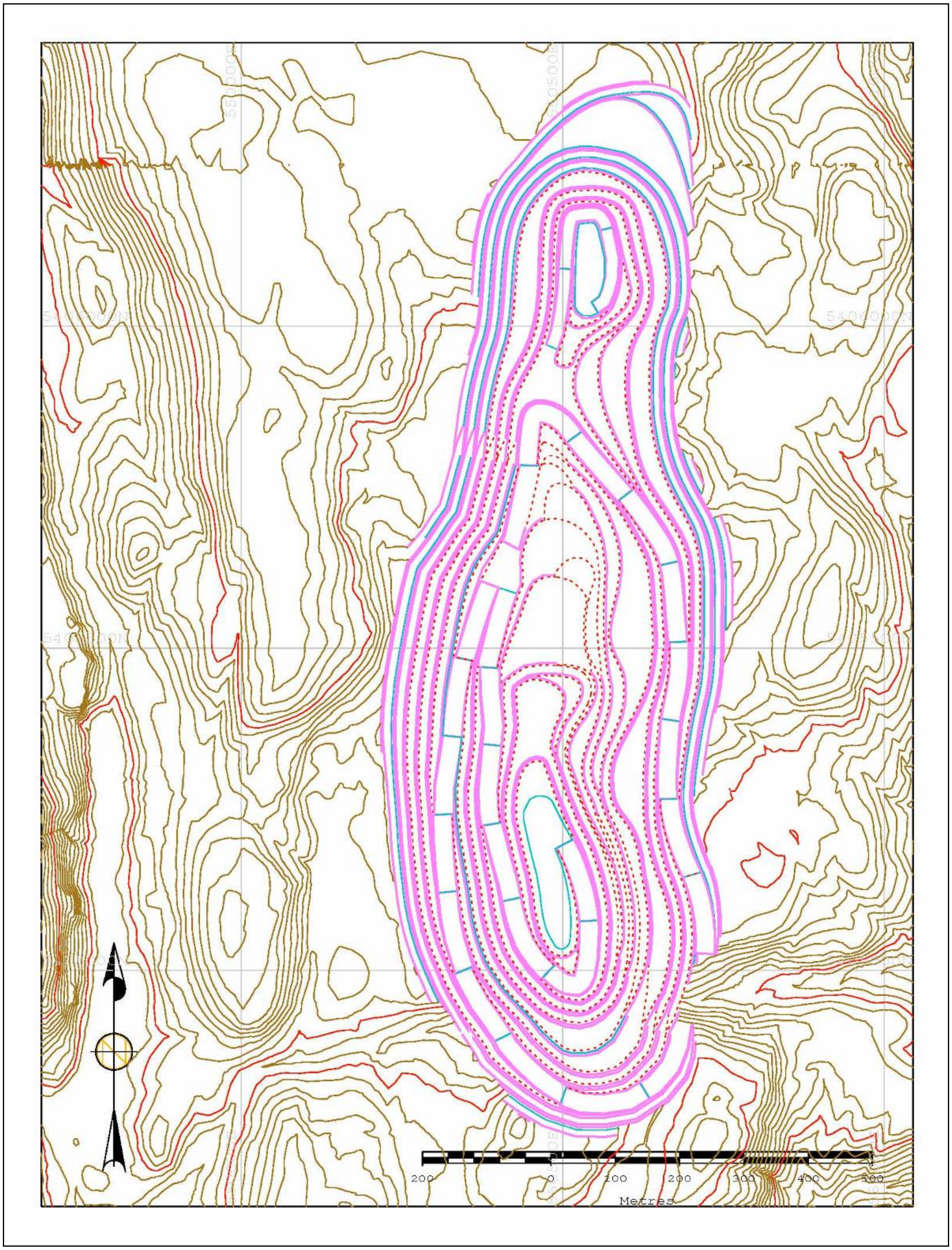


Figure 18.4
Marathon North Pit Phase 3 Design
(Post Mining)

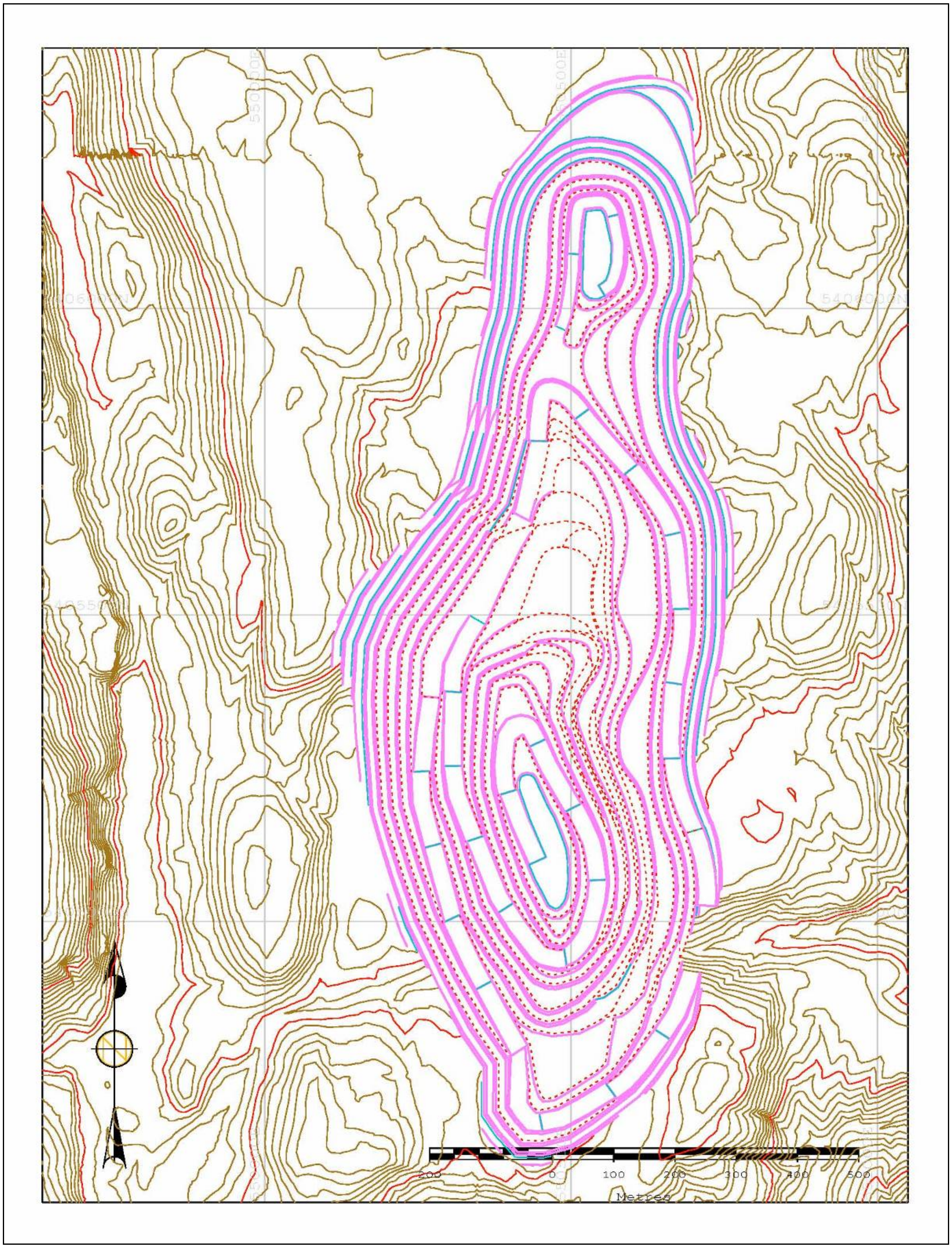


Figure 18.5
Marathon North Pit Phase 4 Design
(Post Mining)

Figure 18.6
Marathon South Pit Design
(Post Mining)

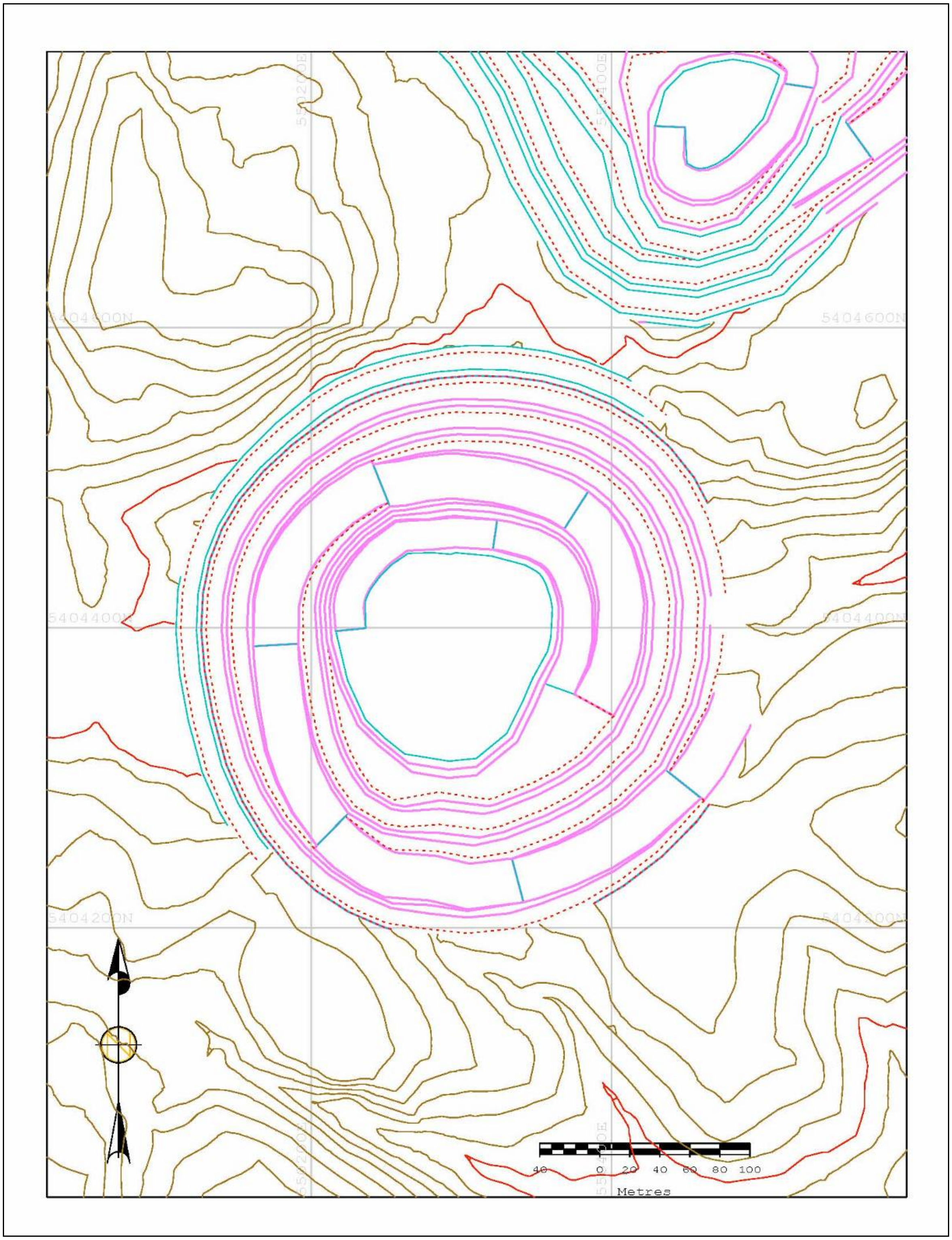
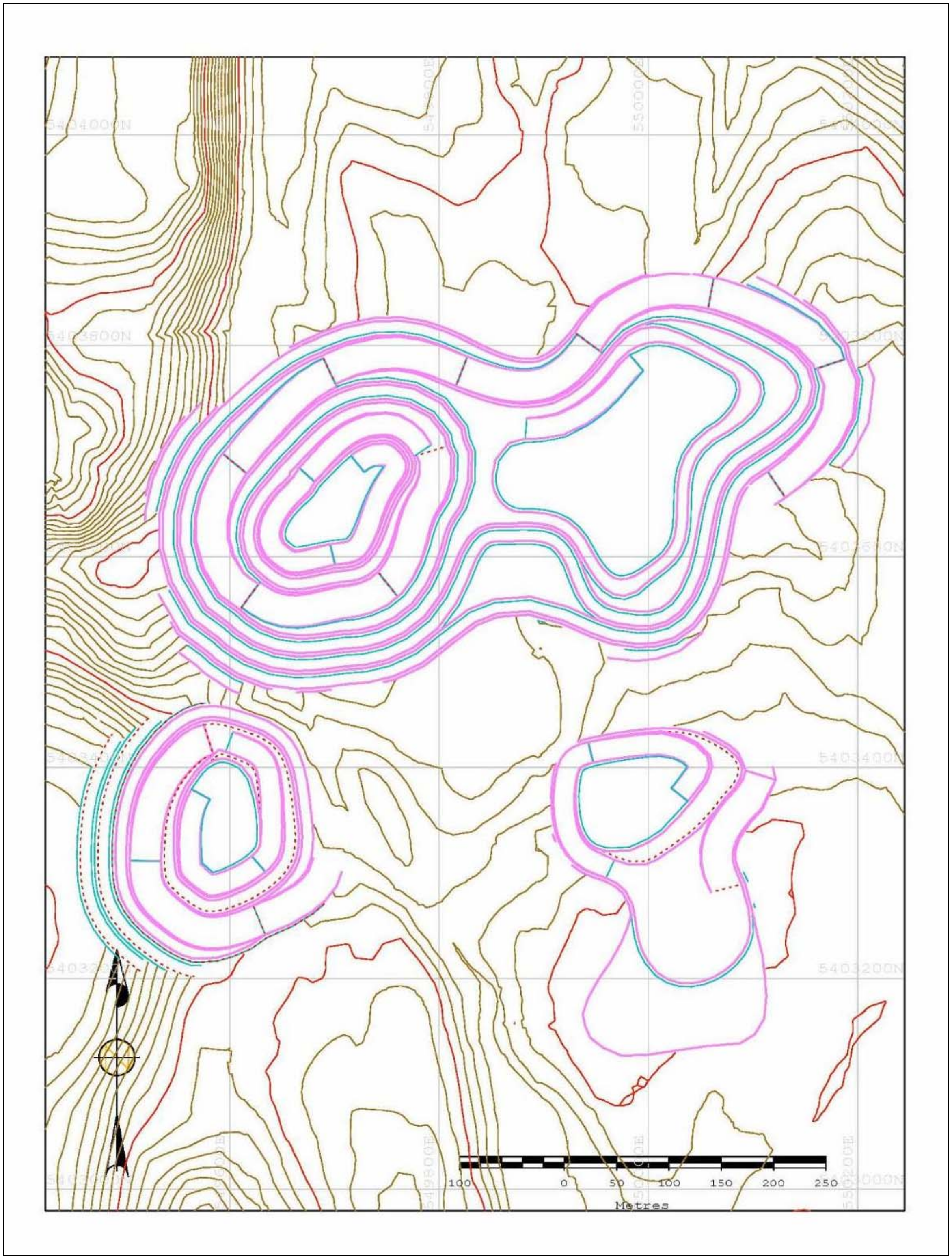


Figure 18.7
Marathon Malachite Pit Designs, Phases 1 through 3
(Post Mining)

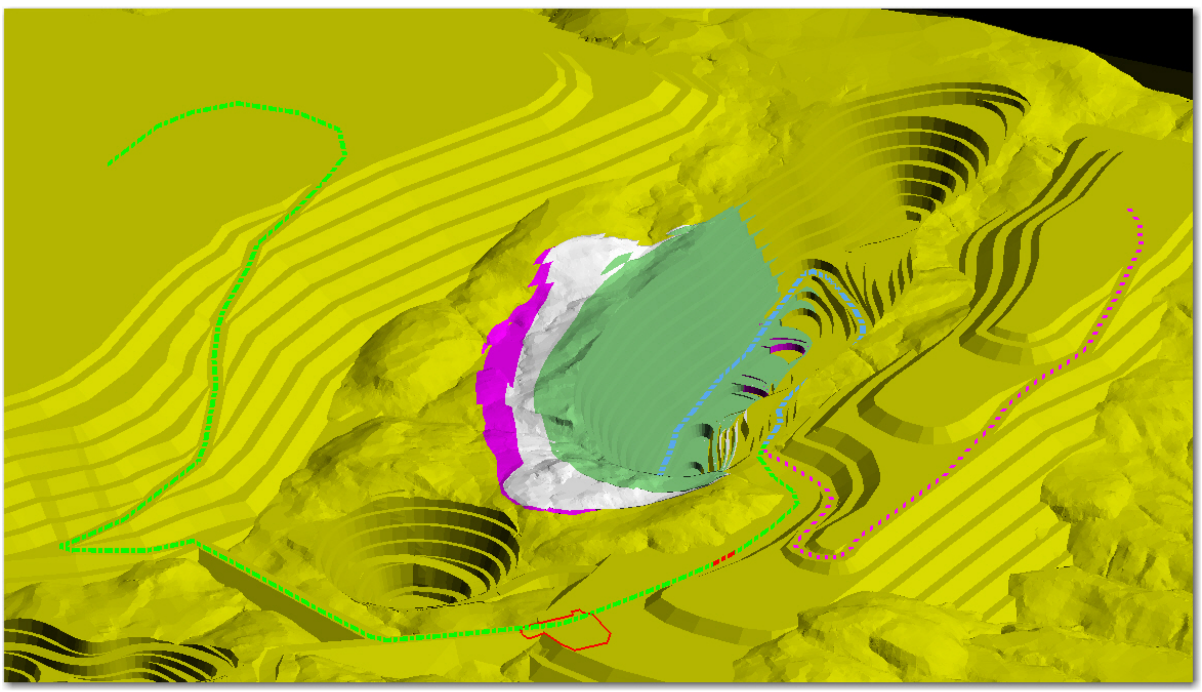


18.1.4 Haulage Simulation

In order to determine the required number of trucks in the production schedule it is necessary to complete a truck haulage simulation. For the Marathon PGM-Cu project pit designs, the Haulage Profile program within the Vulcan Mine Planning software was utilized to accomplish this task. This simulator assigns haulage information into the block model based on a haulage profile to each possible destination for each block. The haulage information is input into the program consisting of loading, spot, dumping, and delay times. Information regarding the selected haul truck performance loaded and empty is also entered into the program.

The haulage profile consists of the haul segments to various possible destinations. For the Marathon PGM-Cu, possible destinations included the crusher, west MRSA, and east MRSA. These hauls are digitized into Vulcan and registered to a topographic surface. An example of the haulage profile for the North Pit phase 1 design is shown in Figure 18.8 below.

Figure 18.8
Typical Haulage Profile for North Pit Phase 1

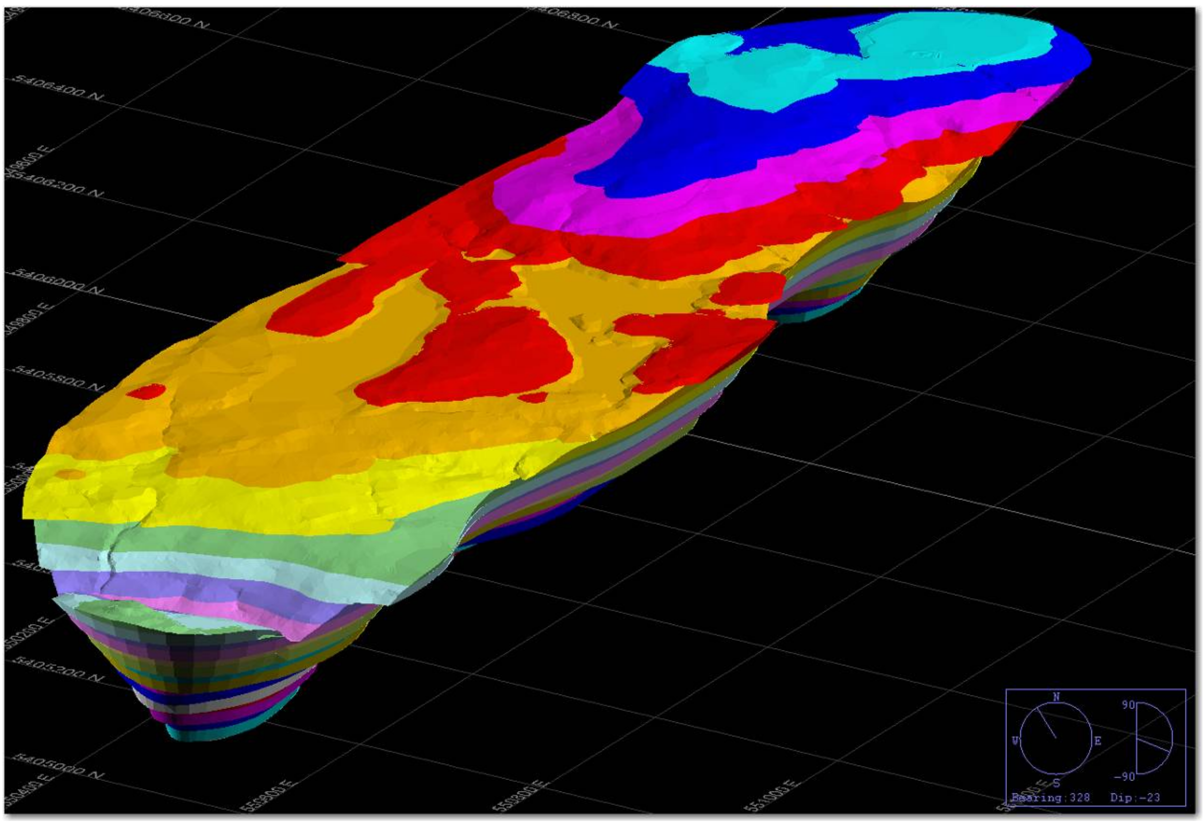


Once all of the profile data has been setup in the program, the haulage simulation is run. Output from the simulation is the average cycle time in minutes, distance in meters, and total time in minutes for the block to be moved to its assigned destination. This data is later extracted during production scheduling where it is used as a constraint during scheduling optimization.

18.1.5 Open Pit Production Schedule

An open pit production schedule was constructed using Maptek's Chronos scheduling software. The Chronos scheduler extracts reserve data from diluted block model and the scheduling solids developed in Vulcan. The scheduling solids for North Pit phase 1 are shown below in Figure 18.9.

Figure 18.9
Scheduling Solids for the North Pit Phase 1



All of the scheduling solids that make up the Marathon pit design have a single solid per bench per phase. Once the scheduling solid reserves have been extracted from the diluted block model, they are imported into a Chronos reserve spreadsheet. This spreadsheet is Excel based with special add-ons that function as a link back to the graphics within Vulcan. In the Chronos spreadsheet, relationships are developed between the reserve data and the structure of the final schedule built.

When the scheduling spreadsheet internal relationships have been established, the schedule can be constructed. For the Marathon PGM-Cu project production schedule, an optimizing program is used to determine the best economic solution while constraining according to truck hours, filling the mill, and keeping the mine rock stripping as balanced as possible. During the pre-production phase, a maximum of 500,000 t of mill feed ore is stockpiled to be consumed during the first year of production. During the first year of production a reduced annual capacity is assumed to allow the plant start-up period. Table 18.5 shows the life of mine production schedule.

18.1.6 Open Pit Operations

The Marathon PGM-Cu project is envisioned as a large scale open pit mining operation. Large loading and haulage equipment will be used to move material from the open pit to the crusher and the MRSAs. Mill feed from the pit will be hauled to the crusher. Below cut-off grade material, along with high sulphide waste, is to be hauled to the east MRSA. All other waste materials are to be hauled to the west MRSA.

Mining starts with pre-production in the quarter before the mill is ready to start operations. Limited ore production is envisioned for the first full year of operation. Starting in the second year of operation the full annual concentrator capacity is achieved at 8,030,000 t. Mining continues until the deposit is exhausted during year 12 of production.

Mining is started in North pit phase 1and continues in the North pit complex throughout the mine life. The Malachite pit is initially developed starting in year 3 and continues limited production throughout the mine life. The South pit is developed towards the end of the mine life.

18.1.6.1 Drill and Blast

Drill and blast operations are assumed to take place using a 12 m bench. In ore, burden and spacing is 5.25 m by 5.5 m. In waste, burden and spacing is 5.1 m by 7.1 m. All drill holes have a 2 m sub-drill. Final walls are pre-sheared. Production drilling is accomplished using a Sandvik D55SP drill. Blasting costs are estimated at $0.38/t ore and $0.32/t waste. Drilling costs are estimated at $0.17/t ore and $0.15/t waste. During the first several years of production, 3 to 4 production drills are required.

18.1.6.2 Loading and Haulage

Loading of blasted ore is planned to be handled by two O&K RH200 EL front hydraulic shovels. A secondary large wheel loader, a Cat 994F, is used as a backup primary loading unit. Haulage is by Cat 793F rear dump haul trucks. The maximum required trucks are 14 with a typical requirement of 13 during the first five years of mine operations. Loading and haulage costs are estimated to be $0.76/t of all material.

18.1.6.3 Ancillary Pit Equipment

To support mine operations, ancillary pit equipment has been selected that includes backhoes, utility haul trucks, utility wheel loader, dozers, motor graders, service trucks, water trucks, light plants, and small vehicles. Ancillary pit equipment is expected to cost $0.16/t all material.

Table 18.5
Marathon Life of Mine Production Schedule

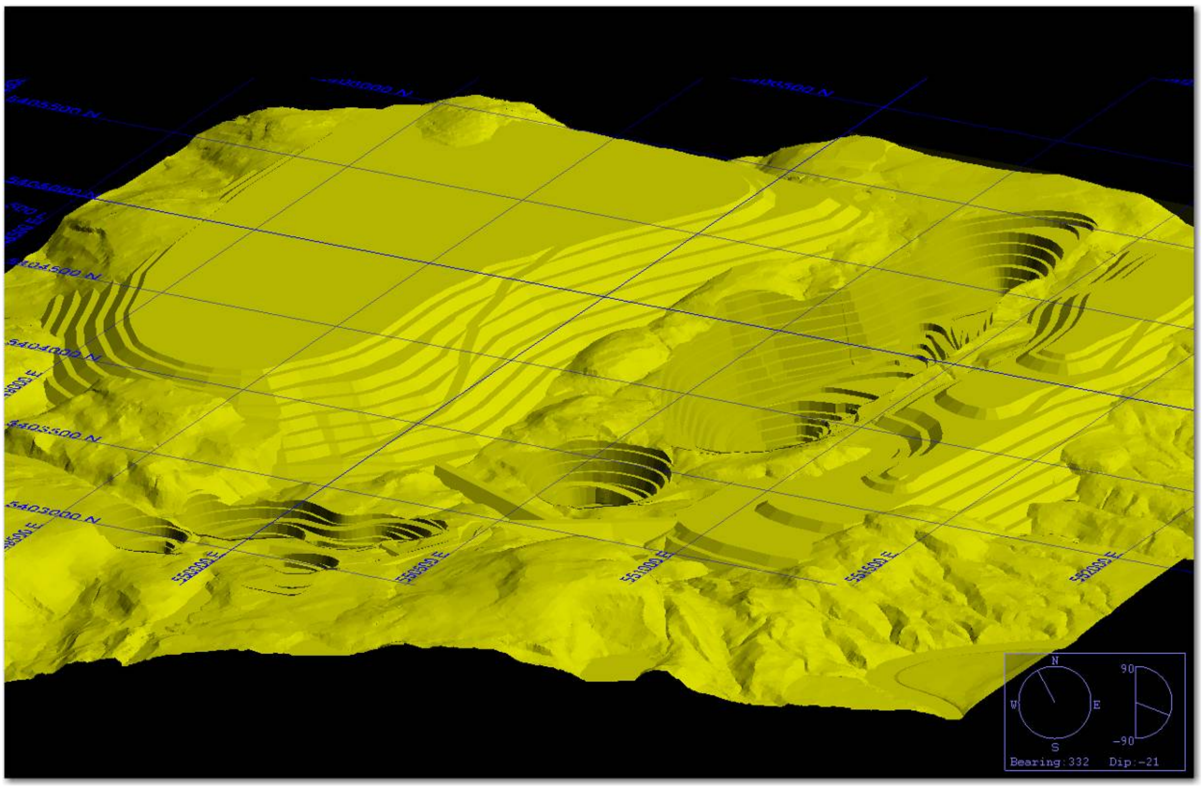
Period Timeframe	1 Q4 Yr -1	Total Yr -1	2 Q1 Yr 1	3 Q2 Yr 1	4 Q3 Yr 1	5 Q4 Yr 1	Total Yr 1	6 Q1 Yr 2	7 Q2 Yr 2	8 Q3 Yr 2	9 Q4 Yr 2	Total Yr 2
To stockpile tonnes	500,000	500,000	0	0	0	0	0	0	0	0	0	0
From stockpile tonnes	0	0	200,000	100,000	200,000	0	500,000	0	0	0	0	0
Ore tonnes	0	0	921,000	1,300,000	1,500,000	2,007,000	5,728,000	2,008,000	2,007,000	2,007,000	2,008,000	8,030,000
Total mill feed	0	0	1,121,000	1,400,000	1,700,000	2,007,000	6,228,000	2,008,000	2,007,000	2,007,000	2,008,000	8,030,000
Mine rock tonnes	2,399,000	2,399,000	4,979,000	7,051,000	7,950,000	8,950,000	28,930,000	8,551,000	8,224,000	7,730,000	6,919,000	31,424,000
Total tonnes	2,899,000	2,899,000	6,100,000	8,451,000	9,650,000	10,957,000	35,158,000	10,559,000	10,231,000	9,737,000	8,927,000	39,454,000
Strip ratio	---	---	4.44	5.04	4.68	4.46	4.65	4.26	4.10	3.85	3.45	3.91
Mill feed grades												
Cu (%)	0.000	0.000	0.235	0.222	0.227	0.238	0.231	0.246	0.251	0.256	0.258	0.253
Pd (g/t)	0.000	0.000	1.120	1.047	1.073	1.051	1.069	1.052	1.052	1.052	1.043	1.050
Pt (g/t)	0.000	0.000	0.316	0.276	0.275	0.262	0.279	0.262	0.262	0.265	0.265	0.264
Au (g/t)	0.000	0.000	0.111	0.097	0.097	0.093	0.099	0.092	0.091	0.090	0.090	0.091
Ag (g/t)	0.000	0.000	1.372	0.933	0.843	0.696	0.911	0.680	0.677	0.709	0.780	0.711
Rh (g/t)	0.0000	0.0000	0.0071	0.0099	0.0112	0.0091	0.0095	0.0066	0.0049	0.0036	0.0036	0.0047
NSR ($/t)	$0.00	$0.00	$32.73	$30.00	$30.44	$30.43	$30.75	$30.84	$31.14	$31.42	$31.46	$31.22

Period Timeframe	10 Yr 3	11 Yr 4	12 Yr 5	13 Yr 6	14 Yr 7	15 Yr 8	16 Yr 9	17 Yr 10	18 Yr 11	19 Yr 12		Totals
To stockpile tonnes	0	0	0	0	0	0	0	0	0	0		500,000
From stockpile tonnes	0	0	0	0	0	0	0	0	0	0		500,000
Ore tonnes	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	4,918,000		90,946,000
Total mill feed	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	8,030,000	4,918,000		91,446,000
Mine rock tonnes	28,000,000	28,000,000	28,000,000	28,000,000	18,000,000	18,000,000	18,000,000	18,000,000	12,056,000	4,663,000		263,472,000
Total tonnes	36,030,000	36,030,000	36,030,000	36,030,000	26,030,000	26,030,000	26,030,000	26,030,000	20,086,000	9,581,000		355,418,000
Strip ratio	3.49	3.49	3.49	3.49	2.24	2.24	2.24	2.24	1.50	0.95		2.88
Mill feed grades												
Cu (%)	0.254	0.255	0.262	0.265	0.268	0.253	0.200	0.225	0.258	0.223		0.247
Pd (g/t)	0.989	0.916	0.880	0.813	0.834	0.875	0.786	0.682	0.555	0.434		0.834
Pt (g/t)	0.260	0.249	0.242	0.229	0.230	0.258	0.255	0.212	0.185	0.152		0.237
Au (g/t)	0.087	0.089	0.090	0.089	0.087	0.092	0.086	0.078	0.067	0.062		0.085
Ag (g/t)	1.010	1.326	1.512	1.546	1.565	1.584	1.990	1.566	2.027	1.402		1.442
Rh (g/t)	0.0038	0.0050	0.0057	0.0048	0.0058	0.0093	0.0158	0.0080	0.0052	0.0062		0.0069
NSR ($/t)	30.73	30.00	29.99	29.28	29.60	29.93	26.17	25.34	25.37	21.29		28.53

18.1.6.4 Mine Rock Storage

Two MRSAs are envisioned for the Marathon PGM-Cu project. The first is the west MRSA with a capacity of 151.3 Mm^3 or 227 Mt. Total surface area impacted by this MRSA is 270 ha. The second mine rock storage area is the east MRSA with a capacity of 40.7 Mm^3 61 Mt. Total surface area impacted by this facility is 106 ha. Total mine rock storage capacity is 192.0 Mm^3 or 288 Mt. Figure 18.10 shows a view of the designed MRSAs.

Figure 18.10
Mine Rock Storage Areas



18.1.6.5 Pit Operations and Maintenance Personnel

Total pit operations and pit maintenance personnel ranges from a low of 94 in the last year of operations to a high of 177 during the fourth quarter of the first year of operations.

18.2 PROCESSING PLANT

The processing plant flowsheet and design criteria are based on the results from the metallurgical testwork, program discussed in Section 16.0 of this Technical Report.

The design of the 22,000 t/d concentrator comprises primary crushing, secondary crushing, HPGR, ball milling, flotation, concentrate dewatering and tailings disposal. The concentrator is designed to produce a copper sulphide flotation concentrate containing valuable PGMs and gold.

Micon was responsible for the high level process design including the general flowsheet, process design criteria, and major process equipment sizing. Met-Chem provided the detailed flowsheets and engineering input for the process facilities and related infrastructure.

18.2.1 Process Design Criteria

The basis for the design criteria includes the mineralogical and metallurgical testwork, with particular reference to the 2007/08 SGS-L program and 2009 XPS work (see Section 16.0), ore reserve and mining plan by Micon (see Sections 17.0 and 18.1) and Micon's in house process and engineering experience.

Table 18.6 summarizes the process design criteria. The operation is designed to treat 22,000 dry t/d of Marathon PGM mineralization from the open pit mine on the basis of a 24 hour per day, 7 day per week operation. The utilization factors used for the calculation of the nominal hourly flow rates are 60% for the primary and secondary crushing circuits and 90% for the remainder of the process facilities.

Table 18.6
Summary of the Process Design Criteria

Parameter	Units	Value	Source
Operating time	d/y	365	Micon
Operating time	h/d	24	Micon
Primary/secondary crusher operating criteria	d/week	7	Micon
Primary/secondary crusher utilization	%	60	Micon
Plant operating criteria	d/week	7	Micon
Plant utilization	%	90	Micon
Primary grind product size (80% passing)	µm	110	Testwork/Micon
Regrind mill circuit product size (80% passing)	µm	20	Testwork/Micon
Total rougher retention time	min	65	Testwork/Micon
Throughput			
Nominal annual throughput	kt	365	Micon
Design daily throughput	t	22,000	Micon
Primary / secondary crusher design throughput rate	t/h	1,528	Micon
Nominal plant feed rate	t/h	1,019	Micon
Run-of-Mine Ore Characteristics (plant design only)			
Maximum rock size	mm	500	Micon
Ore specific gravity		3.1	Micon
Ore moisture	wt %	3	Micon
Bond ball mill work index	kWh/t	16.5	Testwork
Metallurgical Efficiency (plant design only)			
Total recovery - Cu	%	90.8	Testwork
Total recovery - Pt	%	71.0	Testwork
Total recovery - Pd	%	80.1	Testwork
Total recovery - Au	%	79.9	Testwork
Total recovery - Ag	%	74.5	Testwork
Concentrate Cu grade	%	22.0	Micon
Concentration ratio		87	Calculation
Final concentrate production – nominal	dry t/d	251	Calculation

18.2.2 Process Description

Process flowsheets have been developed by Met-Chem.

The process selected for the concentration of the Marathon PGM-Cu mineralization is based on design criteria summarized in the previous section and the design, engineering and equipment selection by Micon and Met-Chem.

Mined ore-grade material is hauled by 215.5-t capacity Cat 793F rear dump mine trucks to the primary crusher situated on the eastern side of the North pit. The trucks dump the ore into a hopper over a 54-in by 75-in gyratory crusher. The primary crusher facility includes a dust collector and is enclosed within a partially winterized building. A rock breaker is provided at the crusher to break oversize rock.

Primary crushed ore (-150 mm) is conveyed onto a 22,000 t live capacity coarse ore stockpile from which it is reclaimed to the secondary crushing and screening plant, comprising a MP800 cone crusher. Product from the secondary crushing plant (-42 mm) is fed to the HPGR feed storage bins situated at the main plant facility.

The product from the secondary crusher circuit feeds the HPGR circuit The HPGR (1.7 m diameter by 1.8 m wide) product is screened at 4 mm, the screen oversize is recycled to the HPGR bins and the undersize feeds the 7.32 m diameter by 10.36 m long (24 ft by 34 ft) circuit, which operating in closed circuit with a cyclone pack. The cyclone overflow product, sizing 80% passing 110 µm, is routed to the flotation circuit.

The flotation circuit comprises two conditioners, a primary rougher stage, a primary cleaner stage, a secondary rougher stage, a secondary cleaner stage and a cleaner scavenger stage. The primary cleaning circuit comprises one stages of cleaning and two stages of secondary cleaning. The flotation circuit is based on the flowsheets developed by SGS-L and XPS, and is shown in Figure 16.2.

The final concentrate is thickened, filtered in a continuous vertical plate type pressure filter and stored in a stockpile located on the ground floor of the mill building, under the pressure filter.

The concentrate is periodically loaded into trucks and transported to the concentrate storage and rail load-out area, which is situated in Marathon.

The reagents used in the flotation circuit are:

- Aerophine 3418A as a Cu and PGM collector; consumption 8.7 g/t of float feed or 191 kg/d.
- PAX as a copper sulphide collector; consumption 58.0 g/t of float feed or 463 kg/d.

- Hercules 7M as a gangue depressant; consumption 205.0 g/t of float feed or 4,447 kg/d.

- MIBC as a frother; consumption 7 g/t of float feed or 292 kg/d.

- Polyfroth W34 as a frother; consumption 7 g/t of float feed or 292 kg/d.

- Aerophine 3406A as a Cu and PGM collector; consumption 171 g/t of float feed or 3,772 kg/d.

A flocculant will be used for thickening the concentrate. The dosing rate is based on concentrate settling tests undertaken during the pilot plant work completed by SGS-L in 1986.

There are two distinct air systems designed for the process plant, the flotation air blower and the high pressure compressed air system.

Storage, pumping and piping systems have been included for the process water, fresh water and potable water systems.

Six process samples for the on stream analyzer are taken from the flotation circuit. The on-stream analyzer measures copper, iron and percent solids.

Two process solids (tailings) disposal systems have been considered, these are discussed in more detail in Section 18.4, Process Solids and Water Management. The base case option assumes sub-aqueous storage and the alternative considers separate land storage of non-acid and acid producing tailings. The latter case assumes the flotation of the acid producing sulphides from the tailings stream and separate pumping, piping and storage systems.

Met-Chem produced a detailed process equipment list and mass balance. The equipment selection and sizing was based on the flowsheets, design criteria and the mass balance.

18.3 INFRASTRUCTURE

The overall site layout (Met-Chem Drawing No. 29041-0102-L) for the base case process solids disposal system, which shows the mine, process solids management facility (PSMF), mine rock storage areas, process plant facilities and surface infrastructure, is presented in Figure 18.11. The site layout for the alternative land based process solids management Option A1 is presented in Figure 18.12 (Met-Chem Drawing No. A1-29041-0002-L-OC). The plant site arrangement showing process infrastructure is shown in Figure 18.13 (Met-Chem Drawing No. 26103-0001-L-OG).

Figure 18.11
General Site Plan Layout – Sub-aqueous Process Solids Disposal Option

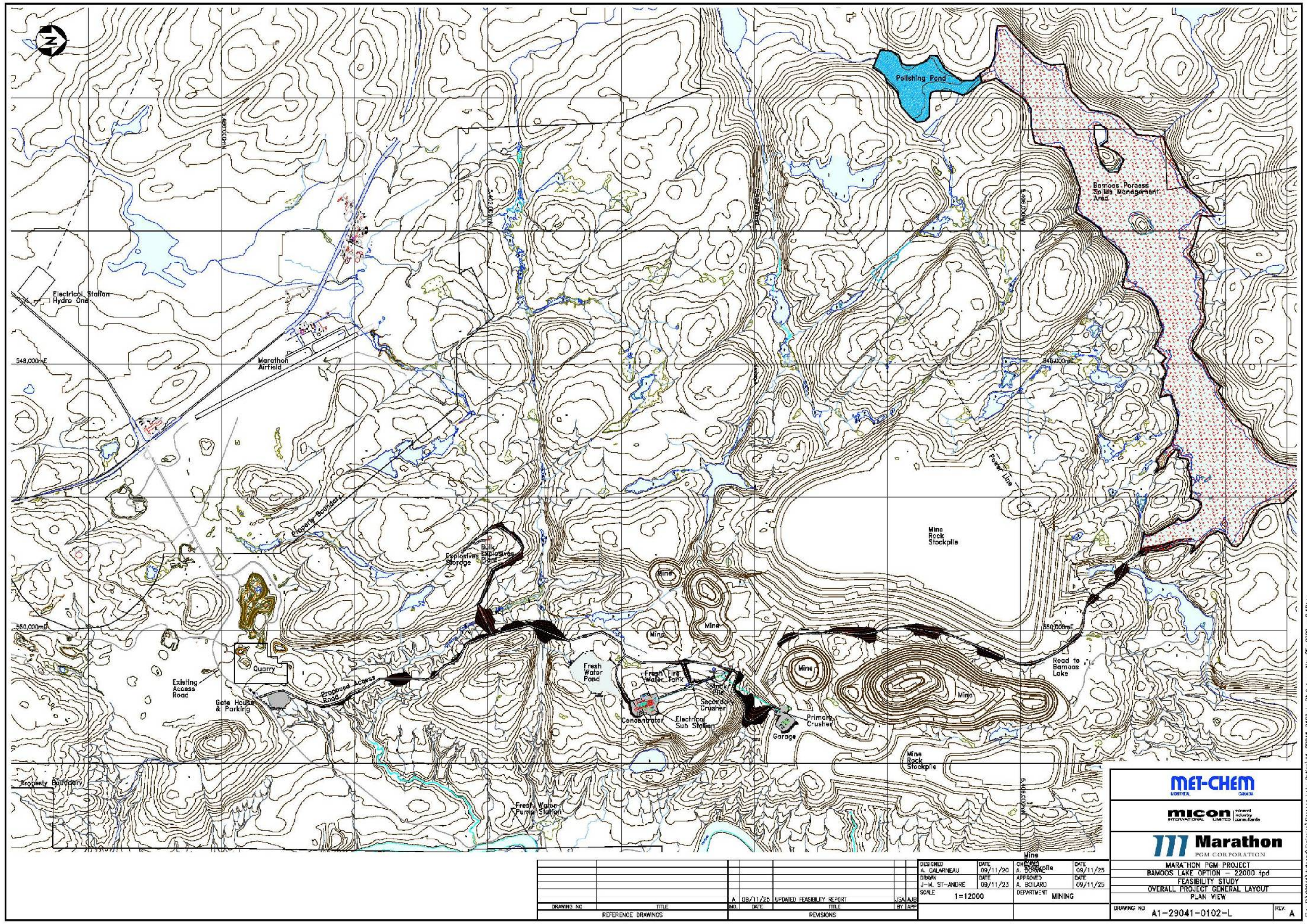


Figure 18.12
General Site Plan Layout – Land-based Process Solids Disposal

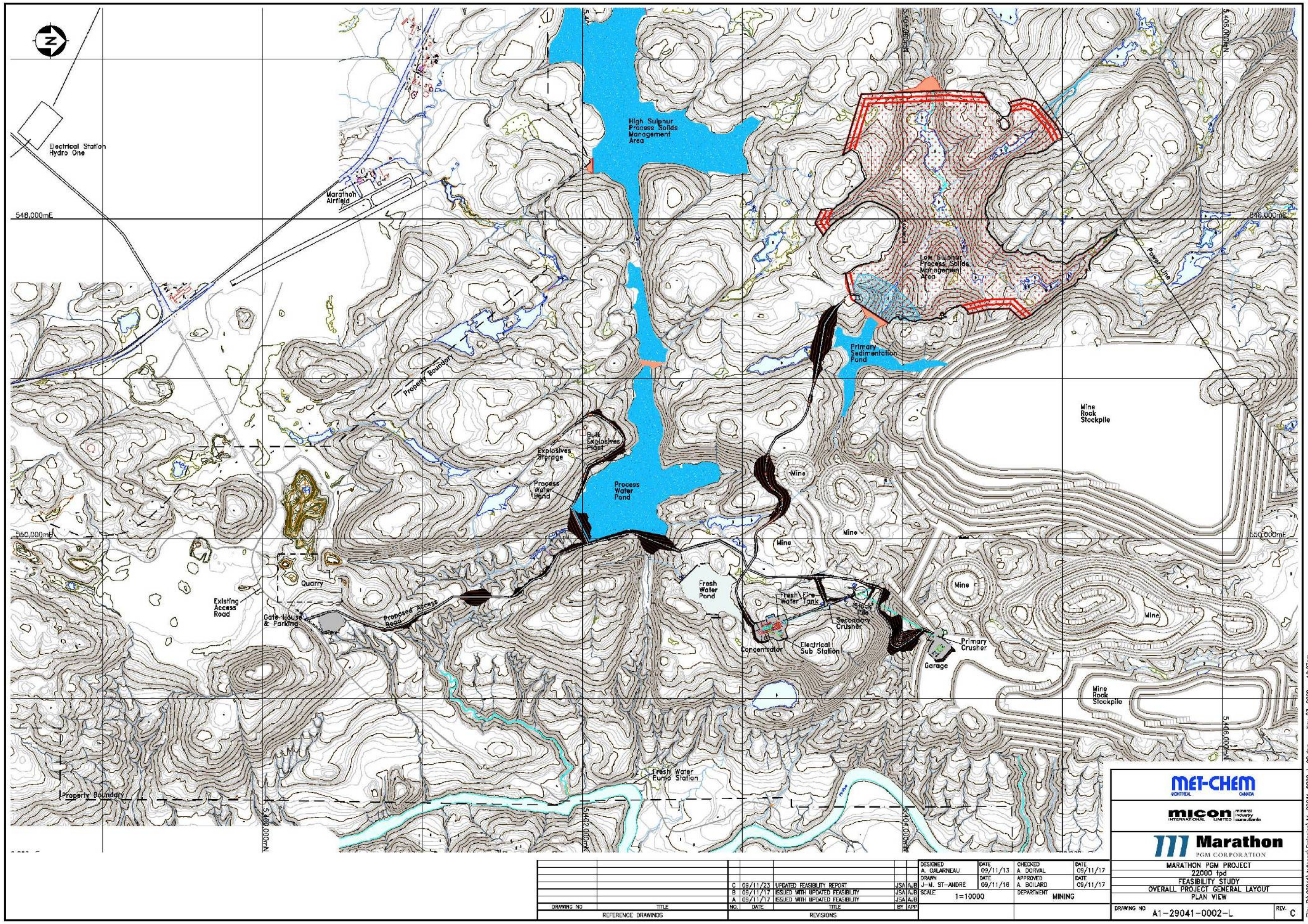
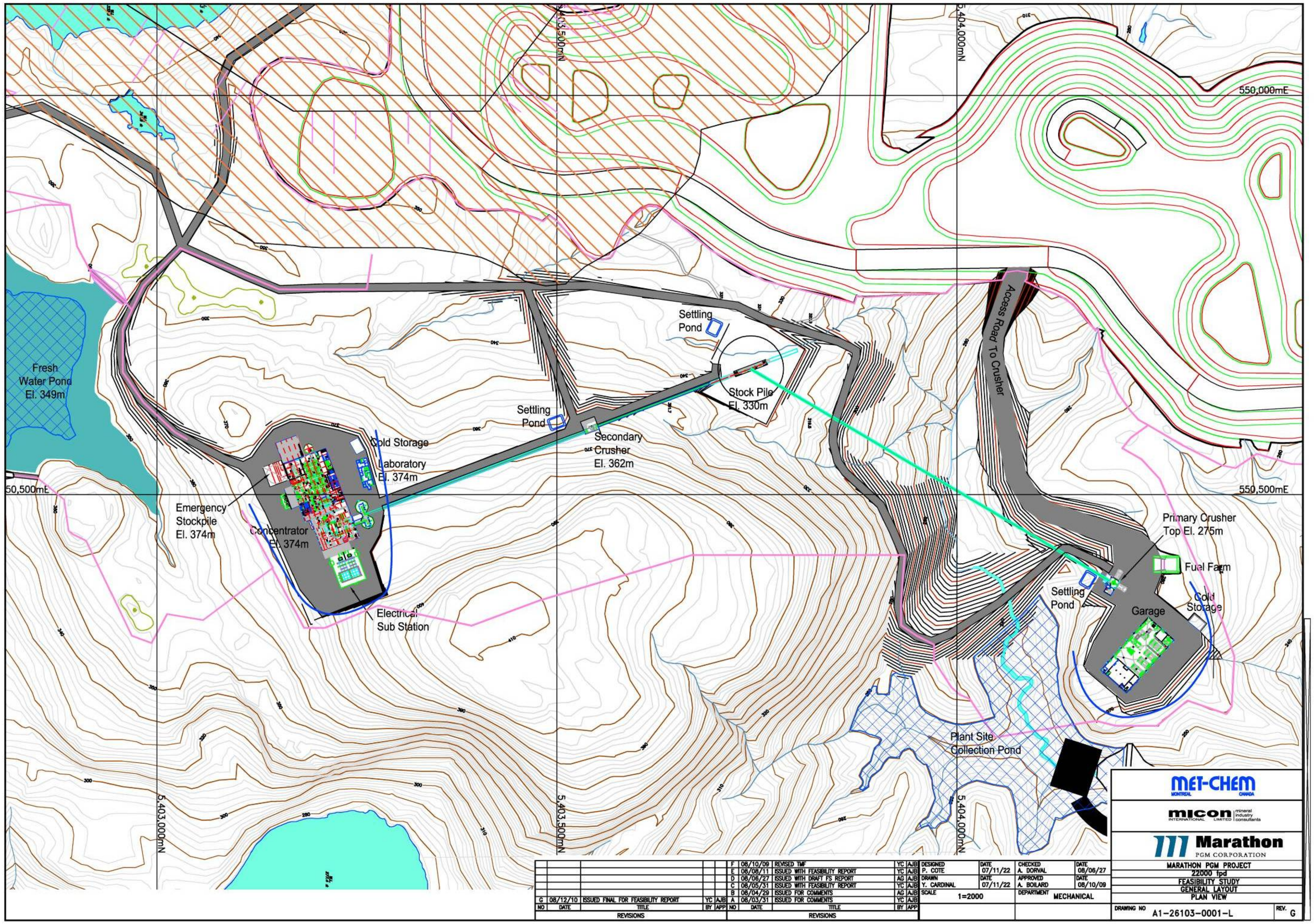


Figure 18.13
Process Area Site Plan Layout



18.3.1 Roads

Access road to the site will be routed in a northeast direction from the extension of Peninsula road branching north from the Trans Canada Highway No. 17 at the Marathon Town intersection. At 2.2 km north of the highway intersection on the existing gravel road, a new access road will be constructed to the mine site. The existing road is considered to be suitable for site access without significant upgrading. A new security gate and parking lot will be erected at the branching point with the existing road.

The new access road branching northwest from the existing road to the Marathon PGM-Cu project site will be around 3.0 km long. The construction is planned using on-site material to form the base complemented by a coat of non-acid generating mine rock fill and road surfacing consisting of 150 mm of borrowed pit material.

The site access gate, unmanned gate house and parking area (14,000 m^2), with the capacity for approximately 150 vehicles, are located at the end of the new access road. The gate will be remotely operated and the facility will include CCTV and radio communication.

The site roads included in Met-Chem's scope for the Feasibility Study were the following:

- The road from the gate to the plant site facilities.
- The fresh water intake road from the Pic River (existing to be slightly upgraded).
- The mine haulage road (from pit edge to the primary crusher and maintenance garage).
- The process solids (tailings) management area access road.
- The mine bulk explosive plant and explosive storage road.
- The road from the concentrator plant to the primary crusher.

Other site roads, including mine roads, PSMF area roads and the effluent treatment plant roads are included in the scope of work for mine and PSMF design.

All site roads are constructed from non-acid generating mine rock covered by 150 mm of borrow pit run material. Drainage culverts are included. Rock cut quantities kept to a minimum and maximum slopes of between 8 to 10% for roads were used in the design.

18.3.2 Construction Camp

Due to the limited availability in the Marathon area, a construction camp has been included to accommodate up to 300 workers/supervisors/technical assistance personnel for the construction period. A preliminary investigation has shown that the rental option will be more economical than the purchase of a camp as renting permits the possibility to ramp up the availability to match the construction schedule. The camp will comprise 49 single room units (6 units at peak), a 7.3 m by 18.3 m recreation complex and 7-unit kitchen complex.

18.3.3 Plant Buildings and Facilities

18.3.3.1 Primary Crusher Building

The primary crusher building is constructed of mainly concrete with foundations sitting on bedrock. Access is provided at the haulage road elevation, which is the top of the building and at the bottom via a service road at elevation 262 m.

18.3.3.2 Ore Stockpile and Reclaim Tunnel

A concrete structure is designed to house the reclaim system underneath the 22,000 t live capacity coarse ore stockpile. The structure is two stories high and is connected at both ends by tunnels leading to the open surface.

18.3.3.3 Secondary Crusher Building

The secondary crusher building is a conventional structural steel structure mounted on spread concrete footings with grated elevated floors.

18.3.3.4 Concentrator Building

The concentrator building is a conventional ore processing type design. The dimensions are 56 m by 119 m and the concentrator building houses the grinding, flotation, reagent and filtration areas, the air compressors and the process solids pumping system.

The high voltage electrical room (9.5 m by 35 m) is located on the east side of the concentrator and the low voltage electrical room (9.5 m by 20.7 m) is located at the middle of the south wall.

The concentrate load-out area, including a 100-t capacity truck weigh scale, is located at the south west corner of the building.

The mill dry, electrical and mechanical shops are located on the basement floor and occupy the northwest corner of the mill. The mill dry is sized to accommodate 60 men and 20 women.

Mill offices, conference room, administration, engineering, metallurgist offices lunchroom and washrooms are provided for on the second floor and third floors

18.3.3.5 Laboratory

The laboratory building is located approximately 20 m north of the concentrator building. The design is a pre-engineered single storey 13.4 m wide by 35 m long steel structure.

18.3.4 Mine Equipment Maintenance Building

The mine equipment maintenance building comprises a standard steel structure with siding and concrete foundations, typically used at similar mine sites. The building includes a wash bay, six major equipment maintenance bays, a tire shop and service area, a drill repair and bit dressing area. The building is designed for the Hitachi EH4400ACII haul truck or similar unit such as a Komatsu 830E AC model. It can also accommodate the Hitachi EX5500-6 diesel-hydraulic shovels, the Letourneau L1850 wheel loader, as well as smaller equipment.

18.3.5 Site Water Systems

The site water systems comprise the fresh water supply system, the process plant site surface water run-off collection systems and the process water collection and pumping system.

The fresh water is supplied from two sources, the Pic River and the fresh water pond, which is located immediately south west of the concentrator. The planned Pic River pumping station is situated about 2.9 km south east of the plant and is accessible by the existing road.

The process water pond located approximately 1 km southwest of the concentrator is designed to collect the main plant area surface water drainage. The process water pumping system consists of a building/pumping package that includes two vertical turbine pumps.

Three relatively small collection ponds will be provided to catch surface run-off from the secondary crusher area, the coarse ore stockpile area and the maintenance garage/primary crusher area.

Pit wall seepage and in-pit run-off will be collected using conventional pit sumps and stage pumped to the water collection pond to be located near the mine rock storage facility, and thereafter pumped to the water treatment plant and/or reclaimed for use in the plant. Perimeter ditches around the pit will be used to intercept run-off.

18.3.6 Potable Water Treatment

Fresh water will be pumped to the water treatment system which comprises a series of four multimedia filters (sand-anthracite). Filtered water will either be used for gland seal of further treated using a membrane filter unit (nanofiltration) to provide potable water suitable for approximately 120 workers.

18.3.7 Heating, Ventilation and Air Conditioning

Exhaust fans are installed for each building to ensure proper air change and temperature control in the summer.

The major buildings and facilities are heated by propane direct fired units. These buildings include the main concentrator building, the concentrator electrical rooms, and the mine equipment maintenance building.

The primary and secondary crusher buildings and the reclaim tunnel are partially heated with the use of electric space heaters located at specific location in the buildings. The heating requirements of these buildings are designed to prevent electric and hydraulic unit rooms from freezing.

Support buildings are heated by electrical baseboard heaters.

18.3.8 Fuel Storage and Fuelling Systems

18.3.8.1 Diesel Fuel Storage

The diesel fuel storage is located northwest of the maintenance garage facilities on the north side of the pit access road. The fuel storage system consists of dual wall, horizontal aboveground storage tanks, installed complete with dispensing pump, tank monitoring package, and a control system.

18.3.8.2 Propane Storage

Two separated propane systems have been allowed for heating the concentrator building and the maintenance garage.

18.3.8.3 Gasoline Storage and Distribution System

The gasoline storage facility is located on the northwest corner of the concentrator pad close to the main access road entrance. One gasoline storage tank has been allowed for, comprising an 11,500-L capacity dual wall horizontal tank. The storage facility will be installed with a dispensing pump package.

18.3.9 Fire Protection System

The fresh water tank has been sized to accommodate the fire water requirements. The heated fire water pump house, which is adjacent to the fresh water tank, includes two diesel fire pumps, one electric fire pump, one electric jockey pump and a diesel storage tank.

The fire water distribution system comprises an underground (2.13 m deep) 10-in diameter piping loop connecting the concentrator, the maintenance garage and the process solids thickening plant. The piping loop feeds 10 hydrants and 12 fire water lines feeding the buildings.

The fire water alarm system comprises a state of the art intelligent multiplex detection system complete with emergency signalling.

18.3.10 Plant Mobile Equipment

The following mobile equipment has been included in the scope of the study to service the plant and surface infrastructure requirements.

18.3.10.1 Site Service Equipment

- One service truck - boom truck 20-t.
- One fuel tanker truck.
- One water tanker truck.
- One lube truck.
- One rescue truck.

18.3.10.2 Handling Equipment

- Two skid steers, JD.
- Two fork lifts, propane.
- One articulated man lift 60-ft.

18.3.10.3 Earthworks & Grading Equipment

- One load-out loader.
- One loader and snow blower.
- One grader 14-ft blade.
- One sand truck.

18.3.10.4 Transportation Vehicles

- Six pick-up trucks.
- One bus – 40 passengers.
- One bus - 24 passengers.

18.3.11 Waste Disposal

The sanitary waste water system consists of collecting and treating all sanitary waste water from each building via an underground piping network. The waste discharges in a modularized sanitary waste water treatment unit utilizing a rotating biological contactor (RBC). One sanitary waste water treatment unit will be located near the primary crusher and the maintenance facilities while the other treatment unit will be located near the concentrator building. The RBC systems have been selected to meet the effluent quality and natural environment discharge standards of 15 mg/L biological oxygen demand (BOD) and 15 mg/L suspended solids.

It has been assumed that garbage disposal will be handled by a local contractor. No allowance is included in the capital cost estimate.

18.3.12 Explosives Plant and Storage

It is assumed that a bulk explosive supplier will erect a bulk explosive mixing plant on the Marathon PGM property. The required services (power, water etc.) will be provided by Marathon PGM and these costs are included in the capital cost estimate. Explosives to be used for pre-stripping and secondary blasting will be stored in a magazine provided by the explosive supplier. The magazine for detonators, also provided by the supplier, will be located nearby. The magazine is located approximately 500 m to the west of the open pit and the design is based on a maximum storage quantity of 30,000 kg, as stipulated by the National Resources Canada (NRCan) explosive quantities/distances regulations.

18.3.13 Concentrate Load-out Facility

Concentrate will be filtered, stored and loaded into trucks at the concentrator. These trucks will deliver the concentrate to a concentrate rail load-out facility located in the Town of Marathon. The proposed conceptual storage and load-out track layout system assumes that Marathon Pulp Inc. in Marathon will grant access to its property to Marathon PGM.

The area of the concentrate storage building at the railway load-out facility is1,645 m² and is sized to store about 7 days of production or approximately 1,800 dry t.

18.3.14 Electrical Power Supply and Distribution

18.3.14.1 Power Supply

The mine site will be fed by a new 115 kV overhead electrical power line connected to the existing Hydro One power line which feeds White River. The proposed connection point is located near the Hydro One distribution substation just north of the town of Marathon. The main site 115 kV power supply line will be a three phase single circuit line, will be about 8 km long and installed on wooden type structures.

Power transport capacity of the line is designed to fully provide the maximum electrical demand of the mine, which is estimated at about 31.7 MVA using a power factor of 95%.

18.3.14.2 Main Substation

Main power substation is located outdoors near to the major electrical loads of the concentrator.

Two power transformers are included in the design. A 25 MVA, 115 kV to 13.8 kV transformer feeds the high voltage loads and a 15 MVA transformer supplies the medium voltage electrical loads. Both transformers are pad-mounted, oil-filled with forced air ventilation.

18.3.14.3 Electrical Distribution

Electrical room No.1 contains the main switchgear to distribute power at 13.8 kV and 4.16 kV.

The major concentrator electrical loads, such as the 13,500 kW ball mill and the 3,725 kW motors of the HPGR are fed at 13.8 kV. Also, a feeder at 13.8 kV feeds the water services (process water, fresh water, reclaim water and process solids thickening facilities). Medium range power loads (150 kW to 2,250 kW) are fed at 4.16 kV.

The electrical design criteria incorporate systems to limit inrush power during the starting of major electrical equipment. This design philosophy meets the requirements of Hydro One for the connection of the plant equipment onto the Hydro One network.

Three pad-mounted, oil-filled 2,000 kVA transformers reduce the 4.16 kV power to 0.6 kV to feed the low voltage (LV) distribution centres installed in electrical room No.2.

The transformers and the LV electrical room No. 2 are installed as close as possible to the LV loads located in the concentrator building in order to minimize distribution costs.

Power at 4.16 kV is fed from electrical room No. 2 to the secondary crusher, the stock pile and the primary crusher/mine areas. Cables are sized to provide capacity for starting of the gyratory crusher and the other important equipment (conveyors, secondary crusher) direct on line. The mining area power requirements are estimated at around 1,000 kW.

A 13.8 kV line from electrical room No. 1 feeds the water pump houses. This line supplies power for process water, fresh water, reclaim fresh water, process solids thickening plant and the bulk explosive plant.

Due to the gate distance from the concentrator plant, Met-Chem has suggested to feed the power requirements for the gate area directly from Hydro One’s medium voltage distribution network.

18.3.14.4 Emergency Electrical System

An emergency motor control centre (MCC) is included in order to supply emergency power in case of interruption of the main power supply. The emergency MCC is located in the low voltage electrical room No. 2 and will supply the emergency loads near the main concentrator building. When a power failure occurs, an automatic switch transfers the MCC to the emergency circuit and a signal is given to start the 660 V, 1,200 kW emergency diesel generator.

18.3.15 Automation and Control Systems

Twenty piping and instrumentation drawings (P&IDs) covering all areas of the process have been developed by Met-Chem. These drawings are based on process flow sheets, potential supplier preliminary drawings, technical information and Met-Chem's in-house database.

The project instrument list was developed from the P&IDs. All the instruments will be integrated in the programmable logic controller (PLC) and will be wired to the PLC analogue input/output module located in different electrical rooms through the plant with a standard 4-20 mA signals. The control loops for the entire process will be integrated in the PLC control system with manual and automatic modes available at all the supervisory control and data acquisition (SCADA) interface operator stations in the plant.

The design of the PLC control system includes one system server, one historian server, two operator's workstations located in the central control room and three remote field units to supervise and control the plant operation. The engineer station is located in the ore processing electrical room for system programming and maintenance debugging.

The main PLCs in the ore processing plant for the grinding area and the flotation/reagent/services areas will have remote I/O rack cabinets installed to collect data.

The SCADA system will include development license and run time license for the supervision and control of the entire plant operation and have the capacity to communicate with most management computer packages available on the market.

18.3.16 Communications

The ore processing plant, the crushing and mine areas will be interconnected through a network via a redundant ethernet communication fibre optic cable system, regrouping the PLC communication system and the plant communication system. This includes telephones, personnel computers, CCTV and alarms.

The proposed communication system for the plant is based on Met-Chem's estimate of the typical requirements for an operation of this size.

18.4 PROCESS SOLIDS AND WATER MANAGEMENT

18.4.1 Introduction

AMEC of Pointe Claire, Quebec, was mandated by Marathon PGM to study and propose conventional slurry deposition PSMF options for the Marathon PGM-Cu project. AMEC's report presents three options based on criteria related to: the production objectives proposed by Marathon PGM; process data obtained by other consultants; the sulphur content of the process solids; the available meteorological data for the region; and the environmental criteria in effect (AMEC, 2009). AMEC's conclusions were based on basic design elements and criteria, preliminary analysis of potential sites, water assessments, evaluation of typical sections of dykes and dams, fill plans, material borrow areas and capital costs estimates.

The three options considered were:

1. Base case - Sub-aqueous storage of process solids in Bamoos Lake (see Figure 18.14).

2. Option 1A - Land-based separated low and high sulphur PSMF area with excess treated water discharge to the environment through the operational/emergency spillway of the high sulphur PSMA into Stream 6. (Figure 18.15).

3. Option 1B - Modified version of Option 1A to let out water to the environment directly to Hare Lake via Hare Creek.

18.4.2 Basic Design Elements and Criteria

Design elements consist of process solids characteristics, water assessment and infrastructure evaluation.

18.4.2.1 Process Solids

The following list shows the principal design elements and criteria concerning process solids:

- Life of mine is 11.5 years.
- Ore reserves are 91.156 Mt.
- Process solids to ore ratio is 0.986.
- Deposition slope is 0.3%.
- Void ratio is 1.1.
- Dry density of deposited process solids is evaluated at 1.48 t/m^3.
- Total volume of process solids is 60.73 Mm^3.
- 10% of process solids are high sulphur process solids.

Figure 18.14
Plan of Bamoos Lake Process Solids Management Facility Area

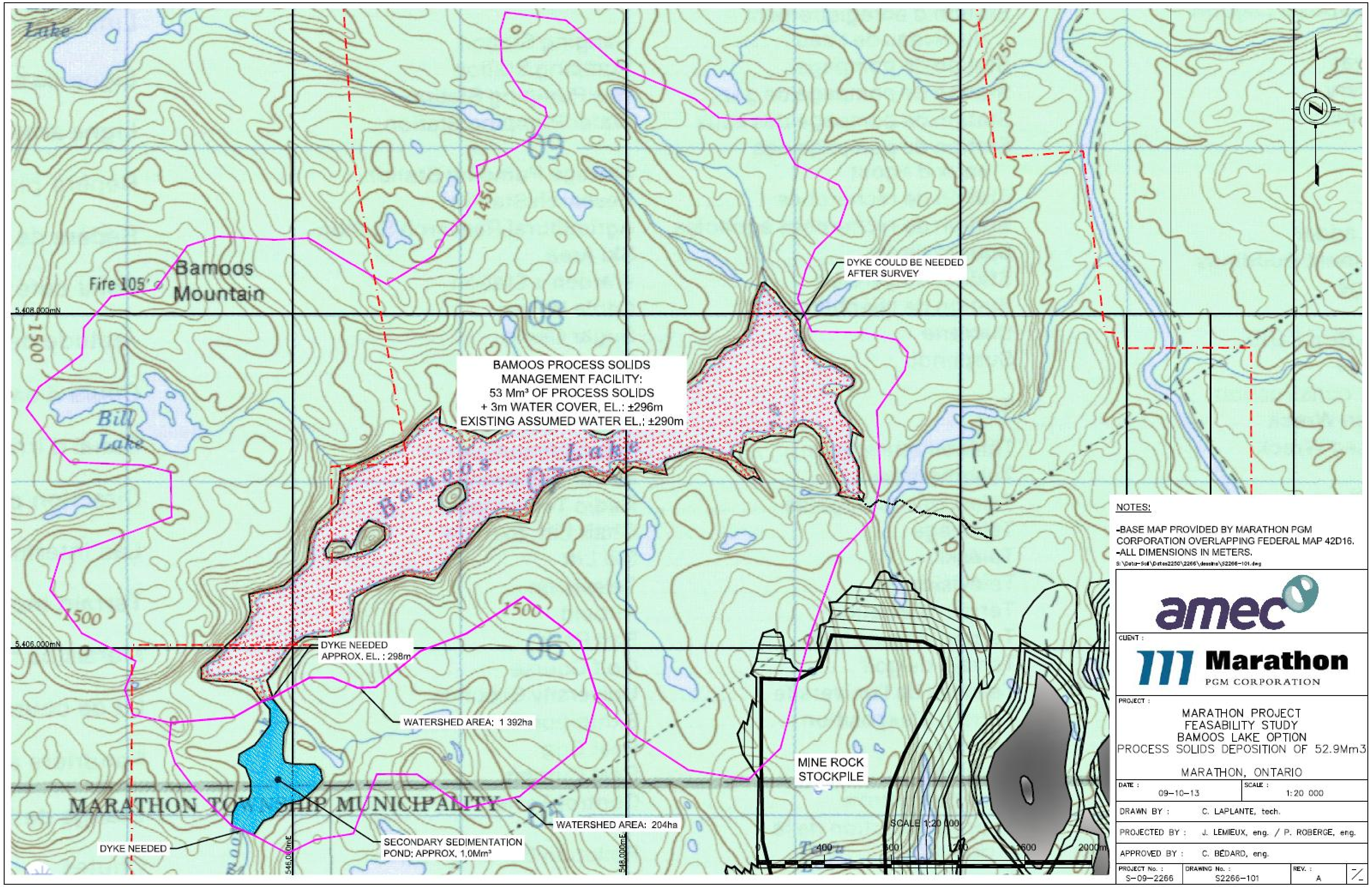
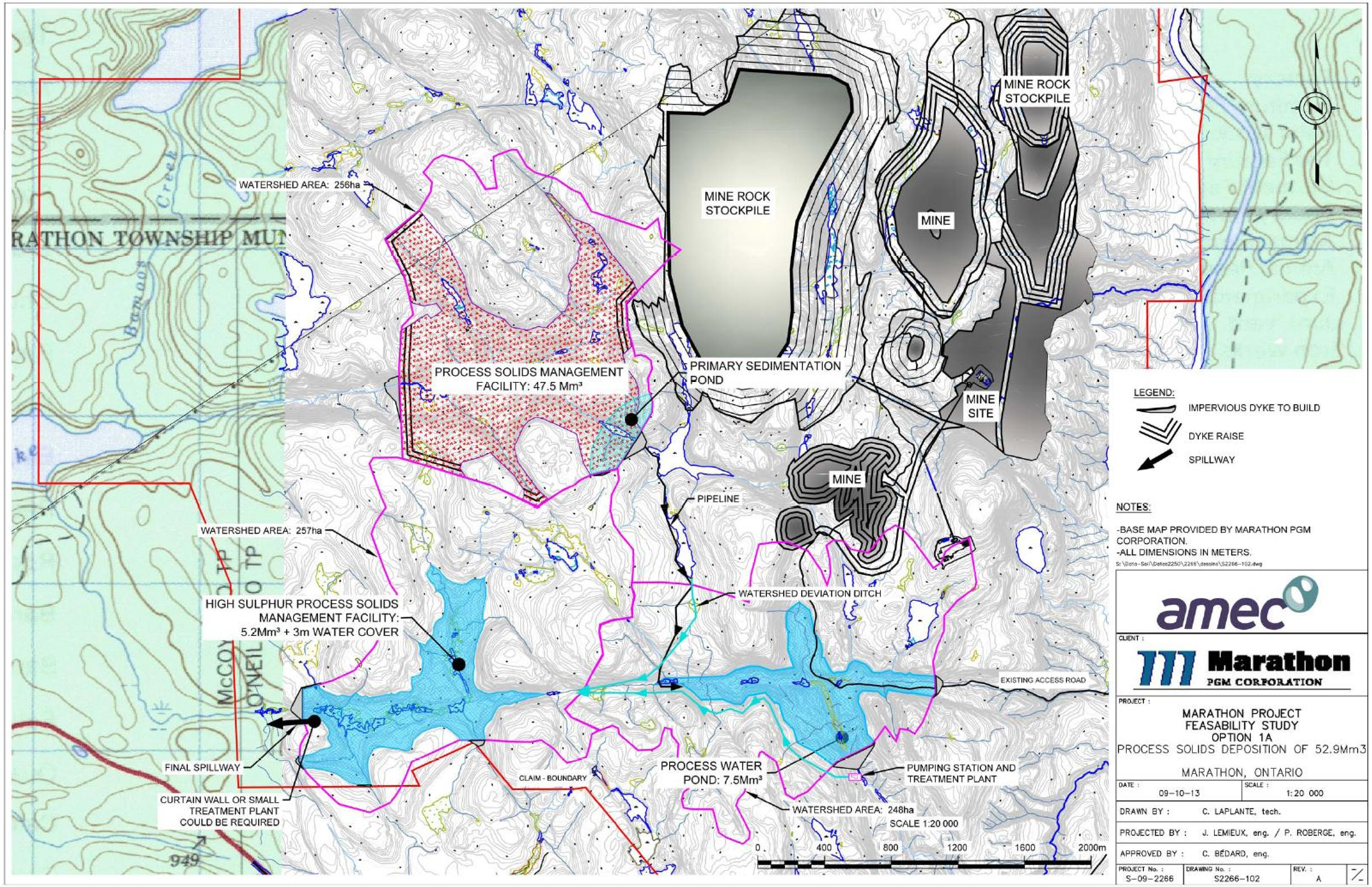


Figure 18.15
Plan of Option 1A Process Solids Management Facility Area



18.4.2.2 Water Assessment

Water Management Policies Guidelines and Provincial Water Quality Objectives of the Ontario Ministry of Environment and Energy must be respected. Since the waste control requirements for the proposed regulated sources of pollution are established on a case by case basis and that the Ministry of Environment and Energy may stipulate requirements for these activities related to the quality and quantity of the discharge in a Certificate of Approval, AMEC has chosen to base its water assessment on a dictated criteria use in the province of Quebec. Therefore, for environmental water management, the return period described in "Directive 019 d'avril 2005 du Ministère du Développement durable, de l'Environnement et des Parcs (MDDEP)" was chosen to calculate the inflow precipitation water. Using the specifications of "Directive 019" allows for a greater flexibility in future planning.

The water balance between the various ponds, influencing the construction of various infrastructure items, was evaluated from the inflow and outflow. The inflow includes the following:

- Precipitation (rain and snow).
- Process water present in the pulp coming from the mill.
- Water from a nearby system (if necessary).

The outflow includes the following:

- Water from the outlets (control structure, pumping stations, treatment plant and polishing pond outlet).
- Exfiltration water from the dams.
- Loss of water by evaporation.
- Water pumped by the mill.
- Pore water from the process solids retained in the PSMA.
- Unavailable water transformed in ice.

18.4.2.3 Dykes, Dams and Cyclone

Infrastructure requirements depend on the option evaluated. Dam and spillway design has been based on the Canadian Dam Association and the Ontario Dam Safety Guidelines.

Three types of dams will be used in this project: impervious dams built in one phase; impervious dams built in many phases; and dams raised with low sulphur process solids.

Dams raised with process solids are very economical. However, these dams can only be used to distance the water pond from the dam crest. For this to be achievable with the particle size evaluated for this project, a portion of the slurry will need to be passed through a cyclone to separate the water from the process solids. This will allow the silty material to drain, to be manipulated and trucked to the construction site. Table 18.7 presents the volumes of process solids required for dam raises in Option 1A.

Table 18.7
Volume of Process Solids Required for Dam Raises, Option 1A

	Required by Year -1	Required by Year 4	Required by Year 7	Required by Year 11.4
Cumulative volume (m^3)	0	177,000	572,000	1,234,000

18.4.2.4 Spillways and Transfer Structures

Operational spillways will be built for every pond that can drain by gravity, designed to prevent overtopping of the dams during the critical spring melt.

In Option 1B, a control structure can be integrated in the dam holding the process water pond since water can be transferred by gravity. The other options do not consider any storage in the primary sedimentation pond.

The emergency spillways are built in the rock next to the water pond to evacuate water flow from a simulated Timmins rainfall considering the time of concentration for the watershed.

Other infrastructure like a process water transfer pipe culvert, pumping stations, control structures and ditches are required in different options. Note that Option 1B requires a cyclone to be able to stockpile low sulphur process solids for dam raises.

18.4.3 Preliminary Analysis of Potential Sites

Preliminary analysis of potential sites was undertaken in four phases. These phases allowed AMEC to determine the best PSMA according the elements of design.

Phase 1: Eight options were evaluated. After analysis considering underlying ore bodies, mining property and economical and environmental reasons, only three passed to Phase 2.

Phase 2: Comparison of low sulphur PSMA preliminary cost evaluations helped to identify the best PSMA. Option 1 was identified as the best on-land option.

Phase 3: Preliminary design of high and low sulphur PSMA, and water pond capacity evaluation. Option Bamoos Lake East was also considered and rejected because it did not hold all process solids under water.

Phase 4: Preliminary evaluation of combined high and low sulphur process solids under water in Bamoos Lake. Option 1 was evaluated with deposition from opposite sides, creating two different effluents and resulting in Options 1A and 1B.

These three options (Bamoos Lake, Options 1A and 1B) were then analyzed in detail.

18.4.4 Description of Potential Sites

18.4.4.1 Option 1A

Option 1A holds all process solids from the present mining plan and final elevation of process solids would be at 382 m (see Figure 18.15). Option 1A allows deposition of low sulphur process solids from west to east, creating a primary sedimentation pond in the south-east corner of the management area. The impervious dam at this end (#6) will include separate operational and emergency spillways. The tallest impervious dam in the high sulphur PSMA would be 43 m high, due to the narrowness of the valley. For the first few years, water will be pumped into a 1,350 m long 1.5-m diameter pipe culvert where it will be transferred by gravity from the primary sedimentation pond to the process water pond. After a few years of operation, pumping will no longer be required and water will pass through the operational spillway of dam #6.

The process water pond, located to the south of the mine, will be able to contain the spring melt with two impervious dams (#PA1 and #PA2). These dams (40 and 15 m in height) will be impervious and built in two phases. The mine access road will have to be built beside this water pond and could use the water retaining dykes as a passage way to access the mine site.

The excess water from the process water pond will be transferred by pumping to the high sulphur PSMA. A treatment plant could be included in the pumping station to help settlement of fine particles.

Settlement of mine water will take place in the open pits. It will then be pumped and, if necessary, treated in Canoo Lake. That watershed will be diverted southward through a ditch to the high sulphur PSMA. In order to do so, a dyke will be needed on the eastern side of the lake (#CE1).

High sulphur process solids will be deposited underwater in a pond to the west of the process water pond. Water from the high sulphur PSMA is assumed of acceptable quality for release in the environment, since underwater deposition reduces suspended matter in water. Consequently, the watershed of the high sulphur PSMA can be left to flow freely through the spillway without treatment. Only one final outlet is created and discharge in the environment is through the operational/emergency spillway of the high sulphur PSMA into Stream 6. Stream 6 can easily be diverted into Hare Lake if required. A final control method, such as silt curtain or closing gate, can be integrated in the spillway to augment retention time in the pond. Alternatively, AMEC considers the land downstream from the final outlet ideal for a polishing pond and, if required, a small treatment plant could be built. For this evaluation, a silt curtain price has been evaluated.

18.4.4.2 Option 1B

Option 1B is a modified version of Option 1A, to let out water to the environment directly to Hare Lake through Hare Creek. This option also holds all process solids from the present mining plan, and final elevation of process solids would be at 382 m.

Option 1B requires deposition from east to west, creating a primary sedimentation pond to the west of the management area with an 89-m impervious dam at final elevation. Water is then be transferred by gravity from the primary sedimentation pond to the process water pond directly downstream through an operation/emergency spillway excavated in rock.

The process water pond will contain the spring melt with one impervious 65-m dam which would be at final elevation at year -1. The excess water from the process water pond will be transferred by gravity through a control structure to a polishing pond if required. The polishing pond could be created downstream of the process water pond before final outlet to the environment. A treatment plant could be included between the two ponds to help settlement of fine particles.

Location and assumptions for the high sulphur PSMA are the same as in Option 1A. However, no excess process water is added to the pond. As in Option 1A, mine water is managed in the open pit and Canoo Lake, and then transferred to the high sulphur PSMA.

Option 1B has two final outlets to the environment: Creek 6 for the high sulphur PSMA and Hare Lake for the low sulphur PSMA.

18.4.4.3 Option Bamoos Lake

The Bamoos Lake option was initially looked at because it is the best available natural containment area in the region. It is also a preferred option for combined high and low sulphur process solids because most of the process solids can be placed under the natural water elevation. It is situated at the top of the watershed and natural water flow to the receiving waters (Bamoos Creek and Hare Lake) would not be affected because the excess water would be left to flow freely through the spillway since the water is assumed free of suspended solids since deposition is done under water. Estimated water level at closure would be about 6 m higher than the present water elevation. Construction costs would be very low and water management fairly simple. A polishing pond is evaluated in this option and provision is made for a treatment plant that might be required depending on how well the fine particles settle. Mine water will also be pumped to Bamoos Lake, so no other management areas are required.

18.4.5 Critical Water Assessment

Water assessment was the key item in the AMEC study, to confirm feasibility of the project and to identify the required infrastructure. The water assessment was analyzed for five distinct project needs at the feasibility stage:

1. Spring melt: Retention volume required to ensure water quality released in the environment in spring according to critical rain and snow melt (6.1 Mm^3).

2. Mill water requirements during winter: Water reserves needed in the fall in order to ensure water availability during winter (4.5 Mm^3).

3. Excess process water to release to the environment (0.55 m^3/s for 5 summer months and 1.7 Mm^3 during the winter).

4. Start-up water assessment: Water availability for operation in the beginning of mine operation during the proposed staged construction.

5. Water for high sulphur process solids: Water volumes required to flood the high sulphur process solids at all times.

No water assessment are presented for Option Bamoos Lake since all excess water is assumed to flow freely to the environment and because the available natural water volume in the lake assures easy recirculation of water back to the mill.

The water assessment of Option 1A is presented below. Option 1B is similar to 1A (same high sulphur PSMA and same low sulphur PSMA, and water transfer between different water ponds is all done by gravity), hence no detailed analysis on water management is done for Option 1B.

Spring water balance evaluates the required water volumes to hold the critical spring melt in respect to Water Management Policies Guidelines and Provincial Water Quality Objectives of the Ministry of Environment and Energy. This shows that capacity of 6.1 Mm^3 needs to be available during the spring melt. Note that water transferred to the environment during this month is not taken into account because no treatment plant is presently incorporated in the pumping station. Therefore, if the water quality is not adequate for release, it will have to be held.

The fall water balance allows the evaluation of the available water volume in fall required to feed the mill with reclaim water during the winter. This shows that approximately 4.5 Mm^3 of water needs to be available in fall.

Every year, excess water from precipitation will need to be pumped from the process water pond to the environment. In the case of the critical spring melt filling the process water pond to the maximum, a pumping capacity of 0.55 m^3/s is required to release the summer rain and the gradual melting of accumulated ice in the management area. The rest of the excess water between spring and fall can be released during winter.

Assessment of the start up period confirms the feasibility of water management during the first years of operation while the recirculation process still requires pumping from low sulphur PSMA to process water pond. The assumption is that start-up infrastructure construction is designed for one year of deposition, fall water balance requires a minimum of 4.5 Mm^3 of available water and the process water pond holds a maximum of 6.1 Mm^3 of water. Between these three assumptions, water can be transferred between the two water ponds and water can be released in the environment. Analysis shows that water management is achievable in wet and dry years with the given assumptions and construction schedules.

Assessment of the high sulphur PSMA start-up water has also been considered. Year-1 requires capacity for one year of deposition under water. The combination of pulp water and one year of rain is sufficient to submerge the high sulphur process solids. During the following years, water from the mine, the Canoo Lake watershed and, in Option 1A, the excess water from process water pond, will be added to the management area.

18.4.6 Typical Sections of Dykes and Dams

Preliminary stability analysis was done on typical sections of five different dam designs to confirm feasibility. A typical dam cross-section is shown in Figure 18.16. A combination of these dams will optimize construction costs. They are:

1. Impervious geomembrane, 15-m high dams.

2. Impervious asphalt center core dams built in one phase.

3. Impervious asphalt center core dams built in many phases with process solids upstream.

4. Dams raised with process solids.

5. Impervious clay center core dams built in one phase.

18.4.7 Fill Plans

Many fill plans were then drawn for the three options to detail the predicted evolution of the PSMA. These fill plans, along with the start up water assessment, were used to prepare the construction and capital expenditure schedule over the life of the mine.

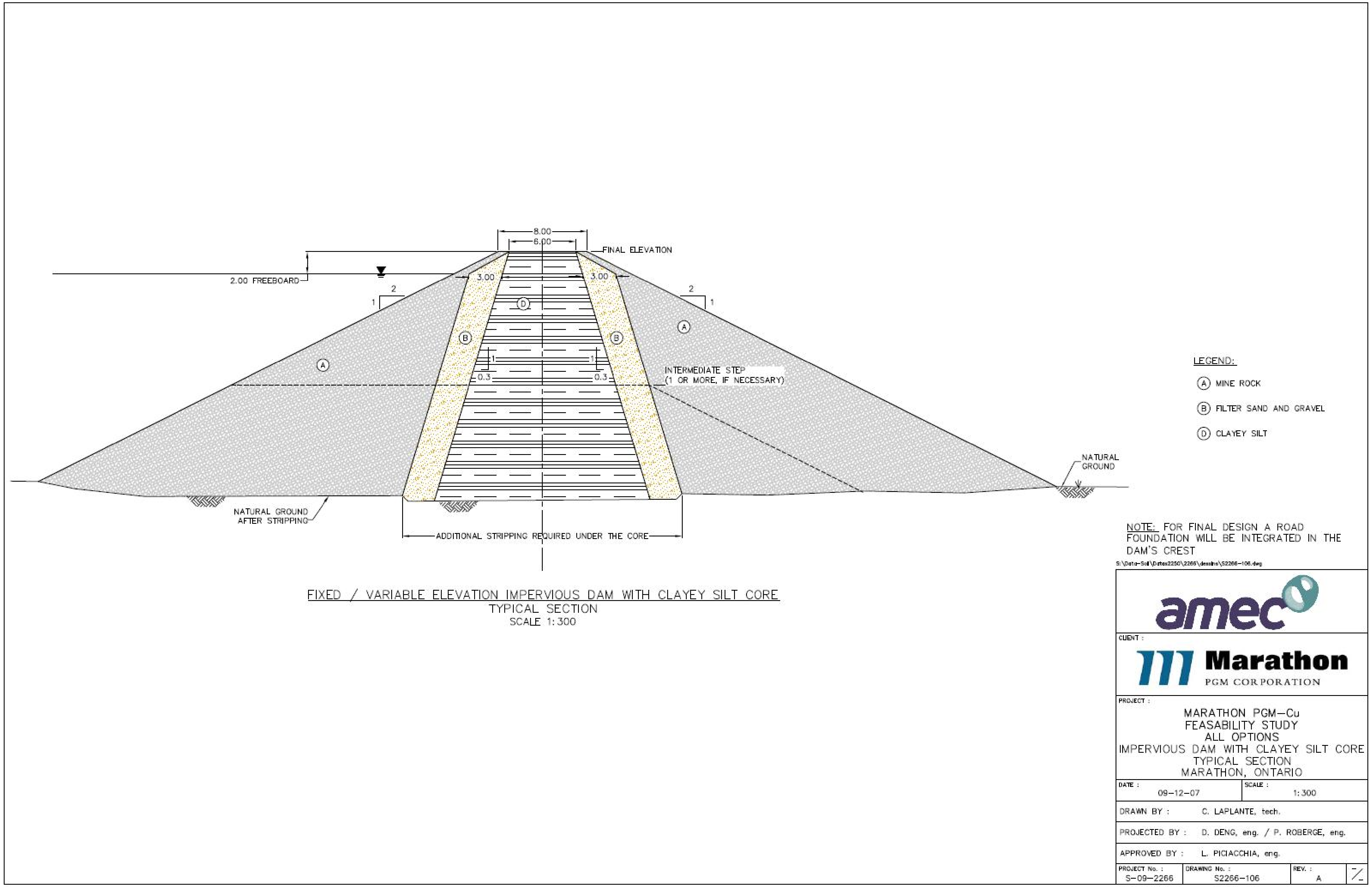
This study assumes the following materials for construction:

- The impervious material used for dam core construction in AMEC's evaluation is clay. No campaign was undertaken by Marathon PGM to qualify the available volumes for this material. Some areas have shown presence of the material but available quantities have not been estimated. A substitute for the clay would be an asphalt cement core. The unit cost for this material is high, although quantities are reduced.

- Sand and gravel used for filter and drain materials will be available in two aggregate pits near the existing access road.

- Dykes and dams will be mainly built of rock fill from the mine rock.

Figure 18.16
Typical Dam Cross-section



- As mentioned previously, the process solids required for dam raises in the PSMA require that a cyclone create a stockpile of dewatered process solids available for dam raises.

18.4.8 Site Water Management

18.4.8.1 Site Water Balance

A site-wide water balance model has been developed for the Marathon PGM-Cu project site by EcoMetrix using data provided by AMEC.

The model was used to estimate the annual flow between the elements on the property during the mine life and at the end of the operations. The model can be used to demonstrate the variability of the flows for varying precipitation and various stages of the life of mine.

18.4.8.2 Water Quality

Direct runoff from the land based process solids alternatives is expected to be neutral for the low sulphide material and acidic for high sulphide material. Water from the base case process solids sub aqueous disposal area will not need treatment beyond settling/polishing (i.e., there are no data that indicate that metal removal and pH control will be necessary).

It is likely that the runoff from the mine rock may reflect small zones where acidic conditions occur. Concentrations of some base metals are also expected to be elevated.

Mine water (pit infiltration, runoff) is likely to be in the order of 3.5 Mm^3/y. It is thought that during the initial stages of mine development mine water will be of sufficient quality that no treatment (metal removal and/or pH control) will be necessary. It is possible that as mine development advances mine water quality may diminish (increased metal levels, low pH) to the point that treatment (metal removal and pH control) will be necessary.

Loading sources, such as runoff from the open pit, runoff from the plant site and process solids water will also will also influence the overall site water quality. It is assumed therefore that excess water that will be discharged to the surrounding environment will require treatment or settling to reduce the total metal concentrations.

18.4.8.3 Water Treatment

The potential water treatment options and associated capital expenditures at the Marathon PGM were estimated by EcoMetrix based on discussions with AMEC and Micon.

The current estimate is that about 24 Mm^3 of water annually (all sources) will have to be managed on site during mine operations. Of this, approximately 3.5 Mm^3 of mine water will be generated (and subsequently treated) on an annual basis the treatment plant design will need to accommodate a flow of between about 7 and 14 m^3/min where water is treated and discharged over a 6-month or a 12-month period, respectively. The updated Feasibility Study assumes that it will be necessary to eliminate mine water on an accelerated schedule (6 months) and it will not be practical to discharge year round.

For the purposes of attempting to estimate a cost for a mine water treatment plant it was assumed that:

1. The system would need to provide for metal removal and pH control (e.g., a high density sludge (HDS) plant as a conservative assumption).

2. The system will have a design flow of 14 m^3/min.

18.5 ENVIRONMENTAL PERMITTING AND MANAGEMENT

N.A.R. Environmental Consultants Inc. (NAR) was retained by Marathon PGM to conduct environmental baseline studies in 2005 and 2006 (N.A.R. Environmental Consultants Inc., 2007). In 2007, Golder was retained by Marathon PGM to undertake a gap analysis of all of the environmental data collected and then to prepare baseline studies for 2007 and 2008 (Golder Associates Ltd. 2009a). In 2009 Marathon employed EcoMetrix and True Grit to provide the environmental research relevant through 2009 and into 2010, and beyond. The overarching objective of this research is to provide the necessary information to develop an EIA and ultimately deliver the EIS for the Marathon PGM-Cu project to the government. The detailed results from these field studies will form part of the EIS.

The project description, prepared by EcoMetrix and True Grit (EcoMetrix and True Grit, 2009), has been submitted by Marathon PGM to the relevant government agencies.

18.5.1 Environmental Baseline Study

The following sections summarize the field and desk-based research that has been completed (or is in midst of being completed), with the aim of providing an understanding of the scope, methodology, and objective of each research activity.

18.5.1.1 Meteorology

The Marathon area is (or has been) serviced by a number of meteorological stations whose data are relevant to the project site). Data, either historical or current, are available from a total of eight stations within a 50 km radius. The most proximate station to the project site is at Marathon Airport which is located about 4 km west. Data available include wind speed and direction, air temperature and relative humidity, solar radiation, rain or snow water equivalent (SWE) precipitation (depending on the season) and barometric pressure.

18.5.1.2 Air Quality

The primary objective of the air quality baseline study is to collect baseline ambient air quality data representative of the Project area in terms of concentrations of particulate matter (PM - less than 10 µm in diameter [PM_{10}] and less than 25 µm in diameter [PM_{25}]), inorganic parameters (in particular metals) and greenhouse gases.

MPGM will set up an appropriate, local air quality monitoring network prior to the commencement of mine construction, which will be active over the life of mine operations. The design and nature of the monitoring network will be commensurate with the results of the air dispersion modeling that will be completed as part of the formal assessment of anticipated air emissions from the project.

18.5.1.3 Noise

For the purposes of the EA process, noise is evaluated not only within the context of worker health and safety but also within the context of potential exposure of local biota and communities to noise generated by project-related activities. The overall objective of the noise baseline study is to collect background noise data that can be used to evaluate the potential effects and cumulative effects of sensitive receptors to noise emissions associated with the Project. Baseline noise monitoring was completed at the site in 2009 under varying conditions to characterize existing noise levels. Baseline noise measurements were also taken at this site in June and September, 2006.

18.5.1.4 Geology

Understanding the geology of the project site is of fundamental importance both from the mine feasibility point of view and the mine design perspective. The baseline geology data collection program has included both desk-top and in-field activities. This has included a review of existing material, such as that summarized in maps and publications provided by Ontario Geological Survey and the Ontario Ministry of Northern Development Mines and Forests, as well as the data gleaned from the comprehensive drilling completed over the last few years as described in Section 11.0 of this Technical Report.

18.5.1.5 Terrain and Soils

The objectives of the review of the existing or baseline terrain conditions on site are to identify environmental constraints and limitations as it pertains to all facets of mine design and subsequent development. For the purposes of the baseline environmental program, features of the terrain within the project area have been assessed with the aid of available topographic maps and recent aerial photographs.

The soil investigation program addresses both soils directly affected within the footprint, as well as soils within the anticipated airshed. The information collected as part of the baseline soils study will also be relevant when assessing the effects of the project, developing soil handling plans for soil salvage, storage, and reclamation and final closure planning.

18.5.1.6 Ecosystem Mapping and Vegetation

An ecosystem mapping and vegetation baseline study is being undertaken for the project area that focuses on ecosystem mapping and inventory of the proposed development property.

The objectives of the baseline studies are to:

1. Characterize the terrestrial ecology of the project area and adjacent areas to facilitate an assessment of the potential effects of development on vegetation and ecosystems within the Project area.

2. Provide the necessary information for the development of wildlife habitat suitability maps.

3. Identify the ecological communities and plants in the project area that are afforded status either at the local, provincial or federal levels.

Initial ecosystem mapping focused on acquiring and compiling existing map data and aerial and lidar imagery for the study area. These data were supplemented by information collected via a number of systematically conducted aerial (helicopter) surveys of the project site and surrounding area.

Vegetation features were assessed on the ground in 2007, 2008 and 2009. Unique or sensitive ecosystems, locally, provincially or federal important, and potentially at-risk plants identified in the field were described and photographed and voucher specimens were collected. Collected vegetation information included an overall assessment of the cover (as a percent) of shrubs, herbs, and grasses, as well as a list of predominant species (with a corresponding percent cover). Secondly, incidental observations of vegetation in the area (i.e., visual encounter survey) were made (and noted) coincident with other baseline sampling activities.

Observations recorded during the terrestrial field program in 2007, 2008 and 2009 indicate that the habitat observed within the project site is similar to that described by Environment Canada for the Abitibi Plains Ecoregion. Mixed forest habitat on the site was dominated by white birch, white spruce, black spruce, and balsam fir, with occasional barren bedrock knobs. Four ecosites and eight vegetation communities were identified within the project site.

A total of 291 vascular plant species have been observed in the study area during baseline studies. Most species identified are typical boreal forest plants that are common throughout northwestern Ontario.

18.5.1.7 Wildlife

The wildlife baseline program is designed to identify terrestrial wildlife and critical wildlife habitat near the project area. Identification of wildlife species (including mammals, birds, amphibians and invertebrates) and critical wildlife habitats is an important initial step required to fill information gaps, and to develop options for mitigation to meet the obligations of federal and provincial standards for species protection.

The wildlife program consists of two main parts, comprising the compilation of existing information on wildlife species and wildlife habitat information from outside sources and fieldwork. Extensive baseline field work was completed in 2007, 2008 and 2009 and covers all seasons. A summary of field activities that comprise the wildlife survey components is provided in Table 18.8.

Table 18.8
Summary of Terrestrial Baseline Field Sampling Undertaken in 2007, 2008 and 2009

Date	Nature of Surveys	Comments
August 2007	Vegetation, wildlife	Visual encounter surveys across project site.
September 2007	Vegetation, Wildlife	Visual encounter surveys across project site.
April 2008	Aerial (wildlife [ungulates], Vegetation [assemblage level]), raptor stick nest	Aerial survey across project site and adjacent areas.
May 2008	Raptor call-back	42 plots were sampled across project site at locations representing different vegetation types (ecosite phase and wetlands type).
June 2008	Breeding birds, vegetation	A total of thirty-six breeding bird survey plots were established on and around the proposed project footprint. The plots were located in habitat types that were representative of the project area and included coniferous forests of various seral stages, marshes, disturbed areas, and forest edges accordign to the Canadian Wildlife Service methodology (2008). Vegetation was surveyed by visual encounter.
March 2009	Aerial (wildlife [ungulates], vegetation [assemblage level])	Aerial survey across project site and adjacent areas.
June 2009	Aerial (wildlife [ungulates], vegetation [assemblage level]), birds, species at risk habitat	Aerial survey across project site and adjacent areas. Bird monitoring was conducted at 33 stations following the Forest Bird Monitoring Program (FBMP) protocol. Special attention was paid to areas that could represent habitat for species at risk.
July 2009	Vegetation, species at risk	Vegetation was surveyed by visual encounter. Targeted species at risk surveying in previosuly identified candidate areas.
August (4 to 7) 2009	Vegetation, species at risk	Vegetation was surveyed by visual encounter. Targeted species at risk surveying in previosuly identified candidate areas.

Date	Nature of Surveys	Comments
August (24 to 25) 2009	Vegetation, species at risk	Vegetation was surveyed by visual encounter. Targeted species at risk surveying in previosuly identified candidate areas.

Observations of wildlife (or wildlife sign, e.g., tracks, scat) around the project site have included a range of mammal, bird, amphibian and invertebrate species. A total of 15 mammal species, 64 bird species, seven species of amphibians, 12 species of butterflies and 19 species of dragonflies and damselflies (odonates) have been observed on the project site during baseline studies.

18.5.1.8 Species of Special Interest

The most up to date versions of provincial and federal data bases listing species of special interest were consulted and cross-referenced with the geographic location of the project site to develop a candidate list of species of special interest to the proposed project. The species were also characterized as to the likelihood that they would be found on or near the project site given that nature of the available habitat on site. Targeted field surveys for species of special interest in candidate habitat locations were undertaken as described above.

Based on the review of species range information, there is potential for six federally listed species and six provincially listed species to occur in the region in which the project site is located. Table 18.9 provides the species listed by either the federal or provincial governments and characterizes the potential for these species to occur within the project area.

Table 18.9
Species at Risk

Species Specifics		Listed By		Potential
Scientific Name	Common Name	Federal	Provincial	
Falco peregrinus	Peregrine falcon	Special Concern	Threatened	Low - Cliffs with "marginal" peregrine falcon habitat have been identified on site but no evidence of nesting.
Haliaeetus leucocephalus	Bald eagle	Not at risk	Special Concern	Low - Bald eagles are known to occur within the Nipigon MNR District but general not in the vicinity of the Project site.
Chordeiles minor	Common Nighthawk	Threatened	Special Concern	Moderate - Although not observed during baseline surveys they are known to occur within the area.
Wilsonia canadensis	Canada Warbler	Threatened	Special Concern	Observed in 2009
Contopus cooperi	Olive-sided Flycatcher	Threatened	Special Concern	Observed in 2009
Euphagus carolinus	Rusty Blackbird	Special Concern	Not at risk	Observed in 2009
Rangifer tarandus	Woodland caribou	Threatened	Threatened	Moderate - Woodland caribou are known to occur within the Nipigon

Species Specifics		Listed By		Potential
Scientific Name	Common Name	Federal	Provincial	
				MNR District (NHIC 2005), and specifically along the Lake Superior coast and islands near the Project area.

Two fish species have ranges that overlap with the project area, in that they are found in the Pic River. The Northern Brook Lamprey (*Icthyomyzon fossor*) (special concern nationally; special concern in Ontario) is known to occur in the lower reaches of the Pic River downstream of the Project site. Lake Sturgeon (*Acipenser fulvescens*) (threatened nationally; special concern in Ontario) is also known to occur in the Pic River.

Natural Resource Value data provided by MNR indicate that there are no known sites of occurrence of, and/or high value habitat for species of flora listed as threatened or endangered, and no known sites of occurrence of flora identified as species of special concern within the Big Pic Forest Management Area.

18.5.1.9 Hydrology

The surface hydrology assessment encompasses all aspects of surface water quantity within the project area. Information on the surface hydrology of the project area is required to evaluate potential hydrological changes related to the development and operation of the proposed mine. The objectives of the hydrology baseline study are to:

1. Measure stream flows and lake water levels throughout the project area.

2. Develop estimates of annual runoff, seasonal distribution of runoff, and return period extreme (high and low) flows of the project area.

3. Develop a water balance for the site.

Flow data are available from the Water Survey of Canada (WSC) for the Pic River for a period of 36 years from a station location approximately 1 km east of the property. Also, an extensive stream flow monitoring network, comprising 41 individual monitoring stations, was established within the project area in 2007. Flow data at these stations has been collected monthly through the open water season.

Surface water on the project site drains either to the Pic River (to the east) or to Lake Superior (to the west). The Pic River forms the eastern boundary of the Marathon property. It is a large, low- to medium-gradient river, approximately 20 to 30 m wide as it traverses the property. Water in the Pic River is highly turbid and carries a significant suspended sediment load.

There are a total of 14 water courses that drain the project area to the Pic River. A number of these (eight) have relatively small watersheds, tend to be high-gradient and have flow that is dependent on rainfall and snow melt.

18.5.1.10 Hydrogeology

The overall objective of the hydrogeological work program was to develop an accurate representation (model) of ground water flow (deep and shallow) for the project area so as to be able to: establish water quality and water level baseline data for all aquifers, and to develop monitoring and sampling schemes for all aquifers; identify and evaluate potential sources of water supply for mining operations; and identify potential sites for mine infrastructure.

Hydrogeological and hydrogeochemical data have been collected on a monthly basis through 2008 and 2009, generally between May and November as conditions permit. Refinement of the preliminary, current understanding of ground water flow on site is ongoing as more data become available.

Based on a preliminary assessment of the groundwater well data collected on site to date it appears that groundwater flow mirrors site topography and generally follows surface water flow paths. That is, the primary groundwater divide runs in a north-south direction more or less along the long axis of the primary mine rock storage area. Water to the east of the divide drains to the Pic River, whereas water to the west of the divide drains towards Lake Superior.

18.5.1.11 Aquatic Resources

The characteristics of waterbodies in the project area form part of the aquatic resources baseline study program. Included are physical and chemical characteristics of water and sediment, and biological characteristics such as benthic invertebrates, fish and fish habitat. The objectives of the aquatic life baseline program are to:

1. Quantify the water quality of key waterbodies in the project area, including spatial and seasonal variability (spring high-flow period, early summer high-flow period, the late summer low-flow period, and under-ice in winter).

2. Quantify the sediment quality of key waterbodies in the project area.

3. Quantify the species composition, abundance, spatial distribution and biological characteristics of the fish and benthic invertebrate communities of the local aquatic ecosystem.

4. Identify the characteristics of fish habitat in the project area.

An extensive network of water quality monitoring stations has been established that includes headwater and downstream areas of all of the watersheds (and sub-watersheds) that traverse the Project area. In total the network includes 58 stations in total comprising 13 lake stations, four Pic River stations and 41 stream stations. Water quality sampling on the project site began in 2001 but the sampling program as it is currently constituted began in the spring of 2008, and is ongoing. The range of parameters for which analyses (or measurements) are undertaken on these samples is shown in Table 18.10.

Table 18.10
Description of the Analyses Completed on Routine Surface Water Samples

Parameter Category	Analytes
Physical tests	Colour, Conductivity, Hardness (as $CaCO_3$), pH, TSS, TDS, Turbidity, DO, Temperature
Anions and nutrients	Alkalinity, Total (as $CaCO_3$), Ammonia-N, Total Bicarbonate, Carbonate, Chloride, Fluoride, Hydroxide, Nitrate-N, Nitrite-N, TKN, Phosphorus (total), Sulphate
Carbon	DOC
Metals	Total Metals (full ICP-MS scan), Dissolved Metals (full ICP-MS scan), Mercury, Hexavalent Chromium
Aggregate organics	BOD, Tannin and Lignins
Radionuclides	Radium-226

Surface waters on the project site can be characterized as being relatively weak tea-stained (from humic acids) in appearance, as having low to moderate hardness (< 25 mg/L as $CaCO_3$) and as being neutral to slightly basic in nature (pH 7 to 8, though some bog areas have pH in the 5.5 to 6.0 range). Waters draining to the project site to the east to the Pic River tend to have higher turbidity and total suspended solids levels than waters draining to the west to Lake Superior. This seems likely attributable to the nature of the overburden over which the streams flow.

Fish and fish habitat characterization has been conducted within the project area and water bodies into which on-site water courses drain (e.g., Pic River, Lake Superior) in 2006 (NAR, 2007), 2007 (Golder, 2008) and 2009 (EcoMetrix, in progress). Effort has been expended within each of the water bodies (lakes, streams) within the project footprint and has been completed on a seasonal basis (where appropriate) to reflect potential differences in habitat utilization relating to high and low flow conditions, as well as seasonal differences in fish activity (e.g., spawning). On-site data collected as part of field collections between 2006 and 2009 have been supplemented by records (limited) available from local Ministry of Natural Resources (MNR) offices (Terrace Bay, Manitouwadge).

Benthic invertebrates have been collected from project area (and environs) lakes and streams in 2006 (NAR, 2007), 2007 (Golder, 2008) and 2009 (EcoMetrix, in progress) so that benthic data are available for the entire study. Coincident with benthic invertebrate sampling (in depositional environments) sediment samples have been collected for analysis of of metals, grain size and nutrients (carbon, nitrogren and phosphorus).

18.5.1.12 Socioeconomic Issues

The objective of the socio-economic/cultural and infrastructure baseline study is to provide key information on the social, economic, cultural, community and health environments that could be potentially affected by the proposed development. The socio-economic/cultural baseline study will focus on the following types of data: local and regional demographics; social indicators, health indicators, cultural indicators; economic indicators; community indicators; and services and infrastructure. Up to date information will be collected through literature searches of existing social, economic, health and cultural studies produced by academics, government agencies, and other organizations.

The primary focus of the baseline study based on proximity is the Town of Marathon, located only a few kilometers from the Project site. First Nations communities of interest based on historical land use and land claims include the Ojibways of the Pic River First Nation (PRFN, the group most proximate to the project site), the Pic Mobert First Nation (PMFN), the Bingwi Neyaashi Anishinaabek First Nation (BNA), the Pays Plat First Nation (PPFN), the Long Lake No. 58 First Nation (LL58), and the Biinjitaawabik Zaaging Anishinaabek First Nation (BZA).

18.5.1.13 Land Use

Information on local and regional land and resource use, for public and First Nations, will be developed as a means of capturing existing conditions and anticipating potential effects on land users, any land use plans, tenure holders and stakeholders.

18.5.1.14 Traditional Knowledge

Traditional knowledge, where available and appropriate, is used as a source of knowledge to inform all dimensions of an EA process and project design. This data is used to develop baseline conditions; identify Valued Ecosystem Components (VEC); understand “values” associated with a study area; identify traditional use of areas that may be lost or affected; identify potential environmental and social effects and their significance; and inform the development of mitigation measures. Discussions with local stakeholders regarding traditional knowledge, in terms of its availability and its potential use in the EA process, have occurred and are ongoing.

Based on information available from First Nations to date, traditional use of the project area was, and continues to be, largely limited to the water corridors defined by the Pic River and the Bamoos Lake-Hare Lake-Lake Superior system.

Marathon PGM continues to consult with First Nations in vicinity of the project area, and in particular the PRFN, to develop a means by which more specific traditional knowledge relevant to the Project area could be utilized as part of the EA process.

18.5.1.15 Archeology

Projects that have the potential to disturb archaeological sites require an archaeological assessment prior to the project being developed. In Ontario, archaeological assessments are completed in a staged fashion, as required by provincial government regulations. Stage I assessments have been carried out in 2007. Two separate Stage II archaeological assessments of the Marathon PGM-Cu project area have been completed according to established guidelines and sampling guidance. The first, which was conducted in 2007, focused on the Pic River corridor and the interior of the project site. The second Stage II study was completed in 2009 and was focused on the Bamoos Lake-Hare Lake-Lake Superior and Angler Creek-Lake Superior corridors.

18.5.2 Environmental Assessment and Permitting

18.5.2.1 Federal Environmental Assessment Process

The Canadian Environmental Assessment Act (CEA Act) is the legislative basis for the federal environmental assessment process. The CEA Act came into force in 1995 under the direction of the Minister of Environment. It applies to the federal government and federal lands where there are specific federal decisions and approvals required to permit a proposed project to move forward. Specifically, the CEA Act applies to a proposal when all four of the following criteria are met:

1. The proposal meets the definition of “project” under the Act.

2. The project is not excluded from having to undergo an EA.

3. The project will necessitate an action or decision of a federal authority.

4. The specified federal action or decision “triggers” an obligation to ensure that an EA is conducted.

Based on the above criteria it has been established that the CEA Act does apply to the Marathon PGM-Cu project. There are four different types of assessments that may be used for the review of a proposed project. A track decision regarding which one of these assessment types will be made by the Minister of Environment (federal) following submission and review of the Project Description Report prepared by AMEC on behalf of Marathon PGM.

18.5.2.2 Provincial Environmental Assessment Requirements

Along with being subject to the federal environmental assessment process, the Project will also be subject to the requirements of the Ontario Environmental Assessment Act (OEA Act), triggered by the requirements of the Ontario Electricity Regulation (OER) under the OEA

Act. According to the OER the Project will be subject to the Class Environmental Assessment (Class EA) for Minor Transmission Facilities.

An Environmental Assessment Screening Report that is consistent with the Class EA for provincial transportation facilities will be required, in particular as the result of likely improvements to highway 17 at the site access road intersection that will be necessary.

The project is likely to include the disposition of certain or all rights to Crown resources where activities are proposed on Crown lands. These activities will be subject to the requirements of the Ministry of Natural Resources Class EA for Resource Stewardship and Facility Development processes.

18.5.2.3 Coordination of the Environmental Assessment Process

As the proposed project is subject to both federal and provincial EA processes, Marathon PGM intends to work in a coordinated way with provincial and federal governments, both governments having formally agreed to coordinate their respective EA processes pursuant to the Canada-Ontario Agreement on EA Cooperation (November, 2004). The Canada-Ontario Agreement on EA Cooperation establishes administrative mechanisms and guides federal-provincial cooperation for the environmental assessment of projects subject to both the CEA Act and the OEA Act. Under this bilateral Agreement, projects that require a review under both federal and provincial EA legislation will undergo a single, cooperative assessment meeting the legal requirements of both governments while maintaining their respective roles and responsibilities.

18.5.2.4 Provincial and Municipal Approvals

As the project proceeds past the EA stage, various provincial and municipal approvals and permits will be required for site preparation and construction activities, as well as for the operational phase of the project the Project.

18.5.2.5 Environmental Assessment Framework and Methodology

The proposed project is subject to a federal environmental assessment (likely a Comprehensive Study) under the CEA Act and an EA is also required under equivalent provincial legislation (OEA Act). The CEA Agency will lead the coordinated EA process.

During the EA process the environmental effects of the proposed project will be assessed in a logical and practical manner. For the purposes of the assessment, environmental effects refer to potential effects or changes to components of the biophysical environment related to the implementation of the proposed project. The assessment is intended not only to be used to support the Marathon PGM-Cu project environmental assessment, but also to function, in combination with follow-up monitoring programs they may be recommended, as a planning tool to ensure continued protection of the environment.

18.6 PROJECT IMPLEMENTATION

18.6.1 Project Schedule

The project implementation schedule has been developed by Micon and Marathon PGM. The main areas of the schedule include environmental assessment and permitting, equipment procurement, construction and commissioning. The schedule has been developed using the best information available as of November, 2009.

The time-line for the environmental assessment and permitting is assumed to be two years from the date of submission of the project description.

The project schedule is presented in the Gantt chart included in Figure 18.17. This chart shows the main project activities throughout the schedule, including the main engineering, procurement and construction activities. One of the most important assumptions for this schedule is that access for construction of certain infrastructure and civil works will be granted before the EA has been formally approved.

It is assumed that construction week will be 6 days per week at 10 hours per day. No winter shut-down allowances have been included in the schedule for civil works as it has been assumed that all concrete work will be undertaken during the summer months.

18.6.2 Project Priorities and Milestones

A list of the key project development milestones is provided below:

- Complete updated Feasibility Study — November, 2009
- Project Description for EA issued — December, 2009
- EA Report issued to authorities — April, 2010
- Process optimization and basic engineering start — May, 2010
- Detailed engineering start — July, 2010
- Long lead equipment purchased — September, 2010
- Process optimization and basic engineering complete — February, 2011
- Mobilization on site — February, 2011
- Environmental assessment approved and all permits granted — December, 2011
- Detailed engineering complete — December, 2011
- Ball mill delivery to site — February, 2012
- Construction complete — January, 2013
- Wet commissioning start — January, 2013
- Production start-up completed — May, 2013

Assuming that basic construction access is granted prior to the final approval of the EA the estimated production start-up date is May, 2013. If access is only acceptable after all permits are in place and the EA has been approved, which is the scenario currently assumed in the Project Description document, then the estimated start-up date is December, 2013.

18.6.3 EPCM Approach

In order to optimize the project schedule and cost the following approach is suggested for engineering, procurement and construction management (EPCM).

- Sufficient basic and detailed engineering will be provided early to secure the procurement and delivery of critical long lead items.

- Basic engineering and advanced procurement of equipment will be started early. This would allow project optimization and lock in equipment pricing providing protection against volatile pricing market. It would also promote supplier participation/commitment in the project.

- Gaining access to vendor certified information as early as possible in the project will largely improve detailed engineering efficiency. Receiving vendor information on time has always been one of the most challenging tasks for any project of this magnitude.

- Infrastructure and site preparation engineering will be completed in advance to satisfy any pre-stripping and construction activities.

- Detailed engineering of all elements of this project, including the process plant, will be synchronized to match delivery of the long lead equipment to site and to manage construction in the most efficient manner.

- Marathon PGM will be an integral part of the EPCM team and will participate actively in the development of the project.

18.6.4 Gantt Chart

A simplified Gantt chart showing the main project activities throughout the schedule is included in Figure 18.17.

18.7 CONCENTRATE MARKETING

Marathon PGM retained Andrew Falls, Principal of Exen Consulting Services (Exen), to provide an analysis of the marketing and transportation of concentrates from the project. Mr. Falls prepared an initial report in July, 2008 that was updated in September, 2009 (Exen Consulting Services, 2008 and 2009).

Concentrate to be produced from the Marathon PGM-Cu project is considered a copper concentrate from a marketing perspective, notwithstanding the relatively high PGM content. In this respect, the concentrate is relatively unusual but the copper content, at 21-22% Cu, is low compared to the majority of copper concentrates, and will have to be blended in order to meet the requirements of almost all smelters.

Figure 18.17
Preliminary Project Schedule

micon INTERNATIONAL LIMITED | mineral industry consultants

Falls' analysis has resulted in the identification of a small number of potential buyers which are able to handle copper-PGM materials in their smelting/refining facilities and which may be anticipated to provide reasonable credit for precious metals in the Marathon PGM concentrate.

Projected output averages 83,000 t/y (dry tonnes) concentrate over a mine of about 11.5 years. Falls' analysis of the market for concentrate from Marathon PGM-Cu project was based on the grades shown in Table 18.11.

Copper is derived from both primary (mined) and secondary (recycled materials and scrap copper) sources. Approximately 10% of the world's total refined metal output is accounted for by secondary sources and a further 20% is produced by solvent extraction/electrowinning (SX-EW) which does not require smelting/refining.

Platinum group metals (principally platinum, palladium and rhodium) are also derived from both primary and secondary sources. Primary output accounts for approximately 80-85% of total output. Secondary supply is recovered from recycling of catalysts, electronic scrap and jewellery.

Treatment of copper concentrates is undertaken by operators of smelters that are:

- Fully integrated with upstream mining capacity that satisfies all feedstock requirements.

- Custom facilities which are largely independent of suppliers of concentrates and which source feedstocks from the market.

- Partially integrated facilities.

Table 18.11
Projected Concentrate Grade

Element	Unit	Grade	Element	Unit	Grade
Copper	%	21.9	Chlorine	ppm	84
Gold	g/t	6.63	Cobalt	%	0.06
Silver	g/t	127	Chromium	ppm	44
Platinum	g/t	16.7	Fluorine	%	0.025
Palladium	g/t	67.9	Potassium	ppm	650
Rhodium	g/t	0.95	Lithium	ppm	<5
Ruthenium	ppm	0.1	Magnesium oxide	%	3.64
Iridium	ppm	0.06	Manganese	ppm	350
Iron	%	29	Molybdenum	ppm	33
Sulphur	%	24.1	Sodium	%	0.29
Zinc	%	0.12	Nickel	%	0.52
Lead	%	0.061	Phosphorus	ppm	<200
Arsenic	%	0.004	Selenium	%	0.008
Antimony	%	<0.001	Silica	%	6
Bismuth	%	<0.002	Tin	ppm	<20
Mercury	ppm	<0.03	Strontium	ppm	110

Element	Unit	Grade	Element	Unit	Grade
Aluminum oxide	%	2.8	Titanium	ppm	650
Barium	ppm	60	Tellurium	ppm	<30
Beryllium	ppm	<0.2	Vanadium	ppm	40
Calcium oxide	%	1.9	Yttrium	ppm	1.9
Cadmium	ppm	10	Water	%	7-10

Output from the Marathon PGM-Cu project will be destined for treatment by partially integrated or fully custom smelters.

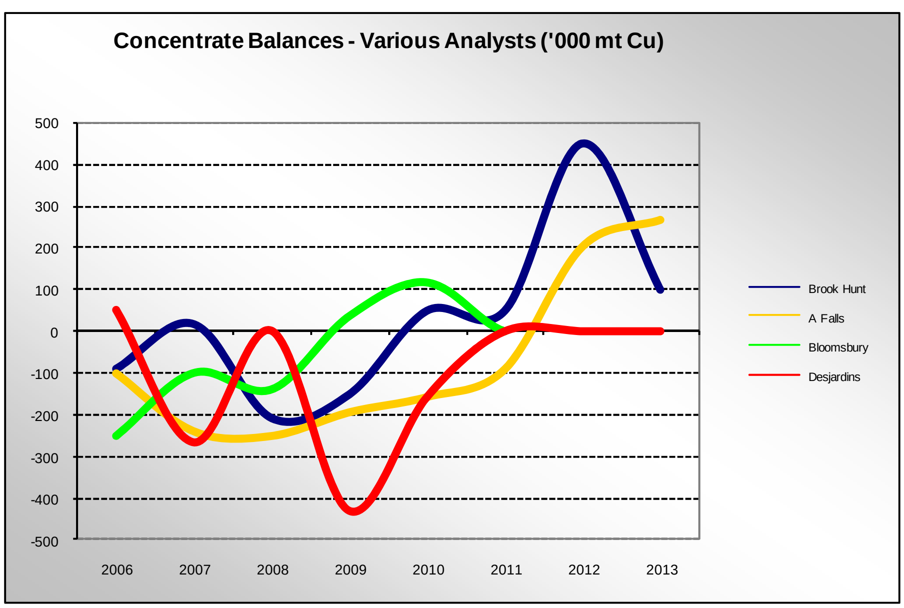
It is estimated that the custom copper concentrate market in 2009 was approximately 23 Mt of concentrate, equivalent to around 45-50% of global smelting capacity.

18.7.1 Copper Concentrate Market

The market for copper concentrate functions somewhat independently of the market for refined copper and, in general, has been less influenced by factors such as currency fluctuations and investment fund activity.

Over the period since 2000, rapid expansion in smelter capacity, especially in China, has led to overcapacity and a relatively tight market for concentrates. Although mine production is expected to increase over the next few years, little will reach the traditional custom smelter markets in Western Europe and Asia. Figure 18.18 summarizes the outlook of a number of analysts, including Falls, for the copper concentrate market to 2013. Although there is significant difference in the analysts' outlook, it is generally expected that the present tightness in the concentrate market will ease as new projects come onstream.

Figure 18.18
Projected Copper Concentrate Balance



In order to meet projected demand for copper, significant new sources of supply will be required and, in order to ensure sufficient supply is brought forward, Falls and other analysts project long term copper prices in the range of US$1.75-2.00/lb.

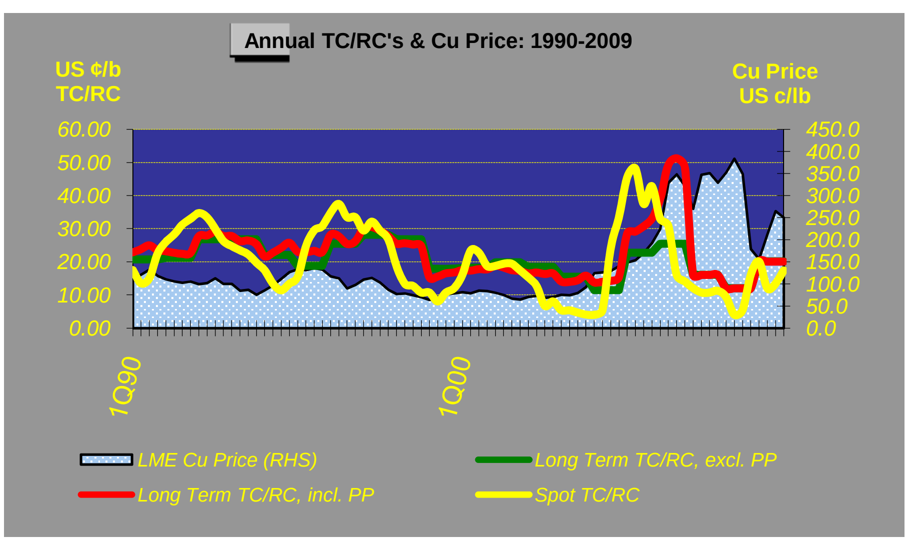
18.7.2 Outlook for Smelting and Refining Charges

The trend of treatment and refining charges (TC/RCs) and LME copper price over a period of nearly 20 years, since 1990, is shown in Figure 18.19.

The rapid expansion of smelter capacity in both China and India, coupled with strong prices for by-product metals and elimination of price participation in 2007 and 2008, resulted in TC/RCs in 2008 that were the lowest in both real and nominal terms over the past 20 years. Development of new capacity in China and India also resulted in a reduction of the influence of the custom smelters in Japan and Korea which, historically, had dominated the market. Annual benchmark TC/RCs established for the beginning of 2009 were settled at levels that did not fully reflect the tightness in the concentrate market but were influenced by the collapse in credit markets in the second half of 2008 and lack of transactions in the spot market. As this situation eased through 2009, benchmark charges fell.

As a result of his analysis, Falls concluded that, over the longer term, TC/RCs will be equivalent to around US$75-85/dmt for smelting and US$0.075-0.085/lb for refining, somewhat lower than the 20% of the copper price that has pertained since 1990. It is generally anticipated that the copper concentrate market will remain tight through 2011, or longer in the major custom concentrate areas such as Western Europe and Asia.

Figure 18.19
Treatment and Copper Refining Charges



18.7.3 Concentrate Quality

Concentrate quality is an issue to smelters/refiners for both environmental and metallurgical reasons.

In general, for a copper concentrate, the contents of copper, iron and sulphur affect heat balance and slag characteristics. Ideal specifications are:

Copper	27-33%
Iron	25-30%
Sulphur	30-35%

While concentrate grades which fall outside these specifications can be blended with other feedstocks, smelters may be less interested for that reason.

Although precious metals and/or PGMs in copper concentrates do not generally pose either metallurgical or environmental issues, not all smelters are capable of economic recovery of these metals and will not offer payment or accountability that is satisfactory to the concentrate seller.

Copper concentrates with high PGM values, therefore, have relatively limited markets. Copper smelters which process electronic and other scrap have sufficient volumes of PGMs going through their systems that sampling and recovery data are well understood. It should be noted, however, that rhodium is seldom, if ever, paid for by the smelter since input levels are generally low and the element is both difficult and expensive to analyze for.

18.7.4 Concentrate Distribution

A total of 16 smelters located in North America, Japan, Korea and Europe were contacted for their interest in the copper-PGM concentrate to be produced by Marathon PGM and feedback was received from 13 of these. As of the end of 2009, three possible purchasers had been identified in Canada, Europe and Japan.

An analysis of transportation options was undertaken as part of Falls' market study. It was concluded that concentrates would likely be shipped by rail in leased covered gondola cars from a loading facility at or near Marathon for transportation to a Canadian smelter, or offshore via Montreal. Concentrate will be trucked from the mine site to the loading facility, if this is not located at site.

Potential sites for rail loading were examined. The rail line which runs through the town of Marathon is owned and operated by Canadian Pacific Railway. Canadian Pacific owns lines to Montreal (including the port) and Vancouver. Interchange agreements with CN Rail and Ontario Northland would allow delivery to Quebec City and the port of Vancouver, for offshore shipment.

18.7.5 Terms and Costs

Based on the responses from the smelters contacted and on the transportation analysis, it was concluded that potentially viable alternatives include a smelter in Canada, and/or shipment via Montreal to Europe or, possibly, to Japan.

Falls provided concentrate treatment terms and costs for the purpose of the Feasibility Study and analysis of project economics. These remain confidential to Marathon PGM.

18.8 CAPITAL EXPENDITURES

The estimated pre-production project capital costs are summarized in Table 18.12.

Table 18.12
Summary of Estimated Pre-production Project Capital Costs

Area	Cost ($ thousand)
Mining pre strip	5,762
Mine equipment[1]	18,536
Process plant and infrastructure	261,695
PSMF and water treatment	8,396
Owner's costs	7,202
Contingency	49,531
Pre-production total	**351,122**

[1] Assumes a 10% down payment on mining equipment and financing of the balance over 5 years at 9%/y interest rate.

The life-of-mine capital cost estimate is $495 million comprising $351 million of pre-production capital and $144 million of sustaining and closure capital. The sustaining capital consists of mainly $103 million for mining, which includes a credit for mine equipment salvage.

18.8.1 Mining Capital Costs

A breakdown of the estimated life-of-mine project capital costs is presented in Table 18.13.

Table 18.13
Summary of Estimated Life of Mine Mining Capital Costs
($ thousand)

Cost Area	Total	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6
Mine equipment - full value	120,846	55,935	63,090	797	157	107	90	40
Deposits paid on delivery	12,089	5,594	6,309	80	16	11	9	4
Capital payments	108,556	8,412	18,656	20,455	22,320	24,345	13,607	240
Mine equipment (not depreciated)	**120,846**	**14,005**	**24,965**	**20,535**	**22,336**	**24,356**	**13,616**	**244**
Pre-production (capitalized interest)	4,531	4,531						

Cost Area	Total	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6
Pre-production (capitalized Opex)	5,762	5,762						
Equipment salvage	-3,897		-	-	-	-	-	-
Total Mining Capital	**127,242**	**24,298**	**24,965**	**20,535**	**22,336**	**24,356**	**13,616**	**244**
Interest to operating costs (G&A)	26,479		8,884	7,269	5,441	3,441	1,257	36

Cost Area	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13
Mine equipment - full value	105	105	105	105	105	105	
Deposits paid on delivery	11	11	11	11	11	11	
Capital payments	93	80	79	81	94	94	
Mine equipment (not depreciated)	**103**	**91**	**89**	**91**	**105**	**105**	**205**
Pre-production (capitalized interest)							
Pre-production (capitalized Opex)							
Equipment salvage	-	-	-	-	-	(3,897)	-
Total Mining Capital	**103**	**91**	**89**	**91**	**105**	**(3,792)**	**205**
Interest to operating costs (G&A)	23	23	24	26	27	27	-

The estimated mine preproduction capital expenditure totals $24.3 million. This amount covers:

- Initial site clearing and overburden removal by contractor(s).
- Initial mine rock stripping and ore stockpiling by the mine department.
- The initial procurement of the pit mobile and ancillary equipment. For the financial evaluation, it is assumed that mining capital equipment is leased based on a 10% down payment and financing at 9%/y interest of the balance over 5 years.
- Following the pre-production period, the cost of financing the mining equipment is included in the G&A operating costs.

The unit equipment costs include equipment base cost, applicable tires, transportation, assembly, commissioning and training.

The total life of mine non-discounted cost of mining capital is $120,846,000.

18.8.2 Process Plant and Infrastructure Capital Costs

The process and infrastructure capital cost estimate, which was developed by Met-Chem, includes the work required to construct the ore processing facilities and establish all the infrastructure and services necessary to support the mine site. The accuracy of the estimate is deemed to be ±15% except for the rail siding area, which is considered to have an accuracy of only ±25%. Two alternatives were costed, the base case assumes sub-aqueous disposal of the process solids and the other case considers storage of the process solids on land.

18.8.2.1 Basecase Capital Cost Estimate

The total base case process and infrastructure capital cost estimate, including contingencies, is \$305.6 ± \$45.8 million (between \$259.8 million and \$351.4 million) of which \$220.6 million are direct costs and \$85.0 million are indirect costs.

The base date for the cost estimate is the second quarter 2009. The estimate is expressed in Canadian dollars. No allowances for escalation or currency fluctuation are included.

The exchange rate used is US\$1.00/\$1.10 when quotations were received in US dollars.
The exchange rate used is €1.00/\$1.58 when quotations were received in Euros.

Labour rate used in the estimate is based on all-inclusive hourly rates quoted by a local contractor for the tradesman, foreman and superintendent. The contractor's mobilization and demobilization as well as workers traveling were established separately. The calculation method and figures were then confirmed with a qualified local contractor for such a project.

The working calendar was defined as 10 hours a day, 6 days a week. Productivity was estimated at 0.95 when factoring site conditions, climate and labour skills.

Quantities for civil work, structural steel, steel deck, stairs, handrails and grating were calculated from layouts. Unit costs, including material, freight, equipment and installation work, were quoted by qualified local contractors.

Only new process equipment included is included in the design. Quotes were obtained for 98% of the equipment in 2008 and 75% in 2009. For most of the major equipment such as the ball mills, crushers, flotation cells and conveyors, at least three quotations were received. Equipment freight was either quoted by the suppliers or estimated as 7.5% of the equipment value.

Equipment installation hours were either quoted by the suppliers or estimated from Met Chem's in-house database for similar projects.

Quantities for large bore process and water piping were calculated by material take-offs from flowsheets and layouts. The cost of service piping and small bore lines was estimated by applying a factor to the total equipment direct cost.

The total base case estimated direct capital cost is \$221 million, which includes \$82 million for equipment, \$78 million for materials and contracts, \$50 million for installation and \$11 million for freight. These costs cover only the processing facilities and site infrastructure. They exclude the mining, PSMF and the MRSAs. A summary of the process and infrastructure direct capital cost estimate is provided in Table 18.14.

Table 18.14
Processing and Infrastructure Direct Capital Cost Estimate by Area

Area	Estimate ($ thousand)
Site preparation and roads	6,909
Power supply	10,255
Metallurgical facilities	158,046
Process solids and water management	5,125
Ancillary buildings	14,643
Auxiliaries and services	9,515
Service equipment and surface vehicles	2,311
Rail siding (concentrate handling)	4,927
Construction camp	8,841
Total direct costs	**220,571**

A summary of the estimated indirect capital costs for the Marathon PGM-Cu project ore processing and site infrastructure is presented in Table 18.15.

Table 18.15
Processing and Infrastructure Indirect Capital Cost Estimate

Indirect Costs	Estimate ($ thousand)	% of Direct Cost
EPCM [1]	17,341	7.9
Total capital spares [2]	6,265	7.7
First fills [2]	2,040	2.5
Construction indirects [1]	9,117	4.1
Wet and dry commissioning [2]	1,846	2.0
Training manuals [1]	100	0.0
Owner's project team [1]	2,000	0.9
Project insurances [1]	2,206	1.0
Contingency [1]	44,114	20.0
Total indirect costs [1]	**85,029**	**38.5**

[1] Percentage of total direct costs.
[2] Percentage of delivered equipment costs.

The indirect capital cost estimate covers only the processing facilities and site infrastructure. It excludes the indirect costs associated with mining, PSMF and the mine rock storage facilities.

EPCM costs were estimated as a specific percentage of each item in the direct cost table and represent 7.7% of the total direct costs. Miscellaneous equipment spares, commissioning, insurance and construction indirect costs were estimated individually as a percentage of the total direct cost. Certain major equipment capital spares were specifically quoted by the supplier.

Construction indirects include site power, temporary construction facilities, road maintenance, site security, health and safety, site management personnel and workers transportation.

The contingency estimate was calculated as a specific percentage of each item included in the direct cost table. The contingency represents 20.0% of total direct costs.

18.8.2.2 Alternative Case Capital Cost Estimate

The total process and infrastructure capital cost estimate for the alternative process solids disposal option, including contingencies, is $309.4 of which $223.3 million are direct costs and $86.1 million are indirect costs. Compared to the base case this option includes an additional flotation circuit to recover sulphides for separate disposal from the process solids (tailings).

18.8.3 Process Solids (Tailings) and Water Management Costs

Construction costs were developed by AMEC in order to compare the feasibility of three process solids (tailings) management options. The unit costs used to estimate the total cost were based on numbers developed for similar materials at other process solids management facilities.

Of the three options studied by AMEC, the sub-aquatic option (Bamoos Lake) was selected as the base case by Marathon PGM. The other two alternatives were land based deposition systems termed Option 1A and 1B (see Section 18.4 for a description of each alternative). Option 1A was the preferred land deposition option.

The results of AMEC study showed that the base case (Option Bamoos Lake) requires the least investment ($2.1 million) due to the fact that minimal infrastructure is required to create the management area, and has the lowest overall average cost of $0.10/m^3 of process solids. By comparison, Option 1A results in an average cost of $0.67/m^3 of process solids and an optimized pre-operational investment of $18 million.

The following table presents the capital cost estimates for the PSMF, the base case (Bamoos Lake Option) and Option 1A.

Table 18.16
PSMF Direct Capital Cost Comparison

	Unit	Option	
		1A	Bamoos Lake
Initial investment - Year -1	$ million	17.80	2.10
Long term investment - 11.4 years	$ million	40.70	6.20
Unit cost (process solids)	$/m^3	0.67	0.10
Restoration costs	$ million	4.10	0.20

The PSMF capital cost estimates also include an allowance for geotechnical design (3% of direct costs) and construction indirects (10% of pre-production and 5% of sustaining direct costs).

The cost of a water treatment plant was estimated at $6 million. This assumes that approximately 3.5 Mm^3 of excess mine water will be generated (and subsequently treated) on an annual basis and therefore the treatment plant will need to accommodate a flow of about 14 m^3/min where water is treated and discharged over a 6 month period. The selection of the type of plant assumes that the system will need to provide for metal removal and pH control. A high density sludge type facility was the conservative basis for the cost estimate. The cost of the water treatment facility was estimated by EcoMetrix and reviewed/accepted by Micon.

18.8.4 Owner's Costs

The estimated Owner's pre-production capital costs total $7.2 million. A summary of these costs is shown in Table 18.17.

Table 18.17
Owner's Pre-production Capital Cost Estimate

Area	Estimate ($ thousand)
Permitting and environmental assessment	5,000
Technical studies	1,000
Communications, IT, software etc.	100
Marathon PGM office costs	50
Reclamation bond [1]	1,052
Total	**7,202**

[1] $10M bond, discounted at 3%/y, with logarithmic growth of liability from 15% to 100% over LOM. This number is for the pre-production period only.

18.8.5 Project Sustaining Capital

The estimated life-of-mine sustaining capital costs are presented in Table 18.18.

Table 18.18
Life-of-Mine Sustaining Capital Costs

Area	$ thousand
Mine equipment (not depreciated)	106,841
Mine salvage	(3,897)
Plant and infrastructure	18,349
Plant salvage	(10,683)
Process solids and water management	4,417
Reclamation Bond	(3,625)
Reclamation costs (process solids)	196
Reclamation costs (other)	6,090
Closure	15,290
Contingency	10,545
Total	**143,523**

18.9 OPERATING COSTS

The total average life-of-mine unit operating costs are presented in Table 18.19.

Table 18.19
Estimated LOM Unit Operating Cost

Component Cost	$/t milled
Mining	5.67
Processing	6.79
Water treatment	0.05
General and administration - site	0.58
General and administration – mine equipment financing	0.29
Total on-site cost	**13.39**
Concentrate transportation, smelting and refining	3.25
Total operating cost	**16.64**

18.9.1 Mining Operating Costs

The open pit operating costs were developed from first principles based on the pit plan and production schedule haul road layouts, the MRSAs and primary crusher locations; projected equipment performances; information for similar operations; suppliers input; and quotes for consumables. The diesel fuel price was assumed to be $0.673/L based on a supplier quote.

The annual estimated mine operating costs are summarized in Table 18.20. The average life-of-mine estimated unit mine operating cost is $1.477/t mined or $5.734/t processed.

The mining costs include operating and maintenance labour, supervision, fuel and lubricants, maintenance and repair parts and consumables including tires and tracks as applicable, and indirect costs. The operating costs cover pit drilling, blasting, excavating and loading and haulage operations.

Table 18.20
Mine Operating Cost

Timeframe	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Totals
To stockpile (kt)	500	0	0	0	0	0	0	0	0	0	0	0	0	500
From stockpile (kt)	0	500	0	0	0	0	0	0	0	0	0	0	0	500
Ore tonnes (kt)	0	5,728	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	4,918	90,946
Total mill feed (kt)	0	6,228	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	4,918	91,446
Mine rock tonnes (kt)	2,399	28,930	31,424	28,000	28,000	28,000	28,000	18,000	18,000	18,000	18,000	12,056	4,663	263,472
Total tonnes (kt)	2,899	35,158	39,454	36,030	36,030	36,030	36,030	26,030	26,030	26,030	26,030	20,086	9,581	355,418
Strip ratio	---	4.65	3.91	3.49	3.49	3.49	3.49	2.24	2.24	2.24	2.24	1.50	0.95	2.88
Drill & blast $/t	0.541	0.487	0.488	0.487	0.487	0.487	0.487	0.520	0.520	0.520	0.520	0.530	0.596	0.502
Load & haul $/t	0.990	0.783	0.688	0.695	0.698	0.711	0.746	0.778	0.810	0.860	0.749	0.919	0.938	0.762
Support $/t	0.320	0.133	0.124	0.136	0.136	0.136	0.136	0.171	0.171	0.171	0.171	0.222	0.387	0.158
Overheads $/t	0.137	0.046	0.040	0.044	0.044	0.044	0.044	0.061	0.061	0.061	0.061	0.079	0.165	0.055
Total mining cost $/t	1.988	1.449	1.340	1.362	1.365	1.378	1.413	1.530	1.562	1.612	1.501	1.749	2.086	1.477
Total manpower	99	160	163	155	155	155	160	138	138	143	133	126	94	

[1] For the financial evaluation Year -1 costs are included as pre-production capital.

18.9.2 Process Operating Costs

The total average on-site annual process operating cost for the 22,000 t/d base case operation is estimated at $55 million per year or $6.79/t of ore processed. The areas included in the process operating cost estimate are crushing, grinding, flotation, concentrate dewatering, air services and process solids pumping. Table 18.21 presents a summary of the estimate divided into the six major components which are labour, electrical power, consumables and reagents, maintenance supplies, heating costs and concentrate handling.

These estimated process operating costs are based on the process design criteria, supplier information and quotes, Met-Chem's database and factors from similar operations.

Table 18.21
Processing Operating Cost Summary

Operating Cost Area	Cost ($ thousand/y)	Cost ($/t)
Labour	5,084	0.63
Electrical power	13,435	1.67
Process consumables	30,925	3.85
Maintenance supplies	2,964	0.37
Heating costs	1,764	0.22
Concentrate handling	378	0.05
Total operating costs	**54,551**	**6.79**

In the process plant, it is estimated that there will be 58 employees. This includes the supervisory staff, and hourly production and maintenance personnel for the crushing, process plant and process solids thickener areas. The annual cost per employee is based on current labour rates in the Marathon area, 5% shift premium for hourly works and an employment burden of 40%.

The average electrical power cost has been estimated by Met-Chem at $0.082/kWh and the estimated total installed process electrical load is 30.1 MW.

Process consumables make-up 57% of the total estimated process operating costs. The total estimated cost for the process consumables is $30.9 million per year or $3.85/t of ore processed. These processing costs can be broken down into three major components, namely liners and wear parts ($0.34/t), grinding media ($1.37/t) and reagents ($2.14/t).

The estimated cost of maintenance supplies is based on a 3% factors applied to the direct capital equipment cost estimate. The estimated annual maintenance supplies cost is $3.0 million or $0.37/t of ore processed.

Heating cost has been estimated by Met-Chem using the calculated number of building air changes per hour and a propane unit cost of $0.632/L.

The cost of transporting the concentrate from the mine site to trail siding, storage, handling and loading onto a train is estimated at $378,414, or $3.25/t of concentrate, or $0.05/t of ore processed.

18.9.2.1 Alternative Process Solids Management Case

The estimated processing costs for the alternative process solids disposal system discussed in Section 18.4 of this Technical Report is $7.08/t milled. Compared to the base case, this estimate includes additional power and additional reagents required for the flotation of sulphide material from the process solids stream.

18.9.3 General and Administration Operating Costs

General and administration (G&A) department includes general management, accounting, environmental, security, safety, human resources, training, First Nations and community relations, infrastructure maintenance, and information technology functions.

The G&A personnel structure consists of the general management, accounting, cost control, purchasing, human resources, safety, community relations, environmental, security, information technology, administration and medical departments. The estimated number of G&A personnel required amounts to 25 with a total annual cost of $2,446,500. The G&A labour cost includes a 10% overtime allowance for hourly paid labour and a 40% burden allowance for all personnel.

The G&A expense component is estimated at $2,213,000/y and comprises items relating to the following:

- Office supplies.
- Consultants.
- Travel, meetings and conferences located off site.
- Training.
- First aid.
- Safety equipment and supplies.
- Insurance.
- Legal expenses.
- Warehouse expenses.
- Licenses and permits.
- Access road maintenance.
- Information technology and communications.
- Community and labour relations.
- Assay laboratory supplies.
- Waste management

The total estimated annual G&A operating costs amount to $4,659,500 or $0.58/t of ore processed.

In addition to the site G&A costs, the project cost includes the cost of financing the mining equipment. The Feasibility Study assumes that mining equipment is procured with a 10% down payment and financing the balance over 5 years at an interest of 9%/y. The average life-of-mine cost financing cost is $26.5 million or $0.29/t processed.

18.9.4 Off-Site Costs

The concentrate transportation, smelting and refining costs are discussed in the marketing Section of this Technical Report. Other off-site costs included concentrate shipment insurance at 0.06% of the net invoice value and an allowance for concentrate losses at 0.6%. The average life-of-mine off-site costs amount to $319/t of dry concentrate produced or $3.25/t of ore processed.

18.10 FINANCIAL ANALYSIS

The overall level of accuracy of the Feasibility Study is ±15%.

18.10.1 Basis of Valuation

Micon has prepared its assessment of the Marathon PGM-Cu project on the basis of a discounted cash flow model, from which NPV, IRR, payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to evaluate the potential for establishing a viable open pit mine and concentrator to exploit the Marathon PGM-Cu deposit. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

18.10.2 Macroeconomic Assumptions

18.10.2.1 Exchange Rate and Inflation

All results are expressed in Canadian dollars ($). Cost estimates and other inputs to the cash flow model for the project have been prepared using constant, mid-2009 money terms, i.e., without provision for inflation.

18.10.2.2 Taxation Regime

Canadian federal and Ontario provincial corporate income and mining taxes have been allowed for. The computation of income tax assumes that prior-period losses of approximately $8.60 million are carried forward and are available to off-set project income, and that combined Canadian Development Expense (CDE) and Canadian Exploration Expense (CEE) allowances of $19.6 million are also utilized. Initial capital expenditure for the establishment of the mine is assumed to be eligible for accelerated depreciation. Thereafter, for income tax, ongoing capital is depreciated at an annual rate of 25% using the declining balance method, with a limit of 50% claimable in the year of acquisition. For the computation of the Ontario mining tax liability, ongoing capital is depreciated at 30% for mining assets and 15% for processing assets.

18.10.2.3 Royalty

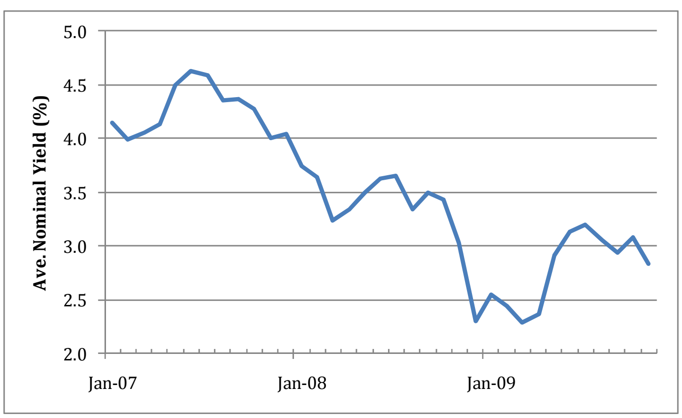
A small part of the mineral resource at the project is subject to a royalty. Between 3.0% and 4.0% of the recovered metal is affected, on which a royalty of 4.0% of the NSR must be paid. This royalty has been provided for in the cash flow model.

18.10.2.4 Weighted Average Cost of Capital

In order to find the NPV of the cash flows forecast for the project, an appropriate discount factor must be applied which represents the weighted average cost of capital (WACC) imposed on the project by the capital markets.

The cash flow projections used for the valuation have been prepared on an all-equity basis. This being the case, WACC is equal to the cost of equity. Figure 18.20 illustrates the drop in nominal 5-10 year Canadian bond yields to an average of less than 3.0% in 2009. These historically low nominal rates, in the light of an inflation rate target of 2.0%, suggest an underlying real risk-free rate of around 1.0%. Assuming the risk premium for equity to be around 5%, Micon has taken a 6.0% real rate of return as its estimate of the cost of equity for the project.

Figure 18.20
Yield on Canadian Marketable 5-10-year Bonds



Bank of Canada, www.bankofcanada.ca.

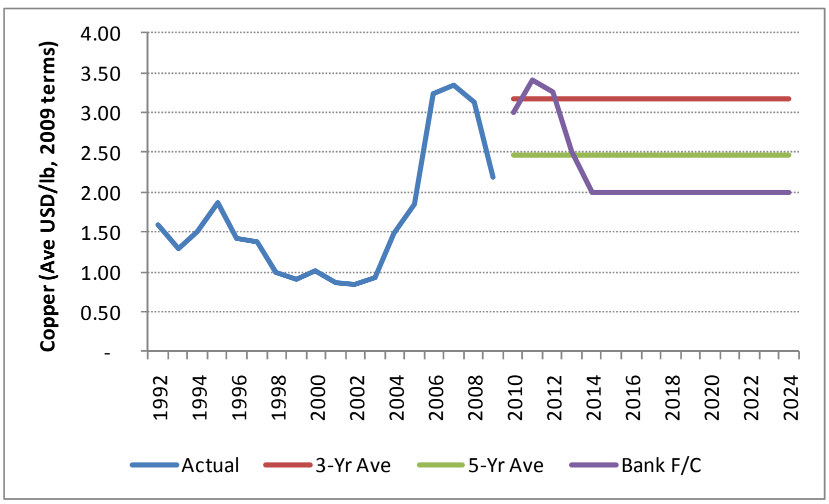
18.10.2.5 Expected Metal Prices

Owing to the volatility observed during 2008 and 2009 in the rate of exchange with the US dollar and in prices quoted in the precious and base metal markets, an unusually high level of uncertainty in measuring the economic viability of mining projects has resulted. Accordingly, Micon considered it appropriate to assess the project using a variety of price forecasts applied to the same production and cost schedule.

Three alternative price forecasts were used to prepare a discounted cash flow forecast for the purposes of project evaluation. The base case scenario considered the average spot prices, in nominal US dollar terms, for the 36 months ending October 31, 2009, i.e., the 'three-year trailing average'. The results of evaluating the project using these prices were then compared to the results obtained using a five-year trailing average and a forecast published by TD Newcrest, the research division of a leading Canadian bank [1].

Figure 18.21, Figure 18.22, Figure 18.23 Figure 18.24 and Figure 18.25 compare the three annual forecasts with actual prices since 1992 presented in real 2009 money terms. (Note: actual 2009 averages are for the 10 months ending October 31, 2009).

Figure 18.21
Copper Price
(Actual and Forecast)



[1] TD Newcrest, 'Base Metals Outlook' p7 and 'Precious Metals Outlook', p11; both dated October 26, 2009.

Figure 18.22
Platinum Price
(Actual and Forecast)

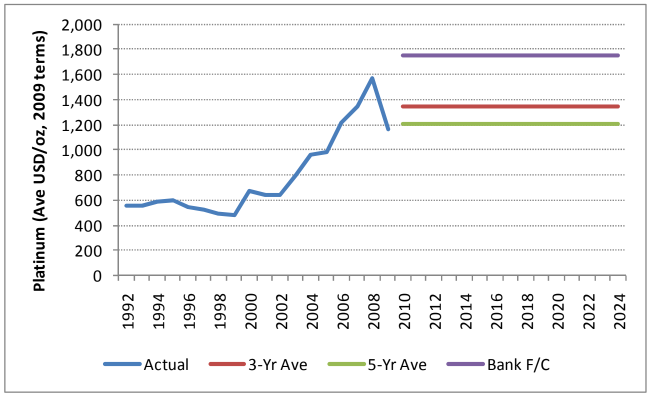


Figure 18.23
Palladium Price
(Actual and Forecast)

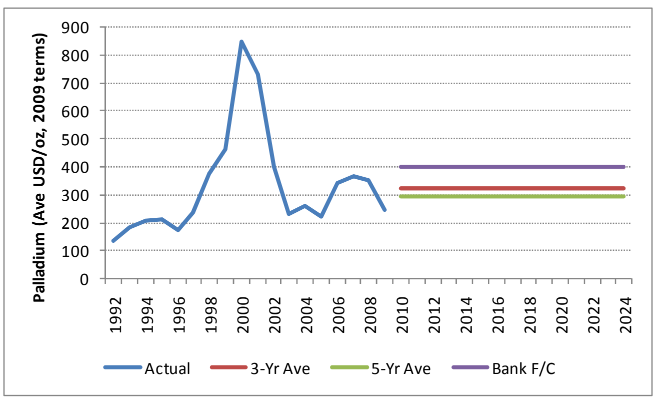


Figure 18.24
Gold Price
(Actual and Forecast)

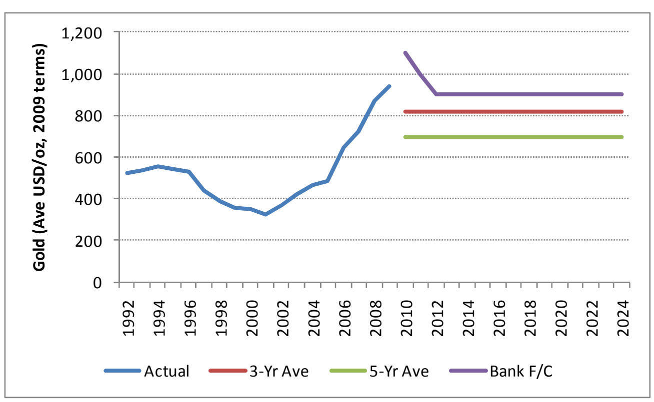


Figure 18.25
Silver Price
(Actual and Forecast)

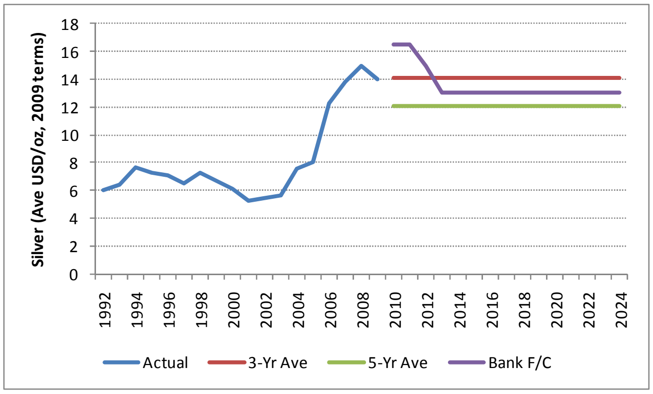


Table 18.22 presents the LOM averages for the operating period. As part of its sensitivity analysis, Micon also tested a range of prices 30% above and below these base case values.

Table 18.22
Metal Price Forecasts
(LOM Averages)

Item	Units	3-y trailing	Bank forecast	5-y trailing
Copper	US$/lb	2.91	2.03[1]	2.63
Platinum	US$/oz	1,346.65	1750.00	1205.73
Palladium	US$/oz	321.44	400.00	293.23
Gold	US$/oz	819.22	900.00	695.11
Silver	US$/oz	14.10	13.00	12.04
Exchange rate	$/US$	1.099	1.10	1.131

[1] US$2.50/lb Cu in 2013 (Yr 1), US$2.00/lb Cu long term.

18.10.3 Technical Assumptions

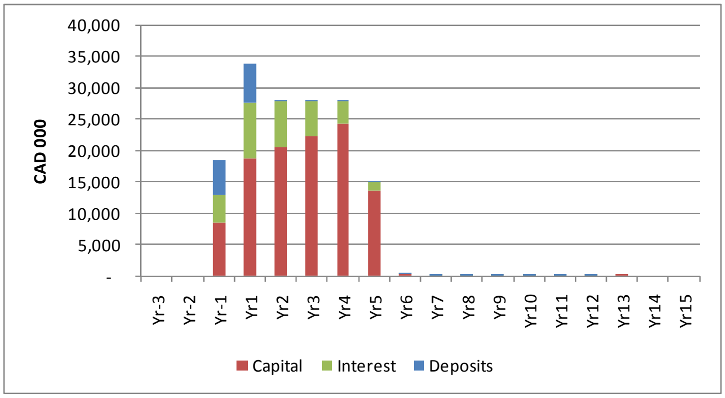
18.10.3.1 Mining Fleet Procurement

Marathon PGM's strategy in developing the project involves optimising the return on capital invested to achieve the planned level of production. To this end, Marathon PGM requested Micon to consider whether leasing the required mining equipment might offer an economic advantage over outright purchase of the fleet.

For the purposes of this study, Micon has assumed that: a deposit of 10% would secure lease-to-purchase finance over the remaining 90% of the fleet value; lease payments take into account an annual interest rate of 9.00% over a five year lease period; at the end of the lease, Marathon PGM takes ownership of the equipment, there being no residual value payable at that time.

Comparing the cash purchase and leasing options suggests that the latter offers an improved rate of return on a reduced capital investment. Leasing of the fleet has therefore been selected for inclusion in the base case. Annual payments associated with the procurement of the mining fleet through this leasing arrangement are shown in Figure 18.26, below.

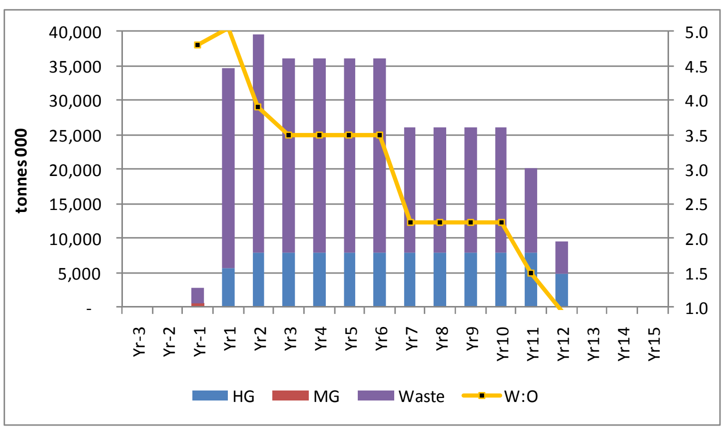
Figure 18.26
Mining Equipment Lease Payments



18.10.3.2 Mine Production Schedule

The production of ore and mine rock (waste) follows the schedule shown in Figure 18.27. The reduction of annual mine rock tonnage (and hence in the mine rock:ore ratio) after Year 2, and again in Year 7, is clearly visible.

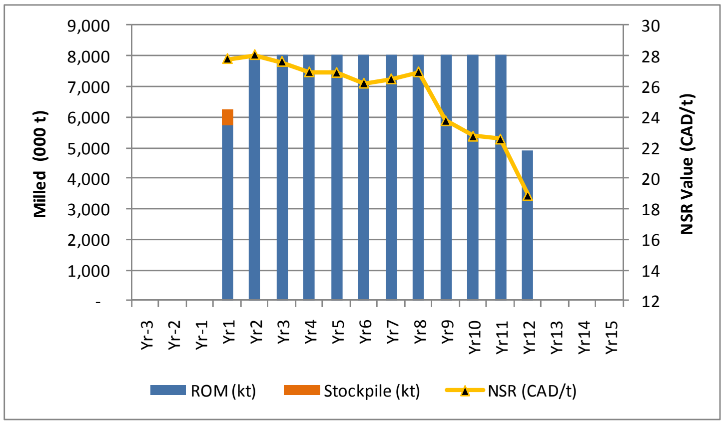
Figure 18.27
Mining Production Schedule



18.10.3.3 Processing Schedule

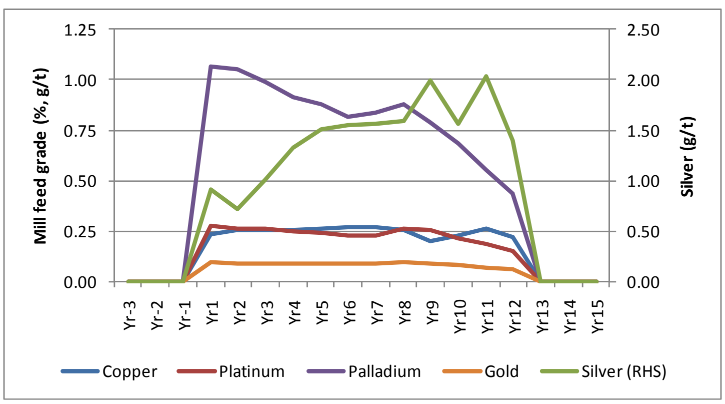
Grade and tonnage of feed to the mill is optimized through the mining schedule, which minimizes the use of stockpiles. Figure 18.28 shows the treatment of 500,000 t of material reclaimed from the stockpile during startup, to more fully utilize milling capacity in that year.

Figure 18.28
Processing Schedule



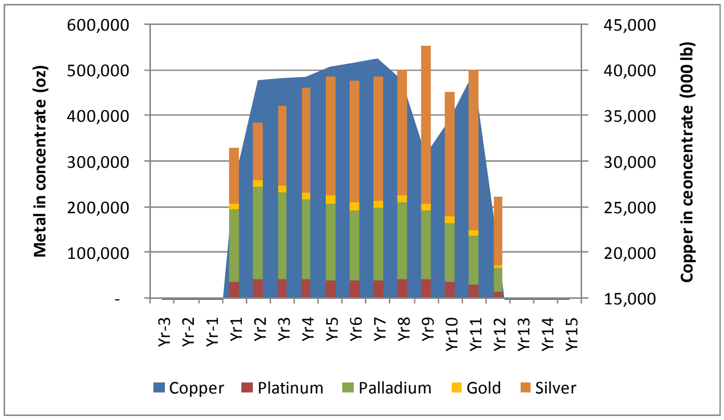
A significant decline in the NSR value per tonne of ore milled in noted from Year 9 onward. The reason for this drop in NSR value can be seen in Figure 18.29, which shows falling palladium, platinum and gold grades over this period, during which copper grades dip briefly while silver grades rise from less than 1.0 g/t to almost 2.0 g/t Ag.

Figure 18.29
Process Feed Grade



The annual amounts of metal recovered to concentrate are shown in Figure 18.30. Copper output rises steadily from Year 2 to Year 7, and despite falling back to around 30 Mlb in Year 9, average output during 10 years of steady state operations is 38.2 Mlb. Contained oz of PGM plus gold peaks at 260,000 oz in Year 2, and averages 225,000 oz over the first nine years of operation.

Figure 18.30
Metals in Concentrate



18.10.3.4 Net Smelter Return

Project revenues assume that a bulk concentrate product is sold and shipped to a smelter outside North America. Treatment and refining charges, metal payability and settlement terms are assumed on the basis of Micon's recent experience with similar concentrate products.

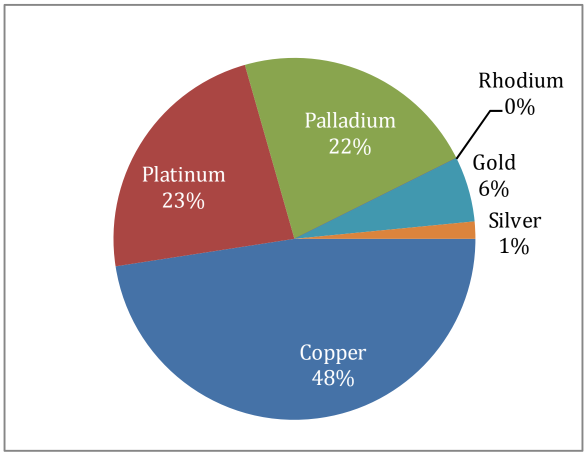
Assumed NSR terms and base case values are presented in Table 18.23, below.

Table 18.23
LOM Total Net Smelter Return

		Copper	Platinum	Palladium	Gold	Silver	Total
Average grade	% or g/t	0.25	0.24	0.83	0.09	1.44	
Recovery	%	90.80	71.00	80.10	79.90	74.50	
Concentrate grade	% or g/t	22.00	16.50	65.47	6.69	105.40	
Payability	%	95.50	88.00	92.50	90.00	90.00	
Payable metal	Mlb, oz	431,220	434,509	1,812,633	180,126	2,839,418	
Gross value	US$ 000	1,253,556	585,132	582,653	147,563	40,036	2,608,939
Price participation	US$ 000	19,556	-	-	-	-	19,556
Smelting	US$ 000	36,051	16,600	16,468	4,200	1,160	74,478
Refining	US$ 000	34,498	6,518	27,189	901	1,136	70,241
Transport	US$ 000	50,862	24,277	23,207	6,171	1,660	106,176
Net Smelter Return	US$ 000	1,112,590	537,738	515,789	136,292	36,080	2,338,488
NSR/t	US$/t	12.17	5.88	5.64	1.49	0.39	25.57

Using the base case price assumptions (i.e., 3-year trailing average), the contribution of each of the above metals to the NSR over the LOM period is shown in Figure 18.31.

Figure 18.31
Contribution of Metals to NSR



18.10.3.5 Product Shipping and Sales

Concentrate transport costs are estimated at a total of US$96.56/dmt.

For the purposes of the cash flow model, it has been assumed that the concentrate product will be sold on despatch, with product inventory and accounts receivable totalling 36 days, approximately equivalent to 10% of annual revenue.

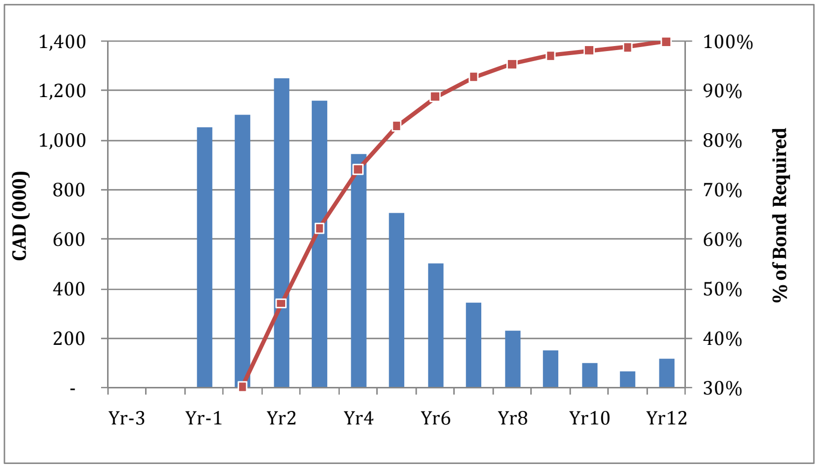
18.10.3.6 Closure Costs

For the purposes of the financial evaluation, it has been assumed that:

- Bonding will be required to the value of $10 million at the time of mine closure.
- Funds contributed to the bond will grow at an annual rate of 3% in real terms.
- At the commencement of operations, approximately 30% of the final liability will have been incurred, growing to 60% by the end of Year 3, almost 90% by the end of Year 6 and the remainder over the following 6 years of operation.

The amounts required to be contributed to the bond are then as shown in Figure 18.32. These amounts have been provided for in the cash flow projection.

Figure 18.32
Annual Bond Contributions

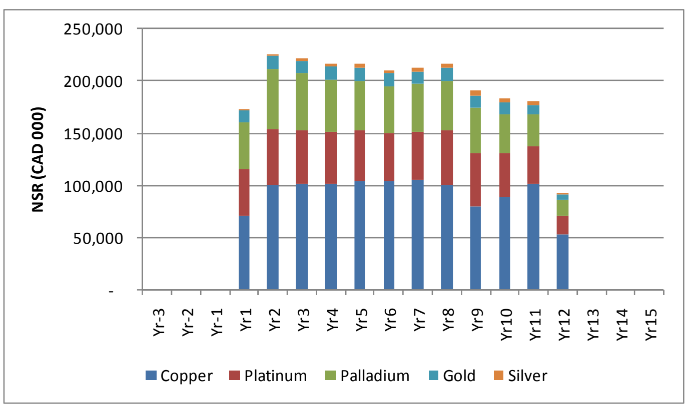


18.10.4 Summary of Base Case Economics

18.10.4.1 Sales Revenue

The net revenue generated annually from concentrate sales is shown in Figure 18.33.

Figure 18.33
Annual Net Revenue



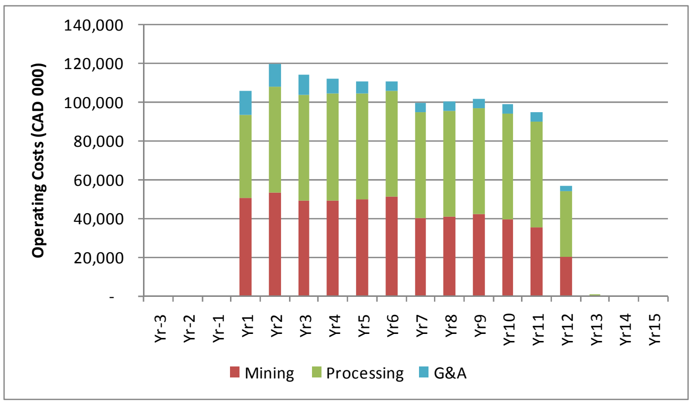
18.10.4.2 Operating Costs

Estimates life-of-mine total and unit operating costs are shown in Table 18.24. Annual cash operating costs vary in line with the volume of mine rock that must be removed from the open pit each year. The annual cash operating costs are shown in Figure 18.34.

Table 18.24
Operating Cost Summary

	Cost ($/t mined)	Cost ($/t treated)	$ thousand
Mining	1.46	5.67	518,591
Processing		6.85	625,962
G&A		0.87	79,524
Total		13.39	1,224,078

Figure 18.34
LOM Cash Operating Costs



18.10.4.3 Capital Expenditures

Mining equipment is assumed to be leased. Initial capital therefore includes only the deposits paid and the capital portion of lease payments made in Year-1. Sustaining capital costs include the capital portion of subsequent lease payments made during the production period.

At the end of the mine life, a provision of $3.9 million is made in the cash flow for equipment salvage arising from the disposal of surplus mining equipment. Process plant salvage of $10.6 million is allowed for, being 5% of the original cost, excluding contingency, EPCM, Owners' costs and construction indirect costs.

Decommissioning and reclamation costs of $21.5 million are provided for at the end of the mine life. Following completion of this work, release of the rehabilitation bond (plus interest earned) is provided for in the cash flow.

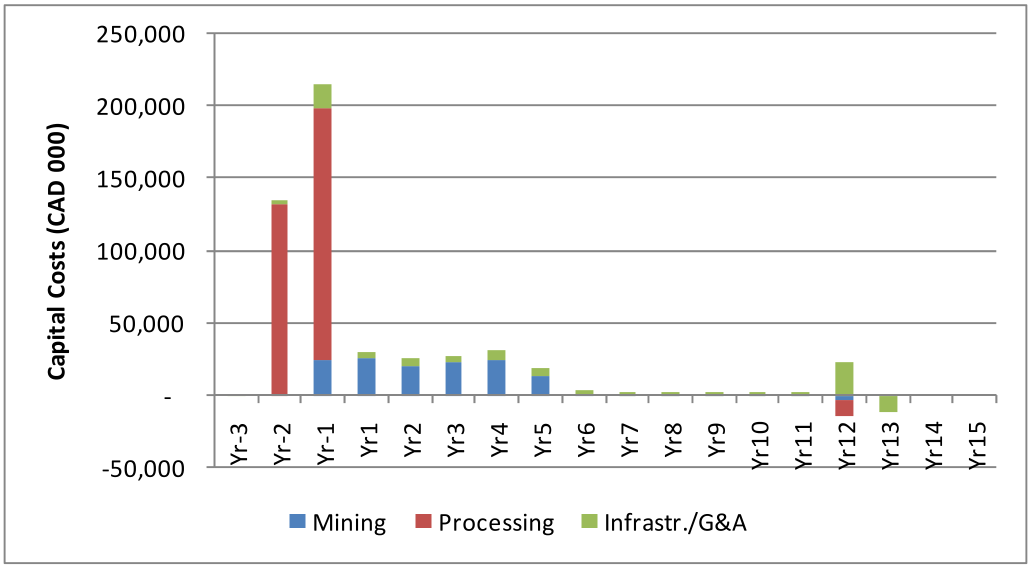
A summary of the life-of-mine capital expenditure for the project is given in Table 18.25.

Table 18.25
Capital Cost Summary
($ thousand)

	Initial	Sustaining	LOM Capital
Mining			
Pre-production (capitalized opex)	5,762	-	5,762
Pre-production (capitalized interest)	4,531	-	4,531
Mine equipment	14,005	106,841	120,846
Equipment salvage	-	(3,897)	(3,897)
Processing			
Site preparation and roads	6,909	-	6,909
Power supply	10,255	-	10,255
Process plant	158,046	-	158,046
Tailing/water management	5,125	-	5,125
Ancillary buildings	14,643	-	14,643
Auxiliaries and services	9,515	-	9,515
Service equipment and surface vehicles	2,311	-	2,311
Rail siding	4,927	-	4,927
Camp	8,841	-	8,841
Travel expenses	2,500	-	2,500
Spare parts	6,373	-	6,373
First fills	2,094	-	2,094
Construction indirects	6,617	-	6,617
Commissioning (dry/wet); manuals	1,993	-	1,993
Owner’s team	2,000	-	2,000
Insurances	2,206	-	2,206
EPCM	17,341	-	17,341
Contingency	44,114	-	44,114
Plant salvage	-	(10,683)	-10,683
Infrastructure			
Permitting	5,000	-	5,000
Technical studies	1,000	-	1,000
Water treatment plant (provision)	6,000	-	6,000
PSMF dam - construction	2,120	4,090	6,210
PSMF dam - geotechnical	64	123	186
PSMF dam - indirects	212	204	416
PSMF dam - contingency	318	613	931
Information and communications technology	100	-	100
Marathon office costs	50	-	50
Reclamation Bond	1,052	(3,625)	(2,573)
Reclamation costs (tailings)	-	196	196
Reclamation costs (other)	-	6,090	6,090
Sustaining capital provision	-	18,349	18,349
Decommissioning	-	15,290	15,290
Contingency (excl plant + tailings)	5,099	9,932	15,031
Total	**351,122**	**143,523**	**494,645**

Figure 18.35 shows the capital expenditure over the life of mine period. Note that the cash flow reflects the capital portion of equipment lease instalments in the year of payment.

Figure 18.35
LOM Capital Costs



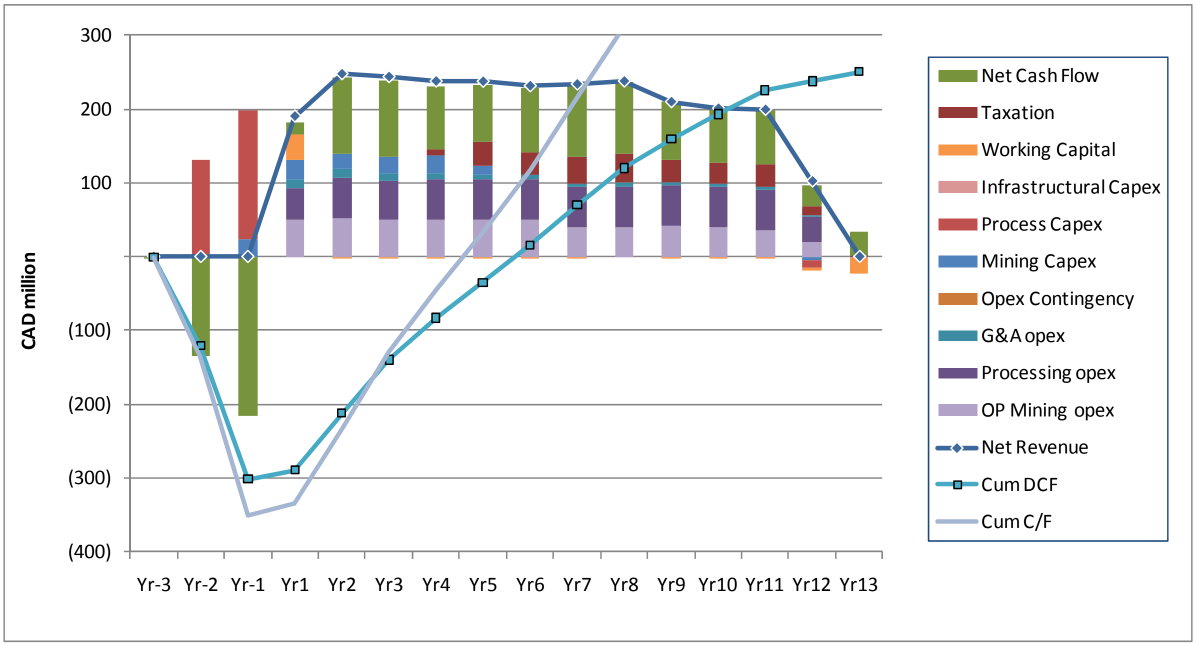
18.10.4.4 Discounted Cash Flow Evaluation

Using the assumptions described above, a cash flow and net present value (NPV) projection was prepared for the base case. This projection is summarised in Table 18.26 and Figure 18.36, below, based on a discount rate of 6%/y (NPV_6).

Table 18.26
Cash Flow Projection

	LOM Total ($ 000)	$/t treated	US$/lb Cu	NPV_6 ($ 000)
NSR copper only	1,222,847	13.37	2.58	723,170
NSR co-products	1,347,385	14.73	2.84	812,451
less Royalty	4,928	0.05	0.01	3,715
Sub-total Net Revenue	**2,565,304**	**28.05**	**5.41**	**1,531,906**
Operating costs				
Mining costs - open pit	518,591	5.67	1.09	314,610
Processing costs	625,962	6.85	1.32	368,129
General & Administrative costs	79,524	0.87	0.17	50,683
Contingency	-	-	-	-
Total cash operating cost	**1,224,078**	**13.39**	**2.58**	**733,422**
Net operating margin	1,341,226	14.67	2.83	798,484
Capital expenditure	494,645	5.41	1.04	415,104
Pre-tax cash flow	846,581	9.26	1.79	383,380
Taxation	249,768	2.73	0.53	132,663
Net cash flow after tax	**596,813**	**6.53**	**1.26**	**250,718**

Figure 18.36
LOM Cash Flow Projection



The results show that the project generates an IRR of 21.2% before tax and 17.4% after tax. The undiscounted payback period is 4.4 years, and the discounted cash flow is positive after 6 years. The NPV_6 is $250.7 million after tax.

The annual cash flow for the base case is presented in Table 18.27.

18.10.5 Sensitivity Analysis

18.10.5.1 Variation in Base Case Assumptions

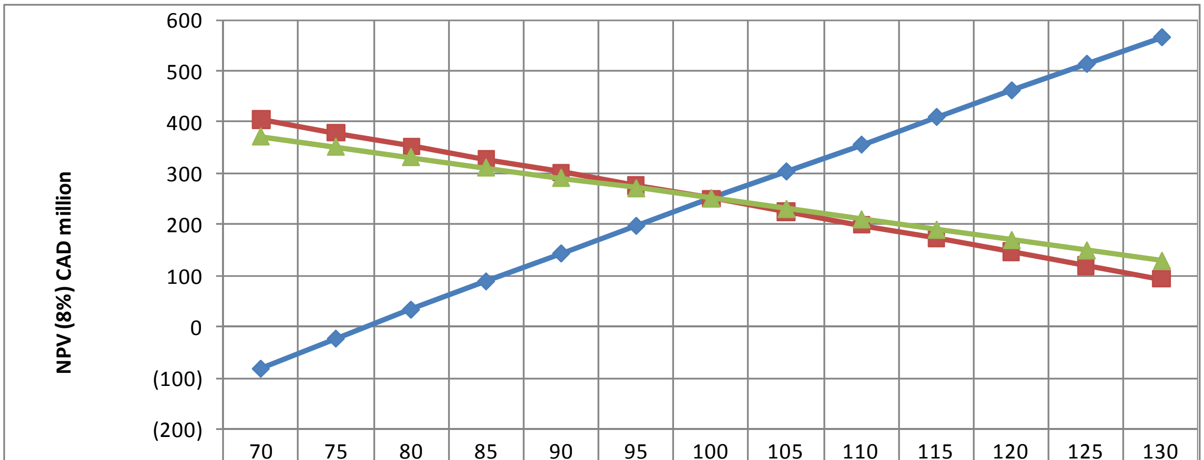
Figure 18.37 shows the sensitivity of the project cash flow NPV_6 to variation over a range of 30% (favourable and adverse) in metal price, operating costs and capital expenditure. In this context, metal prices may be used as a proxy for ore grade and recovery to concentrate.

Table 18.27
Base Case Project Annual Cash Flow

Cash Flow Forecast - Ave Prices to 2009/10/31	3-yr trailing	LOM	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15
Open Pit Mine Production		TOTAL																		
High Grade Ore		90,946	-	-	-	5,728	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	8,030	4,918	-	-	-
Medium Grade Ore		500	-	-	500	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Low Grade Ore		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL ORE (kt) mined (HG+MG+LG)		91,446	**-**	**-**	**500**	**5,728**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**4,918**	**-**	**-**	**-**
Total Waste mined (kt)		263,472	**-**	**-**	**2,399**	**28,930**	**31,424**	**28,000**	**28,000**	**28,000**	**28,000**	**18,000**	**18,000**	**18,000**	**18,000**	**12,056**	**4,663**	**-**	**-**	**-**
W/O ratio		2.88	-	-	4.798	5.051	3.913	3.487	3.487	3.487	3.487	2.242	2.242	2.242	2.242	1.501	0.948	-	-	-
Cumul. W/O ratio		-	-	-	4.798	5.030	4.401	4.072	3.917	3.827	3.768	3.543	3.375	3.246	3.143	2.991	2.881	2.881	2.881	2.881
Processing Plant Production		**91,446**	**-**	**-**	**-**	**6,228**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**4,918**	**-**	**-**	**-**
Copper	%	0.247	-	-	-	0.231	0.253	0.254	0.255	0.262	0.265	0.268	0.253	0.200	0.225	0.258	0.223	-	-	-
Platinum	g/t	0.237	-	-	-	0.279	0.264	0.260	0.249	0.242	0.229	0.230	0.258	0.255	0.212	0.185	0.152	-	-	-
Palladium	g/t	0.832	-	-	-	1.069	1.050	0.989	0.916	0.880	0.813	0.834	0.875	0.786	0.682	0.555	0.434	-	-	-
Gold	g/t	0.085	-	-	-	0.099	0.091	0.087	0.089	0.090	0.089	0.087	0.092	0.086	0.078	0.067	0.062	-	-	-
Silver	g/t	1.440	-	-	-	0.911	0.711	1.010	1.326	1.512	1.546	1.565	1.584	1.990	1.566	2.027	1.402	-	-	-
Payable Metal in Conc (imperial)																				
Copper	000 lbs	431,220	-	-	-	27,475	38,798	39,025	39,160	40,261	40,706	41,180	38,845	30,658	34,514	39,658	20,940	-	-	-
Platinum	oz	434,509	-	-	-	34,877	42,524	41,960	40,140	39,036	36,983	37,077	41,664	41,085	34,268	29,852	15,042	-	-	-
Palladium	oz	1,812,633	-	-	-	158,536	200,864	189,229	175,213	168,311	155,589	159,615	167,429	150,411	130,416	106,178	50,842	-	-	-
Gold	oz	180,126	-	-	-	14,187	16,833	16,238	16,438	16,649	16,561	16,174	17,107	15,912	14,474	12,499	7,054	-	-	-
Silver	oz	2,839,418	-	-	-	122,330	123,150	174,838	229,484	261,803	267,584	270,844	274,244	344,426	271,152	350,894	148,667	-	-	-
Copper	% in concentrate	22.00				22.000	22.000	22.000	22.000	22.000	22.000	22.000	22.000	22.000	22.000	22.000				
Moisture	% in concentrate	8.00				8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00				
Concentrate	t 000 dry basis	930.98				59.32	83.76	84.25	84.54	86.92	87.88	88.90	83.86	66.19	74.51	85.62				
	t 000 wet basis	1,011.93				64.48	91.05	91.58	91.90	94.48	95.52	96.64	91.16	71.94	80.99	93.06				
Overall Payability (metal in Conc)		89.63%				**90.51%**	**90.01%**	**89.89%**	**89.73%**	**89.55%**	**89.31%**	**89.29%**	**89.82%**	**90.54%**	**89.51%**	**88.48%**				
NET SMELTER RETURN (USD 000)		2,338,488	**-**	**-**	**-**	**173,008**	**225,142**	**221,618**	**216,170**	**215,844**	**210,324**	**212,534**	**216,218**	**190,750**	**182,674**	**181,201**	**93,003**	**-**	**-**	**-**
Copper	USD 000	1,112,590	-	-	-	71,578	100,588	101,041	101,207	103,854	104,741	105,952	100,447	79,814	88,951	101,135	53,282	-	-	-
Platinum	USD 000	537,738	-	-	-	43,535	52,828	52,059	49,711	48,254	45,604	45,716	51,625	51,249	42,324	36,487	18,346	-	-	-
Palladium	USD 000	515,789	-	-	-	45,495	57,364	53,968	49,880	47,824	44,099	45,236	47,689	43,133	37,024	29,826	14,250	-	-	-
Gold	USD 000	136,292	-	-	-	10,829	12,788	12,320	12,449	12,586	12,488	12,196	12,963	12,138	10,932	9,343	5,261	-	-	-
Silver	USD 000	36,080	-	-	-	1,570	1,573	2,231	2,922	3,328	3,393	3,434	3,494	4,418	3,443	4,410	1,864	-	-	-
Cash Flow Forecast		**LOM**	**Yr-3**	**Yr-2**	**Yr-1**	**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**Yr9**	**Yr10**	**Yr11**	**Yr12**	**Yr13**	**Yr14**	**Yr15**
		TOTAL																		
Net Smelter Return	CAD 000	2,570,232	-	-	-	190,153	247,453	243,581	237,593	237,235	231,167	233,596	237,645	209,654	200,778	199,158	102,220	-	-	-
Less Royalties	CAD 000	4,928	-	-	-	2,836	844	582	372	194	88	13	-	-	-	-	-	-	-	-
Net Revenue	**CAD 000**	**2,565,304**	**-**	**-**	**-**	**187,318**	**246,609**	**242,998**	**237,221**	**237,041**	**231,080**	**233,583**	**237,645**	**209,654**	**200,778**	**199,158**	**102,220**	**-**	**-**	**-**
Operating Costs	**CAD/t ore**	**1,224,078**	**-**	**-**	**-**	**105,496**	**119,574**	**113,943**	**112,066**	**110,531**	**110,539**	**99,465**	**100,294**	**101,599**	**98,716**	**94,775**	**56,681**	**400**	**-**	**-**
Mining	5.67	518,591	-	-	-	50,220	52,876	49,072	49,195	49,664	50,894	39,832	40,661	41,965	39,081	35,139	19,991	-	-	-
Processing	6.85	625,962	-	-	-	42,779	54,771	54,771	54,771	54,951	54,951	54,951	54,951	54,951	54,951	54,951	33,810	400	-	-
G&A	0.87	79,524	-	-	-	12,497	11,927	10,099	8,099	5,915	4,694	4,681	4,681	4,682	4,684	4,685	2,880	-	-	-
Operating Margin	14.67	1,341,226	-	-	-	81,821	127,035	129,056	125,156	126,510	120,541	134,119	137,351	108,055	102,061	104,383	45,539	(400)	-	-
Capital Costs	5.41	**494,645**	**1,430**	**134,590**	**215,102**	**30,386**	**25,677**	**27,555**	**30,852**	**18,778**	**3,843**	**2,020**	**1,883**	**1,796**	**1,740**	**1,717**	**8,380**	**(11,104)**	**-**	**-**
Engineering Studies	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining Capital	1.39	127,242	-	-	24,298	24,965	20,535	22,336	24,356	13,616	244	103	91	89	91	105	(3,792)	205	-	-
Processing Capital	3.23	295,126	-	131,493	174,316	-	-	-	-	-	-	-	-	-	-	-	(10,683)	-	-	-
Infrastructure Capital	0.79	72,277	1,430	3,097	16,487	5,420	5,142	5,220	6,497	5,162	3,599	1,917	1,792	1,707	1,649	1,612	22,855	(11,309)	-	-
Change in Working Cap	-	-	-	-	-	35,584	(1,395)	(995)	(909)	(341)	(698)	(672)	468	(2,653)	(1,112)	(484)	(3,528)	(23,264)	-	-
Pre-tax c/flow	**9.26**	**846,581**	**(1,430)**	**(134,590)**	**(215,102)**	**15,852**	**102,753**	**102,495**	**95,213**	**108,074**	**117,396**	**132,770**	**135,001**	**108,912**	**101,433**	**103,149**	**40,688**	**33,968**	**-**	**-**
Tax payable	2.73	249,768	-	-	-	-	-	-	9,182	31,960	30,653	36,673	39,128	30,533	29,113	30,203	11,005	603	408	308
C/flow after tax	**6.53**	**596,813**	**(1,430)**	**(134,590)**	**(215,102)**	**15,852**	**102,753**	**102,495**	**86,031**	**76,114**	**86,743**	**96,097**	**95,873**	**78,380**	**72,320**	**72,947**	**29,683**	**33,365**	**(408)**	**(308)**
Cumulative C/Flow			(1,430)	(136,020)	(351,122)	(335,270)	(232,517)	(130,022)	(43,991)	32,122	118,866	214,962	310,835	389,215	461,534	534,481	564,164	597,529	597,121	596,813
Discounted C/Flow (6%)		250,718	(1,349)	(119,785)	(180,604)	12,556	76,783	72,255	57,215	47,755	51,343	53,660	50,505	38,952	33,906	32,264	12,386	13,134	(152)	(108)
Cumulative DCF			(1,349)	(121,134)	(301,737)	(289,181)	(212,398)	(140,143)	(82,928)	(35,173)	16,170	69,830	120,334	159,287	193,193	225,457	237,843	250,977	250,826	250,718
Max funding reqmt to positive cashflow		(417,092)	(1,430)	(136,020)	(351,122)	**(417,092)**	(359,552)	(259,078)	(169,147)	(94,388)	(1,675)	-	-	-	-	-	-	-	-	-

It can be seen that the NPV_6 of the cash flow is most sensitive to changes in price. An adverse change of less than 25% in the base case prices reduces NPV to zero. Sensitivity to operating costs less marked, with a 30% adverse change resulting in a positive NPV of $93 million. Capital costs are the least sensitive of the parameters tested. An adverse change of 30% reduces NPV to $130 million.

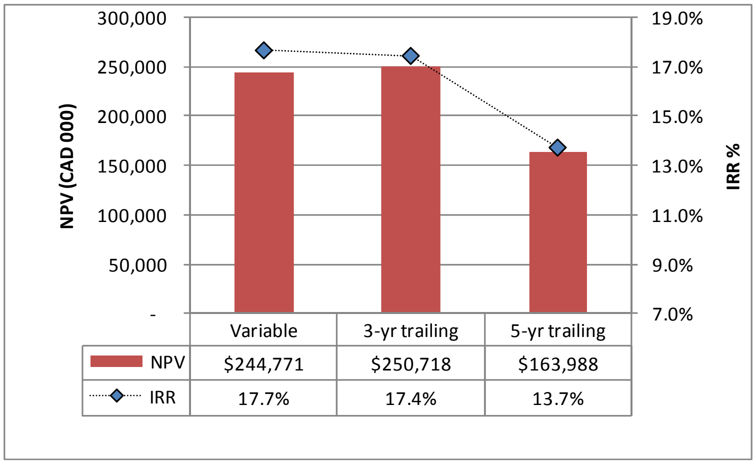
Figure 18.37
NPV Sensitivity Diagram



	70	75	80	85	90	95	100	105	110	115	120	125	130
Product Price	(81)	(23)	34	89	143	197	251	303	356	409	461	513	565
Opcosts	405	379	353	327	302	276	251	224	199	173	146	120	93
Capex	372	351	331	311	291	271	251	231	210	190	170	150	130

Given the sensitivity to price assumptions, and the volatility in metal prices observed in the market, Micon tested the cash flow using several other price scenarios, as described in Section 18.10.2. The results of applying these price scenarios to the base case production and cost assumptions used in the base case are presented in Figure 18.38. It is apparent that the project provides an attractive return when using the base case '3-year trailing' average prices obtaining during the 36 months to October 31, 2009. Similar returns are seen when using the independent forecast of a leading Canadian commercial bank, published in October, 2009. Returns using the 5-year trailing average are also positive, but less attractive.

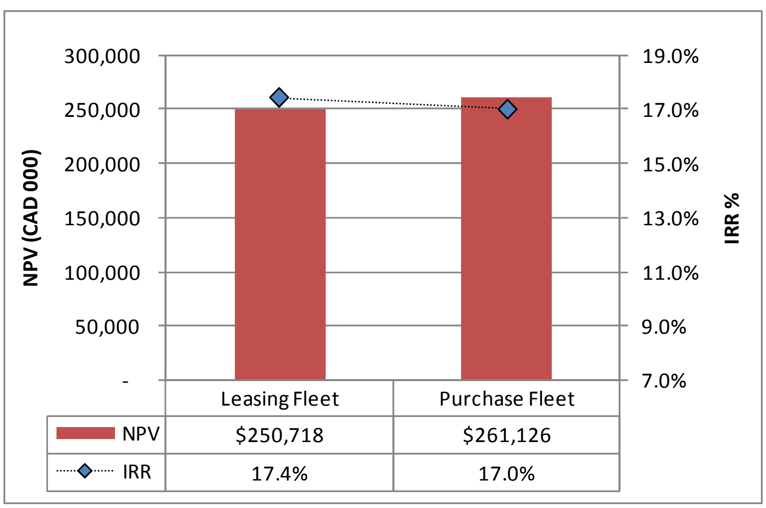
Figure 18.38
Economic Sensitivity to Price Assumptions



18.10.5.2 Mining Fleet Purchase versus Lease Finance

The base case cash flow considers the leasing of the mining fleet. Micon also considered the outright cash purchase of the equipment as an alternative strategy. Since the leasing option represents the introduction of leverage into an otherwise all-equity cash flow, and the after-tax cost of that finance is less than the rate of return in the project as a whole, it is to be expected that the NPV of that option should be less than in the all-equity (fleet purchase) scenario. Figure 18.39 shows this to be the case: NPV_6 for the all equity fleet purchase option is \$261 million, an increase of \$10 million versus the leasing scenario, though at the same time the project IRR falls from 17.4% to 17.0%.

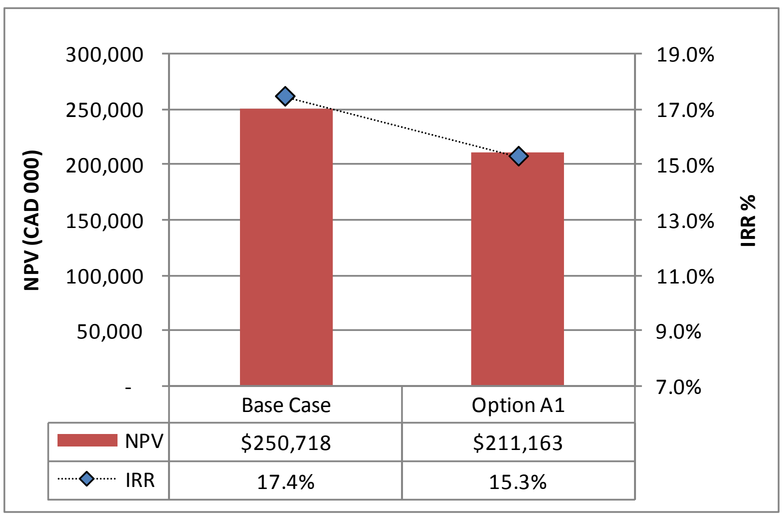
Figure 18.39
Mining Fleet Purchase versus Lease Finance



18.10.5.3 Process Solids Storage Options

The base case cash flow provides for the sub-aqueous deposition of process solids within Bamoos Lake. An alternative process solids disposal option was also considered. In Option A1, sub-aerial deposition of process solids is assumed. This requires a higher initial investment and increased operating costs compared to the base case. The impact of this on project economics is reflected in Figure 18.40, which shows a reduction in NPV from $251 to $211 million, and a reduction in project IRR from 17.4% to 15.3%.

Figure 18.40
Process Solids Storage Options



19.0 INTERPRETATION AND CONCLUSIONS

The updated Feasibility Study completed on the Marathon PGM-Cu project demonstrates the potential to generate strong cash flow under appropriate metal price assumptions. The base case results show that the project generates an IRR of 21.2% before tax and 17.4% after tax. The undiscounted payback period is 4.4 years, and the discounted cash flow is positive after 6 years. The NPV_6 is $250.7 million after tax. The sensitivity studies demonstrate that the project is quite sensitive to adverse changes in price assumptions and moderately sensitive to changes in operating cost or capital expenditure.

A total of 705 drill holes totaling 130,560 m of drill core were used to delineate the Marathon PGM-Cu deposit mineral resource estimate. This measured plus indicated material included in the estimate totals 115 million tonnes at an average grade of 0.24% Cu and 1.09 g/t of PGM+Au, as summarized in Table 19.1. Additional mineral resources within the Coldwell Complex which complement the Marathon PGM-Cu deposit includes the 25 million tonnes grading 0.35% Cu and 0.63 g/t PGM+Au at the Geordie Lake Deposit (refer to Marathon PGM press release of October 9, 2008.

Table 19.1
Marathon PGM-Cu Pit Shell Mineral Resource (Diluted Block Model)

Total Resource (Lower and Higher Grade) above $10.50/t NSR Cut-off

Category	Pit Shell 46 Mineral Resource						Contained Metal				
	Tonnes millions	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	Pd (oz 000)	Pt (oz 000)	Au (oz 000)	Cu (lb million)	Ag (oz 000)
Measured	94.3	0.846	0.243	0.088	0.262	1.599	2,564	736	266	545	4,847
Indicated	20.5	0.451	0.160	0.062	0.140	1.421	386	133	50	73	976
Measured + Indicated	**114.8**	**0.775**	**0.228**	**0.083**	**0.241**	**1.567**	**2,950**	**869**	**316**	**618**	**5,823**
Inferred	6.2	0.306	0.104	0.047	0.151	1.459	61	21	9	21	290

1. The mineral resources presented above are the subject of the Feasibility Study discussed in the present Technical Report.
2. The quantity and grade of reported inferred resources in this estimate are conceptual in nature and there has been insufficient exploration to define them as indicated mineral resources. It is uncertain if further exploration will result in their conversion to indicated or measured mineral resources.

The proven and probable mineral reserves estimated by Micon total 91.4 million tonnes of ore averaging 0.25% Cu and 1.15 g/t PGM+Au, as summarized in Table 19.2. These mineral reserves contain 497 million pounds of Cu, 3.4 million ounces of PGM+Au and 4.2 million ounces of Ag. The estimated life of mine rock mined is 263.5 million tonnes which gives an average mine rock to ore ratio of 2.88.

Table 19.2
Mineral Reserves for the Marathon PGM-Cu Deposit

Classification	Tonnes	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	Cu (M lb)	Pd Oz (000's)	Pt Ozs (000's)	Au Ozs (000's)	Ag Ozs (000's)
Proven	76,461,000	0.910	0.254	0.090	0.268	1.464	452	2,237	625	222	3,600
Probable	14,986,000	0.435	0.147	0.060	0.138	1.318	46	209	71	29	635
Total	91,447,000	0.832	0.237	0.085	0.247	1.440	497	2,447	696	251	4,235

1. The mineral reserves for the Marathon PGM-Cu deposit, as shown in Table 19.2 are included within the mineral resources shown in Table 19.1.

The effective date of the mineral resource and mineral reserve estimates presented in Table 19.1 and Table 19.2 is 24 November, 2009.

The Marathon PGM-Cu project comprises open pit mining and processing at an average rate of 22,000 t/d of ore to produce a saleable flotation concentrate containing Cu, Pd, Pt, Au, Ag and Rh. The life of the operation is estimated at approximately 11.5 years. The average annual metal production for the first five years is 37 million tonnes of Cu, 234 thousand ounces of PGM+Au and 182,000 ounces of silver.

The project schedule suggests that production of copper/PGM/Au concentrate could commence at the end of 2013. The present critical path item is the environmental assessment approval process and associated receipt of the required construction and operating permits.

20.0 RECOMMENDATIONS

20.1 MINERAL RESOURCES AND RESERVES

The immediate efforts of Marathon PGM will be concentrated on securing the required funding to proceed with the development of the deposit. Throughout the process, the company will undoubtedly be restructuring toward a producing mining company, with exploration geared toward reserve and resource sustainability.

20.2 PROJECT DEVELOPMENT

As a result of its Feasibility Study on the Marathon PGM-Cu Project, Micon recommends that Marathon PGM proceeds with the development of the project.

The life-of-mine capital cost for the Marathon PGM-Cu project is $495 million, including estimated initial capital costs of $351 million, as shown in Table 18.25. The estimated annual expenditures over the first three years of project development (Years -3 through -1) are detailed in Table 18.27, which provides the base case annual cash flows for the project.

The metallurgical testwork programs completed to date were used to design the process used in the updated Feasibility Study. This work includes a pilot plant run in 1986, bench scale tests including locked cycle tests (LCT) at SGS-L in 2004, 2007 and 2008, and LCT and a 6-day continuous mini pilot plant run completed by XPS in 2008 and 2009. Although Micon believes that the metallurgical testwork completed to date on the Marathon PGM-Cu deposit provides ample proof that good metallurgical performance can be achieved using conventional flotation, it is suggested that additional work may be worthwhile in order to try and reduce the reagent costs. This could entail reducing reagent dosage rates or substituting the existing reagent suite with less expensive chemicals.

Three feasible process solids (tailings) management areas (PSMA) for the Marathon PGM-Cu project were evaluated by AMEC. AMEC concluded that the sub-aquatic option (Bamoos Lake) seems to be the best PSMA since capital investment will be the lowest, no separation process between high/low sulphur process solids will be required and the risks associated to this option are low. However, this option utilizes an existing lake for containment which may be difficult to permit. AMEC commented that Option 1A represented the best on-land PSMA and should continue as an alternative during the advanced development and permitting process. AMEC further recommends the following:

- Detailed operational water management will need to be evaluated to take into account the detailed mining schedule.
- An extended geotechnical investigation is required for detailed design of the PSMA infrastructure. Furthermore, detailed evaluation of available clay deposits is required to determine dam design and cost.

The Marathon PGM-Cu project will likely be subject to both federal and provincial Environmental Assessment processes, and Marathon PGM intends to work in a coordinated way with both governments in order to drive the process forward with regard to achieving the necessary approvals in a timely manner.

21.0 SIGNATURES

The estimates of mineral resources and mineral reserves, and the results of the Feasibility Study are effective as of 24 November, 2009.

"Charley Murahwi" [signed and sealed]

Charley Murahwi P.Geo., Micon International Limited
January 08, 2010

"Sam Shoemaker" [signed]

Sam Shoemaker MAusIMM, Micon International Limited
January 08, 2010

"John Lemieux" [signed and sealed]

John Lemieux, ing., AMEC Earth & Environmental
January 08, 2010

"Richard Gowans" [signed and sealed]

Richard Gowans, P.Eng., Micon International Limited
January 08, 2010

"Christopher Jacobs" [signed]

Christopher Jacobs, CEng MIMMM, Micon International Limited
January 08, 2010

22.0 REFERENCES

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23.0 CERTIFICATES

CERTIFICATE OF AUTHOR
CHARLEY MURAHWI P.GEO.

As a co-author of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada" dated 08 January, 2010, I Charley Z. Murahwi do hereby certify that:

1. I am employed as a Senior Geologist by, and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, telephone 416 362 5135, fax 416 362 5763, e-mail cmurahwi@micon-international.com.

2. I hold the following academic qualifications:

    a. B.Sc. (Geology) University of Rhodesia, Zimbabwe, 1979;

    b. Diplome d´Ingénieur Expert en Techniques Minières, Nancy, France, 1987;

    c. M.Sc. (Economic Geology), Rhodes University, South Africa, 1996.

3. I am a registered Professional Geoscientist of Ontario (membership # 1618), a member of the Australasian Institute of Mining & Metallurgy (membership # 300395) and am also a registered Professional Natural Scientist with the South African Council for Natural and Scientific Professions (membership # 400133/09).

4. I have worked as a mining and exploration geologist in the minerals industry for over 28 years;

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 14 years on gold, silver, copper, tin and tantalite projects (on and off- mine), 12 years on Cr-Ni-Cu-PGE deposits in layered intrusions/komatiitic environments and the remainder as a consultant.

6. I visited the Marathon PGM project site from 16 to 17 October, 2009.

7. I am responsible for the preparation of Sections 7 to 15 of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated 08 January, 2010.

8. I have had no prior involvement with the mineral property in question.

9. I am independent of Marathon PGM as defined in Section 1.4 of NI 43-101.

10. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with this Instrument.

11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective dated: 24 November, 2009.

Signing date: 08 January, 2010.

"Charley Z. Murahwi" {Signed and sealed}

Charley Z. Murahwi, M.Sc., P. Geo. MAusIMM.

CERTIFICATE OF AUTHOR
SAM SHOEMAKER, Jr.

As a co-author of this report entitled “Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada”, dated 08 January, 2010, I, Sam J. Shoemaker, Jr. Member AusIMM do hereby certify that:

1. I am retained as a Senior Mining Engineer by, and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, tel. (416) 362-5135, fax (416) 362-5763, e-mail sshoemaker@micon-international.com.

2. I hold the following academic qualifications:

   B.Sc (Mine Engineer) Montana College of Mineral Science and Technology 1983

3. I am a Qualified Person as defined in the Instrument.

4. I am a member of Australasian Institute of Mining and Metallurgy (Member Number 229733); as well, I am a member in good standing of other technical associations and societies, including the Society for Mining, Metallurgy, and Exploration, Inc.

5. I have worked as a mining engineer in the minerals industry for 27 years. My experience includes resource estimation, mine development, open pit production, environmental compliance, financial evaluation, mine commissioning, long and short range mine planning, and open pit optimization with a variety of deposit types including gold, silver, copper, zinc, lead, uranium, nickel, platinum-group metals, iron, and industrial minerals.

6. I visited the Marathon PGM project site on August 24, 2009.

7. I am responsible for Section 17, mineral resources and mineral reserves, and Section 18.1 of this report.

8. I am independent of the issuer for which this report is required, other than providing consulting services, as defined by Section 1.4 of NI 43-101.

9. I have had no prior involvement with the mineral property in question.

10. I have read the Instrument and those portions of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Effective date: 24 November, 2009

Signing date: January 8, 2010

“Sam Shoemaker” {signed}

Sam Shoemaker, MAusIMM

CERTIFICATE OF AUTHOR
RICHARD M. GOWANS, P.Eng.

As a co-author of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated 08 January, 2010, I, Richard M. Gowans, P. Eng. do hereby certify that:

1. I am employed by, and carried out this assignment for
Micon International Limited
Suite 900, 390 Bay Street
Toronto, Ontario
M5H 2Y2
tel. (416) 362-5135 fax (416) 362-5763
e-mail: rgowans@micon-international.com

2. I hold the following academic qualifications:
B.Sc. (Hons) Minerals Engineering, The University of Birmingham, U.K. 1980

3. I am a registered Professional Engineer of Ontario (membership number 90529389); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have worked as an extractive metallurgist in the minerals industry for over 28 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the management of technical studies and design of numerous metallurgical testwork programs and metallurgical processing plants.

6. I visited the Marathon PGM project site on August 7, 2007.

7. I am responsible for the preparation of Sections 1, 2, 3, 4, 5, 6, 16, 18.2, 18.3, 18.6, 18.8, 18.9, 19 and 20 of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated 08 January, 2010.

8. I am independent of Marathon PGM Corporation, as defined in Section 1.4 of NI 43-101.

9. I was a QP and author of the Technical Report entitled "Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada" dated 02 February, 2009.

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date: November 24, 2009
Signing date: January 8, 2010

"Richard M. Gowans" {signed and sealed}

Richard M. Gowans, P.Eng.

CERTIFICATE OF AUTHOR
CHRISTOPHER A. JACOBS, C. Eng.

As the author of portions of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada" dated January 08, 2010, I, Christopher A. Jacobs, do hereby certify that:

1. I am employed by, and carried out this assignment for:
   Micon International Limited, Suite 900 – 390 Bay Street, Toronto, ON, M5H 2Y2
   tel. (416) 362-5135
   fax (416) 362-5763

2. I hold the following academic qualifications:
   B.Sc. (Hons) Geochemistry, University of Reading, 1980;
   M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.

3. I am a Chartered Engineer registered with the Engineering Council of the U.K. (registration number 369178);

4. Also, I am a professional member in good standing of: The Institute of Materials, Metals and Mining (Member); and The Canadian Institute of Mining, Metallurgy and Petroleum (Member);

5. I have worked in the minerals industry for 28 years;

6. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 10 years as an exploration and mining geologist on gold, platinum, copper/nickel and chromite deposits; 10 years as a technical/operations manager in both open pit and underground mines; 3 years as strategic (mine) planning manager and the remainder as an independent consultant;

7. I have not visited the Marathon PGM property;

8. I am responsible for the preparation of Section 18.10 and portions of Sections 19 and 20 of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated 08 January, 2010.

9. I am independent of Marathon PGM Corporation, as defined in Section 1.4 of NI 43-101;

10. I was an author of the Technical Report entitled "Technical Report on the Updated Mineral Resource Estimate and Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada" dated 02 February, 2009

11. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;

12. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date: 24 November, 2009

Signing date: January 8, 2010

"Christopher A. Jacobs" {signed}

Christopher A. Jacobs, C.Eng MIMMM

CERTIFICATE OF AUTHOR
JOHN LEMIEUX, ING.

As the author of portions of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada" dated January 08, 2010, I, John Lemieux, do hereby certify that:

1. I am employed by, and carried out this assignment for:
   AMEC, Suite 400 – 1868 Blvd. des Sources, Pointe-Claire, QC, H9R 5R2
   tel. (514) 429-6555
   fax (514) 429-6550

2. I hold the following academic qualifications:
   B.Eng. Mining, University of McGill, 1994;

3. I am a Chartered Engineer registered with the "Ordre des Ingénieurs de Québec" (registration number 113626);

4. Also, I am a professional member in good standing of The Canadian Institute of Mining, Metallurgy and Petroleum;

5. I have worked in the minerals industry for 15 years;

6. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 15 years as a geotechnical engineer for the mining industry, specifically in mining environement for containment of process water and solids including field investigation, conceptual design, detailed engineering, construction supervision and management.

7. I have not visited the Marathon PGM property, however, my assistant engineer and unit manager have on the 22 and 23 of September 2009;

8. I am responsible for the preparation of Section 18.4 of this report entitled "Technical Report on the Updated Feasibility Study for the Marathon PGM-Cu Project, Marathon, Ontario, Canada", dated January 08, 2010.

9. I am independent of Marathon PGM Corporation, as defined in Section 1.4 of NI 43-101;

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;

11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date: 24 November, 2009

Signing date: January 8, 2010

"John Lemieux" {signed and sealed}

John Lemieux, B.Eng Mining