FEASIBILITY STUDY

CANADIAN MALARTIC PROJECT (MALARTIC, QUÉBEC)

OSISKO

A FRESH OUTLOOK ON MINING

PREPARED BY

BBA

COLLABORATION

Golder Associates

GENIVAR

G Mining Services Inc.

micon

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

The Feasibilty Study Technical Report of the Canadian Malartic project (Feasibility Study) has been prepared at the request of Osisko Mining Corporation (Osisko). The Feasibility Study was prepared and compiled by BBA in a collaborative effort between BBA and Osisko together with a number of specialized consultants. This Technical Report was prepared according to the guidelines set out under “Form 43-101F1 Technical Report” of National Instrument 43-101 Standards of Disclosure for Mineral Projects adopted pursuant to Section 143 of the Securities Act (Ontario).

BBA has worked in collaboration with the following firms:

- Genivar - Environmental Impact Study and Community Considerations
- Micon International Limited - Mineral Resource Estimate
- Belzile Solutions Inc. - Mineral Resource Estimate
- G Mining Services Inc. - Mining and Mineral Reserves and Financial Analysis
- Golder Associates - Pit Geotechnical Evaluation and Tailings Water Management.

The mineral resource estimate used in this Feasibility Study is based on the NI 43-101 compliant “Technical Report for the Canadian Malartic Gold Project” authored by B. Terrence Hennessey et al. of Micon, dated October 23, 2008.

All costs in this report are expressed in United States Dollars unless specifically stated otherwise.

1.1.1 Property Description and Location

The Canadian Malartic gold deposit is located in Québec, Canada, immediately south of the town of Malartic and approximately 25 km west of the town of Val-d’Or. It is centred on Latitude 48º 7’ 45” N and Longitude 78º 7’ 00’’ W. The deposit straddles the southern margin of the eastern portion of the Abitibi Gold Belt which is host to a number of large gold deposits.

1.1.2 History

The original Canadian Malartic deposit was discovered in 1926. Underground development began in 1928 following a 26 hole surface drill program and production started in 1935. The mine closed in 1965 after producing 1,076,000 oz of gold from 9.93 Mt of ore grading 3 to 6 g/t Au. Gold was finely disseminated and occured in the native state, and was recovered in a mill by standard cyanide leach process (90% average recovery over the mine life). Following closure of the mine, the property remained idle until purchased by Lac Minerals, Ltd (Lac Minerals) in 1979.

From 1980 to 1988, Lac Minerals' exploration program led to the definition of 5 near-surface gold zones. Metallurgical tests carried out by Lac Minerals indicated a non-refractory ore with recoveries of 86%-90% on a standard cyanide leach circuit.

The project was suspended when Barrick Gold Corporation (Barrick) acquired Lac Minerals in the early nineties. Barrick sold the property to McWatters Mininc Inc. (McWatters) in 2003. McWatters went bankrupt in 2004, and in late 2004, Osisko purchased a 100% interest in the property (initially 6 claims and one mining concession) through the McWatters liquidating trustee. A 2 to 3% sliding scale net smelter royalty is payable to Royal Gold Inc. (Royal Gold), half of which can be purchased back by Osisko for CDN$1.5 M. Since the initial acquisition, Osisko has successfully acquired additional claims and the property now comprises 126 claims and 1 mining concession with a total surface area of 5,654 hectares.

A 3,000 tpd mill with cyanide and flotation circuits (East Malartic mill) was located immediately east of the property, which was last operated in 2002 to treat ore from Barrick's Bousquet Mine. The inactive mill was asquired by Osisko in 2007 and has been dismantled in the fall of 2008 and the existing tailings responsibility remains with the Québec Government.

1.2 Geology and Exploration

The geology of the Canadian Malartic project is largely defined by meta-sedimentary units of the Pontiac Group and lies immediately south of the Cadillac Tectonic Zone. The north-central portion of the property covers an approximately 3.5 kilometre-long section of the fault corridor and is underlain by mafic-ultramafic metavolcanic rocks of the Piché Group cut by porphyritic intrusions, as well as metasediments of the Cadillac Group to the north of the fault zone. In the region of the Canadian Malartic project, the fault corridor experiences a change in orientation from south-east to easterly. The easterly portion of the fault zone is referred to as the Malartic Tectonic Zone. Two major structures, the Malartic (Cadillac) and

Sladen Faults, define the northern and southern boundaries of the tectonic zone in the immediate Malartic area. The Malartic fault is oriented N260°E – N280°E and dips 75° to the north, whereas the Sladen fault is oriented N090°E – N100°E and dips variably from 70°S to sub-vertical.

The rocks of the Pontiac and Piché Groups are intruded by a number of epizonal felsic porphyritic bodies, variously described as syenites, quartz syenites, quartz monzonites, granodiorites and tonalites. The geometries of these felsic intrusions are highly variable, and occur on the property as sills, dykes, discontinuous lenses or small isolated stocks. The gold deposits of the Malartic area are related to porphyries that are generally considered to be syenitic (alkaline) in composition and of Timiskaming (syntectonic) age.

Mineralization in the Canadian Malartic deposit occurs as a continuous shell of 1-5% disseminated pyrite with fine (<20 microns) native gold and traces of chalcopyrite, sphalerite and tellurides. The mineralization is mostly hosted by altered clastic sediments of the Pontiac Group (turbiditic greywacke, mudstone and minor siltstone) overlying an epizonal dioritic porphyry intrusion. Mineralization also occurs in the upper portions of the porphyry body. The porphyry intrusion pinches out in the Sladen Malartic Mine and disseminated mineralization continues in the silicified greywacke, forming a subvertical tabular body that is truncated by the Cadillac fault at the western extremity of the East Malartic mine.

The close spatial association between voluminous, low-grade, disseminated gold mineralization and an epizonal, intermediate porphyry intrusion, as well as the presence of widespread potassic alteration throughout the system, suggest the Canadian Malartic deposit may be an Archean porphyry gold system.

Three distinct phases of drilling have occurred on the Canadian Malartic deposit since its discovery in 1926. During the first phase, from 1928 to 1963 by Canadian Malartic Mines Ltd., over 5,000 surface and underground holes were drilled. These drill holes were predominantly drilled from underground as grade control drilling. In the second phase from 1981 to 1985, Lac Minerals completed 502 drill holes for 43,495 m of drilling. These drill holes were drilled from surface and defined shallower resources (mostly less than 200 m below surface). In the third phase since 2005 and at the end of May 2008, Osisko had drilled approximately 1,700 drill holes for 386,760 m of NQ diamond core. These drill holes were mostly drilled on nominal 30m x 30m grid spacing, and have been the main source of data for the latest resource estimate.

The quality control data routinely submitted as part of the recent Osisko-managed exploration programs includes certified standards and duplicate data. Analysis of the standards data sent to Chemex indicates that assaying is within industry acceptable limits of accuracy. The blank samples did not display evidence of significant contamination. In addition, internal laboratory standards and blanks provided by Chemex were reviewed and also were within industry acceptable limits of accuracy. Check assay programs supervised by independent consultants also adequately reproduced the original assays.

1.3 Mineral Resource Estimates

The Canadian Malartic deposit has already been the subject of two previous NI 43-101 compliant resource estimates (Gossage 2007a and 2007b). These reports, completed by RSG Global Consulting Ltd. (RSG) of Perth, Australia, detailed the exploration completed on the project and presented inferred mineral resource estimates for the Canadian Malartic deposit. The second estimate (Gossage 2007b) of 286.2 Mt at 0.92 g/t Au represented 8.43 million inferred oz of gold at a 0.4 g/t Au lower cut-off grade.

Runnels (2008) presented the results of a Preliminary Assessment report including economic analysis for the deposit using the second inferred resource estimate. This analysis indicated an in-pit inferred resource of 287.7 Mt grading 0.84 g/t Au, equivalent to 6.55 M oz of contained gold associated with an 84% recovery rate and a diluted cut-off grade of 0.294 g/t Au. The study also indicated that over the first three years of production, Canadian Malartic could average 572,000 oz of gold per year at an average head grade of 1.05 g/t Au with cash costs averaging $314 per oz (including royalties). Over the first three years, the operation could generate pre-tax cash flow of $731 M. Over a projected 14 year mine life the deposit could produce an average of 457,800 oz of gold per year at an average cash cost of $381 per oz (including royalties), generating pre-tax operating cash flow of $2.58 billion, with over $1 billion pre-tax cash flow generated in the first five years of production. The study showed a potential Internal Rate of Return of 22.2% and a payback period of 39 months.

As a result of the success in developing a large inferred mineral resource at the Canadian Malartic project, Osisko embarked upon a program of definition drilling to infill the previous programs and to provide sufficient data to upgrade the mineral resource to the higher confidence categories of measured or indicated. As of May 2008, the deposit was infilled with a 30m x 30m pattern of diamond drill holes, totalling 1,327 holes for approximately 316,230 metres of drilling, in addition to approximately 229,000 metres of historic drilling (original Canadian Malartic Mines and Lac Minerals), the data from which a Measured and Indicated (M&I) resource estimate was calculated by Hennessey et al. (2008) of Micon and used in the present report.

In the process of completing this latest resource estimate, Micon validated and verified the database and interpreted the available data. This led to the following conclusions:

- For the updated estimate Micon chose to use a portion of the historic drilling data available from the original Canadian Malartic Mines. After a comparative study between the historical and recent data, a bias was identified for values in the low grade domains of the geological model (<1.0 g/t Au) and all of the historical data in those low grade zones were discarded. In the higher grade populations of the high grade domains (>1.0 g/t Au), no evident bias was observed and these data were retained and used for the resource estimation presented herein.

- The completion of the infill drilling on a 30m x 30m pattern successfully confirmed continuity of the early inferred resource estimates identified from the first phases of drilling. The majority of the mineral resources are now classified in the indicated category.

- Outlier values in the assay population were analyzed and top cuts were generally applied, resulting in a decrease in grade of 0% to 2.35% depending on the mineralized zone.

- The block size dimensions chosen for the model (20m x 10m x 10m) were based on the existing drilling pattern (30m x 30m) and mine planning considerations (10 m bench heights).

- The underground drift and stope workings for the deposit were modeled based on 3D DWG files and drill intersections of these workings. In several cases the stope workings were extrapolated based upon the surrounding drill hole information and likely mining patterns, even if the complete stope outlines were not present in the supplied files. The underground workings model was used to deplete the resource block model.

- The resources were estimated using ordinary kriging (OK). A parallel estimate was conducted using the inverse distance squared (ID^2) method as a check. The discrepancy between the OK model and the ID^2 model is less than 2%.

- A total of nine domains were modeled to define the zones of mineralization. One broad domain based upon a nominal 0.2 g/t Au lower cut-off was created to define the extents of the low grade mineralization. This zone was coded as domain 210. Eight higher grade zones, based upon a nominal 1.0 g/t Au lower cut-off, were created to constrain any high-grade zones within the broader mineralized envelope. These eight zones were coded as domains 110 to 180.

- In order to evaluate the economic potential of the resources, a pit optimization was generated using the Measured and Indicated resources only. Base case pit parameters include a maximum 55 degree slope angle, ore base cost of $6.38/tonne milled (based on a 55,000 t/d milling rate), variable mining cost depending on the zone and depth ($1.46 to $1.77/t mined with increments of $0.018/t mined per 10 m bench) and variable processing recoveries depending on the grade (from 75% to 92%). The gold price used was $775/oz. These parameters were taken from the recent Preliminary Assessment study and/or updates to it.

The resulting in-pit mineral resources for the Canadian Malartic deposit are presented in Table 1.1 below.

Table 1.1: Canadian Malartic Project In-Pit Mineral Resources

Category	Mineral Resources (Mt)	Grade (g/t Au)	Ounces (Moz Au)
Measured (M)	4.5	1.29	0.19
Indicated (I)	173.7	1.12	6.23
Total M & I	**178.2**	**1.12**	**6.42**
Inferred	5.0	0.81	0.13

1.4 Mining and Mineral Reserves

Conventional open pit mining methods are chosen to exploit the Canadian Malartic deposit because of its low grade and proximity to surface. Given the important quantities of low grade mineralization, it was determined that a high milling production rate of 55,000 tpd would be economically feasible and would maximize the value of the deposit. Due to the large width of the mineralized zones, selective mining was not considered as an option.

A resource model was constructed using Gemcom with large blocks measuring 20mE x 10mN x 10m in elevation. This resource model was then used to perform open pit optimization using the Whittle software, which is based on the Lerchs-Grossman algorithm. For this feasibility-level study only Measured and Indicated resource blocks were considered for optimization using the parameters described in Table 1.2. The optimization parameters used to establish the optimal shape of the pit are slightly different than those used for financial analysis as presented in Section 15. However, the differences do not impact the validity of the optimization work performed since there is no change to the cut-off grade.

Table 1.2: Summary of Pit Optimization Variables

Parameters	Units	Value
Gold price	$/oz Au	775.00
Transportation & refining cost	$/oz Au	2.00
Royalties (1.5%)	$/oz Au	11.63
Net gold price	$/oz Au	761.37
Process Gold recovery	%	Variable by zone (79 to 92)
Processing cost	$/t milled	5.30*
General and Administration Costs	$/t milled	0.70*
Rehabilitation	$/t milled	0.18
Sustaining capital	$/t milled	0.20
Mining dilution	%	7.6%*
Diluted cut-off grade	g/t Au	0.36
Reference mining cost	$/t mined	Variable by bench (1.46 to 1.77)
Incremental bench cost per 10 m	$/t mined	0.018
Overall slope angle	Degrees	Variable by sector 46 to 55

*For open pit optimization only.

The process recovery is estimated using equations developed for each metallurgical zone based on extensive testing. The following recovery equations were developed and used in the optimization:

- Recovery equation 1 (for west zone & north zone below RL160)
  - Recovery = [74.41 + (12.300 x grade)] /100 (maximum 92%)
- Recovery equation 2 (for east zone & north zone above RL160)
  - Recovery = [76.15 + (7.339 x grade)] /100 (maximum 89%)
- Recovery equation 3 (for south zone)
  - Recovery = [78.50 + (1.654 x grade)] /100 (maximum 86%)

For simplification purposes a 1.5% royalty was applied to the overall resource in the Preliminary Assessment of the Canadian Malartic project. For this feasibility-level study a more precise calculation was performed by coding all blocs falling within the perimeter of mining titles subject to royalty agreements.

The optimal pit shells generated in Whittle were used to construct detailed pit designs using the design tools in Gemcom. The pit designs implement the pit slope profiles recommended by the Golder pit slope assessment study which concluded that the rock quality and structural conditions generally favour the development of steep inter-ramp slopes in all areas of the pit except the Northeast sector.

Due to the potential for structurally-controlled planar failures along moderately south-dipping structures in the Northeast sector, bench face angles of 60 degrees, with 8 m catch benches at 20 m intervals, was recommended for an inter-ramp slope angle of 46 degrees. With the implementation of an effective pre-split the development of steeper slopes with a designed 8.5 m catch bench, a 75 degree bench face angle for an inter-ramp angle of 55 degrees is recommended for the remaining pit sectors.

Access to the pit is via a 10% decline ramp which was designed with a width of 35 m to accommodate the Caterpillar 793F haul trucks.

Mining reserves were estimated by designing ore outlines or mining shapes around economic mineralization for every 10 m bench. The ore outlines include a 1 m dilution envelope around economic ore blocks and also enclose marginal material surrounded by economic mineralization.

The final pit contains a Proven and Probable reserve of 183.3 Mt at 1.07 g/t Au for an in-situ gold content of 6.28 Moz with the breakdown presented in Table 1.3. The average strip ratio for the pit is 1.78:1 (Waste:Ore). The phasing technique was evaluated using the pushback chooser module in Whittle which iteratively searches for the best combination of pitshells to maximize Net Present Value (NPV). Mining will be accomplished with three phases to achieve the final pit limits with the content of each phase presented in Table 1.4. The results of the pushback designs demonstrate that with every phase the strip ratio increases and the average grade decreases.

Table 1.3: Proven & Probable Reserves

Category	Reserves (kt)	Grade (g/t Au)	In-Situ Ounces (koz Au)
Proven	5,156	1.14	189
Probable	178,173	1.06	6,094
Proven & Probable	**183,329**	**1.07**	**6,283**

Table 1.4: Pit Design Results

Mining Phase	Total (kt)	Waste (kt)	Strip Ratio (W/O)	Ore (kt)	Grade (g/t Au)	Ounces (koz Au)
Phase 1	75,338	43,845	1.39	31,493	1.19	1,200
Phase 2	159,086	103,717	1.87	55,368	1.17	2,079
Phase 3	275,706	179,239	1.86	96,467	0.97	3,004
Total	**510,130**	**326,801**	**1.78**	**183,329**	**1.07**	**6,283**

The final pit design has the ramp entry near the center on the south side of the pit and ramps around counter clockwise. This location was selected to coincide with the mill site and waste dump location to minimize haulage distances. Two switchbacks are introduced in the ramp system to exploit a natural plunge of the mineralization and to keep the ramp system away from the open stopes that are found within the pit. The final pit measures 2000 m along strike and 920 m in width with a final depth of 380 m.

Waste material will be co-disposed with tailings material in a central facility. The waste pile will serve to contain the thickened tailings on the north side of the facility. Over time waste placement will occur over previously deposited tailings. A total of 326.8 Mt of waste are to be placed on the waste pile which will reach a height of 97 m.

The mine production schedule was developed to feed the mill at a nominal rate of 55,000 t/d beginning in the second quarter of 2011 (Year 1). The annual mining rate is initially set at 51 Mt (142.5 kt/d) for the first year of operation and increases to 64 Mt (178.8 kt/d) for the following four years and then decreases. An operating ore inventory is implemented at the beginning of the production to improve economics by delaying the processing of lower grade material as much as possible. During the pre-production period a total of 25.1 Mt will be mined, including 6.4 Mt of ore at an average grade of 1.07 g/t Au and 18.7 Mt of waste. The ore stockpile built up during pre-production will be drawn down in part during Year 2 (2012) and Year 3 (2013).

The main highlights of the pit design are the following:

- Total amount of 510 Mt mined from the pit
- 183.3 Mt milled @ 1.07 g/t Au (average)
- In-situ gold content of 6.28 Moz
- Average waste to ore stripping ratio of 1.78:1
- Mine life of 9.1 years.

In order to maximize productivity and limit the number of units operating in the pit, large scale equipment were selected for the mine operation. The mine will operate 24 hours per day, 365 days per year with four crews working 12 hour shifts on rotation. It is expected that 5 days per year will be lost due to weather.

Loading of waste and ore will be carried out with two 28 m^3 electric driven hydraulic shovels (O&K RH340-B) and transported by 227 t class rigid trucks (CAT 793F). During the second year of operation a 21 m^3 hydraulic shovel (O&K RH200) is planned to increase the mining rate from 51 Mt/y to 64 Mt/y. In addition, a front-end wheel loader (CAT 994F HL) will complement the shovels and re-handle ore from the stockpiles. The haulage fleet increases over time as the haulage distances increase due to a deepening of the pit and higher dump levels for waste. The haulage fleet consists initially of 12 units and increases to 22 units by the sixth year of operation. Ancillary equipment includes two graders (CAT 16 M), four track-type dozers (CAT D10T), one wheel dozer (CAT 834H), two water trucks and other support equipment.

Drilling and blasting activities have been designed to control blast induced vibrations and airblast overpressure on the neighbouring town of Malartic. Several blast patterns will be implemented whereby the explosives charge per delay will increase as mining progresses to the south with an increase in hole diameter and bench height. For the zone nearest to the town, smaller 114 mm diameter holes and 5 m benches will be mined to reduce the explosive charges. However, for 78.1% of the tonnage, larger 216 mm diameter holes and 10 m benches will be drilled.

Three drill models will be required to drill the four patterns. Drilling of 114 mm diameter holes will likely be executed by a contractor given that there is a limited amount of drilling to do and that it will not be required for the duration of the mine. Intermediate patterns will be drilled using diesel powered, self-contained crawler mounted drilling rigs designed for Down the Hole (DTH) blasthole drilling. The angle drilling capability of this drill rig type also makes it ideal for drilling the pre-split holes. The large 216 mm diameter holes will be drilled with a diesel powered, self-propelled crawler mounted blast-hole drills for mining. This rotary type blasthole rig is capable of DTH drilling with the addition of a high pressure air compressor (1450 cfm/350 psi).

Bulk emulsion will be used and delivered in the hole with explosive pump trucks. Initiation will be accomplished with electronic detonators and boosters. The precision of electronic detonators will allow for a better control of ground vibrations. Powder factors will range from 0.26 to 0.30 kg/t of rock. Pre-splitting will be implemented to maximize stable bench face and inter-ramp angles along final walls with a targeted charge weight of 1.0 kg/m^2 of face area.

The total mine department workforce is 262 the first year of operation and reaches a peak of 322 persons by the fifth year for an average of 297 over the mine life.

1.5 Mineral Processing and Metallurgical Testing

1.5.1 Metallurgical Test Program

Initial testwork was conducted in 2006 by Resource Development Inc, on 6 t of drill core material from separate core holes within the Canadian Malartic ore body. The metallurgical testwork consisted of grinding, leaching, gravity concentration, flotation and heap leach tests, and cyanide destruction studies.

Additional testwork carried-out by Metso also in 2006, provided initial values for the design of the comminution circuit. Those grinding tests were carried out on a bulk sample of 5.4 t and included a JKtech simulation of the grinding circuit with the goal to establish the expected capacity of the 38ft x 21ft EGL SAG mill and the two 24ft x 36.5ft EGL ball mills.

The amenability of the low grade ore (0.6 g/t Au) to heap leach was studied at different crushed sizes. The gold recovery ranged from 32% to 39%. This option was discarded.

SGS was then contracted in 2007, to carry out a more comprehensive test protocol under the supervision and guidance of BBA and Osisko. Equipment and reagent suppliers also participated in the verification program for specialized tests. Results were reported to and reviewed by BBA and Osisko when they became available. The test protocol included the sample preparation, gold deportment studies and the following tests: grinding, flotation, leach on whole ore, heap leach, environmental program and cyanide destruction. Those investigations formed the basis of the Preliminary Assessment study.

For the Feasibility Study an even more comprehensive metallurgical test program was required to complement the previous testwork in order to finalize the flowsheet and equipment selection for the project. The continuation of the testwork was carried out principally by SGS and also by URSTM with specialized tests related mostly to equipment sizing and reagent selection carried out by FLSmitdh, Outotec, PSI, Knelson, Falcon, SNF, Ciba, Cyplus, and others.

The majority of the deposit consists of four ore types, namely potassic altered porphyry with carbonate (CPO), silicified porphyry (SPO), potassic altered greywake with carbonate (CGR) and silicified greywake (SGR). The metallurgical samples were selected to give a good representation of the ore zones in the deposit. A total of approximately 338 drill holes were used for testwork sampling.

A deportment study was conducted by SGS with the objectives of determining the bulk mineralogy and the occurrence of gold in the various lithologies in order to evaluate any mineralogical factors that may affect recoveries. The findings of this examination were that gold mainly occurred as liberated native gold fine particles in the range 10 to 20 microns with some inclusions in pyrite which is the major sulphide mineral.

An additional deportment study was carried out including a series of diagnostic leach tests on composites of leach tails. The objective of this investigation was to understand the occurence of gold in the leach tails and to identify any mineralogical factors that could affect the dissolution of gold. The results showed that the gold grains not leached in the tails were very small (averaged 2 microns in size) and that 77% of the gold in the tails was associated with sulphides. This led to the conclusion that a high percentage of the gold that is not leached is ecapsulated in sulphides and that a finer grind of that material would have the most effect on the recovery of gold into solution in the leach circuit.

An additional set of grinding testwork was carried out by SGS in order to provide values for an updated design of the comminution circuit. Both ball mill work index (BWi) and Sag Mill Comminution (SMC) determinations indicated that the ore is moderately hard to hard. Those results proved to be very consistent within each set of tests and between the independent test facilities (SGS and Metso). Grinding circuit simulations, using the SAG model predicted by the SMC determinations and using the Bond Wi determinations were carried out.

The grinding circuit simulations resulted in a prediction that a grinding circuit consisting of one 26,000 hp SAG mill and three 16,000 hp ball mills could produce a product of P_{80} = 64 microns at a daily throughput of 55,000 tpd which became the new design criteria for the process plant.

Initial bulk sulphide rougher flotation testwork was conducted with the goal to promote high gold recovery in the concentrate while rejecting the flotation tails to final plant tails. The major conclusions indicated that the overall recovery, with leaching of the flotation concentrate, was projected to be in the range of 78%. This option was discarded. Additional flotation testwork was carried out to assess the viability of the leach of both the flotation concentrate and the flotation tails. The flotation was carried out at a coarse grind of P_{80} = 130 microns for flotation feed. The flotation concentrate was reground to a P_{80} of approximately 30 microns before leaching. The overall recovery from the test results was determined to be in the range of 86 to 90% depending on head grade and location within the pit. Because the grind for flotation feed is coarse and only the flotation concentrate is reground, the existing ball mill grinding capacity would be sufficient. This process route however was not chosen for the project initiation as significantly more testwork is required to prove that the initial metallurgical findings are robust enough to justify the additional investment required. The initial plant design will accommodate a future flotation circuit and re-arrangement of the grinding circuit, if and when additional testwork results in a decision to change the process to add flotation. The design of the process plant is therefore based on the whole ore leach process flowsheet.

A series of scoping tests for whole ore leach were conducted on individual lithologies, composites of lithologies and overall composite samples, to study the effect of process variables on gold extraction. The main conclusions were that the majority of the gold is extracted at 24 hours and that the gold recovery responds to finer grinding. Considering the average head grade of the deposit of 1.07 g/t Au with material from all locations a gold recovery of slightly above 84% was achieved with a leach feed at P_{80} = 64 microns and a leach time of 28 hours.

One of the findings from the initial leach tests was that the behavior of the ore to leaching was different for each zone in the mine. Based on the results from initial tests, the orebody was divided into four main zones, namely: West, South, East and North. The North deep part of the North zone behaved similarly to the West zone and the North shallow part behaved similarly to the East zone. The partition of the mine into zones was then based on the main zones defined as West (including North deep), South and East (including North shallow). Leach testing then focused on defining the ore response to leaching on a zone basis and developing grade-recovery curves for each of the ore zones in the mine. In doing this it was determined that, based on a weighted average calculation, the recovery for the overall mine was 85.9%. This recovery was obtained by attributing the respective recoveries to each block of the pit model design according to its head grade and location within the mine.

The environmental verification program included acid based accounting (ABA) tests on tails from leach tests by lithologies. The results indicated that CPO, SPO and CGR have a low net acid neutralizing potential and also suggested that SGR could have a potential to be acid generating. The NAG tests indicated that only SGR could have the potential to be acid generating, while each of CPO, SPO, CGR had high neutralizing pH values suggesting that the sulphide is unavailable for oxidation and these lithologies are not acid generating in nature. An additional series of tests, started in March 2008, have resulted in similar findings. Results from humidity cell tests on leach tails, started in November, 2007 and tests on various leach tails and waste ore material, started in March 2008, are indicating that the material should not be acid generating. All cells are running at neutral pH. The humidity cell tests are ongoing.

1.5.2 Process Plant Design

Run of mine ore is transported to a gyratory crusher. The ore is crushed, stockpiled in a covered pile then conveyed to the SAG mill. The SAG mill is in a closed circuit with a pebble crusher and feeds a two stage ball mill grinding circuit, including three identical ball mills. Each ball mill is close circuited with hydro-cyclones.

The tertiary grinding circuit cyclone overflow is screened for trash removal and thickened to about 50% solids before being fed to the leach tank circuit where oxygen is injected to increase the kinetics of the dissolution of gold in the leach circuit. The slurry then flows by gravity to two parallel sets of CIP pump cell carousels where activated carbon absorbs the gold in a counter-current flow arrangement. The loaded carbon is screened from the slurry and transferred to the stripping circuit where the gold is stripped into a gold pregnant solution and deposits, in the form of a sludge, onto stainless steel cathodes. The deposited gold sludge is pressure washed to the bottom of the electrowinning cells and is subsequently filtered, dried and poured into gold dore bars.

The stripped carbon is reactivated in two kilns and then re-used in the carousel pump cell CIP circuit. The slurry from the last stage of the tank series in the carousels is barren in gold, so is considered as final tailings and is directed to the tailings thickener. The slurry is thickened to approximately 68% solids by weight and pumped to the Detoxification plant where cyanide content is significantly reduced to less than 20 ppm using a CombinOx® Process Cyanide Destruction method. The slurry is subsequently pumped to the tailings retention area where most of the water remains contained in the slurry and a small percentage drains out to be reclaimed back to the process.

A final effluent treatment plant will be used prior to discharging any excess water into the environment. The plant will be located at the tailings area and will use the peroxygen based process.

Process design criteria are based on a processing plant of 55,000 tpd capacity with a plant utilization of 92%. The average gold head grade for plant feed will be 1.07 g/t Au with an overall gold recovery of 85.9%. The average silver head grade will be 1.68 g/t Au with an overall recovery of 69.3%. The mill production for life of mine is shown in Table 1.5.

Table 1.5: Total Mine Life Mill Production

Tonnes Milled	Average Gold Grade (g/t Au)	Gold Production (oz)	Average Silver Grade (g/t Ag)	Silver Production (oz)
183,329,000	1.07	5,397,000	1.69	6,880,000

A manpower level of 90 people, split into 21 staff and 69 workers will be required to operate and maintain the mill.

1.6 Environmental and Permitting

1.6.1 Permitting

In anticipation of the environmental permitting needs for the project, Osisko's consultants initiated an environmental baseline data collection program in June 2007. This data collection program lasted until February 2008. Since then, Osisko with the assistance of Genivar, Golder, BBA and G Mining have been working towards the final objective of obtaining an environmental permit for the project by May, 2009. A summary of the environmental assessment process and proposed activities leading up to the permitting of the project are presented below:

- June 2007 to February 2008 - Environmental baseline data collection (**Completed**)
- February 2008 to September 2008 - Environmental Impact Assessment (**Completed**)
- September 2008 - EIA documentation forwarded to the Québec Ministry of Sustainable Development, Environment and Parks (MSDEP) (**Completed**)
- Review of the EIA by the Government – September to December 2008 **(in Progress)**
- Acceptance of the EIA by MSDEP – December 2008
- Public hearings - January to April 2009
- Approval to proceed (Order in Council) - May 2009
- Certificates of authorization – June 2009.

Note that the Canadian Malartic project (mining, processing) is subject to the EIA approval. The houses and institutions relocation, street infrastructure, East Malartic tailings pond site closure, power line and land preparation (named satellite projects) are not part of the EIA. Those sub-projects have either received their permits or their permitting is in progress according to schedule.

1.6.2 Environmental Highlights

Archaeology and built heritage – The study held in 2007 covers an area of approximately 14 km^2. The archaeological component was completed using previously known archaeological data and eco-geomorphological criteria. No known archaeological sites are present within the boundaries of the studied area. The founding of the Town of Malartic and its current occupation are direct consequences of gold and silver discoveries in the 1920s and 1930s. Malartic does not have any classified cultural property.

Community – The last survey held in December 2007 shows a high level of support for this project. In fact, 84% of the population was in favour, 6% not in favour but accepted Osisko's project, only 3% were against the project and 7% were neutral. Public meetings are held every month and will be conducted until final approval. Construction of new lots was begun in fall 2007. At this time, 97% of the population involved in the relocation process has accepted to either move to the new location or have sold their property to Osisko. The balance 3% are still in the process of negotiating with Osisko. The plant (industrial complex) will be located 2.5 km from the town.

Wildlife and Aquatic Habitat – Mammals: the study zone includes a diversity of small and large mammals (red-backed mole, several species of shrew, deer mice, red squirrel, snowshoe hare, black bear, moose, etc.). Amphibians and reptiles (herpetofauna) are scarce. None of the documented species are at risk.

Fisheries and Aquatic Habitat - Among these, walleye, sauger, pike, yellow perch, northern pike, burbot and brown bullhead are species that are popular for sport fishing, with northern pike and the percidae (walleye, sauger, yellow perch) attracting major interest in the region. None of these fish represent species at risk. The overall quality of water and sediments is acceptable for maintaining aquatic life. The water is slightly cloudy, highly productive, moderately mineralized, and exhibits a weak buffering capacity. In the sediments, only chromium at a single station exceeded the threshold level above which effects on aquatic life are probable. The benthic community in the sampled water bodies is relatively diversified and abundant.

Hydrology and water supply - The major waterway potentially affected by the project is Piché River (which is the same river impacted by previous mining operations); the surface area for the watershed is

194.8 km^2. Based on hydrologic studies, the current watershed supplemented by water from the depleted old underground mine workings has the potential to supply the water needed for the processing plant.

Climate - The climate in the project area is characterized by long, cold winters and relatively short summers. Total precipitation reaches 914 mm. Winds are generally from the south or southwest from June through January, and mainly from the northwest from February through May. Evaporation amounts to 652 mm per year, most of which occurs during the summer season when the water balance experiences an average deficit.

1.7 Tailings and Water Management

During the mining operations, it will be necessary to dispose of and manage 122 Mm^3 of dry, settled tailings, and 161 Mm^3 of waste rock. The proposed tailings sites will be mainly located over the old East Malartic tailings area southeast of the planned pit. The waste rock dump will be located between the tailings area and the open pit, southeast of the urban centre of Malartic. Part of the dump may also be extended over the tailings pond once the tailings will be dried and hardened.

The Canadian Malartic tailings area will cover more than 600 ha. The thickened tailings will be delivered to the Tailings Management Facility (TMF) at approximately 68% solids (w/w) The tailings will be deposited from different central points, either along an axis or on top of a cone, to create a series of cones that will overlap progressively and will be characterized by gentle slopes. The whole area will be developed by sectors using a pre-defined placement sequence. The tailings will be contained at the boundaries of the TMF by starter perimeter berms built with non acid generating waste rock. The external slopes in areas that have reached the final configuration will be covered with a 5 m layer of mine waste rock and a vegetative cover on top of the waste rock. This will allow progressive closure of the TMF and it is planned that 65% of the surface of the tailings will be capped and re-vegetated by the end of mine operations.

Using the footprint of East Malartic as a basis for the future Osisko tailings and waste rock area provides a series of advantages: It would allow the rehabilitation of one of the most important orphan sites in Québec by the placement of non-acid generating tailings on top of acid generating material. It would minimize the footprint of zones impacted by tailings placement in the area of Malartic by concentrating all tailings in one location. It would offer a site located close to the mine site and at an adequate distance from the town, and it would not infringe on non-impacted watersheds.

Thickened tailings are not expected to release significant quantities of water. Water management of a facility using thickened tailings is consequently typically designed to collect and transfer or discharge runoff water. The main element of the water management system of the Canadian Malartic project consists of the construction of a new polishing pond that will be located south of the existing sedimentation and polishing ponds of the old East Malartic property. Mainly, the water management will include the following:

- Runoff from the local watershed – This water will accumulate directly in the new polishing pond.

- Runoff from the existing tailings pond as well as sedimentation and polishing ponds – Water from the tailings pond is currently transferred to the sedimentation pond. Water will be transferred to the new polishing pond by pumping. A separate pumping station will be required for water transfer from the polishing pond to the new polishing pond.

- Water from the underground mines – This water is currently pumped at an annual rate of 1.4 Mm^3. In the future operation, the underground water will be pumped at a rate of 2.84 Mm^3 and will be mostly discharged directly into the new polishing pond and also piped directly to the process plant for fresh water requirements.

The process make-up water requirements will be met with the addition of the fresh water from the underground mines and the water from the natural catchment area of the project.

1.8 Infrastructure and Support Facilities

The Canadian Malartic project is not located in a remote area and the existing infrastructure in the vicinity of the Canadian Malartic project provide major advantages for the implementation and operation of the future open pit operation.

The following infrastructure facilities are already in place to support the project:

- Trans-Canadian highway 117
- Canadian National Railroad line
- Regional Airport facilities
- Hydro-Québec 120kV electrical power grid
- Natural Gas pipeline.

During the construction of the project, the following infrastructure and ancillary facilities will be erected:

- Electrical:
  - 120kV electrical power line (19 km) from Cadillac
  - Main substation
  - Electrical site distribution.
- Building and support facilities:
  - Road construction
  - Municipal site work
  - Administration/Warehouse complex
  - Mine/Truck shop building
  - Control gate
  - Fuel storage
  - Mine explosives plant.

A new 19 km long 120kV electrical power line from the Hydro-Québec 120kV local grid will be built and connected to the main substation adjacent to the mill. The new line will be commissioned by Hydro-Québec in September, 2010. The main substation at the plant facilities will have three 120-13.8kV power transformers each with a capacity of 42/56/70 MVA. The load will be distributed equally amoung the three transformers (aproximately 28 MW each). In case one transformer is taken off line or in case of a failure, two transformers will handle 100% of the load. A 25kV electrical distribution network will be connected to various areas on the site.

A paved road running north-south from the town of Malartic towards Lake Mourier cuts through the central area of the Canadian Malartic property. A control gate building will be erected at the main entrance to supervise the personnel and merchandise transport entering and exiting the site.

In order to complete the project, additional facilities will be required to support the operation and maintenance aspects of the mine and process plant. The major support facilities include the administration/warehouse complex, the mine truck/heavy equipment shops for mobile equipment and a mine explosives plant for storage and preparation of explosives for the mine operation.

The ancillary facilities will be located near the process plant and all facilities will be connected to the same communication network.

1.9 Community Resettlement

A portion of the mineralized zone of the Canadian Malartic project is located underneath the southern portion of the town of Malartic, and for the exploitation of the open pit mine, approximately 205 residences and five institutional buildings and social housing units need to be relocated.

An area to the north of the town was selected for the expansion area. Municipal infrastructure works are presently underway to complete the urban designs (roads and housing lots).

Negotiations have been ongoing to allow for the selection of lots for the owners or the acquisition of their properties by Osisko.

Regarding the various institutional buildings, Osisko has completed discussions with provincial ministries and municipal and regional institutions to define and agree on the final conceptual design. The principal institutional buildings are:

- Elementary School
- Cultural and Recreation Center
- Adult Learning Facility
- Long-Term Care Hospital Center
- Daycare Center
- Social Housing.

The community resettlement has been initiated in advance of the main project in order to involve and gain support of the local community in the project development at an early stage. In addition, the resettlement should be completed before the start of mine pre-production works in Q4 of 2009.

1.10 Capital and Operating Costs

1.10.1 Capital Cost Summary

Capital expenditures (CAPEX) are estimated at $723.3 M plus a provision for contingency of $65.6 M for a total CAPEX of $788.9 M or $146 per recoverable ounce. This places the Canadian Malartic project within current industry norms as one of the best undeveloped gold projects in the world.

CAPEX to completion (CAPEX less sunk costs, less capital lease financing during operation) is estimated at $642.9 M. In order to accelerate the development of the project, Osisko entered into contracts for fabrication of long-lead delivery equipment and initiated work on certain development activities, including the relocation program. As of September 30 2008, Osisko has invested $82.0 M. Furthermore, Osisko negotiated a capital lease financing agreement with CAT Finance for mining equipment for an amount of $83.0 M, which will reduce the capital requirements during the construction phase by a net $64.0 M.

Osisko has entered into commitments on many long-lead items, construction contracts and fixed price quotations for an estimated cost of $286 M. No contingency has accordingly been applied to those amounts. A contingency provision of 15% has been provided on the remaining project outlays. The level of accuracy of the total capital investment estimate is ± 10%.

The capital cost summary is shown in Table 1.6 below.

Table 1.6: Capital Cost Summary

Description	Total Cost ($M)
General administration	14.5
Community Resettlement	87.0
Mining	136.7
Electrical and Communication	19.5
Infrastructure	29.7
Processing	348.0
Tailings and Water Management	15.3
Indirects	72.7
Sub-total	**723.3**
Contingency	65.6
CAPEX Total	**788.9**
Less • Outlays to September 30, 2008	(82.0)
Less • Net Capital Lease for Mining Equipment	(64.0)
CAPEX to completion	**642.9**

The investment program is scheduled over a 2½ year period. Sustaining capital is estimated at $95.0 M and is mainly for additional mining equipment. Closure cost provisions amount to $45.0 M.

Third quarter 2008 market prices for all materials and labor were applied and a CDN/USD exchange rate of 1.18 was used for all Canadian expenditures.

1.10.2 Operating Cost Summary

Mining Costs have been estimated at an average of $1.41/t mined ($3.73/t milled). Fuel price assumption is based on $70 per barrel of oil. The average annual fuel consumption is estimated at 23 million litres. Mine operating costs have been based on a 9.1-year mine life. The average waste to ore stripping ratio over the life of mine is 1.78:1.

Processing costs have been estimated at $4.96/t milled. In addition, $0.09/t milled for transportation and refining has been estimated. Milling operating costs consist mainly of electrical power, mill grinding media, reagents, consumables and labour.

General and administration services are estimated at $0.65/t milled and consist of support services at site as well as municipal taxes and insurance costs.

Total operating costs are estimated at $9.43/t milled as shown in Table 1.7.

Table 1.7: Operating Cost Summary

Description	Cost ($/t milled)
General Administration	0.65
Mining	3.73
Processing	4.96
Transportation/Refining	0.09
Total	**9.43**

1.11 Project Execution Plan

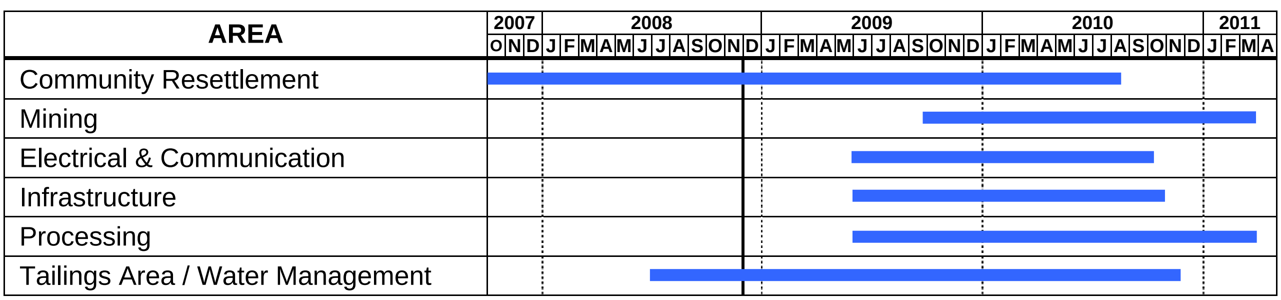
The Canadian Malartic project execution will be directly managed by the Osisko project management team. The engineering, procurement and construction works will be contracted out to qualified firms under the direct supervision of the Osisko project team. Project control functions such as scheduling, cost control, project logistics and site supervision will be executed directly by Osisko personnel.

Major project milestones are presented in Table 1.8 and the project summary schedule is presented in Figure 1.1.

Table 1.8: Major Project Milestones

Major Project Milestones		
Description	**Start Date**	**Completion Date**
Neighbourhood Relocation	October 2007	December 2009
Institutions Relocation	October 2008	March 2010
Project Financing	Q4 2008	Q1 2009
Detail Engineering	June 2008	September 2009
Public Hearings (BAPE)	January 2009	April 2009
Permitting	-	May 2009
Polishing Pond	June 2009	September 2010
Administration/Truckshop Building	June 2009	March 2010
Mill Construction	June 2009	March 2011
120 kV Transmission Line	June 2009	September 2010
Mine Pre-stripping	October 2009	March 2011
Mill Commercial Production	-	April 2011

Figure 1.1: Summary Schedule



1.12 <u>Financial Analysis</u>

The base case for the Canadian Malartic project assumed mining reserves of 6.28 Moz at $775/oz of gold. Silver production is included in the financial model as a by-product. A price of silver of $10/oz has been used.

The Feasibility Study assumed a CDN/USD exchange rate of 1.18 and an oil price of $70 per barrel.

During the 9.1-year mine life, the average gold and silver production is estimated at 591,000 oz per year and 754,000 oz per year respectively. The average operating cash cost is established at $319/oz of gold after royalties and silver credit are applied.

The capital expenditures (CAPEX) are estimated at $788.9 M. CAPEX to completion (CAPEX less investments to September 2008 and less capital lease financing repayments during the operations period) is estimated at $642.9 M.

Capital and operation costs were developed for a nominal 55,000 tpd mine throughput.

Sustaining capital costs total $94.7 M and the mine closure cost is estimated at $45.0 M. Process and mining equipment salvage values are included in the financial calculations and are estimated at $59.9 M. In addition, an initial working capital requirement of $22.7 M has been considered for accounts payable, accounts receivable, production inventory and supplies.

Under the base case scenario, the Internal Rate of Return (IRR) on the CAPEX to completion is estimated at 28.8% before taxes and 25.1% after taxes (un-leveraged).

A summary of the financial highlights is shown in Table 1.9.

Table 1.9: Summary of Financial Highlights

Description	Quantity
Proven and Probable Gold reserves (oz)	6,283,000
Estimated Net Recoverable Gold (oz)	5,397,000
Average Annual Gold Production (oz)	591,000
Average Annual Silver Production (oz)	754,000
Cash Cost per oz	
- before royalties	$320
- with royalties, net of silver revenues	$319
Total Investment (CAPEX)	$789 M
CAPEX per recoverable oz of gold	$146
CAPEX to completion	$643 M
CAPEX to completion per recoverable oz of gold	$119
Sustaining Capital	$95 M
Closure Costs	$45 M
Operating Cash Flow Pre-tax	$2,463 M
IRR - Pre-tax (CAPEX to completion)	28.8%
IRR - After-tax (CAPEX to completion)	25.1%
Payback (Total project)	36 months
Payback (CAPEX to completion)	32 months
Mine Life	9.1 years

On a CAPEX to completion basis, project sensitivity to gold price, direct cash cost, process recovery, capital costs and exchange rate is shown in Tables 1.10 to 1.14.

Table 1.10: Gold Price Sensitivity

CAPEX to Completion						
Gold Price ($/oz)	**After-tax IRR**	**After-tax NPV 5% ($ M)**	**After-tax NPV 0% ($ M)**	**Pre-tax IRR**	**Pre-tax NPV 5% ($ M)**	**Pre-tax NPV 0% ($ M)**
650	16.6%	399.4	768.2	19.0%	536.3	990.7
700	20.1%	532.4	954.1	23.1%	722.4	1,256.5
750	23.5%	664.8	1,140.0	27.0%	908.4	1,522.3
775	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
800	26.7%	796.6	1,325.9	30.7%	1,094.4	1,788.1
850	29.7%	928.4	1,511.8	34.2%	1,280.5	2,053.9
900	32.6%	1,059.4	1,697.7	37.5%	1,466.5	2,319.7

Table 1.11: Direct Cash Cost Sensitivity

CAPEX to Completion						
Cash Costs ($/oz)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
272	28.1%	859.3	1,414.5	32.3%	1,182.8	1,914.4
288	27.1%	816.4	1,354.0	31.2%	1,122.3	1,828.0
320	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
352	23.0%	644.9	1,111.9	26.4%	880.5	1,482.4
368	22.0%	602.0	1,051.4	25.2%	820.1	1,396.0

Table 1.12: Process Recovery Sensitivity

CAPEX to Completion						
Recovery	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
0.790	20.9%	562.0	995.7	24.0%	764.1	1,316.4
0.816	22.5%	625.4	1,084.7	25.8%	853.1	1,443.4
0.859	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
0.902	27.6%	835.9	1,381.3	31.7%	1,149.7	1,867.0
0.928	29.1%	899.1	1,470.3	33.4%	1,238.7	1,994.0

Table 1.13: Initial Capital Expenditures Sensitivity

CAPEX to Completion						
CAPEX to Completion ($ M)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
578.61	28.4%	779.8	1,282.5	32.5%	1,066.2	1,725.9
610.76	26.7%	755.2	1,257.7	30.6%	1,033.8	1,690.5
642.90	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
675.05	23.6%	706.2	1,208.2	27.2%	969.0	1,619.9
707.19	22.3%	681.6	1,183.5	25.7%	936.7	1,584.5
739.34	21.1%	657.0	1,158.7	24.4%	904.3	1,549.2

Table 1.14: Exchange Rate Sensitivity

CAPEX to Completion						
Exchange Rate (C$/US$)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
1.00	19.7%	573.6	1,034.3	22.6%	783.3	1,368.1
1.10	22.8%	666.6	1,151.5	26.2%	912.0	1,537.4
1.18	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
1.25	26.9%	777.8	1,292.5	30.9%	1,067.1	1,741.3
1.30	28.2%	808.0	1,330.7	32.3%	1,109.1	1,796.5

The results indicate that the project financial return is most sensitive to gold price and gold recovery (revenues) and less sensitive to initial capital cost, operating cost and exchange rate. The prices of silver and crude oil have a limited affect.

1.13 Conclusions and Recommendations

The pit design contains a proven and probable reserve of 183.3 Mt of ore at a head grade of 1.07 g/t Au for an in-situ gold content of 6.28 Moz. Given the important quantities of low grade mineralization and a favorable waste to ore strip ratio (1.78:1), it was determined that a high production rate of 55,000 tpd would result in a financially viable and robust project.

The results of the Feasibility Study indicate that the Canadian Malartic project warrants proceeding to implementation stage of the project. It does not present significant technical difficulties and economic indicators resulting from this study are very positive. Capital expenditures (CAPEX) are estimated at $788.9 M or $146 per recoverable ounce placing the Canadian Malartic project within current industry norms as one of the best undeveloped gold projects in the world.

Based on the CAPEX to completion scenario ($642.9 M), the internal pre-tax rate of rate (IRR) is estimated at 28.8% and the pre-tax NPV (discount 5%) is estimated at $1,001 M. Results of the Feasibility Study demonstrate that the Canadian Malartic project is robust in the current market.

BBA concludes that Osisko should advance the project to the final stage of implementation including the following recommendations:

- Secure the project financing
- Complete the relocation program by the end of 2009
- Secure required permits and authorizations from Government and regulatory agencies
- Continue the detail engineering program
- Move to the construction phase following environmental permitting.

2.0 GENERAL INFORMATION

2.1 Introduction

2.1.1 Scope of Work

The Feasibilty Study Technical Report of the Canadian Malartic project (Feasibility Study) has been prepared at the request of Osisko Mining Corporation (Osisko). The Feasibility Study was prepared and compiled by BBA in a collaborative effort between BBA and Osisko together with a number of specialized consultants. This Technical Report was prepared according to the guidelines set out under “Form 43-101F1 Technical Report” of National Instrument 43-101 Standards of Disclosure for Mineral Projects adopted pursuant to Section 143 of the Securities Act (Ontario).

The scope of work for BBA while writing the Feasibility Study Report included the following:

- Preparation of a final process plant design with complete operating and capital cost estimates and metallurgical test program description.
- Preparation of project area descriptions.
- Review and provide comments for the mining operation section.
- Gather, review and combine all necessary information, data and documents for the completion of this Feasibility Study.

2.1.2 Qualified Persons and Personnel

The following companies and the qualified persons contributing to this report are listed below:

- BBA Inc. - David Runnels, Eng.
- Genivar - André-Martin Bouchard, Eng.
- Micon International Limited - B. Terrence Hennessey, P. Geo.
- G Mining Services Inc. - Louis-Pierre Gignac, Eng.
- Golder Associates - Michel Julien, Eng.
- Belzile Solutions Inc. - Elzéar Belzile, Eng.

The respective certificates for people listed as Qualified Persons may be found in Section 18.0 – Qualification Certificates. Refer to Table 2.1 for a detailed list of qualified persons and their respective sections of responsibility.

Table 2.1: List of Detailed Scope for Each Qualified Person

Section	Title of Section	Qualified Person
1.0 to 1.13	Executive Summary	David Runnels, BBA
2.0 to 2.3.4	General Information	David Runnels, BBA
2.4 to 2.4.5	History	B. Terrence Hennessey, Micon
3.0 to 3.5.5	Geology and Exploration	B. Terrence Hennessey, Micon
3.6 to 3.6.7	Data Verification and Summary	B. Terrence Hennessey, Micon Elzéar Belzile, Belzile Solutions
4.0 to 4.11	Mineral Resource and Summary	Elzéar Belzile, Belzile Solutions
5.0 to 5.8	Mining and Mineral Reserves	Louis-Pierre Gignac, G Mining Services
6.0 to 6.2.4	Mineral Processing and Metallurgical Testing	David Runnels, BBA
7.0 to 7.3.3	Tailings and Water Management	Michel Julien, Golder
8.0 to 8.12.3.4	Infrastructure and Support Facilities	David Runnels, BBA
9.0 to 9.6	Community Resettlement	David Runnels, BBA
10.0 to 10.10	Environmental Considerations	André-Martin Bouchard, Genivar
11.0 to 11.8.6	Community Considerations	André-Martin Bouchard, Genivar
12.0 to 12.4	Project Organization and Execution	David Runnels, BBA
13.0 to 13.4	Capital Cost Estimate	David Runnels, BBA
14.0 to 14.4	Operation Cost Estimate	David Runnels, BBA
15.0 to 15.12	Financial Analysis	David Runnels, BBA Louis-Pierre Gignac, G Mining Services
16.0	Conclusions and Recommendations	David Runnels, BBA
17.0	References	David Runnels, BBA
18.0	Qualification Certificates	David Runnels, BBA

2.1.3 Terms of Reference

This report has been prepared in compliance with National Instrument 43-101 respecting the Standards of Disclosure for Mineral Projects. In addition, the Resource and Reserve definitions are as set forth in Canadian Institute of Mining, Metallurgy and Petroleum, CIM Standards on Mineral Resources and Mineral Reserves - Definitions and Guidelines adopted by CIM Counsel on December 11, 2005.

The units of measure (volume, distance, etc.) used in this report, unless otherwise noted, are in metric. The currency used for all costs is presented in United States Dollars, unless specified otherwise. When costs were derived in Canadian dollars, a conversion rate of 1.18 Canadian dollars to 1.00 US dollars was used.

Table 2.2: List of Main Abbreviations

Abbreviation	Full Description	Abbreviation	Full Description
Barrick	Barrick Gold Corporation	LCM	Loose cubic meters
BBA	BBA Inc.	LG 3D	Lerchs-Grossman 3D algorithm
BCM	Bank cubic metres	M	millions
BMWI	Ball mill work index	m	metres
BWI	Bond work index	m^2	square metres
°	degrees	m^3	cubic metres
˚C	degree centigrade (celsius)	McWatters	McWatters Mining Inc.
CIM	Canadian Institute of Mining, Metallurgy and Petroleum	mm	millimetres
cm	centimetres	Micon	Micon International Limited
Chemex	ALS Chemex Laboratory	MRNF	Ministère des Ressources Naturelles et de la Faune
CoG	cut-off grade	MSDEP	Ministry of Sustainable Development, Environmental Parks
Cygnus	Cygnus Consulting Inc.	NSR	Net Smelter Return
dBA	average decibel	NPV	net present value
dB	decibel	Osisko	Osisko Mining Corporation
G Mining	G Mining Services Inc.	oz	troy ounce (31.1035 grams)
Genivar	GENIVAR	ppm	parts per million
Au	gold	ppb	parts per billion
Golder	Golder Associates	PGE	Precious metal group elements
g	grams	ROM	run of mine
g/t	grams per tonne	RQD	rock quality designation
ha	hectares (10,000 m^2)	RSG	RSG Global Consulting Ltd.
hp	horsepower	SG	specific gravity
h	hours	SGS	SGS Lakefield
IRR	internal rate of return	S/R	stripping ratio
kg	kilograms	t (or tonnes)	metric tonne (1000 kg)
km	kilometres	t/d or tpd	tonnes per day
km^2	square kilometres	t/h	tonnes per hour
km/h	kilometres per hour	t/m^3	tonnes per cubic metre
kPa	kilopascals	t/m	tonnes per metre
kt	1000 tonnes	t/y	tonnes per year
kV	kilovolts	URSTM	Unité de Recherche et de Service en Technologie Minérale
kW	kilowatts	$	United States dollar
Lac Minerals	Lac Minerals Ltd.	y	year
l	litres		

2.1.4 Source Documents

The major source of information used for this Feasibility Study Report comes from the Technical Report by Micon, dated October 23, 2008. Additional information used includes reports from studies provided to Osisko by other consultants such as Golder, WJP Pennstrom Consulting, SGS and G Mining. All other supplementary information used for this study consists of internal reports and studies generated by BBA or Osisko.

2.1.5 Disclaimer and Reliance on Other Experts

The lead author of this report, Mr. David Runnels, is not qualified to comment on issues related to mining and exploration titles and land tenure, royalties, permitting and legal and environmental matters. The lead author has accordingly relied upon the representations of experts employed by the issuer, Osisko, for Sections 1.1.1 and 2.2 of this report and has not verified the information presented in those sections. In addition, there is no assurance that the project implementation will be realized.

The comments in the Report reflect BBA's best judgment in light of the information available to it at the time of preparation. BBA reserves the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to BBA subsequent to the date of this report.

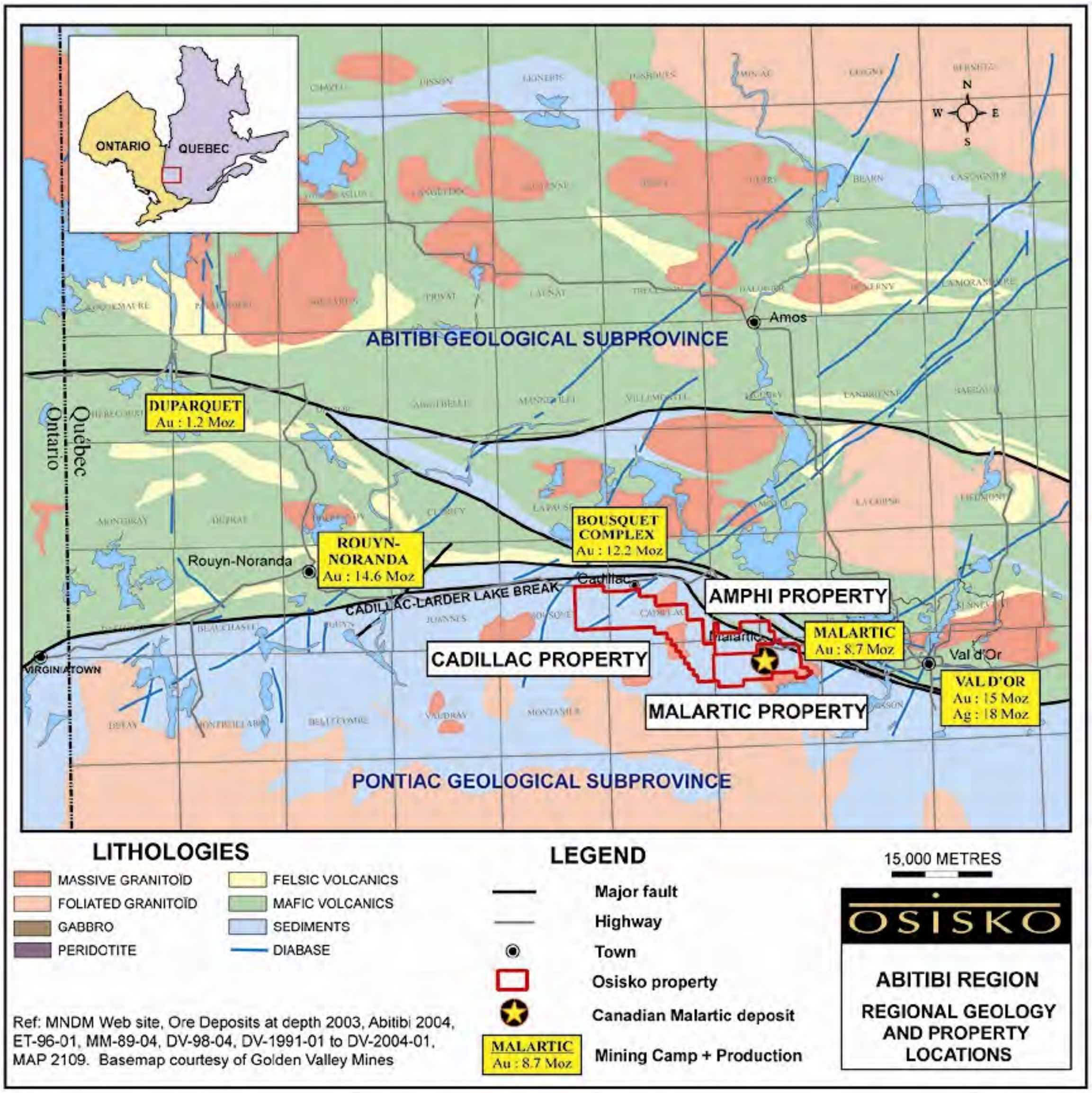
Use of this Report acknowledges acceptance of the foregoing conditions.

2.2 Property Description and Location

2.2.1 Property Location

The Canadian Malartic property is located in the Abitibi region of north-western Québec in National Topographic System (NTS) sheet 32D/01. It lies entirely within the Fournière Township, immediately south of the town of Malartic, about 25 km west of Val-d'Or, Québec (Figure 2.1) and around 550 km northwest of Montreal, Québec . The property is of roughly rectangular shape, extending about 13 km east-west and around 4 km north-south, approximately centered at UTM grid coordinates 713000 E and 5333000 N (Latitude 48º 7' 45" N and Longitude 78º 7' 00'' W). The property also covers the southern portion of the town itself.

Figure 2.1: Property Location



2.2.2 Property Ownership and Agreements

The Canadian Malartic property is comprised of 127 contiguous mining titles, including 114 Map Designated Claims (CDC), 12 claims (CL) and one Mining Concession (CM) covering a total of 5,654.1 ha (Figure 2.2). A small area towards the western end of the Canadian Malartic property is covered by two claims forming a square block with a total area of 23.98 ha. One of these claims is owned by an independent prospector, while the other has lapsed and has been map-staked by an unknown third party.

The Canadian Malartic property was acquired by Osisko in stages between 2004 and 2006. The majority of the mining titles of the Canadian Malartic property were map-staked by Osisko or its appointed intermediaries, and is not subject to any encumbrances. Others were purchased outright from independent parties, without royalties or other obligations. Of the 127 mining titles comprising the Canadian Malartic property, 21 are subject to agreements presented in Table 2.3 below:

Table 2.3: Mining Claims Agreements and Encumbrances

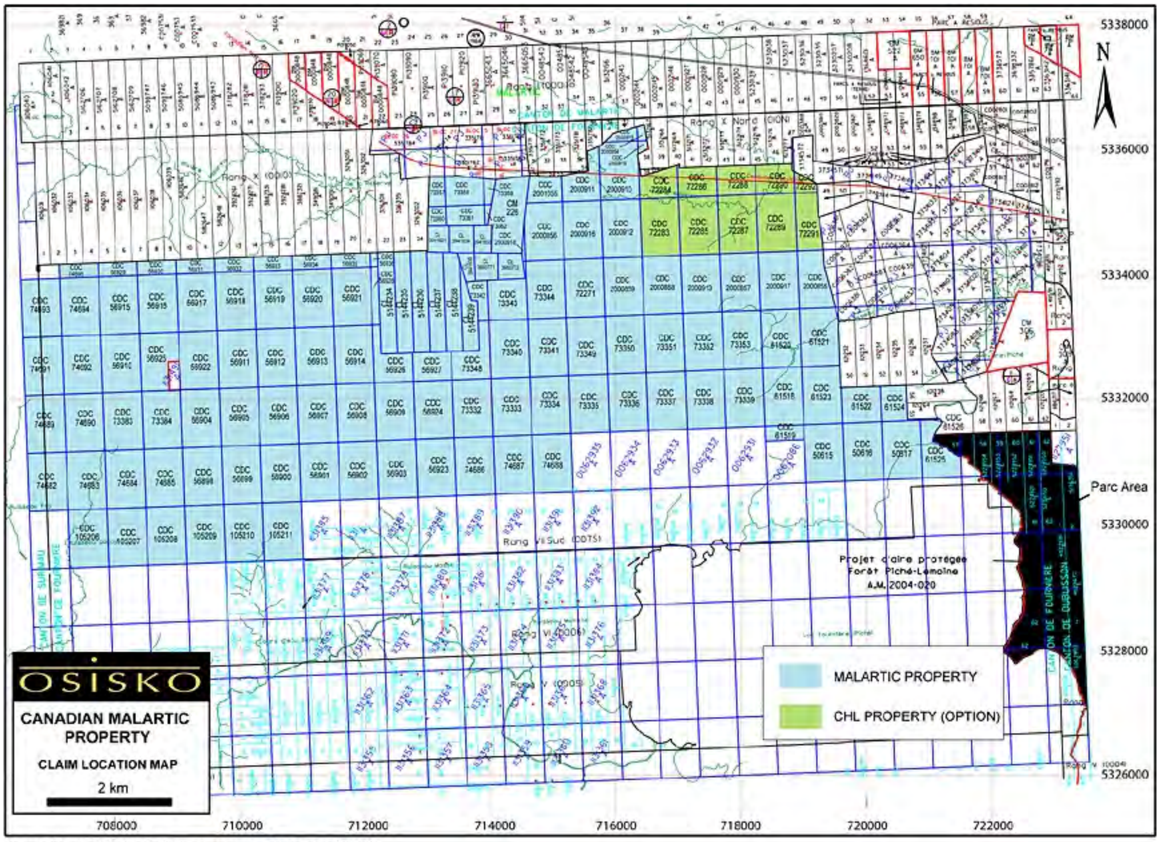
Mining Titles	Agreements and Encumbrances
CM 226 CL 3941621 CL 3941633 CL 3941634 CL 3941635 CL 3950771 CL 3950772	Mining Titles 100% owned by Osisko. Claims purchased from McWatters Mining Inc. liquidating trustee in consideration of a cash payment (paid). Titles are subject to a sliding 2% - 3% Net Smelter Royalty payable to Royal Gold. The royalty rate is tied to the price of gold, with the higher rate taking effect if the gold price is greater than 350 $/oz. Half of the royalty can be purchased back by Osisko for CDN$1,500,000.
CL 5144234 CL 5144235 CL 5144236 CL 5144237 CL 5144238 CL 5144239	Mining Titles 100% owned by Osisko. Claims acquired from Dianor Resources Inc. and subsidiary Threegold Resources Inc. for cash and shares (paid in full). Claims are subject to a 2% Net Smelter Royalty payable to a private individual. The entire royalty may be purchased back by Osisko for CDN$2,000,000.
CDC 72271	Mining Title 100% owned by Osisko. Claim purchased from Golden Valley Mines for a cash consideration (paid). Claim is subject to a 2% Net Smelter Royalty payable to Golden Valley Mines.
CDC 2000854 CDC 2000855 CDC 2000856 CDC 2000857 CDC 2000858 CDC 2000859	Mining Titles 100% owned by Osisko. Claims purchased from a private individual in consideration of a cash payment (paid). Claims are subject to a 2.5% Gross Overriding Metal Royalty.
CDC 2001055	Mining Title 100% owned by Osisko. Claim purchased from a private individual in consideration of a cash payment (paid). Claim is subject to a 2.5% Gross Overriding Metal Royalty.

2.2.3 Adjacent Properties

Exploration rights immediately north of the Canadian Malartic property are owned by Osisko under a separate property (East Amphi) and by Niogold Mining Corp. Rights to the east are owned by Golden Valley Mines Ltd. (Golden Valley Mines), to which Osisko has an option to earn a 70% interest, and by

Northern Star Mining Corp. Rights to the south of the Canadian Malartic property are owned by an individual prospector and C2C Inc., whereas rights to the west are owned by Osisko under a separate property (Cadillac).

Figure 2.2: Plan Displaying the Malartic Property Mining Claims



Source: Quebec Ministry of Natural resources Claim Map 32 D/01

2.2.4 Mineral, Surface and Water Rights

A detailed list of the mining titles of the Canadian Malartic property, including claim numbers, size, expiration dates, work expenditure requirements, renewal fees and exploration credits has been compiled. At the time of finalization of the Feasibility Study report, all mining titles were registered with 100% ownership to Osisko, and were in good standing. The limits of the various mining titles are not physically defined in the field. The majority of the Canadian Malartic property titles are Map Designated Claims (CDCs). CDCs are pre-defined cells conforming to a system of electronic map-staking established by the Québec Government in 2000. The map-designated coordinates of the cells are the legal limits of the said claims. The physical limits of each claim can be established by surveying and positioning the map-designated coordinates of the claims in the field. Twelve claims on the Canadian Malartic property were originally acquired by field-staking, but have since been surveyed by the Government and incorporated into the electronic map-staking system. The boundaries of the Mining Concession have been similarly surveyed and recorded. Detailed positions of CDC, claims and other mining titles for most of the NTS sheets in Québec are available in PDF format maps at the following web site of the MNRW: ftp://ftp.mrnfp.gouv.qc.ca/public/gestim.

Each claim provides a right of access, though not surface rights, to a designated parcel of land on which exploration work may be undertaken. Access to land that has been granted, alienated or leased by the Crown for non-mining purposes requires the permission of the current rights-holder. Additionally, claims that lie within town boundaries or lands identified as State reserves may be subject to further conditions and obligations concerning the work to be performed on the claim.

A mining concession is initially granted for a twenty year period, with the possibility of renewal for additional ten year periods. There is no obligation or work requirement needed to maintain the concession other than the payment of an annual fee based on the size of the concession. The mining concession provides some surface rights to the owner, limited to those necessary for mining activities.

2.2.5 Royalties

Provided in Section 2.2.2 – Property Ownership and Agreements.

2.2.6 Environmental Exposures

There are no environmental liabilities, obligations or responsibilities associated with the Canadian Malartic property, other than the adherence to the regulations of the MSDEP concerning exploration activities. Several non-remediated tailings ponds, particularly those of the past-producing East Malartic mine, occur on the property but as long as these are covered by exploration rights (CDC's), the environmental liabilities related to these ponds are the responsibility of the MRNF.

2.2.7 Civil Responsibilities

The mining activities at the past producing original Canadian Malartic, Sladen and East Malartic mines resulted in a connected network of open and/or backfilled stopes and other underground workings, which are now flooded. Two pumps at the East Malartic mine shaft are operated by Osisko under the supervision and responsibility of the MRNF in order to control the local water table.

Portions of the southern limits of Malartic were built over the original Canadian Malartic mine, and a number of houses are located over open stopes. In certain cases, the remaining crown pillars are as little as ten metres thick, posing an ever-present risk of collapse. The highest risk areas were expropriated and fenced off by the Québec Government in 1981, with 11 homes being relocated to other parts of the town. In 2000, Barrick expanded the fencing and built additional fences around areas determined as potentially dangerous. As the current owner of the Canadian Malartic property, it is incumbent upon Osisko to maintain the fences in good condition. The fence perimeters must be cleared of brush, and the fences need to be clearly marked with signs indicating the danger within the enclosed zones.

Community Development (CD) work has progressed since 2006 with the mission to provide Osisko and the town of Malartic with a sustainable development initiative for the relocation of a total of 205 residences and 5 institutional buildings from the vicinity of the planned open pit. The CD program mission can be defined as the information, education and involvement of the community on one hand, and the inventory, planning as well as the relocation and construction of real estate assets on the other, to create the new housing subdivision in Malartic. This is further discussed in more detail in Section 9.

2.2.8 Environmental Approvals and Permits

The environmental approval process is well advanced and the Environmental Impact Assessment (EIA) has been submitted for government approval. Following the public hearings scheduled for the 1st quarter of 2009, the final approval is expected in May 2009.

Requisite permissions for use of water and incidental logging for drill-access have been acquired from various government agencies such as the MRNF and the MSDEP.

2.3 Accessibility, Local Resources and Infrastructure

2.3.1 Accessibility

The northern extents of the Canadian Malartic property can be accessed directly from the Trans Canadian Highway 117. A paved road running north-south from the town of Malartic towards Lake Mourier cuts through the central area of the Canadian Malartic property. The Canadian Malartic property is further accessible by a series of logging roads and trails, as well as a network of gravel roads associated with the past producing mines. Malartic is also serviced by a rail-line which cuts through the middle of the town. The nearest large airport is located in Val-d’Or, about 25 km east of Malartic.

2.3.2 Climate

The climatological information concerning the temperature and precipitation presented in this report is based on data collected at the Val-d'Or meteorological station between 1970 and 2001, as reported by the Centre de Ressources en Impacts et Adaptation au Climat et à ses Changements (CRIACC). Data on wind velocity and direction is based on records from 1961 to 1991.

Mean annual temperature for the Val-d'Or/Malartic area is 1.2°C, with average daily temperatures ranging from -17.2°C in January to 17.2°C in July. The average total annual precipitation is 914 mm, peaking in September (102 mm) and at a minimum in February (40.5 mm). Snow falls between October and May, with most occurring between November and March. Peak snowfall occurs in December, averaging 610 mm, equivalent to 54 mm of water. Winds are generally from the south or southwest from June through January, and mainly from the northwest from February through May. Average wind velocities are in the order of 11 to 14 km/h.

2.3.3 Local Resources and Infrastructure

The Canadian Malartic property is located in the southern portion of the town of Malartic. The town has a population of around 3,500 people and hosts a variety of commercial establishments, including motels, restaurants, service suppliers, retailers and a community health clinic, as well as elementary and high schools. The city of Val-d'Or, some 25 km east of Malartic, hosts a large number of manufacturers and suppliers who serve the mining industry.

Mining-related activities ceased in 2002, when the East Malartic mill, used by Barrick to process ore from their Bousquet Mine was shut down and left on a care and maintenance basis.

Osisko is currently using the East Malartic administrative complex as an exploration office, with portions of the facility dedicated to equipment storage and core logging facilities.

The town's principal employer, the Domtar Sawmill, closed in 2006, resulting in a pool of local manpower trained in heavy equipment/industrial operations. Skilled workers may also be available from the areas within an approximate 25 km radius of Malartic, specifically Cadillac to the west and Val-d'Or to the east, where a number of mines are still in operation. It should be noted, however, that the recent increase in mining activity in the Abitibi region may result in a temporary shortage of certain classes of experienced mining personnel.

2.3.4 Physiography

The Canadian Malartic property is situated in the Abitibi lowlands and is relatively flat, consisting of plains with a few small hills. The topographic relief on the Canadian Malartic property is subtle, with a difference of about 95 m between maximum and minimum elevations. Most of the area is sparsely wooded with secondary growth black spruce, larch and birch as the dominant species. The central, east-central and west-central parts of the Canadian Malartic property are cut by a number of small streams, generally oriented east-west and connecting bogs or swampy areas. The south eastern extremity of the Canadian Malartic property partially overlaps onto Lake Fournière, which covers about 28 km^2.

Overburden is characteristically a thin layer of till, typically only a few metres thick, with local development of organic-rich boggy material. Outcropping exposures are relatively rare, generally restricted to localized zones in which the lithology is silica-altered and more resistant to erosion.

2.4 History

2.4.1 Prior and Current Ownership

Gold was first discovered in the Malartic area in 1923 by the Gouldie Brothers at what is now designated the Gouldie Zone. In 1925, a new showing was discovered and staked by an Ottawa-based prospecting syndicate, located about 1.6 km northwest of the Gouldie prospect. This property was sold to the newly-incorporated Malartic Gold Mines in 1927. Malartic Gold Mines undertook drilling, trenching and limited underground development on the deposit until 1929, when the project was suspended with the onset of the Great Depression.

In 1933, the original Canadian Malartic Gold Mines Ltd. took possession of the Malartic Mines property as well as the claims covering the Gouldie prospect. Production at the original Canadian Malartic Mine began in 1935 and continued uninterrupted until 1965. The original Canadian Malartic success prompted additional exploration, discovery and development immediately to the east. The Barnat/Sladen Mines and East Malartic Mine independently went into production in 1938 and continued with only minor interruptions until 1970 and 1983, respectively.

In 1964, Falconbridge Nickel Ltd. purchased the original Canadian Malartic Mine and, following cessation of gold production in 1965, refurbished the mill to process nickel ore from its Marbridge Mine. These operations ceased in 1968, after which the original Canadian Malartic mill was decommissioned and removed.

In 1974, the mining titles covering a portion of the historic Canadian Malartic holdings were purchased by East Malartic Gold Mines. The rest of the gold camp, covering the balance of the original Canadian Malartic ground, as well as the past-producing Barnat/Sladen and East Malartic Mines, was acquired in 1979 by Long Lac Exploration Ltd. These two companies, as well as a third Ontario-based company, merged in 1982 to form Lac Minerals, which continued to explore the property over the next decade with the objective of defining a near-surface gold resource amenable to open-pit mining methods.

Control of the property fell to Barrick in 1994 when it acquired Lac Minerals. Barrick did not explore the property but completed a number of environmental and stope-stability studies during the 1990's. Barrick's principal activity was to process pyrite-rich ore from its Bousquet Mine at the East Malartic mill, which lasted until 2002 and resulted in the production of acid-generating mill tailings. Barrick sold all of its interests in the Malartic camp, including environmental and reclamation liabilities, to McWatters in February, 2003.

In November 2004, Osisko, through an intermediary, purchased a 100% interest in six claims and one CM (Mining Concession) covering the past-producing original Canadian Malartic Mine. The mining titles were purchased from the McWatters liquidation trustee, following the bankruptcy of McWatters earlier in 2004. A sliding 2% to 3% Net Smelter Royalty is payable to Royal Gold for these titles, half of which can be purchased for CDN$1.5 M. The titles have since been transferred, and are registered with 100% ownership in favour of Osisko. Osisko elected not to purchase the CM covering the past producing Barnat, Sladen and East Malartic Mines from the liquidation trustee, due to concerns over acquired environmental liabilities. Control of this portion of the property was assumed by the Québec Government (MRNF) in December, 2004, after the liquidation trustee failed to find a buyer.

On December 29, 2004, Osisko announced the signing of a letter of intent with Dianor and its wholly-owned subsidiary Threegold to acquire a 100% interest in a block of six claims to the southwest of, and contiguous with, the property purchased from the McWatters trustee. These claims are subject to a 2% Net Smelter Royalty payable to a private individual, but the royalty may be purchased for CDN$2.0 M. Official documents for the transfer of these claims were filed on December 29, 2005, and the claims are now registered with 100% ownership in favour of Osisko.

Between February and June, 2005, 92 additional claims were staked by Osisko or its appointed intermediaries, surrounding the original block of seven mining titles and the Dianor block. In December, 2005, Osisko staked six more claims along the southern margin of the property. The transfer of these claims has been processed, and all are now registered with 100% ownership in favour of Osisko.

On February 3, 2006, Osisko announced the signing of a letter of intent with Golden Valley Mines to purchase a 100% interest in a single claim contiguous to the property. The claim is subject to a 2% Net Smelter Royalty payable to Golden Valley Mines. The finalization of the agreement was announced on June 21, 2006. The transfer of this claim has been processed and is now registered with 100% ownership in favour of Osisko.

In late 2005, the Québec Government cancelled the CM and claims covering the portion of the McWatters property that was transferred from the McWatters liquidation trustee and converted the area to 16 CDCs. The conversion of mining titles to CDCs effectively freed any eventual owner of the titles of the associated environmental liabilities and encumbrances.

The claims were made available through the Government electronic map staking system, and eight separate parties simultaneously submitted applications for the titles. The ownership situation was resolved by a claim-by-claim lottery conducted on February 15, 2006. Osisko succeeded in acquiring two of the claims at the lottery. On March 2, 2006, Osisko announced that it had signed letters of intent with a group of four independent parties to purchase 100% interest in the remaining 14 titles. Seven of these titles were purchased outright from two individuals, without additional encumbrance. The remaining seven claims were purchased from two other individuals and are subject to a 2.5% Gross Overriding Royalty. The transfer documents for these claims have been processed, and all are now registered with 100% ownership in favour of Osisko.

2.4.2 <u>Exploration History</u>

The past-producing Canadian Malartic, Barnat/Sladen and East Malartic gold mines located on Osisko's property went into production between 1935 and 1938, and ceased production in 1965, 1970 and 1983, respectively. Relatively little exploration work was done before development began on those deposits and, during mining operations, essentially all reports of geological work, drilling, development and production were internal, unpublished documents. The collective archives of the past-producing mines were acquired by Lac Minerals at the time it took control of the property, and stored in the administrative offices of the East Malartic Mine. The mine office and the archives fell under the control of the Québec Government (MRNF) when it assumed responsibility for that portion of the property from the McWatters liquidation trustee.

The first geological maps of the Malartic area (Fournière Township) were produced by James and Madsley (1925) of the Geological Survey of Canada. Geological reports including detailed mapping in the area of the Canadian Malartic Mine were produced in 1928 by Gill and Murdoch and in 1929 by Keading (unpublished internal reports of Canadian Malartic Mines Ltd.). Québec Mines Service geologist O'Neill (1935) remapped the Canadian Malartic property at a detailed scale, and provided the first petrographic descriptions of the mineralized rocks. Derry (1939) and, Derry and Herz (1948) published papers detailing the structure, alteration as well as a metallogenic model for the Canadian Malartic gold deposit.

Geoscientific reports on the Malartic gold camp have been published by Gunning and Ambrose (1940), Eakins (1962), Sansfaçon (1986a, 1986b), Grant et al. (1987), Sansfaçon and Hubert (1990), Trudel and Sauvé (1992) and Fallara et al. (2000). Other geological information specifically pertinent to the Canadian Malartic deposit is available in reports by the Geological Survey of Canada (Descarreaux, 1978), the Québec Mines Service (Dresser and Denis, 1951) as well as by the Ministry of Energy and Resources of Québec (Latulippe, 1963, 1967 and 1976; Germain, 1983).

After the closing of the East Malartic Mine, Lac Minerals continued exploration work on the property, including drilling of approximately 500 surface drill holes on or near the Canadian Malartic deposit in various campaigns dating from 1981 to 1985. Several other drill campaigns were completed on the Barnat/Sladen and East Malartic portions of the property until 1990 when Lac Minerals discontinued exploration on the property. Most of the drill data generated by Lac Minerals was filed for assessment with the Québec Government and is publicly available.

Lac Minerals undertook limited ground geophysical surveys on the property between 1980 and 1983. The first geophysical survey entailed an induced polarization (IP) survey over a single line transecting known structures hosting disseminated pyrite-gold mineralization, yielding inconclusive results. In 1981, magnetic-HLEM (horizontal loop electro-magnetic) surveys were performed over 66 km of cut-lines, in an attempt to define the geological contacts of various lithologies and mineralized zones, but again yielded poor results. Similar inconclusive results were generated that same year by a VLF (very low frequency EM) survey over the same lines. In 1982, IP surveys were conducted over 12.8 km of line over other areas of known mineralization peripheral to the main Canadian Malartic deposit. Five priority anomalies detected in this survey were tested by drilling or trenching. The results were negative and the source of the IP anomalies was not determined. In 1993, a new IP technique was employed to survey an area of known mineralization near the western end of the Canadian Malartic deposit. Four localized zones of heightened geophysical response were identified within the surveyed sector, but could not be correlated to any particular source.

Given the poor response of the various geophysical survey techniques, Lac Minerals targeted its exploration drilling program based on results of historic drilling, underground development and surface geological mapping. This approach led to the discovery of a new mineralized zone (the Charlie Zone), located under the tailings to the south of the Sladen mine.

During the time Barrick owned the property (1994 - 2003), no exploration work was done. Efforts focused on partial recompilation of historical data for resource estimate purposes, and on stope stability and environmental assessment. Barrick drilled a limited number of geotechnical holes to determine the thickness and stability of crown pillars of the Canadian Malartic Mine, in the area underlying houses in the southern part of the town of Malartic. After the 2003 acquisition, there is no public record of McWatters performing any exploration work on the property.

2.4.3 Historic Drilling

The vast majority of historic drilling on the property was undertaken by the past-producing Canadian Malartic, Barnat/Sladen and East Malartic gold mines during development and production. Drill records for these operations were mostly internal, unpublished documents. A subset of these historic archives has been compiled by Osisko, pertaining to the ground covered by the Canadian Malartic Mine.

Two distinct phases of historical drilling have occurred on the Canadian Malartic deposit. During the first phase, from 1928 to 1963 by Canadian Malartic Mines Ltd., records indicate that over 5,000 surface and underground holes were drilled on this portion of the property. These holes were predominantly drilled from underground as grade control drilling. The surviving archives only include data for about 4,000 of these holes (Canadian Malartic S-series and U series holes), from which a total of 3,838 drill holes (159,056 m of drilling) were included in Osisko's digital database. The remainder was discarded as data were incomplete, illegible or had unreliable collar location information. There are no descriptions of the drill procedures, equipment employed, core diameter or drilling quality available in the documents. Orientation data on drill holes are limited to sporadic acid tests for dip. Data for drilling in the areas of the past-producing Barnat/Sladen and East Malartic Mines have been compiled by Osisko for inclusion in the resource update.

Lac Minerals drilled approximately 502 surface holes (43,495 m of drilling) on the Canadian Malartic property between 1981 and 1985. Drill logs indicate the core was BQ diameter, but information pertaining to drilling procedures, drill equipment is not available. Orientation data on drill holes are limited to sporadic acid tests for dip and rare measurements of azimuth and dip using unspecified instrumentation.

2.4.4 Production History

The Canadian Malartic property includes four past-producing gold mines. Three of these, the Canadian Malartic, Sladen and East Malartic Mines, are portions of a 3,000 m-long, continuous mineralized system, exploited from west to east respectively. The Barnat Mine is part of the Malartic gold camp but is considered a separate deposit located within the Cadillac Fault Zone. During the period from 1935 to 1983, these mines produced a total of 159,451 kg (5,126,462 oz) of gold, mostly from underground operations. Three small open pits (Buckshot and Mammoth zones) were excavated at the Barnat and East Malartic Mines, to recover mineralization from crown pillars after the backfilling of underground stopes. The production figures reported here were obtained from Grant et al. (1987), Sansfaçon et al. (1986, 1990) or Trudel and Sauvé (1992).

The Canadian Malartic Mine operated between 1935 and 1965. The deposit was mined mostly by underground long-hole stoping methods, making it the only underground bulk tonnage gold mine in Québec. Mining was limited to higher grade (greater than 3 g/t Au) mineralized zones within a larger, lower grade mineralized envelope, along nine levels extending to a depth of approximately 350 m. Development continued along four additional levels (to level 13) but there is no evidence of production at these deeper levels. A total of 9,931,376 tonnes of ore at an average grade of 3.37 g/t Au were extracted, for an aggregate production of 33,468.3 kg of gold (1.076 Moz Au). Mineralization occurs as finely disseminated native gold within altered sediments and porphyry, and was recovered by standard milling and cyanide-leaching techniques with an 89.4% average recovery reported over the mine life.

The ore from the Canadian Malartic Mine was also anomalously rich in silver relative to the rest of the Malartic gold camp, with gold to silver ratios ranging from about 4:1 to 1:1. Total silver output was approximately 20,000 kg (643,000 oz).

The Barnat/Sladen Mine comprised several ore bodies. The Barnat Mine worked at least three separate ore zones located in tectonized porphyry/diorite masses within the Cadillac Tectonic Zone, while the Sladen Mine, located south of the fault zone, operated within the Pontiac Subprovince along the same mineralized trend as the Canadian Malartic Mine to the west. Production began at the Barnat/Sladen Mines in 1938 and continued until 1970. A total of 8,454,032 tonnes of ore were processed at an average grade of 4.46 g/t Au, yielding a total of 37,743.5 kg Au (1.213 Moz Au). The Barnat/Sladen ore also averaged a little more than 1 g/t silver, yielding a total of approximately 9,000 kg of silver (289,000 oz Ag).

The East Malartic Mine began production in 1935, operated semi-continuously until 1983, and represents the largest historic producer in the Malartic gold camp. Over the lifetime of the mine, a total of

17,948,457 tonnes of ore were extracted, at an average grade of 4.92 g/t Au, yielding 88,239.1 kg Au (2.837 Moz Au).

2.4.5 Past Production Resource and Reserve History

Some of the resource figures reported in this section were prepared under reporting codes other than those acceptable by NI 43-101 and should not be relied upon to conform to current standards and definitions. As such, the data should be interpreted as unclassified historical resource estimates.

From 1980 to 1985, Lac Minerals explored the original Canadian Malartic portion of the Canadian Malartic property with the objective of defining a near-surface (less than 100 m deep) resource amenable to open pit mining. The exploration program led to the definition of five near-surface gold mineralized zones forming an aggregate unclassified resource (pre-NI 43-101) of approximately 8,160,000 tonnes with an average grade of 1.98 g/t Au (520,000 oz Au), using a cut-off grade of 1.03 g/t Au. The mineralized zones defined by Lac Minerals are all the near-surface expression of a much larger, lower-grade, continuously mineralized gold system extending to a depth of at least 350 m.

Sansfaçon (1989) estimated an unclassified resource of 27,210,000 tonnes of “ore” with an average grade of 1.95 g/t Au for the entire original Canadian Malartic deposit. This figure was calculated using a cut-off grade of 1.03 g/t Au and considered mineralization to a vertical depth of 305 m. An updated calculation by Sansfaçon and Hubert (1990) estimated an unclassified resource of 25,637,624 tonnes at an average grade of 2.02 g/t Au, using minimum and maximum cut-off grades of 1.0 g/t Au and 6.9 g/t Au, to a depth of 335 m.

An NI 43-101-compliant Inferred mineral resource was estimated by RSG and released by Osisko on December 7, 2006 with the subsequent Technical Report filed on SEDAR on January 24, 2007. The Inferred estimate was based on historical drilling and partial 60 m x 60 m definition drilling completed by Osisko at that time. Grade estimation was undertaken using a combination of methods including Multiple Indicator Kriging (MIK), Ordinary Kriging (OK) and Inverse Distance (ID^2) weighting.

Categorization of the gold estimates was undertaken on the basis of assessment criteria set out in the NI 43-101, Standards of Disclosure for Mineral Projects of February, 2001 and the classification scheme adopted by CIM Council in August, 2000. Inferred Resources were defined using criteria selected during validation of the grade estimates, with detailed consideration of NI 43-101 categorization guidelines.

Although several different estimation techniques and cut-offs were investigated, RSG recommended that the estimate derived using the MIK approach be reported above a 0.5 g/t Au cut-off as shown in Table 2.4 below.

Table 2.4: 2006 RSG Inferred Resource Estimate (0.5 g/t Au Lower Cut-Off)

Geostatistical Method	Tonnes	Grade (g/t Au)	Moz Au
Multiple Indicator Kriging	178,404,000	1.14	6.539
Ordinary Kriging	177,188,000	1.15	6.551
Inverse Distance Squared	170,806,000	1.20	6.590

An updated NI 43-101-compliant Inferred resource was prepared by RSG and released by Osisko on July 5, 2007 with the subsequent report filed on SEDAR on August 17, 2007. The updated Inferred estimate was based on historical drilling and the completed 60m x 60m definition drilling program, including the Sladen (eastern) extension of the deposit. Grade estimation was undertaken using a combination of methods including Multiple Indicator Kriging (MIK), Ordinary Kriging (OK) and Inverse Distance (ID^2) weighting.

Categorization of the gold estimates was undertaken on the basis of assessment criteria set out in NI 43-101 and the classification scheme adopted by the CIM Council in August, 2000. Inferred Resources were defined using criteria selected during validation of the grade estimates, with detailed consideration of NI 43-101 categorization guidelines.

Although several different estimation techniques and cut-offs were investigated, RSG recommended that the estimate derived using the OK approach be reported above a 0.4 g/t Au cut-off as shown in Table 2.5 below.

Table 2.5: RSG Updated Inferred Resource Estimate (0.4 g/t Au Lower Cut-Off)

Geostatistical Method	Tonnes	Grade (g/t Au)	Au (Moz)
Ordinary Kriging	286,200,000	0.92	8.43
Multiple Indicator Kriging	288,500,000	0.90	8.37
Inverse Distance Squared	277,100,000	0.95	8.44

Finally, a NI 43-101-compliant Preliminary Assessment study compiled by BBA of Montreal with the collaboration of RSG, G Mining Services, Genivar, Golder and the Osisko technical team, was released by Osisko on March 31, 2008 with the full report filed on SEDAR the same day. Only in-pit Inferred resources between surface and a vertical depth of 400 m were considered for the purposes of the study. A gold price of $775 per ounce was assumed for the financial analysis, and current market prices were applied for all materials.

The Preliminary Assessment study indicated an in-pit Inferred resource of 287.7 Mt grading 0.84 g/t Au, equivalent to 6.55 Moz of contained gold with an average recovery of 84%. The study also indicated that over the first three years of production, Canadian Malartic could average 572,000 ounces of gold per year at an average head grade of 1.05 g/t Au and with cash costs averaging $314 per ounce (including royalties). Over the first three years, the operation could generate pre-tax cash flow of $731 M. Over a projected 14 year mine life the then current deposit could produce an average of 457,800 oz of gold per year at an average cash cost of $381 per ounce (including royalties), generating pre-tax operating cash flow of $2.58 billion, with over $1 billion pre-tax cash flow generated in the first five years of production. The study showed a potential Internal Rate of Return of 22.2% and payback period of 39 months.

3.0 GEOLOGY AND EXPLORATION

Information on the geology, mineralization and deposit types at Osisko's Canadian Malartic project has been previously described in NI 43-101 Technical Reports (Gossage 2007a, Gossage 2007b and Runnels et al. 2008) filed by the company on SEDAR (www.sedar.com). The information provided herein is from data provided by Osisko and from the above reports.

3.1 Regional Geology

The Malartic property straddles the southern margin of the eastern portion of the Abitibi Subprovince, an Archaean greenstone belt situated in the southeastern part of the Superior Province of the Canadian Shield. The Abitibi Subprovince is comprised of an older northern volcanic zone (2730 - 2710 MA) and a younger southern volcanic zone (2705 - 2698 MA), separated by the regional Porcupine-Destor Fault Zone (Card and Poulsen, 1998). The Abitibi Subprovince is limited to the north by gneisses and plutons of the Opatica Subprovince, and to the south by metasediments and intrusive rocks of the Pontiac Subprovince. The contact between the Pontiac Subprovince and the rocks of the Abitibi greenstone belt is characterized by a major fault corridor, the east-west trending Cadillac-Larder Lake Tectonic Zone (Figure 2.1). This structure runs from Larder Lake, Ontario through Rouyn-Noranda, Cadillac, Malartic, Val d'Or and Louvicourt, Québec, at which point it is truncated by the Grenville Front. The corridor defined by the Porcupine-Destor Fault Zone and the Cadillac-Larder Lake Tectonic Zone, generally known as the Timmins-Val d'Or camp (Robert et al. 2005), hosts a great number of mineral deposits that account for the bulk of historical and current base and precious metal production from the Superior province (Spooner and Tucker Barrie, 1993).

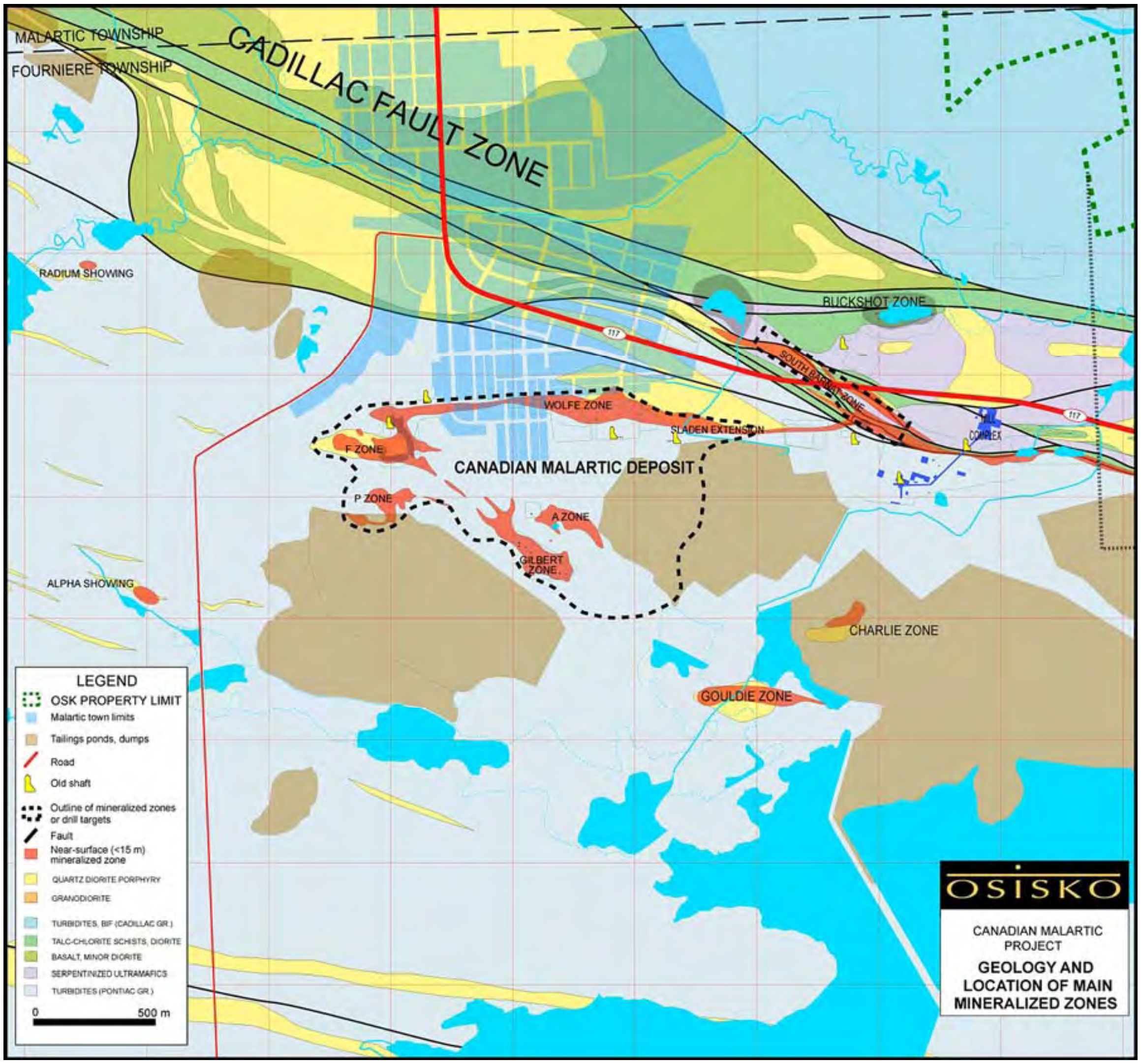
The regional stratigraphy of the southeastern Abitibi area is divided into groups of alternating volcanic and sedimentary rocks, generally oriented at N280° to N330° and separated by faults zones. The main lithostratigraphic divisions in this region are, from south to north, the Pontiac Group of the Pontiac Subprovince and the Piché, Cadillac, Blake River, Kewagama and Malartic Groups of the Abitibi Subprovince. The Pontiac group includes greywackes, shales and minor conglomerates (turbiditic clastic sediments), as well as thin horizons of ultramafic volcanic rocks. The Piché group, confined within the Cadillac Tectonic Zone, comprises abundant talc-chlorite-carbonate schists representing strongly deformed and altered magnesian basaltic to komatiitic volcanics. The schists include abundant irregular, deformed intrusions of diorite and feldspar porphyry, many of which are gold-mineralized. The Cadillac Group consists of greywackes and polymictic conglomerates; the Blake River Group is dominated by basalts; the Kewagama Group includes greywacke, shales, oxide-facies iron formation and conglomerates and the Malartic group comprises ultramafic volcanic rocks.

The various stratigraphic units listed above are folded into a regional synclinal structure variously known as the Malartic or Cadillac Syncline. The fold axis trends west-northwest and dips steeply to the north, with the axial trace located within the Cadillac Group sediments. The various lithological groups within the Abitibi Subprovince are metamorphosed to greenschist facies. Metamorphic grade increases toward the southern limit of the Abitibi belt, where rocks of the Piché Group and the northern part of the Pontiac Group have been metamorphosed to upper greenschist facies. The latter rocks have been subject to retrograde metamorphism, probably due to hydrothermal flux associated with the Cadillac Fault, as evidenced by chloritization of biotite, development of actinolite after hornblende and albitization of more calcic plagioclase. Metamorphism increases rapidly to the south of the Cadillac Tectonic Zone. Pontiac Group sediments at the southern end of the Canadian Malartic property are metamorphosed to staurolite facies. This higher grade metamorphic terrane is also punctuated by frequent peraluminous granite intrusions, derived from partial melting of the metasediments during orogenesis.

3.2 Property Geology

The majority of the Canadian Malartic property is underlain by meta-sedimentary units of the Pontiac Group, lying immediately south of the Cadillac Tectonic Zone (Figure 3.1). The north-central portion of the property covers an approximately 3.5 km-long section of the fault corridor and is underlain by mafic-ultramafic metavolcanic rocks of the Piché Group cut by porphyritic intrusions, as well as metasediments of the Cadillac Group to the north of the fault zone. At the point where the Cadillac Tectonic Zone transects the town of Malartic, it is oriented N320°E, whereas further east it is oriented at N280°E - N290°E. The rapid change in the direction of the fault corridor has been interpreted by Gunning and Ambrose (1940) and Eakins (1962) as a bifurcation of the fault zone. The portion of the fault zone oriented N280°E - N290°E has been referred to as the Malartic Tectonic Zone; it extends about 9 km along strike with a width of 600 to 900 m. The Malartic Tectonic Zone includes many subordinate faults with orientations varying from sub-vertical to sub-horizontal.

Figure 3.1: Property Geology Map



The portion of the Piché Group volcanic belt that transects the Canadian Malartic property is about 650 m wide. Two major structures, the Malartic (Cadillac) and Sladen faults, define the northern and southern boundaries of the tectonic zone in the immediate Malartic area. As it occurs on the property, the Malartic fault is oriented N260°E - N280°E and dips 75° to the north, whereas the Sladen fault is oriented N090°E - N100°E and dips variably from 70°S to sub-vertical. The Piché Group ultramafic metavolcanic rocks do not outcrop anywhere on the property, and are known from historic records, underground workings and drilling. The Piché Group rocks are typically bluish-grey, pervasively foliated with numerous veinlets of talc-carbonate. Less altered variants occur as massive, aphanitic to fine grained serpentinized ultramafic rock.

The Pontiac Group metasediments on the property comprise turbiditic greywacke, mudstone and minor siltstone, generally rhythmically banded with beds of variable thickness ranging from about one millimetre to one metre. The sediments typically have a well-developed foliation and are dark grey to black, occasionally exhibiting a brownish tint caused by development of biotite through metamorphism and/or potassic alteration proximal to porphyritic felsic intrusions.

The rocks of the Pontiac and Piché Groups are intruded by a number of epizonal felsic porphyritic bodies, variously described as syenites, quartz syenites, quartz monzonites, granodiorites and tonalites. The geometries of these felsic intrusions are highly variable, and occur on the property as sills, dykes, discontinuous lenses or small isolated stocks.

The porphyries are all feldspar-phyric (1 to 5 mm wide phenocrysts) with fine-grained to aphanitic, medium to light grey matrices. Within the Pontiac Group, the porphyritic intrusions are particularly abundant within an area bounded to the south by the Raymond Fault. South of the Raymond Fault, a swarm of ultramafic sills (possibly komatiitic flows) occur in the metasediments in the southwestern portion of the Canadian Malartic property. The Fournière granodiorite/tonalite pluton touches the southeastern extremity of the property.

Surface drilling by Lac Minerals in the 1980’s defined several near-surface mineralized zones, all expressions of the larger, continuous mineralized system at depth. In addition to these, the Gouldie and Charlie mineralized zones occur approximately 1.2 km southeast of the main deposit, although the relationship between these zones and the main deposit is presently unknown. Within the Cadillac Tectonic Zones, several near-surface mineralized zones have been documented (South Barnat, Buckshot), generally all associated with shattered felsic intrusions.

3.3 Deposit Types

Base and precious metal deposit types within the southern portion of Abitibi Subprovince, i.e. the Timmins-Val d'Or belt (Robert et al., 2005), include volcanogenic massive sulphide (VMS) deposits, lode gold deposits, komatiite-hosted and gabbro/peridotite-hosted nickel-copper-PGE deposits, small porphyry copper-molybdenum deposits and small vein-type molybdenum deposits (Card and Poulsen, 1998; Spooner and Tucker Barrie, 1993).

Archaean lode gold deposits of the Superior Province include several types but are dominated by epigenetic, structurally-controlled mesothermal vein deposits, i.e. "orogenic" deposits in the sense of Hagemann and Cassidy (2000). Other types include disseminated and stockwork porphyry-related deposits, with or without vein overprints, sulphide-rich breccia and replacement deposits, gold-rich VMS deposits and gold-rich pyritic exhalites (Robert and Poulsen, 1997; Card and Poulsen, 1998).

The gold deposits of the Malartic area are porphyry-related (Issigonis, 1980; Robert 2001) and possibly orthomagmatic in origin. The porphyries are generally considered to be syenitic (alkaline) in composition and of Timiskaming (syntectonic) age (Fallara et al., 2000; Robert, 2001), although recent petrographic examinations of unaltered porphyries suggest a sub-alkaline dioritic to monzodioritic composition. Given the nature of the mineralization (see below) and the close spatial association with high-level porphyry dykes and stocks, Osisko adopted the porphyry gold model (Sillitoe, 2000) as a tool to drive exploration on the property.

3.4 Mineralization

Mineralization in the Canadian Malartic deposit occurs as a continuous shell of 1 to 5% disseminated pyrite with fine native gold and traces of chalcopyrite, sphalerite and tellurides (Eakins, 1962; Fallara et al, 2000). It is mostly hosted by altered clastic sediments of the Pontiac Group (turbiditic greywacke, mudstone and minor siltstone) overlying an epizonal dioritic porphyry intrusion. Mineralization also occurs in the upper portions of the porphyry body. The porphyry intrusion pinches out in the Sladen Malartic Mine and disseminated mineralization continues in the silicified greywacke, forming a subvertical tabular body that is truncated by the Cadillac fault at the western extremity of the East Malartic Mine.

Alteration in the metasediments consists of biotite-sericite-carbonate (potassic alteration) overprinted by cryptocrystalline silica-carbonate. Carbonates include calcite and minor ankerite. Highly silicified zones adopt a “cherty” texture and are commonly brecciated. Potassic alteration in the porphyry consists mostly of alkali-feldspar replacement of plagioclase that is contemporaneous with minor quartz veining. Cryptocrystalline quartz replacement with minor carbonate also overprints potassic alteration in the porphyry. Late, coarse-grained, quartz-feldspar-muscovite veins mineralized with native gold form relatively small, higher grade stockworks along the northern edge of the deposit (Eakins, 1962; Derry, 1939). Retrograde chlorite-calcite alteration of the previous assemblages, particularly the biotite, is ubiquitous throughout the deposit but is particularly intense along ductile shear zones, forming chlorite-calcite schists.

Osisko staff believe that the close spatial association between voluminous, low-grade, disseminated gold mineralization and an epizonal, intermediate porphyry intrusion, as well as the presence of widespread potassic alteration throughout the system, suggests that the Canadian Malartic deposit may be an Archaean porphyry gold system.

3.5 Exploration and Drilling

3.5.1 General Exploration

After acquiring the property in the fall of 2004, Osisko initiated a review and compilation of the Canadian Malartic Mines database received from the Québec Government in January, 2005. Data indicated the presence of a widespread, low-grade mineralized system (minimum 1,200 m long and 350 m wide) to depths of 350 m. The data included drill logs and assays for more than 4,600 surface and underground drill holes, specifically the Canadian Malartic Mines U-series (underground) and S-series (surface) holes, and Lac Minerals D-series (surface diamond drilling) and P-series (surface percussion drilling) holes, as well as hundreds of maps, level plans, vertical sections and longitudinal sections.

A total of 4,468 historic holes were compiled into an electronic database for use in the Inferred resource estimate presented in the first Canadian Malartic Technical Report (Gossage 2007a). The remainder were discarded as they were incomplete, illegible or had missing or unreliable collar data. The percussion-hole assay data were not considered, on the assumption that they were of questionable quality. It should be noted that the maps and sections from the historic archives suggest that more than 5,000 surface and underground holes were drilled on this portion of the property, but data for the missing holes (U-series holes above 4,000) were never found.

Drilling on the property by Osisko commenced in March, 2005 in a series of systematic phases designed to outline the mineralized deposit.

Osisko completed a reconnaissance prospecting/sampling program on the property in the summer of 2005. A total of 404 samples were collected and analyzed for gold. Two new zones of surface mineralization were discovered. The two zones consist of mineralized sedimentary and porphyritic units, similar to those observed in the Canadian Malartic deposit. The new zones, Alpha and Bravo, are respectively located 1 km southwest and 5 km southeast of the main deposit.

Surface sampling on the Alpha zone returned results ranging from 80 ppb Au to 4,840 ppb Au (4.84 g/t Au) from a total of 9 surface grab samples. The Alpha zone samples consist of carbonate-altered sedimentary units with minor disseminated pyrite mineralization. The mineralized samples were collected from a few outcroppings over an area 110 m long by 25 m wide.

The Bravo zone returned results ranging from 150 ppb Au (0.15 g/t Au) to 1,460 ppb Au (1.46 g/t Au) from a total of 17 grab samples collected from sparse outcrops spread over an area 70 m long by 15 m wide. The Bravo zone samples included sedimentary and porphyritic units with carbonate alteration and minor pyrite mineralization. No additional work was done in the area of these showings in 2006. A more detailed sampling program was completed in the summer of 2007, including the immediate areas of Alpha and Bravo showings. No new significant showings were discovered.

Osisko completed a high-resolution (50 to 100 m line-spacing) helicopter-borne geophysical survey over the property in the summer of 2006. The survey included magnetic, radiometric (K-Th-U) and five-channel frequency domain electromagnetic readings. The survey was performed by Fugro Airborne Surveys Corp. of Mississauga, Ontario and total coverage of the survey block amounted to 2,485 km.

In the autumn of 2006, the Québec Government provided Osisko with the archival data for the Sladen/Barnat and East Malartic Mines. Compilation of these data is underway, but the data pertain to a portion of the property outside of the area considered for the current resource estimation.

3.5.2 Osisko Drilling Program

Osisko began exploration drilling on the Malartic property in March, 2005, using a Longyear 38 rig with a wire-line retrieval system, supplied by Forage-Plus of St. Georges, Québec, recovering NQ-diameter core. The drill rig is mounted on a fixed base with steel skids and is surrounded by a fixed drill shack. It is mobilized on the property using a tractor or bulldozer. The Longyear 38 unit is capable of drilling to vertical depths of 800 to 1,000 m with NQ-diameter rods. A second Longyear 38 rig was added to the project in December, 2005, essentially identical to the first and also set up to drill NQ-diameter core. Two more rigs were mobilized to the property in July, 2006, also drilling NQ-diameter core. One of these was another Longyear 38 similar to the first two, while the fourth was a Boyles-17 rig. The Boyles drill is lighter than the Longyear models, and is capable of drilling to a vertical depth of about 350 m with NQ-diameter rods. Additional rigs were added to the program in 2007 such that at the peak of the definition drilling program in the fall of 2007 and winter of 2008, 15 drill rigs were operating on the property.

Core production varies somewhat according to the ground conditions, but is occasionally as high as 100 m for a 12 hour drill shift. With all four units in operation, drilling was proceeding at a maximum rate of approximately 10,000 m per month.

Core is placed in standard wooden core boxes at the drill rig and a cover is affixed. The core is then delivered each shift to the Osisko core logging facility at the Malartic field offices.

Drill access on the property is achieved through a network of dirt roads. In lightly forested areas the trees and brush are cut prior to drill mobilization. Larger logs are cut and stacked at the path-sides while lighter brush and branches are processed through a wood-chipper. Water for the drilling operation is generally supplied by submersible pumps lowered into one of several flooded shafts of the historic mines.

In almost all cases, the drill casing is left in-ground after holes are completed and down-hole surveys have been performed, so that collar position can be precisely measured, and the hole can be extended, if necessary. Casings are plugged with a wooden stopper to keep debris out of the hole and large wooden posts are planted to mark the casing location.

3.5.2.1 Methodology and Planning

Drilling has proceeded in a number of discrete phases. A Phase 1 program (total of 2,190 m), completed during the first quarter of 2005, was designed to evaluate the F zone porphyry at the western end of the deposit between sections 3290E and 3674E. The objective was to test the system for continuous, near-surface, relatively homogeneous low grade mineralization. Results were encouraging with assays of semi-continuous disseminated mineralization returning 72- to 163-m long intersections grading 1.01 to 1.70 g/t Au.

A Phase 2 program (20 holes, 5,198 m) was immediately initiated in order to test the strike length of the system. Drilling of this phase was completed to section 4460E by October, 2005, again with encouraging results.

A 14,700 m Phase 3 program followed and marked the beginning of the definition drill program on the deposit, executed along north-south lines spaced approximately 60 m apart, with collars also spaced about 50 to 60 m apart on individual lines. The majority of the holes were drilled vertically, as the deposit is broad and locally complex, and not limited to any particular discrete horizon. Angled holes were used primarily in areas close to underground workings, to avoid stopes or to investigate mineralization proximal to stope limits. The Phase 3 definition program targeted the deposit to a depth of about 350 m between sections 3444E and 3850E. This program was completed with drill hole CM06-728 in March, 2006, resulting in a total of 22,090 m drilled in the first year of exploration, including three additional peripheral exploration holes.

The 60 m definition drilling program continued to section 4520E with Phases 4 and 5 (93 holes, total of 23,704 m) that was completed in September, 2006. Five additional inclined exploration holes with a total of 945 m were also drilled in order to test a separate deposit (the South Barnat zone) during the summer of 2006. Results were particularly encouraging, the best intersection (BA06-1007) yielding 57.6 m at 4.11 g/t Au.

A 14,095 m Phase 6 program was then designed to extend the definition drill program 480 m further east to section 5000E, i.e. along the Sladen Extension, representing the historic Sladen Mine, which is contiguous with the Canadian Malartic deposit. Phase 6 was initiated in September, 2006, simultaneously with two other programs: A) Phase 7 (5,296 m), designed to test the southern extension of the deposit surrounding the Gilbert Zone; B) Phase 8 (59,400 m), designed to infill the definition program along a 30 m spaced grid in the western part of the Canadian Malartic deposit, between sections 3260E and 3850E. Phase 7 was completed as of the end of 2006.

Phase 6 was completed in February, 2007, whereas Phase 8 was mostly completed by September, 2007. Approximately 1,780 m of this program were delayed until freeze-up in January, 2008 as the planned hole entailed drilling over the historic Canadian Malartic Mines tailings pond. The 30 m infill program was also expanded in 2007 to section 5030E with Phases 9, 10, 11 and 12 (total of approximately 174,800 m), which were collectively completed by the end of March, 2008.

Exploration drilling totalled 12,560 m in 2007 and 21,520 m as of the end of May, 2008. This included reconnaissance testing of the Southeastern Extension of the deposit, testing of the eastern extension of the South Barnat Zone, the Jeffrey Zone as well as testing of targets on surrounding properties owned by Osisko. Condemnation drilling in the projected tailings impoundment area totalled 3,295 m in 2007 and 25,873 m in 2008.

As of the end of May, 2008, Osisko had completed a total of about 1,700 drill holes for 386,760 m of drilling on the Malartic project, in all categories, including 1,327 holes for 316,230 m on the Canadian Malartic deposit itself. These drill and assay data were used for the purposes of this report.

As these programs represent a vast amount of definition and exploration drilling a detailed presentation of individual drill results is beyond the scope of this report.

3.5.2.2 Collar Surveying

With few exceptions, planned hole locations are established and the positions of completed holes are measured using a differential GPS unit. From March, 2005 to September, 2005 surveys were performed by contractors from J. L. Corriveau in Val d’Or, Québec. In October, 2005 a similar system was purchased for use on the project. On the occasions where casings are necessarily removed immediately after completion of a hole (i.e. holes collared on the streets in the south-central area of Malartic), a marker or artefact is left to indicate the position of the collar to within a few centimetres.

Surveys are performed using a Sokkia Radian IS real-time-kinetic differential GPS system. A base-station unit set up near the Malartic exploration office broadcasts data to a mobile unit so that precise real-time positions can be calculated. At the beginning and end of each survey day, readings are taken on a brass-medallion survey benchmark anchored into a sidewalk in the south-central area of Malartic. Precise coordinates for this benchmark are available from the Québec Government. Repeated measurements at this point indicate that the Sokkia DGPS has a horizontal precision of about ± 1 cm and a vertical precision of approximately ± 2 cm. The Universal Transverse Mercator (UTM), 1983 North American Datum (NAD83) coordinate system is used for recording position data. In the Malartic area, the northing of the NAD83 coordinate system is oriented about 2.18°E of geographic north.

Planned hole locations are marked by a steel rod, and at least two offset pickets along UTM north or south of the rod. The offsets are used to properly locate the drill if the hole-marker rod is displaced during drill setup, or as front sight/back sight markers in the case of inclined holes. The offsets also serve as an independent check that the GPS is working properly, as the marker rod and offset pickets will line up precisely if the unit is operating normally.

Completed holes are resurveyed using the DGPS equipment. Measurements are taken at the centre of the top of the casing, as well as at ground level at the side of the casing. In the case of inclined holes, the ground-level measurement is taken at the leading edge of the casing. In any case, reported positions of the completed drill holes are considered to be accurate to within 15 cm in X, Y and Z directions.

3.5.2.3 Down-Hole Surveying

Procedures for down-hole surveying have changed over the evolution of the project. Initially, down-hole dip-deviation data was acquired by acid-tests performed at approximately 100 m intervals. The drill contractor has since acquired a Flexit tool for measuring down-hole deviation. Holes are routinely surveyed immediately after they are completed. The initial series of holes were resurveyed.

The Flexit probe is a self-contained unit, including batteries, control and synchronizing electronics, internal radio link antenna, three orthogonally mounted accelerometers, three orthogonally mounted magnetometers and a temperature sensor. The probe simultaneously measures azimuth (± 0.3°), inclination (± 0.2°), total magnetic field (± 50 nT), magnetic dip (± 0.3°) and hole temperature (± 0.2°C).

Data from the probe are transferred to a mobile data collection unit, and then downloaded to a computer for incorporation into drill logs.

3.5.3 Sampling Approach and Methodology

Sampling of gold mineralization from the Canadian Malartic property has been essentially limited to the collection of samples of diamond drill core. A limited amount of surface sampling on the property was performed by independent consulting geologists during the summers of 2005 and 2007; these samples were submitted for assay using the same general protocol as that employed for core samples.

All samples are analyzed for gold by Chemex of Val-d'Or, Québec, a laboratory which is certified ISO 9001:2000. Samples are analyzed by standard 50 g fire assay with atomic absorption finish and any samples yielding greater than 10 g/t Au are reanalyzed with a gravimetric finish. Density measurements are performed on one in twenty five of the assayed samples.

All aspects of the sampling method and approach were reviewed by Micon during its site visit. The QA/QC procedures for ensuring the security of core samples, the integrity of chain-of-custody for samples and the accuracy of laboratory analyses are in line with current industry practice.

3.5.3.1 Diamond Core Logging and Sampling

During logging detailed descriptions of the drill core are made by experienced and qualified geologists under the employ of Cygnus of Montreal who are members in good standing of the OGQ (Québec Order of Geologists) or the OIQ (Québec Order of Engineers). Drill log data are recorded directly onto laptop computers using a multi-page core logging template designed for the specific needs of the project. The core logging protocol is described below.

Core boxes are laid out 9 to 12 at a time. Marker blocks are temporarily removed and their position marked on the core with a grease pencil. The core is aligned as well as possible so as to consolidate any gaps and the core is then measured and marked metre-by-metre with a grease pencil. Any discrepancies between marker blocks and measured core length are immediately addressed and resolved. The drill hole interval ('from' and 'to') for each box is recorded in the log.

The next step involves recording the RQD (rock quality designation, a geotechnical logging parameter) using a reference spacing of 2 m and discounting core pieces less than 10 cm long. This is followed by a visual estimate of core recoveries, done on a 0.5 m basis. Marker blocks are then replaced.

Core is then marked up for sampling. All material with even slight alteration (either carbonate, silica or hematite), sulphide mineralization or veining is sampled. Samples are a minimum of 1.0 m long and a maximum of 1.5 m long and must respect lithological contacts or the interface between zones of significantly different alteration style. Monotonous sections of barren, unaltered rock are marked up at 1.5 m intervals for description, and every fifth such interval marked for check sampling purposes.

Sampling intervals (from/to) are recorded in serially-numbered tripartite sample tag booklets. These measurements are recorded on the first and second portion of the tag, with the third left blank. The second and third portions are torn from the book and tucked into the core box at the beginning of the appropriate sample interval mark, for use by the core sampling technician. Sampling and mark-up intervals are recorded in the logs, along with sample tag numbers, when applicable.

Geological descriptions are made of each marked interval whether sampled or not. Separate columns in the log allow for codes describing the protolith, alteration style, overall intensity of alteration, relative degree of carbonate and silica alteration, sulphide percentage, rock colour, vein type and veining density. A separate column is reserved for written notes on lithology, mineralization, structure, vein orientations/relations, etc.

Once core logging is completed on an interval-by-interval basis, a summary geological description of the hole is completed using a simplified code. The header page, listing the hole number, collar coordinates, final depth, start/end dates and the name of the core logging geologist is completed. Core is then photographed in batches of up to 5 boxes at a time, both dry and wet. Logs and photographs are then submitted to the project geologist.

The project geologist is responsible for verification of information in all logs as they are completed. The final steps are to incorporate the down-hole survey data into the log, and record the security tag numbers for each sample (refer to Section 3.5.3.2) provided by the core-sampling technician. Once the assay results have been received, these are also incorporated into the logs.

3.5.3.2 Core Sampling, Security and Chain-of-Custody

Core samples collected at the drill site are stored in closed core boxes sealed with fibre tape and are delivered to the exploration offices at shift change. All core logging, sampling and storage takes place at the exploration offices, located in the administrative complex of the historic East Malartic Mine site. The compound is surrounded by chain-link fence, monitored by closed-circuit video cameras and has a security guard posted at all times at the entrance. Employees, directors, officers and associates of Osisko are not involved in any aspect of the logging, manipulation or sampling of core.

Following the logging and core marking procedures described above, the core passes to the sampling facility. At this point, the core is no longer handled by on-site geologists. Core sampling is performed by qualified technicians of Cygnus and quality control is maintained through regular verification by on-site geologists.

Core is broken, as necessary, into manageable lengths. Pieces are removed from the box without disturbing the sample tags, cut in half lengthwise with a diamond saw, and then both halves are carefully repositioned in the box. When a complete hole has been processed in this manner, one half of the core is collected for assay while the other half remains in the core box as a witness.

The technician packs one half of the split core sample intervals into vinyl sample bags that are sequentially numbered to match the serial number sequences in the tag booklets used by the core-logging geologists. The blank portion of the triplicate sample tag is placed in the bag with the sample, while the portion marked with the sample interval is stapled into the bottom of the core box at the point where the sample interval begins. Sample bags are sealed with tamper-proof, serially numbered, yellow plastic security tags. The technician notes the beginning and end of the security tag sequence for a particular sampling run, and reports this to the project geologist so that the drill logs can be finalized.

Sealed sample bags are packed into sturdy plastic barrels with locking lids or in large weaved nylon shipping bags (rice bags). When full, the barrels or shipping bags are sealed with tamper-proof, serially numbered, red plastic security tags. Barrels/bags are assigned sequential numbers which are matched against the security tags and loaded on sequentially numbered, plastic-wrapped wood pallets. This information is also forwarded to the project geologist.

Aluminum tags embossed with the hole number, box number and box interval (from/to) are prepared and stapled onto the ends of each core box. Core boxes are then moved to permanent on-site storage in steel core racks. Rejects and pulps from the laboratory are sent back to the Canadian Malartic site and stored in large domed structures with limited access.

The project geologist prepares the sample submission form for the assay laboratory. This form identifies the barrels/shipping bags by number and security tag number, as well as the sequence of samples packed in each. Couriers from Chemex arrive once or twice per week at the core-processing facility to transport the pallets of sealed barrels/bags directly back to the laboratories. Once at the laboratory, a manager checks the barrel and security tag numbers against those that are on the submission form, and initializes each if the corresponding numbers are correct. Copies of these forms are then returned to the exploration offices for verification, and any discrepancy is investigated and corrected as necessary.

3.5.3.3 Conclusions

It is Micon's opinion that the logging and sampling protocols used by Osisko and Cygnus at the Canadian Malartic project are conventional industry standard ones conforming to generally regarded best practices. Micon is confident that the system is appropriate for the collection of a database suitable for the estimation of an NI 43-101 compliant mineral resource estimate.

3.5.4 Sample Preparation and Analytical Procedures

3.5.4.1 Analytical Laboratories

All primary and duplicate assay work for the Malartic project has been performed by Chemex located in either Val d'Or, Québec or Reno, Nevada. To facilitate turnaround time of the large volume of samples submitted, sample pulverizing is done primarily at Chemex's preparation facility located in Timmins, Ontario.

All Chemex laboratories are certified ISO 9001:2000 for the "supply of assays and geochemical analysis services" by BSI Quality Registrars. Certification for ISO 9001:2000 requires evidence of a quality management system covering all aspects of the organization. Chemex also takes part in the "Proficiency Testing Program - Minerals Analysis Laboratories" and holds a certificate demonstrating its success in the program for analysis of gold, silver, copper, zinc, lead, nickel and cobalt. Samples for umpire assaying were submitted to Accuracy Laboratories in Thunder Bay, Ontario or Acme Laboratories in Vancouver, British Colombia.

3.5.4.2 Sample Preparation and Analytical Procedures

All samples received by Chemex are processed through a sample tracking system that is an integral part of that company's Laboratory Information Management System (LIMS). This system utilizes bar coding and scanning technology that provides complete chain-of-custody records for every stage in the sample preparation and analytical process and limits the potential for sample switches and transcription errors.

Samples are dried, and then crushed to 70% passing -10 mesh (1.7 mm). A 250 g subsample is split off the crushed material, and pulverized to 85% passing -200 mesh (75 microns). A 50 g split of the pulp is used for assay. Crushing and pulverizing equipment is cleaned with barren wash material between sample preparation batches and, where necessary, between highly mineralized samples. Sample preparation stations are also equipped with dust extraction systems to reduce the risk of sample contamination.

As part of the standard internal quality control procedures used by the laboratory, each batch of 84 fire-assay crucibles includes one blank, two internal (laboratory-generated) standards and three duplicate samples along with 78 client samples. In the event that any reference material or duplicate result falls outside the established control limits, an error report is automatically generated. This ensures that the person evaluating the sample set for data release is made aware that a problem may exist with the dataset and an investigation can be initiated.

Pulps and coarse rejects from the samples are returned to the Malartic exploration offices on a regular basis. These materials are securely stored in a locked facility for future reference.

3.5.4.3 Gold Analyses

Prepared samples are analyzed by fire assay with atomic absorption finish. Samples returning assays in excess of 10 g/t Au are re-analyzed with a gravimetric finish.

The Lac Minerals samples were assayed at the Bousquet Mine site assay laboratory with a detection limit of 0.069 g/t Au reported in the database with 0.034 g/t Au precision steps. The results were originally reported in ounces/short ton. The majority of the Lac samples have been reassayed by Osisko as described previously.

The original Canadian Malartic samples were assayed by fire assay although details of the approach are unknown. The original data were recorded in pennyweights (dwt) with a detection limit of 0.2 dwt (approximately 0.34 g/t Au). Data precision steps are approximately 0.17 g/t Au.

3.5.4.4 Specific Gravity

Specific gravity (bulk density) measurements are performed on approximately one out of 25 of the submitted samples. For bulk, non-porous material, a piece of the sample is weighed, and its volume determined by immersion. Porous materials tested in this way are treated with a coat of paraffin wax to seal them prior to testing.

3.5.5 Quality Assurance/Quality Control (QA/QC) procedures

Accuracy and potential contamination of the analytical procedure at the laboratory are monitored by the introduction of blanks and blind certified reference standards into the sample stream by Osisko. For the (QA/QC) program five different Rocklabs certified reference standards were obtained from Mine Assay Supplies of Canada, ranging from 0.583 g/t Au to 8.543 g/t Au. The bulk standards were split into 120 g bags on site with different internal codes for introduction into the sample stream. Blanks consist of locally purchased decorative marble stone purchased in 30 kg bags. One standard or one blank is assayed per 18 to 20 samples in a batch.

Duplicate and umpire assaying was performed using different protocols as the drill program evolved. For drill holes 651 to 800 (the beginning of program), recommendations from Roscoe Postle Associates of Toronto were followed (the RPA Protocol), consisting of:

- Duplicate assaying of 1 in 20 rejects from the mineralized zone
- Duplicate assaying of 1 in 20 original pulps from the mineralized zone
- Umpire assaying of 1 in 20 original pulps from the mineralized zone (different from the original pulps above)
- Quarter split assaying from higher grade mineralized zones (stopped after hole 682)
- Introduction of 1 in 20 blind certified standards or blanks into the entire sample stream.

For drill holes 801 to 1250, the RPA Protocol was modified in order to accelerate the checking procedure and consisted of:

- Duplicate assaying of 1 in 20 rejects from the mineralized zone
- Duplicate assaying of 1 in 20 original pulps from the mineralized zone
- Umpire assaying of rejects from the reject samples above
- Introduction of 1 in 20 blind certified standards or blanks into the entire sample stream.

For drill holes 1251 to 1499 a new protocol was introduced following recommendations by RSG. This new protocol (the RSG Protocol) was introduced gradually and overlapped with the modified RPA Protocol, resulting in the following procedure:

- Duplicate assaying of 1 in 20 rejects from the mineralized zone
- Duplicate assaying of 1 in 20 rejects from the entire sample stream, performed automatically at Chemex (CD sample series)
- Umpire assaying of rejects from the rejects from the entire sample stream above
- Introduction of 1 in 20 blind certified standards or blanks into the entire sample stream

Finally, starting with drill hole 1500 (and currently the practice), the RSG Protocol was fully implemented, consisting of:

- Duplicate assaying of 1 in 20 rejects from the entire sample stream, performed automatically at Chemex (CD sample series)
- Umpire assaying of rejects from above
- Introduction of 1 in 20 blind certified standards or blanks into the entire sample stream.

Cygnus staff monitor the results of the duplicate, blank and reference standard assay results visually looking for significant discrepancies in duplicate results, anomalously high values for the blank samples or sample results and significant deviations from the accepted values for the standards and using the 95% confidence limits provided by Rocklabs as a guideline. Any anomalous result is followed up with the laboratory and a significant amount of reassaying has been conducted in order to produce the final database.

It is Micon’s opinion that Osisko is running an industry standard QA/QC program for its insertion of control samples into the stream of core samples for the Canadian Malartic project. However, in future Micon recommends that the results of the blank, standard and duplicate sample be monitored by means of control charts rather than visually.

For reference standard samples the control chart should consist of the assay results for each standard plotted on the y axis against time on the x axis. Superimposed on this chart should be five horizontal reference lines representing the accepted value for the standard and the accepted-value-plus-two and -three standard deviations (SD) and minus-two and -three standard deviations. The accepted value of the standards, and all of the individual assay results from the round robin program used to determine it, are available on the Rocklabs website. From this the standard deviation can be calculated.

An analysis of a standard would be considered a QA/QC failure if the result came back outside of the +/- 3 SD lines or if two samples in a row came back between the 2 SD and 3 SD lines on the same side of the graph. Such charts can also show trends of drift over time indicating problems with calibration of instruments or, if the accepted value is returned repeatedly, that the standard has been identified and its value is being faked.

For the blanks a background value should be determined from the early assay results at which point it should be confirmed that it truly is a blank with very low, near-detection-limit values. Using this value any subsequent result greater than about 10 times background should be considered a failure. A simple graph similar to the analytical standard one with the blank assay results and the 10 times background line should be prepared to monitor the results.

Duplicates should be plotted on x-y scatter plots and compared against a 45º line. Linear regression lines with r^2 values can also be plotted to monitor bias and scatter in the results.

3.6 Data Verification

The database used for the resource estimate presented herein was, in part, used previously by RSG for the earlier Inferred resource estimates at the Canadian Malartic project (Gossage, 2007a, 2007 b and Runnels et al. 2008). That database, with additional drilling from 2008, was used for the Micon resource estimate. RSG conducted its own data verification program which is included in those reports which are available on SEDAR (www.sedar.com) under Osisko’s filings.

3.6.1 Drill Hole Database

A drill hole database in Microsoft Access format was supplied to Micon by Osisko. The database contained the following tables and fields:

- Collar information - hole ID, maximum depth, xyz coordinates of collar
- Down-hole survey - hole ID, down-hole depth, dip, azimuth

- Assay - hole ID, depth from, depth to, sample ID, gold value in g/t Au
- Geology - hole ID, depth from, depth to, sample ID, rock code and alteration type.

The combined database was then reviewed and validated prior to being finalized into an appropriate format for resource evaluation. The following general activities were undertaken during database validation by Micon:

- Cross check total hole depth and final sample depth data
- Check for overlapping and missing sampling intervals
- Replace unsampled intervals and entries for less than detection limits with -999. (During the compositing process these samples were given a grade of 0.001 g/t Au.)
- Check drill hole survey data for unusual or suspect down-hole deviations
- Check lithology and alteration codes.

Checks were made on 85 underground (Canadian Malartic Mines) drill holes to compare the values registered in the historic paper copies of the logs with the values in the database and to check for transcription errors. A total of 5,078 assay data entries were verified and only 2 minor errors were found.

Additionally some 34 assay certificates were downloaded by Micon directly from Chemex's internet-based "Webtrieve" system for comparison with the results from the Osisko-generated samples in the database. The certificates retrieved were selected randomly from the entire period of Osisko drilling. Some 4,030 entries were checked and no errors were found.

3.6.2 Comparison of Data Types

The Canadian Malartic database contains surface and underground diamond drill holes drilled by different operators over a period of about 80 years. Every time different sample populations are present in the same database, it is important to compare them to be sure that there is no bias between the populations. There is little concern about the recent Osisko drilling results since the drilling program was conducted with modern industry standards (drilling, sampling and assaying).

The Lac Minerals core was still available and the second part of the core was re-assayed by Osisko (for about 90% of the Lac Minerals' database) and the new assay results were used for resource estimation instead of the original assays. As the core was recently re-assayed with modern techniques, this population is also considered reliable (after QA/QC review) and can be used for resource estimation.

There are more concerns about the old Canadian Malartic assay database. For these early operators, the accuracy of the low grade assays was of much lower importance since they were mining only the high grade portion of the deposit.

The quality of historical assay data is often difficult to assess. For the old Canadian Malartic data set, there is no independent verification possible since the samples and drill core are not available. The verification can only be done by comparing samples or groups of samples not only in close proximity to each other but also in the same geologic environment.

From the examination of the drilling data on vertical sections and level plans, some observations can be made:

- Historical underground drilling was generally focused on the higher grade portion of the deposit and often even limited to the high grade core of the high grade domains. There is an important sampling bias when compared to the recent drilling (important clustering of the data for the historical holes).
- It is crucial to discriminate between the low and high grade domains to compare the data since samples may be very close (within a few metres) but very different in grade because they are located in different domains.
- Even in high grade domains (> 1.0 g/t Au), the grade was not evenly distributed. There was often a higher grade core (> 3.0 g/t Au) that was the focus of the underground operation. Recent surface drilling did not cross this high grade core very often, especially in the areas close to stoping.
- Even after the removal of the samples within the old stopes, there are many more "old" samples in the high grade core than recent samples.

Based on these observations, it is important to realize that for comparison of the data sets to identify a potential analytical bias, the samples must be in the same "geological environment" and not only in proximity to each other. This is difficult to reach only with a statistical approach, especially within the high grade domains where there is no indication if a sample is close to the high grade core or not.

3.6.2.1 Comparative Statistics (Recent vs. Historical Data)

Table 3.1 compares the population statistics of 5 m composites (with cut assays) for historical and recent data sets within the main domains used for resource estimation (void samples removed).

Table 3.1: Comparative Statistics (Recent vs. Historical Data)

Statistic	Low Grade Domains (10)		Pervasive Domains (70)		Main High Grade Domains (20)		Other High Grade Domains (30-60)	
	Historical	Recent	Historical	Recent	Historical	Recent	Historical	Recent
Number	7,380	28,597	2,005	8,433	10,074	5,831	1,969	3,766
Mean g/t Au	0.50	0.29	0.79	0.72	2.07	1.58	1.72	1.42
Min g/t Au	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Max g/t Au	7.00	7.00	9.80	8.28	22.00	22.00	15.00	15.00
Median g/t Au	0.38	0.14	0.63	0.56	1.64	1.34	1.38	1.08
Std Dev g/t Au	0.61	0.47	0.80	0.67	1.77	1.27	1.38	1.38
Coeff. Variation	1.23	1.63	1.02	0.92	0.85	0.80	0.80	0.97
Comparison of Means Recent/Historical		**-42.0%**		**-8.6%**		**-24.0%**		**-17.7%**

As presented in Table 3.1, the Canadian Malartic drilling has a higher mean than the more recent drilling (Osisko + Lac Minerals) for all domains. The difference is more pronounced in the low grade domains with a discrepancy of -42% while it is around -20% for the high grade domains.

Even if this table compares data within the same domains, it is difficult to establish a real measure of a potential analytical bias between the sample populations because the drill holes of the two sets are not equally distributed within the various domains.

3.6.2.2 Comparative Study Within a Relatively Big Solid

To be able to compare the two data sets in the same geological environment, a relatively large 3D solid was constructed in an area where both Osisko and Canadian Malartic drilling was present and where there is no stoping (Figures 3.2, 3.3 and 3.4). The dimension of the solid is about 250 m (east-west) x 325 m (along dip) x 35 m (in thickness). The volume of the solid is close to 2.7 Mm^3 representing a tonnage of about 7.4 Mt (at a specific gravity of 2.75 t/m^3). Gemcom software (GEMS 6.1.3) was used for the study. This solid is part of Domain 20 (sub domain 22) and represents about 10% of the domain. This domain is by far the most important high grade domain in the model (about 80% of the volume of all high grade domains).

Figure 3.2: Plan View of the Solid Used for the Comparative Study

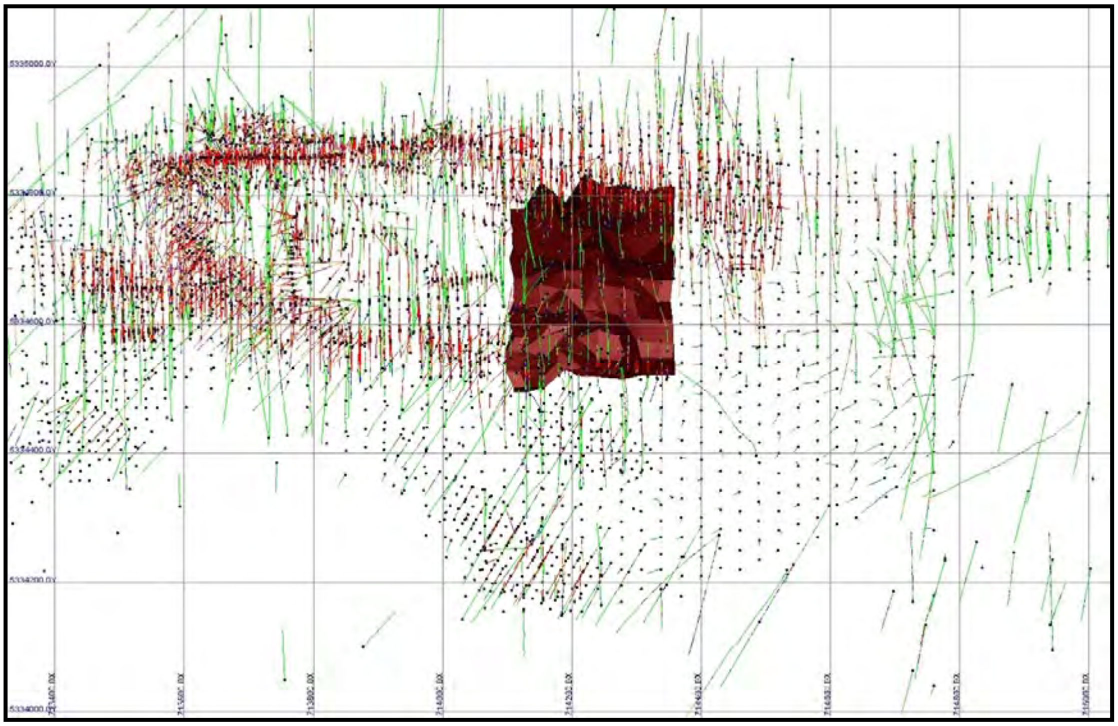


Figure 3.3: Section 714340E (Looking West)

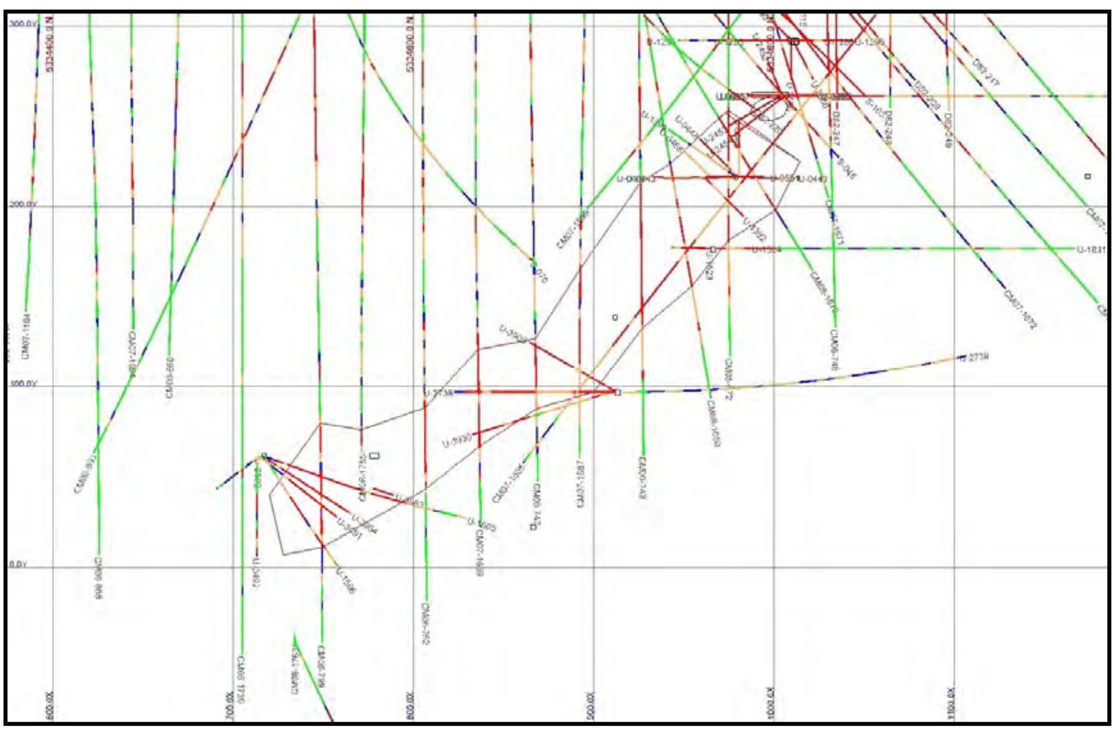
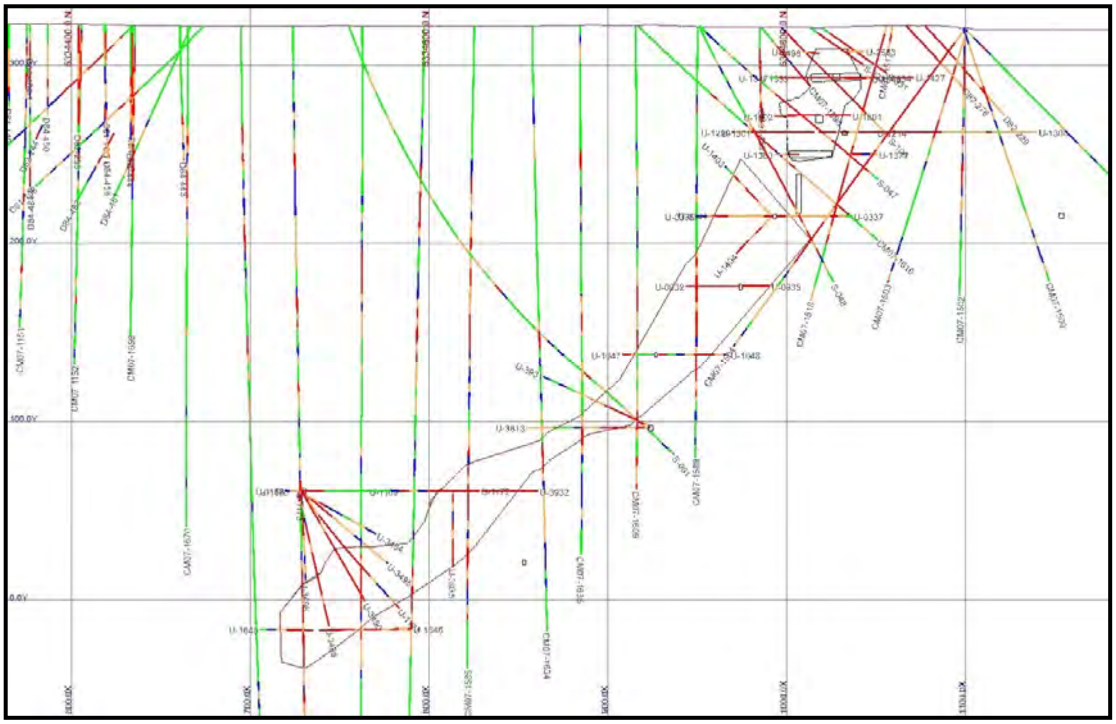


Figure 3.4: Section 714250E (Looking West)



Within the solid, 3 m composites were created starting at the intersection of the drill hole within the solid. Composites less than 3 m were rejected. The purpose of the study was to compare the statistics of the two populations (mean, median, coefficient of variation, etc).

Table 3.2 presents the statistics of the composites from historical holes (uncut and cut) and from the recent Osisko drilling. A total of only 6 composites were top cut (from the historical data) representing only 0.3% of the population and only to avoid the presence of few outliers.

As shown in Table 3.2 the mean of the Osisko drilling is 7.6% and 9.4% lower than the cut and uncut historical composites respectively. This represents a much lower discrepancy than seen when comparing the entire high grade envelope (between 25 to 30%) with its uneven distribution of older drilling.

Table 3.2: Statistics of 3 m Composites

Statistic	Historical (uncut)	Historical (cut to 10.0 g/t Au)	Osisko (uncut)
No holes	178	178	72
No composites	1,744	1,744	994
Min g/t Au	0	0	0
Max g/t Au	33.18	10.00	9.35
Mean g/t Au	1.67	1.64	1.51
Median g/t Au	1.35	1.35	1.30
Std dev g/t Au	1.66	1.29	1.08
Variance	2.76	1.67	1.16
Coeff. Variation	1.00	0.79	0.71

3.6.2.3 Comparative Study Within Small Volumes

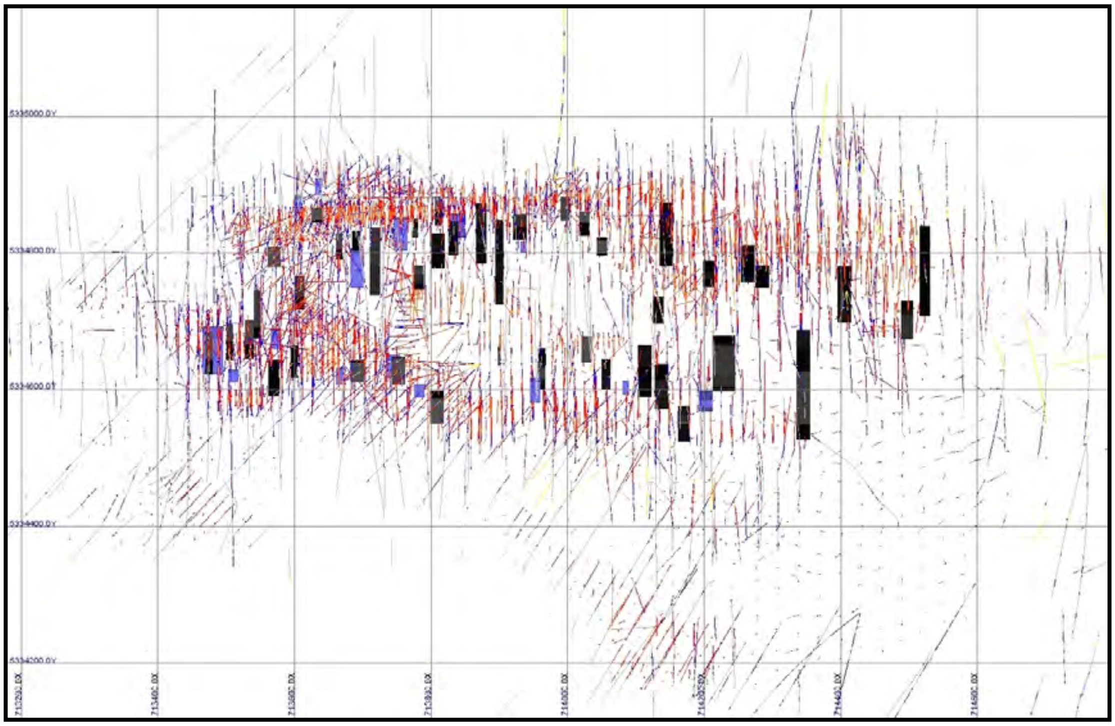
In May, 2008 comparisons between historical (Canadian Malartic) and recent drilling (Osisko) were made using smaller volumes to be able to compare data within the same geological environment. At that time, the study was done using 5 m composites

For this comparison, after looking at the drilling sections, relatively small volumes were identified in which both Canadian Malartic and Osisko assay data are present not only in close proximity but also in the same “environment”. Volumes have been identified in both high and low grade domains even if it is more difficult to identify low grade volumes since little Canadian Malartic drilling is located in the low grade domain.

A total of 43 different volumes in high grade domains and 17 in low grade domains have been identified. Some of them compared only two holes while most of them compared many holes. Usually, there are more underground Canadian Malartic holes than Osisko holes because there are many fans of underground holes starting at the same location. Additionally individual underground hole intersections are shorter than the surface holes. For the comparison, 5 m composites (uncut) were used for all drilling. From these the average of the composite grades (uncut) for the two data sets within each individual volume was calculated.

Figure 3.5 shows the location of the constructed solids (high grade in black and low grade in blue) while Figures 3.6 and 3.7 show two of the solids in a sectional view.

Figure 3.5: Plan View of the Solids



(high grade in black and low grade in blue)

Figure 3.6: Section 714350E Solid (Black) in High Grade Domain

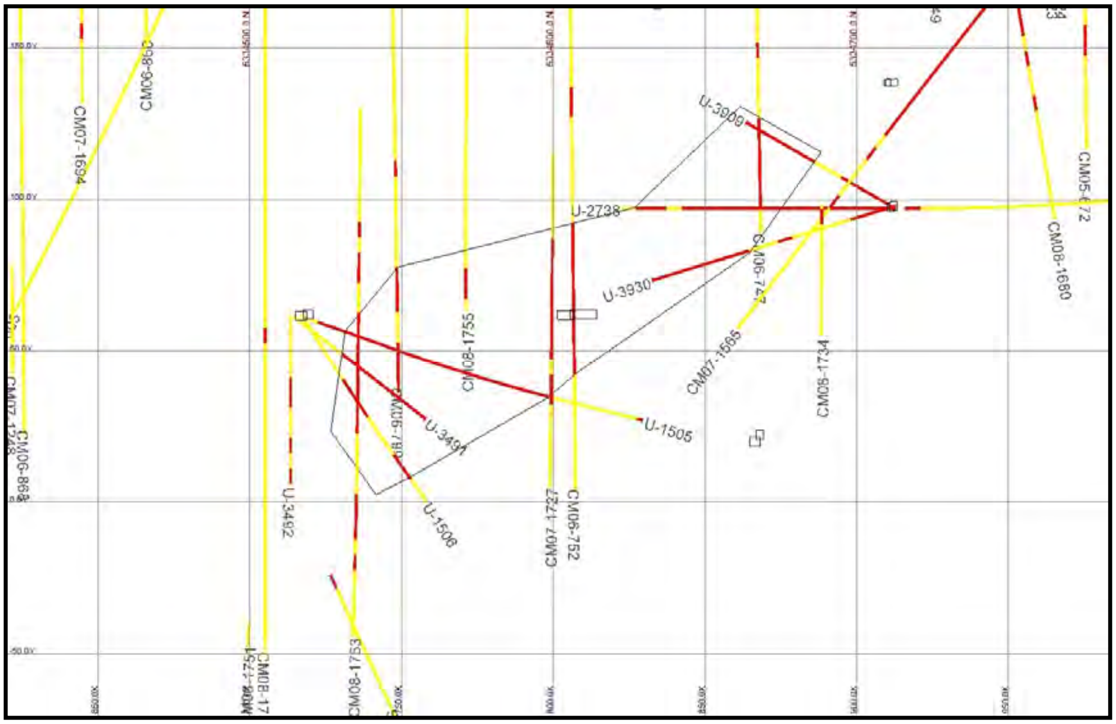
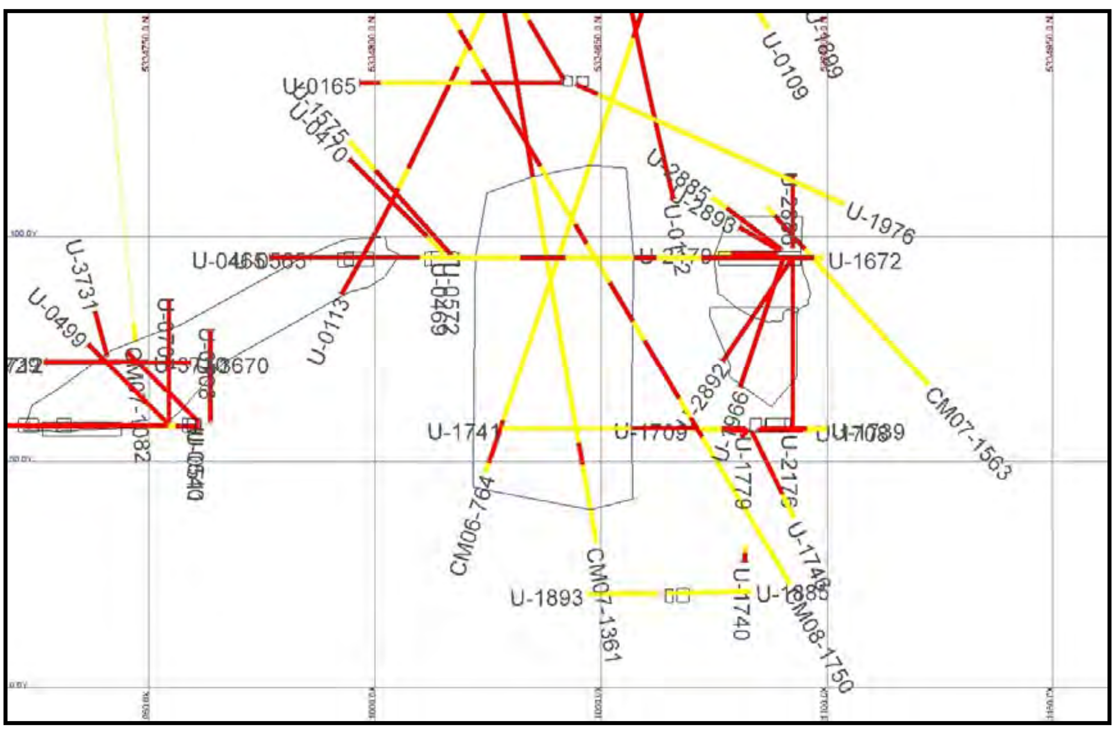


Figure 3.7: Section 713840E Solid (Blue) in Low Grade Domain



3.6.2.4 Comparison in High Grade Domains

Table 3.3 summarizes the results of the comparison of the gold grade assays for the two populations within the volumes for the high grade domain.

One can see that within the high grade material (>1.0 g/t Au), even if there are important individual discrepancies, the average grade, either mathematical or weighted by volume, is about 5% in favour of the recent Osisko drilling.

Also, the scatter plot (Figure 3.8) shows that the data are quite well distributed around the 45 degree line (red), except for one volume highly in favour of the Osisko drilling. If this “outlier” were removed the average would be the same for the two populations.

Figure 3.9 shows that there is no obvious relationship between the size of the volumes compared and the level of variance. It is interesting to observe that the highest variances (>50%) are all in favour of the Osisko drilling.

Table 3.3: Comparison Between the Original Canadian Malartic and Osisko Drill Holes in the High Grade Domains

		Historical UG holes (CM)			Recent Surface Holes (Osisko)			Variance (%)
Solid	**Volume (m^3)**	**No. of Holes**	**No. of 5 m composites**	**Average (g/t Au)**	**No. of Holes**	**No. of 5 m composites**	**Average (g/t Au)**	**Recent/ Historical**
713480-1	27,989	7	26	1.08	2	20	1.03	-4.6
713510-1	25,581	6	23	1.29	1	22	1.63	26.4
713540-1	31,832	2	16	1.23	1	11	0.94	-23.6
713540-2	38,063	1	12	1.19	5	48	1.14	-4.2
713570-1	53,176	3	18	0.89	4	56	0.76	-14.6
713570-2	19,844	6	17	1.36	1	10	1.36	0.0
713600-1	14,720	1	8	1.43	1	8	1.36	-4.9
713600-2	22,612	4	15	0.88	4	27	0.74	-15.9
713630-1	15,252	5	8	4.67	1	6	4.25	-9.0
713660-1	21,631	4	23	1.61	3	28	2.07	28.6
713690-1	7,991	3	14	2.26	1	4	3.29	45.6
713690-2	30,520	14	49	3.29	1	10	5.41	64.4
713720-1	36,058	2	9	2.31	2	9	2.58	11.7
713750-1	34,629	3	12	1.93	1	8	2.56	32.6
713780-1	7,438	1	10	2.77	1	10	2.70	-2.5
713780-2	17,763	4	14	1.67	1	9	2.09	25.1
713810-1	73,057	10	40	1.55	2	33	2.36	52.3
713810-2	26,209	3	12	1.52	1	7	2.56	68.4
713810-3	12,919	5	11	2.37	1	9	2.11	-11.0
713840-1	21,388	5	20	1.62	1	8	1.85	14.2
713870-1	32,845	5	12	1.10	4	19	1.51	37.3
717900-1	2,845	1	5	1.93	1	4	1.80	-6.7
713900-2	22,577	4	10	2.50	2	6	2.24	-10.4
713930-1	22,288	7	14	1.19	1	8	1.40	17.6
713960-1	12,605	3	17	1.97	2	10	2.22	12.7
713990-1	8,220	4	8	2.05	1	6	1.80	-12.2
714020-1	17,659	10	27	2.73	1	7	2.04	-25.3
714020-2	19,016	7	40	2.01	1	8	3.36	67.2
714050-1	7,104	5	14	2.60	1	4	2.23	-14.2
714050-2	9,234	1	4	1.58	1	3	1.12	-29.1
714110-1	37,956	2	14	1.75	1	5	1.91	9.1
714140-1	31,233	7	33	2.04	2	9	2.26	10.8
714140-2	11,076	1	5	2.73	1	4	2.18	-20.1
714140-3	108,999	8	36	2.01	2	23	1.70	-15.4
714170-1	24,530	6	35	2.15	2	14	1.91	-11.2
714200-1	13,574	1	5	1.78	2	12	2.15	20.8
714230-1	111,166	7	59	1.75	2	18	1.57	-10.3
714260-1	26,408	4	11	1.79	1	14	1.48	-17.3
714290-1	14,332	3	11	1.91	1	8	1.90	-0.5
714350-1	158,974	6	48	1.62	3	25	1.58	-2.5
714410-1	56,158	4	20	1.82	3	17	1.17	-35.7
714500-1	26,149	2	8	1.53	2	10	1.23	-19.6
714530-1	110,534	10	33	3.56	3	35	3.92	10.1
Volume weighted avg.	**1,424,154**	**197**	**826**	**1.90**	**75**	**612**	**2.00**	**5.2**
Math. average				**1.93**			**2.03**	

Figure 3.8: Scatter Plot of Comparative Assays in High Grade Domains

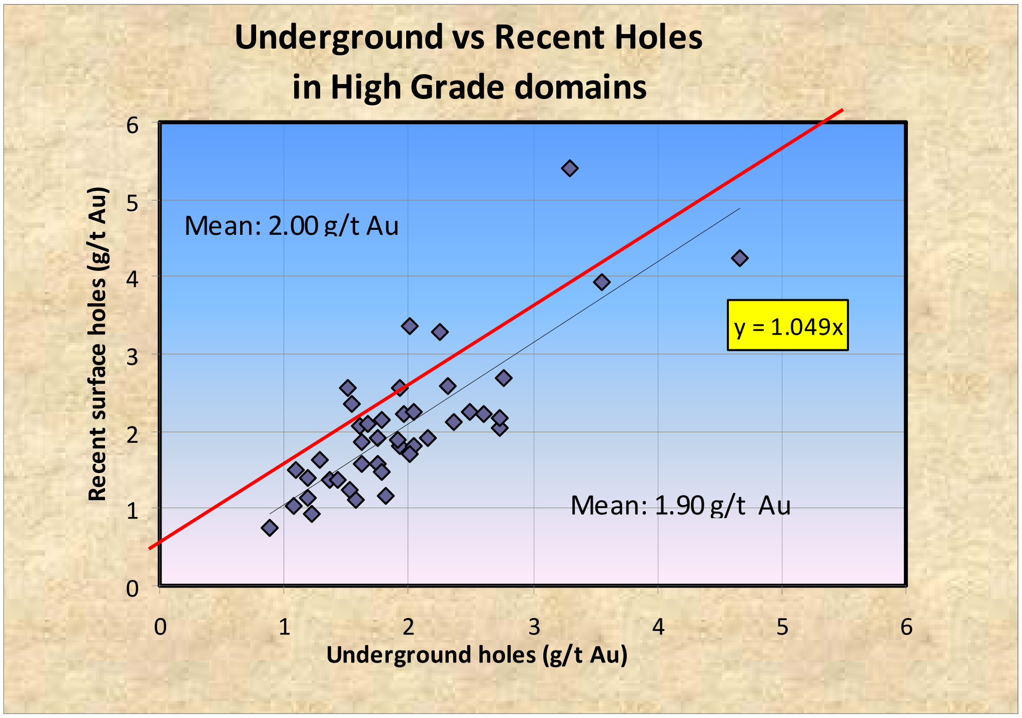
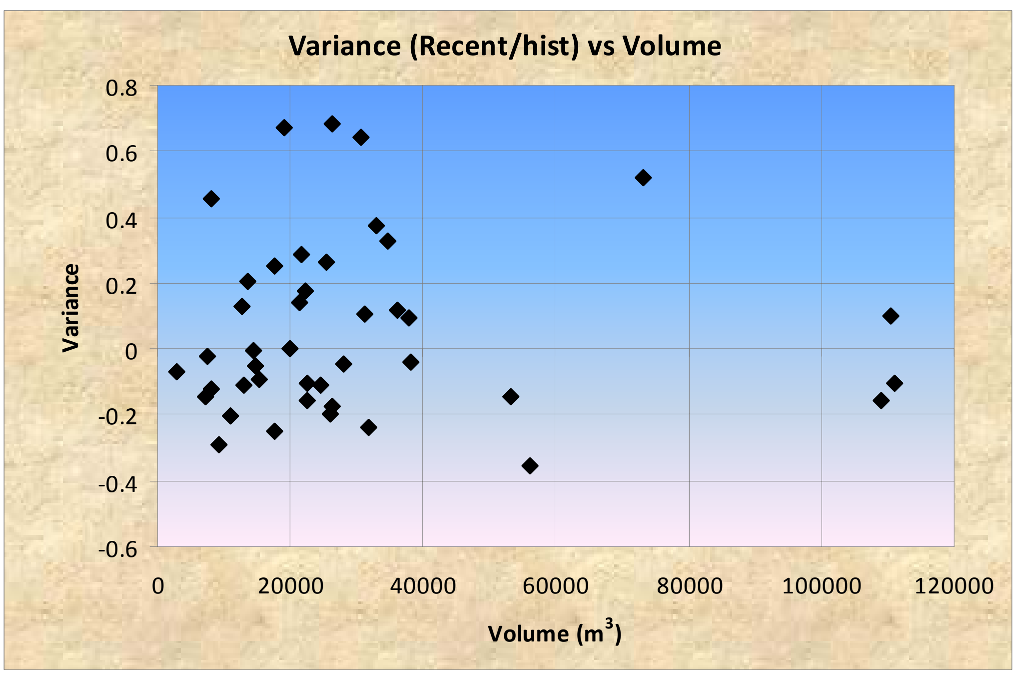


Figure 3.9: Relationship Between Variance and Volume Size in High Grade Domains



3.6.2.5 Comparison in Low Grade Domains

The same exercise was completed for volumes identified in the low grade domains. Table 3.4 summarizes the results of the comparison of the gold grade assays within the volumes for the low grade domains.

For the volumes in this domain, the result is very different. It can be seen that almost all the comparisons are in favour of the original Canadian Malartic drilling. On average, the mathematical and volume weighted averages of the original Canadian Malartic drilling are respectively 32% and 24% higher than the Osisko drilling.

Table 3.4: Comparison Between the Original Canadian Malartic and Osisko Drill Holes in the Low Grade Domains

		Historical UG holes			Recent surface holes			Variance (%)
Solid	**Volume (m^3)**	**No. of Holes**	**No. of 5 m composites**	**Average (g/t Au)**	**No. of Holes**	**No. of 5 m composites**	**Average (g/t Au)**	**Recent/ Historical**
713480-1	114,315	3	22	0.50	3	35	0.40	-20.0
713510-1	4,726	1	4	0.35	1	4	0.20	-42.9
713570-1	12,702	1	6	0.46	1	6	0.27	-41.3
713600-1	15,521	3	11	0.40	1	14	0.19	-52.5
713630-1	16,142	2	8	0.26	1	3	0.31	19.2
713660-1	3,331	1	7	0.46	1	9	0.33	-28.3
713690-1	61,648	2	20	0.24	2	24	0.25	4.2
713750-1	46,340	3	22	0.51	2	22	0.40	-21.6
713750-2	19,744	1	6	0.48	2	13	0.36	-25.0
713780-1	4,328	1	5	0.41	1	4	0.11	-73.2
713810-1	10,858	2	9	0.58	1	7	0.55	-5.2
713840-1	62,154	2	13	0.59	2	30	0.43	-27.1
713930-1	8,913	1	4	0.52	1	3	0.36	-30.8
713960-1	2,693	1	7	0.58	2	7	0.46	-20.7
714080-1	3,720	1	5	0.19	1	4	0.07	-63.2
713170-1	6,489	1	7	0.27	1	6	0.02	-92.6
713200-1	26,681	3	16	0.38	2	20	0.18	-52.6
Based on volume	**420,305**	**29**	**172**	**0.45**	**25**	**211**	**0.34**	**-23.9**
Math. average				**0.42**			**0.29**	**-31.9**

The scatter plot of the results (Figure 3.10) shows that almost all the points are below the 45 degrees line. It can also be noted that the highest variances seem to be associated with the smaller volumes (Figure 3.11).

Figure 3.10: Scatter Plot of Comparative Assays in low Grade Domains

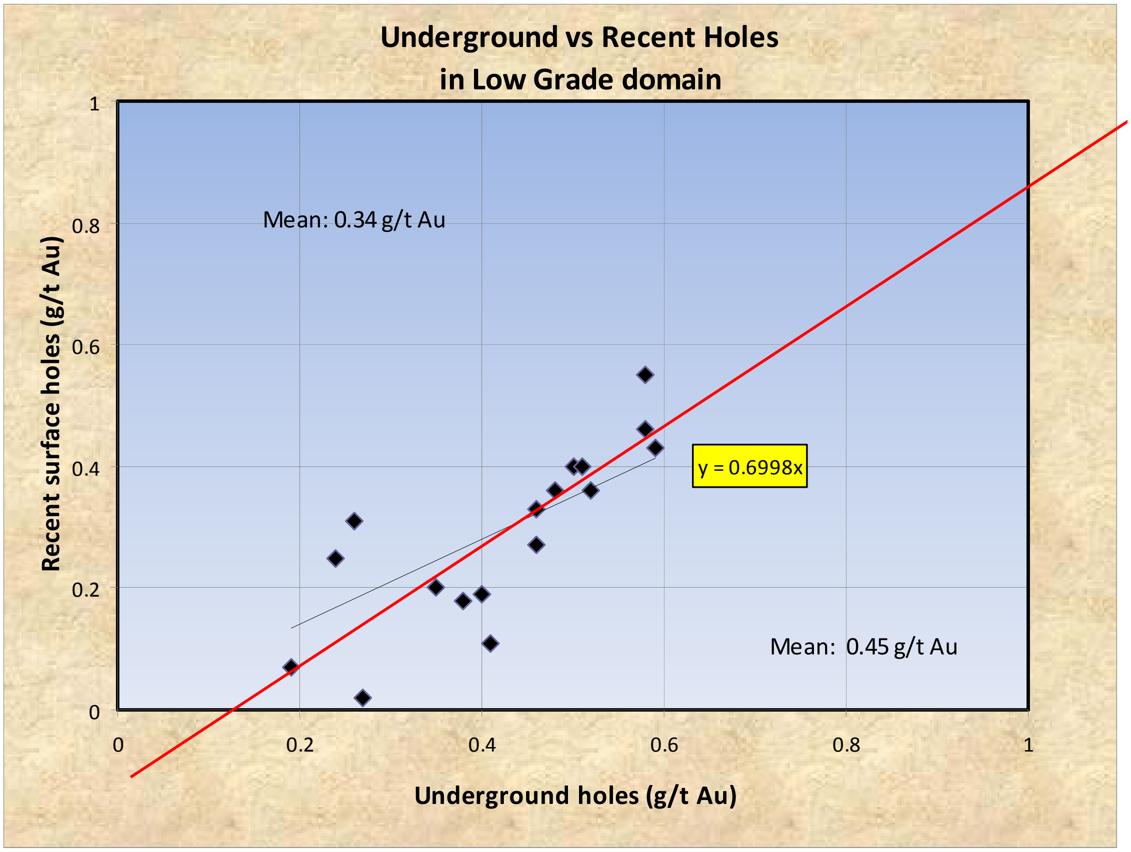
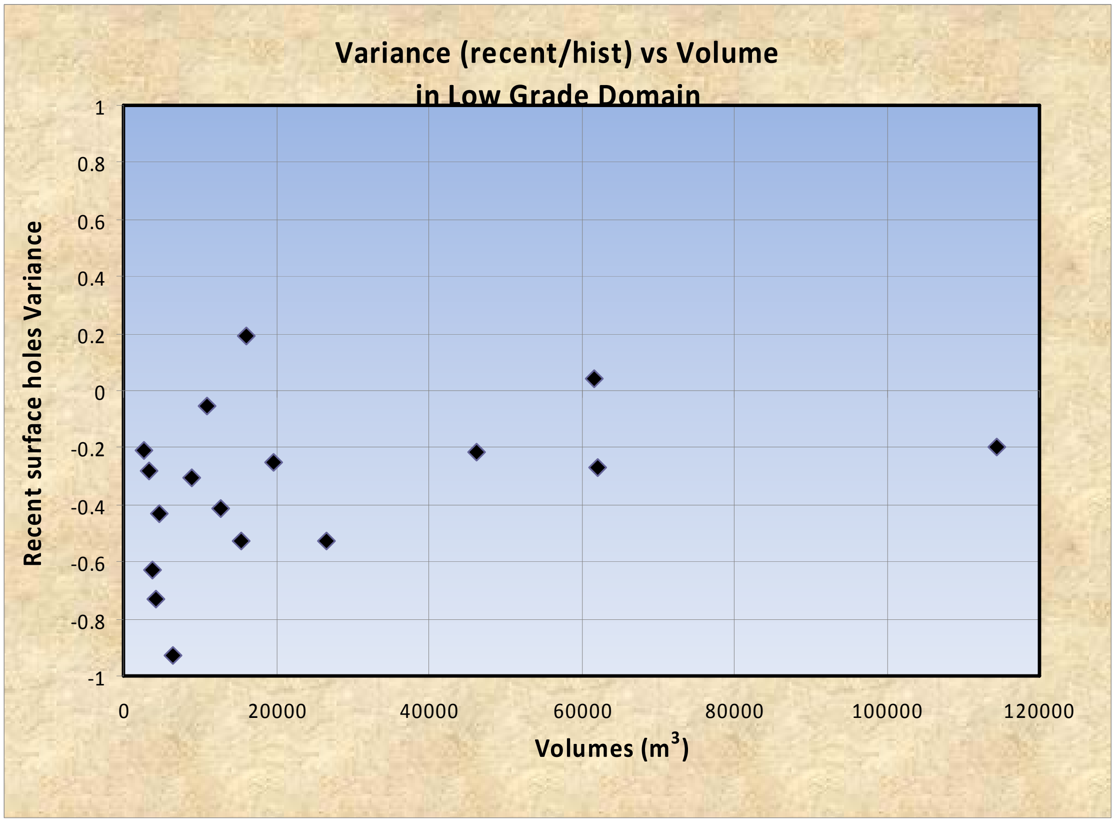


Figure 3.11:Graph Showing Relationship Between Variance and Volume Size in Low Grade Domains



3.6.2.6 Conclusions on Comparisons of Historical and Recent Data

The comparative study of gold grade between the Canadian Malartic and Osisko drilling clearly identifies a bias between the two populations in the low grade domains. It is of course difficult to quantify the bias but the discrepancy seems to be significant, a minimum of 25% to 30%. The historical data may be affected by the fact that the detection limit for gold assays was higher at the time and a lot of assays between 0.20 and 0.60 g/t Au are not consistent with recent drilling and assaying.

For the high grade material (envelopes >1.0 g/t Au), the situation is different. Even if, within the global high grade envelopes, the discrepancy is in favour of the historical data, there is little evident bias when the comparison is done within volumes in the same geologic environment. Within small volumes, the average of the composites is about the same or even in favour of the recent drilling.

In conclusion, the result of the comparative studies led to the rejection of all historical data in the low grade domains and the pervasive alteration domains. In these domains, only the Lac Minerals and Osisko drilling data were used for grade interpolation and resource estimation.

For the high grade domains (> 1.0 g/t Au), as there is no evidence of bias for samples in the same geological environment, the historical data were used for resource estimation along with the Lac Minerals and Osisko data.

3.6.3 Survey Control

The survey control on recent drilling has been well established. A detailed topographic surface has been generated based on ground surveys and is considered sufficient for the current level investigations. Recent detailed survey methods and results are also considered to be industry standard. Less survey control is available for historical data which significantly lowers the confidence in some of these data. A portion of the historical drilling was not used for this reason.

3.6.4 Drill Hole Recovery

Core recovery data are available for the Osisko series of drilling completed since 2005. About 96% of the core has recorded minimum recoveries of 96% to 97%. The core recoveries at the Canadian Malartic deposit are considered excellent overall and appropriate for use in the resource estimation completed herein.

The correlation between core recovery and gold values for the raw gold grades was analyzed for the Osisko series of drill holes within the low grade mineralized halo. No relationship was evident between the core recoveries and gold assay data, indicating that the data were not biased by recoveries.

3.6.5 Analysis of Assay QA/QC Data

RSG completed a detailed analysis of QA/QC results available and used for its August, 2007 resource estimate. The results are described in Gossage, 2007b. RSG summarized its findings as follows:

“In summary, based on the available standards assay quality control data, RSG Global concludes that the standards sent to Chemex are within industry acceptable limits of accuracy. The blank samples do not display evidence for significant contamination.”

"Internal laboratory standards and blanks provided by Chemex also are within industry acceptable limits of accuracy. The check assay programme adequately reproduced the original assays and RSG Global considers the original assay data to be a reasonable representation of the sampled core."

"For future QA/QC studies, collated Osisko duplicates (coarse repeats and field duplicates), umpire assaying and Laboratory standards, duplicates and repeats should be obtained and analysed".

Micon has reviewed the subsequent QA/QC results from the Chemex internal control samples and the blanks, standards and duplicates submitted by Osisko. It has discovered no material problems which would prevent the data being used for a mineral resource estimate.

3.6.6 Check Sampling

3.6.6.1 RSG Sampling

RSG completed a program of check sampling of original Osisko core in January, 2007. RSG presented its findings as follows:

- "RSG Global supervised a check sampling programme at the Canadian Malartic Project, Québec, Canada, completed on January 23, 2007. Randomly selected intervals of diamond drill core from the Canadian Malartic Project were selected by RSG Global. Quarter core from these intervals were collected with the core cutting, sampling and sample submission completed by Osisko technical staff and their contractors under the supervision of RSG Global. All aspects of the sampling and assaying were completed as per the Osisko standard protocols."
- "The check sampling programme consisted of 19 check intervals, collected from four drill holes, plus two certified standards. The intervals were selected to ensure a representative mix of lithology and alteration were considered, and that a reasonable grade range was investigated."
- "The following lithology / alteration types were sampled:"
  - "CM05-686 - Silicified or partly silicified schist"
  - "CM06-700 - Schist (+/- chlorite / carbonate alteration)"
  - "CM06-767 - Silicified or partly silicified schist"
  - "CM06-790 – Altered porphyry".

"Table 4.7.2.3 (Table 3.5 in this report) presents the sampled intervals and the results of the check assaying".

Table 3.5: RSG Global Check Assay Program Intervals and Results

Hole ID	From (m)	To (m)	Sample Number	Check Sample Number	Au (ppm)	Au Check (ppm)	Sample Type
CM05-686	145.5	147.0	142903	142903A/ML45952	5.550	5.000	Duplicate
CM05-686	147.0	148.5	142904	142904A/ML45953	3.740	3.700	Duplicate
CM05-686	148.5	150.0	142905	142905A/ML45954	2.290	2.170	Duplicate
CM05-686	150.0	151.0	142906	142906A/ML45955	1.480	1.685	Duplicate
CM05-686	151.0	152.0	142907	142907A/ML45956	1.115	0.880	Duplicate
CM06-700	100.1	101.6	144119	144119A/ML45957	2.240	2.430	Duplicate
CM06-700	101.6	103.1	144120	144120A/ML45958	7.600	7.630	Duplicate
CM06-700	103.1	104.6	144121	144121A/ML45959	5.740	6.000	Duplicate
CM06-700	104.6	106.1	144122	144122A/ML45960	4.790	4.410	Duplicate
CM06-767	231.3	232.8	224767	224767A/ML45945	0.103	0.102	Duplicate
CM06-767	232.8	234.3	224768	224768A/ML45946	0.480	0.982	Duplicate
CM06-767	234.3	235.8	224769	224769A/ML45947	1.090	0.598	Duplicate
CM06-767	284.5	285.3	224806	224806A/ML45948	2.660	2.690	Duplicate
CM06-767	285.3	286.8	224807	224807A/ML45949	1.370	1.450	Duplicate
CM06-767	286.8	288.3	224809	224809A/ML45950	0.693	0.667	Duplicate
CM06-790	311.5	313.0	226869	226869A/ML45941	3.190	2.590	Duplicate
CM06-790	313.0	314.5	226870	226870A/ML45942	1.705	2.110	Duplicate
CM06-790	314.5	316.0	226871	226871A/ML45943	0.218	0.191	Duplicate
CM06-790	316.0	317.5	226872	226872A/ML45944	1.450	1.295	Duplicate
CM05-686	145.5	147.0	142903	142903S/ML45951	*1.326	1.305	Standard
CM06-700	104.6	106.1	144122	144122S/ML45961	*0.996	1.005	Standard

Note: * represents the certified standard value

3.6.6.2 Micon Sampling

During the August, 2008 site visit Micon completed its own program of check sampling of duplicate half core (field duplicates) from a number of drill holes at a variety of grade ranges throughout the Canadian Malartic deposit. A total of ten samples were taken and submitted to the Chemex laboratory for analysis. The samples were prepared and assayed using the same protocols as employed by Osisko. Micon maintained custody of the samples until they were handed over to Chemex staff at the laboratory receiving door.

The results of Micon's field duplicate sample program can be seen in Table 3.6 below.

Table 3.6: Micon Check Sampling Program and Results

Hole-ID	From (m)	To (m)	Original Sample No.	Check Sample No.	Original (g/t Au)	Check (g/t Au)
CM08-1754	46.00	47.50	692608	330401	1.975	1.980
CM07-1710	291.00	292.50	722328	330402	4.460	3.790
CM08-1789	260.50	262.00	726957	330403	0.963	0.955
CM08-1823	117.50	119.00	800525	330404	2.040	1.840
CM07-1674	83.00	84.50	691388	330405	3.810	3.530
CM08-1691	235.00	236.50	642535	330406	7.870	7.960
CM07-1649	81.00	82.50	642008	330407	1.485	0.845
CM08-1875	136.00	137.50	744492	330408	4.660	3.050
CM08-1847	96.50	98.00	801507	330409	1.490	1.095
CM07-1614	68.00	69.50	658749	330410	3.080	2.670
				Mean	**3.183**	**2.772**
				Difference		**-12.9%**

Micon considers the assays results returned to be an acceptable variance for field duplicate samples. The low variance is consistent with the low coefficient of variation seen in the database and the apparent lack of nugget effect in the deposit.

3.6.7 Summary

It is Micon's opinion that the Canadian Malartic drill hole database is robust and suitable enough for use in mineral resource estimation studies.

4.0 MINERAL RESOURCE ESTIMATES

4.1 Data

Three different distinct phases of drilling have occurred at the Canadian Malartic project, the historical drilling by the Canadian Malartic mine, programs carried out by Lac Minerals-Barrick and the drilling by Osisko. The statistics for the three phases are summarized in Table 4.1.

Table 4.1: Drilling and Sampling Statistics

Company	Dates	No. of Holes	Length (m)	No. of Samples	Average Length (m)
Canadian Malartic	1928 - 1963	3,838	159,056	128,618	1.18
Lac Minerals-Barrick	1987 - 1990	630	69,738	34,660	1.51
Osisko	2005 - 2008	1,327	316,230	198,173	1.44
Total		**5,795**	**545,024**	**361,451**	**1.35**

All the data used for the resource estimate presented herein are derived from this drilling database. The drill hole density is quite sufficient to develop a reasonable picture of the distribution of mineralization, and to quantify its volume and quality with a good degree of confidence.

The core size used is different for the three main phases of drilling. While Osisko recently used NQ-sized core (47.6 mm) and Lac Minerals BQ-sized (36.5 mm), it is uncertain what the core dimension of the Canadian Malartic drilling was. Considering the time at which this drilling occurred, the core size was likely to be AX (30 mm) and possibly less at the early stages of that program.

4.2 Modeling

Osisko provided Micon with completed modeling of the different mineralized zones and the porphyry unit, a 3D wireframe of the surface topography and of the interpreted base of the overburden. The different surfaces and solids were reviewed and judged reasonable for the interpretation of the deposit and, therefore, can be used for resource estimation.

To ensure a constant direction and dip of the mineralization, the mineralized zones were divided into sub-domains for variographic studies and subsequent grade interpolation.

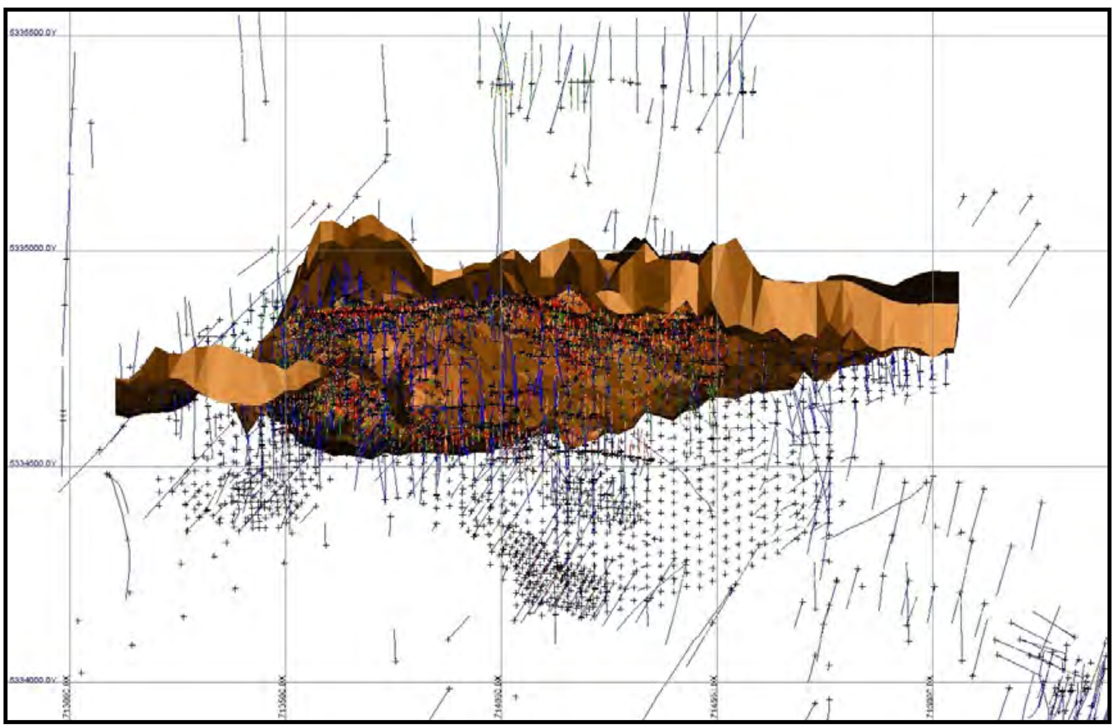
4.2.1 Porphyry Unit

In the geological logging available, there is no distinction between the different types of sediments. Therefore, the only geological units that can be modeled close to the deposit are the porphyry units that intrude the sediments.

The 3D model of the main porphyry unit was based on interpreted sections and geological logging only from the Osisko drilling program. Figure 4.1 shows the extent of the modeled porphyry unit.

This unit is not the only porphyry present in the area. There are many additional small porphyries but they are not easy to model since geological information is only present in the Osisko geological logging. Geological information from Lac Minerals and Canadian Malartic drilling is not available in the database. The modeling would benefit from this information if it would be available in electronic format.

Figure 4.1: Plan View Showing Extent of the Main Porphyry Unit



Mineralization is present in both porphyry and sedimentary rocks but the presence of the porphyry is important since the main high grade zones are located very close to or at the contact between sediments and porphyry.

4.2.2 Mineralization Modeling

Based on a cut-off of 1.0 g/t Au, five different high grade domains were modeled along with one pervasive alteration envelope (cut-off 0.4 g/t Au) and one broader low grade envelope (based on a 0.2 g/t Au cut-off) that defines the extents of the mineralization.

The pervasive alteration envelope is based on the intensity of the rock alteration (porphyry and sediments) and also on the mineralization. Rocks are altered in the high grade zone but alteration is also associated with lower grade material (between 0.4 and 1.0 g/t Au), especially to the east side of the deposit. The continuity of the mineralization associated with alteration is good and can be differentiated from the wider low grade halo (0.2 g/t Au).

All the mineralized zones are included within the larger, low grade envelope. Figures 4.2 and 4.3 show the location of the mineralized zones (isometric and plan views) and Figures 4.4 and 4.5 are two cross-sections (north-south) through the deposit.

Figure 4.2: Isometric View Showing the Interpreted Mineralized Zones

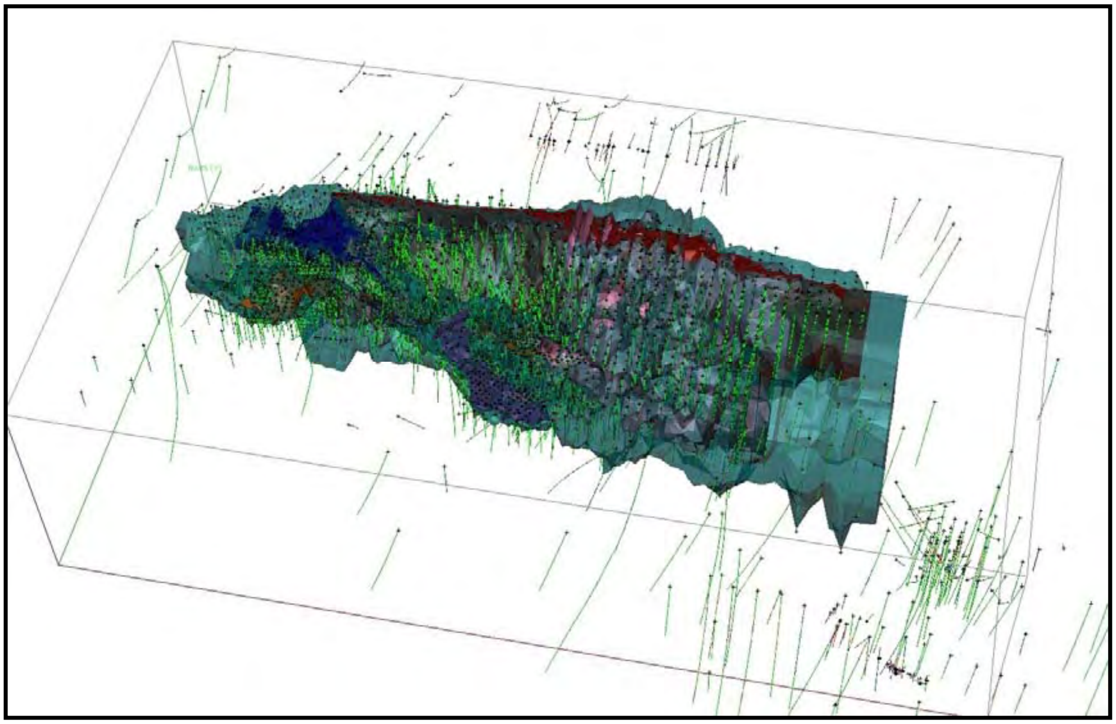


Figure 4.3: Plan View Showing the Interpreted Mineralized Zones

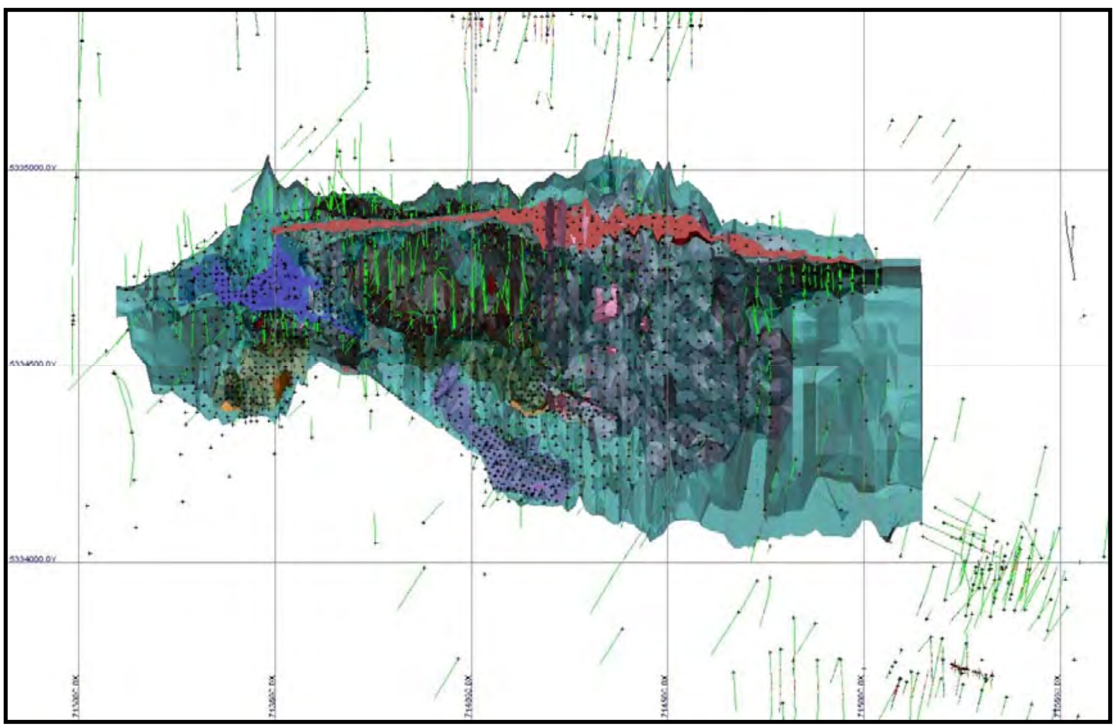


Figure 4.4: Sectional View (713680E)

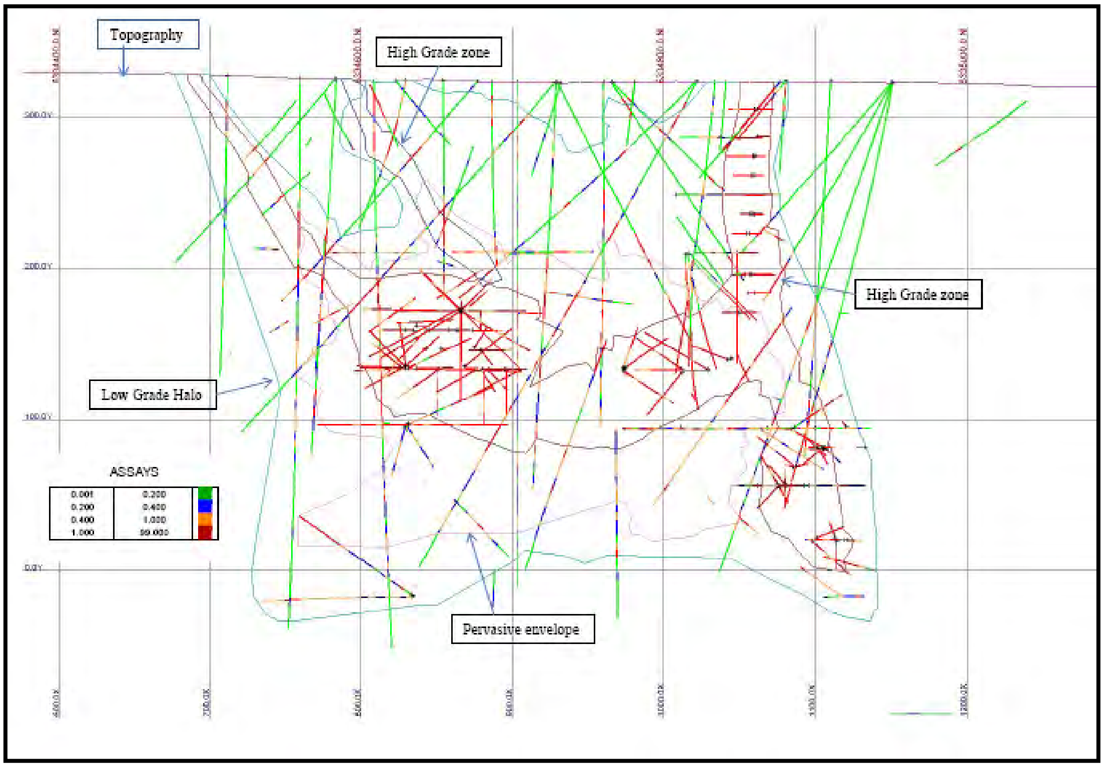
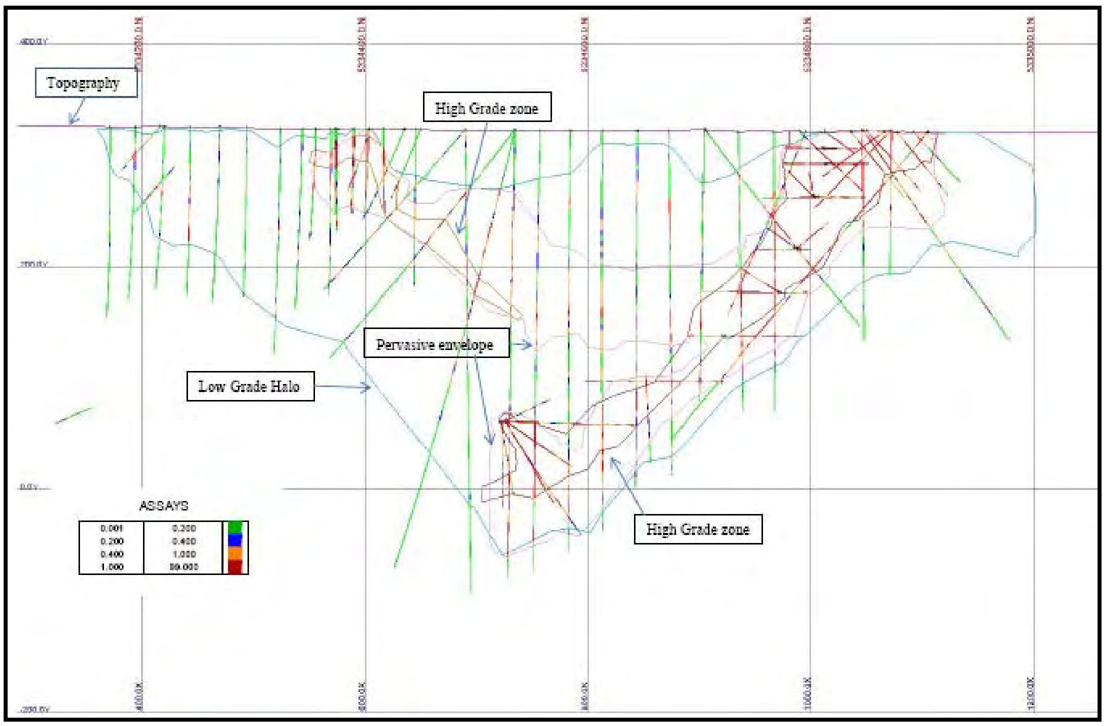


Figure 4.5: Sectional View (714280E)



4.2.3 Surface Topography and Overburden

Osisko provided a 3D wireframe model of the surface topography based on a topographic pick up of the central area of the project and drill hole collars in the areas beyond the extents of the pickup. The model was reviewed and showed a good agreement with the location of the drill hole collars.

A 3D wireframe that defines the base of overburden in the area was also provided by Osisko. This surface is based on the information taken from drill logs. As the base of overburden is not always clearly identified in the logs, this surface is interpretive and is not as accurate as the topographic surface. Nevertheless, as the overburden thickness is predominantly less than a few metres (2 to 3 m), the interpretative nature of the surface is not considered material for this study (the resource estimation) but would need to be refined for detailed studies.

4.3 Underground Development and Stoping

For the geological modeling, Osisko provided Micon with 3D dxf files of the development and stoping of the former Canadian Malartic and Sladen Mines (Figure 4.6). The model was reviewed and compared with the surrounding drill hole information, especially where recent Osisko surface drilling intercepted open stopes. Generally, the openings seem to fit reasonably well with the recent drilling (Figures 4.7 and 4.8).

It can be observed that the vast majority of the underground drill hole collars are located within drives and stopes. Nevertheless, it seems that some sub-levels do not feature in the supplied DWG file since some hole collars are located where there is no development. These sub-levels should not represent an important volume of voids for the resource estimate but further investigation is important to confirm that all the underground workings have been identified from the available information.

Based on the compilation of the percent model in Gemcom, the total volume of the excavations is slightly higher than 5 Mm^3 (5,086,761 m^3).

Figure 4.6: Isometric View of the Underground Workings

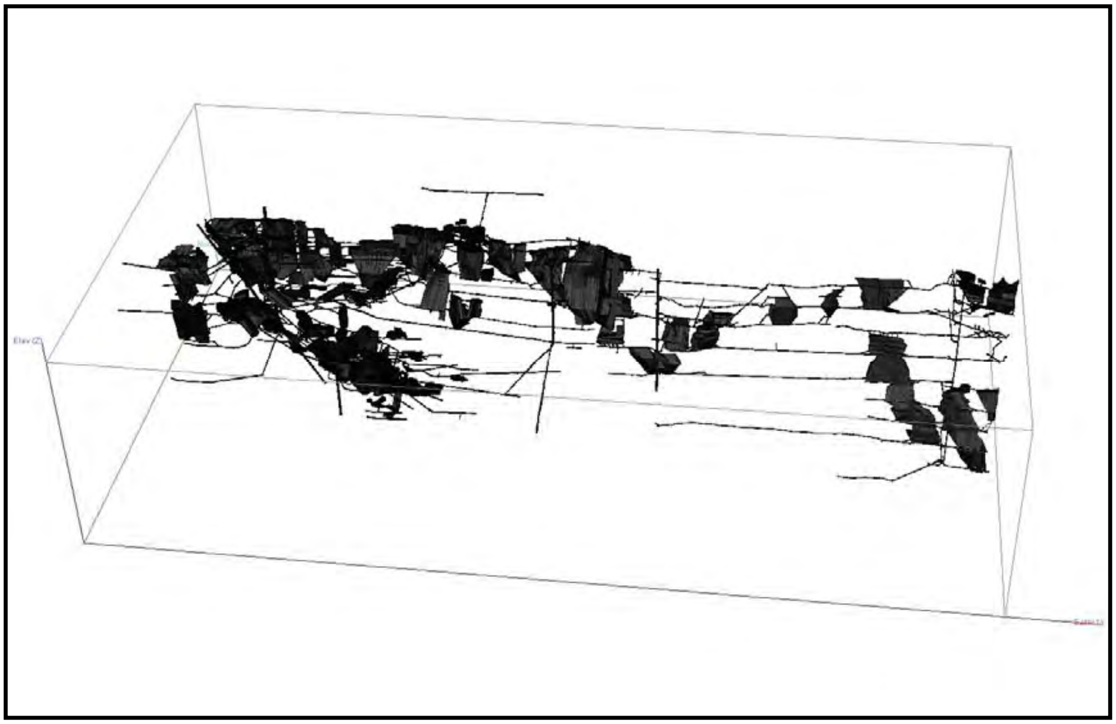


Figure 4.7: Section 713740E Showing Recent Drill Holes Intersecting Old Workings

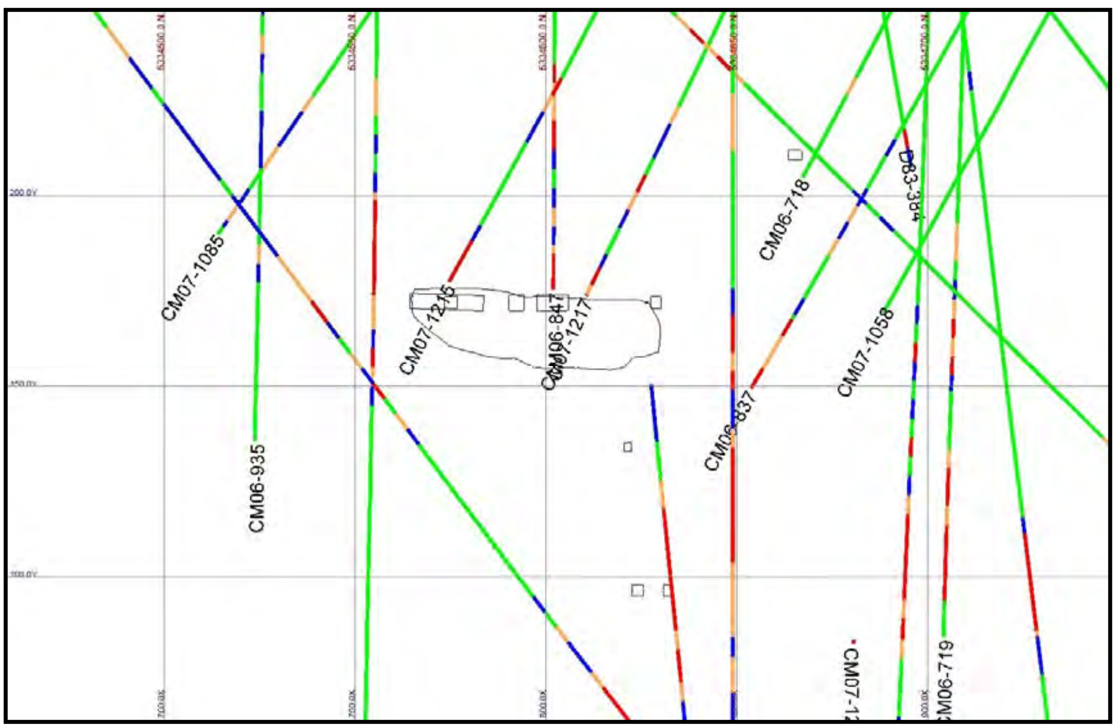
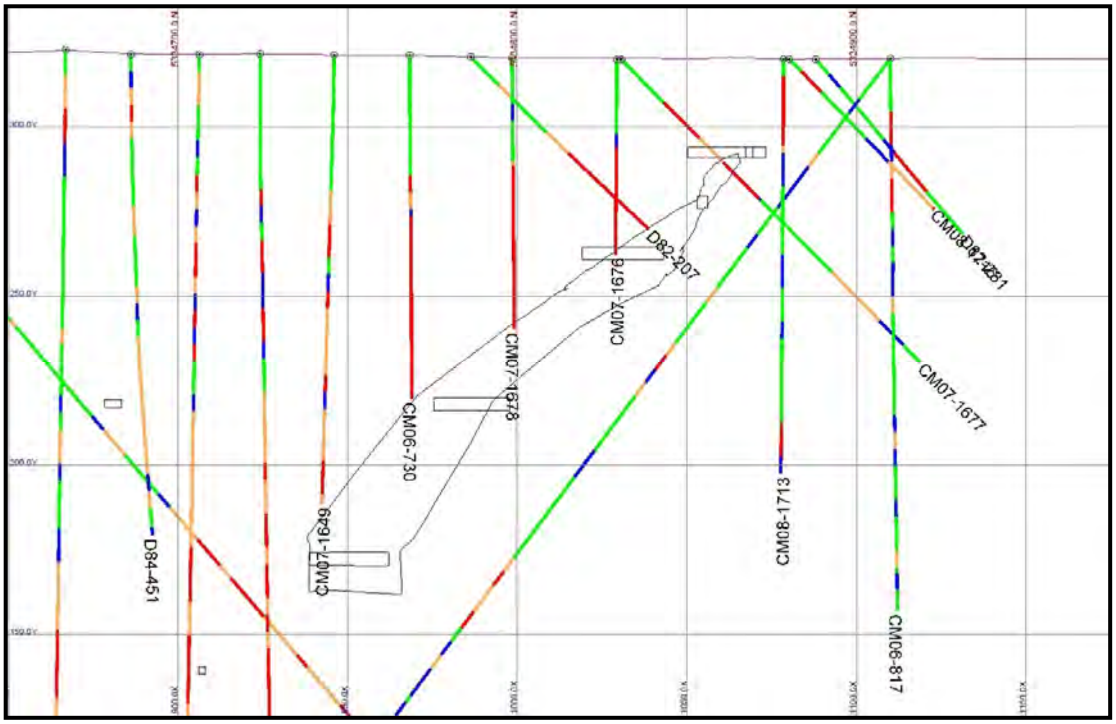


Figure 4.8: Section 714460E Showing Recent Drill Holes Intersecting Old Workings



For the resource estimate, the stope volumes were expanded by 1.0 m to take into account the fact that some openings could still be unknown at the moment and that there could be some caving during and after the mining operation that is not represented in the plans of the mines.

The development and the expanded stopes were used to deplete the resource block model. All assay data within the openings were coded as voids and therefore completely excluded from use in the resource estimation process.

4.4 Statistical Analysis

Drill hole assay intervals intersecting interpreted domains and underground voids were coded in the database (by solid precedence) and used to analyze sample lengths, generate composites, statistics and variography.

4.4.1 Compositing

As the original database is a mix of samples of different assay lengths (from 0.2 m to up to 2.0 m), it was judged to be better to standardize the assay lengths before performing statistical analyses.

Even if almost all drill holes have a north-south direction, they are drilled at a wide variety of dip angles, ranging from horizontal (underground holes) to -90^o for most of the Osisko drilling. It is therefore difficult to have an equal support, even with equal lengths. Nevertheless, because of the size of the potential operation that would use 10 m benches, it was decided to composite the data with a regular 5.0 m run length (down-hole) within each interpreted domain. Composites of less than 2.5 m were excluded from the database. Unsampled intervals were given a grade of 0.001 g/t Au while samples below detection limit were given 0.002 g/t Au.

4.4.2 Statistics

Descriptive statistics, histograms and probability plots were compiled for each of the mineralized domains. These were used to assess the statistical characteristics of the datasets and help in the selection of the high grade assay cut-off, if necessary. All statistical analyses have been completed based on the 5.0 m composites.

Descriptive and distribution statistics of the 5.0 m composites were generated and grouped by mineralized domain. The grade datasets for the various estimation domains are characterized by a generally low (in high grade zones) to moderate (in low grade zones) coefficient of variation (CoV) for a gold deposit. This indicates that high grade values contribute moderately to the mean grades (compared to average gold deposits), except in the low grade domains where high grade is more erratic.

The effect of the higher grade composites was investigated along with probability and histogram plots. Based upon this analysis, it was decide to apply a top cut to the assays in most of the domains, even if the impact is not really material. Real outliers are rare and the maximum loss of gold after cutting represents 2.35% of the metal content in one of the low grade domains.

Examples of histograms and probability plots are shown for sub-domain 22 (main high grade zone) in Figures 4.9 and 4.10 while Table 4.2 shows the summary statistics within the domains for uncut and cut values.

It must be noted, as explained earlier, that all historical data from the original Canadian Malartic drilling have been removed from the composite database for the low grade and pervasive domains.

Figure 4.9: Log Normal Histogram - Zone 22

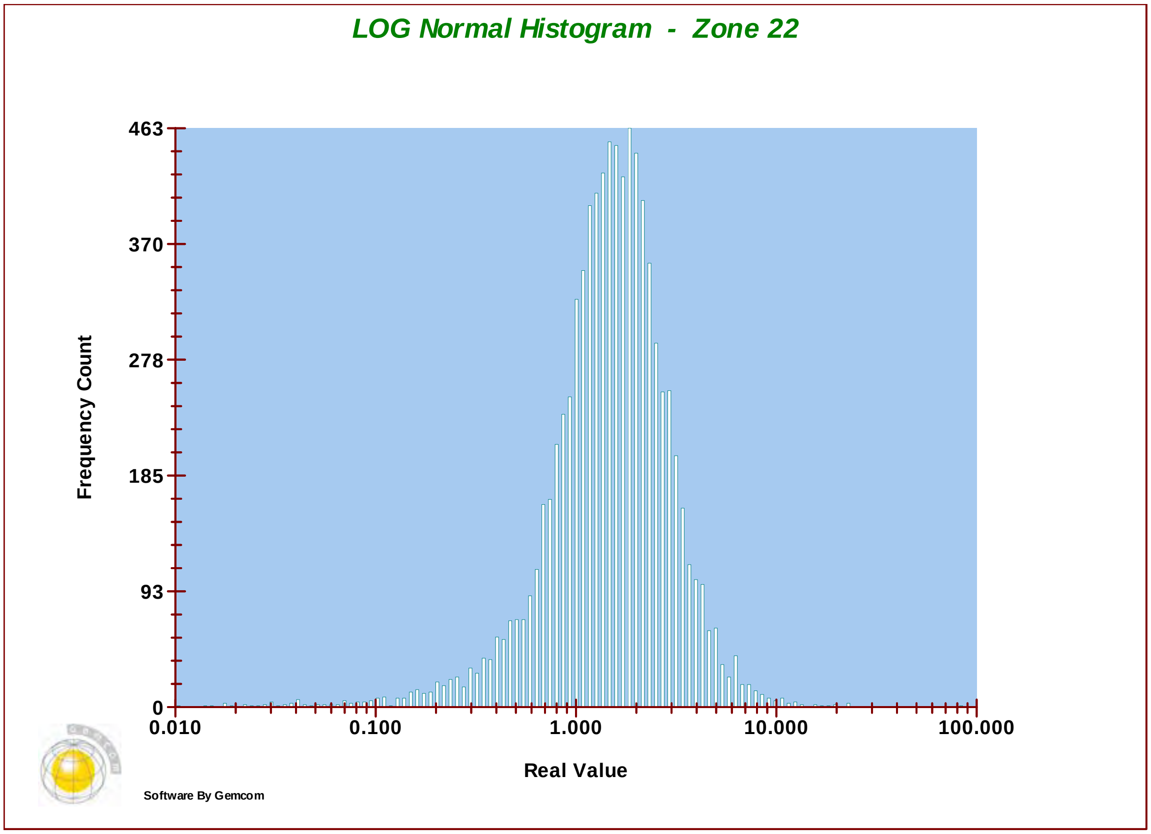


Figure 4.10: Log Normal Probability Plot - Zone 22

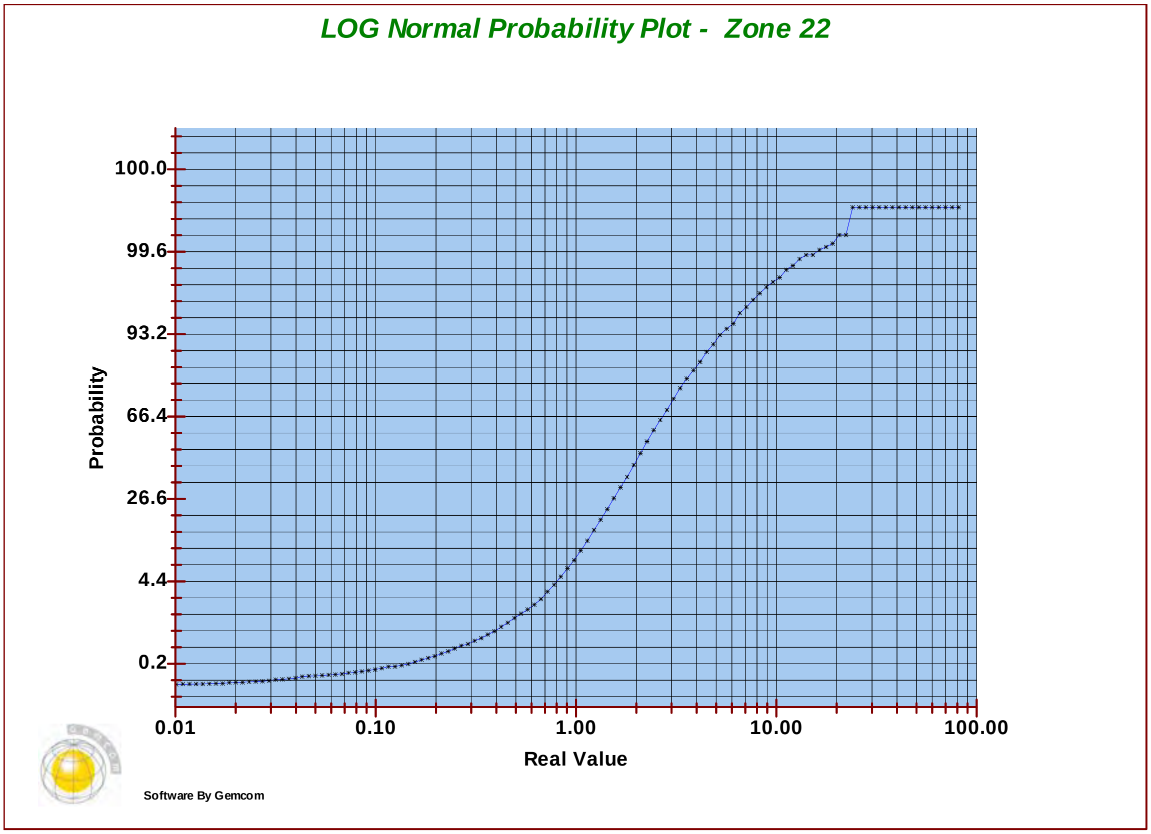


Table 4.2: Summary Statistics of 5 m Composites by Zone (Sub-Domains)

Domain	Sub-Domain	Number of 5 m Comp.	Max Value (g/t Au)	Uncut Mean (g/t Au)	Std. Dev.	CoV	Upper Cut (g/t Au)	Cut Mean (g/t Au)	Cut Std. Dev.	Cut CoV	No. of Data Cut	% Loss Metal Content
Low Grade Zones (10)	11	400	2.43	0.32	0.34	1.05	No	0.32	0.34	1.05	0	0.00
	12	7,787	22.07	0.28	0.60	2.15	6.00	0.27	0.47	1.75	11	2.35
	13	3,245	11.19	0.38	0.63	1.67	7.00	0.38	0.61	1.60	2	0.53
	14	1,137	3.00	0.27	0.40	1.44	2.00	0.27	0.37	1.37	8	1.74
	15	9,790	33.99	0.26	0.53	2.05	5.00	0.26	0.41	1.60	5	1.19
	16	6,238	8.70	0.31	0.50	1.6	6.00	0.31	0.49	1.57	5	0.30
High Grade Zones 20-30 40-50 60	21	1,041	21.57	1.75	1.73	0.99	12.00	1.73	1.54	0.89	6	1.35
	22	8,705	87.76	1.83	1.68	0.92	15.00	1.82	1.35	0.74	10	0.73
	23	1,183	46.38	2.27	2.21	0.97	15.00	2.24	1.83	0.82	2	1.20
	24	2,348	41.62	2.03	2.54	1.25	22.00	2.01	2.30	1.14	5	1.08
	26	2,628	42.09	1.97	1.89	0.96	15.00	1.95	1.60	0.82	7	1.17
	31	295	8.65	1.83	1.21	0.66	No	1.83	1.21	0.66	0	0.00
	32	423	10.25	1.98	1.64	0.83	No	1.98	1.64	0.83	0	0.00
	41	1,685	15.82	1.61	1.36	0.85	12.00	1.61	1.33	0.82	3	0.26
	42	1,901	27.29	1.19	1.39	1.17	15.00	1.18	1.19	1.01	3	1.15
	51	244	7.73	1.67	1.37	0.82	No	1.67	1.37	0.82	0	0.00
	52	298	9.58	1.48	1.27	0.86	No	1.48	1.27	0.86	0	0.00
	60	889	13.69	1.74	1.67	0.96	No	1.74	1.67	0.96	0	0.00
Pervasive Alteration Zones (70)	71	3,874	9.93	0.72	0.62	0.86	8.00	0.72	0.61	0.84	2	0.09
	72	3,156	14.63	0.73	0.77	1.05	8.00	0.73	0.74	1.01	2	0.35
	73	1,403	8.28	0.70	0.65	0.93	No	0.70	0.65	0.93	0	0.00

4.5 Bulk Density Data

A total of 3,109 density measurements were performed on core samples at Chemex from samples in the 2006 and 2007 drilling campaigns.

Table 4.3 summarizes the statistics of the two main lithologies and all the others are grouped. For the two main lithologies, the standard deviation and the coefficient of variation are very low. Based on these statistics, an average density of 2.75 t/m³ for sediments and 2.69 t/m³ for porphyry was used to estimate the tonnage in the resource estimation.

Table 4.3: Statistics of Density Measurements by Lithology

Parameters	Sediments	Porphyry	Other
Number	2,196	780	133
Mean (t/m³)	2.75	2.69	2.80
Minimum (t/m³)	2.54	2.52	2.65
Maximum (t/m³)	3.27	3.38	3.07
Median (t/m³)	2.75	2.69	2.80
Std dev	0.03	0.04	0.09
CoV	0.01	0.02	0.03

4.6 Variography

Grade variography was generated and modeled in preparation for the estimation of gold grades. The variography was completed based directly on the cut 5 m down-hole composite data.

A standard approach was used to generate and model the variography for each of the sub-domains. The steps taken are summarized below:

- Examination of the orientations and dips of the solids representing the domains to be studied to help in the determination of the axes of better continuity.
- Generate and model the down-hole direction correlogram, which allows the determination of the nugget effect (close spaced variability).
- Calculate and model the major, semi-major and minor axes of continuity.

Correlograms were generated every 30 degrees of azimuth and at 15 degree dip increments for all the sub-domains using Sage 2001 software, which uses regressions to determine optimal anisotropy directions.

All the variography was modeled with a nugget effect and 2 structures representing the larger scale spatial variability of the datasets.

The modeled correlograms for each of the domains are summarized in Table 4.4. The rotation angles use the Gemcom convention around the ZYZ axes based on the orientation of the block model used. The resulting orientations were visualized in Gemcom to see if the directions of the axes were consistent with the solid orientations.

Generally, the nugget effect is moderate for a gold deposit at between 20 to 30%. For all domains the first structure generally comprises a significant amount of the variance, between 45% to 65%. So the nugget effect and the first structure together usually represent 75% to 90% of the total variance. The range of the major axis of the first structure is generally close to 30 m.

Table 4.4: Variography Statistics

Correlogram	Domain	Nugget	Ranges (m)		Rotation (degree)		
			1st Structure	2nd Structure	Z	Y	Z
11-13 One	11 - 13	0.25	X: 9 Y: 9 Z: 25 Sill: 0.54	X: 25 Y: 80 Z: 92 Sill: 0.21	-49	-5	-37
12 One	12	0.30	X: 17 Y: 9 Z: 19 Sill: 0.56	X: 280 Y: 49 Z: 81 Sill: 0.14	-10	-38	-2
14-16 One	14 - 16	0.25	X: 6 Y: 6 Z: 26 Sill: 0.62	X: 97 Y: 37 Z: 99 Sill: 0.13	24	1	-58
15 One	15	0.20	X: 48 Y: 6 Z: 14 Sill: 0.61	X: 45 Y: 237 Z: 97 Sill: 0.19	-113	16	0
21-23-26	21-23-26	0.25	X: 9 Y: 13 Z: 19 Sill: 0.52	X: 80 Y: 103 Z: 37 Sill: 0.23	35	21	58
22 One	22	0.25	X: 20 Y: 21 Z: 19 Sill: 0.67	X: 93 Y: 200 Z: 166 Sill: 0.08	40	73	18
24 One	24	0.40	X: 7 Y: 14 Z: 14 Sill: 0.49	X: 31 Y: 67 Z: 92 Sill: 0.11	11	30	-5
31 One	31	0.20	X: 23 Y: 14 Z: 4 Sill: 0.44	X: 81 Y: 150 Z: 45 Sill: 0.36	-58	-30	101
32 One	32	0.30	X: 5 Y: 41 Z:6 Sill: 0.40	X: 13 Y: 120 Z: 76 Sill: 0.30	3	86	-53
41 One	41	0.30	X: 20 Y: 14 Z: 45 Sill: 0.47	X: 11 Y: 71 Z: 95 Sill: 0.23	-16	-31	-11
42 One	42	0.30	X: 6 Y: 6 Z: 15 Sill: 0.57	X: 109 Y: 41 Z: 87 Sill: 0.13	-111	-25	70
51 One	51	0.25	X: 17 Y: 12 Z: 39 Sill: 0.73	X: 12 Y: 214 Z: 148 Sill: 0.02	17	-18	18
52 One	52	0.20	X: 10 Y: 30 Z: 21 Sill: 0.60	X: 12 Y: 191 Z: 26 Sill: 0.20	-80	13	16

Table 4.4: Variography Statistics (continued)

Correlogram	Domain	Nugget	Ranges (m)		Rotation (degree)		
			1st Structure	2nd Structure	Z	Y	Z
60 One	60	0.25	X: 11 Y: 51 Z: 16 Sill: 0.73	X: 28 Y: 82 Z: 75 Sill: 0.02	50	107	-7
71 One	71	0.30	X: 27 Y: 14 Z: 18 Sill: 0.57	X: 147 Y: 172 Z: 65 Sill: 0.13	-75	13	36
72 One	72	0.25	X: 16 Y: 25 Z: 4 Sill: 0.31	X: 80 Y: 60 Z: 38 Sill: 0.44	63	49	-40
73 One	73	0.20	X: 14 Y: 17 Z: 8 Sill: 0.60	X: 30 Y: 51 Z: 98 Sill: 0.20	6	-28	-16

4.7 Block Modeling

A block model was constructed (July, 2008) within the Canadian Malartic July08 Gems 6.1.3 database. The block model extents were designed to be large enough to facilitate pit optimizations and associated pit slopes. The block model parameters are summarized in Table 4.5.

Table 4.5: Canadian Malartic Block Model Parameters

Parameters	East	North	Elevation (m)
Minimum coordinates	712700	5333400	-400
Maximum coordinates	715700	5335800	360
Block Size (m)	20	10	10
Number of blocks	150	240	76
Rotation	0	0	0

The block dimension (20m x 10m x 10m) is based on the existing drilling pattern (30m x 30m) and mine planning considerations (10 m bench height).

The domain coding (rock type model) was based on the various wireframe constraints. Table 4.6 presents the domain coding of the various wireframes, solids and surfaces used in the block model.

Also, within the block model project, a series of models were incorporated for recording the different attributes assigned and calculated in the block model development. The attributes of the block model project are listed in Table 4.7.

Table 4.6: Block Model Coding

Type	Solid or Surface Name	Description	Block Model Code
Surface Surface	Topo_RSG_July2008	Topographic surface - Air	0
Lithology	Overburd_RSG July08	Base of the overburden	101
	geol_porph_dyke	porphyry unit	80
	0p2_nl_horiz	Low grade sub-domain	11
	0p2_nl_mod	Low grade sub-domain	12
	0p2_nl_vert	Low grade sub-domain	13
	0p2_sl_horiz	Low grade sub-domain	14
	0p2_sl_shallow	Low grade sub-domain	15
	0p2_sl_steep	Low grade sub-domain	16
	1p0_Main_nl_horiz	High grade sub-domain	21
	1p0_Main_nl_mod	High grade sub-domain	22
	1p0_Main_nl_mod_e	High grade sub-domain	23
	1p0_Main_nl_vert	High grade sub-domain	24
	11p0_Main_sl_steep	High grade sub-domain	26
	1p0_S_sl_horiz	High grade sub-domain	31
	1p0_S_sl_shall	High grade sub-domain	32
Grade Envelopes	1p0_WC_nl_horiz	High grade sub-domain	41
	1p0_WC_nl_vert	High grade sub-domain	42
	1p0_WS_N_dip	High grade sub-domain	51
	1p0_WS_sl_horiz	High grade sub-domain	52
	1p0_cs	High grade domain	60
	Perv_nl_mod	Pervasive sub-domain	71
	Perv_sl_shallow	Pervasive sub-domain	72
	Perv_sl_steep	Pervasive sub-domain	73
	0p2_main_2008	Low grade domain	10
	1p0_main	High grade domain	20
	1p0_S	High grade domain	30
	1p0_WC	High grade domain	40
	1p0_WS	High grade domain	50
	0p4_perv	Pervasive domain	70
Stopes	cm***	Stoping voids	998
Stopes	ijm***	Stoping voids	998
Stopes	sm***	Stoping voids	998
Development	drive***	Drive development voids	999

Table 4.7:Block Model Attributes

Model Name	Description
Rock Type	Sub-domain coding
Density	Specific gravity
% solid	% of the block outside voids
Au_ok	Ordinary kriging grade Au (cut)
Au_ID2	Inverse distance squared grade Au (cut)
Au_OK_recent	Ordinary kriging Au (cut) recent drilling only
Class	Resource classification
Comp vs. OK	Average value of the composites inside blocks

4.8 Grade Estimation Methodology

Grade estimation was done using Ordinary Kriging (OK) as the reported method while inverse distance to the second power (ID^2) was used for comparison. Gems 6.1 software was used for the estimates.

The grade estimates were generated using the cut 5 m composites. The blocks that are included in a particular sub-domain are estimated only with the composites coded within that sub-domain (hard boundary), except for the low grade domains where the sample populations were similar in average grade (soft boundary). The two estimates (OK and ID^2) have been made using a similar sample search approach as summarized below:

- First pass: minimum of 5 and maximum of 12 composites with data in a minimum of 4 octants within the search ellipse. The search ellipse corresponds to the ranges of the first structure identified by variographic studies (generally <30 m for the major axis).
- Second pass: minimum of 3 and maximum of 12 composites with data in a minimum of 3 octants within the search ellipse. The size of the ellipse corresponds to the 2/3 of the ranges of the second structure identified by variography (generally close to 100 m for the major axis).
- Third pass: minimum of 1 and maximum of 12 composites within a search ellipse corresponding to the ranges of the second structure identified by variography.

The minimum data in 4 and 3 octants for the first and second pass (with a maximum of 4 composites per octant) were used to help to decluster the data, especially for underground drilling where fans of drilling are often present. Using maximum composites per octants will limit the weight given to underground holes that are often in fans from drive levels.

Also, for the low grade domains, a high grade distance restriction was applied to restrict composite data higher than 2.0 g/t Au to a search distance corresponding to half of the search ellipse dimension. Even though somewhat arbitrary, this measure is judged prudent since the continuity of high grade values is very limited in the low grade domains and it is desirable to restrict the extrapolation of the higher grade composites within these zones.

Search ellipse parameters are described in Table 4.8.

Table 4.8: Sample Search Parameters

Interpolation Profile	Rock Code	Pass	Rotation			Sample Search			Sample Selection			Minimum Octants With Data	Maximum Samples Per Octant
			Z	Y	Z	X	Y	Z	Min	Max	Max Per Hole		
1113_1	11-13	1	-49	-5	-37	9	9	25	5	12	2	4	4
1113_2	11-13	2	-49	-5	-37	17	53	60	3	12	2	3	4
1113_3	11-13	3	-49	-5	-37	25	80	92	1	12	2	-	-
12_1	12	1	-10	-38	-2	17	9	10	5	12	2	4	4
12_2	12	2	-10	-38	-2	130	33	53	3	12	2	3	4
12_3	12	3	-10	-38	-2	200	50	80	1	12	2	-	-
1416_1	14-16	1	24	1	-58	6	6	25	5	12	2	4	4
1416_2	14-16	2	24	1	-58	65	25	65	3	12	2	3	4
1416_3	14-16	3	24	1	-58	100	37	100	1	12	2	-	-
15_1	15	1	-113	16	0	48	6	14	5	12	2	4	4
15_2	15	2	-113	16	0	30	150	65	3	12	2	3	4
15_3	15	3	-113	16	0	45	230	100	1	12	2	-	-
212326_1	21-23-26	1	35	21	58	10	13	19	5	12	2	4	4
212326_2	21-23-26	2	35	21	58	53	67	25	3	12	2	3	4
212326_3	21-23-26	3	35	21	58	80	100	37	1	12	2	-	-
22_1	22	1	40	73	18	20	21	19	5	12	2	4	4
22_2	22	2	40	73	18	62	150	110	3	12	2	3	4
22_3	22	3	40	73	18	93	500	166	1	12	2	-	-
24_1	24	1	11	30	-5	7	14	14	5	12	2	4	4
24_2	24	2	11	30	-5	21	44	61	3	12	2	3	4
24_3	24	3	11	30	-5	32	67	92	1	12	2	-	-
31_1	31	1	-58	-30	101	23	14	5	5	12	2	4	4
31_2	31	2	-58	-30	101	54	100	30	3	12	2	3	4
31_3	31	3	-58	-30	101	80	150	45	1	12	2	-	-
32_1	32	1	3	86	-53	5	40	6	5	12	2	4	4
32_2	32	2	3	86	-53	10	80	50	3	12	2	3	4
32_3	32	3	3	86	-53	13	120	75	1	12	2	-	-
41_1	41	1	-16	-31	-11	20	14	45	5	12	2	4	4
41_2	41	2	-16	-31	-11	20	45	63	3	12	2	3	4
41_3	41	3	-16	-31	-11	20	71	95	1	12	2	-	-
42_1	42	1	-111	-25	70	6	6	15	5	12	2	4	4
42_2	42	2	-111	-25	70	72	28	59	3	12	2	3	4

Table 4.8: Sample Search Parameters (continued)

Interpolation Profile	Rock Code	Pass	Rotation			Sample Search			Sample Selection			Minimum Octants With Data	Maximum Samples Per Octant
			Z	Y	Z	X	Y	Z	Min	Max	Max Per Hole		
42_3	42	3	-111	-25	70	109	41	87	1	12	2	-	-
51_1	51	1	17	-18	18	17	12	39	5	12	2	4	4
51_2	51	2	17	-18	18	25	107	75	3	12	2	3	4
51_3	51	3	17	-18	18	35	214	148	1	12	2	-	-
52_1	52	1	-80	13	16	10	30	21	5	12	2	4	4
52_2	52	2	-80	13	16	17	125	20	3	12	2	3	4
52_3	52	3	-80	13	16	25	190	30	1	12	2	-	-
60_1	60	1	50	-73	7	11	51	16	5	12	2	4	4
60_2	60	2	50	-73	7	19	55	50	3	12	2	3	4
60_3	60	3	50	-73	7	28	82	75	1	12	2	-	-
71_1	71	1	-75	13	36	27	14	18	5	12	2	4	4
71_2	71	2	-75	13	36	100	115	43	3	12	2	3	4
71_3	71	3	-75	13	36	150	172	65	1	12	2	-	-
72_1	72	1	63	49	-40	16	25	5	5	12	2	4	4
72_2	72	2	63	49	-40	53	40	25	3	12	2	3	4
72_3	72	3	63	49	-40	80	60	38	1	12	2	-	-

The tabulated unclassified mineral resources for the Canadian Malartic gold deposit are presented in Table 4.9 for various cut-off grades and for the two grade interpolation methodologies.

Table 4.9: Unclassified Mineral Resources, Canadian Malartic Gold Deposit

Cut-off (g/t Au)	OK Model			ID^2 Model		
	K Tonnes	Grade (g/t Au)	K Ounces Au	K Tonnes	Grade (g/t Au)	K Ounces Au
0.001	650,951	0.51	10,606	650,902	0.51	10,642
0.10	561,282	0.58	10,453	537,556	0.60	10,455
0.20	431,751	0.71	9,823	412,421	0.74	9,855
0.30	318,276	0.87	8,920	311,936	0.90	9,056
0.36	269,689	0.97	8,407	268,346	1.00	8,595
0.40	244,038	1.03	8,094	244,951	1.06	8,310
0.50	194,454	1.18	7,381	198,628	1.20	7,644
0.60	158,561	1.32	6,749	163,339	1.34	7,022
0.70	132,456	1.46	6,204	137,266	1.47	6,477
0.80	114,862	1.57	5,781	118,231	1.58	6,019
0.90	101,197	1.66	5,410	104,380	1.68	5,641
1.00	90,844	1.74	5,094	92,971	1.77	5,294

As shown in the table, the results are very comparable. At a low cut-off (<0.36 g/t Au), the discrepancy is less than 2% in ounces while at higher cut-offs (>0.6 g/t Au), the difference reaches a maximum of 4% in favour of the ID^2 model.

4.9 Estimation Validation

The block model, ore zones, voids and drill hole database were sliced on 30 m intervals and the attributes of the block model (rock code, grade, % solid) were visually compared with the source data throughout the strike length of the deposit. The block model reproduces the wireframes well and the ordinary kriging-based resource estimate was found to provide an appropriate local estimate of drill hole grades.

4.9.1 Composites vs. Interpolated Blocks

A common way to validate grade estimation is to compare the average grade of the composites used in the estimate with the estimated grade of the blocks interpolated. If the drilling pattern is regular (no clustering of the data) and there is no distortion in the grade distribution, the two populations should show about the same mean. Table 4.10 presents the average grade of the cut 5 m composites and of the blocks interpolated for each individual sub-domains of the deposit.

Table 4.10: Comparison Between Composites and Interpolated Blocks

Zone	Grade, Composites (g/t Au)	Grade, OK Model (g/t Au)	Difference (OK/Comp) (%)	K Tonnes
11	0.32	0.27	-16.4	11,294
12	0.27	0.26	-4.0	132,308
13	0.38	0.33	-15.8	41,466
14	0.27	0.27	-0.1	10,258
15	0.26	0.22	-17.1	203,530
16	0.31	0.29	-5.8	81,285
21	1.73	1.54	-12.7	4,402
22	1.82	1.70	-6.9	41,844
23	2.24	1.99	-12.5	3,516
24	2.01	1.89	-6.2	8,092
26	1.95	1.88	-3.5	7,515
31	1.83	1.60	-14.3	1,017
32	1.98	2.02	1.9	1,144
41	1.61	1.67	3.8	5,441
42	1.18	1.22	3.0	6,132
51	1.67	1.65	-1.5	958
52	1.48	1.51	2.3	797
60	1.74	1.70	-2.3	4,239
71	0.72	0.73	1.0	38,570
72	0.73	0.74	1.5	35,898
73	0.70	0.66	-5.6	11,247
Total	**0.54**	**0.51**	**-6.3**	**650,951**

From the table, it can be seen than the OK estimate is on average about 6% lower than the corresponding grade of the composites. This is something that can be expected for two main reasons as discussed below.

The first reason is that the drill hole data are clustered in some of the high grade zones because the underground drilling (the original Canadian Malartic drilling) is concentrated in the highest grade portions of the deposit. The restrictions that are imposed in the resource estimate (maximum composites per octants) help to decluster the data. Thus, the weight of the underground drilling is more pronounced in the average grade of the composites than in the OK estimate.

The second reason is the distance restriction employed for the high grade composite values in the low grade zone during grade interpolation. Therefore, the influence of these higher values is higher in the average grade composites than in the kriged estimate.

4.9.2 Mean Composite Grade Within a Block vs. Kriged Grade

Mean drill hole sample composite grades that fall within a block were calculated for each block. This value is compared with the grade interpolated for the block. A successful grade interpolation protocol will result in block grade estimates that demonstrate a minimum amount of bias. Table 4.11 shows the comparison for blocks within each zone.

Table 4.11: Comparison Between Mean Composite Grade Within a Block and the Kriged Grade

	Zone	No. Blocks With Composites	Ordinary Kriging Estimate (g/t Au)	Composites, Mean Within Blocks (g/t Au)	Difference OK/Comp (%)
Low Grade	11	164	0.31	0.31	-1.45
	12	3,559	0.26	0.28	-5.62
	13	1,410	0.35	0.37	-4.17
	14	477	0.28	0.27	2.48
	15	4,600	0.25	0.25	-1.72
	16	2,676	0.30	0.31	-3.65
High Grade	21	373	1.72	1.69	1.45
	22	3,396	1.85	1.84	0.44
	23	401	2.21	2.18	1.13
	24	871	2.01	1.97	1.67
	26	886	1.95	1.95	-0.20
	31	110	1.70	1.73	-1.30
	32	155	2.11	2.07	1.94
	41	573	1.65	1.64	0.74
	42	615	1.21	1.20	0.72
	51	89	1.66	1.62	2.08
	52	90	1.52	1.51	0.26
	60	310	1.75	1.70	3.28
Pervasive Alteration	71	1,587	0.73	0.72	1.29
	72	1,402	0.74	0.74	0.20
	73	566	0.70	0.71	-1.28
Total		**24,310**	**0.84**	**0.84**	**-0.02**

It is apparent from the table that there is no significant bias between the grade of the composites and the estimated grade. The analysis demonstrates that the mineral resource model provides a reasonable estimate of the Canadian Malartic project mineral resources.

4.10 Classification

The Mineral Resources estimated for the Canadian Malartic gold deposit were classified according to the "CIM Standards on Mineral Resources and Reserves - Definitions and Guidelines" as adopted by CIM Council on December 11, 2005.

The mineral resource classification is based on the robustness of the various data sources available, including:

- Quality and reliability of drilling and sampling data
- Distance between sample points (drilling density)
- Confidence in the geological interpretation
- Continuity of the geologic structures and the continuity of the grade within these structures
- Variogram models and their related ranges (first and second structures)
- Statistics of the data population
- Tonnage factor.

Based on these criteria, the resources have been classified according to the data search used to estimate each block and also on the type of data used for the estimate.

Measured resources are limited to the blocks estimated in the first estimation pass and only within mineralized zones for which the recent drilling represents a high majority of the data (>65%).

Indicated resources correspond to the blocks estimated in the second estimation pass plus the blocks estimated in the first pass but not classified as Measured.

Inferred resources correspond to the blocks estimated in the third estimation pass.

4.10.1 Global Resources

Table 4.12 provides the resource estimation tabulation by category, at different cut-offs, for the ordinary kriged model (the official model used for reporting). Table 4.13 presents the results for the ID^2 model.

Based on economic parameters explained later in this report (see Section 4.10.2), Micon has calculated the break-even cut-off grade for Canadian Malartic project to be 0.36 g/t Au. At this cut-off, the global Measured + Indicated mineral resources total 232.2 Mt at a grade of 1.03 g/t Au, representing 7.69 Moz of gold. The Inferred resources represent 37.4 Mt at 0.60 g/t Au, for 0.72 Moz of gold (see highlighted row in Table 4.12).

Table 4.12: Canadian Malartic Project Global Resources By Cut-Off - OK Model

	Measured			Indicated			Total: Meas. & Ind.			Inferred		
Cut-off (g/t Au)	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**
0.001	5.1	1.22	0.20	494.6	0.57	9.10	499.7	0.58	9.30	151.3	0.27	1.31
0.10	5.1	1.22	0.20	442.4	0.76	9.00	447.5	0.64	9.20	113.8	0.43	1.25
0.20	5.0	1.23	0.20	347.1	0.76	8.54	352.2	0.77	8.73	79.6	0.43	1.09
0.30	4.9	1.25	0.20	263.2	0.93	7.87	268.1	0.94	8.07	50.2	0.53	0.85
0.36	**4.8**	**1.27**	**0.20**	**227.4**	**1.02**	**7.49**	**232.2**	**1.03**	**7.69**	**37.4**	**0.60**	**0.72**
0.40	4.8	1.28	0.20	208.1	1.08	7.26	212.9	1.09	7.45	31.1	0.64	0.64
0.50	4.4	1.34	0.19	170.0	1.23	6.71	174.4	1.23	6.90	20.0	0.75	0.48
0.60	4.1	1.41	0.19	142.0	1.36	6.21	146.0	1.36	6.40	12.5	0.87	0.35
0.70	3.6	1.51	0.18	121.9	1.48	5.80	125.5	1.48	5.97	7.0	1.04	0.23
0.80	3.2	1.61	0.17	106.7	1.58	5.43	109.9	1.58	5.60	4.9	1.17	0.18
0.90	2.9	1.69	0.16	95.0	1.67	5.11	97.8	1.67	5.27	3.4	1.32	0.14
1.00	2.7	1.75	0.15	85.7	1.75	4.83	88.4	1.75	4.98	2.5	1.45	0.12

Table 4.13: Canadian Malartic Project Global Resources By Cut-Off - ID^2 Model

	Measured			Indicated			Total: Meas. & Ind.			Inferred		
Cut-off (g/t Au)	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**	**Tonnes (Mt)**	**Grade (g/t Au)**	**Ounces (Moz Au)**
0.001	5.1	1.22	0.20	494.6	0.57	9.14	499.6	0.58	9.34	151.3	0.27	1.30
0.10	5.1	1.22	0.20	423.9	0.76	9.02	429.0	0.67	9.22	108.6	0.43	1.24
0.20	5.0	1.23	0.20	332.9	0.76	8.58	338.0	0.81	8.78	74.5	0.43	1.08
0.30	4.9	1.25	0.20	259.3	0.93	7.99	264.1	0.96	8.19	47.8	0.53	0.86
0.36	**4.8**	**1.27**	**0.20**	**226.5**	**1.02**	**7.65**	**231.3**	**1.05**	**7.84**	**37.1**	**0.60**	**0.75**
0.40	4.7	1.28	0.20	208.6	1.08	7.43	213.2	1.11	7.62	31.7	0.64	0.69
0.50	4.4	1.34	0.19	172.2	1.23	6.91	176.6	1.25	7.10	22.1	0.75	0.55
0.60	4.0	1.41	0.18	145.1	1.36	6.43	149.1	1.38	6.61	14.3	0.87	0.41
0.70	3.6	1.51	0.18	125.0	1.48	6.01	128.6	1.50	6.18	8.7	1.04	0.29
0.80	3.2	1.61	0.17	109.3	1.58	5.63	112.5	1.60	5.80	5.7	1.17	0.22
0.90	2.8	1.69	0.16	97.3	1.67	5.31	100.2	1.70	5.46	4.2	1.32	0.18
1.00	2.6	1.75	0.15	87.2	1.75	5.00	89.8	1.78	5.14	3.2	1.45	0.15

4.10.2 In-Pit Mineral Resources

The OK block model was exported to the Whittle pit optimization software where a pit was generated on Measured and Indicated resources only. Base case pit parameters include a maximum 55 degree slope angle, ore based cost of $6.38/t milled (based on a 55,000 t/d milling rate), variable mining cost depending of the zone and depth ($1.46 to $1.77/t mined with incremental $0.018/t mined per 10 m bench) and variable processing recoveries depending on grade (from 75% to 92%). The gold price used was $775/oz. These costs are derived from the 2008 Preliminary Assessment (Runnels et al, 2008) and the most recent feasibility study work currently underway (Sections 5.2.4 to 5.2.7).

Table 4.14 presents the resource estimation tabulation, by category, and at different cut-off grades for the ordinary kriged model within the pit shell generated.

Based on an ore based cost of $6.38/t milled, the corresponding in-pit cut-off grade for the base case $775/oz Whittle shell is 0.36 g/t Au. At this cut-off, the official Measured + Indicated In-Pit resources for the Canadian Malartic project total 178.2 Mt at an undiluted grade of 1.12 g/t Au, representing 6.42 Moz of gold (see highlighted row in Table 4.14).

Mineral resources that are not mineral reserves do not have demonstrated economic viability. The stated mineral resources are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the authors. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this mineral resource estimate, at this time.

4.10.3 Impact of the Use of Historical Drilling

In order to quantify the impact of using the historical data for the high grade zones in the resource estimate, a block model using only recent data (5 m composites from Lac Minerals and Osisko drilling only) was created (using the OK method) and compiled for comparison on the global and In-Pit resources. The same parameters as for the OK model above were used.

Table 4.15 compares the estimates, using the same classification, at a cut-off of 0.36 g/t Au and shows the impact on the grade.

As shown in the table, using the historical data for the resource estimate has an impact of about 4 to 5% on the grade and ounces.

Table 4.14: Canadian Malartic Project In-Pit Resources - By Cut-Off - OK Model

	Measured			Indicated			Total: Meas. & Ind.			Inferred		
Cut-off (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)
0.30	4.6	1.27	0.19	192.1	1.04	6.43	196.8	1.05	6.62	5.8	0.74	0.14
0.36	**4.5**	**1.29**	**0.19**	**173.7**	**1.12**	**6.23**	**178.2**	**1.12**	**6.42**	**5.0**	**0.81**	**0.13**
0.40	4.5	1.31	0.19	163.4	1.16	6.11	167.9	1.17	6.29	4.6	0.84	0.13
0.50	4.2	1.37	0.18	140.5	1.28	5.78	144.7	1.28	5.96	3.7	0.94	0.11
0.60	3.8	1.44	0.18	120.8	1.40	5.43	124.7	1.40	5.61	2.8	1.08	0.10
0.70	3.4	1.54	0.17	106.0	1.50	5.12	109.4	1.50	5.29	1.9	1.26	0.08
0.80	3.0	1.63	0.16	94.2	1.60	4.84	97.2	1.60	5.00	1.6	1.37	0.07
0.90	2.8	1.70	0.15	84.6	1.68	4.57	87.4	1.68	4.73	1.3	1.48	0.06
1.00	2.6	1.76	0.15	76.9	1.75	4.34	79.5	1.76	4.49	1.1	1.59	0.06

Table 4.15: Comparison Between OK Models Using and Not Using Historical Drilling in High Grade Zones

	Measured			Indicated			Total: Meas. & Ind.			Inferred		
Method	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)
Global with Hist. Drilling	4.8	1.27	0.20	227.4	1.02	7.49	232.2	1.03	7.69	37.4	0.60	0.72
Global without Hist. Drilling	4.8	1.25	0.19	227.2	0.98	7.16	232.0	0.99	7.36	37.1	0.59	0.70
Difference	0.1%	-1.3%	-1.2%	-0.1%	-4.2%	-4.4%	-0.1%	-4.2%	-4.3%	-1.0%	-1.2%	-2.2%
In-Pit with Hist. Drilling	4.5	1.29	0.19	173.7	1.12	6.2	178.2	1.12	6.42	5.0	0.81	0.13
In-Pit without Hist. Drilling	4.5	1.27	0.19	173.2	1.06	5.9	177.8	1.07	6.11	5.0	0.78	0.13
Difference	0.0%	-1.4%	-1.4%	-0.3%	-4.7%	-5.0%	-0.3%	-4.6%	-4.9%	0.2%	-3.3%	-3.2%

4.11 Interpretation and Conclusions

Previous exploration work by Osisko has demonstrated the existence of a large, lower-grade mineralization envelope around the old underground mine workings of the Canadian Malartic mine in Malartic, Québec. This work led to the release of two Inferred mineral resource estimates for the deposit in early- and mid-2007 (Gossage 2007a and 2007b).

These estimates have led Osisko to work on an extensive 30m x 30m infill definition drill program in order to upgrade the mineral resources to higher confidence categories. That drilling has now reached a stage where an updated estimate can be completed. Micon has performed that estimate and it has been largely successful in upgrading most of the mineral resources to higher categories.

In the process of completing the estimate Micon has interpreted the available data and come to the following conclusions:

- For the updated estimate Micon has chosen to use a portion of the significant amount of historic drilling data available from Canadian Malartic Mines. After a comparative study between the historical and recent data, a bias has been identified for values in the low grade domains of the geological model (<1.0 g/t Au) and all of the historical data in those low grade zones have been discarded. In the higher grade populations of the high grade domains, no evident bias was observed and these data were retained and used for the resource estimation presented herein.
- The impact of using the historical data in the high grade zones is between 4 to 5% on the grade.
- The completion of the infill drilling on a 30m x 30m pattern has successfully confirmed continuity of the early Inferred resource estimates identified from the first phases of drilling. The majority of the mineral resources are now classified in the Indicated category.
- Outlier values in the assay population have been analyzed and top cuts were generally applied, resulting in a decrease in grade of from 0% to 2.35% depending on the zone.
- The block size dimensions chosen for the model (20m x 10m x 10m) were based on the existing drilling pattern (30m x 30m) and mine planning considerations (10 m bench heights).
- The resources were estimated using ordinary kriging (OK). A parallel estimate was conducted using the inverse distance squared (ID^2) method as a check. The discrepancy between the OK model and the ID^2 model is less than 2%.

- In order to evaluate the economic potential of the resources, a pit optimization was generated using the Measured and Indicated resources only. Base case pit parameters include a maximum 55 degree slope angle, ore based cost of $6.38/t milled (based on a 55,000 t/d milling rate), variable mining cost depending on the zone and depth ($1.46 to $1.77/t mined with increments of $0.018/t mined per 10 m bench) and variable processing recoveries depending on the grade (from 75% to 92%). The gold price used is $775/oz. These parameters were taken from a recent preliminary assessment and/or updates to it for an on-going feasibility study.

The resulting in-pit mineral resources for the Canadian Malartic deposit are presented in Table 4.16 below.

Table 4.16: Canadian Malartic Project In-Pit Mineral Resources

Category	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)
Measured	4.5	1.29	0.19
Indicated	173.7	1.12	6.23
Total: Meas. & Ind.	**178.2**	**1.12**	**6.42**
Inferred	5.0	0.81	0.13

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.

5.0 MINING AND MINERAL RESERVES

This section contains results and supporting details of optimal pit limit evaluation, mine design and mine planning for the Canadian Malartic project, which include:

- Pit optimisation
- Detailed pit and phase design
- Production schedule optimisation
- Waste dump design
- Annual production plan and schedule
- Equipment selection and requirements.

5.1 Resource Model

The resource model for the Canadian Malartic project was supplied as a Gemcom project file with the bloc model containing the following items: a geological rock code, in-situ gold grade, resource category, density, and a percent solid value. The percent solid value represents the percentage of intact rock mass with the remaining percentage representing the proportion of void created from the past underground workings. The block information was used for mine planning purposes with other block model elements added for optimization purposes as further described in Section 5.2.

The block model created covers an area of 3.0 km by 2.4 km and extends beyond the mineralization in all directions for a total of just over 2.7 million blocks (Table 5.1).

Table 5.1: Block Model Dimensions

Direction	Model Origin	Block Size (m)	Nb. Blocks
X (col)	712,700	20	150
Y (row)	5,333,400	10	240
Z (level)	360	10	76

5.2 Open Pit Optimization

Open pit optimization was conducted to determine the optimal economic shape of the open pit in three dimensions. This task was undertaken using the Whittle software which is based on the Lerchs-Grossmann algorithm. The method works on a block model of the ore body, and progressively constructs lists of related blocks that should, or should not, be mined. The method uses the values of the blocks to define a pit outline that has the highest possible total economic value, subject to the required pit slopes defined as structure arcs in the software. This section describes all the parameters used to calculate block values in Whittle. The optimization parameters used to establish the optimal shape of the pit are slightly different than those used for financial analysis as presented in Section 15.0. The differences in parameters do not impact the validity of the optimization work performed since there is no change to the cut-off grade.

For this Feasibility Study only measured and indicated resources were considered for optimization purposes and for reserve calculations.

5.2.1 Geotechnical Assessment

A pit slope design evaluation was completed by Golder in order to support feasibility-level pit slope designs.

The formulated pit slope recommendations are based on:

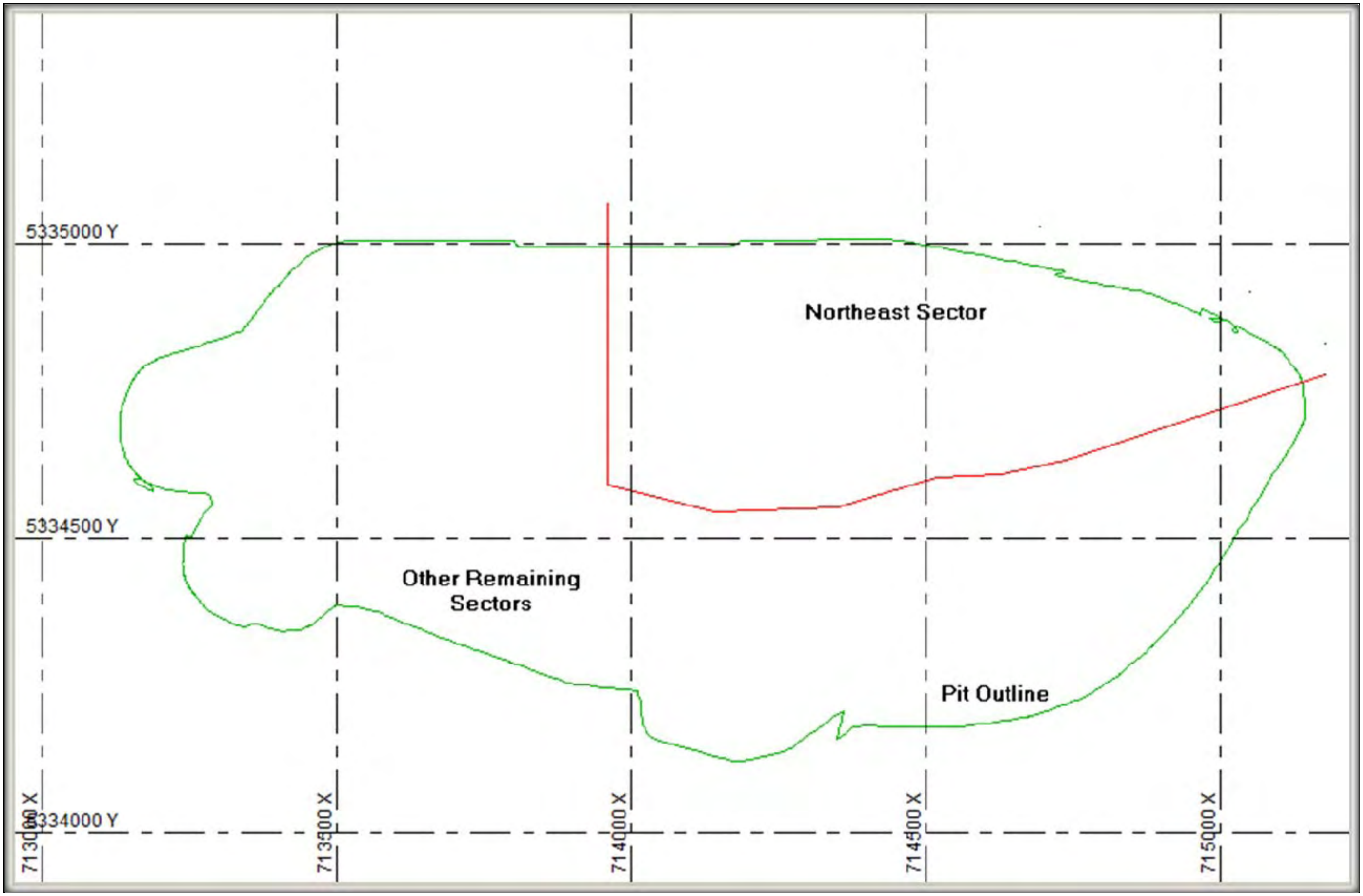
- the examination and interpretation of historical reports and underground mappings
- core logging and strength testing of five coreholes specifically drilled for geotechnical purposes
- structural information collected from five additional coreholes drilled for exploration purposes.

5.2.1.1 Rock Mass Characterization

The rockmass classifies as Very Strong Rock to Strong Rock according to the unconfined compressive strength (UCS) laboratory testing for the majority of the rockmass units. The UCS ranges from 100 to 205 MPa. Other laboratory testing performed includes triaxial compressive strength testing and direct shear testing. The Rock Mass Rating (RMR) classification which incorporates UCS, RQD, joint spacing, joint condition and groundwater conditions classifies the geotechnical units as Good Quality rock masses except for the schists which classify as Fair Quality rock mass.

The rock quality and structural conditions generally favour the development of steep inter-ramp slopes in all areas of the pit except the Northeast sector. Separate recommendations have therefore been formulated for the Northeast sector and for the remaining pit sectors. The pit sectors are presented in Figure 5.1.

Figure 5.1: Geotechnical Domains Modeled for Optimization



5.2.1.2 Northeast Sector Design Recommendations

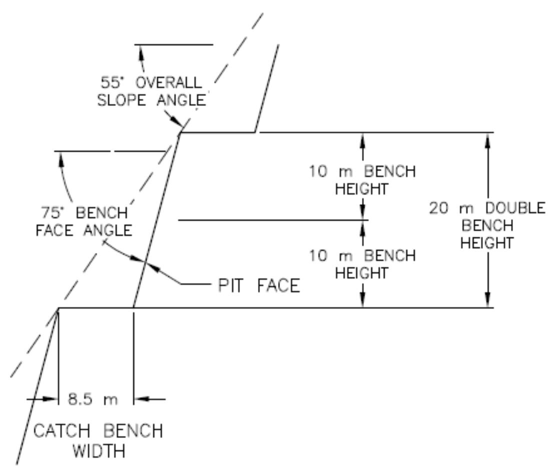
Kinematic stability analyses indicate the potential for significant structurally-controlled planar failures along moderately south-dipping structures in the Northeast sector. The Sladen fault/greywacke-porphyry contact dips 60 degrees South in this sector. For this reason, it is recommended that bench faces in this sector be designed at 60 degrees to reduce back-break and allow for the development of effective catch benches. A double bench configuration is suggested with 8 m catch benches at 20 m intervals, for an inter-ramp slope angle of 46 degrees.

5.2.1.3 Remaining Sector Design Recommendations

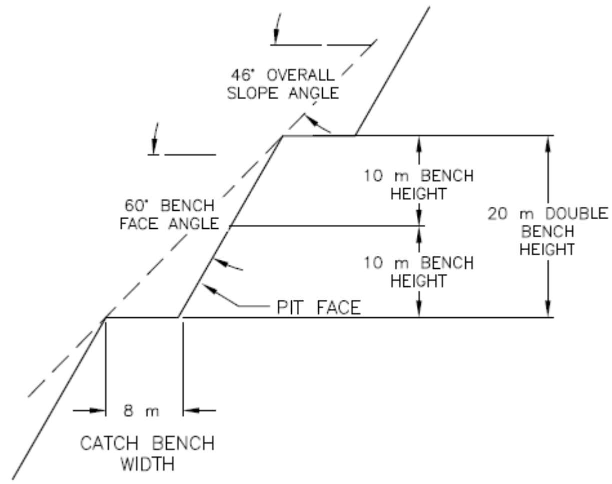
Structural data is favourable to the excavation of moderately steep to steep slopes in all other areas except the Northeast sector.

Two slope profiles are recommended depending on the implementation of an effective pre-split or not. A design recommended with only buffer blasting would include a 9 m catch bench at 20 m intervals (double bench) and designed bench face angles of 69 degrees for an inter-ramp slope angle of 50 degrees. A pre-split would enable the development of steeper slopes with a designed 8.5 m catch bench, 20 m intervals, 75 degree bench face angle for an inter-ramp angle of 55 degrees. The implementation of a pre-split is the preferred approach to pit slope development. The recommended slope configurations to be implemented are presented in Figure 5.2.

Figure 5.2: Slope Configurations (Source: Golder Associates)



All Other Sectors except Northeast
with effective pre-split



Northeast sector
with effective pre-split

5.2.2 Hard Boundaries

A hard boundary was established to limit pit shell expansion towards the town of Malartic. The Canadian Malartic project has established an area in the southern part of the town to be relocated in order to allow for the development of the open pit. However, mineralization at the north end of the deposit is economic and results in an optimal pit shell extending beyond the relocation area. For this reason, a hard boundary was established by assigning arbitrarily high costs to blocks in a buffer zone to be created between the open pit and the town.

5.2.3 Selling Price, Selling Cost and Royalties

A base case gold price of $775/oz was used for pit optimization and for economic evaluation. However, a range of pit shells at various gold prices were generated and used to schedule the pit.

Transportation and refining costs for other gold mines in the Abitibi region were used as a reference. Transportation costs from the Val d'Or area to nearby Canadian refineries is estimated at $0.65/oz Au and refining costs are variable between $0.75-1.00/oz Au depending on gold purity and contaminants in the doré. A transportation and refining cost of $2.00/oz Au has therefore been assumed for optimization.

Of the 127 mining titles comprising the property, 21 are subject to royalties established either on a net smelter return (NSR) or a gross overriding metal royalty. Mining titles subject to royalties are outlined in Table 5.2 and can be regrouped into five royalty packages. The royalty land packages are presented in Figure 5.3.

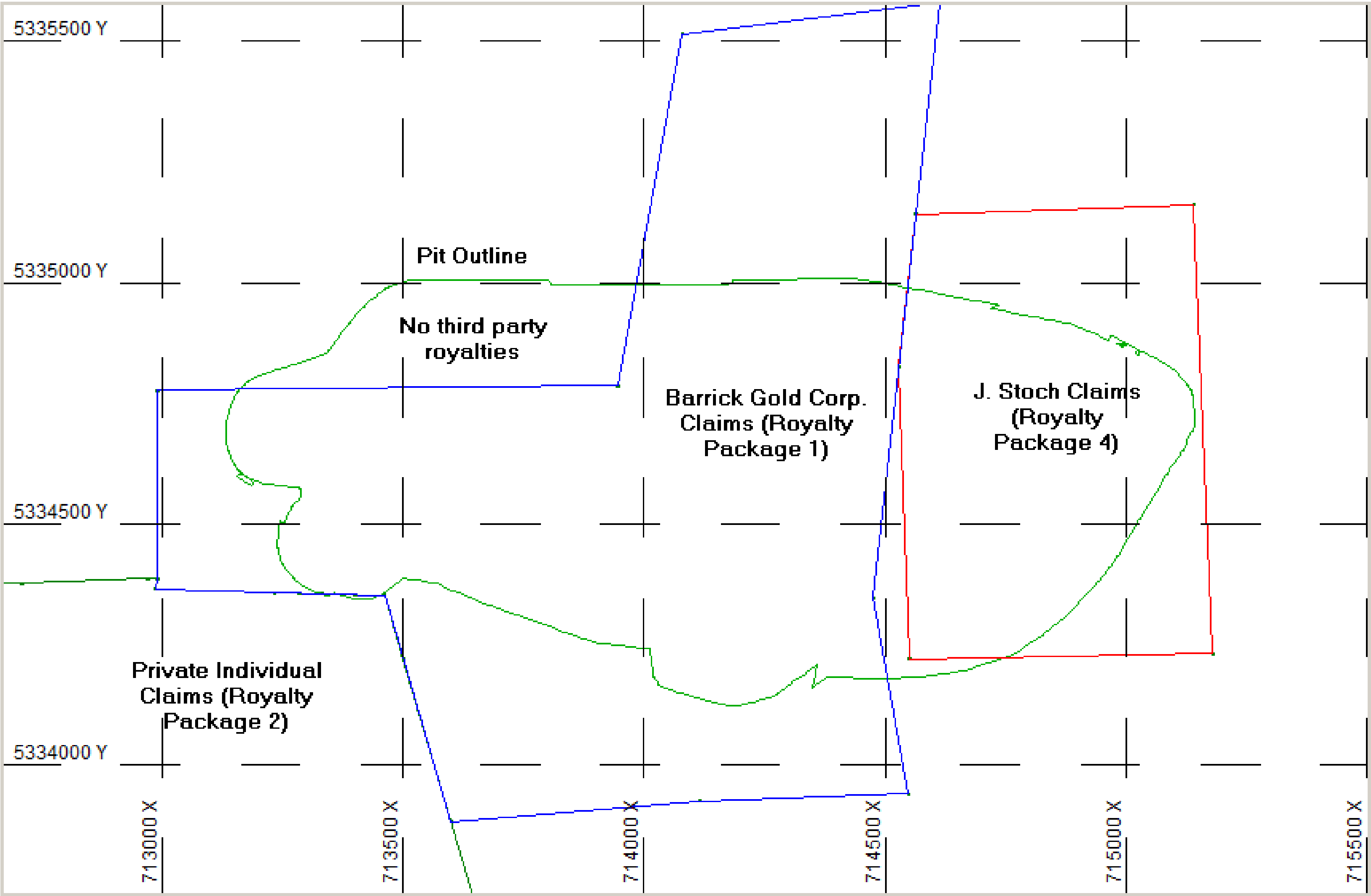
For simplification purposes a 1.5% royalty was applied to the overall resource in the Preliminary Assessment Study of the Canadian Malartic project. For this Feasibility Study a more precise calculation was performed by coding all blocs falling within the perimeter of mining titles subject to royalty agreements. The codes were then subsequently used to calculate the appropriate royalty rate during optimization.

Only two royalty packages currently fall within the current open pit limits. These are royalty packages, 1 and 4 as presented in Table 5.2. Half of the royalty payable to Royal Gold can be purchased for CDN$1.5 M. This study assumes that the Canadian Malartic project will purchase half of this royalty payable to Royal Gold with a remaining 1.5% Net Smelter Return taking effect if the price of gold is greater than $350/oz.

Table 5.2: Mining Claims Subject to Royalty Agreements

Nb.	Mining Title	Agreements and Encumbrances	Royalty Package
1 2 3 4 5 6 7	CM 226 CL 3941621 CL 3941633 CL 3941634 CL 3941635 CL 3950771 CL 3950772	Titles are subject to a sliding 2% to 3% NSR payable Royal Gold. The royalty is tied to the price of gold, with the higher rate taking effect if the gold price is greater than $350/oz. Half the royalty can be purchased back by Osisko for CDN$1.5 M.	1
8 9 10 11 12 13	CL 5144234 CL 5144235 CL 5144236 CL 5144237 CL 5144238 CL 5144239	Claims are subject to a 2% NSR payable to a private individual. The entire royalty may be purchased back by Osisko for CDN$2 M.	2
14	CDC 72271	Claim is subject to a 2% NSR payable to Golden Valley Mines	3
15 16 17 18 19 20	CDC 2000854 CDC 2000855 CDC 2000856 CDC 2000857 CDC 2000858 CDC 2000859	Claims are subject to a 2.5% Gross Overriding Metal Royalty payable to a private party.	4
21	CDC 2001055	Claim is subject to a 2.5% Gross Overriding Metal Royalty	5

Figure 5.3: Royalty Agreements Modeled for Optimization



5.2.4 Cost Structure Basis for Mine Plan Development

5.2.4.1 Mining Costs

Detailed mining costs were estimated for all activities of the mining cycle. Drilling and blasting costs have been determined for certain zones of the pit to limit environmental nuisances as further described in Section 5.6.1. The reference mining cost is presented in Table 5.3. In addition to the reference hauling cost from the pit rim to the waste dump an incremental haulage cost with depth has been estimated for ramping out of the pit. The incremental cost is estimated at $0.018/t mined per 10 m bench.

Table 5.3: Reference Mining Cost ($/t mined)

Mining Activity	Zone 1 5 m bench with 114 mm Φ hole	Zone 2 5 m bench with 140 mm Φ hole	Zone 3 10 m bench with 140 mm Φ hole	Zone 4 10 m bench with 216 mm Φ hole
Production Drilling	0.27	0.22	0.22	0.13
Blasting	0.48	0.41	0.38	0.31
Pre-shear Drill & Blast	0.04	0.04	0.04	0.04
Grade Control	0.03	0.03	0.03	0.03
Loading	0.15	0.15	0.15	0.15
Hauling	0.46	0.46	0.46	0.46
Road & Dump Maintenance	0.14	0.14	0.14	0.14
Support Equipment	0.03	0.03	0.03	0.03
Dewatering	0.02	0.02	0.02	0.02
Mine Mgmt & Tech services	0.15	0.15	0.15	0.15
Total Reference Cost	**1.77**	**1.65**	**1.62**	**1.46**

A mining cost model was constructed in Gemcom that was subsequently exported for optimization in Whittle. The unit mining costs by elevation are presented in Table 5.4.

Table 5.4: Mine Operating Costs ($/t mined) by Surface RL

Surface Reference Level (RL)	Zone 1 5 m bench with 114 mm Φ hole	Zone 2 5 m bench with 140 mm Φ hole	Zone 3 10 m bench with 140 mm Φ hole	Zone 4 10 m bench with 216 mm Φ hole
320	1.77	1.65	1.62	1.46
310	1.78	1.67	1.64	1.48
300	1.80	1.68	1.66	1.50
290	1.82	1.70	1.67	1.52
280	1.84	1.72	1.69	1.54
270	1.86	1.74	1.71	1.55
260	1.87	1.76	1.73	1.57
250	1.89	1.77	1.75	1.59
240	1.91	1.79	1.76	1.61
230	1.93	1.81	1.78	1.63
220	1.95	1.83	1.80	1.64
210	1.96	1.85	1.82	1.66
200	1.98	1.86	1.84	1.68
190	2.00	1.88	1.85	1.70
180	2.02	1.90	1.87	1.72
170	2.04	1.92	1.89	1.73
160	2.05	1.94	1.91	1.75
150	2.07	1.95	1.93	1.77
140	2.09	1.97	1.94	1.79
130	2.11	1.99	1.96	1.81
120	2.13	2.01	1.98	1.82
110	2.14	2.03	2.00	1.84
100	2.16	2.04	2.02	1.86
90	2.18	2.06	2.03	1.88
80	2.20	2.08	2.05	1.90
70	2.22	2.10	2.07	1.91
60	2.23	2.12	2.09	1.93
50	2.25	2.13	2.11	1.95
40	2.27	2.15	2.12	1.97
30	2.29	2.17	2.14	1.99
20	2.31	2.19	2.16	2.00
10	2.32	2.21	2.18	2.02
0	2.34	2.22	2.20	2.04
-10	2.36	2.24	2.21	2.06
-20	2.38	2.26	2.23	2.08
-30	2.40	2.28	2.25	2.09
-40	2.41	2.30	2.27	2.11
-50	2.43	2.31	2.29	2.13
-60	2.45	2.33	2.30	2.15
-70	2.47	2.35	2.32	2.17
-80	2.49	2.37	2.34	2.18

5.2.4.2 Processing Costs

Processing costs including tailings management costs that were used for the pit optimization were estimated at $5.30/t milled based on a milling rate of 55,000 tpd. The actual unit cost per tonne milled after further refinements was determined to be lower at $4.96/t milled as used in the financial analysis of this study.

5.2.4.3 General and Administration Costs

The general and administrative costs used for the pit optimization were estimated at $0.70/t milled for an estimated annual expense of $14.0 M per year. The general and administrative costs includes the costs associated with support functions such as purchasing and warehousing, accounting, environmental management, health and safety, human resources, insurance and general management. The actual cost per tonne milled after further refinements was determined at $0.65/t milled for an annual expense of $13.1 M.

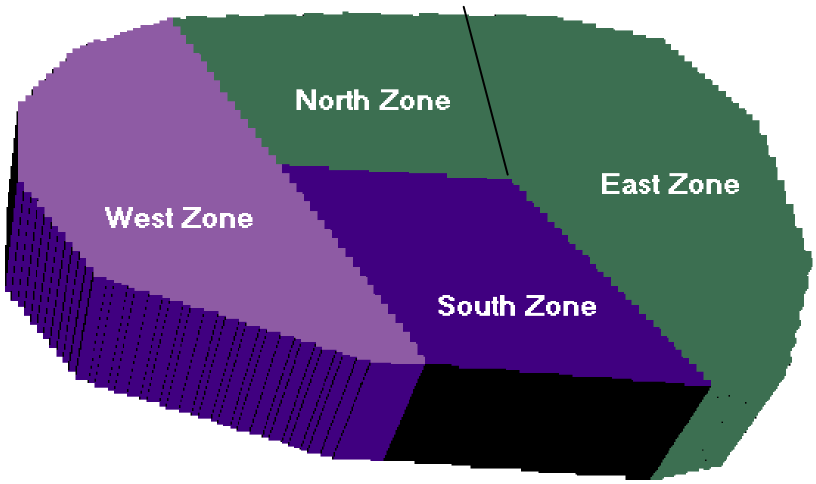
5.2.5 Mill Recovery

Metallurgical recoveries have been established as a function of head grade and by domains in the pit based on extensive metallurgical testing. Testwork results were analysed based on the location of samples in the ore body which led to the establishment of four domains (Figure 5.4) characterized by three metallurgical recovery equations as follows:

- Recovery equation 1 (for west zone & north zone below RL160)
  - Recovery = [74.41 + (12.300 x grade)] /100 (maximum 92%)
- Recovery equation 2 (for east zone & north zone above RL160)
  - Recovery = [76.15 + (7.339 x grade)] /100 (maximum 89%)
- Recovery equation 3 (for south zone)
  - Recovery = [78.50 + (1.654 x grade)] /100 (maximum 86%)

The recovery equations were used to create a recovered gold grade model that was subsequently used during the optimization process.

Figure 5.4: Metallurgical Zones Modeled for Optimization

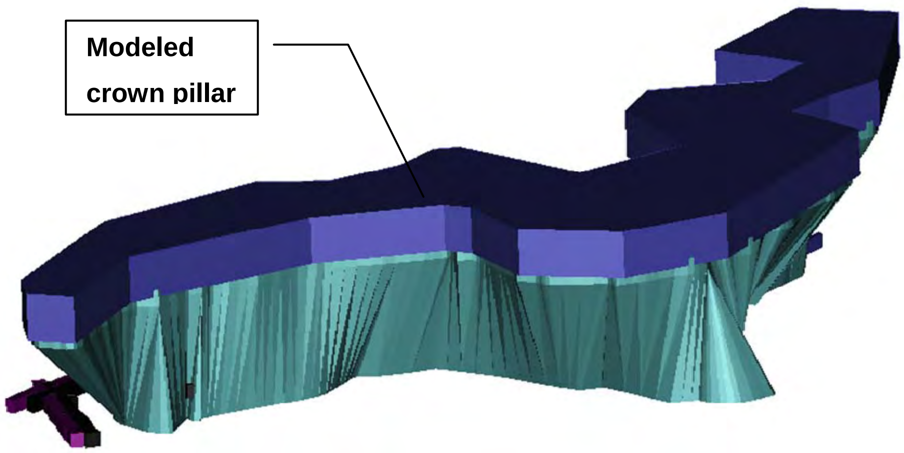


5.2.6 Mining Dilution and Ore Losses

A routine was used to estimate mining dilution by calculating the number of waste blocks which touch an ore block in the 3D Block Model. The dilution edge value is coded with an integer item. This routine only assesses neighbouring blocks on a bench and does not consider the bench above or below. Whole block dilution should cover most of the effects of material above and below whereas this evaluation is considering the recovery and dilution effects as the shovel extracts ore while it advances across a bench. This estimate provides a reasonable evaluation across the bench. The program checks for waste blocks east, west, north and south of the ore block. An ore block completely surrounded by other ore blocks experiences no mining dilution. The other contact blocks will be diluted along their edges since they will be mixed in the blast throw along their edges and the shovels will not be able to separate the material as defined in the pre-blasted digging packet. More edges will result in more dilution. The weighted dilution estimate from using this routine is 7.6%. This dilution factor was applied in Whittle during the optimization process with a zero dilution grade.

Mining of the orebody will have to contend with the voids left from the past underground mines. Mining of the crown pillars above the open stopes (Figure 5.5) will occur during open pit mining where blasted mineralization will fall into the open stopes. To account for this loss, crown pillars were modeled and mineralization in these was disregarded. Economic mineralization in crown pillars is estimated at 1.8 Mt at an average grade of 1.66 g/t Au. Recovery of this ore will be possible on lower benches but was not considered and is therefore treated as an ore loss. No additional mine recovery factors were applied.

Figure 5.5: Example of Modeled Crown Pillar



5.2.7 Cut-Off Grades

CoGs are calculated using parameters described in the above sections and presented in Table 5.5. The CoG is calculated on the basis of after-mining costs. CoG is that grade where revenue from produced gold equals the cost of treating the ore, refining and selling the gold. The CoG is defined as follows:

$$g = \frac{Cp + Ca + Cr + Com + Cscap + Cmc}{r * (P - Cs)}$$

g	Cut-off grade (g/t Au)
r	Metallurgical recovery (%)
P	Price of gold $/oz
Cs	Cost of selling (including royalties) $/oz Au
Cp	Total cost of processing $/t milled
Ca	Administration and services cost $/t milled
Cr	Re-handling cost $/t milled
Com	Additional ore mining cost $/t milled
Cscap	Sustaining capital in $/t milled
Cmc	Closure cost during life of mine in $/t milled

Table 5.5: Cut-Off Grade

Gold price	$/oz	775.00
Royalty rate	%	1.5
Royalty	$/oz	11.63
Refining cost	$/oz	2.00
Cost of selling	$/oz	13.63
Total processing cost	$/t milled	5.30
Metallurgical recovery	%	79
Mining dilution	%	7.6
Additional ore mining cost	$/t milled	-
Re-handling Cost	$/t milled	-
Administration & Services	$/t milled	0.70
Rehabilitation	$/t milled	0.18
Sustaining capital	$/t milled	0.20
Total ore based cost	**$/t milled**	**6.38**
Cut-Off Grade	**g/t Au**	**0.36**

Note: The metallurgical recovery is calculated on a block by block basis as a function of head grade and location in the pit as previously described. However, the average recovery at the cut-off grade for the four metallurgical domains is estimated at 79%.

Included in the ore based cost is the estimated rehabilitation cost per tonne of ore treated and a sustaining capital provision to maintain the operation such as the purchase of replacement equipment. These combined factors account for $0.38/t milled.

There is no systematic re-handling cost required to feed the mill. A re-handling cost is taken into account for stockpiling strategies.

The CoG is variable depending on the applicable royalty rate. The in-situ CoG based on the above ore based cost structure and gold price assumption varies between 0.35 g/t Au and 0.36 g/t Au. The CoG calculation using the same cost parameters as used in the financial analysis section remains at 0.36 g/t Au.

5.3 Optimization Results

Using the Whittle program, a series of potential pit shells were designed. The amount of waste and ore was determined for each shell along with the operating cash flow discounted at 5%. Two values were calculated for each pit shell. These calculations generated a best case scenario that assumes every internal pit shell is mined out sequentially and a worst case scenario that assumes mining takes place bench by bench for the given pit shell. The best case provides the upper NPV boundary limit while the worst case provides the lower NPV boundary limit. An intermediate result between these NPV boundary limits is obtained in practice by phasing the pit with push backs.

The pit shell that maximizes the NPV for the best case (pit shell 36) was used as a guideline for designing the final pit. The results for this pit shell are as follows and as illustrated in Figures 5.6 to 5.9 and Table 5.6:

- Total tonnes 482 Mt
- Ore tonnes 192.8 Mt
- Avg. Grade 1.03 g/t Au
- In-situ Ounces 6.39 M oz
- Strip ratio W:O 1.50 : 1.0
- Best Case (5% DCF) $1,751 M
- Worst Case (5% DCF) $1,512 M.

Table 5.6: Whittle Shells by Revenue Factor

Pit Number	Revenue Factor	Gold Price($/oz)	Tonnes Rock ('000)	Tonnes Ore ('000)	Tonnes Waste ('000)	Grade (g/t Au)	Ounces (k oz)
1	0.30	233	27,303	13,439	13,864	1.83	790
2	0.32	248	32,717	15,884	16,833	1.78	910
3	0.34	264	39,062	18,593	20,469	1.73	1,034
4	0.36	279	71,757	26,885	44,872	1.71	1,477
5	0.38	295	96,816	34,076	62,740	1.67	1,825
6	0.40	310	112,632	39,750	72,882	1.62	2,068
7	0.42	326	144,249	48,357	95,892	1.58	2,462
8	0.44	341	159,190	53,321	105,869	1.55	2,664
9	0.46	357	203,864	62,073	141,791	1.55	3,100
10	0.48	372	290,152	82,467	207,685	1.51	4,001
11	0.50	388	297,226	87,081	210,145	1.48	4,133
12	0.52	403	326,166	95,326	230,840	1.45	4,451
13	0.54	419	328,428	97,930	230,498	1.43	4,512
14	0.56	434	334,011	101,296	232,715	1.41	4,599
15	0.58	450	343,405	105,695	237,710	1.39	4,721
16	0.60	465	351,536	109,621	241,915	1.37	4,825
17	0.62	481	356,766	113,638	243,128	1.35	4,916
18	0.64	496	362,513	116,923	245,591	1.33	4,993
19	0.66	512	367,662	120,951	246,710	1.31	5,078
20	0.68	527	378,644	126,659	251,985	1.28	5,210
21	0.70	543	382,985	130,526	252,459	1.26	5,285
22	0.72	558	390,617	135,626	254,992	1.24	5,389
23	0.74	574	395,730	139,656	256,074	1.22	5,465
24	0.76	589	399,572	143,134	256,438	1.20	5,527
25	0.78	605	418,858	150,573	268,286	1.18	5,695
26	0.80	620	424,377	154,461	269,916	1.16	5,766
27	0.82	636	427,730	158,123	269,608	1.15	5,825
28	0.84	651	431,686	161,209	270,477	1.13	5,877
29	0.86	667	437,632	165,038	272,593	1.12	5,943
30	0.88	682	444,502	169,722	274,780	1.10	6,021
31	0.90	698	452,786	173,573	279,213	1.09	6,092
32	0.92	713	463,120	178,110	285,011	1.08	6,175
33	0.94	729	466,670	181,379	285,292	1.07	6,223
34	0.96	744	470,567	184,307	286,261	1.06	6,267
35	0.98	760	478,661	189,755	288,906	1.04	6,349
36	1.00	775	481,785	192,847	288,938	1.03	6,391
37	1.02	791	493,681	197,703	295,978	1.02	6,475
38	1.04	806	496,020	200,703	295,317	1.01	6,512
39	1.06	822	498,470	203,882	294,588	1.00	6,551
40	1.08	837	507,854	208,504	299,351	0.99	6,621
41	1.10	853	550,573	219,990	330,583	0.97	6,837
42	1.12	868	552,603	222,974	329,629	0.96	6,870
43	1.14	884	554,013	225,821	328,192	0.95	6,901
44	1.16	899	564,459	230,307	334,151	0.94	6,967
45	1.18	915	567,117	233,452	333,665	0.93	7,001
46	1.20	930	583,162	238,183	344,979	0.92	7,080
47	1.22	946	584,153	241,096	343,057	0.92	7,108
48	1.24	961	588,059	244,160	343,899	0.91	7,144
49	1.26	977	594,435	247,538	346,897	0.90	7,186
50	1.28	992	595,434	250,059	345,375	0.90	7,209
51	1.30	1,008	602,641	253,440	349,202	0.89	7,252
52	1.32	1,023	606,879	257,274	349,604	0.88	7,292
53	1.34	1,039	613,235	260,403	352,831	0.88	7,331
54	1.36	1,054	614,285	262,855	351,430	0.87	7,352
55	1.38	1,070	618,824	266,640	352,184	0.86	7,391
56	1.40	1,085	620,550	269,747	350,803	0.86	7,419
57	1.42	1,101	622,851	272,638	350,213	0.85	7,445
58	1.44	1,116	624,362	275,094	349,268	0.84	7,466
59	1.46	1,132	626,881	278,166	348,715	0.84	7,494
60	1.48	1,147	631,384	280,766	350,618	0.83	7,521

Figure 5.6: Pit by Pit Results at $775/oz Au

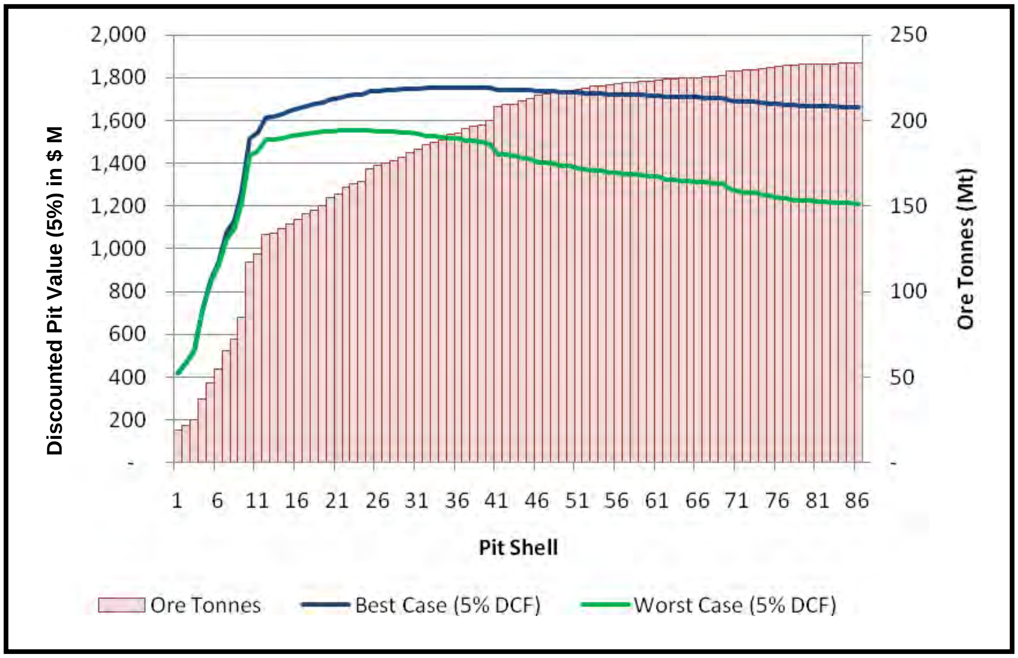


Figure 5.7: Size vs. Value at $775/oz Au

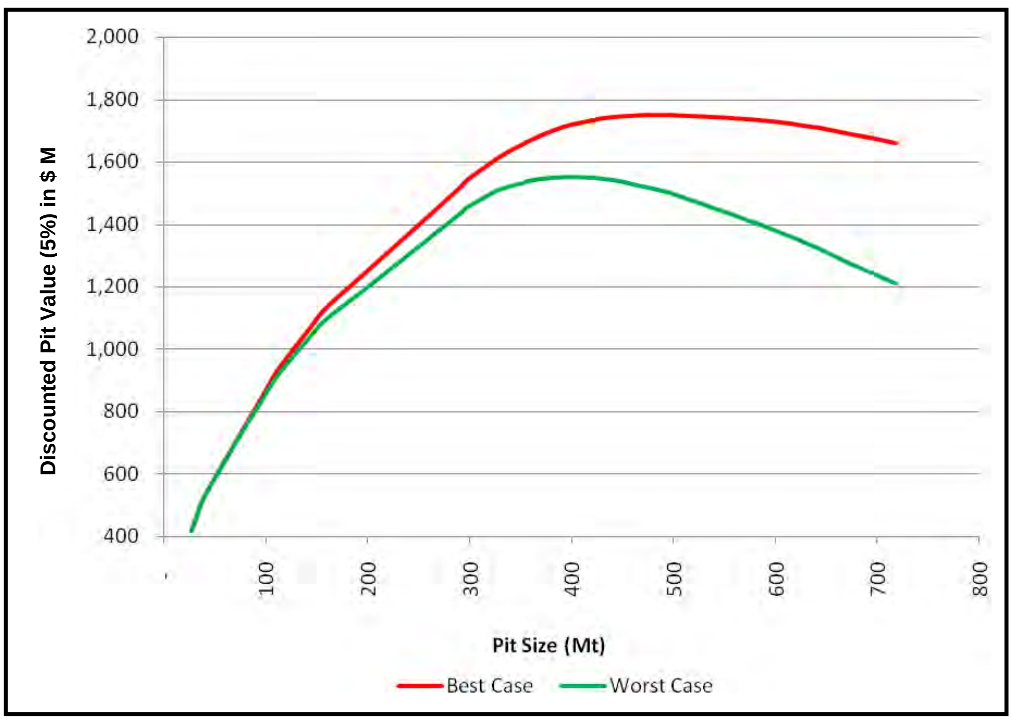


Figure 5.8: Sensitivity of Pit Shells to Gold Price

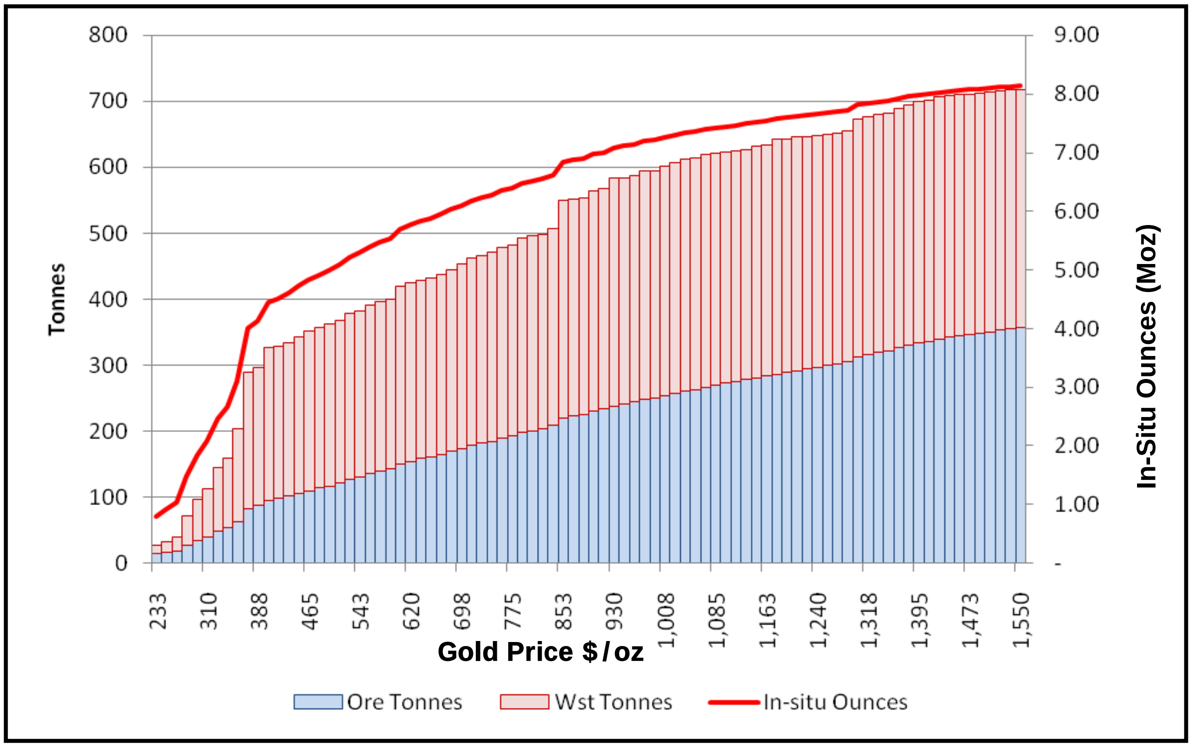
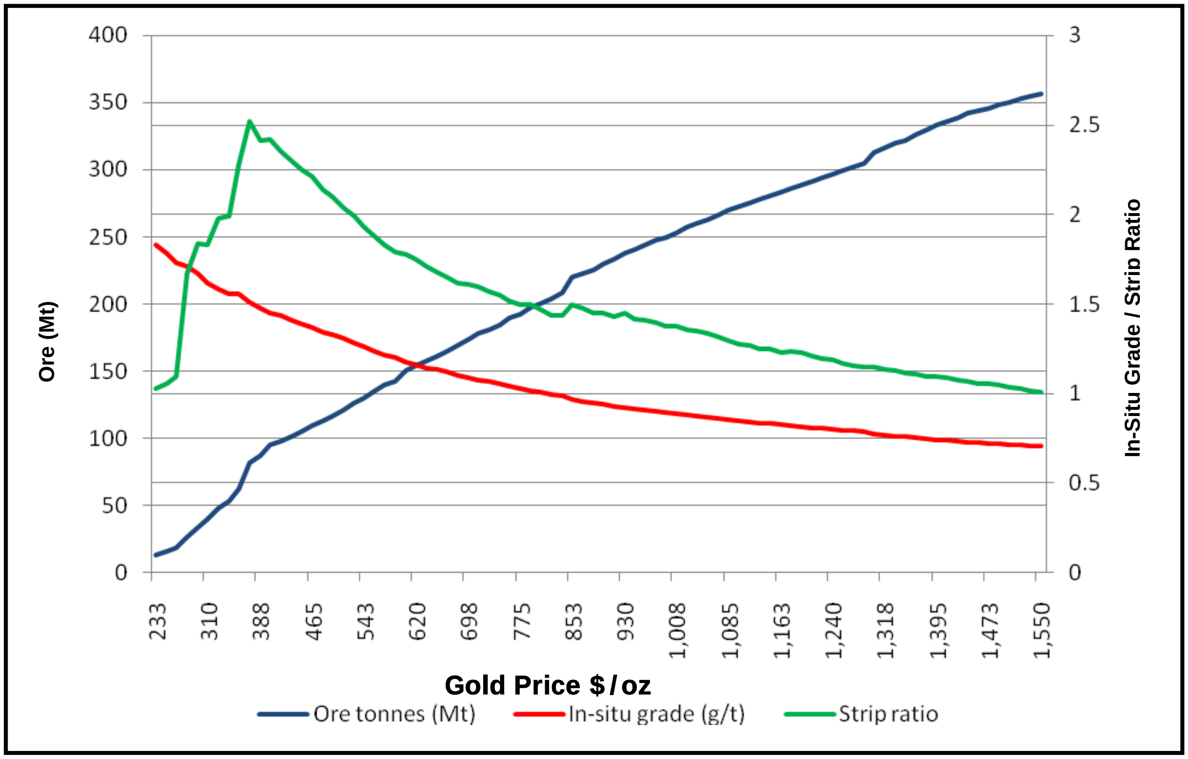


Figure 5.9: Sensitivity of Ore Tonnes, Grade and Strip Ratio to Gold Price



5.4 Mine Design

The optimal pit shells produced with the Lerchs-Grossmann algorithm were used as a guideline for the pit design. The pit design process consists of designing ramp accesses to the bottom of the pit using the geotechnical recommendations guiding the bench geometry.

The shell selection process involves analyzing a series of graphs, tables and figures generated from Whittle and Gemcom. The NPV graphs generated from Whittle have distinct characteristics showing major changes to the economics of the pit.

Phasing of the pit was investigated by using the pushback chooser module in Whittle which iteratively searches for the best combination of pushbacks to maximize NPV. A mining width is specified between pushbacks to assure enough working room for equipment. Considering the anticipated size of mining equipment, a mining width of 70 m was assured.

The Whittle shells selected are further analyzed in Gemcom to address mining practicalities of the selected shells such as the distances from the underground openings.

5.4.1 Bench Geometry

Design geometry of the mining benches has been recommended by Golder in their report titled "Feasibility-Level Pit Slope Design Criteria", July 2008. The bench geometry is specific to the design sectors established by Golder and previously described. The bench configuration is presented in Table 5.7.

Table 5.7: Bench Configuration

Wall	Operating Practices	Bench Configuration and Height (m)	Catch Bench Width (m)	Bench Face Angle (Deg.)	Design Inter-Ramp Slope Angle (deg.)
All sectors except Northeast sector	Buffer Blasting	Double Bench 2 X 10 m 20 m between catch benches	9	69	50
All sectors except Northeast sector	Pre-Split	Double Bench 2 X 10 m 20 m between catch benches	8.5	75	55
Northeast sector	Controlled Blasting to Break Cleanly Along Structure	Double Bench 2 X 10 m 20 m between catch benches	8	60	46

Pre-splitting is recommended where development of steeper, stable bench faces, with design bench face angles of 75° is required, accompanied by slightly narrower catch benches. A slope designed with 8.5 m catch benches at 20 m intervals is required for steep regions. Catch benches must always be sufficient to provide effective protection against rockfall.

The Bench Face Angle (BFA) and Internal Ramp Angle (IRA) are limited to structural control for slopes oriented within plus or minus 20° of the dip direction of structures involved in the planar failure modes, and plus or minus 45° of the trend of potential wedge failure modes.

5.4.2 Ramp and Haul Road Design

The ramps and haul roads are designed for the largest equipment being the Cat 793F haul truck. For double lane traffic, industry best-practice is to design a travelling surface of at least 3.5 times the width of the largest vehicle. Ramp gradients will be established at 10%.

A shoulder barrier or safety berm on the outside edge will be constructed of crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the Cat 793F tire (40R57) is 1.8 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.5:1 (H:V).

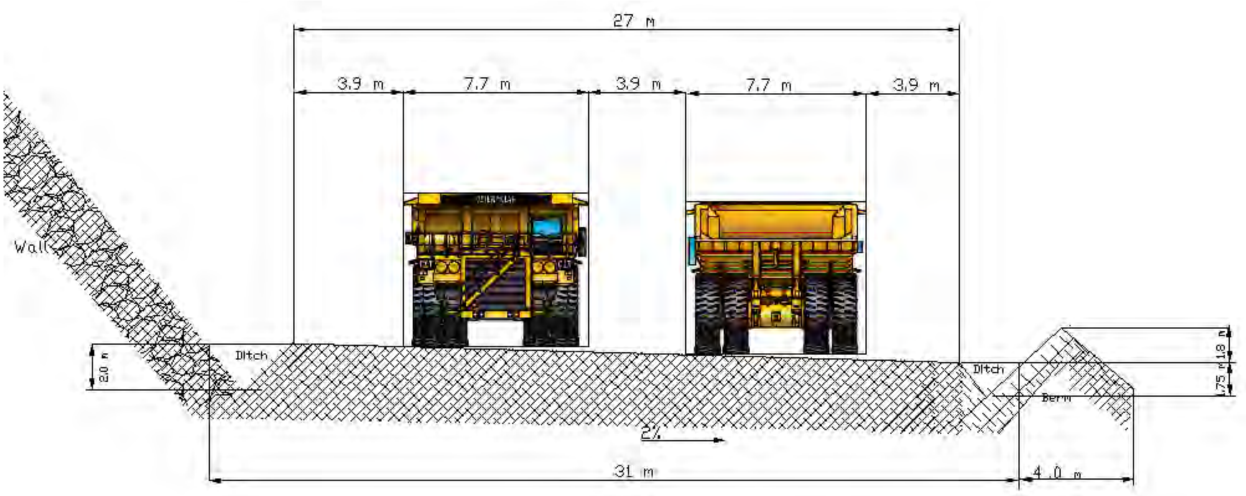
Ditches are planned on each side of the road to assure proper drainage of the running surface. The ditches along the ramp will be 2.0 m deep near the high wall and 1.75 m deep near the berm. To facilitate drainage of the roadway a 2% cross slope on the ramp is planned.

A summary of the ramp design parameters is presented in Table 5.8. Figure 5.10 presents an illustration of the in pit haul road design.

Table 5.8: Haulage Road Design Parameters

Parameter	Dimension (m)
Design Vehicle Overall Width (227-tonne truck)	7.7
Rolling Radius of Tire (40.00R57)	1.8
Berm Height	1.8
Berm Width	4.0
Ditch Width	2.0
Double Lane High Wall Haul Road (no drop off)	31
Double Lane High Wall Haul Road (drop off)	35
Single Lane High Wall Haul road	20

Figure 5.10: In-Pit Haul Road Design



5.4.3 Pit Designs

Mining will be accomplished with three phases to achieve the final pit limits. The objective of pit phasing is to improve economics by feeding the highest grade during the earlier years and to defer some stripping towards the latter years. Pit designs for each mining phase were realized using identical bench configurations as described in Section 5.4.1. The initial phase is centered on the historical underground stopes as the higher grade material is located in this vicinity.

The initial mining phase consists in effect of two pits with the eastern pit reaching a depth of about 100 m and the western pit reaching a depth of about 130 m from surface. The eastern pit measures approximately 700 m in length (along strike) and 340 m in width. The western pit measures 550m x 450m. During this initial phase the northern side of these pits are mined to the final limits requiring that pre-split blasting be practiced along these final walls. The design for the initial mining phase is presented in Figure 5.11.

The second mining phase establishes final pit limits along the north and west walls. The pit design is such that the ramp is common with the final ramp from elevation 220RL downwards which will reduce ramp development work. The second phase leaves an area in the southeast corner where lower grade material is present. The second phase has two pit bottoms that continue to center on the open stopes at the same location as the initial phase but reaches a depth of 220 m as presented in Figure 5.12.

The ramp location takes advantage of the natural plunge of the mineralization in the northeast corner. The location of the ramp in this sector produces minimal offsets from the optimal shell as evidenced by the increased catch bench widths left along the northeast wall. The location of the ramp in this sector also has the benefit of avoiding open stopes located within the pit. Two switchbacks are introduced in the ramp system, one at elevation 160RL and the other at elevation 120RL.

The final pit measures 2000 m along strike and 920 m in width with a final depth of 380 m. The final pit has two pit bottoms with the west pit bottom consisting of a 40 m cut from elevation 60RL. This pit bottom provides the opportunity of in pit dumping of waste from lower benches extracted from the central portion of the pit. However, this operational possibility was not assumed for planning and costing purposes. The ramp accessing the deepest central portion of the pit is narrowed to a single lane (20 m) for the last two benches from RL0. The final mining phase is presented in Figure 5.13.

The differential (design allowances) between the optimal $775 pit limit and the ultimate pit design is approximately 6% (510 Mt of rock in design vs. 481.9 Mt of rock in shell). This differential is for inclusion of ramps and final wall geometries in consideration of current geotechnical constraints. This low differential is evidence that the overall slope angles used for optimization allow for the inclusion of the ramp system and respect the recommended inter-ramp angles. Figures 5.14 and 5.15 illustrate the different profiles of the whittle shell and final pit design.

Figure 5.11: Initial Mining Phase

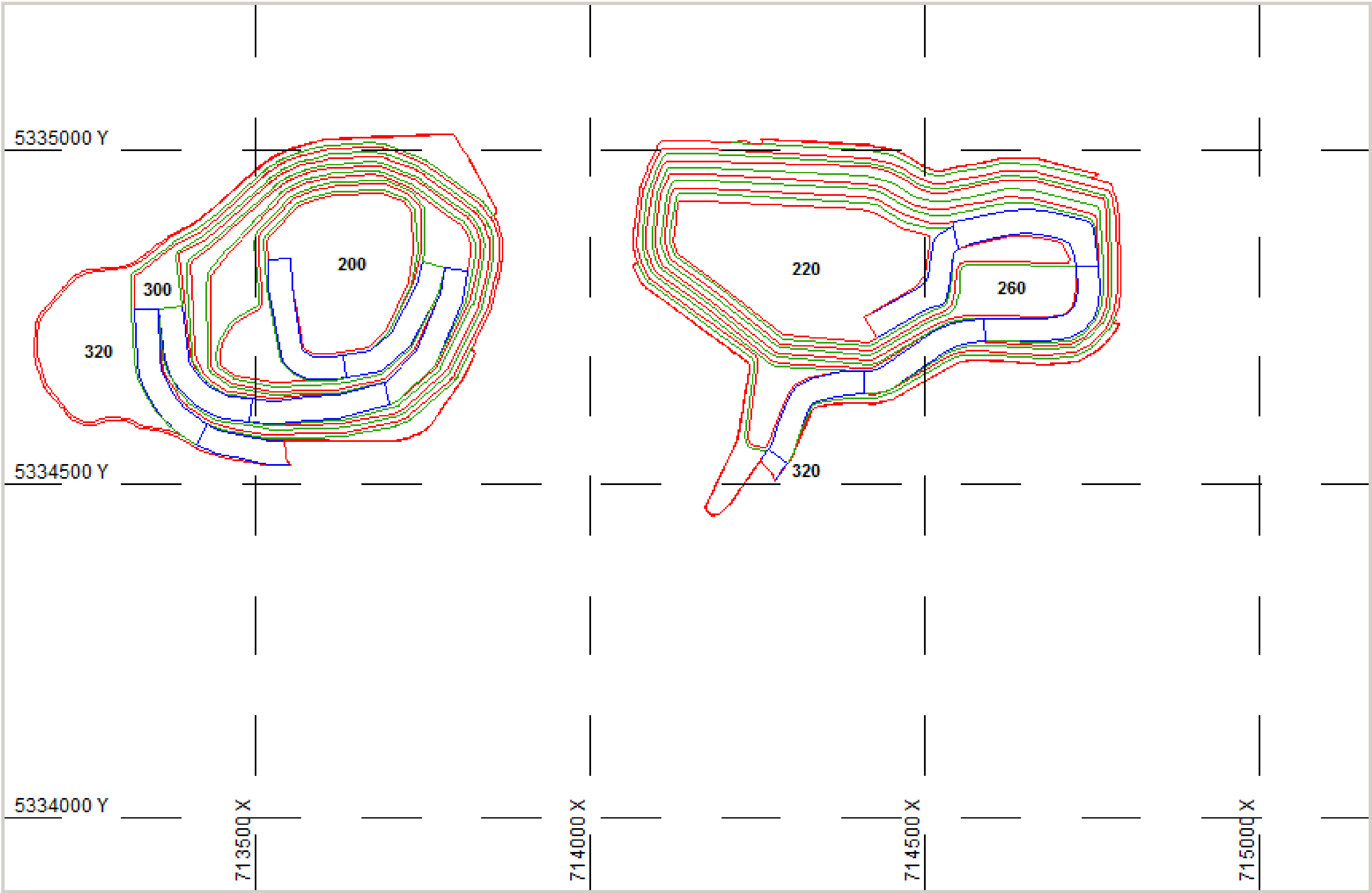


Figure 5.12: Second Mining Phase

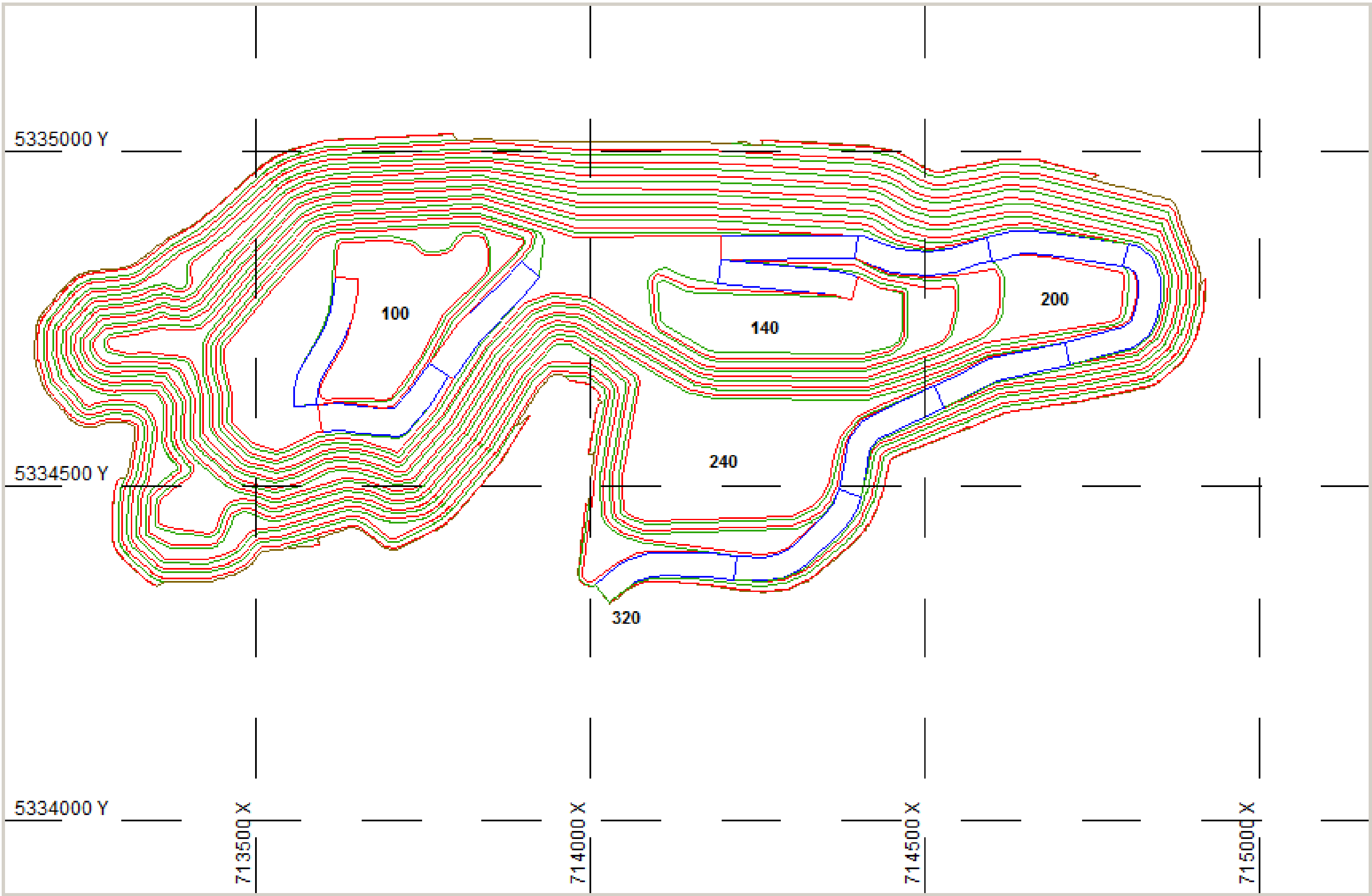


Figure 5.13: Final Mining Phase

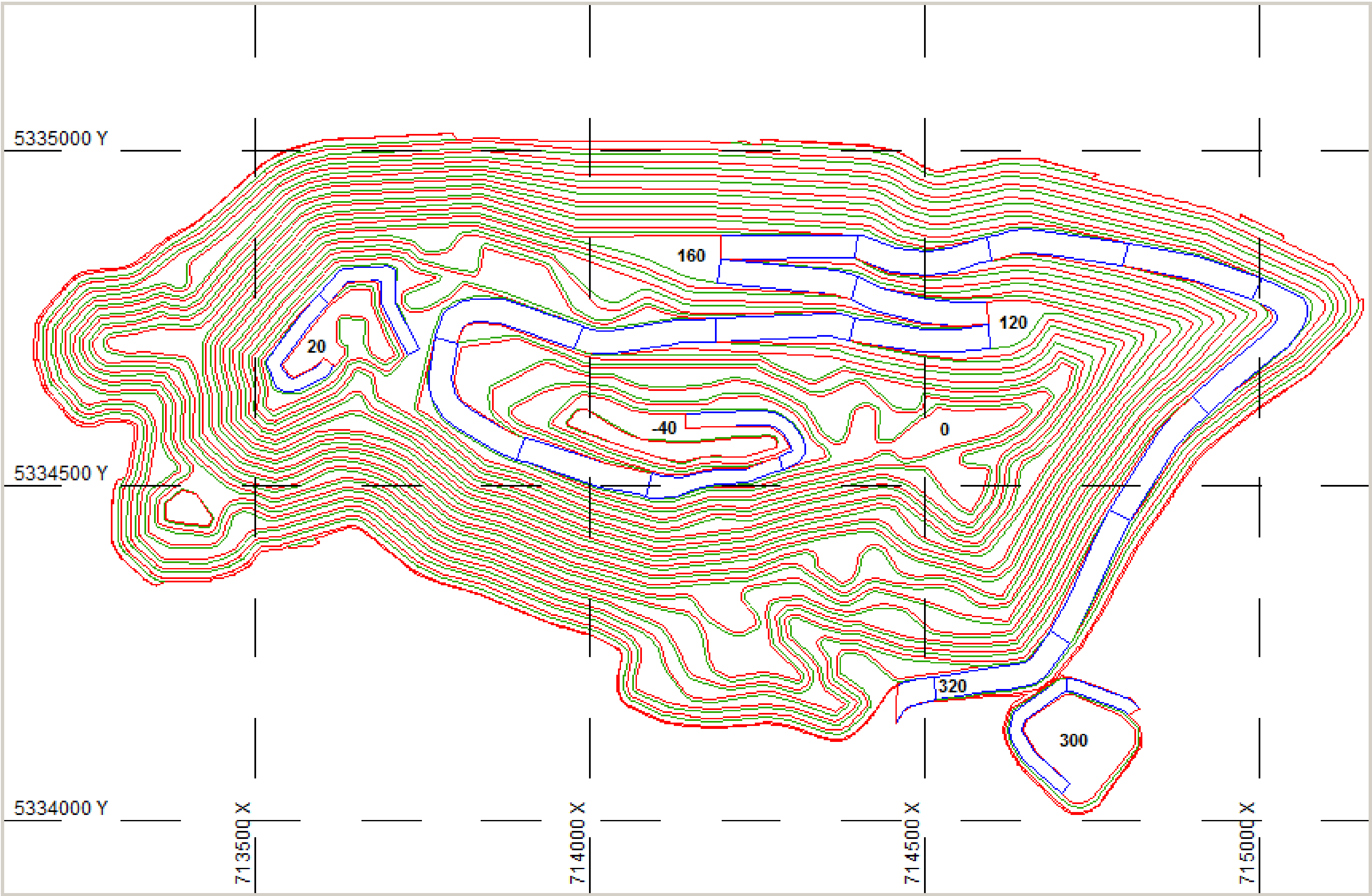


Figure 5.14: Section 714,190 Looking West of Final Pit Design and Optimal Whittle Shell

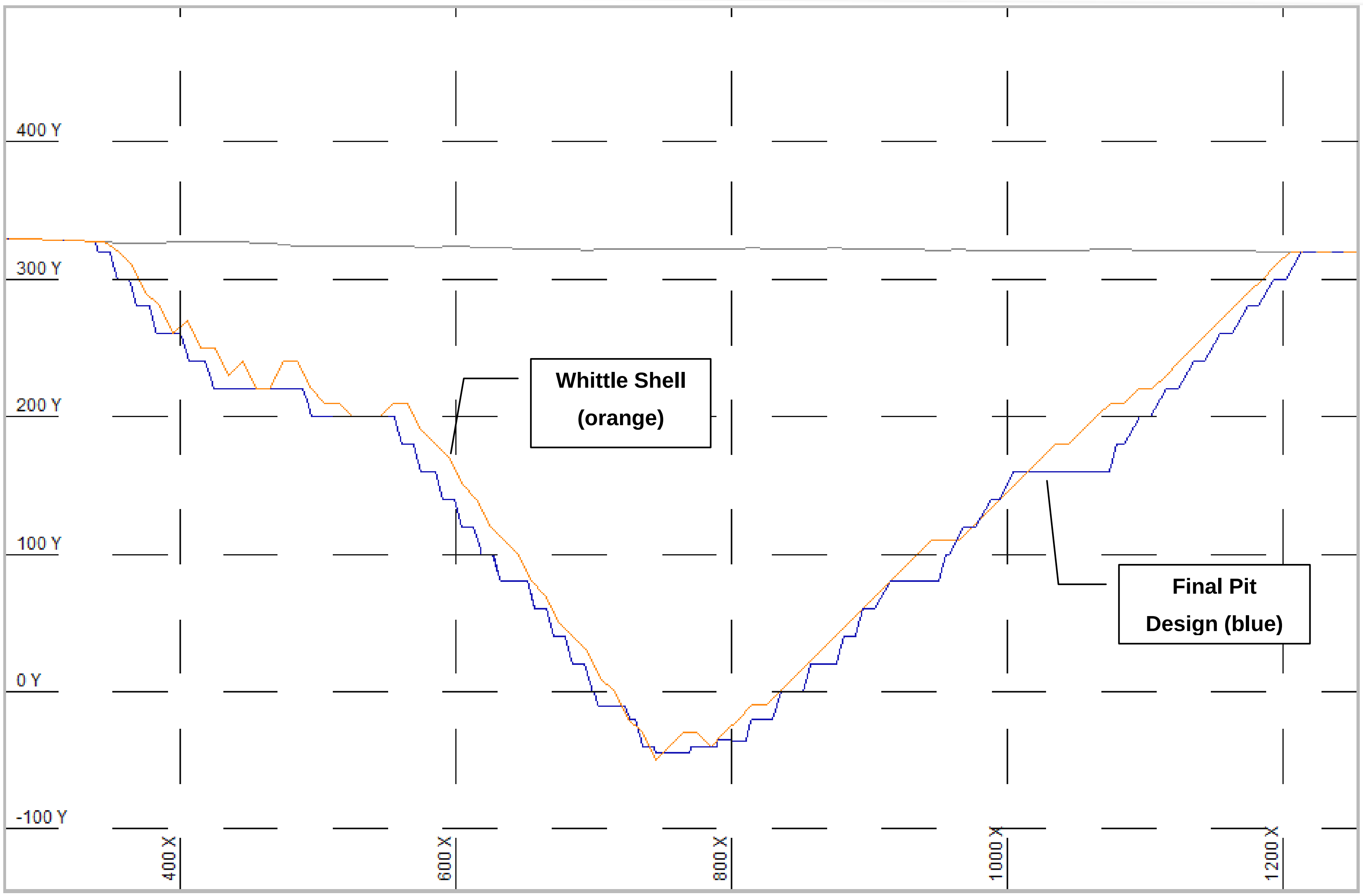
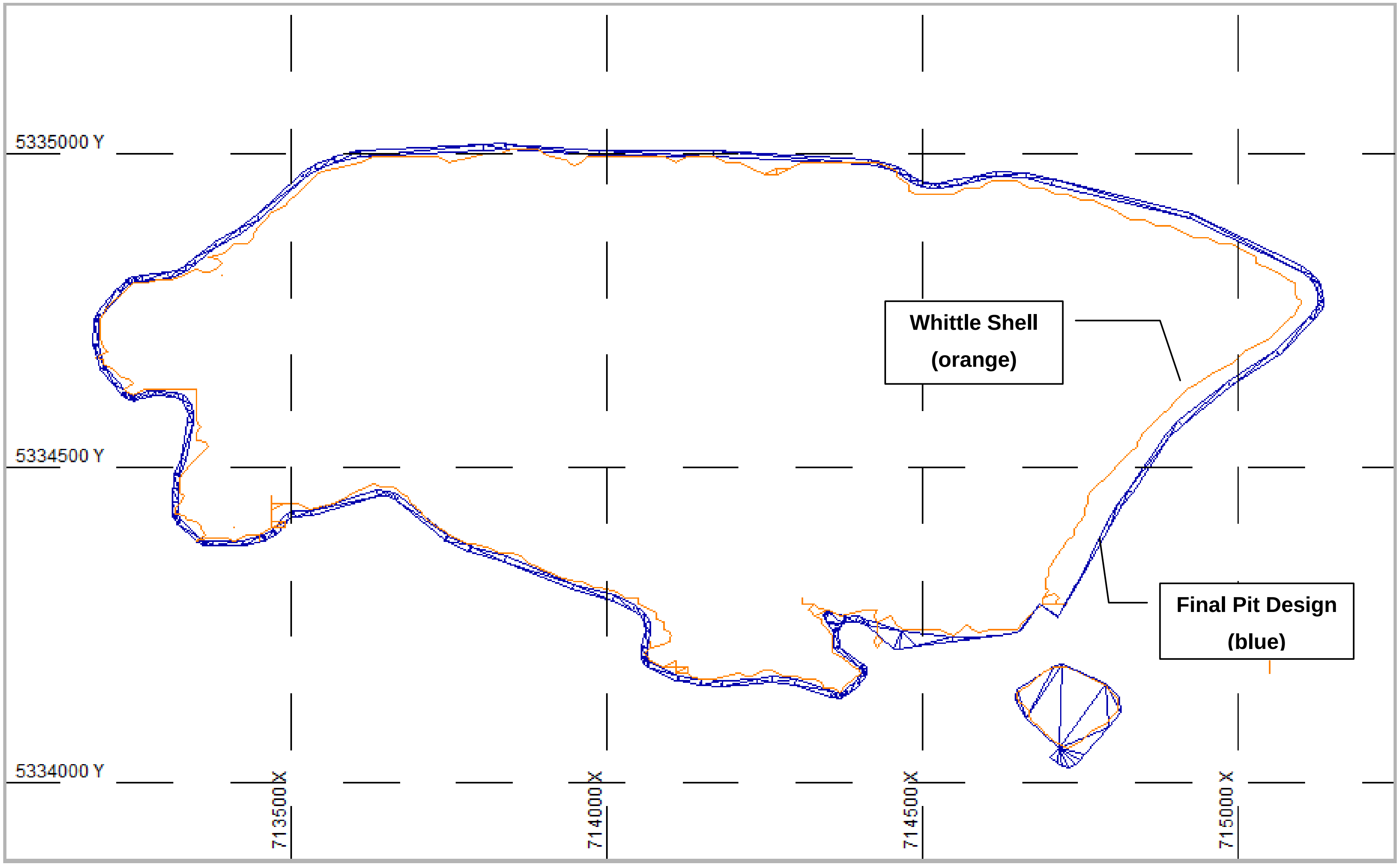


Figure 5.15: Plan View of Final Pit Design and Optimal Whittle Shell (300RL)



5.4.4 Mine Reserves

The reserves for the surface mine design are reported according to the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) standards. According to these standards, Resource Model blocks classified as Measured and Indicated are reported as proven and probable reserves. Owing to the above reporting standards, the Inferred Resources cannot be included as Reserves and so have not been included in the life of mine (LOM) schedule.

Mining reserves were estimated by defining ore outlines or mining shapes around economic mineralization for every 10 m bench. Only mineralization classified as measured and indicated was considered when designing the ore outlines. Tables 5.9 and 5.10 illustrate mine reserves respectively by category and by bench.

The ore outlines include a 1 m dilution envelope around economic ore blocks and also enclose marginal material surrounded by economic mineralization. The dilution envelope and enclosed waste in most cases is mineralized, with an associated dilution grade. The digitized ore outlines disregard isolated ore blocks which are treated as an ore loss. The ore tonnes and grades reported in the following tables are inclusive of dilution as estimated from the detailed mining shapes.

Figure 5.16 shows bench 200RL with the designed mining outline. Each block is 20m x 10m. Minimum economic ore widths are about 40 m and can be as wide as 250 m or wider.

The total proven and probable mine reserves are estimated at 183.3 Mt at 1.07 g/t Au for 6,283,290 in-situ ounces. The majority of these reserve tonnes (97.2%) are from Indicated resources.

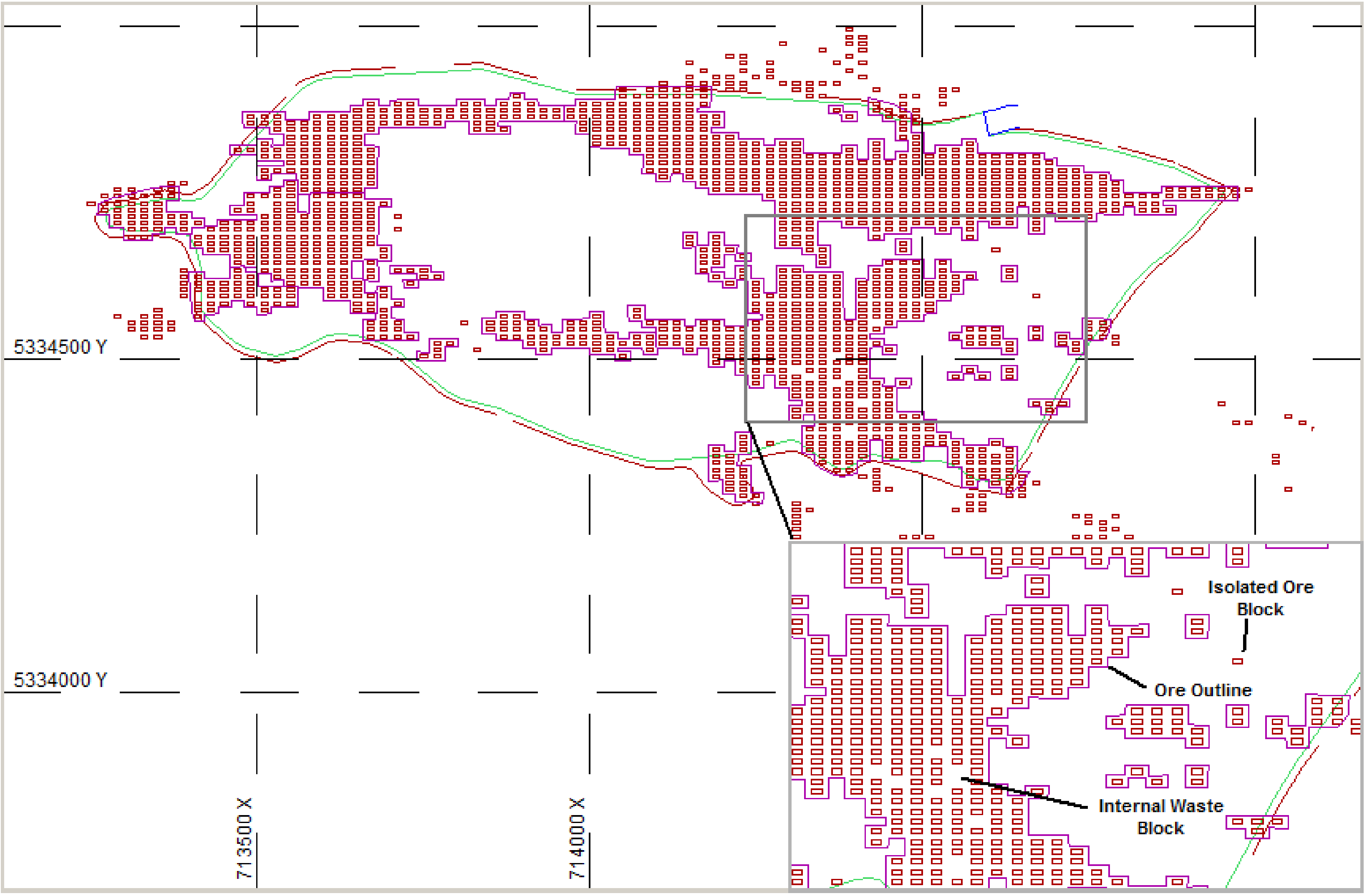
Table 5.9: Mine Reserves by Category

Category	Tonnes (kt)	Grade (g/t Au)	In-Situ Ounces (oz Au)
Proven	5,156	1.14	188,792
Probable	178,173	1.06	6,094,499
Proven & Probable	183,329	1.07	6,283,290

Table 5.10: Mine Reserves by Bench

Bench	Total (kt)	Waste (kt)	Strip Ratio (W/O)	Ore (kt)	Grade (g/t Au)	In-situ Gold (koz)
330	2,486	1,971	3.83	514	0.90	15
320	11,449	9,350	4.46	2,099	0.98	66
310	30,401	25,841	5.67	4,560	1.07	156
300	31,068	23,651	3.19	7,417	0.96	228
290	28,810	21,136	2.75	7,674	0.98	241
280	28,245	20,467	2.63	7,777	1.05	263
270	26,478	18,678	2.39	7,801	1.09	274
260	25,835	17,904	2.26	7,931	1.04	265
250	24,055	16,127	2.03	7,928	1.02	259
240	23,499	15,640	1.99	7,860	1.01	254
230	21,767	14,498	1.99	7,269	1.06	247
220	21,269	13,956	1.91	7,313	1.03	242
210	19,527	12,462	1.76	7,065	1.04	236
200	19,146	12,135	1.73	7,011	1.01	227
190	17,587	10,921	1.64	6,666	1.07	230
180	17,007	10,357	1.56	6,650	1.09	233
170	15,450	9,090	1.43	6,359	1.09	223
160	15,093	8,503	1.29	6,589	1.03	219
150	13,693	7,593	1.24	6,100	1.07	210
140	13,374	7,274	1.19	6,100	1.06	208
130	12,309	6,866	1.26	5,443	1.06	185
120	12,043	6,492	1.17	5,551	1.06	189
110	10,673	5,719	1.15	4,953	1.05	167
100	10,314	5,404	1.10	4,910	1.04	164
90	9,102	4,638	1.04	4,464	1.02	146
80	8,766	4,496	1.05	4,270	0.96	131
70	7,357	3,317	0.82	4,040	1.02	132
60	6,893	2,702	0.64	4,191	1.11	149
50	5,597	1,925	0.52	3,672	1.22	144
40	5,232	1,901	0.57	3,331	1.28	137
30	3,967	1,518	0.62	2,449	1.27	100
20	3,587	1,699	0.90	1,888	1.30	79
10	2,207	808	0.58	1,399	1.27	57
0	2,072	677	0.49	1,395	1.42	64
-10	1,246	212	0.21	1,034	1.61	54
-20	1,105	353	0.47	752	1.59	38
-30	713	213	0.42	501	1.83	29
-40	597	244	0.69	353	1.69	19
-50	111	62	1.25	49	1.56	2
Total	**510,130**	**326,801**	**1.78**	**183,329**	**1.07**	**6,283**

Figure 5.16: Ore Outline Bench 200RL



5.4.5 Mine Phases

The content of each mining phase is presented in Table 5.11. Phase 1 design is based on shell 4 (pit number) for a gold price of 279 $/oz. The strip ratio is lower with a higher in-situ grade than the subsequent two mining phases. The strip ratio of phase 2 and 3 are similar, but the last phase has a lower mined grade.

Table 5.11: Tonnage Mined by Phase

Mining Phase	Total (kt)	Waste (kt)	Strip Ratio (W/O)	Ore (kt)	Grade (g/t Au)	Ounces (koz Au)
Phase 1	75,338	43,845	1.39	31,493	1.19	1,200
Phase 2	159,086	103,717	1.87	55,368	1.17	2,079
Phase 3	275,706	179,239	1.86	96,467	0.97	3,004
Total	**510,130**	**326,801**	**1.78**	**183,329**	**1.07**	**6,283**

5.4.6 Overburden

An overburden surface was created from the drill hole information which corresponds to the ending position of drill hole casings installed and indicated on driller logs. The drill casings often extend into the bedrock and tend to overestimate the depth and therefore volume of overburden. The volume of overburden contained within the pit limits is estimated at 7.6 Mm^3 for an average thickness of 7 m. Mining costs including drilling and blasting were applied for the overburden which would be considered conservative.

Some of the overburden excavated will be used to construct a perimeter berm to serve as a visual barrier on the North pit rim (the linear park). The quantity of material required for this perimeter berm is estimated at 1.5 Mm^3 of fill material. The remaining overburden will be stockpiled on the outer edge of the planned waste dump such that this material can be used for future reclamation work if suitable.

5.4.7 Stockpile Design

Two stockpiles have been planned, one high grade (HG) and one low grade (LG). The low grade stockpile is located to the south east of the pit and the high grade to the south west of the pit. Material above 0.8 g/t Au is stockpiled in the HG pile and below 0.8 g/t Au in the LG pile. The stockpiles have a design height of 30 m and an angle of repose of 37 degrees.

The HG stockpile has a footprint of 19 hectares and a potential storage volume of 5.5 Mm^3 at full height. The LG stockpile has a footprint of 16.5 hectares with a potential storage volume of 4.1 Mm^3. These storage volumes are slightly greater than what is required by the plan and will allow for some operational flexibility. The HG stockpile peaks at 6.5 Mt at the end of the first year of operations for an estimated 3.1 Mm^3 assuming a 30% swell factor. The LG stockpile peaks at 6.0 Mt at the end of the seventh year of operation for an estimated volume of 2.8 Mm^3.

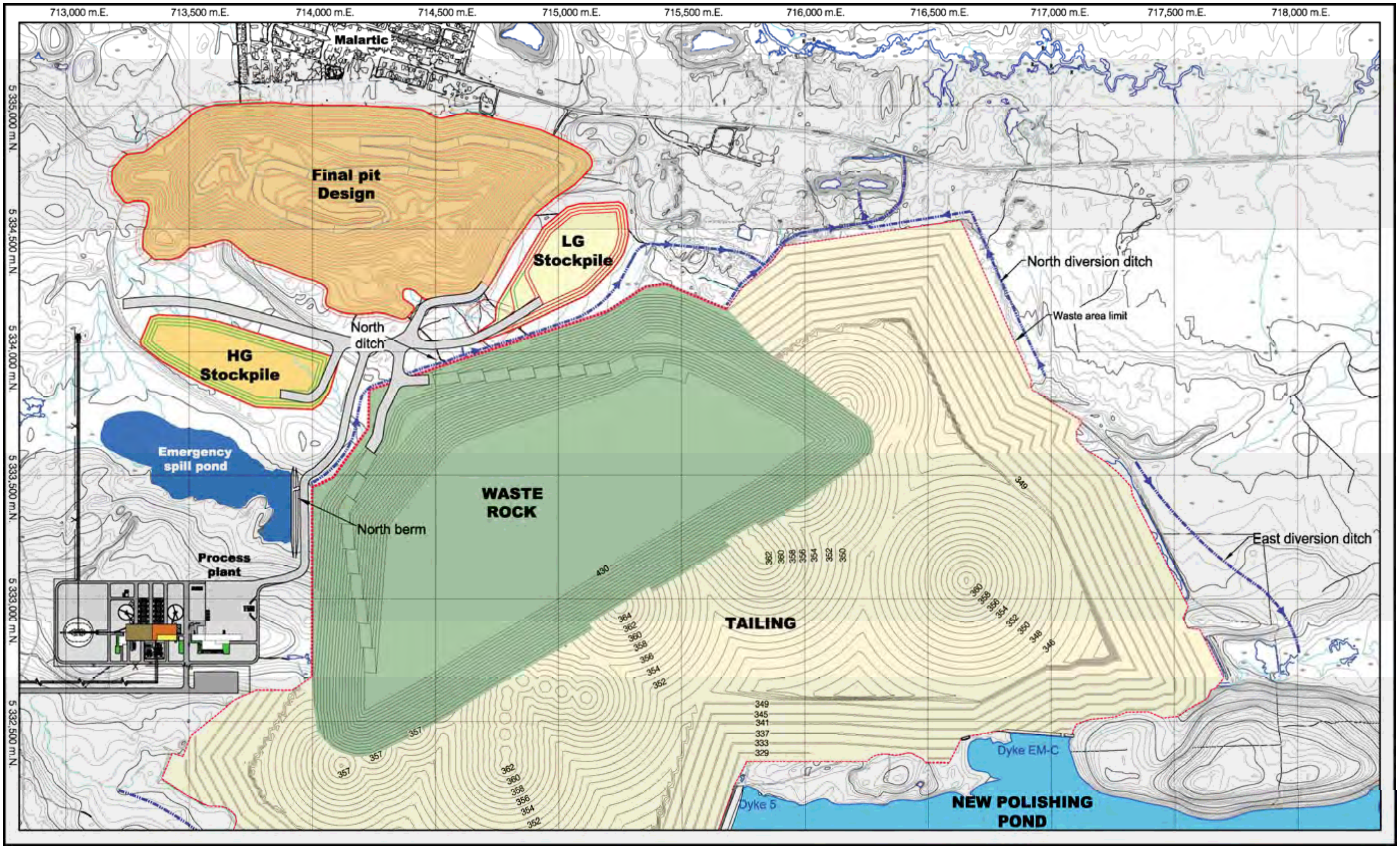
5.4.8 Waste Dump Design

Waste material will be co-disposed with tailings material in a central facility. The waste pile will serve to contain the thickened tailings on the north side of the facility. Over time the waste placement will occur over previously deposited tailings. A total of 326 Mt of waste are to be placed on the waste pile. An in-situ compacted density of 2.03 t/m^3 was used to estimate the storage volume required of 161 Mm^3.

The toe of the waste dump is located about 290 m from the pit crest. At the entrance of the waste dump two ramps with a 10% gradient will be developed leading in opposite directions which will reduce haulage distances on the dump.

The dump will be constructed in 10 m lifts with 12 m terraces, or step-ins between lifts for an overall slope angle of 22 degrees (2.4V:1H). A total of 10 lifts are required to store all waste material. The base of the waste dump is at about elevation 330RL and reaches elevation 427RL for a height of 97 m. The final waste dump design is illustrated in Figure 5.17.

Figure 5.17: Mine General Arrangement with Stockpiles and Waste Dump



5.5 Mine Production Schedule

The mine production schedule was developed to feed the mill at a nominal rate of 55,000 t/d beginning in the second quarter of 2011 (Year 1). Therefore, the pre-production period as presented in the following Tables and Figures includes the first quarter of 2011.

The mine production schedule includes a twelve month pre-production period during which time the overburden will be pre-stripped and some ore will be stockpiled. During the pre-production period a total of 25.1 Mt will be mined, including 6.4 Mt of ore at an average grade of 1.07 g/t Au and 18.7 Mt of waste.

The ore stockpile built up during pre-production will be drawn down in part during Year 2 (2012) and Year 3 (2013).

The annual mining rate is initially set at 51 Mt (142.5 kt/d) for the first year of operation and increases to 64 Mt (178.8 kt/d) for the following four years and then decreases. The following tables (Table 5.12 to 5.15) present the mine production schedule, stockpiling schedule, stockpile inventory status and mill feed schedule. Table 5.16 presents the bench extraction schedule per period. The mine plans from the end of the pre-production period to the end of mine life in year 10 are shown in Figures 5.18 to 5.23.

Table 5.12: Mine Production Schedule

Period	Ore (kt)	Ore Grade (g/t Au)	Waste Rock (kt)	Ore Reclaim (kt)	Material Moved (kt)	Strip Ratio (W/O)
Pre-prod (Yr -1)	2,829	1.10	9,481	-	12,310	3.35
Pre-prod (Yr 1)	3,553	1.05	9,194	-	12,747	2.59
Sub-total Pre-prod	**6,382**	**1.07**	**18,675**	**-**	**25,057**	**2.93**
Yr 1	16,538	1.20	21,759	0	38,298	1.32
Yr 2	18,685	1.10	45,286	1,390	65,361	2.42
Yr 3	16,124	1.05	47,893	3,951	67,968	2.97
Yr 4	20,253	1.07	43,697	0	63,950	2.16
Yr 5	22,539	1.10	41,472	0	64,011	1.84
Yr 6	21,418	0.93	38,239	0	59,657	1.79
Yr 7	19,662	0.88	34,137	1,213	55,012	1.74
Yr 8	21,673	1.01	23,044	0	44,717	1.06
Yr 9	17,366	1.22	11,517	2,709	31,592	0.66
Yr 10	2,689	1.66	1,083	4,983	8,755	0.40
Sub-total Prod.	**176,947**	**1.07**	**308,126**	**14,247**	**499,320**	**1.74**
Total	**183,329**	**1.07**	**326,801**	**14,247**	**524,376**	**1.78**

Table 5.13: Tonnage from Pit to Stockpile

Period	HG Stockpile		LG Stockpile		Combined Stockpiles	
	(kt)	(g/t Au)	(kt)	(g/t Au)	(kt)	(g/t Au)
Pre-prod (Yr -1)	1,436	1.59	1,393	0.60	2,829	1.10
Pre-prod (Yr 1)	3,553	1.05			3,553	1.05
Sub-total Pre-prod	**4,989**	**1.20**	**1,393**	**0.60**	**6,382**	**1.07**
Yr 1	1,482	1.20			1,482	1.20
Yr 2						
Yr 3						
Yr 4	178	1.07			178	1.07
Yr 5			2,464	0.56	2,464	0.56
Yr 6			1,343	0.57	1,343	0.57
Yr 7			800	0.58	800	0.58
Yr 8	1,598	1.01			1,598	1.01
Yr 9						
Yr 10						
Sub-total Prod.	**3,258**	**1.10**	**4,607**	**0.56**	**7,865**	**0.78**
Total	**8,247**	**1.16**	**6,000**	**0.57**	**14,247**	**0.91**

Table 5.14: Stockpile Inventory

Period	HG Stockpile		LG Stockpile		Combined Stockpiles	
	(kt)	(g/t Au)	(kt)	(g/t Au)	(kt)	(g/t Au)
Pre-prod (Yr -1)	1,436	1.59	1,393	0.60	2,829	1.10
Pre-prod (Yr 1)	4,989	1.20	1,393	0.60	6,382	1.07
Yr 1	6,471	1.20	1,393	0.60	7,864	1.10
Yr 2	5,081	1.20	1,393	0.60	6,474	1.07
Yr 3	1,130	1.20	1,393	0.60	2,523	0.87
Yr 4	1,308	1.19	1,393	0.60	2,701	0.88
Yr 5	1,308	1.19	3,857	0.57	5,165	0.73
Yr 6	1,308	1.19	5,200	0.57	6,508	0.69
Yr 7	95	1.19	6,000	0.57	6,095	0.58
Yr 8	1,693	1.02	6,000	0.57	7,693	0.67
Yr 9	0	0.00	4,983	0.57	4,983	0.57
Yr 10						

Table 5.15: Mill Feed Schedule

Period	Tonnes from Pit		Tonnes from Stockpile		Mill Feed	
	(kt)	(g/t Au)	(kt)	(g/t Au)	(kt)	(g/t Au)
Yr 1	15,056	1.20			15,056	1.20
Yr 2	18,685	1.10	1,390	1.20	20,075	1.11
Yr 3	16,124	1.05	3,951	1.20	20,075	1.08
Yr 4	20,075	1.07			20,075	1.07
Yr 5	20,075	1.16			20,075	1.16
Yr 6	20,075	0.96			20,075	0.96
Yr 7	18,862	0.89	1,213	1.19	20,075	0.91
Yr 8	20,075	1.01			20,075	1.01
Yr 9	17,366	1.22	2,709	0.85	20,075	1.17
Yr 10	2,689	1.66	4,983	0.57	7,672	0.95
Total	**169,082**	**1.08**	**14,247**	**0.91**	**183,329**	**1.07**

Table 5.16: Bench Extraction Schedule by Period

Tonnes by Bench (kt)	Pre-prod Yr-1	Pre-prod Yr 1	Prod. Yr 1	Prod. Yr 2	Prod. Yr 3	Prod. Yr 4	Prod. Yr 5	Prod. Yr 6	Prod. Yr 7	Prod. Yr 8	Prod. Yr 9	Prod. Yr 10	Total
330	890	1,317	-		279								2,486
320	4,210	2,741	-		4,498								11,449
310	3,496	4,506	2,428	10,250	5,422	4,300							30,401
300	3,714	4,184	2,373	10,769	5,287	4,740							31,068
290			8,713	10,552		9,546							28,810
280			8,180	10,514		9,550							28,245
270			7,129	2,946	6,840	3,570	5,993						26,478
260			6,583	2,899	6,848	3,513	5,992						25,835
250			1,535	3,638	9,618		9,265						24,055
240			1,356	3,381	9,562		9,201						23,499
230				3,539	7,905		5,353	4,970					21,767
220				3,232	7,758		5,318	4,960					21,269
210				1,204		8,518		9,805					19,527
200				1,047		8,332		9,767					19,146
190						6,182	1,568	9,836					17,587
180						5,699	1,577	9,732					17,007
170							5,938	1,798	7,713				15,450
160							5,680	1,827	7,586				15,093
150							4,217		9,476				13,693
140							3,909		9,465				13,374
130								2,447	9,863				12,309
120								2,348	9,695				12,043
110								1,163		9,510			10,673
100								1,005		9,309			10,314
90										9,102			9,102
80										8,766			8,766
70										3,455	3,902		7,357
60										3,152	3,742		6,893
50										769	4,829		5,597
40										655	4,577		5,232
30											3,967		3,967
20											3,587		3,587
10											2,207		2,207
0											2,072		2,072
-10												1,246	1,246
-20												1,105	1,105
-30												713	713
-40												597	597
-50												111	111
Total	**12,310**	**12,747**	**38,298**	**63,971**	**64,016**	**63,950**	**64,011**	**59,657**	**53,799**	**44,717**	**28,883**	**3,772**	**510,130**

Figure 5.18: End of Pre-Production (2010)

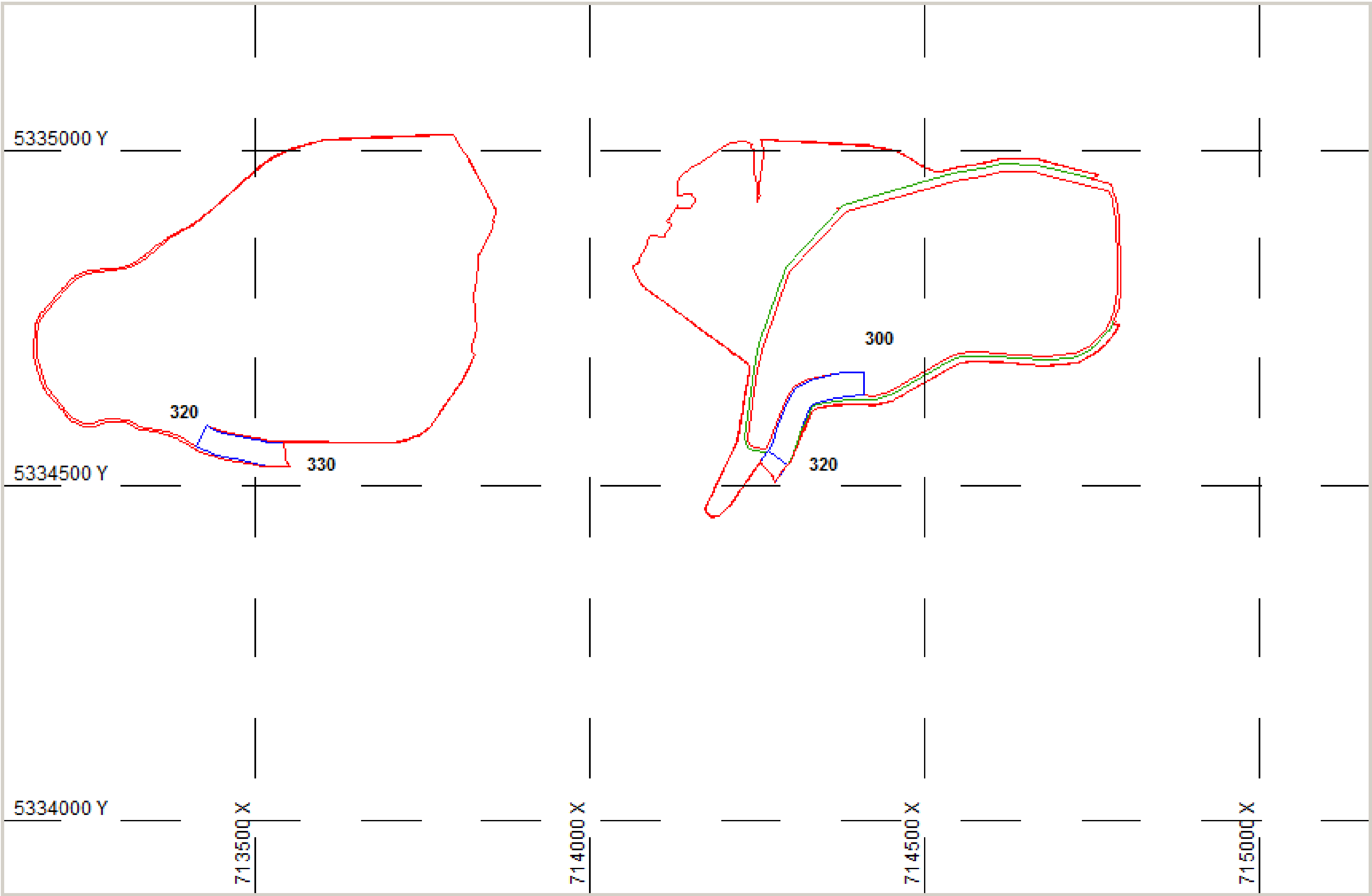


Figure 5.19: End Year 1 (2011)

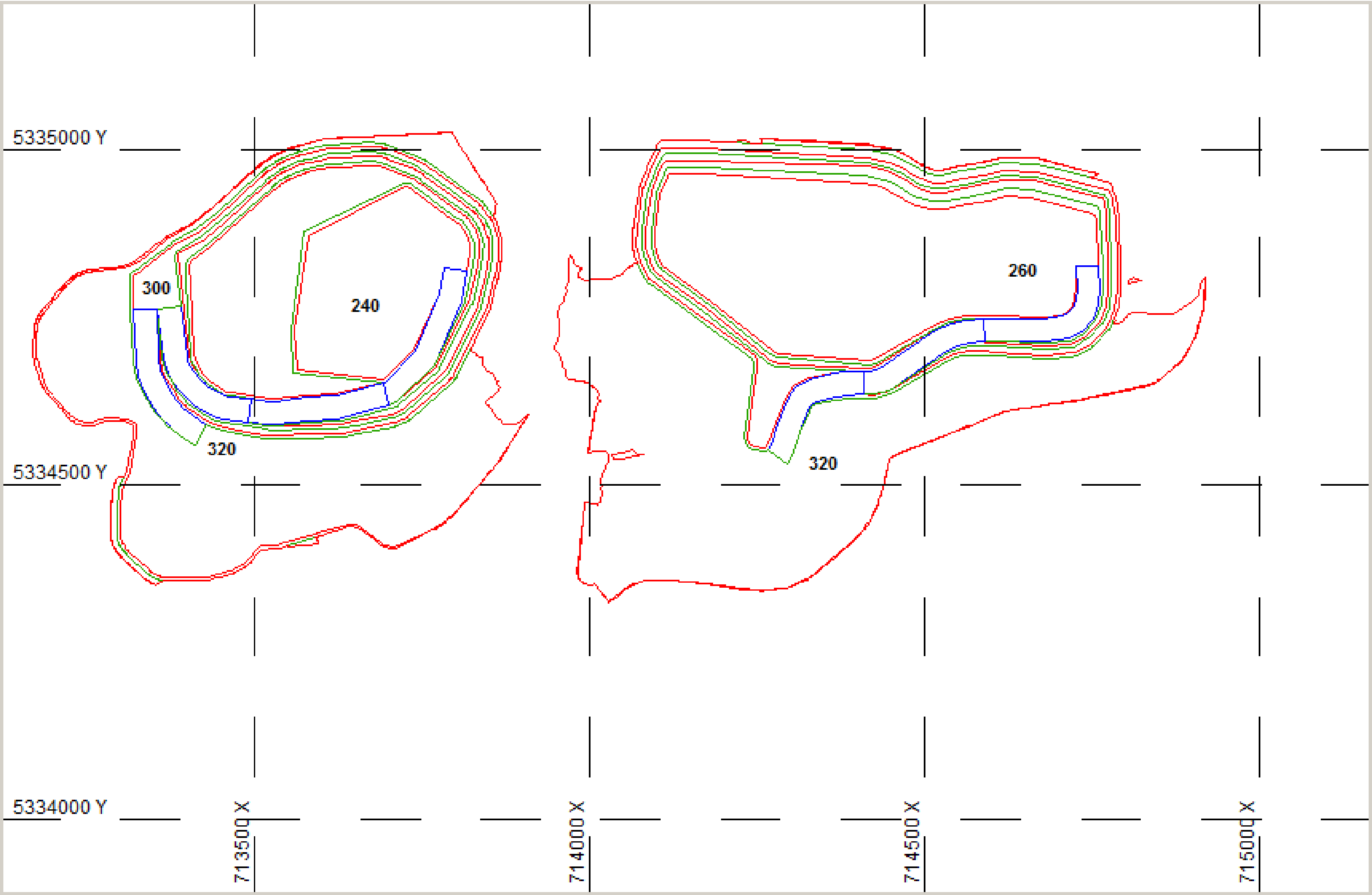


Figure 5.20: End Year 3 (2013)

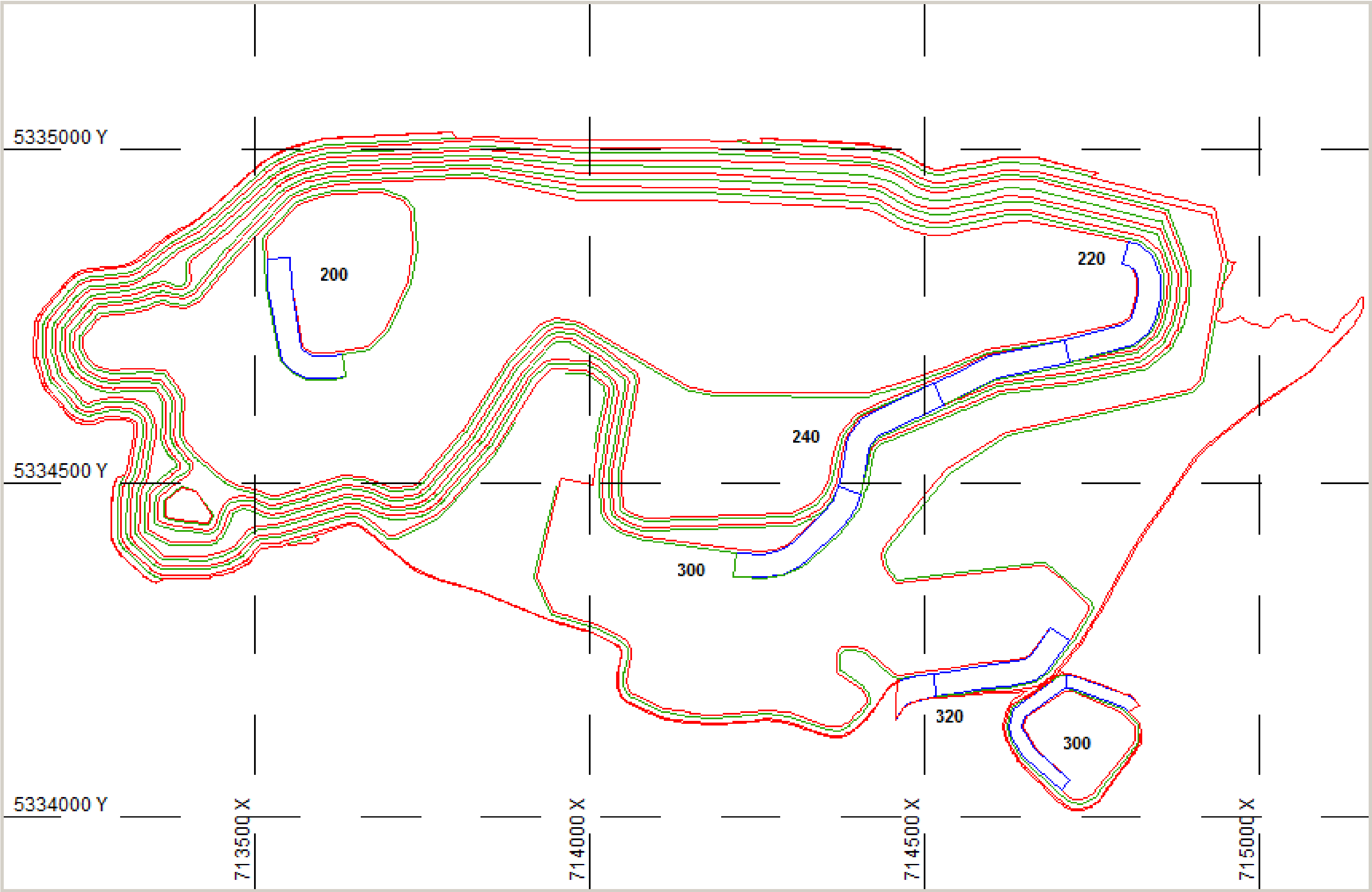


Figure 5.21: End Year 5 (2015)

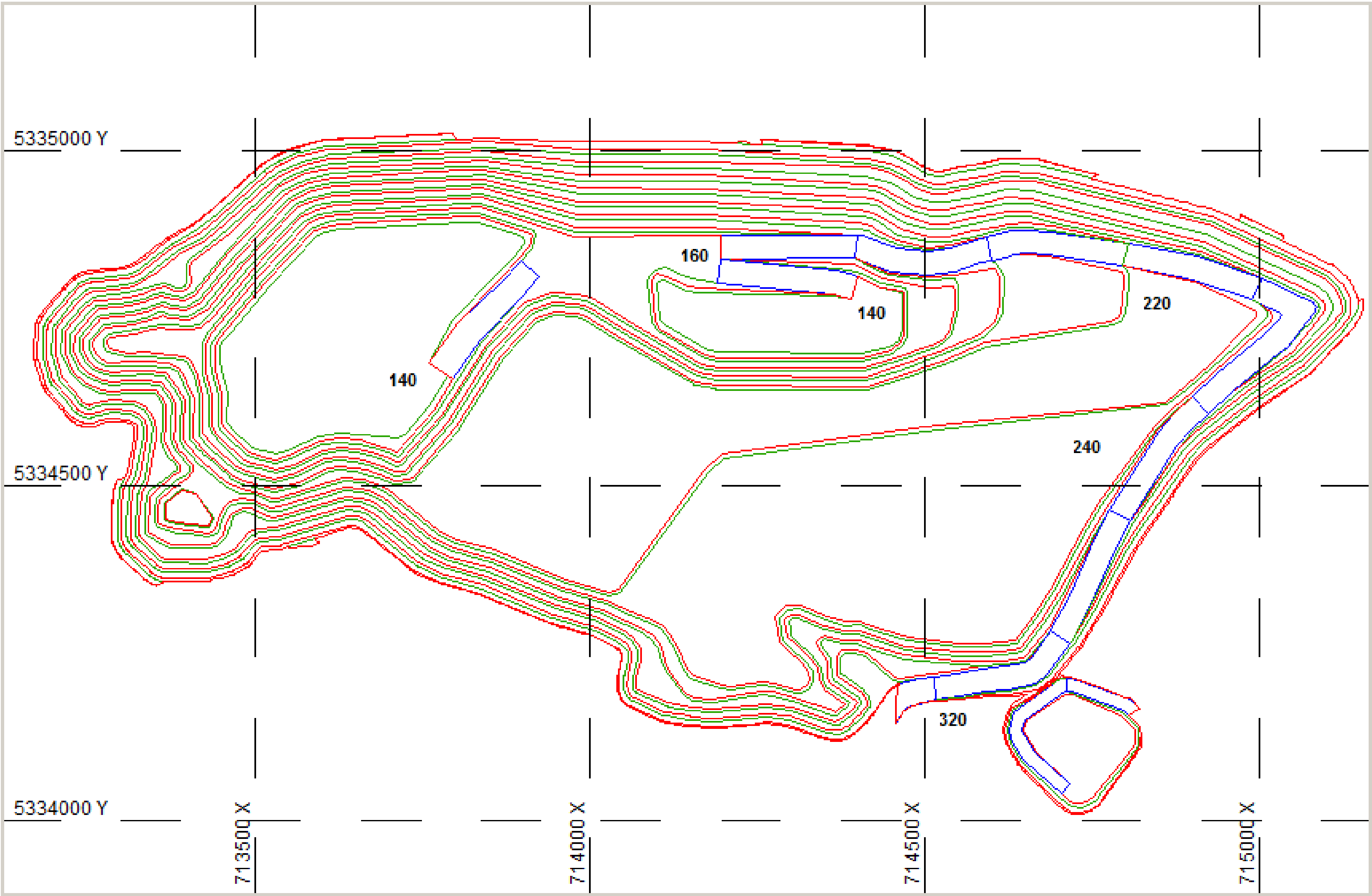


Figure 5.22: End Year 7 (2017)

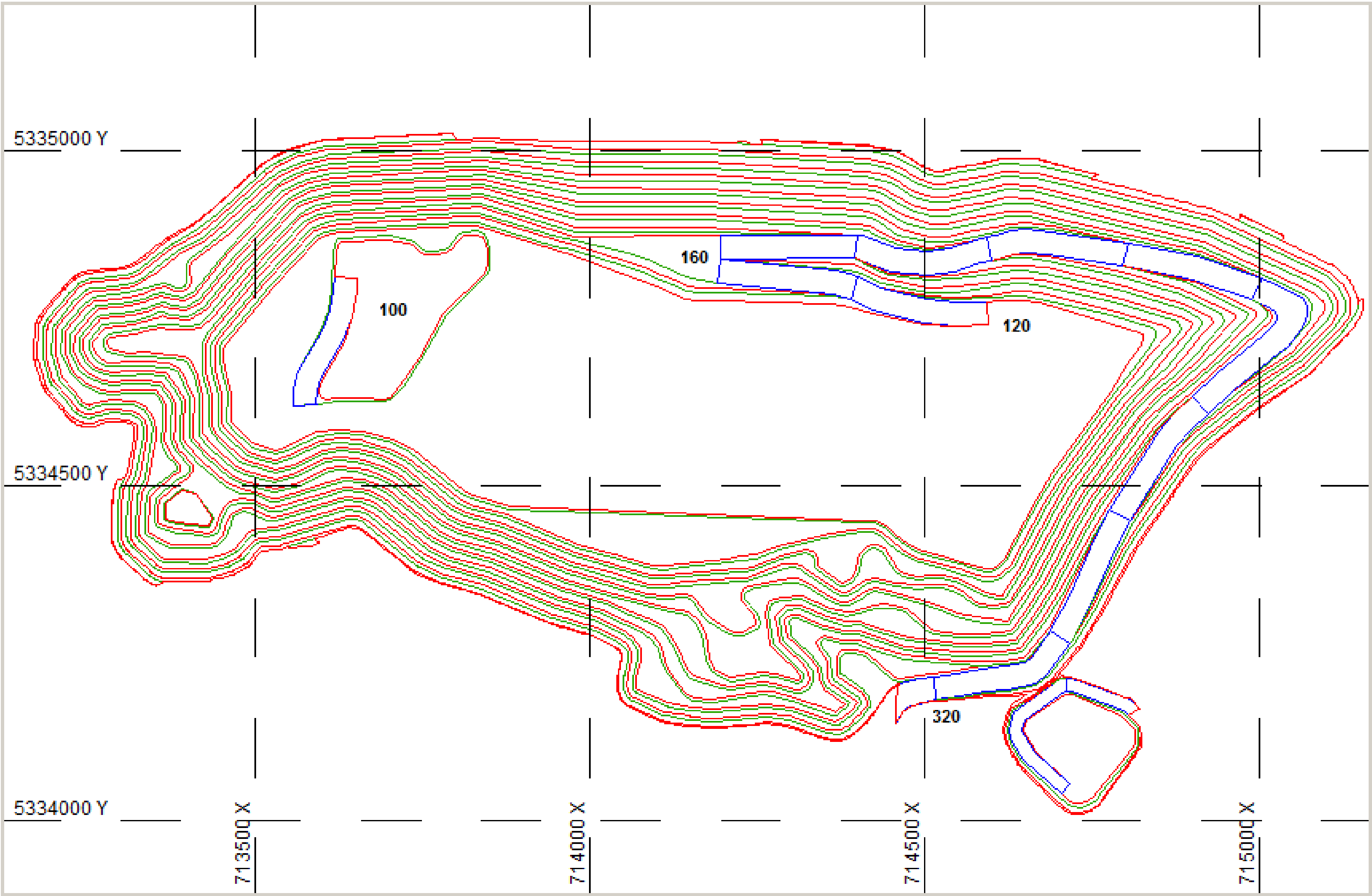
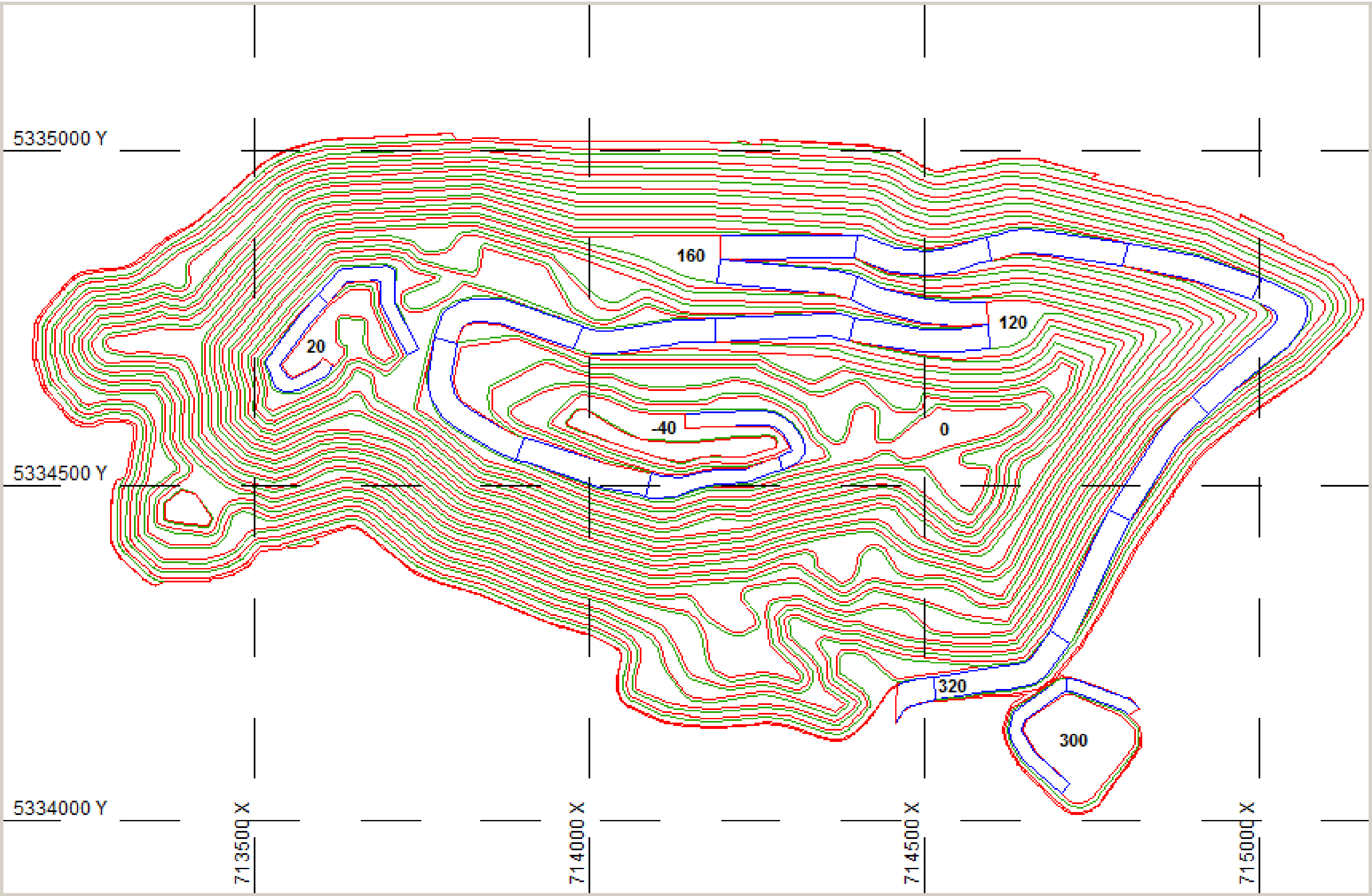


Figure 5.23: End Year 10 (2020), Final Pit



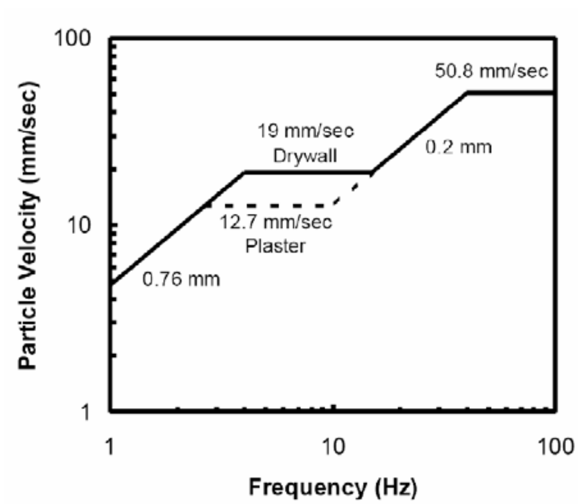
5.6 Mine Operations and Equipment Selection

5.6.1 Drilling and Blasting

Drill pattern design has been dictated by the need to control blast induced vibrations and airblast overpressure on the neighboring town of Malartic. Ground vibrations and airblast overpressure are regulated by Directive 019 of the MSDEP. The regulations of Directive 019 are highly influenced by the US Bureau of Mines (USBM) research on the effect of vibrations on residential buildings.

Figure 5.24 provides a threshold damage limit, defined as cosmetic damage (e.g. cracking) within the structure, categorized by both frequency ranges and particle velocity.

Figure 5.24: USBM Vibration Criteria (after Siskind et al, 1980)



Directive 019 requires that the maximum particle velocity measured at the point of impact be less than 12.7 mm/s and that the maximum airblast overpressure be less than 128 dB. These levels can be maintained by limiting the amount of explosives detonated per delay interval.

A study conducted by Géophysique GPR International Inc. as part of the Environmental Impact Assessment has provided guidelines as to the permissible amount of explosives per delay in order to meet the criteria of Directive 019.

The technical services groups of the leading explosives providers were consulted to provide their recommendations for the best blast patterns to implement in order to conform to the regulations. To do so, several blast patterns will be implemented whereby the explosives charge per delay will increase as mining progresses to the south by increasing hole diameter and bench height.

5.6.1.1 Production Blast Patterns

Four zones have been designated based on the distance from the nearest houses where the drill pattern provides a suitable explosives charge. The distances of the blast zone limits from the town are presented in Table 5.17 and the blast zone locations are shown in Figure 5.25.

Table 5.17: Blast Zones and Properties

Blast Limit	Distance (m)	Hole Diameter (mm)	Bench Height (m)
Zone 1	<159	114	5
Zone 2	<236	140	5
Zone 3	<337	140	10
Zone 4	>337	216	10

Blast zones 1 and 2 are developed with 5 m bench heights in order to reduce the explosives column height. Blast zone 1 is further limited with the use of 114 mm diameter holes which increase to 140 mm for zones 2 and 3. Bench height increased to 10 m for zones 3 and 4. Blast zone 4 sees the implementation of the larger 216 mm production blast holes. The blast patterns are presented in Table 5.18.

The blast pattern designs and explosives column height results in a powder factor ranging from 0.26 to 0.30 kg/t of rock. The powder factor is higher at 0.30 kg/t for the 10 m bench heights.

Table 5.18: Production Blast Pattern Parameters

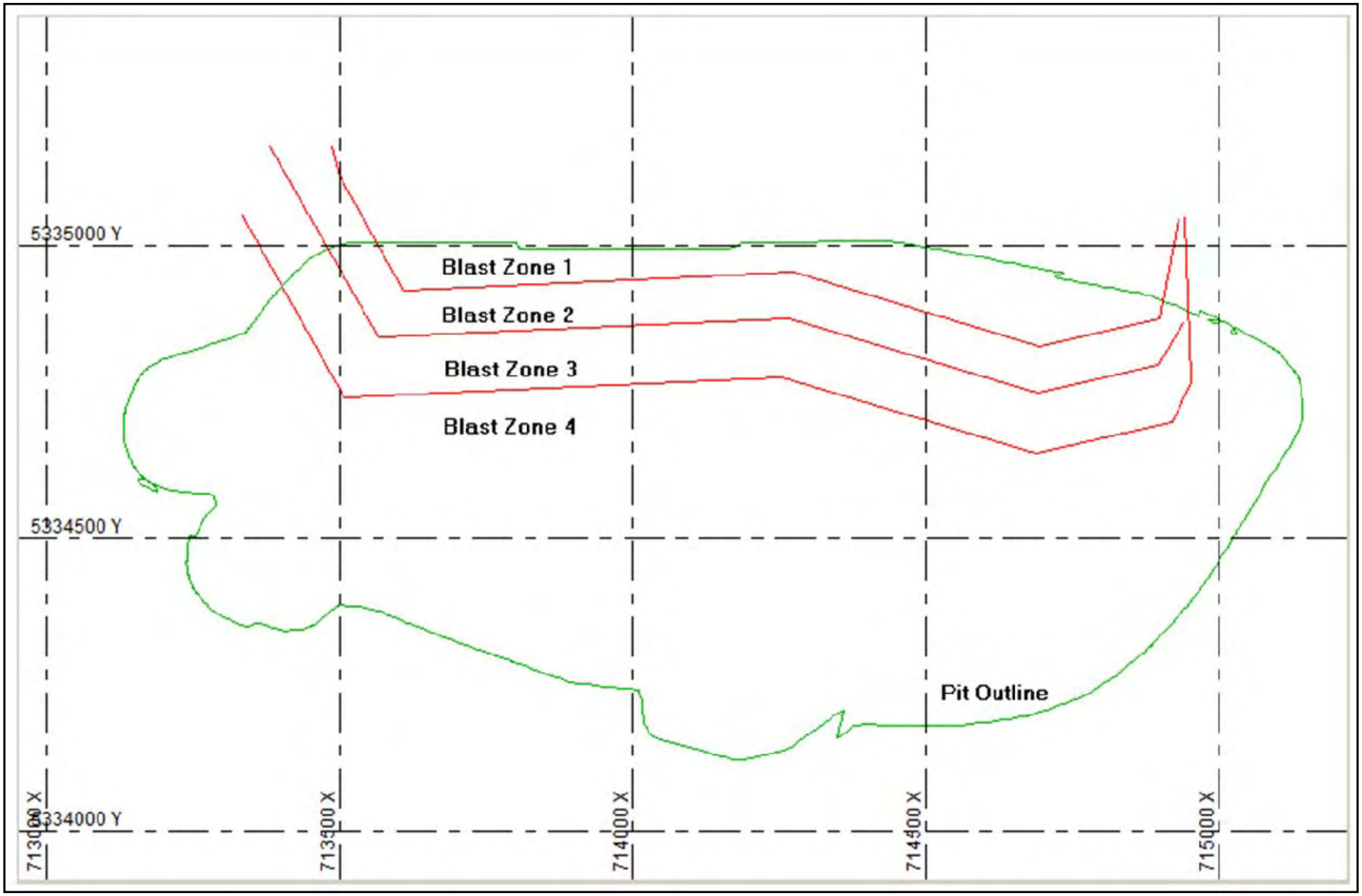
Design ratios	Units	Pattern 1	Pattern 2	Pattern 3	Pattern 4
Explosive type		Emulsion	Emulsion	Emulsion	Emulsion
Explosive density	g/cm^3	1.25	1.25	1.25	1.25
Hole diameter	mm	114	140	140	216
Burden (B)	m	3.20	3.80	3.80	6.00
Spacing (S)	m	4.20	4.80	4.80	7.00
Subdrill (J)	m	0.60	0.60	1.20	1.20
Stemming (T)	m	1.77	2.04	3.38	3.60
Bench height (H)	m	5.00	5.00	10.00	10.00
Blasthole length (L)	m	5.60	5.60	11.20	11.20
Powder factor	kg/t	0.26	0.27	0.30	0.30

The blast pattern yields based on an average rock density of 2.76 t/BCM are presented in Table 5.19. The yield ranges from 33 to 104 t/m drilled.

Table 5.19: Blast Pattern Yields

Pattern yield	Units	Pattern 1	Pattern 2	Pattern 3	Pattern 4
Rock density	t/bcm	2.76	2.76	2.76	2.76
BCM/hole	bcm/hole	67	91	182	420
Yield per hole	t/hole	185	252	503	1,159
Yield per meter drilled	t/m drilled	33	45	45	104

Figure 5.25: Plan View of Blasting Zones



5.6.1.2 Pre-Split Blast Patterns

Pre-split blasting is recommended from the geotechnical assessment in order to maximize stable bench face and inter-ramp angles along final walls. The pre-split consists of a row of closely-spaced holes along the design excavation limit. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves.

The pre-split row is fired in advance of the adjacent trim blast, which is designed to limit damage beyond the pre-split line. The trim blast includes a “buffer” row adjacent to the pre-split row to break back to, but not beyond the pre-split, by ensuring that the buffer row is fired with good horizontal relief.

The following general guidelines have been used in designing the pre-split:

- Blasthole diameter of 140 mm
- Blasthole length of 20 m for double bench pre-split when sufficiently far from the town
- Hole spacing of 1.5 m equivalent to about 11 times the blasthole diameter
- Targeted charge weight of 0.8 kg/m^2 of face area
- Detonation of 5 holes per delay.

The pre-split design parameters are outlined in Table 5.20. The hole diameter is identical to production patterns 2 and 3 which can be drilled with the same drill rig. The pre-split loading consists of a 5 kg toe charge of emulsion with a continuous length decoupled charge of 25 kg above. For simplicity and practical reasons, the decoupled charge corresponds to a complete case of packaged pre-split explosive product. A pre-split explosive with an internally traced high strength detonating cord of 10 g/m for initiation is preferred for fast loading and reduced manipulations.

Table 5.20: Pre-Split Design Parameters

Design Parameters	Units	Pre-Split
Hole diameter	mm	140
Spacing (S)	m	1.5
Stemming (T)	m	2.7
Bench height (H)	m	20.0
Blasthole length (L)	m	20.7
Face area	m^2	31.1
Explosives charge	kg	30.0
Toe charge	kg	5.0
Decoupled charge	kg	25.0
Decoupled charge length	m	17.0
Decoupled charge loading	kg/m	1.47

5.6.1.3 Drilling

Drill rig requirements were established from the yield per meter drilled (from the pattern), the pure penetration rates and the actual drilling time described by an overall drilling factor. The instantaneous penetration rate is a function of the bit size and the rock mass unconfined compressive strength. An average value of 175 MPa was assumed to estimate the instantaneous penetration rates.

The drilling cycle includes the time for drilling the hole, retracting the drill steel and tramming and levelling between holes. Other factors affect drilling efficiency such as reaming the hole, waiting on patterns to be laid out, moving between patterns, bit changes, walk around inspections and moving in and out for blasting. Weighing all these factors, and comparing with other operations, drill productivity is captured by the overall drilling factor to arrive at an overall penetration rate that is achievable over the long term.

Typically, the longer the drill hole the greater the overall drilling factor (or time spent drilling). The overall drilling factor ranges from 48% to 60% depending on the drill pattern. Table 5.21 summarizes the drilling productivities assumed.

Table 5.21: Drill Productivity by Pattern

Drill productivity	Units	Pattern 1	Pattern 2	Pattern 3	Pattern 4
Pure penetration rate	m/h	46.0	44.9	44.9	35.0
Overall drilling factor (%)	%	48	48	57	60
Overall penetration rate	m/h	22.1	21.6	25.6	21.0
Drilling efficiency	t/h	731	969	1,150	2,174
Drilling efficiency	holes/h	3.94	3.85	2.29	1.88

Three drill models will be required to drill the four patterns. Drilling of pattern 1 would likely be given to a contractor since there is a limited amount of drilling and it will not be required for the duration of the mine. Patterns 2 and 3 would be drilled using a diesel powered, self-contained crawler mounted drilling rig designed for DTH blasthole drilling. The angle drilling capability of this drill rig also makes it ideal for drilling the pre-split holes. Pattern 4 would be drilled with a diesel powered, self-propelled crawler mounted blast-hole drill for mining. This rotary type blasthole rig is capable of DTH drilling with the addition of a high pressure air compressor (1450 cfm/350 psi).

Table 5.22 presents the tonnage mined by drill pattern over the mine life. Only 2.5% of the total tonnage is to be mined with 114 mm diameter holes during the first three years of operation. The proportion of large diameter holes (216 mm) increases over time as the pit deepens with 78.1% of the total mined with this diameter drill hole.

Table 5.22: Tonnage Mined by Drill Pattern

Tonnes Mined (kt)	Pattern 1	Pattern 2	Pattern 3	Pattern 4	Total
Pre-Prod	1,655	2,138	2,911	5,607	12,310
Yr 1	6,463	11,026	12,846	20,709	51,045
Yr 2	3,323	8,204	13,206	39,238	63,971
Yr 3	1,322	6,750	8,416	47,528	64,016
Yr 4	51	3,467	11,391	49,041	63,950
Yr 5		343	8,556	55,111	64,011
Yr 6		293	4,694	54,670	59,657
Yr 7		126	4,191	49,482	53,799
Yr 8		0	447	44,270	44,717
Yr 9		0	0	28,883	28,883
Yr 10				3,772	3,772
Total	**12,814**	**32,348**	**66,658**	**398,310**	**510,130**
%	**2.5**	**6.3**	**13.1**	**78.1**	**100**

A re-drill factor of 6% has been assumed in estimating the number of drill holes and total meters to be drilled. This re-drill percentage is deemed adequate given the good ground conditions described by the geotechnical investigation. Table 5.23 presents the number of drill holes by pattern inclusive of the re-drill holes.

Table 5.23: Holes Drilled by Pattern

Drill Holes	Pattern 1	Pattern 2	Pattern 3	Pattern 4	Total
Pre-Prod	9,458	9,002	6,129	5,127	29,715
Yr 1	36,938	46,434	27,048	18,937	129,357
Yr 2	18,990	34,549	27,806	35,881	117,225
Yr 3	7,557	28,426	17,721	43,461	97,165
Yr 4	290	14,601	23,986	44,844	83,721
Yr 5		1,446	18,016	50,394	69,857
Yr 6		1,234	9,883	49,992	61,109
Yr 7		530	8,824	45,248	54,602
Yr 8			941	40,481	41,422
Yr 9				26,411	26,411
Yr 10				3,449	3,449
Total	**73,233**	**136,223**	**140,353**	**364,224**	**714,034**

5.6.1.4 Blasting

A high-energy bulk emulsion explosive specifically developed for surface mines will be used in wet or dry conditions. The bulk emulsion would consist of a 70% emulsion, 30% ANFO blend with an in-hole density of 1.25 g/cm^3. Initiation of the bulk emulsion requires direct contact with a 400 g booster.

Production blasting will use a system of electronic detonators which are more precise and allow for a better control of ground vibrations. A conservative approach of using two detonators per hole has been planned and is not absolutely necessary from a technical point of view. The use of only one electronic detonator per hole would reduce costs by an estimated $0.035/t mined. For planning and budgeting purposes it has been planned that blasting of 5 m benches would use one electronic and one NONEL detonator for cost reduction reasons as opposed to two electronic detonators for 10 m benches.

The advantages of electronic initiation are numerous:

- Precise timing of each detonator
- No overlapping of seismic waves and amplification problems

- Assurance of the proper functioning of each detonator prior to firing
- Firing is only possible with the detonating device
- Elimination of cut-offs
- Improved fragmentation.

Bulk and packaged explosives costs are subject to rise and fall formula based on the quarterly price of natural gas and diesel.

Blasting activities will likely be outsourced to an explosives provider who will be responsible for supplying and delivering explosives in the hole through a shot service contract. The mine engineering department will be responsible for designing blast patterns and implementing them in the field.

The shot service contract includes:

- Hole measurement
- Transport of blasting accessories
- Placement of wedge blocks in breakthrough holes where voids are encountered
- Explosives loading
- Stemming placement
- Tie-in and firing
- Blast reporting
- Loading and firing of perimeter holes.

Blast hole loading and firing activities would be performed on day shift only. Two blast crews working 12 hour shifts and rotating on a 4 days on, 4 days off schedule is proposed. Each crew is composed of a master blaster, 2 helpers and 2 explosives truck operators, for a total of 10 employees.

A fleet of 3 explosive pump trucks having a 16 t to 18 t capacity is required for the delivery of bulk emulsion in order to have 2 units in operation at all times. The shot service cost includes the capital recovery of the emulsion pump trucks.

Table 5.24 presents the typical hole loading for each drill pattern. Table 5.25 presents the explosives cost per hole.

Table 5.24: Typical Hole Loading

Explosives products	Units	Pattern 1	Pattern 2	Pattern 3	Pattern 4
Bulk emulsion	kg	49	68	150	348
NONEL (6 m)	ea	1	1		
Electronic detonator (6 m)	ea	1	1	1	1
Electronic detonator (15 m)	ea			1	1
Booster (16 oz)	ea	2	2	2	2
Harness wire	m	4.2	4.8	4.8	6.4

The cost per tonne of explosives product decreases from 0.37 $/t mined for drill pattern 1 (114 mm) to 0.22 $/t mined for drill pattern 4 (216 mm). Drill patterns 2 and 3 have an intermediate cost of 0.32 and 0.29 $/t mined respectively.

Table 5.25: Explosives Cost per Hole

Explosives cost/hole	Units	Pattern 1	Pattern 2	Pattern 3	Pattern 4
Bulk emulsion	$	[redacted]	[redacted]	[redacted]	[redacted]
Fuel	$	[redacted]	[redacted]	[redacted]	[redacted]
NONEL (6 m)	$	[redacted]	[redacted]	-	-
Electronic detonator (6 m)	$	[redacted]	[redacted]	[redacted]	[redacted]
Electronic detonator (15 m)	$	-	-	[redacted]	[redacted]
Booster (16 oz)	$	[redacted]	[redacted]	[redacted]	[redacted]
Harness wire	$	[redacted]	[redacted]	[redacted]	[redacted]
Other	$	[redacted]	[redacted]	[redacted]	[redacted]
Total	**$**	**64.89**	**74.29**	**136.87**	**233.47**
Cost	**$/t**	**0.37**	**0.31**	**0.29**	**0.21**

5.6.1.5 Explosives Plant and Depot

The explosives plant is sized to produce the required quantity of explosives for the project estimated at about 19,500 tonnes of emulsion per year. The explosives depot would include a magazine for the storage of 40,000 kg of 1.1D class explosives (pre-split packaged explosives and boosters) and another magazine for the storage of 20,000 detonators classified 1.1B.

The explosives plant would operate 7 days a week on 12 hour shifts. The explosive plant requires 1 operator working on the same schedule as the blasting crew and a supervisor plus a mechanic working on a 5 days on, 2 days off schedule, for a total of four employees.

5.6.2 Ore Control

The ore control program will consist of establishing ore/waste boundaries in the field to guide loading unit operators. This information can be relayed to the digital screens placed in the shovels which indicate bucket location with respect to the digging packets.

Information for the establishment of this ore control program would be provided from blasthole cuttings. Pre-engineered patterns will be relayed to the drills. Ore control methods are designed for simplicity, low cost, and adaptability. This method is recognised as having the lowest direct cost, allowing single-pass, dual-use holes (sampling and blasting) in a mine that will need to advance through up to six twenty meter benches per year to maintain production schedules. Although this drill method is unable to identify vertical grade boundaries, the ability to maintain a one-week cycle-time between drill / assay / blast / design / load is considered critical.

Grade control costing has assumed that samples will be collected and assayed on 10 m downhole intervals in the case of patterns 3 & 4 and every second hole on 5 m downhole intervals in the case of patterns 1 & 2. Single samples will be taken per blast hole via a mechanical sample collector / splitter or from drill cuttings collected in a sample pan placed beneath the hood.

Drill helpers will supervise and control the sampling operation. A minimum sample of 5 kg will be prepared at the mill laboratory for assaying. Gold fire assays performed at the mill laboratory will determine gold content. Sampler duties will be to record the sample and document the sample locations (sample ID, drill hole, depth, etc) as well as ensuring that the retained sample is put into a labelled polyethylene sample bag and placed beside the drill pads for collection. They will also be required to clean the hood between holes as required.

5.6.3 Loading and Hauling

The primary loading tools will consist of hydraulic excavators, with a wheel loader added as a secondary loading tool. The hydraulic excavator model selected is the O&K RH340-B with an operating weight of 567 t and is fitted with a 28 m^3 heavy-duty rock bucket. This heavy-duty rock bucket is specified for a maximum loose density of 2.2 t/m^3 according to O&K datasheets. The normal rock bucket has a heaped capacity of 34 m^3, specified for a maximum loose density of 1.8 t/m^3. In light of this, it is anticipated that high fill factors will be achieved with the smaller heavy-duty rock bucket. The unit will be electrically driven as opposed to diesel driven, given the availability of low-cost electric power and the large size of the open pit.

A Caterpillar 994F HL front-end wheel loader (FEL) will complement the primary loading fleet. Should ore feed from the pit be interrupted for extended periods, the plant will be fed solely from stockpiled ore on the ROM pad by a FEL. To reduce concerns relating to operating rubber tire front-end loaders in a hard rock environment, given the current high cost environment for tires, chains have been planned on front tires to extend tire life. Pewag chains have been costed and the tire life adjusted according to feedback and performance history of users of these chains. The chains and front end tires are estimated to last for 7,500 hours whereas the rear tire is estimated at 4,500 hours for costing purposes.

During the second year of operations an O&K RH200 is added to the loading fleet to increase loading capacity. This unit would be purchased used and was considered to be diesel driven for this study.

A fleet of Caterpillar 793F rigid trucks with a 227 t payload has been selected to provide a good pass-match with the O&K RH340-B shovels. The RH340-B will require 4 passes on average which is dependent on the bucket fill factor and swell factor.

The FEL is configured in a high lift arrangement in order to clear the sideboard of the 227 t class truck. The FEL is equipped with a 15 m^3 bucket requiring 8 passes to load the trucks to capacity. The number of passes required is more than is typically considered ideal, but other factors favoured this model such as having a sole provider of major mining equipment. The loading productivity assumptions for the RH340-B, FEL and RH200 are presented in Table 5.26.

The swell factor used to determine loose density is an important factor in estimating loading unit productivities. A typical swell factor for rock of 40% has been assumed for determining loading cycles. Loose density information verified through testing would be required to validate this important assumption. However, 40% is thought to be a conservative estimate.

Table 5.26: Loading Unit Productivity Estimates

Loading unit	Units	O&K RH340-B	CAT 994F HL	O&K RH200
Loading unit weight	t	567	195	525
Haulage unit		**Cat 793F**	**Cat 793F**	**Cat 793F**
Rated payload	t	227	227	227
Heaped volume	m^3	149	149	149
Production Parameters				
Bucket capacity	m^3	28	15	21
Bucket fill factor	%	100	93	95
In-situ dry density	t/bcm	2.76	2.76	2.76
Moisture	%	3	3	3
Swell	%	40	40	40
Wet loose density	t/m^3	2.03	2.03	2.03
Actual load per bucket	t	56.9	28.94	40.51
Passes (decimal)	#	3.99	7.84	5.60
Passes (whole)	#	4.0	8.0	5.5
Actual truck wet payload	t	227.4	231.5	222.8
Actual truck dry payload	t	220.8	224.7	216.3
Actual heaped volume	m^3	112.0	114.0	109.7
Payload capacity	%	100	100	98
Heaped capacity	%	75	75	74
Cycle time				
Hauler exchange	min	0.70	0.70	0.70
First bucket dump	min	0.10	0.10	0.10
Average cycle time	min	0.65	0.70	0.63
Load time	min	2.75	5.70	3.64
Cycle efficiency with 20% wait time	%	80	80	80
Number of trucks loaded per hr	#	17.5	8.4	13.2
Production / Productivity				
Productivity dry tonnes / op. hr	t/h	3,854	1,853	2,856

Table 5.27: Tonnage Loaded by Period by Loading Unit

Tonnes Mined (kt)	O&K RH340 (2 Units)	O&K RH200 (1 Unit)	CAT 994 (1 Unit)	Total Tonnes Moved
Pre-Prod	9,473	-	2,837	12,310
Yr 1	40,999	-	10,046	51,045
Yr 2	39,944	16,329	9,088	65,361
Yr 3	42,037	15,753	10,178	67,968
Yr 4	39,254	13,772	10,924	63,950
Yr 5	40,390	14,650	8,971	64,011
Yr 6	41,269	14,564	3,824	59,657
Yr 7	39,052	13,763	2,196	55,012
Yr 8	39,298	5,418	-	44,717
Yr 9	28,883	-	2,709	31,592
Yr 10	1,890	-	6,866	8,755
Total	**362,488**	**94,250**	**67,638**	**524,376**
%	**69.1**	**18.0**	**12.9**	**100**

Fleet productivity and unit requirements were estimated by using the Fleet Production and Costing (FPC) software developed by Caterpillar. Haulage profiles and fleet productivities were estimated by elevation in the pit (Table 5.28 and Table 5.29). The ore haul distance on the upper bench is 2.1 km and increases with the deepening of the pit to a distance of 4.9 km. The waste haul on the upper bench is 2.0 km and increases to a distance of 5.74 km at the bottom of the pit. The cycle times gradually increase with the deepening of the pit with the ore haul ranging from 10 minutes to 41 minutes (Table 5.30) and between 11 and 49 minutes for the waste haul (Table 5.31).

The waste haul is influenced by the waste dump development strategy. The development of the waste dump assumes the establishment of an entire lift before increasing the dump height. The advantage of this strategy is that it enables progressive rehabilitation of the waste dump over time. Under this strategy, the ramp segment of the waste dump haul progressively increases over time.

Table 5.28: Ore Haulage Profiles

Bench	In-pit Segment (m)	Uphill Ramp Segment (m)	To Crusher (m)	Total Dist. (m)
320	950	0	1,160	2,110
300	950	200	1,160	2,310
280	900	400	1,160	2,460
260	850	600	1,160	2,610
240	800	800	1,160	2,760
220	785	1,000	1,160	2,945
200	750	1,200	1,160	3,110
180	720	1,400	1,160	3,280
160	700	1,600	1,160	3,460
140	680	1,800	1,160	3,640
120	650	2,000	1,160	3,810
100	630	2,200	1,160	3,990
80	600	2,400	1,160	4,160
60	465	2,600	1,160	4,225
40	455	2,800	1,160	4,415
20	375	3,000	1,160	4,535
0	245	3,200	1,160	4,605
-20	195	3,400	1,160	4,755
-40	150	3,600	1,160	4,910

Table 5.29: Waste Haulage Profiles

Bench	In-Pit Segment (m)	Pit Ramp Segment (m)	Pit to Dump (m)	Dump Ramp (m)	On Dump Segment (m)	Total Dist. (m)
320	950	0	290	0	800	2,040
300	950	200	290	100	800	2,340
280	900	400	290	200	800	2,590
260	850	600	290	300	800	2,840
240	800	800	290	400	800	3,090
220	785	1,000	290	500	800	3,375
200	750	1,200	290	600	800	3,640
180	720	1,400	290	600	800	3,810
160	700	1,600	290	700	800	4,090
140	680	1,800	290	700	800	4,270
120	650	2,000	290	700	800	4,440
100	630	2,200	290	700	800	4,620
80	600	2,400	290	800	800	4,890
60	465	2,600	290	800	800	4,955
40	455	2,800	290	900	800	5,245
20	375	3,000	290	900	800	5,365
0	245	3,200	290	900	800	5,435
-20	195	3,400	290	900	800	5,585
-40	150	3,600	290	900	800	5,740

Table 5.30: Ore Haulage Cycle Times

Bench	Load (minutes)	Haul (minutes)	Return (minutes)	D&M (minutes)	Wait (minutes)	Total (minutes	Avg. Speed (Km/h)
320	3.26	1.99	1.64	1.50	1.54	9.93	25.49
300	3.26	4.82	3.99	1.50	2.25	15.82	17.52
280	3.26	5.88	4.23	1.50	1.99	16.86	17.51
260	3.26	6.94	4.47	1.50	1.85	18.02	17.38
240	3.26	8.00	4.72	1.50	2.70	20.18	16.41
220	3.26	9.13	5.03	1.50	2.48	21.40	16.51
200	3.26	10.22	5.30	1.50	3.36	23.64	15.79
180	3.26	11.32	5.59	1.50	3.10	24.77	15.89
160	3.26	12.44	5.89	1.50	2.96	26.05	15.94
140	3.26	13.56	6.19	1.50	3.78	28.29	15.44
120	3.26	14.53	6.47	1.50	3.51	29.27	15.62
100	3.26	15.64	7.12	1.50	4.44	31.96	14.98
80	3.26	16.72	7.12	1.50	4.19	32.79	15.22
60	3.26	17.60	7.13	1.50	4.02	33.51	15.13
40	3.26	18.73	7.46	1.50	5.00	35.95	14.74
20	3.26	19.72	7.64	1.50	4.71	36.83	14.77
0	3.26	20.61	7.72	1.50	4.61	37.70	14.66
-20	3.26	21.66	7.96	1.50	4.40	38.78	14.71
-40	3.26	22.71	8.22	1.50	5.23	40.92	14.40

Table 5.31: Waste Haulage Cycle Times

Bench	Load (minutes)	Haul (minutes)	Return (minutes)	D&M (minutes)	Wait (minutes)	Total (minutes)	Avg. Speed (Km/h)
320	3.33	2.00	1.65	1.50	2.39	10.87	22.52
300	3.33	5.55	4.54	1.50	2.03	16.95	16.57
280	3.33	7.17	4.94	1.50	2.86	19.80	15.70
260	3.33	8.79	5.33	1.50	2.47	21.42	15.91
240	3.33	10.42	5.72	1.50	3.30	24.27	15.28
220	3.33	12.11	6.18	1.50	2.95	26.07	15.54
200	3.33	13.76	6.61	1.50	3.68	28.88	15.12
180	3.33	14.85	6.89	1.50	3.46	30.03	15.22
160	3.33	16.53	7.34	1.50	3.15	31.85	15.41
140	3.33	17.64	7.65	1.50	3.21	33.33	15.37
120	3.33	18.73	7.93	1.50	3.87	35.36	15.07
100	3.33	20.41	8.38	1.50	4.48	38.10	14.55
80	3.33	21.50	8.66	1.50	5.38	40.37	14.54
60	3.33	22.38	8.74	1.50	5.26	41.21	14.43
40	3.33	24.08	9.21	1.50	6.00	44.12	14.27
20	3.33	25.07	9.39	1.50	5.75	45.04	14.29
0	3.33	25.96	9.48	1.50	5.59	45.86	14.22
-20	3.33	27.00	9.72	1.50	6.54	48.09	13.94
-40	3.33	28.06	9.97	1.50	6.27	49.13	14.02

Table 5.32: Haulage Truck Productivity and Haulage Hours Required by Year

Haulage Productivity	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Ore Haulage												
Ore tonnes hauled	2,829	20,091	18,685	16,124	20,253	22,539	21,418	19,662	21,673	17,366	2,689	183,329
Ore hauling hours	2,726	24,934	26,230	22,910	33,273	39,988	40,322	41,222	53,176	46,681	7,964	339,427
Ore haul kilometers	28,351	223,120	223,461	195,070	270,771	322,796	319,527	322,752	402,262	346,972	58,225	2,713,305
Ore haul cycles	12,859	91,325	84,933	73,289	92,059	102,448	97,356	89,373	98,512	78,936	12,222	833,312
Avg. Cycle time (min)	12.72	16.38	18.53	18.76	21.69	23.42	24.85	27.67	32.39	35.48	39.09	24.44
Avg. Haul dist (km)	2.20	2.44	2.63	2.66	2.94	3.15	3.28	3.61	4.08	4.40	4.76	3.26
Waste Haulage												
Wst tonnes hauled	9,481	30,953	45,286	47,893	43,697	41,472	38,239	34,137	23,044	11,517	1,083	326,801
Wst hauling hours	9,706	41,707	63,929	75,744	72,236	79,637	83,702	85,780	67,816	37,865	3,960	622,081
Wst haul kilometers	93,534	348,743	523,208	612,229	571,576	616,357	638,389	655,768	494,656	271,331	27,708	4,853,500
Wst haul cycles	43,094	140,697	205,845	217,695	198,623	188,509	173,813	155,167	104,745	52,349	4,922	1,485,459
Avg. Cycle time (min)	13.51	17.79	18.63	20.88	21.82	25.35	28.89	33.17	38.85	43.40	48.27	25.13
Avg. Haul dist (km)	2.17	2.48	2.54	2.81	2.88	3.27	3.67	4.23	4.72	5.18	5.63	3.27
Re-handling Haulage												
Rehandled tonnes	-	-	1,390	3,951	-	-	-	1,213	-	2,709	4,983	14,247
Rehandling hours	-	-	1,369	3,892	-	-	-	1,195	-	2,668	4,908	14,031
Rehandle km	-	-	13,329	37,898	-	-	-	11,633	-	25,983	47,796	136,639
Rehandle cycles	-	-	6,317	17,961	-	-	-	5,513	-	12,314	22,652	64,758
Avg. Cycle time (min)	-	-	13.00	13.00	-	-	-	13.00	-	13.00	13.00	13.00
Avg. Haul dist (km)	-	-	2.11	2.11	-	-	-	2.11	-	2.11	2.11	2.11

Table 5.33 presents the haulage truck productivity achieved by period by haulage destination. The number of haulage hours increases over time as the haulage cycle increases with a total of 22 trucks required by the sixth year of operation.

Table 5.33: Haulage Truck Productivity and Truck Requirements

Tonnes Mined	Waste Haul (tph)	Ore Haul (tph)	Stockpile Haul (tph)	Haulage Hours	Trucks Required
Yr -1	977	1,038	0	12,433	12
Yr 1	742	806	0	66,641	12
Yr 2	708	712	1,015	91,527	16
Yr 3	632	704	1,015	102,546	18
Yr 4	605	609	0	105,509	19
Yr 5	521	564	0	119,626	21
Yr 6	457	531	0	124,025	22
Yr 7	398	477	1,015	128,196	22
Yr 8	340	408	0	120,992	21
Yr 9	304	372	1,015	87,213	15
Yr 10	273	338	1,015	16,831	3
Avg / Tot.	**525**	**540**	**1,015**	**975,539**	

5.6.4 Equipment Availability and Usage

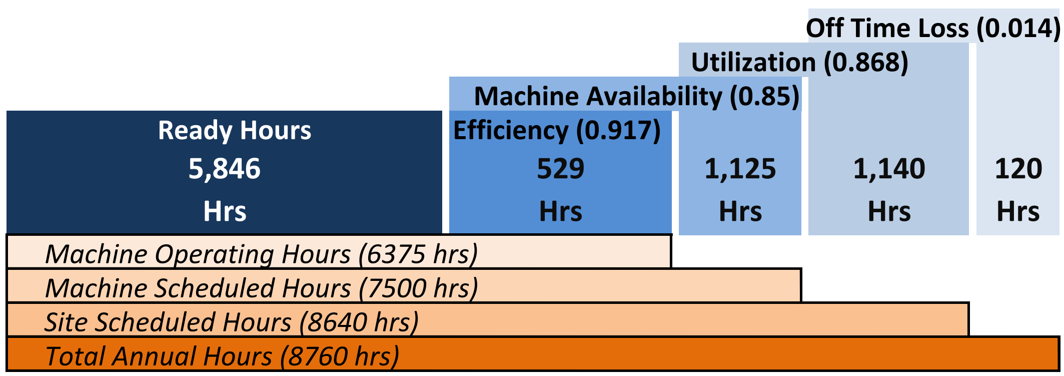
Major equipment availability assumptions vary between 88% and 80%. A lower availability of 80% has been assumed for production drill rigs and ancillary equipment. An availability of 88% for the RH340-B units has been assumed given that they are electric drive and require less maintenance than their diesel drive equivalents. A constant 85% availability has been assumed for the haulage trucks, which is an industry acceptable value for the life of the mining equipment selected.

Usage for the major mining fleet, including drills, assumes a 365-day per year operation with two 12 hour shifts less the following time lost per day:

- Mobilise and start-up checks: 15 min/shift
- Meal break including start & stop: 60 min/shift
- End of shift blast & move: 20 min/shift
- Total time lost per shift: 95 min/shift (1.58 h/shift)
- Total time lost per day: 190 min/day (3.17 h/day)

It is assumed that 5 days will be lost to weather and other stoppages during the year for an off time loss of 1.4% (120/8760). Using the above assumptions for time lost per day, an average utilization of 87% is calculated (7500/8640). With an anticipated mechanical availability of 85%, 6375 operating hours per year are planned. Finally, a job efficiency factor is applied in order to estimate effective work hours or ready hours. A typical hourly usage breakdown for major equipment is presented in Table 5.34.

Table 5.34: Equipment Usage Breakdown



5.6.5 Support Equipment

Caterpillar equipment was specified where applicable for support equipment. Support equipment requirements are based on a typical open pit mine operation and maintenance requirements to safely support the loading, hauling and drilling fleets.

Support equipment includes several units as follows:

- 2 x motor graders (CAT 16M) for roadway maintenance
- 4 x tracked dozers (CAT D10T) with single shank ripper for bench face clean up, dumps and stockpile management
- 1 x wheel dozer (CAT 834H) for rock spillage cleanup, working around the shovels for maintaining floor grades / cleanup
- 2 x water truck (76 kl capacity) for bench face and roadway dust suppression
- 1 x tool carrier, 1 x boomtruck & 1 x lowboy (150 t)

- 3 x maintenance service trucks for field repairs
- 2 x fuel & lube trucks for servicing in the field
- 3 x utility excavators (CAT 345D) for various clean-up jobs, pit wall scaling, secondary breakage, blast mat manoeuvring and ditching activities
- 5 x CAT 740 articulated trucks, 1 x CAT 980H wheel loader, 2 x CAT D6T tracked dozers, 1 x CAT CS56 compactor and 1 x CAT 14M motor grader used for site maintenance and topsoil management.

5.6.6 Fleet Management

A high-precision global positioning system (GPS) for machine guidance is planned for the operation. A high precision system is considered to mitigate the associated risk of working around underground workings. This will enable shovel operators to navigate safely in potentially hazardous areas. In addition to protecting people and equipment, the GPS system will improve the productivity and bench grade control. The results and usefulness of such a system have proven to be worthwhile at other mines where past underground mines have been developed.

Proposals for a fleet management system have been obtained from Wenco and from Minestar (Caterpillar). A final decision on the system provider has not been made. The capital cost for such a system includes on-board hardware, telemetry hardware, servers, workstations, software, training and installation.

5.6.7 Mine Dewatering

The open pit dewatering requirements were investigated by Golder. The strategy will consist of using the underground openings and the connectivity of the past underground mines (Canadian Malartic, Sladen, Barnat and East Malartic) to keep the water level 50 m below the working benches. A numerical model was created by Golder to simulate the underground water flows.

Three pumping stations are proposed at the following locations with the estimated minimum and maximum pumping rates:

- Pumping station at the Canadian Malartic no. 1 shaft (348 meters deep)
  - 3,400 to 5,300 m^3/day (620 to 970 usgpm).
- Pumping station at the Sladen No2 shaft (545 meters deep)

  - 1,000 to 2,300 m^3/day (180 to 420 usgpm).
- Pumping station in the East Malartic No. 3 shaft (545 meters deep)
  - 1,800 to 7,300 m^3/day (330 to 1300 usgpm).

The total pumping rates vary between 6,200 and 14,900 m^3/day. The distribution of pumping rates at the pumping stations is based on the connectivity of the underground workings and may vary over time with the deepening of the pit. Water pumped from the underground workings will be used in the process plant.

5.6.8 Mine Maintenance

The Canadian Malartic project does not intend to enter into a maintenance and repair contract (MARC) for its mobile equipment fleet. Consequently, the maintenance department has been structured to fully manage this function, performing maintenance planning and training of employees. However, the Canadian Malartic project will rely on dealer and manufacturer support for major components and would seek to enter into a component exchange program with Caterpillar for major components such as engines, transmissions and final drives.

The maintenance department will require specialized tools for the specific equipment models on site such as diagnostic tools, pin pullers, hydraulic torque wrenches and general shop tools such as presses, nitrogen charging kits, air tools, lift stands and kidney looping machines. The cost of shop tools and specific equipment maintenance tools required has been estimated with the equipment dealer.

A maintenance control system will be used to manage maintenance and repair operations. This system will keep up to date status, service history and maintenance needs of each machine. The specific software package is yet to be selected as this software package will require an interface with the parts management and inventory system.

5.6.9 Mine Management and Technical Services

The operations team is responsible for achieving production targets in a safe manner. The engineering and geology team will provide support to the operations team by providing short term and long term planning, grade control, surveying, mining reserves estimation and all other technical functions. Operating costs for this group includes salaries, office supplies, and survey and grade control supplies. The manning schedule is considered appropriately sized for a single pit operation.

The mine is headed by a mine manager who is responsible for the overall management of the mine. Superintendent positions in engineering, geology, operations and maintenance report directly to the mine manager.

The operations department is composed of two supervisors on each shift for a total of eight. A mine dispatcher is also planned on each shift. To increase operator level performance and organize structured training programs two mine trainers are planned on day shift only.

5.7 Mine Manpower

5.7.1 Work Schedule

Mine crews will work 12 hour shifts on a 5 day / 4 off / 4 night / 5 off schedule. Four crews are required to operate 24 hours per day all year round. This is a common schedule used in the region and results in 2,190 scheduled hours of work per year excluding vacation time.

Mine technical staff will work on a standard office schedule consisting of a 5 day work week.

Table 5.35 presents the mine staffing levels over the mine life with a reduction occurring when the tonnage decreases during the 8th year of operation. The last year, 2020, is a fractional year and explains the reduction in number of employees. The total mine department workforce is 262 the first year of operation and reaches a peak of 322 persons by the fifth year for an average of 297 over the mine life.

The mine technical services group consisting of engineering, surveying and geology is relatively constant over the mine life with 15 people required.

Table 5.35: Manpower Requirements

Area/ Position	Roster	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
Operations												
Mine manager	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Mine superintendant	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Senior shift foreman	Rotations	3	8	8	8	8	8	8	8	8	8	1
Dispatch supervisor	Rotations	0	4	4	4	4	4	4	4	4	4	1
Trainer	Dayshift	2	2	2	2	2	2	2	2	2	2	0
Mine secretary	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Shovel operator	Rotations	4	8	12	12	12	12	12	12	12	8	4
Wheel loader operator	Rotations	2	4	4	4	4	4	4	4	0	4	4
Haul truck operators	Rotations	24	48	64	72	76	84	88	88	84	60	12
Driller	Rotations	14	36	36	36	36	32	28	24	24	12	4
Drill helpers	Rotations	6	12	12	12	12	12	12	12	12	12	2
Track dozer operator	Rotations	8	16	16	16	16	16	16	16	16	12	2
Wheel dozer operator	Rotations	2	4	4	4	4	4	4	4	4	4	1
Grader operator	Rotations	4	8	8	8	8	8	8	8	8	8	1
Water truck operator	Rotations	2	4	4	4	4	4	4	4	4	4	1
Labourer	Dayshift	1	2	2	2	2	2	2	2	2	2	0
Utility equipment operator	Dayshift	6	8	8	8	8	8	8	8	8	8	1
Sub-Total Operations		**81**	**167**	**187**	**195**	**199**	**203**	**203**	**199**	**191**	**151**	**34**
Maintenance												
Maint. superintendant	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Maint. foreman	Rotations	3	8	8	8	8	8	8	8	8	8	1
Maint. trainer	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Maint. planner	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Maint. shift planner	Dayshift	2	4	4	4	4	4	4	4	4	4	1
Mechanical engineer	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Mechanic	Rotations	10	40	56	56	56	56	56	56	52	32	8
Electrician	Rotations	2	8	8	8	8	12	12	12	12	12	2
Welder	Rotations	2	8	8	8	8	12	12	8	8	8	1
Helper	Rotations	2	8	8	8	8	8	8	8	8	8	1
Sub-Total Maintenance		**25**	**80**	**96**	**96**	**96**	**104**	**104**	**100**	**96**	**76**	**14**
Technical Services												
Chief engineer	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Senior mine engineer	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Drill & blast engineer	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Geotechnical engineer	Dayshift	0	1	1	1	1	1	1	1	1	1	0
Junior mine engineer	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Chief surveyor	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Surveyor	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Surveyor rodmen	Dayshift	2	2	2	2	2	2	2	2	2	2	0
Chief geologist	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Senior geologist	Dayshift	1	1	1	1	1	1	1	1	1	1	0
Geologist	Dayshift	1	2	2	2	2	2	2	2	2	2	0
Geology technician	Dayshift	2	2	2	2	2	2	2	2	2	2	0
Sub-Total Technical Services		**13**	**15**	**15**	**15**	**15**	**15**	**15**	**15**	**15**	**15**	**2**
Grand-Total Mine Department		**119**	**262**	**298**	**306**	**310**	**322**	**322**	**314**	**302**	**242**	**50**

5.8 Mine Equipment

The main factors which influenced the selection of the major mine equipment included the annual production requirements, optimization of the fleet size, as well as support.

Mine equipment requirements are summarized in Table 5.36. All equipment is purchased new in the equipment purchase schedule except for the O&K RH200 shovel to be purchased in 2011 as presented in Table 5.36. The prices for the major equipment are based on negotiated prices obtained with Hewitt Equipment (CAT dealer) for the loading, hauling and support equipment fleet. Purchase Orders for the all major equipment except drills have been given in order to obtain equipment in the required time frame to execute the proposed mine schedule.

The purchase of mining equipment has been financed through a lease buyback agreement entered into with CAT Finance for support equipment and major mining equipment including O&K loading units.

Construction type equipment including two CAT 345D hydraulic excavators, CAT 980H wheel loader, five CAT 740 articulated dump trucks, two CAT D6T dozers and one CAT 14M grader are purchased, commissioned and are performing site preparation work at the present time.

Given the duration of mining operations, no replacement units are planned over the mine life for loading units and haulers. Replacement units are planned for support equipment such as track-type dozers wheel dozers, pickup trucks, small excavators, service trucks and fuel/lube trucks as presented in Table 5.37: Equipment Purchase Schedule. The capital cost schedule for this equipment is presented in Table 5.38.

Unit purchase prices are not published in Table 5.38 in order to keep the confidentiality agreement with the equipment suppliers.

Table 5.36: Equipment Requirements

Equipment	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
Major Equipment													
Rotary Drill (216 mm)	-	-	2	2	4	5	5	5	5	5	4	3	1
Rotary Drill (140 mm)	-	-	5	5	5	4	3	2	2	2	1	1	-
Shovel (RH340)	-	-	2	2	2	2	2	2	2	2	2	2	1
Wheel Loader (994)	-	-	1	1	1	1	1	1	1	1	-	1	1
Shovel (RH200)	-	-	-	1	1	1	1	1	1	1	-	-	-
Haul Truck (240t class)	-	-	12	12	16	18	19	21	22	22	21	15	3
Grader (16' blade)	-	-	2	2	2	2	2	2	2	2	2	2	0
Water truck (76kl)	-	-	1	1	2	2	2	2	2	2	2	2	1
Track Dozer - 580 hp	-	-	4	4	4	4	4	4	4	4	4	3	1
Wheel dozer - 525 hp	-	-	1	1	1	1	1	1	1	1	1	1	0
Support Equipment													
Excavator (2.1 m^3) (Rock Breaker)	-	-	1	1	1	1	1	1	1	1	1	1	1
Excavator (2.1 m^3)	2	2	2	2	2	2	2	2	2	2	2	2	2
Wheel Loader (5.5 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1
Articulated Trucks (40 t)	5	5	5	5	5	5	5	5	5	5	5	5	5
Track Dozer (200 hp)	2	2	2	2	2	2	2	2	2	2	2	2	2
Grader (14' blade)	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor	1	1	1	1	1	1	1	1	1	1	1	1	1
Small Loader (Hole Stemming)	-	1	1	1	1	1	1	1	1	1	1	1	1
Boom Truck (22 t)	-	1	1	1	1	1	1	1	1	1	1	1	1
Tool carrier	1	1	1	1	1	1	1	1	1	1	1	1	1
Service truck	-	1	2	3	3	3	3	3	3	3	3	3	3
Fuel/Lube truck	-	1	2	2	2	2	2	2	2	2	2	2	2
Lowboy & Tractor Head (150 t)	-	1	1	1	1	1	1	1	1	1	1	1	1
Pick-up truck	-	12	20	20	20	20	20	20	20	20	20	20	10
Pit Busses	-	2	3	4	4	4	4	4	4	4	4	3	3
Lighting towers	-	4	4	4	4	4	4	4	4	4	4	4	4
Mob. welding mach.	-	1	1	2	2	2	2	2	2	2	2	2	2
Mobile compressors	-	1	1	2	2	2	2	2	2	2	2	2	2
Dewatering	-	1	1	2	2	4	4	6	6	6	6	6	6
Dispatch System	-	0	1	1	1	1	1	1	1	1	1	1	1
Emulsion Plant Infrastructures	-	1	1	1	1	1	1	1	1	1	1	1	1
Blasting Mats	-	-	200	300	300	300	300	300	300	300	300	300	300
Tools for Maintenance Shop	-	1	1	1	1	1	1	1	1	1	1	1	1
Spare Bucket for Shovel	-	1	1	1	1	1	1	1	1	1	1	1	1
Technical Services Equipment	-	1	1	1	1	1	1	1	1	1	1	1	1

Table 5.37: Equipment Purchase Schedule

Equipment	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
Major Equipment													
Rotary Drill (216 mm)	-	-	2	1	1	1	-	-	-	-	-	-	-
Rotary Drill (140 mm)	-	-	5	-	-	-	-	-	-	-	-	-	-
Shovel (RH340)	-	2	-	-	-	-	-	-	-	-	-	-	-
Wheel Loader (994)	-	1	-	-	-	-	-	-	-	-	-	-	-
Shovel (RH200)	-	-	-	1	-	-	-	-	-	-	-	-	-
Haul Truck (240t class)	-	6	6	2	2	2	1	2	1	-	-	-	-
Grader (16' blade)	-	1	1	-	-	-	-	-	-	-	-	-	-
Water truck (76kl)	-	-	1	1	-	-	-	-	-	-	-	-	-
Track Dozer - 580 hp	-	2	2	-	-	-	-	1	2	-	-	-	-
Wheel dozer - 525 hp	-	1	-	-	-	-	-	1	-	-	-	-	-
Support Equipment													
Excavator (2.1 m^3) (Rock Breaker)	-	-	1	-	-	-	-	1	-	-	-	-	-
Excavator (2.1 m^3)	2	-	-	-	-	-	-	-	-	-	-	-	-
Wheel Loader (5.5 m^3)	1	-	-	-	-	-	-	-	-	-	-	-	-
Articulated Trucks (40 t)	5	-	-	-	-	-	-	-	-	-	-	-	-
Track Dozer (200 hp)	2	-	-	-	-	-	-	-	-	-	-	-	-
Grader (14' blade)	1	-	-	-	-	-	-	-	-	-	-	-	-
Compactor	1	-	-	-	-	-	-	-	-	-	-	-	-
Small Loader (Hole Stemming)	-	1	-	-	-	-	-	-	-	-	-	-	-
Boom Truck (22 t)	-	1	-	-	-	-	1	-	-	-	-	-	-
Tool carrier	1	-	-	-	-	-	-	-	-	-	-	-	-
Service truck	-	1	1	1	-	-	1	2	-	-	-	-	-
Fuel/Lube truck	-	1	1	-	-	-	-	1	-	-	-	-	-
Lowboy & Tractor Head (150 t)	-	1	-	-	-	-	-	-	-	-	-	-	-
Pick-up truck	-	12	8	4	6	6	6	6	6	6	6	6	-
Pit Busses	-	2	1	1	-	1	2	-	2	-	-	-	-
Lighting towers	-	4	-	-	-	4	-	-	-	4	-	-	-
Mob. welding mach.	-	1	-	1	-	1	-	-	-	1	-	-	-
Mobile compressors	-	1	-	1	-	1	-	-	-	1	-	-	-
Dewatering	-	1	-	1	-	2	-	2	-	2	-	1	-
Dispatch System	-	0	1	-	-	-	-	-	-	-	-	-	-
Emulsion Plant Infrastructures	-	1	-	-	-	-	-	-	-	-	-	-	-
Blasting Mats	-	-	200	100	-	-	-	-	-	-	-	-	-
Tools for Maintenance Shop	-	1	1	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Shovel	-	1	-	-	-	-	-	-	-	-	-	-	-
Technical Services Equipment	-	1	-	-	-	-	-	-	-	-	-	-	-

Table 5.38: Equipment Capital Cost Schedule (k $)

Equipment	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Major Equipment														
Rotary Drill (8.5")	-	-	[redacted]	[redacted]	[redacted]	[redacted]	-	-	-	-	-	-	-	[redacted]
Rotary Drill (5.5")	-	-	[redacted]	-	-	-	-	-	-	-	-	-	-	[redacted]
Shovel (RH340)	-	[redacted]	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Wheel Loader (994)	-	[redacted]	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Shovel (RH200)	-	-	-	[redacted]	-	-	-	-	-	-	-	-	-	[redacted]
Haul Truck (240t class)	-	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	-	-	-	-	[redacted]
Grader (16' blade)	-	[redacted]	[redacted]	-	-	-	-	-	-	-	-	-	-	[redacted]
Water truck (76kl)	-	-	[redacted]	[redacted]	-	-	-	-	-	-	-	-	-	[redacted]
Track Dozer - 580 hp	-	[redacted]	[redacted]	-	-	-	-	[redacted]	[redacted]	-	-	-	-	[redacted]
Wheel dozer - 525 hp	-	[redacted]	-	-	-	-	-	[redacted]	-	-	-	-	-	[redacted]
Sub-Total	-	52,685	32,999	16,453	8,431	8,431	3,466	9,130	5,949	-	-	-		137,544
Support Equipment														
Excavator (2.1 m^3) (Rock Breaker)	-	-	[redacted]	-	-	-	-	[redacted]	-	-	-	-	-	[redacted]
Excavator (2.1 m^3)	[redacted]	-	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Wheel Loader (5.5 m^3)	[redacted]	-	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Articulated Trucks (40 t)	[redacted]	-	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Track Dozer (200 hp)	[redacted]	-	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Grader (14' blade)	[redacted]	-	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Compactor	[redacted]	-	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Small Loader (Hole Stemming)	-	[redacted]	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Boom Truck (22 t)	-	[redacted]	-	-	-	-	[redacted]	-	-	-	-	-	-	[redacted]
Tool carrier	[redacted]	-	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Service truck	-	[redacted]	[redacted]	[redacted]	-	-	[redacted]	[redacted]	-	-	-	-	-	[redacted]
Fuel/Lube truck	-	[redacted]	[redacted]	-	-	-	-	[redacted]	-	-	-	-	-	[redacted]
Lowboy & Tractor Head (150 t)	-	[redacted]	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Pick-up truck	-	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	-	[redacted]
Pit Busses	-	[redacted]	[redacted]	[redacted]	-	[redacted]	[redacted]	-	[redacted]	-	-	-	-	[redacted]
Lighting towers	-	[redacted]	-	-	-	[redacted]	-	-	-	[redacted]	-	-	-	[redacted]
Mob. Welding mach.	-	[redacted]	-	[redacted]	-	[redacted]	-	-	-	[redacted]	-	-	-	[redacted]
Mobile compressors	-	[redacted]	-	[redacted]	-	[redacted]	-	-	-	[redacted]	-	-	-	[redacted]
Dewatering	-	[redacted]	-	[redacted]	-	[redacted]	-	[redacted]	-	[redacted]	-	[redacted]	-	[redacted]
Dispatch System	-	[redacted]	[redacted]	-	-	-	-	-	-	-	-	-	-	[redacted]
Emulsion Plant Infrastructures	-	[redacted]	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Blasting Mats	-	-	[redacted]	[redacted]	-	-	-	-	-	-	-	-	-	[redacted]
Tools for Maintenance Shop	-	[redacted]	[redacted]	-	-	-	-	-	-	-	-	-	-	[redacted]
Spare Bucket for Shovel	-	[redacted]	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Technical Services Equipment	-	[redacted]	-	-	-	-	-	-	-	-	-	-	-	[redacted]
Sub-Total	5,401	6,129	2,919	725	210	950	605	1,835	330	890	210	460	-	20,664
Grand-Total	**5,401**	**58,814**	**35,918**	**17,178**	**8,641**	**9,381**	**4,071**	**10,965**	**6,279**	**890**	**210**	**460**	**-**	**158,208**

6.0 MINERAL PROCESSING AND METALLURGICAL TESTING

6.1 Metallurgical Test Program

6.1.1 General

Initial testwork was conducted in 2006 by Resource Development Inc, coordinated by Mr. William Pennstrom on behalf of Osisko. The testwork was carried out on 6 t of drill core and the corresponding rejects. These samples were collected from three separate core holes within the Canadian Malartic ore body. The metallurgical testwork consisted of grinding, leaching, gravity concentration, flotation and heap leach tests, and cyanide destruction studies.

Additional grinding testwork was carried-out on a bulk sample of 5.4 t by Metso during 2006, in order to provide initial values for the design of the comminution circuit. The analysis of the results of the JKTech drop-weight tests (DWT) determined the JKSimMet parameters used to carry out a preliminary simulation of the grinding circuit.

For the Preliminary Assessment Study, a more comprehensive sampling and test protocol was required. SGS was contracted to carry out this testwork program under the supervision and guidance of BBA and Osisko. This additional testwork program was conducted mainly for grinding, leaching and flotation, in order to develop the best flowsheet in view of the metallurgical results.

For the Feasibility Study an even more comprehensive study including a complete metallurgical test protocol and numerous verifications for equipment selection and performance were required. The program testing was executed principally at SGS but also at the URSTM and at other locations such as FLSmitdth, Outotec, PSI, Knelson, Falcon, SNF, Ciba, Cyplus, etc. for specialized and equipment-related tests. Some of these reports and some results of testwork were not available at the time of submission of the Preliminary Assement Study report but were continued over time and constitute the basis for this Feasibility Study report. Metallurgical test results were reported to BBA when they became available and were reviewed by BBA and Osisko and data analysis was performed. The data and the analyses were reported to Osisko in the form of reports, emails and telecom meeting updates.

The testwork program included the following activities:

- Selection and preparation of metallurgical composites
- Sample receiving, logging and preparation for metallurgical testing.
- Grinding testwork:
  - 11 Full DWT and 2 abbreviated SAG Mill Comminution (SMC) drop tests
  - 22 BWI determinations.
- Flotation testing:
  - 90 Flotation tests
  - 77 Leaching of flotation concentrates
  - 73 Leaching of flotation tails.
- Leaching testing on whole ore:
  - 956 Leach tests.
- Gravimetric testing :
  - 42 Gravimetric tests (including 8 GRG determinations).
- Heap leach testing:
  - 4 Leaching tests
  - 8 Column tests.
- Environmental related tests:
  - 137 Acid base Accounting (ABA) tests
  - 137 Neutralizing Acid Generation (NAG) tests
  - 20 Humidity cell tests
  - 81 TCLP tests
  - 39 Cyanide destruction tests.

6.1.2 Samples

6.1.2.1 Sample Background

Core samples were collected during the period from 2005 to 2008 by the drilling company representatives and a very detailed and rigorous procedure was carried out for logging, storage and assaying (refer to Section 3.5, Exploration and Drilling). Most of the Assaying was performed by Chemex. The majority of the deposit consists of four ore types, namely CPO, SPO, CGR and SGR (Table 6.1).

Table 6.1: Ore Types Description

Ore Type	Description	Average Occurrence as a Percentage of the Orebody
CPO	Potassic altered porphyry with carbonate.	10 %
SPO	Silicified porphyry.	20 %
CGR	Potassic altered greywake with carbonate.	28 %
SGR	Silicified greywake.	42 %

The metallurgical samples were selected to give a good representation of the ore zones in the deposit. Approximately 338 drill holes were used for testwork. The following Table 6.2 shows a summary of the metallurgical samples sent to SGS for testwork and the drawings nos. A1-301-G-0901, A1-301-G-0902, A1-301-G-0903-, A1-301-G-0904, A1-301-G-0905, A1-301-G-0906, A1-301-G-0907 and A1-301-G-0908 included in Appendix 4 show the maps indicating the location of the drill holes used for the test program.

Table 6.2: Summary Description of the Samples Collected for Testwork

Overall Pit Composite		
Testwork	**Batch Per Lithology**	**Number of Drill Holes**
Composite Metallurgical samples (overall comp 1)	CPO-2007-1	26
	SPO-2007-2	25
	CGR-2007-3	25
	SGR-2007-4	25
Composite Grinding Samples	CPO-2007-5	12
	SPO-2007-6	12
	CGR-2007-7	12
	SGR-2007-8	12
Composite Heap Leach Sample	HLSED-2007-5	26
Test Grinding	REMGR	8
Various Metallurgical Tests Average Waste	CPO-2007-9	3
	SPO-2007-10	2
	SGR-2007-12	4
	CGR-2007-11	3
Various Metallurgical Tests High Sulphide Waste	CPO-2007-9	2
	SPO-2007-10	3
	SGR-2007-12	3
	CGR-2007-11	4
Various Metallurgical Tests Low Grade	CPO-2007-9	2
	SPO-2007-10	4
	SGR-2007-12	4
	CGR-2007-11	3
Various Metallurgical Tests High Sulfite Ore	CPO-2007-9	3
	SPO-2007-10	4
	SGR-2007-12	3
	CGR-2007-11	3
Composite Metallurgical samples (overall comp 3)	SGR-2007-16	10
	CGR-2007-15	8
	SPO-2007-14	4
	CPO-2007-13	3
HG test	CPO-2008-109	2
	SPO-2008-110	3
	CGR-2008-111	3
	SGR-2008-112	5

Table 6.2: Summary Description of the Samples Collected for Testwork (continued)

Variability Series		
VAR-1	**Batch Per Lithology**	**Number of Drill Holes**
West/Shallow	CPO-2008-41	1
	SPO-2008-42	1
	CGR-2008-43	3
	SGR-2008-44	2
West/Deep	CPO-2008-45	1
	SPO-2008-46	2
	CGR-2008-47	2
	SGR-2008-48	2
Middle/Shallow	CPO-2008-49	1
	SPO-2008-50	1
	CGR-2008-51	3
	SGR-2008-52	1
Middle/Deep	CPO-2008-53	1
	SPO-2008-54	1
	CGR-2008-55	3
	SGR-2008-56	1
East/Shallow	CPO-2008-57	1
	SPO-2008-58	2
	CGR-2008-76	1
	SGR-2008-59	1
East/Deep	CPO-2008-60	1
	SPO-2008-61	1
	CGR-2008-62	2
	SGR-2008-63	2
VAR-2	**Batch Per Lithology**	**Number of Drill Holes**
North/Shallow	CPO-2008-101	1
	SPO-2008-102	2
	CGR-2008-103	3
	SGR-2008-104	5
North/Deep	CPO-2008-105	1
	SPO-2008-106	2
	CGR-2008-107	3
	SGR-2008-108	5

Table 6.2: Summary Description of the Samples Collected for Testwork (continued)

Confirmation Per sector, 1st Series		
C-1 (West)	**Batch Per Lithology**	**Number of Drill Holes**
0.3 g/t	CPO-2008-64	1
	SPO-2008-65	1
	CGR-2008-66	1
	SGR-2008-67	1
0.6 g/t	CPO-2008-68	2
	SPO-2008-69	1
	CGR-2008-70	1
	SGR-2008-71	2
0.9 g/t	CPO-2008-72	3
	SPO-2008-73	3
	CGR-2008-74	2
	SGR-2008-75	2
C-2 (South)	**Batch Per Lithology**	**Number of Drill Holes**
0.3 g/t	CPO-2008-77	2
	SPO-2008-78	4
	CGR-2008-79	4
	SGR-2008-80	6
0.6 g/t	CPO-2008-93	2
	SPO-2008-94	4
	CGR-2008-95	5
	SGR-2008-96	5
0.9 g/t	CPO-2008-97	2
	SPO-2008-98	4
	CGR-2008-99	4
	SGR-2008-100	5
C-3 (East)	**Batch Per Lithology**	**Number of Drill Holes**
0.3 g/t	CPO-2008-89	2
	SPO-2008-90	3
	CGR-2008-91	3
	SGR-2008-92	6
0.6 g/t	CPO-2008-93	2
	SPO-2008-94	3
	CGR-2008-95	4
	SGR-2008-96	6
0.9 g/t	CPO-2008-97	2
	SPO-2008-98	3
	CGR-2008-99	4
	SGR-2008-100	6

Table 6.2: Summary Description of the Samples Collected for Testwork (continued)

Confirmation Per Sector, 2nd Series					
West	**Batch Per Lithology**	**Number of Drill Holes**	**East**	**Batch Per Lithology**	**Number of Drill Holes**
0.4 g/t	CPO-2008-161	2	0.4 g/t	CPO-2008-145	2
	SPO-2008-162	3		SPO-2008-146	3
	CGR-2008-163	3		CGR-2008-147	4
	SGR-2008-164	6		SGR-2008-148	4
0.8 g/t	CPO-2008-165	3	0.8 g/t	CPO-2008-149	2
	SPO-2008-166	3		SPO-2008-150	2
	CGR-2008-167	3		CGR-2008-151	4
	SGR-2008-168	5		SGR-2008-152	7
1.2 g/t	CPO-2008-169	2	1.2 g/t	CPO-2008-153	2
	SPO-2008-170	3		SPO-2008-154	2
	CGR-2008-171	3		CGR-2008-155	4
	SGR-2008-172	5		SGR-2008-156	4
1.8 g/t	CPO-2008-173	2	1.8 g/t	CPO-2008-157	2
	SPO-2008-174	3		SPO-2008-158	3
	CGR-2008-175	3		CGR-2008-159	3
	SGR-2008-176	6		SGR-2008-160	5
2.5 g/t	CPO-2008-218	1	2.5 g/t	CPO-2008-210	2
	SPO-2008-219	2		SPO-2008-211	2
	CGR-2008-220	2		CGR-2008-212	2
	SGR-2008-221	3		SGR-2008-213	3
4.0 g/t	CPO-2008-222	1	4.0 g/t	CPO-2008-214	1
	SPO-2008-223	2		SPO-2008-215	2
	CGR-2008-224	2		CGR-2008-216	2
	SGR-2008-225	4		SGR-2008-217	4
North	**Batch Per Lithology**	**Number of Drill Holes**	**South**	**Batch Per Lithology**	**Number of Drill Holes**
0.4 g/t	CPO-2008-129	2	0.4 g/t	CPO-2008-113	2
	SPO-2008-130	3		SPO-2008-114	3
	CGR-2008-131	3		CGR-2008-115	3
	SGR-2008-132	5		SGR-2008-116	5
0.8 g/t	CPO-2008-133	2	0.8 g/t	CPO-2008-117	1
	SPO-2008-134	3		SPO-2008-118	3
	CGR-2008-135	3		CGR-2008-119	3
	SGR-2008-136	7		SGR-2008-120	4
1.2 g/t	CPO-2008-137	1	1.2 g/t	CPO-2008-121	2
	SPO-2008-138	3		SPO-2008-122	3
	CGR-2008-139	3		CGR-2008-123	3
	SGR-2008-140	4		SGR-2008-124	5
1.8 g/t	CPO-2008-141	2	1.5 g/t	CGR-2008-206	4
	SPO-2008-142	3		SGR-2008-207	4
	CGR-2008-143	4	1.8 g/t	CPO-2008-125	1
	SGR-2008-144	5		SPO-2008-126	3
2.5 g/t	CPO-2008-198	2		CGR-2008-127	3
	SPO-2008-199	2		SGR-2008-128	5
	CGR-2008-200	2	2.0 g/t	CGR-2008-208	4
	SGR-2008-201	3		SGR-2008-209	5
4.0 g/t	CPO-2008-202	1			
	SPO-2008-203	2			
	CGR-2008-204	2			
	SGR-2008-205	3			

Table 6.2: Summary Description of the Samples Collected for Testwork (continued)

Environmental Tests 2008		
Average Waste	CPO-2008-238	1
	SPO-2008-239	1
	CGR-2008-240	2
	SGR-2008-241	2
High Sulphide Waste	CPO-2008-242	1
	SPO-2008-243	1
	CGR-2008-244	1
	SGR-2008-245	1
Low Grade	CPO-2008-246	1
	SPO-2008-247	2
	CGR-2008-248	1
	SGR-2008-249	2
High Sulphite Ore	CPO-2008-250	1
	SPO-2008-251	3
	CGR-2008-252	1
	SGR-2008-253	2
Average Grade	CPO-2008-254	2
	SPO-2008-255	1
	CGR-2008-256	3
	SGR-2008-257	3

6.1.2.2 Compositing

In general, the composites for metallurgical testwork at SGS have been prepared as follows:

- Individual samples are combined to prepare a composite. The material is screened at 10 mesh (1.7 mm) and any oversize is crushed to minus 10 mesh (1.7 mm) and is split in 1 kg charges. The samples are blended. The composite is then split into charges using a rotary splitter. The head sample is riffled out of one of the test charges.

- For the large quantity of overall composites that have been prepared, each composite has been blended by placing it in a drum which is rotated end over end for about an hour.

For the Grinding testwork the samples were prepared as follows:

- Drill core samples were received by SGS and each one was sampled and prepared for the sag mill comminution (SMC) testwork where the cores are cut in ¼ cylinders using a diamond saw and the test is subsequently performed as per the standard drop weight test (DWT) procedure, except that only one size fraction is tested. The remainder was crushed to minus 6 mesh (3.4 mm) and 10 kg was removed for BWI determinations. Approximately 20 g of this crushed material was used for a specific gravity (SG) determination by pycnometre.

6.1.2.3 Ore Mineralogy

An initial deportment study was conducted by SGS on four composite samples. The objectives of the investigation were to determine the bulk mineralogy and the occurrence of gold in these samples, and to identify and evaluate any mineralogical factors that may affect recoveries.

The findings of this initial examination were:

- Gold mainly occurs as liberated native gold fine particles with some inclusions in pyrite. The gold particles had an average gold content of 87.8% for CPO, 87.1% SPO, 88.5% for CGR and 85.2% for SGR. Gold grains are measured as a geometric mean of average length and average width. Microscopic gold grains ranged from 2 to 50 microns in size for CPO (arithmetic average 16 microns), 1 to 85 microns in size for SPO (arithmetic average 11 microns), 1 to 35 microns in size for CGR (arithmetic average 7 microns) and 1 to 32 microns in size for SGR (arithmetic average 8 microns).

- Approximately 800 g of material passing 300 microns of each as-received sample was preconcentrated by heavy liquid separation at an SG of 2.9 to generate a sink fraction and a float fraction. The percentage of gold carried in the sink fraction (mainly sulphide and iron-oxide) and considered to be mainly liberated or attached, was approximately 54.7% for CPO, 55.8% for SPO, 33.7% for CGR and 36.6% for SGR. For the combined CPO and SPO, 55% of the gold is recovered in the sink fraction. For the combined CGR and SGR, 35% of the gold is recovered in the sink fraction. The overall sink fraction represents only 3% of the sample weight.

- In the four samples, S^{2-} contents are less than 1.6%, and pyrite is the major sulphide mineral.

- In the four samples Fe-oxide/hydroxides (mainly hematite, magnetite, goethite and limonite) are the major iron minerals and considered to be the source of the soluble iron that could come into solution during cyanidation.

6.1.3 Grinding Testwork

Initial grinding testwork was conducted under the supervision of Metso. Representative samples (5.4 t bulk sample) from the deposit were sent to Metso, who conducted BWI determinations. Metso requested Hazen Research Inc. for a full JKtech DWT of those samples. Further analysis of the results of the DWT was used to determine the JKSimMet parameters. A summary of the results is shown in Table 6.3 below. The complete results are included in the Contract Support Services (CSS) Reports. The initial objective of the testwork at that time was to establish the expected capacity of the 38ft x 21ft EGL SAG mill and the two 24ft x 36.5ft EGL ball mill grinding circuits.

Additional grinding testwork (BWI and SMC) was carried-out by SGS on different representative samples of the ore body in order to update but more so to confirm the parameters required for the design of the comminution circuit. The further analysis of the results of the SMC tests was used to determine the JKSimMet parameters.

The following summarizes the results:

- As a first step, BMWI determinations were conducted for each of the four lithologies (CPO, SPO, CGR and SGR) and two of the lithology composites (PO and GR). Additional tests were executed on composites from three zones (sections) of the deposit namely East, West and South and at two depths in the orebody, top 150 m and bottom 150 m.

- The value of A x b from the SMC tests, is a measure of resistance to impact breakage. A high value of A x b means that an ore is soft whilst a low value means that it is hard. The initial test result values indicate that the Malartic ore is moderately hard to hard.

- The further test results confirmed that the ore fell in the medium to hard range of hardness in terms of resistance to impact breakage (A x b) and in terms of BWI.

Table 6.3: Grindability Test Summary

Sample	Ore Density (g/cm3)		DWT	Parameters	BWI (kWh/t)			AI
Name	**Pycno**	**W.D.**	**A x b**	**DWI**	**150M**	***200M1**	***200M2**	**g**
				Metso test				
SPO	-	2,66	36,2	-	-	12,6	-	-
SGR+CGR	-	2,73	35,5	-	-	16,1	-	-
				SGS test				
Overall Comp	-	-	-	-	-	15,6	15,5	-
PO Comp	-	-	-	-	-	15,2	14,8	0,846
CPO-2007-05	-	2,67	33,0	8,0	13,7	14,9	-	0,604
SPO-2007-06	-	2,66	38,2	6,9	13,0	14,7	-	0,654
GR Comp	-	-	-	-	-	16,1	15,9	0,476
CGR-2007-07	-	2,68	33,8	7,8	15,0	16,2	-	0,454
SGR-2007-08	-	2,69	37,7	7,0	14,7	15,9	-	0,533
REMGR	-	2,67	39,9	6,7	14,5	14,6	-	0,786
Comp G-E/S	2,77	2,75	44,3	6,2	-	15,9	-	0,455
Comp G-E/D	2,75	2,69	42,7	6,3	-	17,9	-	0,485
Comp G-M/S	2,75	2,74	40,8	6,7	-	16,8	-	0,677
Comp G-M/D	2,76	2,74	42,3	6,5	-	17,1	-	0,477
Comp G-W/S	2,75	2,73	35,3	7,7	-	15,1	-	0,473
Comp G-W/D	2,77	2,73	36,4	7,5	-	16,9	-	0,399
Minimum	**2,75**	**2,66**	**33,0**	**6,2**	**13,0**	**12,6**	**14,8**	**0,399**
Maximum	**2,77**	**2,75**	**44,3**	**8,0**	**15,0**	**17,9**	**15,9**	**0,846**
Average	**2,76**	**2,70**	**38,2**	**7,0**	**14,2**	**15,7**	**15,4**	**0,563**

*200M1 and 200M2 tests were done in duplicate.

Pycno: ore specific gravity measured by pycnometre.

W.D.: ore density measured by water displacement technique on rocks -31.5 / +26.5 mm.

It can be observed from Table 6.3 that the results are very consistent within each set of tests and between the independent test facilities (Metso and SGS) both for the impact grinding characteristics as well as the BWI determinations. A total of 16 samples were tested for either BWI or DWT and all results fell in the medium to hard range of hardness.

6.1.3.1 JKSimMet Simulations

Following the initial grinding test work CSS were contracted to perform additional grinding circuit simulations based on the DWT and BWI determinations. Based on JKSimMet's database information the SAG mill performance was modeled. Based on this model and with the BWI values that were determined for the ball mills for the Malartic ore, various grinding circuits were simulated. SMC test (Sag mill comminution) is a test which uses particles that are normally cut from drill core using a diamond saw to achieve close size replication; SMC tests represent a very good indicator of the "grinding behavior" of the ore tested. Different simulations were performed to determine the effect of pre-crushing of the SAG mill feed in order to maximize the tonnage. The simulation assumptions for the bond work indices were at 16.1 kWh/t for the primary ball mill circuit and, for the secondary ball mill circuit a higher value of 17.3 kWh/t was assumed in order to reflect the lower efficiency for finer grinding. The final objective of the test was to find the optimal P_{80} at 55 000 tpd and to limit pebble crusher tonnage to around 600 tph.

The simulation results confirmed that the grinding circuit consisting of one 26,000 hp SAG mill and three 16,000 hp ball mills coupled in a two and one configuration could produce a of P_{80} = 64 microns at a daily throughput of 55,000 tpd or a P_{80} of 60 microns at 53,000 tpd (Table 6.4). It is felt that both these initial simulation results are conservative as the selected feed size fraction curve from crusher suppliers did not contain a significant quantity of fine particles as it will in reality. These fine particles will lower the SAG mill power consumption allowing for either a smaller transfer size to ball mill circuits or more capacity to the SAG. In our current configuration of one SAG and three ball mills (2+1), increasing tonnage of the SAG would put even more pressure on the ball mill circuits producing a coarser feed to the leach which is not wanted as recovery would drop. The latest simulations incorporating a higher ratio of fines in the SAG mill feed indicate the possibility to maintain the throughput at 55,000 tpd producing a finer leach feed, down to 60 microns.

Table 6.4: Summary of Grinding Circuit Simulations

Test	F80	Precrush	*Ball mills	Capacity	Sag	BM	Sag screen	Pebble crusher			P_{80} (um)
#	(inch)	C.S.S. (inch)	Config.	t/d	kW	kW	P_{80} (um)	tph	F80 (mm)	kW	Final
Maximum Sag tonnage with precrushed cone crusher and maximum tonnage and power pebble crusher											
1	7.5	< 3/4"	3 (P)	88 000	17 940	35 733	3402	846	46	1158	148
2	7.5	< 3/4"	2 (P) + 1 (S)	88 000	17 940	35 739	3402	846	46	1158	131
3	7.5	< 3/4"	1 (P) + 2 (S)	88 000	17 940	35 765	3402	846	46	1158	124
4	7.5	< 3/4"	4 (P)	75 100	16 723	47 716	3215	775	48	1077	75
5	7.5	< 3/4"	2 (P) + 2 (S)	85 000	17 585	47 698	3357	831	46	1077	75
Optimal P80 at 55 000 tpd with precrushed and maximum tonnage pebble crusher <600 tph											
6	8	< 3/4"	3 (P)	55 000	14 420	35 625	2205	434	50	635	62
7	8	< 3/4"	2 (P) + 1 (S)	55 000	14 420	35 565	2205	434	50	635	55
8	8	< 3/4"	1 (P) + 2 (S)	55 000	14 420	35 664	2205	434	50	635	52
Optimal P80 at 55 000 tpd no precrushed and maximum tonnage and power available pebble crusher											
9	7.5	No	3 (P)	55 000	17 973	35 763	3227	784	47	1109	72
10	7.5	No	2 (P) + 1 (S)	55 000	17 973	35 668	3227	784	47	1109	62
11	7.5	No	1 (P) + 2 (S)	55 000	17 973	35 756	3227	784	47	1109	59
Optimal P80 at 55 000 tpd no precrushed and maximum tonnage pebble crusher <600 tph											
12	7	No	2 (P) + 1 (S)	55 000	17 908	35 797	3335	597	41	724	65
13	7	No	1 (P) + 2 (S)	55 000	17 908	35 794	3335	597	41	724	64
14	7	No	2 (P) + 1 (S)	52 100	17 587	35 796	3216	577	41	706	60
15	7	No	1 (P) + 2 (S)	53 000	17 680	35 794	3250	582	41	711	60
16	6	No	1 (P) + 2 (S)	55 000	16 439	35 798	2555	492	41	618	60
17	5	No	1 (P) + 2 (S)	55 000	16 379	35 520	2520	491	41	616	60
18	6	No	2 (P) + 1 (S)	55 000	16 679	35 798	2220	550	40	615	60
19	5	No	2 (P) + 1 (S)	55 000	16 454	35 795	2240	568	40	635	60

* Ball mill configurations : (P) for primary and (S) for secondary

Test # 1 to 3: Precrushed and maximum tonnage & power at SAG with 3 BM and K_{80} for flotation

Test # 4 & 5: Same as previous but with 4th BM ==> smaller P_{80}

Test # 6 to 8: 55 000tpd at Sag with precrushed but BM configuration to optimize P_{80}

Test # 9 to 11: 55000tpd at Sag no precrush but smaller close setting at gyratory

Test # 12-13: 55000tpd at Sag with maximum power and Pebble <600 tph (most realistic scenarios)

Test # 14-15: P_{80} at 60 microns and pebble <600 tph

Test #16 to 19: Finer P_{80} assumed for gyratory crusher

6.1.4 Testwork Program

6.1.4.1 Flotation Testwork

The initial goal of the flotation tests was to establish bulk sulphide rougher flotation conditions that promoted high gold recovery in the concentrate and low losses of gold in the tails. The effect of fineness of grind, collectors, modifiers and pH were tested. Preliminary bulk flotation tests were conducted on four lithologies (SPO, CPO, CGR and SGR). In those initial tests, ore was ground to a P_{80} = 75 microns. Additional testwork was conducted for each of the two composites of the two major lithologies (PO and GR). The ore was ground to 73, 90, 113 and 152 microns approximately for PO and 74, 98, 152 and 260 microns for GR. Finally, testwork was conducted on an overall composite of the four lithologies.

The major conclusions indicate that:

- Higher gold recoveries are obtained at finer grind size.
- A mixture of PAX (125 g/t), 3418A (100 g/t) at a grind of P80 = 97 microns recovered 84.3% to 87.2% of gold after 25 minutes of flotation with 19.8% to 21.4% weight recovery.
- Flotation concentrate grades ranged from 4.63 to 4.78 g/t Au.
- Flotation of an Overall composite: gold recovery to flotation concentrate averaged 85.9% and the tails gold assay averaged 0.2 g/t Au. With such a low recovery to the flotation concentrate, the overall recovery, with leaching of the flotation concentrate only, was projected to be in the range of 78% to 80%.

The final conclusion was that, due to the low recovery by this process route, flotation/leach of flotation concentrate was not pursued as a viable process option. In this option flotation tails are rejected as final tails along with the residue from the flotation concentrate leach.

Additional flotation tests were conducted based on flotation of a coarse ground feed (P_{80} = 130 microns), regrind of flotation concentrate and leaching of both flotation concentrate (reground) and flotation tails. For details of these tests, refer to Section 6.1.4.6.

6.1.4.2 Leach Testwork

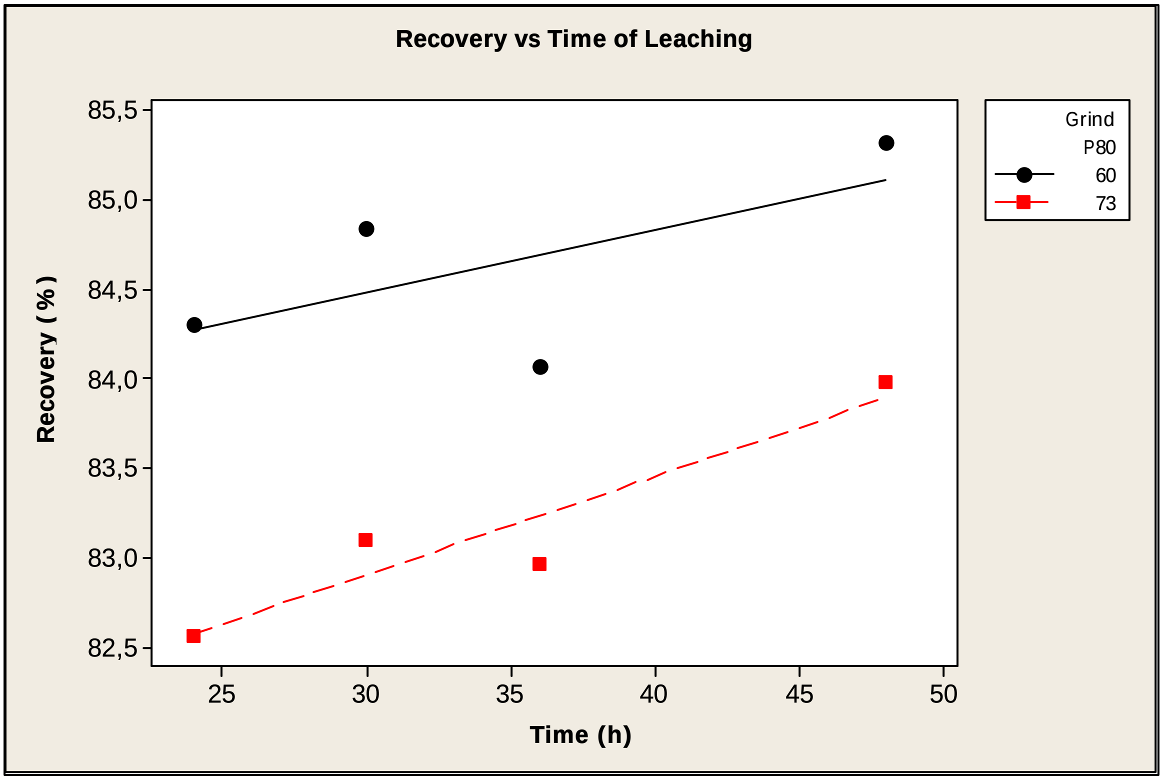
A series of scoping tests for whole ore leach were conducted on individual lithologies (CPO, SPO, CGR and SGR), composites of lithologies (PO and GR) and overall composite (OA) samples, to study the effect of process variables on gold extraction. The variables studied were grind size, pH, leach time and, cyanide, oxygen and lead nitrate addition, cyanide and lime consumption, and carbon in leach. The standard conditions used for leach tests were to grind the material to the proper sizes and pulp with water to 40% solids. The pH was adjusted with quicklime to 11.0 and the cyanide concentration was 0.5 g/l. The slurry was bottle leached at the required leached times. Table 6.5 is showing the relationship between time, grind and recovery on the Preliminary Assessment Study composite sample (overall comp 1).

Table 6.5: Effect of Grinding in Leach for the Preliminary Assessment Study Overall Composite (Overall Comp 1)

Leach Parameters Extraction % Au				
Leach Time (h)	**Grind Size P_{80} (microns)**			
	45	**65**	**75**	**95**
24 h	89	84	83	86
	87	84	82	86
	83	85	84	84
	85	83	82	83
	89	81	82	84
24 h (Average)	86.4	83.4	82.6	84.3
30 h	84.7	84	81.3	83.1
	84	84	80.4	84.2
	86	85	86	81.1
	86	85	85	82.9
	85.7	84	83.9	82.3
30 h (Average)	85.8	84.3	83.3	82.7
48 h	86.2	84.1	82.5	81.8
	87	83.5	81.8	82.9
	86.9	85.3	84.3	81
	86.1	85.2	85	81.5
	87.1	84.7	84.6	82.2
48 h (Average)	86.7	84.6	83.6	81.9
Tails (g/t Au)	0.15	0.17	0.17	0.21
Calc. Head (g/t Au)	1.12	1.12	1.08	1.13

A second series of tests were done on composite pit samples (overall comp 3) to confirm the initial results. Similar behavior when compared with the Preliminary Assessment Study tests were observed except that cyanide consumption was much lower mainly due to grinding in a stainless steel grinding mill. These tests show that extended time of leaching improves the recovery (Figure 6.1).

Figure 6.1: Gold Extraction (%) Versus Leaching Time for Pit Composite Sample (Overall Comp 3)



These tests also proved that finer grinding improves the recovery as shown in Figure 6.2. Overall recoveries obtained from testing the same pit composite, after regrinding flotation concentrate, is also shown on Figure 6.2. As mentioned previously the complete flotation test work is not part of the Feasibility Study report. The idea to include the flotation metallurgical results on the same graph as the leach tests was to show that the relationship between the grind and recovery is still consistent at grind as fine as a P_{80} of 20 microns.

Figure 6.2: Gold Extraction (%) Versus Leach Feed Grind Size (microns) at 30 h Leaching Time for Pit Composite Sample (overall comp 3)

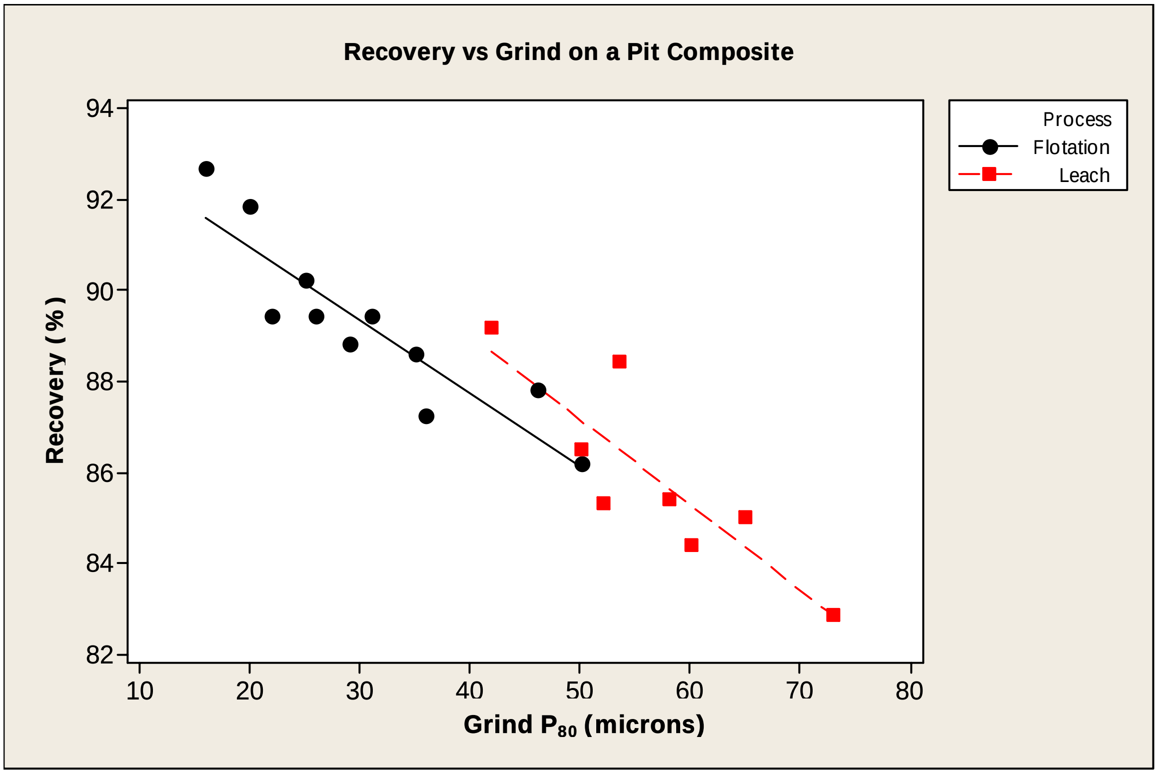


Table 6.6 as well as Figures 6.3 and 6.4 are showing the different consumptions for lime and cyanide according to the various areas tested. The East zone is the area with the highest consumption for both lime and cyanide.

Table 6.6: Lime and Cyanide Consumption for Each Area of the Deposit

Area	Cyanide Consumption (kg/t)	Lime Consumption (kg/t)
Pit Comp.	0,028	0,641
East	0,052	0,813
North	0,029	0,439
South	0,014	0,436
West	0,027	0,656
Average	0,031	0,586

Figure 6.3: Lime Consumption for Each Area of the Deposit

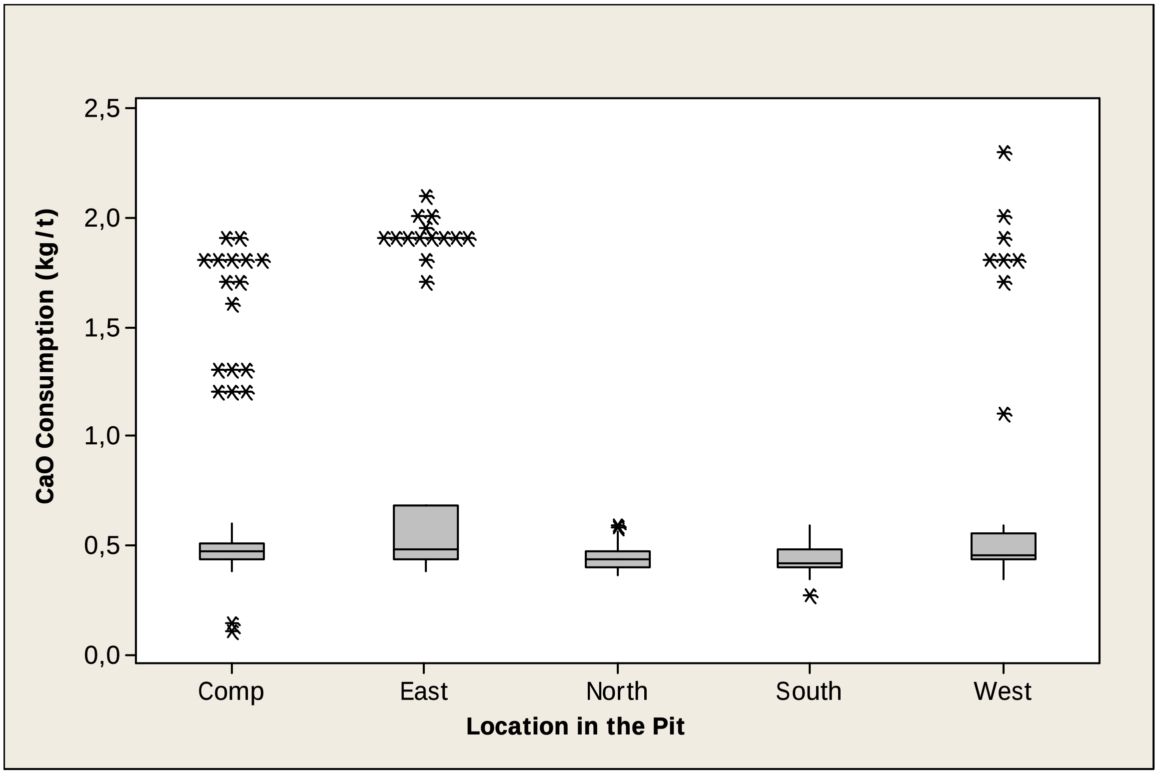
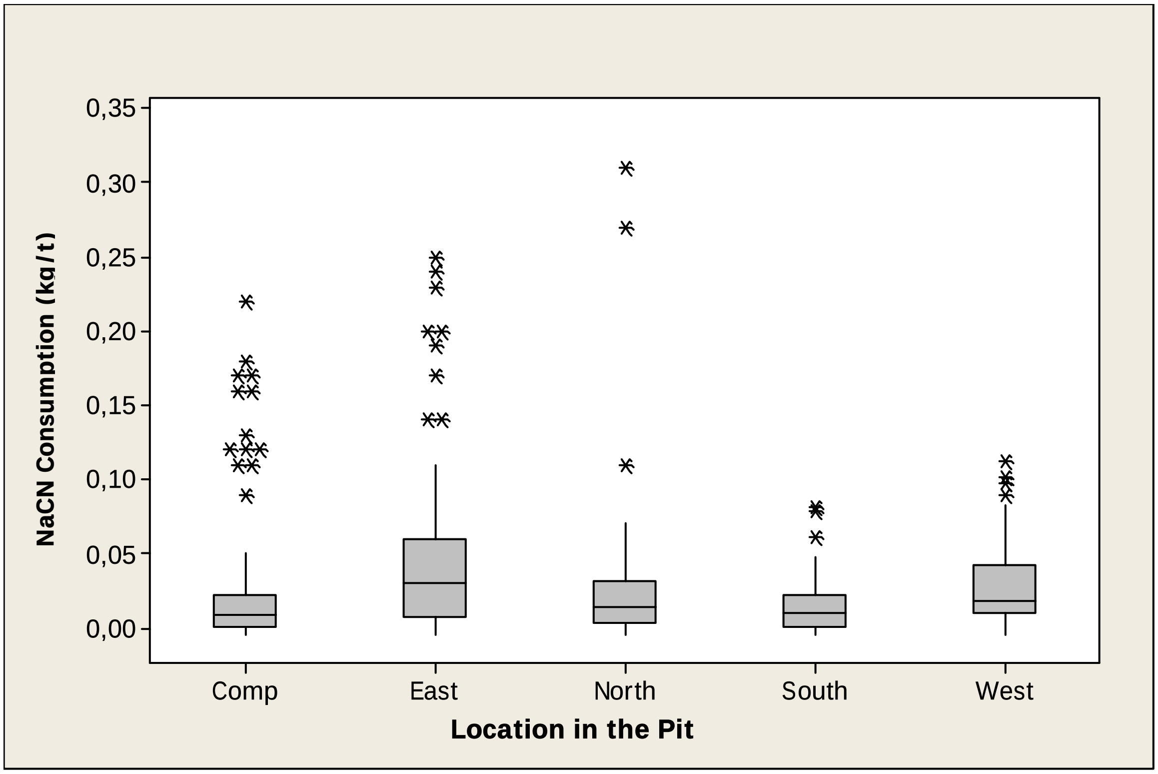


Figure 6.4: Cyanide Consumption for Each Area of the Deposit



The general observations are:

- The majority of the gold is extracted after 24 h.
- Gold recovery is responding to finer grind.
- The NaCN consumption is relatively low and generally does not exceed 0.10 kg/t. Some tests were conducted with fresh carbon addition and in these cases, the cyanide consumption was averaging 0.20 kg/t.
- Leach tests at higher than 500ppm NaCN concentration did not show improvement on recovery.
- The lime consumption averaged 0.60 kg/t when maintaining a pH of 11.
- Lead nitrate addition has not shown any improvement on the recovery or on cyanide consumption for either the whole ore leach process nor for leaching of flotation concentrate.
- Oxygen consumption is relatively low and increasing the DO levels has shown no significant improvement on recovery.

- Higher pH did not show significant improvement on recovery and only affected cyanide consumption, positively.
- Due to the presence of very fine gold particles, gravimetric recovery is not effective.

6.1.4.3 Deportment Study and Diagnostic Leach on Leach Tails

Following the initial metallurgical test program and the Preliminary assessment Study results an additional deportment study was conducted by SGS on a composite sample of leach tails. The objective of the investigation was to determine the occurrence of gold in the sample and to identify any mineralogical factors that could affect the dissolution of the gold.

The protocol and findings of this examination were:

- Approximately 800 g of as-received sample was preconcentrated by heavy liquid separation at an SG of 2.9 to generate a sink fraction and a float fraction. The percentage of gold carried in the sink fraction (mainly sulphide and iron-oxide) was 67.9%. The overall sink fraction represents 7.6% of the sample weight.
- The gold in the sink fraction is mainly locked in sulphides.
- The gold in the float fraction (accounting for 92.4% of the total mass and 32.1% of the total gold) is mainly locked in non-opaque minerals.
- The gold grains observed ranged in size from 0.7 to 5 microns with an average of 2 microns all locked in sulphides (mainly in Pyrite).
- The gold locked in sulphides accounted for over 60%, the remaining gold was considered to be locked in non opaque minerals.

A diagnostic leach was performed on a leach tail residue of the same set of tests. The tail samples were split in two and the first half was pre-leached in hydrochloric acid prior to being leached with cyanide. The second half of the sample was pre-leached with Aqua-Regia then leached in cyanide. The results showed that 8% of the gold was associated with Carbonates, Fe Oxydes or Pyrrhotite (liberated by Hydrochloric acid) and 77% was associated with sulphide. The reminder of the gold was trapped in the gang (not liberated by Aqua-Regia).

Considering that the gold that was not leached is encapsulated it was determined that the grind is the factor that has the most effect on the recovery. At this point, a P_{80} of 64 microns was established as the required grind for the process design.

A series of additional flotation tests was conducted with the intent of first concentrating the sulphide and then regrinding the sulphide prior to cyanadation. The deportment study of the leach tails forms the basis for investigating the concept of flotation plus leach of both reground concentrate and tails. For details of these tests, refer to section 6.1.4.6.

6.1.4.4 Recovery Curves by Area of the Deposit.

Following the metallurgical testwork program of the Preliminary Assessment Study, it was determined that the deposit could be split in 4 main zones; this based on metallurgical results from the tests executed on various specific drill holes.

A new set of samples were selected and prepared from those 4 major areas of the deposit. The 4 zones (North, South, East and West) were split to represent similar metallurgical behavior in each zone throughout the deposit. After several tests it was determined that the North zone could be split in two sub-parts: the deep part linked to the West zone and the shallow part linked to the East zone. Two sets of samples were selected and prepared for a complete new testwork leach program. The first set of samples was selected to represent the East, West and South zones at 3 different grades (3 grades per zone) of 0,3 g/t Au, 0,6 g/t Au and 0,9 g/t Au with the intent to establish recovery curves for each zone of the deposit. The second set of samples was selected to represent the East, West and North zones at 2 depths in the orebody. Shallow samples represent a depth of 150 m below the surface and deep samples are taken at a depth of 150 m from the bottom of the deposit. Leach tests were repeated several times on the same sample and at the same leach conditions to replicate the results. The selected leach conditions were 500ppm NaCN, pH of 11.0, a P_{80} of 64 microns and a leach time of 30 hours. Those results are presented in Table 6.7.

Table 6.7: Gold Recoveries on Several Samples of the Deposit

Sample	Zone	Au Recovery (%)	Head Grade (g/t Au)	Used for Regression
Env/HL-6	Pit Comp.	80,7	0,61	Reg
Env-HL-9	Pit Comp.	83,1	0,89	Reg
Overall-1	Pit Comp.	84,5	1,13	Reg
Overall-3	Pit Comp.	85,2	1,23	Reg
C3-3	East	79,0	0,32	Reg
C3-6	East	79,8	0,64	Reg
C3-9	East	81,9	0,82	Reg
C2-3	South	78,9	0,33	Reg
C2-6	South	80,0	0,70	Reg
C2-9	South	79,9	0,97	Reg
Env-3	West	80,2	0,38	Reg
C1-3	West	77,2	0,31	Reg
C1-6	West	81,7	0,58	Reg
C1-9	West	85,0	0,87	Reg
Var-W/S	West	82,6	0,84	Reg
Var-W/D	West	87,4	0,97	Reg
Var1-D	West	86,4	0,81	Reg
Var2-D	West	81,4	0,89	Reg
Var1-S	East	82,3	0,87	Reg
Var2-S	East	82,8	0,95	Reg
Var-E/S	East	82,5	0,93	Reg
Var-E/D	East	84,5	0,86	Reg
N-04	North	75,7	0,37	-
N-08	North	83,4	0,69	-
N-12	North	86,9	1,20	-
N-18	North	85,6	1,83	-
N-25	North	90,6	2,51	-
N-40	North	90,4	4,07	-
S-04	South	75,5	0,43	-
S-08	South	82,0	0,82	-
S-12	South	83,6	1,26	-
S-15	South	79,5	1,53	-
S-18	South	82,2	1,84	-
S-20	South	85,5	2,01	-
E-04	East	82,3	0,41	-

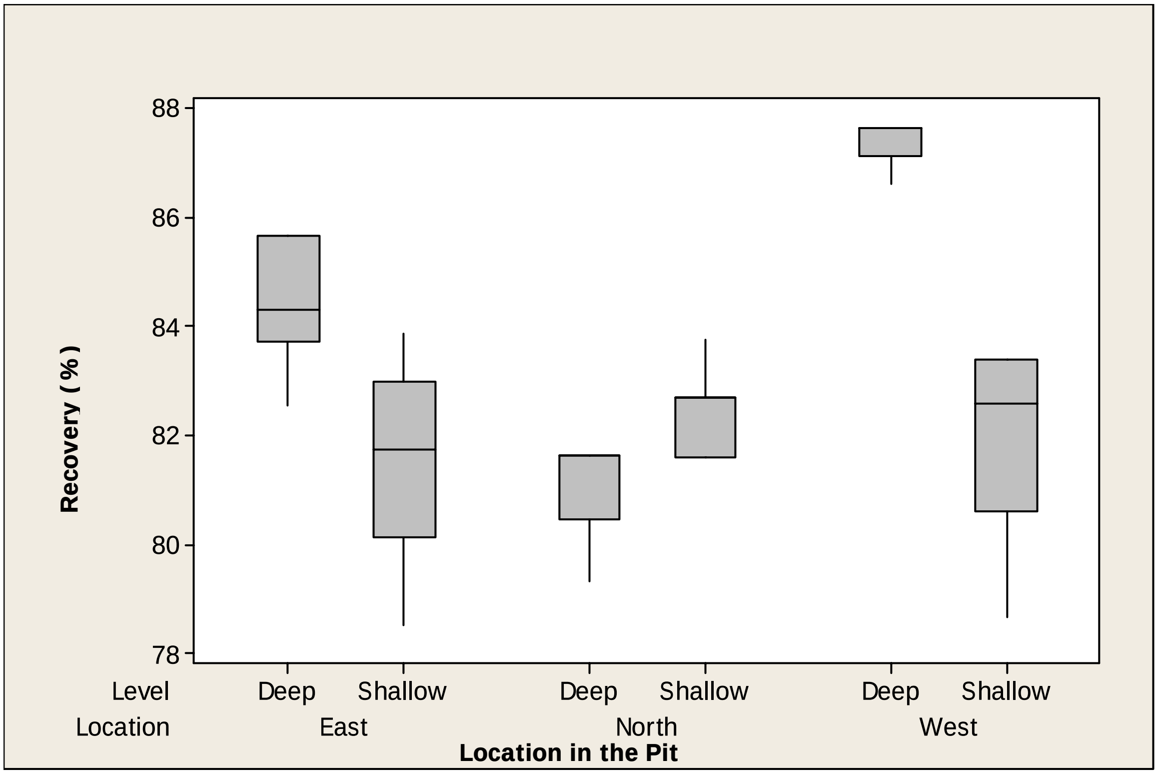
Table 6.7: Gold Recoveries on Several Samples of the Deposit (continued)

Sample	Zone	Au Recovery (%)	Head Grade (g/t Au)	Used for Regression
E-08	East	84,0	0,84	-
E-12	East	80,8	1,20	-
E-18	East	85,9	1,81	-
E-25	East	87,2	2,85	-
E-40	East	90,9	4,13	-
W-04	West	79,7	0,46	-
W-08	West	82,5	0,84	-
W-12	West	87,9	1,20	-
W-18	West	86,1	1,78	-
W-25	West	87,8	2,68	-
W-40	West	93,0	4,11	-
High Grade	Pit Comp.	88,2	2,14	-

Recoveries proved to be lower in the top 150 meters (shallow sample) and generally higher with the deep samples except for the North zone. The larger differential between shallow and deep sample metallurgical results was noticed on the West zone samples with a 5% differential in favor of the deep area. The delta recovery on the other zones showed around a 1% difference.

Figure 6.5 is showing the differential of recovery between the top layer and the bottom layer for 3 zones of the deposit (West, North and East).

Figure 6.5: Recoveries Per Zone and Different Depths of the Deposit

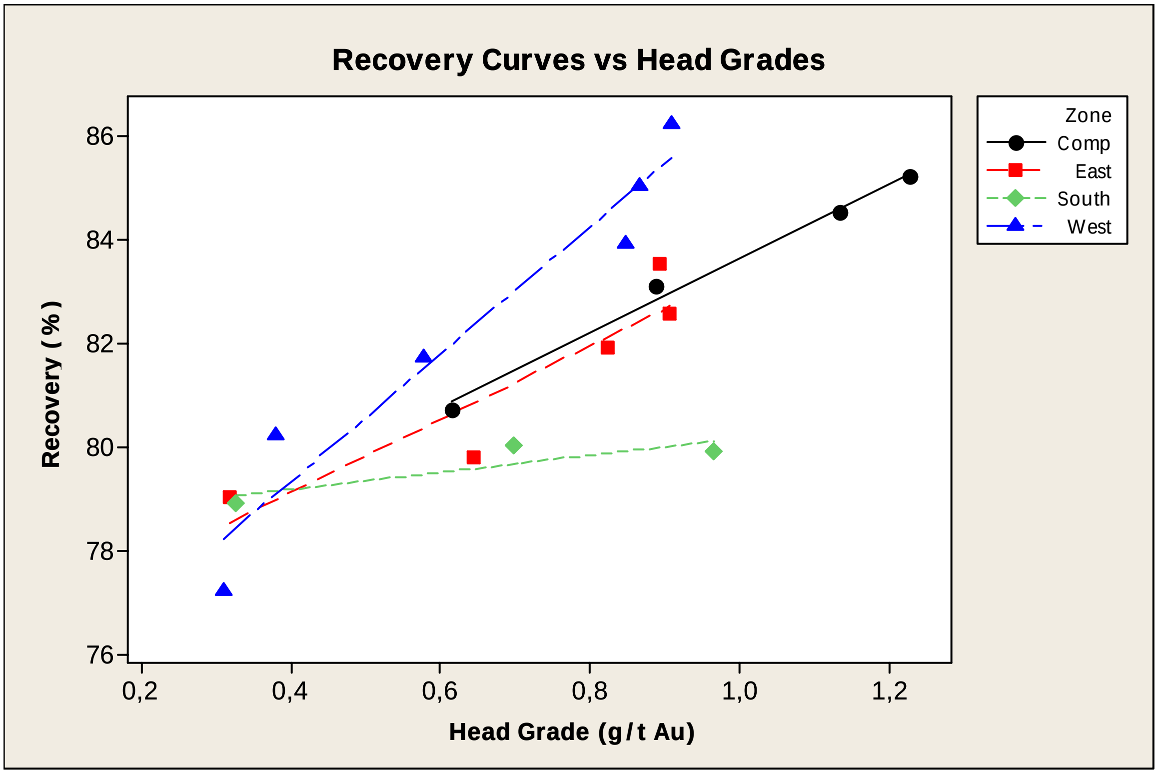


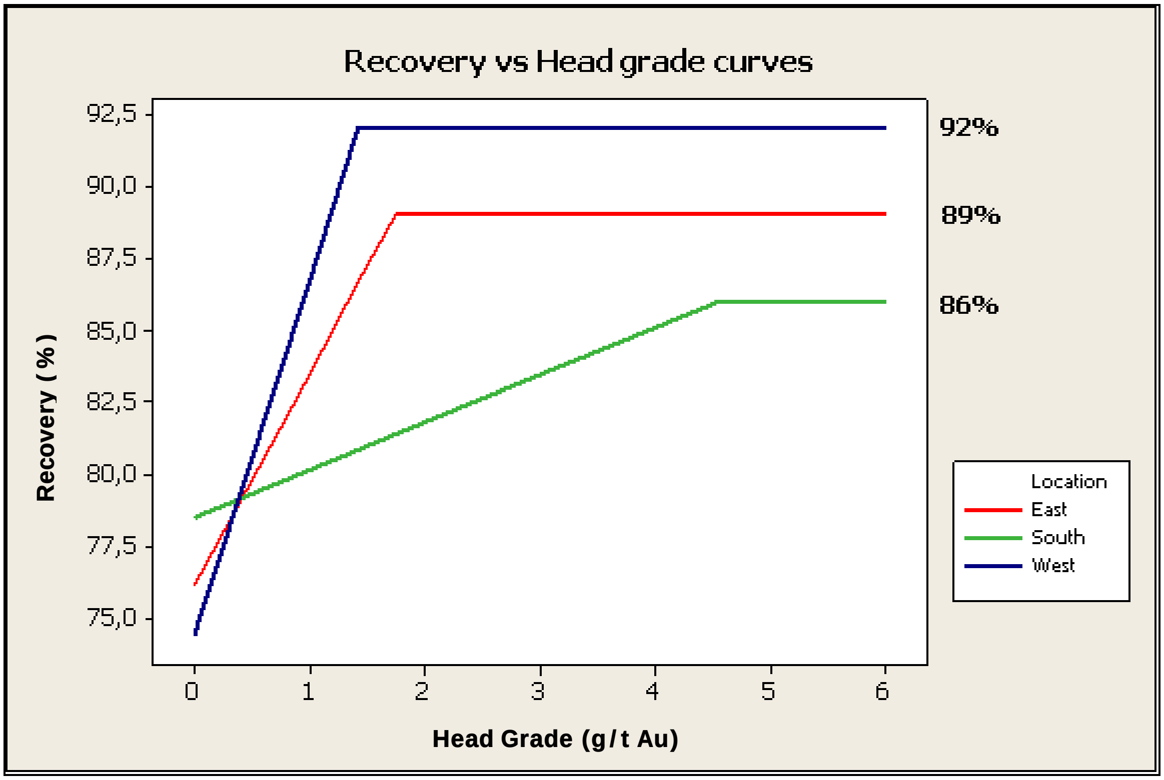
Gold recoveries were calculated using the average recalculated head grade from all leach tests of the database. The average recovery obtained for each of the samples was used to plot the recovery curves of each of the 3 areas. The regression curves were based on recovery results weighted with the gold content of the respective zone. Figure 6.6 is showing 2 graphs of the 3 regression curves established for West, East and South zones; the head grade scale was varied to better show the recovery trend with higher grade. A regression curve from a composite sample selected to represent all zones of the deposit is also included in that same Figure 6.6. A total of 18 different composite samples represent the 3 zones of the deposit:

- West : 85 leach tests from 8 composite samples
- East : 42 leach tests from 7 composite samples
- South : 36 leach tests from 3 composite samples.

The South zone has a significantly different grade-recovery response curve from the other zones. Most of the gold from the South zone is appearing in the top 200 meters.

Figure 6.6: Regression Curves Established for West, East and South Zones





The recovery equation for each of the 3 major zones are:

- West (including North Deep) [74.41 + (12.300 x head grade g/t Au)] (Max 92%)
- East (including North Shallow) [76.15 + (7.339 x head grade g/t Au)] (Max 89%)
- South [78.50 + (1.654 x head grade g/t Au)] (Max 86%).

The highest recoveries were obtained in the West zone and the lowest in the South. The South zone contains the lowest ratio of PO in the deposit which may explain why the recovery is lower. GR material samples proved to have lower recoveries throughout the complete testwork program. A maximum recovery value was attributed to each recovery curve based on historical recovery values and test results of each zone. These equations represent the basis of the recovery calculation used in the pit design model.

Figure 6.7: West Zone Recovery Curve

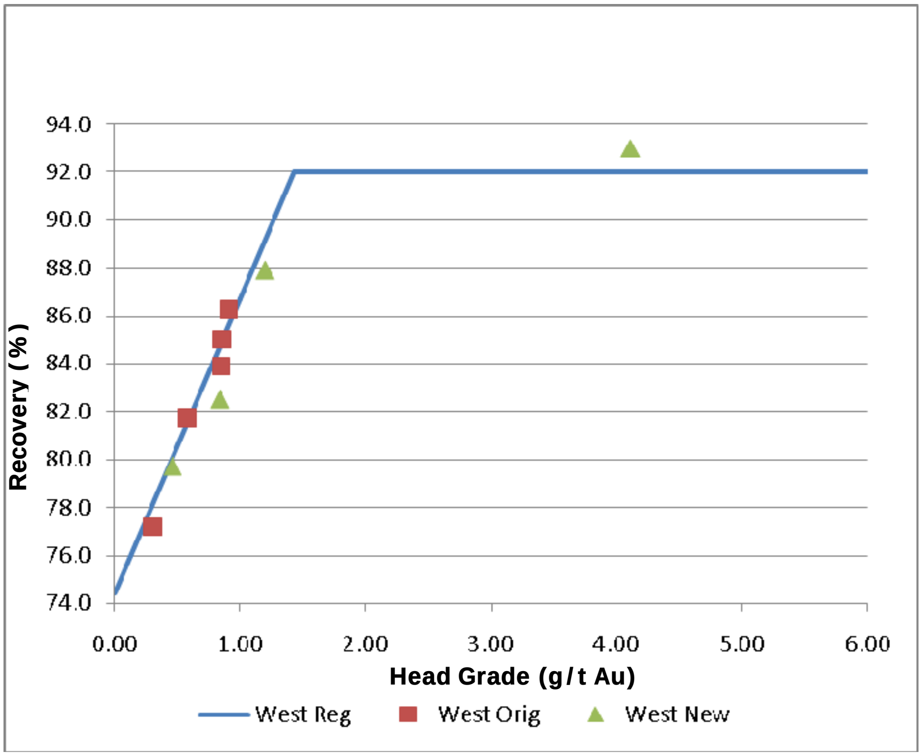


Figure 6.8: East Zone Recovery Curve

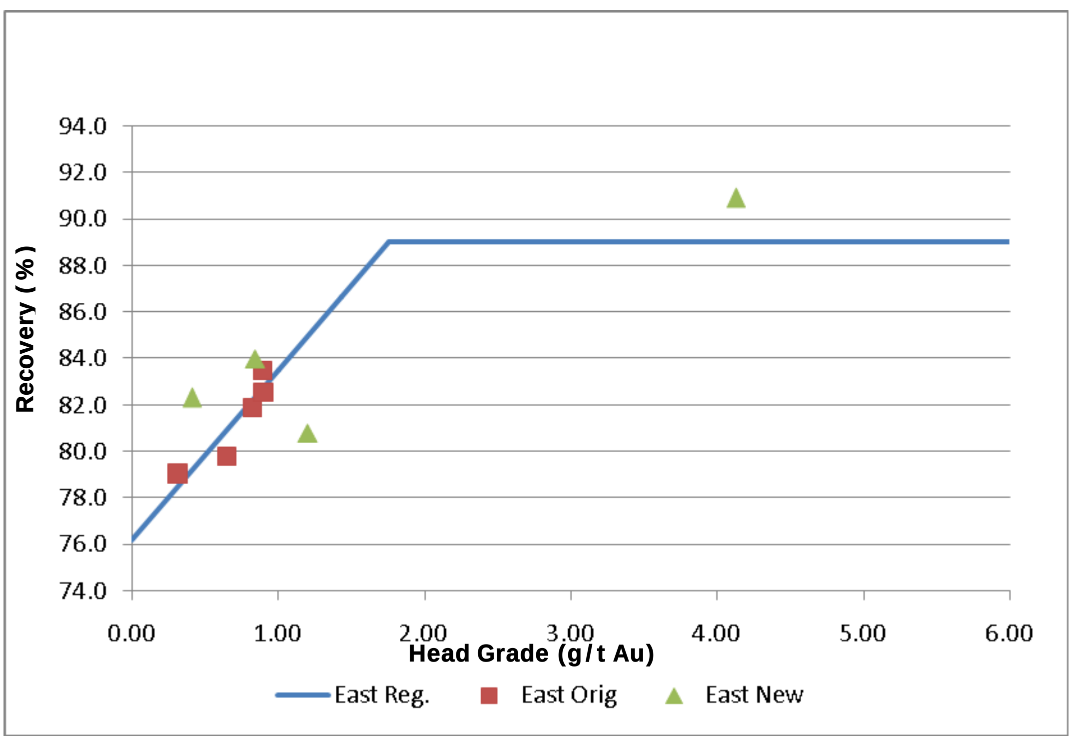
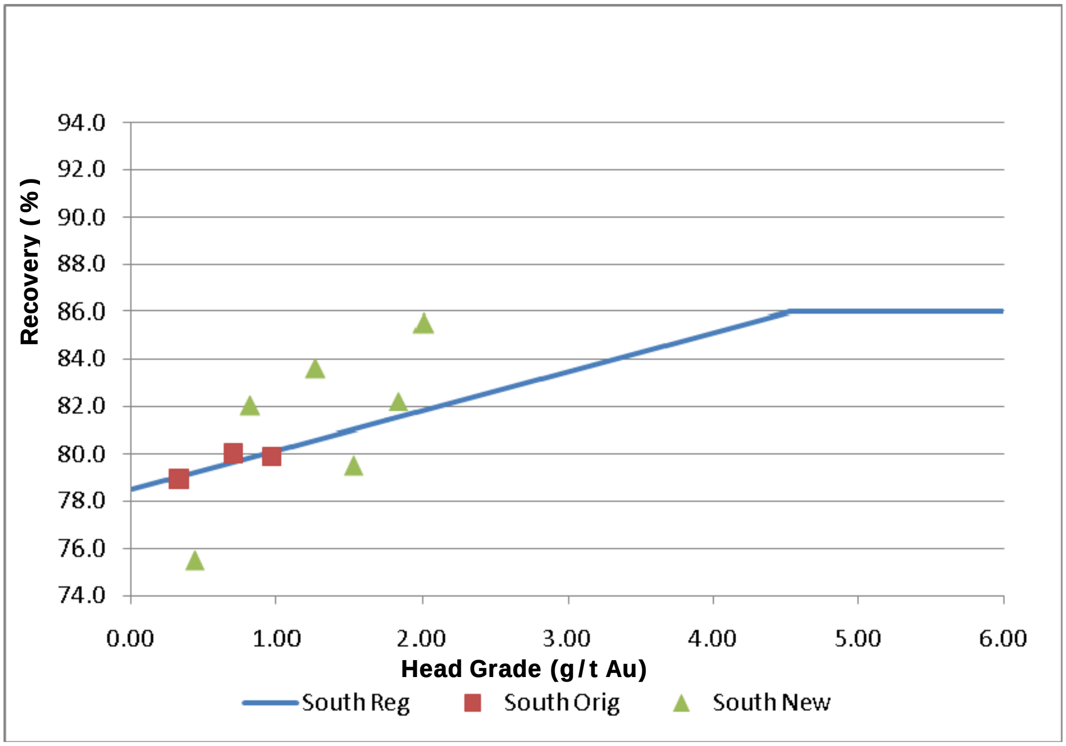


Figure 6.9: South Zone Recovery Curve



6.1.4.5 <u>Extraction of Silver</u>

Assays for silver were done on solids and solutions of the leach tests performed on the 4 zones of the deposit; the results are presented in Table 6.8.

Table 6.8: Silver Recovery on Several Samples in the Deposit

Sample	Zone	Head Grade Ag (g/t Ag)	Recovery Ag (%)	Regression Group
Conf 1st serie 03	West	1,04	70,0	W-N-E
Conf 1st serie 06	West	1,30	72,2	W-N-E
Conf 1st serie 09	West	1,53	85,7	W-N-E
Conf 3rd serie 03	East	0,66	75,6	W-N-E
Conf 3rd serie 06	East	1,72	81,8	W-N-E
Conf 3rd serie 09	East	2,18	86,4	W-N-E
E04	East	1,01	70,3	W-N-E
E08	East	1,77	83,0	W-N-E
E12	East	2,42	87,6	W-N-E
E18	East	3,02	90,1	W-N-E
N04	North	1,38	78,3	W-N-E
N08	North	1,58	81,0	W-N-E
N12	North	2,23	83,8	W-N-E
N18	North	2,61	80,8	W-N-E
S04	South	1,29	58,2	South
S08	South	1,31	62,0	South
S12	South	1,76	71,5	South
S18	South	2,53	74,3	South
W04	West	1,28	76,6	W-N-E
W08	West	1,54	80,5	W-N-E
W12	West	1,83	83,6	W-N-E
W18	West	1,87	84,0	W-N-E

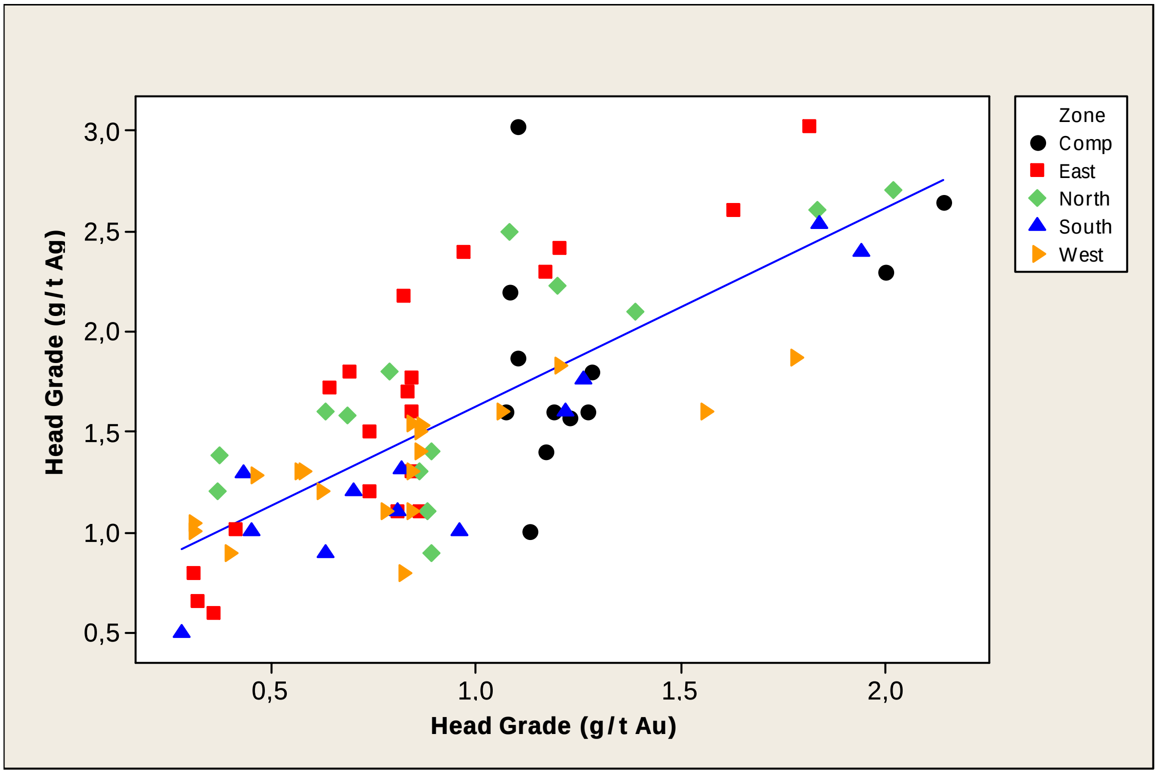
Correlation curves were developed to estimate the silver content in the 4 different areas (North, South, East and West) and the same exercise was repeated for the silver recoveries in each of the 4 zones. It was established that the simplest way to correlate the silver recovery and silver content was to derive one equation for the grade of silver and 2 equations for the recovery. As for the gold, the recovery of silver proved to be significantly lower in the South area of the deposit.

The equation obtained for the estimation of silver content (head grade) is:

- All zones: [0,638 + (0,987 g/t Au)].

The silver head grade regression was calculated using both the calculated head grade based on leach test assay results on solids and solutions and on head assays of a composite sample. Figure 6.10 is showing the regression curve of the relationship silver content vs. gold content.

Figure 6.10: Silver Content vs. Gold Content Regression Curve



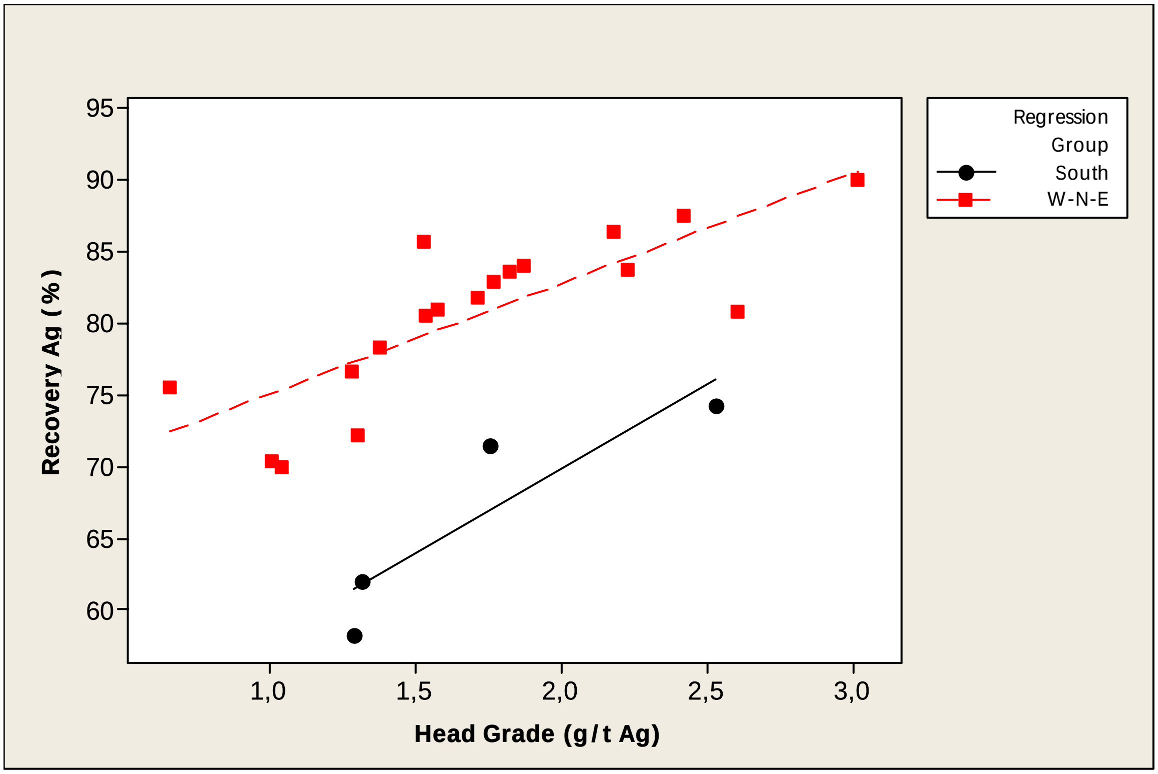
Silver recoveries were calculated using the average recalculated head grade from all leach tests of the database. The average recovery obtained for each sample was used to plot the recovery curves. The estimated loss of silver in full scale operations vs. laboratory dissolution tests was estimated at 10%. The 10% loss has been included (deducted) in the equations.

The equations obtained for the recovery of silver are as follow:

- W-N-E zones: [57.55 + (7.983 x head grade g/t Ag)] (Max 85%)
- South Zone: [36.21 + (11.770 x head grade g/t Ag)] (Max 70%).

Figure 6.11 is showing the regression curve (not including the deduction of 10% for silver recovery).

Figure 6.11: Silver Recovery Regression Curves



The recovery curve regression analysis is based on a total of 22 different composite samples representing the different zones of the deposit:

- West : 38 leach tests from 7 composite samples
- East : 60 leach tests from 7 composite samples
- North : 20 leach tests from 4 composite samples
- South : 19 leach tests from 4 composite samples.

6.1.4.6 Flotation Testwork: Concentration of Sulphides

Initial flotation tests showed that sulphide minerals can be concentrated with rougher flotation conditions even at a relatively coarse grind. Knowing from previous tests that a finer grind is improving the gold recovery and that the leached residual gold is mainly fine-gold-trapped in sulphides, new flotation tests were conducted with the idea that the gold in the flotation tail could be leached at a coarser grind and the flotation concentrate could be reground to liberate the gold entrapped in sulphides and then leached. The energy saved by grinding the flotation feed to a coarser feed could be used to regrind the flotation concentrate.

Composite samples of the various zones and from the entire deposit were tested to compare the improvement in recovery with the new approach. The samples were first ground to a P_{80} of 130 microns and floated. The concentrate obtained represented around 10% of the initial mass and recovers an average of 73% of the gold content. The flotation tails were leached as is and the concentrates were reground to a P_{80} of 25 to 30 microns prior to cyanadation.

The major conclusions (based on an overall composite) indicated that:

- At a P_{80} of 130 microns, flotation concentrate contained 73.5% of the gold in 6.3% of the initial mass.
- Gold content in the flotation tails is 0.30 g/t Au and decrease to 0.06 g/t Au after leaching.
- Gold content in the concentrate is 11.7 g/t Au and is decreasing to 1.34 g/t Au after leaching of the reground concentrate at a P_{80} of 31 microns.
- The combined recoveries gave an average of 89.1% recovery, a significant improvement when compared to whole ore leaching at a P_{80} of 64 microns.

Flotation test work is still ongoing to develop the database and evaluate the gain from the different zones of the deposit (North, South, East and West). The flotation tests were based on a P_{80} of 130 microns for the flotation feed based on JKtech simulations indicating the possibility to produce a 130 microns product at 55,000 tpd using 2 ball mills in the secondary grinding circuit. The third ball mill could then be used to regrind the flotation concentrate to a finer grind prior to leach.

In conclusion, flotation has a high potential to improve recovery and possibly improve the economics of the project. The financial gains of increasing the recovery by approximately 3% would have to be evaluated taking into account the increased capital and operating cost. Additional test work is required to confirm the recovery gain at different grades from various areas of the deposit.

6.1.4.7 Carbon-in-Leach

A series of carbon-in-leach (CIL) tests were conducted on 4 lithologies (SPO, CPO, CGR and SGR) to study the effect of grinding size and carbon addition on gold extraction. The P_{80} considered were 55 to 91 microns. The material was ground to the required size and pulped with water to 40 % solids. The pH was adjusted with quicklime to 11.5 and the cyanide concentration was 0.5 g/l. The slurry was bottle leached for 48 hours. The results show that:

- The majority of the gold is extracted after 24 h.
- A small increase in gold recovery from 0.1% to 1.6 % was found when 10 g/l of activated carbon was added. This improvement was not statistically proven and so is not included in the global recovery estimation but was considered to compensate for the small loss of the carbon circuit not accounted for in the leach tests.
- As expected it was found in the laboratory tests that there was a significant increase in the NaCN consumption and lime consumption. This increase in the consumption does not represent the reality of industrial scale and is explained by the fresh (new) activated carbon used in the experimental tests that has a tendency to adsorb NaCN and lime mainly.

6.1.5 Heap Leach and Column Tests

The amenability of the low grade ore (head grade of around 0.6 g/t Au) to heap leach was studied in fourteen day bottle roll tests conducted at -19.05 mm (-3/4"), -12.7 mm (-1/2"), -9.53 mm (-3/8") and -6.35 mm (-1/4") crushed sizes. The material was crushed to the required size and pulped with water to 40% solids. The pH was adjusted with quicklime to 11.5 and the cyanide concentration was 0.5 g/l. The results show the following:

- The recoveries of gold range from 31.9% [19.05 mm (-3/4")] to 39.2% [-6.35 mm (-1/4")].

In a second test (conducted in duplicate), the amenability of the low grade ore to column leach was studied in sixty three day column tests conducted at -19.05 mm (-3/4") crush size. The material was crushed to the required size. The pH was adjusted with lime ranging 10.5 to 11.0 and the cyanide concentration was 0.5 g/l. The results show the following:

- The average gold recovery is 38.7%

- The calculated gold head was 0.59 g/t Au versus 0.6 g/t Au from size fraction analysis.

No further tests were conducted as the recoveries were too low to be of interest to the project.

6.1.6 Settling Testwork and Rheology Tests

A series of sedimentation tests (mostly static settling tests) were performed in order to determine the settling ore characteristics, flocculent dosage, slurry density achievable and finally, very important, the sizing of the thickener. These tests were performed on a material at a P_{80} of 55 microns and a pH between 10.5 and 11.5 (averaging 11.0). Four (4) thickener manufacturers (Delkor, FLSmidth, Outotech and Westech) and 2 flocculent suppliers (Ciba, SNF) conducted tests on representative samples. Additional settling and rheology tests were performed by Pipeline Systems, FLSmidth, Westech and SGS.

The results of the static settling tests showed an achieved density of up to 64% solids and the rake (compacted) density between 67% and 71% solids. A slurry density at 68% seems to represent the conservative typical limit allowing pumping with conventional centrifugal pumps.

The initial testwork for the sizing of the thickeners (leach and tailings) were targeting 50% solids at the leach and 64% solids at the tailings thickener. The value of 64% solids was then increased to 68%, later in time, to maximize water reclaiming volume and to improve thickened tailings discharge. Material proved to respond well to the increase %solids throughout the settling test program.

Several types of thickeners were evaluated by the various selected suppliers including hi-flow, hi-rate, hi-density, hi-compaction and paste thickeners, and this latest allowing the highest density on a continuous basis. The selected thickener for the leach process is a hi-rate (with dilution feed well) thickener that according to the teswork will allow the production of a 50% solids slurry minimizing flocculent dosage. A hi-density thickener was selected for the tailings thickening. Its increased depth will accommodate the production of a 68% solids slurry. Hi-compaction and paste thickeners would have given the possibility to produce even higher density slurry but are significantly more expensive for both the capital and operating costs in addition to the fact that pumping would become an issue.

The flocculent dosage of the testwork varied between 20 and 35 g/t and a 10% solids diluted feed gave the best settling results. The same flocculent was selected for the Rheology tests allowing a better comparison of the results. Also the same flocculent was used in the various tests to allow the comparison of the tests performed in the different labs.

6.1.7 Environmental Related Testwork

6.1.7.1 Acid Base Accounting (ABA)

ABA tests were conducted on material expected to be stockpiled (ore and waste) and on tails from leach tests which represent material that will be discharged in the tailing pond. Many samples of the same material (ore and waste) were tested: high and average sulphide content ore, and high, average and low grade ore, PO and GR fraction and +/- 75 microns fraction ore, high and average sulphide content waste, PO and GR fraction. For tailings material, average grade leached ore were tested so as +/- 45 microns fraction.

The ABA test results on course -1/4" crushed material (replicating ore and waste to be stockpiled) indicate that:

- Low grade ore Composite -75 micron and PO waste have high enough NP (Neutralizing Potential) vs. sulphide content to be considered not potentially acid generator. The PO lithology is mainly present in the northern part of the pit.

- Both GR waste (high and average sulphide) and composite waste (high and average sulphide) do not have enough NP to neutralize the sulphide content. The humidity cells test work will measure the rate of release for the NP vs. sulphide and the amount available on the long term. The high sulphide waste represents approximately only 2% of the total waste. In operation, the high sulphide and GR waste will not be mined separately and will be included in average waste composite when placed in the waste pile.

- The GR, PO and composite ore (± 75 microns), PO, GR and composite low grade ore, PO, GR and composite high sulphide ore (including -75 microns) and the PO high sulphide waste are classified as "uncertain" meaning that the NP is equivalent or slightly more than the sulphide content. The humidity cells test work will measure the rate of release for the NP vs. sulphide and the amount available on the long term.

- Two additional sets of independent samples covering the different zones of the deposit were tested for ABA. In the first set of samples the pit was split in 3 areas while for the second the pit was divided in the 4 zones. For both samples and in all cases, ABA tests gave an "uncertain status" for acid generation which is the same result as the previous ABA testwork done on a composite sample.

The ABA test results on leached material showed an "uncertain" status and humidity cell tests are ongoing to evaluate the behaving on the long term.

6.1.7.2 Net Acid Generation (NAG)

NAG tests were conducted on the same series of samples as the ABA. Tests gave the following results:

- The high sulphide waste (GR and composite samples) did show acid production when exposed to hydrogen peroxide. High sulphide waste represents approximately 2% only of the total waste so is less of a concern.

- The fraction +75 microns did show a small amount of acid produced in contact with hydrogen peroxide. The material is not expected to be separated by size fraction and will be part of the total residues.

- The high sulphide ore sample showed a pH higher than 8.0 (no acid produced in presence of hydrogen peroxide). The high sulphide ore represents a small portion of the volume of the total ore.

- All other samples showed a pH higher than 9.5 (no acid produced in contact with hydrogen peroxide).

- Two additional sets of independent samples covering the different zones of the deposit were tested for NAG. In the first set of samples the pit was split in 3 areas while for the second the pit was divided in the 4 zones. The NAG test performed so far on all composite samples did not show any acid generation in contact with hydrogen peroxide showing pH higher than 10.

The NAG test done on leach residues of average grade samples showed pH higher than 10.0 (no acid was produced in the presence of hydrogen peroxide).

6.1.7.3 Humidity Cells

Two humidity cells were started in November 2007 on leach residues and a new set of Cells were started in March 2008 including: Waste material (high and average sulphide content), ore (average and low gold content), high sulphide ore, and leach tail residue. The fraction -75 microns on low grade, average grade and high sulphide ore were also tested in the humidity cells started last spring.

After 25 weeks of running, the humidity cell results for -1/4” crushed and -75 microns unleached material are indicating that:

- All cells are running at neutral pH.
- The high sulphide ore shows a bulk NP and sulphide content depleting at a similar rate and keep producing a solution at neutral pH.
- The high sulphide and average waste composite shows a bulk NP depleting at a slightly faster rate than sulphide content. At this point, both cells are producing a solution at neutral pH.
- All other cells, including average and low grade ore (also including the -75 microns fraction) are producing a solution at neutral pH and are showing a bulk NP depleting at a lower rate than sulphide content suggesting that acid will not be produced in the long term.
- For all materials tested, with neutral pH values being maintained in the solutions of all cells running, there are no indications of metals being dissolved in the solution.

After 25 and 40 weeks of humidity cells running on the tail residue:

- Both set of samples, the first leach tails with P_{80} of 75 microns (November 2007) and the second leach tails with P_{80} = 60 microns (March 2008), keep producing a solution at a neutral pH.
- In both cases, the bulk NP is depleting at a slightly faster rate than the sulphide content. Humidity cell tests have to keep running to verify if all sulphide content will be available and if other neutralizing agents will be sufficient in case the carbonate NP get exhausted.

- For both materials tested, with neutral pH values being maintained in the solutions of all cells running, there are no indications of metals being dissolved.

6.1.7.4 Lixiviates Testwork

Samples of ore and waste composite materials representing the whole deposit were submitted for lixiviate testing.

Toxicity Characteristics Leaching Procedure (TCLP) was conducted on High Sulphide ore and HS waste, average waste and GR waste, average ore and low grade ore, average grade and low grade leach tails, average grade and low grade leach tails plus and minus 53 microns. The procedure followed was the MA.100-Lix.com.1.0 from the ‘’Centre d’expertise en analyse environnementale du Québec’’ (CEAQ).

- None of the ore, waste or leach tail materials shows all element contents below the back ground level and cannot be considered as a material of low risk.

- The lixiviate of all material shows at least one element exceeding the criteria Directive 019 and have to be considered as lixiviable.

- On the other hand, all of the materials tested showed lixiviate well below all limits of high risk criteria of Directive 019. None of the material tested is considered at high risk.

- A different set of samples representing 3 areas of the deposit were also tested for TCLP. The results of those tests showed very similar results when compared to the composite pit sample tested previously. This composite sample is being considered as leachable (higher than the low risk level) but is well below the high risk limit of the Directive 019.

6.1.8 Grinding Media and Reagent Consumption

Grinding media and reagent consumption rates for the present study were extrapolated from the interpretation of the results of the various testwork data and from experience for the treatment of similar ores.

6.1.8.1 Grinding Media

SAG Mill Grinding Media

Different quality of grinding media from different suppliers is available to be used as primary SAG grinding media for the mining industry. A high quality 5.25 inch diameter steel balls will be used to start the milling operations. Total consumption is estimated at 0.45 kg/t based on the operation of the SAG mill with approximately 15% of steel charge. The delivery, in bulk, will be by truck of approximately 34 tonnes per shipment. The steel balls will be dumped in a concrete bin containing over 15 days of capacity at 400 tonnes.

Ball Mill (secondary : 2 mills) Grinding Media

Different quality of grinding media is also available for the 2 ball mills of the secondary grinding circuit. A high quality 2.5 or 2.0 inch diameter forged steel balls will be used for that grinding circuit. Total consumption of 0.5 kg/t was estimated based on the operation of 2 parallel ball mills with approximately 28% of steel charge which represents normal overflow from the trunnion (steel charge flush trunnion). The delivery in bulk will be by truck at approximately 34 tonnes per shipment. The grinding media will be dumped in a concrete bin containing over 15 days of capacity at 450 tonnes.

Ball Mill (tertiary : 1 mill) Grinding Media

Different quality of grinding media but more so different type of grinding media is available for the ball mill of the tertiary grinding circuit. An economical good quality 1 inch diameter cast slugs will be used for the final grinding circuit. Total consumption of 0.32 kg/t was estimated based on the operation of the ball mill with approximately 35% of steel charge which calls for a steel grate overflow ball mill type (steel charge over and above the trunnion). A discharge steel grate frame will be developed in order to maintain the steel charge inside the mill. The delivery in bulk will be by truck at approximately 34 tonnes per shipment. The grinding media will be dumped in a concrete bin containing over 15 days of capacity at 275 tonnes.

6.1.8.2 Leaching and Detoxification Reagents

Flocculent (solid in bag)

The selected flocculent is a polymer anionic water-soluble standard and typically in use at other mining operations. Total consumption is estimated at 0.05 kg/t based on two thickener applications. The leach thickener with 0.02 kg/t will allow maintaining an underflow density of 50% required to feed the leach circuit. The tailings thickener consumption at 0.03 kg/t is necessary to obtain a high underflow density of 68% solids. The concept of thickened tailings (Section 7) is to allow cone shape discharging at the tailings pond minimizing at the same time the water management requirements. Flocculent will be delivered by truck of 24 bags of 700 kg so approximately 17 tonnes per shipment.

Quicklime (solid in bulk)

Quicklime will be used as pH modifier in the process. The total consumption of 0.58 kg/t was based on leach testwork for the milling operations at 0.49 kg/t and 0.09 kg/t at the detox plant. The delivery in bulk will be by truck in approximately 33 tonnes by shipment and discharged in a 300 tonnes outside silo.

Sodium Cyanide (solution at 30%)

Sodium cyanide will be delivered as a solution at 30% from the supplier distribution center. The total consumption of 0.20 kg/t was partially based on leach test results and experience in treating relatively fine gold ore with minimum consuming impurities such as copper sulphide, pyrite and the like. The delivery is in bulk by truck of approximately 30 tonnes per shipment discharged into a reservoir containing 3 days retention.

Caustic Soda (solution at 50%)

Caustic soda total consumption of 0.05 kg/t was estimated based on the usage of a solution at 50% and as per standard application in the gold mining industry (stripping circuit). The delivery in bulk will be by truck in approximately 35 tonnes shipments.

Anti-Scalant (liquid)

The selected anti-scalant (scale-guard) was chosen for its ability to resist heat and different water qualities. Anti-scalant will be used in the stripping circuit, to keep the carbon clean, and in the process water distribution system, to minimize maintenance of the piping system. A total consumption of 0.006 kg/t was estimated, based on the assumption to partially replace nitric acid in the stripping circuit. The delivery in bulk will be by truck in approximately 20 tonnes shipments.

Activated Carbon (solid in bag)

Natural coconut shell type activated carbon (dimension 6 mesh x 12 mesh) will be used in the adsorption circuit. The total estimated consumption is 0.03 kg/t, based on operation standards and the utilization of the carbon-in-pulp pump cell carousels circuit minimizing carbon transfer thus loss in fines. The delivery will be by truck of 24 bags of 500 kg so approximately 12 tonnes per shipment.

Nitric Acid Liquid (liquid)

Nitric acid will be sporadically used in the stripping circuit to eliminate the scale in the stripping circuit but more so on the activated carbon. The total consumption of 0.01 kg/t is based on experience and takes into account the usage of an anti-scalant. The delivery in bulk will be by truck of approximately 33 tonnes per shipment.

Oxygen (plant)

The best option, considering the oxygen consumption, was evaluated to be the rental of an oxygen plant (VPSA or VSA) installed at site by the supplier. The plant will produce the required average oxygen requirements. A backup system (LOX) with liquid oxygen delivered in a storage tank will allow continuous oxygen feed during mechanical shut down or during the summer months when maximum consumption occurs (July to September). The average consumption at 0.20 m^3/t is highly variable depending of the season, summer being the high consumption months. The consumption rate was derived from experience gained from other operations in the region. The delivery in bulk will be by truck of approximately 20 000 m^3 of oxygen gas or 27 tonnes of liquid oxygen.

Sulfur Dioxide (liquid)

Sulfur dioxide will be used at the detox plant. The total consumption of 0.064 kg/t was based on the detox testwork to reduce the cyanide content in the tailings water to less than 20ppm. The delivery in bulk will be by truck of approximately 26 tonnes per shipment.

Copper Sulfate (solid in bags)

The total consumption of copper sulfate was estimated at 0.025 kg/t. This was based on the testwork results for the operation of the Detox plant. The delivery will be by truck of 24 bags of 1 000 kg so a total of 24 tonnes per shipment.

Hydrogen Peroxide Solution (liquid)

Hydrogen peroxide will be use as a 50% solution during winter and 70% in the summer time. Total consumption of 0.10 kg/t was based on detox testwork program. The delivery in bulk will be by truck of approximately 20 tonnes per shipment.

6.1.9 Conclusions

The Canadian Malartic ore is composed of four main lithologies that are CPO, SPO, CGR and SGR spread throughout the deposit in an average ratio of 10%, 20%, 28% and 42%. The deposit was studied (metallurgical testwork) along the 3 axes being East-West, North-South and by depth. The main parameters studied were the variability of hardness and abrasion, reagents consumptions, gold and silver recoveries and the environmental characterization of both the ore and waste material.

The Canadian Malartic ore hardness has been evaluated, using the Drop Weight Test protocol, and described as moderately hard to hard when compared to the well known JKtech database. The hardness was found to be fairly constant throughout the deposit. Based on those hardness, abrasion and work index results, several simulations were performed using JKSimMet software in order to size and confirm the performance of the complete grinding circuit. It was found that the current selected primary SAG mill with 2 secondary grinding BM's and 1 tertiary grinding BM will allow the 55,000 tpd rate producing material at a P_{80} of 64 microns size required to maintain good metallurgical results.

The deposit was split into 4 zones (west, north, east and south) based on similar metallurgical behaviour. Recovery curves (recovery vs. head grade) were developed based on the numerous metallurgical test results. The best metallurgical recoveries were obtained in the North-West zone of the deposit where the grade and the PO lithology ratio are higher than the average of the deposit. The lowest results were obtained from the southern zone where grade is lower but more so the PO lithology ratio very low. Three recovery curves were developed for the gold recovery: west including deep north; east including shallow north and south. Two 2 curves were developed for the silver recovery: west-north-east as one group and south. Applying those equations for both the gold and silver recoveries to the various head grades and location for each block of the mining block model and for the complete open pit shell, the average recovery was calculated at an average of 85.9% for the gold and 69.3% for the silver. It is expected to pour a similar quantity of silver as compared to gold, this based on a lower average recovery but higher grade. The average head grade silver was found to be at 1.68 g/t Ag.

Grinding media usage was derived from steel consumption comparison with similar size milling operations and considering the various hardness and abrasion index test results. For the SAG mill a 0.45 kg/t of 5 ¼'' forged balls was estimated. For the two secondary grinding ball mills, a 0.50 kg/t of 2" forged balls was assumed. The tertiary grinding one ball mill consumption was evaluated at 0.32 kg/t of 1" cast slugs. The selection of the grinding media suppliers will be based on past experience of the production of good quality grinding media and the capacity to supply the project. Grinding media will represent a significant portion of the unit based mill operating cost and at the same time could limit production rate if not under control; thus will require a very close follow up.

The reagent consumption of Canadian Malartic ore is relatively low with an expected cyanide and lime consumptions of 0.20 kg/t and 0.58 kg/t respectively. Oxygen demand is estimated to be very low and no lead nitrate addition is required. The cleanness may explain those low figures.

Gold deportment and diagnostic leach demonstrated that the residual gold, after the leach process, is encapsulated mainly in pyrite. The significant proportion of the gold remaining in the tailings after the leach process was characterized as very fine grained gold. It was demonstrated that due to the small grain size, gravimetric processes are inefficient. The grind of the leach feed is the most important parameter observed especially for the gold present (encapsulated) in the sulphide. The finer the grind the higher the recovery.

The sulphide mineral, mainly pyrite, of the Canadian Malartic ore is easily "floatable" and the tests showed that the majority of the sulphide can be concentrated in less than 10% of the mass. Those flotation tests followed by leaching of both flotation concentrate and tails demonstrated that a finer grinding of the sulphide fraction was sufficient to liberate most of the gold. It was proven that the gold in the sulphide fraction needs a finer grind than the gold included in the non-sulphide fraction. Further testwork is ongoing as this concept showed significant potential for improving the economics of the project.

Thickened tailings technique was chosen as the deposition method for the tailings in order to maximize water recycling thus improving the water balance and minimize environmental impacts by reducing the footprint; even if this technique is known to be of higher cost. Several settling and rheology testing were performed to select and size the proper thickeners for both the leach and tailings sector and, to confirm the pumping of the 68% solids slurry at the tailings.

The CombinOx® technology (SO_2/Air/H_2O_2) process was selected for tailings detoxification. CombinOx® process gives the best combination for process efficiency, capital and operating costs. It is also the most flexible process in term of being able to handle different cyanide level slurry.

The Canadian Malartic ore and waste selected materials tested presented what is called to be an “uncertain potential” for acid generation, based on ABA and NAG static test work. Those results called for the next stage of testing which is the kinetic humidity cell testing program. The 9 different materials tested in the humidity cells (kinetic test work) still currently ongoing are to this date behaving as non acid generating material after more than 30 weeks. Further results are expected soon with final conclusion no later than after the maximal period of 56 weeks.

6.2 Process Plant Design

The testwork program initially started with a series of scoping tests to determine the behavior of the minerals in various process configurations. Numerous tests were carried out to determine the response to a flowsheet based on whole ore leach versus a combination of flotation/regrind and whole ore leach. Various combinations of degree of grind, flotation, regrind and leaching of intermediate and whole ore leach products were tested. The conclusion of all of these tests was that the best flowsheet to start the project would be based on whole ore leach. The whole ore leach flowsheet is the simplest, lower risk flowsheet and has been tested to a high degree of confidence.

Due to the predominant mineralogy where a significant amount of fine gold is encapsulated in sulphides (principally pyrite) the route of flotation was considered, mainly ultrafine regrind of sulphides concentrate followed by a whole ore leach showed promise of improved recoveries. The benefits of this route could not be proven to the level required for the feasibility study. It was then decided to initiate the project with a whole ore leaching flowsheet but make some provisions in the design such that an intermediate step of sulphide flotation and sulphide regrind could be introduced in the future without major operations interruption. These modifications would be initiated only if proven to be economically sustainable on their own.

Based on testwork carried out at SGS and URSTM and at other agencies, equipment manufacturers and experience on similar projects a flowsheet for the whole ore leaching route has been developed. This flowsheet reflects the results of testwork carried out to date and it forms the basis for the plant design and mill costs developed in this study. A summary level flowsheet is shown in Figure 6.12. METSIM software was used to balance the mass and water flows throughout the plant. This information was used to size equipment in various unit operations throughout the plant.

6.2.1 Plant Design Criteria

The process plant design criteria, for the whole ore leach flowsheet is developed based on the following sources of information and analysis:

- Testwork as reported in Section 6.1 of this report
- The mill production plan as reported in Section 6.1 of this report
- Information published from public sources
- From Osisko personnel having experience with similar ore types in the Abitibi region
- From previous projects with similar unit process operations
- From equipment manufacturers' recommendations.

Table 6.9 lists a summary of the principal design criteria established for the project. It is important to mention that the plant design criteria include safety factors when compared to the results of the testwork in order to accommodate peak operating conditions in the plant operation. The process design criteria are based on a processing plant of 55,000 tpd capacity based on a plant design utilization of 92%. The basis for plant design assumed a head grade of 1.2 g/t Au and a gold recovery of 86%. If the plant were to operate at this recovery and head grade as an annual average, the plant would produce approximately 665,000 oz/y Au. Over the period of one year, due to the fluctuations in head grade, recovery and tonnage rate, the process plant will have an average head grade of 1.07, an average recovery of 85.9% and produce an average of 590,975 oz/y Au.

Table 6.9: Summary of the Principal Design Criteria

Description	Units	Average Value
Nominal annual throughput	t/y	20,075,000
Process Plant Utilization	%	92.0
Crusher Plant Utilization	%	70.0
Nominal Daily Throughput	t/d	55,000
Average Hourly Crusher Throughput	t/h	3,275
Average Hourly Mill Throughput	t/h	2,491
Gold Content in Ore	g/t Au	1.07
Silver Content in Ore	g/t Ag	1.69
Grind P_{80}	microns	64
Average leach time	h	28
Gold Recovery	%	85.9
Silver Recovery	%	69.3
Annual Gold Production	oz/y	590,975
Annual Silver Production	oz/y	753,000
Carbon Loading	g/t Au	2,520
Weight of Carbon Stripped/day	t/d	20

6.2.2 Process Description Overview

Run of mine ore is transported to the gyratory crusher in 227 t mine haul trucks and each truck dumps in one of two dumping positions in the crusher feed pocket. The crushed ore feeds a conveyor for transportation of the ore to the covered stockpiled. The ore is reclaimed from the pile in an underground reclaim tunnel and is conveyed to feed the primary grinding SAG mill in the concentrator. The SAG mill is in a closed circuit with scalping screens and a pebble crusher. The SAG circuit product is fed to the two secondary grinding ball mills which feed the one tertiary grinding ball mill to produce a final product size suitable for feeding the leach circuit. Each of the two secondary ball mills are close-circuited with one cluster of hydro-cyclones while the tertiary grinding ball mill requires two clusters of hydro-cyclones due to a higher slurry volume to handle.

The slurry is brought to a high pH of 11 with lime, added to the SAG mill feed before adding cyanide (NaCN) to the circuit. Cyanide is added to the grinding circuit to start the leaching process of gold from the ore to the solution phase.

The ground slurry passes through linear screens, before the thickener, to screen out any organic material and any other tramp material that has come into the mill with the ore. The slurry is then thickened to about 50% solids before being fed to the leach tank circuit.

The leach tanks are located outside and consist of four series of five tanks with agitators. Oxygen is added to raise the oxygen level of the solution phase, in order to increase the leaching kinetics. From the leach tanks the slurry flows by gravity to two parallel sets of CIP pump cell carousel systems where activated carbon is added to each tank in order to adsorb the gold from the slurry liquid phase in a counter-current flow arrangement. In contrast to a conventional CIP circuit design the carbon is stationary (stays in each tank) and the slurry is transferred from tank to tank via the inter-stage pump screens. This technology has proven to be more efficient in carbon loading (higher gold loading of carbon) than the conventional CIP design. It also reduces the generation of carbon fines thereby reducing a fraction of the fine carbon losses to tailings. The loaded carbon is pumped from the first stage in the carousel circuit to a loaded carbon screen where the loaded carbon is separated from the slurry. The loaded carbon transfers into the stripping vessels by gravity.

Caustic solution is pumped through the column of carbon in the pressurized stripping vessel to elevate the pH. Cyanide can also be added to improve stripping efficiency. That solution is heated to about 140^o celcius and then is passed through the pressurized stripping vessel, stripping the gold from the loaded carbon back into the solution. The solution, now loaded with gold (pregnant solution), is sent to the electrowinning (EW) circuit where gold in the form of a sludge is precipated onto stainless steel cathodes. The gold sludge is pressure washed from the cathodes to the bottom of the EW cells. The gold sludge precipitate is then filtered, dried and then sent to a refining furnace where the gold is poured into gold dore bars. The gold bars will contain a significant amount of silver as the silver in the ore leaches and is stripped along with the gold and eventually recovered in the EW cells.

The stripped carbon is transferred to the carbon reactivation kilns where it is reactivated by heating to about 650^o Celsius in a reducing atmosphere. The carbon is then re-used in the CIP circuit. Fresh carbon is regularly added to make up for attrition losses. The activated carbon is pumped to the empty tank in the CIP circuit to start a new tank in the carousel sequence which becomes the last tank in the series. Before being added to the last tank in the carousel series the carbon is screened to ensure that there are no fine particles of carbon introduced into the circuit.

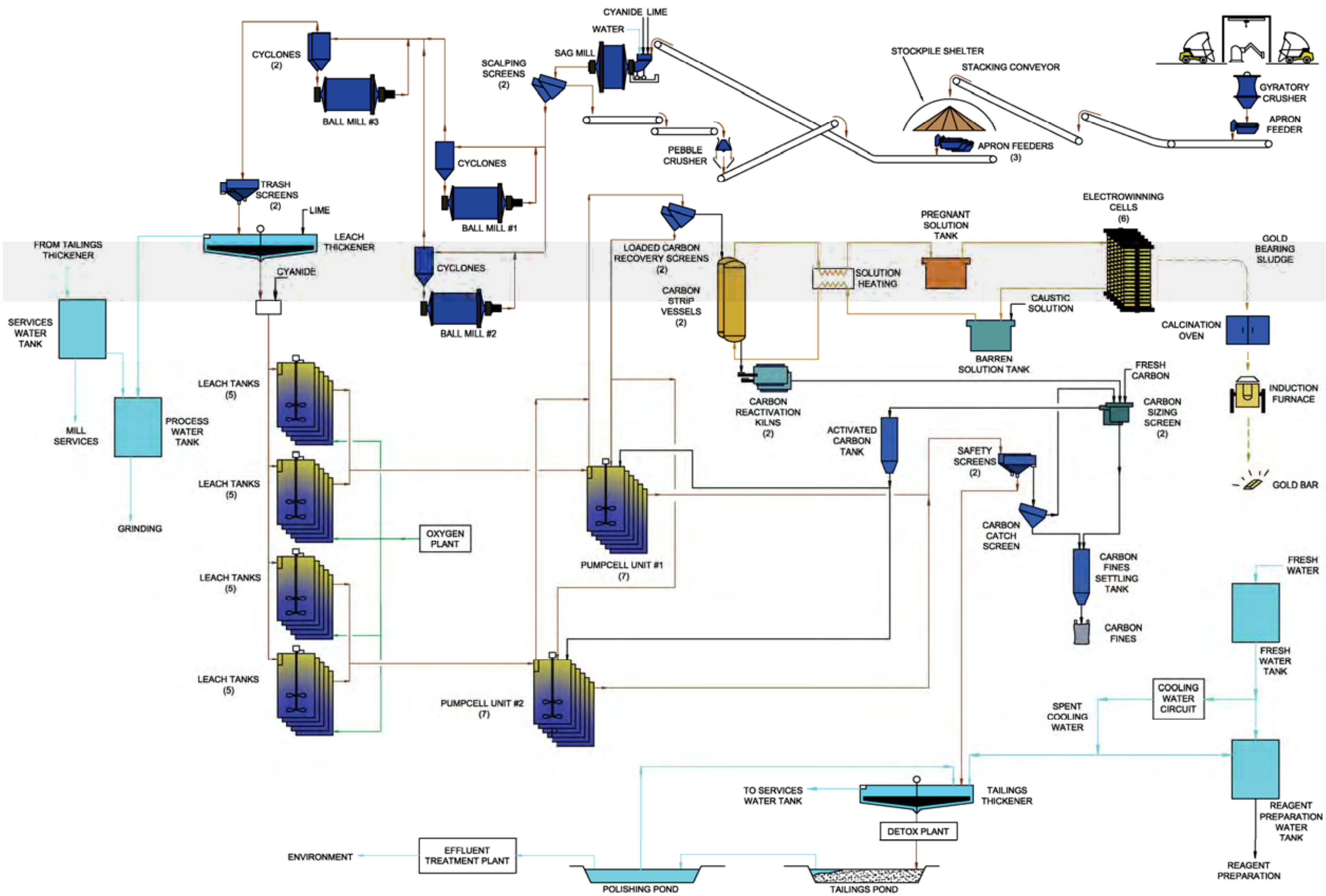
The slurry flowing from the last tank in the series in the carousels is barren in gold and is considered as final process tailings. This slurry is discharged over linear safety screens, as an insurance against coarse carbon losses from the circuit. Any oversize from the linear safety screens is fed to a carbon catch screen. The oversize from the carbon catch screen is returned to the circuit via the carbon sizing screen and the underflow is directed to the carbon settling tank to remove as much carbon as possible. This settled carbon material is collected in bags and sold to the smelter for its gold content.

The tailings slurry is thickened to approximately 68% solids by weight. This thickened tailings slurry is pumped to the detoxification plant where the cyanide content is significantly reduced to less than 20 ppm using the CombinOx® process. This process uses the combination of both the SO_2/air and peroxide water treatment concepts. The detoxified slurry is subsequently pumped to the tailings retention pond where most of the water remains contained in the solids and a small percentage drains out to be reclaimed back to the process.

Excess water from the tailings management facility will pass through a Peroxide based final effluent treatment plant prior to being discharged into the environment.

Sampling of the various process streams is carried out to be able to both quantify the plant performance on a shift and daily basis and to be able to control areas of the process on a continuous/semi-continuous basis. The location and type of sampling is described below in each sub-section of the process plant description.

Figure 6.12: Process Flow Diagram



6.2.3 Plant Facilities Description

6.2.3.1 Plant Location

The area of the proposed mill site is shown on Figure 6.13. The overall site including the mine open pit, waste dumps, process areas, tailings retention and water ponds occupy an area of approximately 2,400 hectares with existing ground surface elevation ranging between elevation 325 and 356 meters above mean sea level. The present site is covered to a large extent by existing tailings retention areas from previous mining/processing plant operations. Small drainage features flow from northwest to southeast across the site.

A geotechnical investigation for the proposed process facilities and mine garage was carried out in September of 2008 and a detailed report was issued. The subsurface conditions encountered in the boreholes and test pits generally consist of the following:

- The bedrock is a grey greywacke generally fresh, medium strong to strong and good to excellent quality with east-west sub-vertical bedding or foliation.

- Measurement of the groundwater elevation showed, in general, the static water level is a few meters below the ground surface at the mill location and close to surface at the crusher location.

The existing geographic high point is to be utilized for the ore storage stockpile, the process facilities and the mine garage and warehouse. These facilities will be constructed on solid rock at an elevation of 351 meters. The primary crusher is located closer to the exit of the mine pit with the crusher dump platform at elevation 345 meters. The bottom elevation of the crusher is founded on solid rock. Compacted fill from the mine pre-production will bring the elevation around the primary crusher to the final dump platform elevation.

The existing tailings impoundment areas from previous mining/processing operations will be used for deposition of the future tailings and for mine waste dumps.

The process facilities are described in detail below and an overview is shown in isometric presentation in Figure 6.14.

Figure 6.13: Area of the Proposed Mill

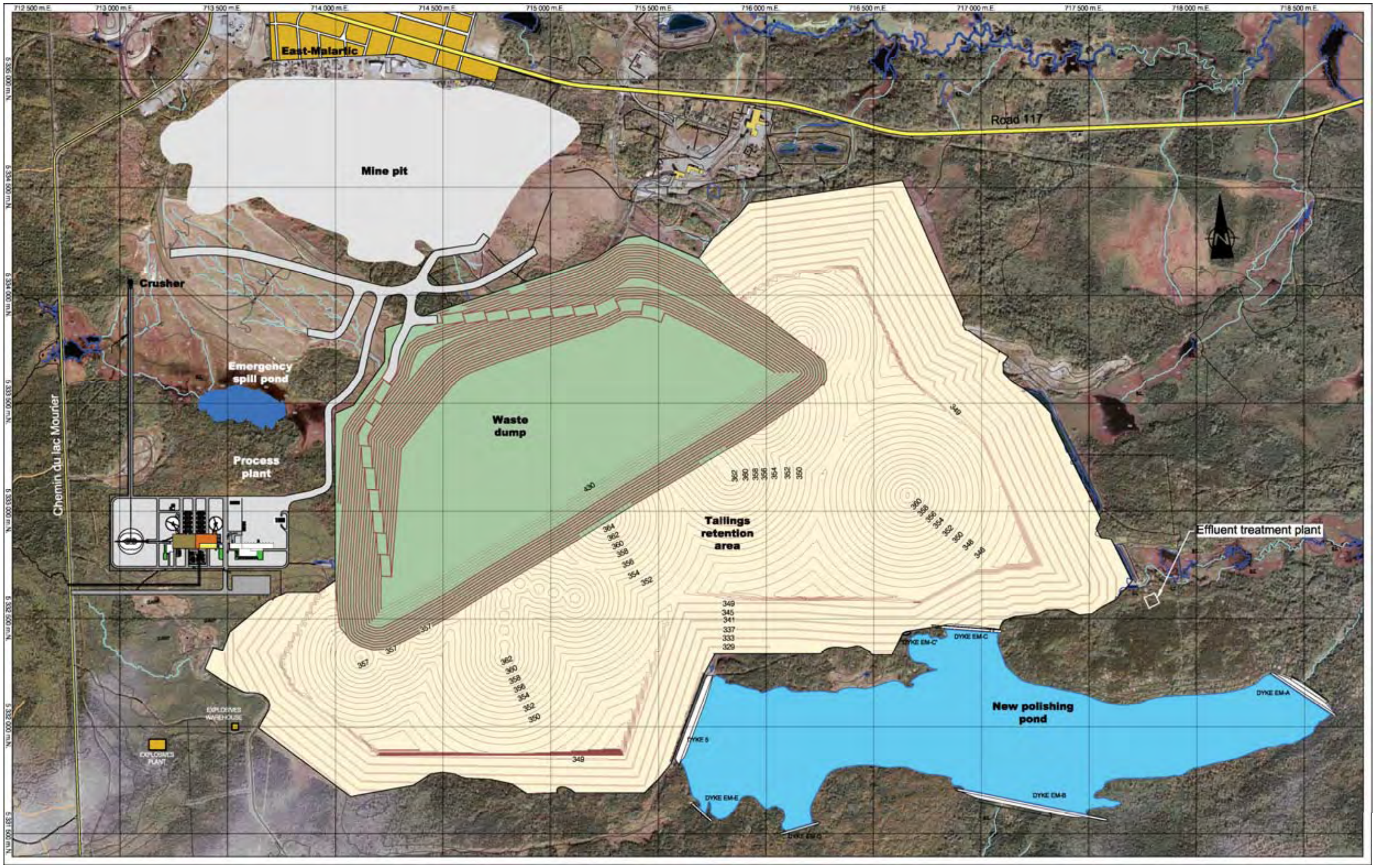


Figure 6.14 : Process Facilities Isometric View

6.2.3.2 Primary Crushing, Conveying and Stockpiling

The primary crusher is a 60" x 89" gyratory crusher with a 800 hp drive motor. The crusher is designed to crush at an average rate of 3,275 tph at a product size of P_{80} = 175mm. The dump pocket is designed with two dumping points for dumping 227 t mine haul trucks and has a live capacity of 440 t or the capacity of two trucks. The control room for the crusher station is located to allow the operator to look directly over the dump pocket and to be able to operate the hydraulic boom/rock breaker to manipulate the dumped ore in the pocket to assist the flow of run of mine ore into the crusher cavity. From time to time, the hydraulic rock breaker will be used to break oversized rocks in the dump pocket. An overhead bridge crane of 75 t capacity will be installed on the crusher area superstructure. This crane will serve to lift and place materials during crusher concave and mantle/mainshaft replacement as well as to transport materials up and down through the hatches inside the crusher building.

The crusher building will contain the dust collection system as well as the air make-up equipment to maintain a safe and clean working environment inside the crusher building. The building is designed with a surge pocket under the crusher with a live capacity of 400 t. A 2.1 m wide x 7.0 m long apron feeder extracts crushed ore from the surge pocket under the crusher to feed the crushed ore conveyor at an average rate of 3,275 tph. The apron feeder is equipped with a 150 hp variable speed drive which will control the loading on the crushed ore conveyor

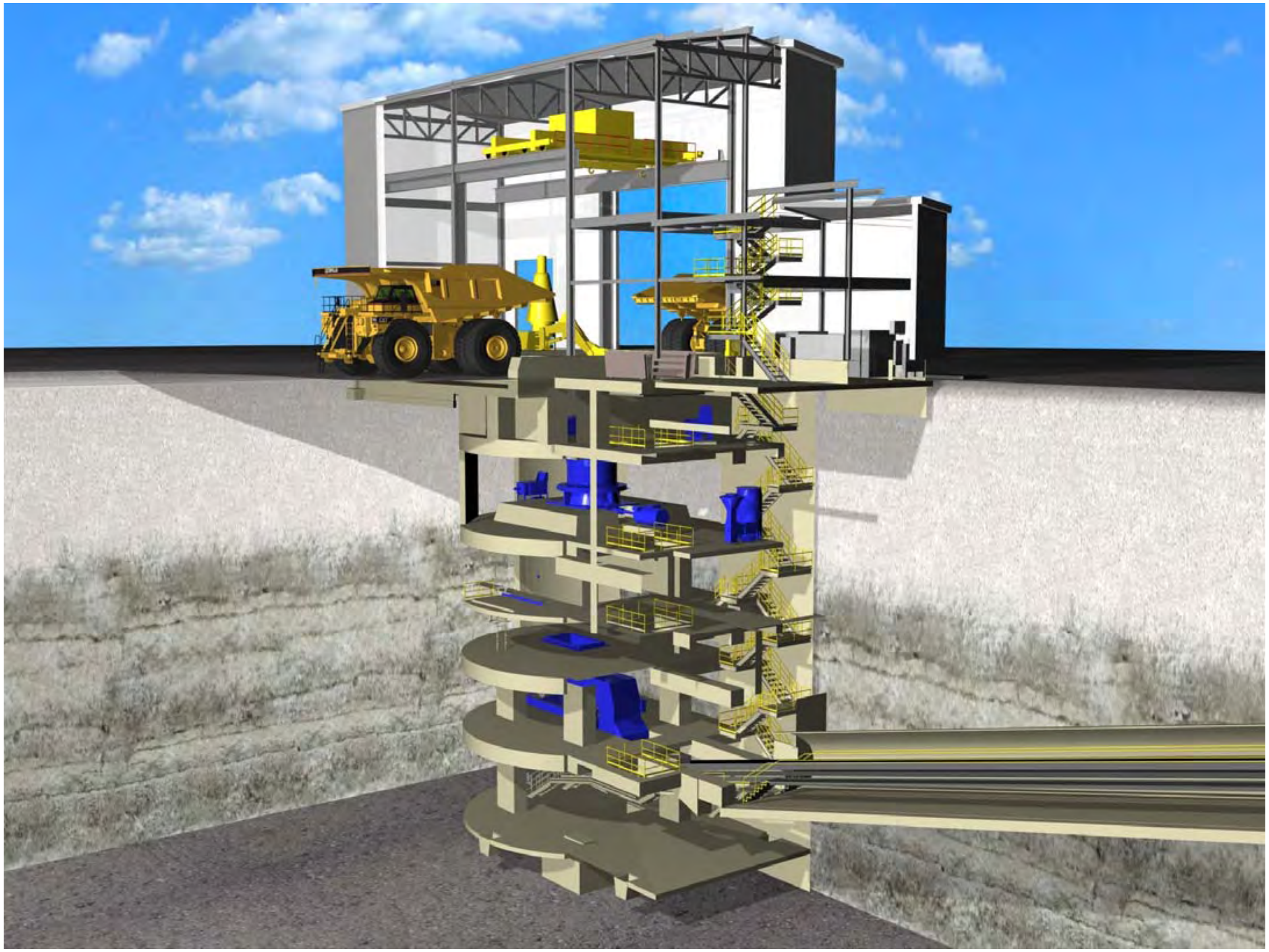
The crushed ore conveyor is approximately 953 m long and feeds the crushed ore stockpile stacking conveyor. The crushed ore conveyor has a 1372 mm wide belt with two drive pulleys, one driven by two (2) 400 hp motors and one drive pulley with a single 400 hp motor. The drives each consist of a motor, a load sharing fluid coupling and a gearbox. The stockpile stacking conveyor is 260 m long and feeds the 32 m high stockpile. The stacking conveyor has a 1372 mm wide belt with two drive pulleys, one driven by two (2) 400 hp motors and one drive pulley with a single 400 hp motor. Each drive consists of a motor, a load sharing fluid coupling and a gearbox. The drive and tail stations for the two conveyors are located outdoors. The electrical equipment for the drive station is enclosed in a climate controlled building located adjacent to the conveyor drives.

The covered stockpile has a live capacity of 28,000 t which represents approximately 12 hours of operation of the process plant. The total live plus dead storage capacity in the crushed ore stockpile is approximately 140,000 t which represents approximately 60 hours of process plant operation. This will allow the process plant to continue operating for the duration of a complete crusher concave/mantle relining.

The stockpile is covered for dust containment from dust that is generated when the crushed ore conveyor is discharging onto the stockpile. The dust containment enclosure is a space-frame type structure that is independent of the crushed ore stacking conveyor structure. The structure will have openings at the bottom to allow mobile equipment to enter inside.

Figure 6.15 illustrates the general arrangement of the primary crusher.

Figure 6.15: Primary Crusher Isometric View



6.2.3.3 Ore Reclaim and SAG Mill Feed Conveyor

Crushed ore will be reclaimed from the crushed ore stockpile by three apron feeders located in a reclaim tunnel under the stockpile. The apron feeders are each 2.1 m wide X 7.0 m long and extract ore from lined openings under the stockpile. Each apron feeder will have the full capacity to feed the SAG mill at an average rate of 2,491 tph. The apron feeders discharge onto the SAG mill feed conveyor. Each feeder is driven by a 150 hp variable speed drive. The variable speed of the apron feeders varies the SAG mill feed rate in response to the control loop demand.

The reclaim tunnel has an additional exit for emergency escape in the event of fire or other emergencies. The reclaim tunnel area includes a dry type dust collector to ensure that dust is collected from inside the apron feeder and mill feed conveyor loading chutes. This will ensure a safe and clean working environment inside the reclaim tunnel. The dust collector is sized for 12,000 cfm capacity and is equipped with a 150 hp fan located on the downstream side of the dust collector.

The mill feed conveyor is approximately 240 m long and conveys the ore from the stockpile reclaim apron feeders to the SAG mill feed chute. The conveyor has an 1829 mm wide belt with a head end pulley driven by a single 500 hp motor. The drive consists of a motor, soft start hydraulic coupling and gearbox. The conveyor has a nominal capacity of 3091 tph including the recirculating load (2491 tph new feed plus 600 tph circulating load). The discharge end of the SAG mill feed conveyor is equipped with a belt wash station to minimize the carry over of fines on the belt thus reducing the cleanup required around the mill feed area.

Sampling of the mill feed is not practical and no on-line sampling is done. From time to time, for the purpose of SAG mill performance modeling, the SAG feed conveyor can be stopped and the material on the conveyor belts manually sampled. The SAG mill feed conveyor and the recirculating load (pebble crusher) conveyor are both equipped with weigh scales.

6.2.3.4 Grinding Circuit

The overall grinding circuit consists of the SAG mill circuit and three ball mills with their respective equipment as described below. The ball mills are configured as a two stage circuit with two secondary grinding ball mills and one tertiary grinding ball mill.

The SAG mill circuit consists of a single 26,000 hp gearless drive SAG mill, 38 ft. diameter X 21 ft. long (EGL) , two 12 ft. x 24 ft. vibrating scalping screens (one operating and one standby), recirculating load conveyors and a 1,200 hp pebble crusher. The SAG mill is fitted with pebble discharge grates and allows a discharge of a top size in the order of 50 to 75 mm. The mill discharge chute directs slurry to one of two scalping screens, one operating and one standby.

The undersize material from the scalping screens, minus 12 mm material, discharges into a pump box which serves as a SAG mill discharge pump box as well as a discharge pump box for the two primary ball mills. The recirculating load conveyors receive the scalping screen oversize and transfer this material (pebbles) to feed the pebble crusher. This recirculating load circuit includes two self cleaning belt magnets to remove tramp material to protect the pebble crusher from tramp, metallic materials. It also includes a metal detector ahead of the pebble crusher which will actuate the crusher feed shuttle to divert feed from the pebble crusher to the discharge conveyor of the pebble crusher when metal is detected on the crusher feed belt. The crushed product from the pebble crusher (P_{80}<12.5 mm size material) is conveyed to the SAG mill feed conveyor.

The final product from the SAG circuit is the feed to the two secondary grinding ball mills and has a product size of P_{80} = 3.3 mm.

Each of the two secondary grinding ball mills is 16,000 HP, 24 ft. diameter X 36.5 ft. long (EGL) and each mill is close-circuited with cyclones to produce a secondary grinding ball mill product size P_{80} = 135 microns. Each mill is driven by two 8,000 HP synchronous motors, each through air clutches to drive a pinion connected to the mill bull gear. The primary ball mill cyclone overflows are the new feed to the single tertiary grinding ball mill. Each secondary grinding ball mill discharges into the SAG mill discharge pump box thus using a common discharge/cyclone feed pump box. There are three secondary grinding ball mill cyclone feed pumps connected to this pump box, two main operating pumps, one for each ball mill cyclone cluster feed, and one standby pump. The standby pump is valved to serve as a standby pump for either secondary grinding ball mill circuit. Each secondary grinding ball mill cyclone cluster feed pump is a 2200 HP, 30 X 26 horizontal slurry pump. The two main operating pumps, one per ball mill, are variable speed and the standby pump is a fixed speed pump. The main pumps are equipped with a variable speed fluid coupling drive. There is one cyclone cluster for each secondary grinding ball mill which consists of eleven, 33 inch diameter cyclones, 9 operating and 2 spare cyclones.

The single tertiary grinding ball mill is 16,000 HP, 24 ft. diameter X 36.5 ft. long (EGL) which is close-circuited with cyclones to produce a final product size P_{80} = 64 microns. The mill is driven by two 8,000 HP synchronous motors, each through air clutches to drive a pinion connected to the mill bull gear. The tertiary grinding ball mill cyclone overflow is the final product of the overall grinding circuit and this product is the feed to the leach circuit. The tertiary grinding ball mill circuit includes three horizontal slurry pumps, 1850 HP, 30 X 26 (two operating and one standby) to feed two cyclone clusters. The two running pumps are variable speed and the standby pump is fixed speed. There are two cyclone clusters for the single tertiary grinding ball mill circuit, each consisting of fifteen 26 inch diameter cyclones with 12 operating cyclones and 3 standby cyclones.

The SAG mill bay is equipped with a 110 t capacity double girder overhead bridge crane with a 20 t auxiliary hoist and the ball mills bay is equipped with a 40 t double girder overhead bridge crane with a 20 t auxiliary hoist. All of the mills and the pebble crusher are equipped with their individual bearing lube systems. The SAG mill feed bay which includes the pebble crusher is equipped with a 35 t capacity double girder overhead bridge crane (no auxiliary hoist). The cyclones are serviced with a 10 t single girder overhead bridge crane.

Three ball pits are required, two for the ball mills for two different ball diameters and one for the large grinding balls for the SAG mill. For replenishing the mill charge the grinding balls are loaded into a 15 t capacity ball bucket with a ball magnet suspended from an overhead crane. The ball bucket is transported by overhead crane to the mills and balls are transferred into the ball charging hoppers located at each mill feed end. Each ball pit has its own access door on the exterior building wall which is opened to provide access for delivery trucks to discharge balls into the ball pits.

The product from the grinding circuit (secondary and tertiary grinding ball mill cyclone overflows) is sampled with in-line samplers and the sample stream flows to an on-line particle size monitor. The information from the on-line particle size monitor is used to control the grind size of the product from the milling circuit (leach circuit feed).

The grinding circuit is shown in isometric view in Figure 6.16.

Figure 6.16: Grinding Circuit Isometric View

6.2.3.5 Leach Feed Thickener

The ground ore from the SAG/Ball mill circuit flows from the cyclone clusters by gravity to two linear trash screens for removal of organic materials, metal and any other miscellaneous tramp materials. The underflow from the two trash screens flows by gravity to the leach feed thickener. The leach feed thickener is a high rate thickener and will control the feed density to the leach circuit to approximately 50% solids (w/w).

The two linear trash screens are 32 m^2 in size and are equipped with a cloth with 700 microns (24 mesh) openings. The two screens are fed in parallel from the tertiary grinding ball mill cyclone overflows

The thickener is a 62 m diameter thickener with a constant slope bottom (9.5 deg. slope) and an auto-diluting feed well. The thickener underflow pumps are located outside the perimeter of the thickener with the suction lines from the thickener center cone buried under the thickener. There are three thickener underflow pumps, 500 HP, 14 X 12 each (two running and one standby). The main operating pumps are variable speed and the standby pump is fixed speed. The main operating pumps are equipped with variable speed fluid coupling drives. The pump speed is varied to be able to control the solids density of the feed to the leach circuit.

The thickener overflow flows by gravity to the process water tank dedicated to this thickener.

6.2.3.6 Leach Circuit and CIP Circuit

The two leach feed thickener underflow pumps feed two leach tank groups. There are a total of 20 agitated leach tanks arranged in four rows of 5 tanks located outside on the north side of the process plant building. The leach tanks are arranged in two groups of 10 tanks, each group made up of two adjacent rows of 5 leach tanks each. Each group of two rows of 5 tanks is fed with a header pipe from a ‘’T’’ split which is fed by one of the leach feed thickener underflow pumps. Two stage samplers are installed at each of the two leach feed lines. The primary samplers are thief-type samplers and the secondary samplers are vezin-type.

The leach circuit is designed to provide a leach residence time of approximately 28 hours. Each tank is equipped with a 200 hp double impeller agitator mounted on the superstructure on the top of each tank. Slurry flows by gravity from tank to tank in each of the four 5-tank series. The tanks are 18 m diameter and vary in height from 22 m to 19.2 m for each row of 5 tanks. Each tank can be bypassed by manually operated valves which will direct the flow through a bypass line traversing the tank. Each of the leach tanks is equipped with a gas sparger, china-hat style for injection of oxygen into the slurry to be able to maintain the oxygen level in the solution required for the process kinetics. The discharges of two adjacent rows of leach tanks (one tank group) are combined to feed a single carousel-type CIP circuit. There are two carousel-type CIP circuits each fed by one group of 10 leach tanks.

The leach tank agitators are accessed by a mobile crane of 200 t capacity. Access is provided for the mobile crane along each row of 5 tanks.

The CIP circuit is configured as two parallel carousel-type arrangements. Each tank is equipped with a pump cell/agitator unit. All of the tanks are at the same height. The carbon remains in each tank and the slurry is pumped in sequence from tank to tank using the pumping screen mounted on the agitator drive. Each CIP carousel group has 7 tanks arranged in a series flow arrangement. Each tank has approximately 330 m^3 of usable volume. Each tank contains 20 t of carbon. One tank per carousel arrangement is emptied approximately every two days and the slurry/carbon mixture is pumped to the loaded carbon recovery screen ahead of the stripping circuit. The loaded carbon recovery screen is a 20 m^2 linear screen. There is one loaded carbon recovery screen feed pump per CIP carousel unit. Each pump is a 75 HP, 8 x 6, horizontal slurry pump. The slurry from the loaded carbon recovery screen (screen undersize) returns to the carousel slurry feed launder by gravity and the loaded carbon is charged directly into the strip vessels from the loaded carbon screen. Loaded carbon screening is located directly above the strip vessels.

The barren slurry from the two CIP carousel units flows continuously by gravity to a single tailings transfer pump box where it is pumped to two linear safety screens. There are two tailings transfer pumps, one running and one standby. Each pump is a 900 HP, 26 x 22 horizontal slurry pump. The main pump has a variable speed fluid drive and the standby pump is fixed speed. The two linear safety screens are fed in parallel and are each 20 m2 in screening area with 700 microns (24 mesh) filter cloths. From the linear safety screen the tailings slurry flows by gravity to the tailings thickener where the slurry is thickened to approximately 68% solids before being pumped to the detoxification plant for disposal to the tailings retention area. Any oversize from the linear safety screens flows by gravity to the carbon catch screen and its underflow is discharged to the tailings carbon settling tank. The bottom of this tank is conical in shape and is discharged from time to time into permeable super sacs. The overflow from this tank flows through a screen basket and then by gravity to the tailings. When a super sac is full, the operator inspects the full sac and decides where to send the contents. The contents are bagged and stored for sale and transport to a smelter. The carbon catch screen oversize is returned to the circuit via the carbon sizing screen over the activated carbon storage tank. The leaching circuit is shown in isometric view in Figure 6.17.

Figure 6.17: Leaching Circuit Isometric View

6.2.3.7 Tailings Thickening

The tailings thickener is a 62 m diameter, high wall thickener with a constant slope bottom (9.5 deg. slope), an auto-diluting feed well and a high torque rake/rake drive system. An emergency escape route is provided for by a ladder which is located in a dry well inside the rake mechanism tower. The thickener underflow pumps are located in a pump room under the thickener center column. The pump room is accessed through a tunnel under the thickener which exits outside of the thickener tank wall. There are two thickener underflow pumps, 600 HP, 20 X 18 each (one running and one standby). Both pumps are variable speed and they are both equipped with variable speed fluid coupling drives. The pump speed is varied to be able to control the solids density of the feed to the detox plant.

In order to minimize the cyanide loss in the plant and hence minimize the detoxification cost, reclaim water is added to the tailings thickener feed well. The thickened tailings at 68% solids (w/w) is pumped to the detox plant for detoxification of the cyanide content of the solution in the slurry. The treatment of the slurry in the detox plant lowers the cyanide content of the tailings slurry down to 20 ppm of cyanide or less. The plant make-up water from the reclaim water pond will enter the process, to the maximum extent possible, through the tailings thickener feed. This flow arrangement minimizes the impact on the environment as a consequence of helping to reduce the cyanide content of the final slurry. In addition, by adding water to the tailings thickener feed, the treatment cost for tailings detox is minimized. The tailings thickener overflow discharges into a process water tank dedicated to this thickener and this water is re-used throughout the process.

6.2.3.8 Stripping Circuit

The slurry containing the loaded carbon from the CIL carousel tank is pumped to the loaded carbon recovery screens located above the stripping vessels. The screened (washed) carbon from the linear screen falls through a chute into one of the two 10 t strip vessels. The slurry from the CIP carousel circuit that was in the feed to the loaded carbon recovery screens flows through the screen deck, is collected in the screen undersize launder and flows back to the CIP carousel circuit feed launder by gravity.

Carbon stripping is accomplished with Zadra pressure stripping technology. The circuit consists of a barren solution tank, two strip vessels, a natural gas fired strip solution heating system and heat exchangers. The carbon stripping operation involves circulation, under pressure and elevated temperature (approx 140^{o} Celsius), of barren solution through a primary recovery heat exchanger, a final trim heat exchanger and then through the loaded carbon bed in the strip vessel. The elution takes place with the incoming solution at a temperature of 140ºC under pressure at a nominal flowrate of 1.5 bed volumes per hour. Pressure in the system is maintained at about 370 kPa by means of an automatically controlled pressure valve located in the eluate pipe downstream of the cooling heat exchanger. The pregnant solution is cooled to about 80ºC in the water cooling heat exchanger and then flows directly to the electrowinning cells located in the gold room.

After stripping, the barren carbon is cooled down by a water wash step in the strip vessel. Stripped carbon is then pumped from the strip vessel either to the carbon reactivation area or to the carbon sizing screen above the 24 t carbon holding tank. Fresh carbon is also added to the circuit via the carbon sizing screen, to replace the carbon lost as fine carbon that leaves the circuit either in the tailings slurry stream or the fine carbon that is filtered from the carbon transport water. The water used to transport carbon is collected in a transport water recovery tank and is re-used as transport water in a closed circuit. The carbon that settles out in this tank is pumped from time to time to a filter press that recovers the fine carbon. This carbon is then collected and sold to the smelter for its gold content. The fresh carbon that is added to the circuit is first scrubbed in an agitated tank to liberate fine carbon and then the carbon/water mixture is pumped to the carbon sizing screen above the 24 t carbon holding tank.

A good proportion of the stripped carbon is pumped from the strip vessel to the carbon reactivation circuit. When the carbon is not passed through the reactivation circuit, it is pumped directly to the sizing screen ahead of the carbon holding tank.

In the carbon reactivation circuit, the carbon is first screened and dewatered. The screen discharge is directed to a kiln feed hopper. Additional dewatering is accomplished by a static drain screen at the bottom of the feed hopper at the screw feeder. Two natural gas fired rotary reactivation kilns, each having a capacity of 10tpd, are provided for carbon reactivation. The heating system is a set of natural gas fired burners enclosed in a refractory chamber. The rotating kiln cylinder is provided with a variable speed drive for production turndown and an emergency drive to maintain cylinder rotation in the event of a power failure. A low differential pressure controlled exhaust fan removes reactivation products while maintaining close to atmospheric pressure inside the kiln cylinder. The activated carbon exits the kiln and falls by gravity into a quench tank. The quenched carbon is then transported to the sizing screen ahead of the 24 t capacity carbon holding tank. The sizing of the carbon is a crucial step in the carbon regeneration circuit. An efficient sizing of the carbon will avoid the hidden loss of the fine carbon reaching the CIP circuit, adsorbing gold and then escaping to the tailings. The carbon tank provides a holding capacity of 24 t and is fitted with a screened drain and screened overflow to prevent carbon from escaping in case of overflow of the tank.

This carbon that has been screened and stored in the 24 t carbon tank is ready to be pumped to the CIP carousel tank that will be emptied next in the sequence of the carousel cycle(approximately one 20 t transfer per day).

Carbon is always transferred using recessed impeller type pumps to minimize the generation of carbon fines.

A carbon transfer water system is provided to conserve water and to recover the fine carbon particles which contain residual gold. A central tank is used to collect all transport water used in carbon movement and the water that drains from vessels and screening (stripping vessels, carbon dewatering ahead of regeneration kilns, water from sizing screen and surge tank, etc.) after each cycle of use. The tank is designed to allow the fine carbon to settle. This water, with the addition of make-up water is used only for the transporting of carbon. This carbon transport water is distributed with two water pumps, one operating and one standby. Periodically the carbon fines are pumped from this central tank, filtered and then put in super sacs for further drainage and transport to the smelter. Credit is given for any residual gold and silver values in the fine carbon.

6.2.3.9 Gold Electrowinning and Refining

The electrowinning (EW) cells are fed directly from the strip vessels. A two way distributor which also serves as a flash tank ensures an even distribution of pregnant solution to each of the two (2) lines of EW cells. There are 6 electrowinning sludge cells each fed at a rate of 4.5 m^3/h. Each EW cell is 3.5 m^3. The cells are fabricated from stainless steel lined with polypropylene and each cell is equipped with a sludge pump. The eluate, after electrowinning, flows to a pump box from where it is pumped to the barren solution storage tank outside of the gold room. A bleed to control solution quality and the addition of reagent is made prior to each stripping cycle. Approximately once per week the cells are cleaned to recover gold in the form of sludge. The cathodes are washed, in place, with a high pressure washer to dislodge the maximum amount of sludge from the cathodes. The sludge is transferred by sludge pumps to a filter feed pump box and pumped to a pressure filter. The filter cake is then transferred to pans and placed in the drying oven prior to mixing with a prescribed flux to prepare for charging the refining furnace.

For refining, an 125 kW, 2.2 ft^3 induction furnace will be used, complete with power pack, hydraulic tilting, power ramping control and fume collection. A wet scrubber will collect the fumes from the furnace. Refined gold is poured into a series of molds and the slag produced is poured into slag molds. The slag, after cooling, is broken up and the high grade is re-poured and the low grade is recycled via the grinding circuit.

Accessories for the gold room include storage racks for the refining furnace additives, bullion scale, charge weigh scale, gold molds and mold stand, slag molds, drying oven for filtered sludge, filter feed pump box, filter feed pump, sludge pressure filter, wire samplers and other miscellaneous items. Two (2) portable mold stands will be supplied with five (5) molds each.

A bridge crane of 5 t capacity will be provided for maintenance over the electrowinning cells and the refining furnace areas.

The entire electrowinning and refining functions are enclosed in a secure area with limited access. A secure room with vault door is provided to store dried electrowinning cell sludge and gold bars.

The gold recovery circuit including CIP, stripping, carbon regeneration and gold room is shown in Figure 6.18.

Figure 6.18: Gold Recovery Isometric View

6.2.3.10 Tailings Detox Plant and Tailings Pumping and Effluent Treatment Plant

Tailings Detox Plant

Cyanide destruction is achieved using the CombinOx® Process Cyanide Destruction Method. The cyanide destruction plant consists of reactor tanks, reagent mix and hold tanks and addition systems, air blower and agitators. The plant is designed to handle 55,000 t/day. Pumping and dosing equipment are located inside the process plant building, while the agitated reactor tanks are located outside.

The process uses both sulfur dioxide (in any of the available product forms) and peroxygen chemicals as reagents.

Soluble copper, from copper sulfate provides the catalyst to the detox reaction; dissociated metals are precipitated as hydroxides and strong cyanide complexes are precipitated as insoluble salts, predominantly in the presence of copper.

Potential benefits of the CombinOx® process when compared to the single SO_2/Air process include: lower capital cost, lower operating cost, flexibility towards changes in achieving a broader range of cyanide target concentrations and the ability to handle cyanide loading fluctuations in the feed to the cyanide detoxification circuit.

A preliminary estimation of the reagent consumption based on laboratory testwork is:

- SO2 (Liquid) = 1,275 t/year
- H2O2 (100%) = 2,032 t/year
- Ca(OH)2 = 1,855 t/year and
- CuSO4.5H2O = 510 t/year.

This estimation was based on laboratory testwork at Cyplus using liquid SO_2 and a design tonnage of 55,000 t/day at 68 wt/wt% and CN_{WAD} feed concentration of 100 mg/l.

The system consists of 2 CombinOx® reactors with agitators, a Copper Sulphate addition system, a Hydrogen Peroxide addition system and a Liquid SO_2 delivery system. These systems include pumps, on-line analysis and control systems and other miscellaneous equipment.

Tailings Pumping

The treated tailings slurry is transferred to the tailings pump box and is pumped at approximately 68% solids to the tailings retention area where the pumped slurry stacks and drains. Two pump trains are provided for tailings pumping, one operating and one standby, each consist of two pumps in series. Both pumps are horizontal slurry pumps, 1000 hp, 20 x 18. The first pump is fixed speed and the second pump is equipped with a variable speed fluid drive.

Effluent Treatment Plant

As previously mentioned and because of the usage of the thickened tailings technology the excess water from the tailings deposition area will be minimized. A very small quantity of water will drain out of the thickened slurry. That excess water will naturally mix with the various site water inflows such as precipitation and also mine dewatering water into the new polishing pond. This low cyanide content water will be re-used in the process as reclaim water. A pumping system will pump the reclaim water back to the process plant. The maximizing of the reclaim water re-use forms the basis of the water management plan of the project. The water management balance is presented in another section if this study but it is expected that the water balance is close to being in equilibrium thus with a minimum quantity to discharge into the environment. The excess water would though require treatment for removing some metals, mainly copper and to reduce cyanide to an acceptable level prior to releasing the water into the environment.

The effluent treatment plant is designed for a flow of 650 m^3/h and is split in two phases, each with two 7.5m diameter x 7.5m high agitated reaction tanks. During the first treatment stage, hydrogen peroxide and copper sulfate are added to oxidize and destroy remaining traces of cyanide complexes. During the second treatment stage, ferric sulfate is added as a coagulent in order to precipitate metals in solution. Then, an anionic flocculant can be added to the treated water to promote settling in the subsequent thickener that is in fact a clarifier. When discharge criteria are satisfied, clean water from the clarifier overflow is discharged back into the environment. The clarifier is periodically cleaned and the recovered solids are pumped back to the tailings impoundment.

6.2.3.11 Plant Water Systems

The plant water systems consist of the process water system which is supplied principally from the plant thickener overflows, the fresh water system which is supplied from the old underground mine dewatering system, reagent preparation water system, the gland water distribution system and the reclaim water from the tailings retention pond area (new polishing pond). The different water systems are used in the plant as described in the following sections:

Process Water

Process water supplied to the process is made up of two systems, the tailings process water system and the leach process water system. The two process water systems each have their separate tanks and pumping systems but are connected to the same process distribution headers. The source of water for the leach process water tank comes from the leach feed thickener overflow. The tailings process water tank receives its water from the tailings thickener overflow. The common process water header provides water principally to the SAG and Ball mill circuits. The principal users of process water are the SAG mill feed, ball mill cyclones feed dilution, scalping screens spray water, floor washdown and other miscellaneous users.

Each of the process water tanks has a retention time of approximately 10 to 15 minutes under average flow conditions. The leach process water flow is the larger of the two and water is supplied by three horizontal water pumps, two running and one standby, each pump is 500 hp, 16 x 14. The tailings process water flow is supplied by two horizontal water pumps, one running and one standby, each pump is 500 hp, 16 x 14.

The principal make up water to the process water systems is recirculated water from the tailings pond. This water is added to the process water system as make-up water, mainly to for the tailings thickener feed (as previously described elsewhere). This make-up water contributes to reducing the cyanide content in the final tailings and hence ends up in the tailings process water tank as tailings thickener overflow.

Fresh Water

Fresh Water system is mainly furnished from fresh water sources that is the old underground mine dewatering. This water is used principally for cooling water for the grinding mill drives and lube systems, the cooling water for variable speed fluid drives on pumps, for the strip solution heat exchanger and for caustic reagent preparation. The fresh water network is supplied by two pumps, one running, and one standby. The pumps are fixed speed 75 HP, 6 x 4 horizontal water pumps.

Reagent Preparation Water

Reagent preparation system water is supplied from the spent cooling water throughout the plant. The cooling water system, as described above, provides cooling water through a piping network and the spent cooling water from each user is piped to a collecting piping network. This spent cooling water is all piped to the feed of the Reagent Preparation Water tank. The distribution of Reagent Preparation water is provided by two pumps, one running and one standby. These pumps are low head, 15 hp, 1.5 x 1.0 horizontal water pumps direct driven.

Fresh water from the fresh water system is bled to the reagent preparation tank if there are peak demands that cannot be met by the cooling circuit return water. Reagent preparation water is used for strip cooling, stripping area sprays and stripping area make-up water in addition to reagent mixing/dilution for the following reagents: flocculent dissolution, Lime slaking and Copper Sulphate mixing.

Gland Water

Gland Water system water is supplied from the process water header. The gland water network supplies all gland water to the gland seals for the slurry pumps in the process. The low pressure gland water is supplied by three pumps, two running and one standby. Each pump is a direct driven, fixed speed, 100 hp, 3 x 2 horizontal water pumps. The pumps will maintain a distribution system pressure of 700 kPa and each pump gland water feed line will be equipped with a control valve to regulate the gland water to the required pressure/flow.

6.2.3.12 Reagent Preparation

The reagent preparation area includes receiving systems, mixing/preparation/metering systems for flocculent, caustic, cyanide, copper sulphate, nitric acid, anti-scalant and slaked lime. The mixing/preparation and dosing systems are all located at the east end of the process building and cover an area of approximately 18m x 24m with additional area for bulk storage tanks outside the building wall (lime silo, cyanide, nitric acid, peroxygen and liquid SO_2). This area is easily accessible for transportation trucks delivering bulk and packaged reagents.

Flocculent

Flocculent is delivered to the plant in super sacs of 700 kg. At the process plant approximately 4 sacs per day are consumed. The flocculant bags are stored indoors. A seven to ten day supply is kept in the mixing area. The sacs are lifted onto a platform over the hopper/feeder which feeds the patented wetting device which transfers the powder into solution. The solution is diluted and mixed in an agitated mixing tank and then transferred to a flocculent holding tank by a progressive cavity type pump.

From the storage tank the flocculent is metered to the two thickener feed wells by progressive cavity pumps, one pump per thickener with one standby pump for both. As the polymer is pumped by the metering pumps, it passes through flocculent dilution boards (static mixers), one per thickener, where it is diluted further for dissolution into the thickener feed slurry.

The same concept will be applied for the effluent treatment plant but at a much smaller scale in terms of quantity.

Caustic

Caustic soda is delivered in liquid form (50% concentration) in bulk transport trucks of approximately 35 tonnes. At the process plant the caustic is unloaded into a storage tank which is equivalent to about 15 to 20 days storage. It is pumped to the barren strip solution tank where it is diluted with fresh water. The same concept will apply at the effluent treatment plant but the storage tank will be much smaller.

Cyanide

Sodium cyanide is delivered in liquid form (30% concentration) in bulk transport trucks of approximately 30 tonnes. It is unloaded into an outside storage tank which is equivalent to about three days of average consumption. Three pumps meter cyanide to the SAG mill feed, the leach feed thickener feed box and to the leach tanks area. The pumps are stainless steel chemical pumps and each of the 3 metered feed lines is equipped with in-line flow meters.

Lime Slaking and Distribution

Quick lime is delivered in bulk carriers containing roughly 33 tonnes, equipped with pneumatic unloading systems. The lime is unloaded from the trucks into the 300 t capacity silo. The silo is equipped with dust collection and a pneumatic unloading system and an unloading hopper at the bottom. Screw feeders at the bottom of the silo convey the quick lime to two lime slakers where water is added and the quick lime dissolves in the water. The lime slurry flows from each slaker to an agitated storage tank from where the lime is distributed to three different distribution loops. Each distribution loop has two horizontal slurry pumps, one operating and one standby, which feed the loop. From each distribution loop lime slurry is metered into its usage point by an automated valve. The unused flow in each loop is piped back to the agitated storage tank forming a close circuit minimizing line plugging occurrences.

The lime slurry distribution loops are piped to the SAG mill area, the leach tank area and to the Detox plant.

Copper Sulphate

At the process plant copper sulphate is delivered in 1000 kg super sacs transported in batches of 24 tonnes per truck load. The super sac is lifted onto the feed bin of the agitated tank and the copper sulphate powder discharges into the water that has been metered into the tank. The mixture is then transferred by a chemical pump to a storage tank from where it is pumped to the Detox plant via a metering chemical pump.

At the effluent treatment plant copper sulfate is delivered in the same bags and mixed with water in a mechanically agitated tank to a solution strength of 18%. The copper sulfate solution is metered from a storage tank to the reactor tankage.

Nitric Acid

Nitric acid is delivered in a 30 t tanker truck and transferred to the nitric acid storage tank which contains approximately 50 t of acid. The nitric acid is used in the stripping circuit for acid washing and cleaning of lines and vessels/heat exchangers in this area. The acid is diluted to 7 to 9% strength in a mixing tank before using.

Anti-Scalant

Anti-scalant is used in the process water reservoir and in the stripping circuit to minimize the scale build up. Each area has its own storage in barrels and its own anti-scalant metering pump. Anti-scalant is delivered in 20 tonne tankers and stored inside in a 25 tonne reservoir

Sulfur Dioxide (SO_2)

SO_2 is delivered in liquid form by tanker truck of approximately 26 tonnes and stored in a 80 tonne horizontal storage tank. The storage tank comes with a pressure regulator to regulate the pressure of the SO_2 gas added to the reactor tank for the Detox plant.

Peroxide

At the process plant and for the detox peroxide is delivered in liquid form by tanker trucks of 20 tonnes and stored in a 30 tonne vertical outside storage tank. Typically peroxygen at 70% will be used in summer time as compared to 50% in wintertime. A system of pumps made from stainless steel deliver the hydrogen peroxide to a dosing tank with level control. The dosing tank has two functions: first of all to prevent liquid running back from the process point to the storage tank and secondly to build up pressure for the downstream dosing pump.

Hydrogen peroxide is delivered to the effluent treatment plant in containers at the same solution strength as the process plant, and transferred to a dosing tank. Hydrogen peroxide is pumped from the dosing tank to the cyanide destruction reactor tank.

Ferric sulfate

Ferric sulfate is delivered to the effluent treatment as moist crystals in big bags. It is mixed with water in an agitated tank to a solution strength of 14%. The ferric sulfate reports to one reactor tank for the second stage reaction to take place.

6.2.3.13 Service Air Compressors

Plant air is required principally for continuous instrument air consumption and for short-term demand for pneumatic tools during maintenance activities. Based on other projects experience and a design demand assessment for this project the requirement for air demand was established at approximately 1000 cfm at 100 psi. To satisfy this demand, three air screw compressors, two operating and one standby, are required. One hundred percent of the air will be dried so that only one distribution loop will be required in the process plant to provide both instrument air requirements and air for pneumatic tools. This air will also be used for the air clutches for the ball mill drives. One main air receiver will be required at the compressor location and two other air receivers for the distribution loop to minimize pressure drops on the compressed air network during peak loads. In addition, each ball mill will have its own air receiver.

The crusher area has a system that is independent from the process plant. The crusher is equipped with an air compressor with 100 cfm capacity at 100 psi. The compressor is a reciprocating type with its own control system and air receiver. The compressor is located in the mechanical room where the crusher lube oil cooling radiators are located (at ground level).

6.2.3.14 Oxygen Plant

The oxygen plant is designed based on VPSA technology (Vacuum Pressure Swing Adsorbtion) to produce 90% pure oxygen. The VPSA technology is an air separation plant using high efficiency molecular sieves to selectively remove the oxygen from air. The VPSA is a technology allowing producing oxygen with a much smaller size plant.

The plant is installed at the mill site and is owned and maintained by the air separation technology supplier. The plant produces oxygen continuously up to a rate of 520 m^3/h which represents approximately 12% above the average yearly oxygen consumption of the process plant. A liquid oxygen supply tank with vaporizer and pressure regulator are also installed to provide additional oxygen (bulk delivery) during peak demand periods occurring in the summer months of July to September. Those three months average an additional 140 m^3/h of oxygen consumed. The liquid oxygen supply tank is also required to provide oxygen during maintenance periods and unscheduled downtime periods of the oxygen plant. The projected availability of the oxygen plant is around 98%. The liquid oxygen storage tank is replenished from time to time by liquid oxygen supply trucks as required, the summer months being the busiest.

The oxygen is distributed to the leach tank oxygen spargers through a piping network internal to each tank using an inverted cone type sparger.

6.2.3.15 Mill Offices, Dry and Lunch Room

Located adjacent to the concentrator building, on the south west corner, the administrative and general service facilities include administration offices, change rooms, toilets and showers, lockers, cafeteria, first aid stations, and laboratories.

This facility is a 3 stores structure with a total floor area of some 1,900 m^2 and has a 14 m height. The ground floor houses the laboratories and has an area of 1,016 m^2 with a 3,000 mm clear interior height.

Sample Preparation and Laboratory

The laboratory is fully equipped and has separate sectors as described below:

- Sample preparation sector with space/bench area for sample receiving, drying ovens, size reduction equipment and adequate bench space for the preparation of mine and geology samples.

- Assay Laboratory including fire assay, scale room, atomic absorption, chemical lab analysis and chemical storage.

- Metallurgical Laboratory including pressure filters, grinding simulation equipment, flotation cells, leaching test equipment and other miscellaneous met lab equipment as required.

Offices and Dry

The Dry (lockers – wet & dry, showers, toilets and cafeteria) is located on the 2nd floor and has an area of some 442 m^2.

The 3rd floor has also an area of 442 m^2 and houses the administrative offices (10 closed offices, 4 open area work stations, 2 conference rooms, toilets and a small coffee and lunch room).

All of these floors are directly connected to the operating floors in the concentrator by 2 service staircases that also serve as emergency exits for both facilities.

This steel structure will be clad with preformed insulated steel panels on the exterior and offer extensive windows for day-lighting and natural ventilation in living and working spaces. Complete mechanical and electrical systems shall be installed for heating, air conditioning, ventilation, sanitary and lighting requirements.

6.2.3.16 HVAC and Dust Collection

The HVAC systems supply the air and heat required for human occupancy and for process air make up needs. The dust collection systems provide the required equipment to clean and exhaust air used in the plant processes and laboratories.

Crusher

In the crusher area, a central HVAC unit supplies fresh air to all the underground floors to ventilate and heat the areas and to provide make-up air for the scrubber exhaust system. The scrubber is connected to all the transfer points for dust control. Winter ventilation is based on an average of four air changes per hour, summer ventilation is based on the heat from the process equipment. The scrubber exhaust capacity is 6085 l/s. Heat is supplied by electrical coils in the ductwork or by electrical terminal units.

Process Plant

The process plant is ventilated according to a winter/summer schedule designed to provide the minimum required fresh air for winter and exhaust the mill's excess heat in summer. The system supplies fresh air at the lowest practical height and exhausts at the roof in order to ensure the cleanest possible air at the worker's level. The winter air change rate is four air changes per hour, calculated for a volume of 3,6 m above the work floors. Fresh air for the process plant is heated using a combination of heat recuperated from the various mill processes and direct fired natural gas burners in the ventilation units located on the roof of the process plant building. The process plant is heated at ground level using electrical unit heaters. Larger unit heaters are located near exterior doors. Heat given off by the process provides a large part of the building's heating needs.

The process plant mill workshop is ventilated by the general process plant ventilation system. A local exhaust system is used to capture welding fumes.

Tailings Thickener

The tailings thickener underflow pumphouse is ventilated by an air make-up unit at the access tunnel exit.

Offices, cafeterias and Control Rooms

All the offices, cafeterias and control rooms are air conditioned. Heating is by electrical coils in the ventilation units, ventilation ductwork and by baseboards along exterior walls.

Laboratories

The laboratory areas are ventilated in order to compensate for the exhaust from the fume hoods. The laboratories and offices are air conditioned. All fresh air is heated with electrical coils that also provide space heating. Exhaust systems treat the air from the various laboratory sectors with dust collectors or scrubbers.

Locker Rooms

Both the mill and the main dry (wet and dry locker areas, toilets, showers) are ventilated and heated, but not cooled. Dedicated HVAC systems provide the ventilation and drying air. Heating is by electrical coils in the HVAC units and ductwork. A humidity sensor control systems reduces ventilation during low occupancy periods.

Electrical Rooms

All the electrical rooms are pressurized, using dedicated HVAC units, to minimize dust from entering the rooms. The rooms are cooled using self contained air conditioners, connected to a water loop or by split-type systems with rooftop air cooled condensers when water is not available.

6.2.3.17 Fire Protection

The fire protection systems provide protection for all of the plant's various areas according to their specific requirements.

Crusher

The crusher is protected by a 1-1/2 inch fire hose system. In addition, the crusher conveyor is protected by a dry type sprinkler system and the lube and hydraulic units by dedicated sprinkler heads.

Stockpile Tunnel

The stockpile tunnel and stockpile tunnel conveyor are protected by a dry type sprinkler system.

Concentrator

The concentrator does not require sprinklers. 1-1/2 inch fire hoses are located so that all areas of the mill can be reached by fire fighting personnel.

Certain specific areas of the concentrator require additional fire protection: conveyors, lubrication units and hydraulic units require local wet type sprinklers. The SAG feed conveyor is protected by a dry type sprinkler system.

Offices, Drys, Cafeterias and Control Rooms

All the offices, drys, cafeterias and control rooms are protected by wet type sprinkler systems as well as by 1-1/2 inch fire hoses. Sprinkler head density is increased in the drys.

Laboratory

The laboratory is protected by sprinklers and 1-1/2 inch fire hoses.

Electrical Rooms

The electrical rooms do not require fire protection. Manual extinguishers will be installed in all electrical rooms.

Transformer Rooms

The main transformer room requires wet type sprinkler protection.

Water Supply and Plant Area Protection

The fire protection water supply provides 340 m^3/h for two hours. An underground loop supplies water to fire hydrants spaced at 100 meters from one another. In the concentrator, fire water booster pumps provide 65 psi at the top of the highest riser. A jockey pump will also be installed.

6.2.3.18 Process Plant Electrical Power System and Site Distribution

There are three (3) electrical rooms inside the concentrator, one main process electrical room between axes 20 and 24, one remote electrical room for the grinding area and mill lab section, between axes L4 and L5, and one remote electrical room for the reagent area between axes 29 to 31.

The main process electrical room is on two (2) floors, ground floor and operating floor, and it includes three (3) 13.8-4.16 kV transformers for the process pumping motors inside the concentrator and all the 600V transformers for smaller motors and services. For the 4.16 kV distribution, there are (3) transformers of 12/16 MVA with enough capacity to cover a first contingency (n-1). The 4.16 kV switchgear and starters are located at the operation floor above the transformers. Six (6) transformers 13.8 kV-600V, 2.5/3.3 MVA and distribution center are located at the operation floor near the 600 Volts motor starters.

Between axes 18 and 19, three (3) other electrical rooms are added to install the Quadramatic drives (GE motors), controls and protection relays for the Ball mills. Those electrical rooms are located at ground floor near the ball mills. For the SAG mill motor supplied by ABB, the cycloconverter (E-House) and cycloconverter transformers are located between axes 5 and 7.

For the crushing and ore handling area, one (1) 4.16 kV feeder has been included with one substation at the stock pile, one at the conveyors drive area and one at the crusher building. Each substation includes 4.16 kV and 600 V distribution equipment for processing and services. The 4.16 kV power distribution cables are installed directly on the conveyor structures.

The administration and garage building is fed by the 13.8 kV service loop by an indoor electrical room located near the garage building. This electrical room includes all the necessary 13.8 kV and 600V equipment for this area.

From the 25 kV line, two (2) local substations consisting of pad mounted and mobile sub-units have been considered to feed the mine shovels, open pit lighting and water pumping.

6.2.3.19 Process Automation and Communication Systems

Process Control system

The Project has adopted PLC's, industrial human machine interface (HMI) as local operator stations and SCADA software to implement overall process control operations.

Sectors of the plant will be fully automated with operation controlled from two control rooms and a minimum operator involvement locally.

Expert system software will be used to improve production and stability of the PID loops of the process. An expert system software is a computer program that solves problems using information and reasoning techniques normally associated with a human expert. The expert system is linked to the process control I/O system and allows control room operators to diagnose and respond to unusual disturbance operating conditions. More specifically in a 55,000 tpd grinding circuit, an expert system will contribute to maximize throughput and to stabilize the control of particle size product which is of great importance for the recovery of fine gold particles. A better control of the grinding circuits will result in significant savings by improving liner protection thus extending the life of the liners.

Historian system software will be used to implement process data collection for production management.

Plant control network architecture will primarily be using Ethernet connectivity for I/O control as well as for supervisory functions. The entire control system is designed such that remote access from any location within the plant or from outside the plant network will be possible; allowing remote troubleshooting, process data collection, or even production analysis.

The operator station software is selected to operate using a client/server configuration. In such a configuration, the engineering station, the plant remote operator panels, portable wireless laptop terminals, palm user interfaces and even remote office desktop computers are all clients of the control application server. Any of these clients, given the proper access rights, will be capable of accessing the server to obtain for example, production orders, Standard Operation Procedures (SOP), laboratory test results, etc. In the suggested architecture, a RAID 5 server configuration is selected to host the control applications configuration and the historical data collected on the process in order to guarantee data integrity.

The engineering station hosts the configuration software for operator stations, plant operator panels, process and I/O controllers and networks configuration. The engineering station also supports remote access from any location, allowing the performing of configuration tasks remotely.

Process controllers and communication networks, like the server and stations, are selected to be redundant-capable in order to minimize system stoppage and plant downtime. Remote controllers required for the tailings, reclaim and fresh water pumping are linked to the main control network through fiber optic cables rather than by regular copper communication links or RF. For all process control, software and communication networks, the products selected are widely used, proven, industrial grade technologies and are open software platforms.

Motor control centers and switchgear are designed to come pre-wired with remote I/O modules and in specific cases, with intelligent O/L relays and power monitoring equipment all of which provide ethernet network communication capabilities. Supervisory functions and critical control operations are communicated over separate networks.

Local control stations will normally be composed of a START (green) pushbutton and a STOP (red) pushbutton. Operating mode selection for given equipment will be done from a virtual LOC-OFF-REMOTE mode selector programmed on the operator interface unit. The actual function of the “Start” pushbutton will either be interpreted by the control system as a regular “Start” function when the mode is “Local”, or else as a “Jog” when the mode is selected to “Remote”. Exceptionally, local control stations may also have an EMERGENCY STOP (red) latching type mushroom pushbutton when associated with moving equipment presenting higher potential hazards to personnel. The emergency stop button when used, will always be hard wired in the equipment starter control (run back to the MCC), just as conveyor emergency pull cord units are.

The data and application server of the control system, operator stations, process controllers, remote I/O modules, process instruments and network components will be powered by a 120 Vca UPS system to ensure an uninterrupted operation.

Process instruments are, whenever possible, “intelligent” instruments (HART-capable for example) and they are wired to the analog inputs/outputs of the process controller by means of traditional, individual 4-20mA signal cables. Exceptionally, instruments that are multi-variables, such as belt scales, or have advanced diagnostics capabilities are also going to be connected with a communication link to the supervisory network.

Phone System

The phone system will be based on Ethernet using Voice Over IP. Each phone will have two (2) Ethernet ports. The phone system shall be host on a dedicated server complete with extension management and voice mail.

Paging System

The paging system will be integrated with the phone system. Speakers will be located in all process and office areas.

Access System

A computer based access system will be used to monitor and grant access to the site using electronic ID cards.

Camera System

Camera system will be web-based using MPEG2 format. The cameras will be connected to the plant LAN. Cameras will be 8 mm color CCD with 480 line horizontal resolution and electronic sensitivity enhancer type cameras.

Communication Network System

The fiber optic network will be used to link, through Ethernet, the different control systems and all auxiliary systems.

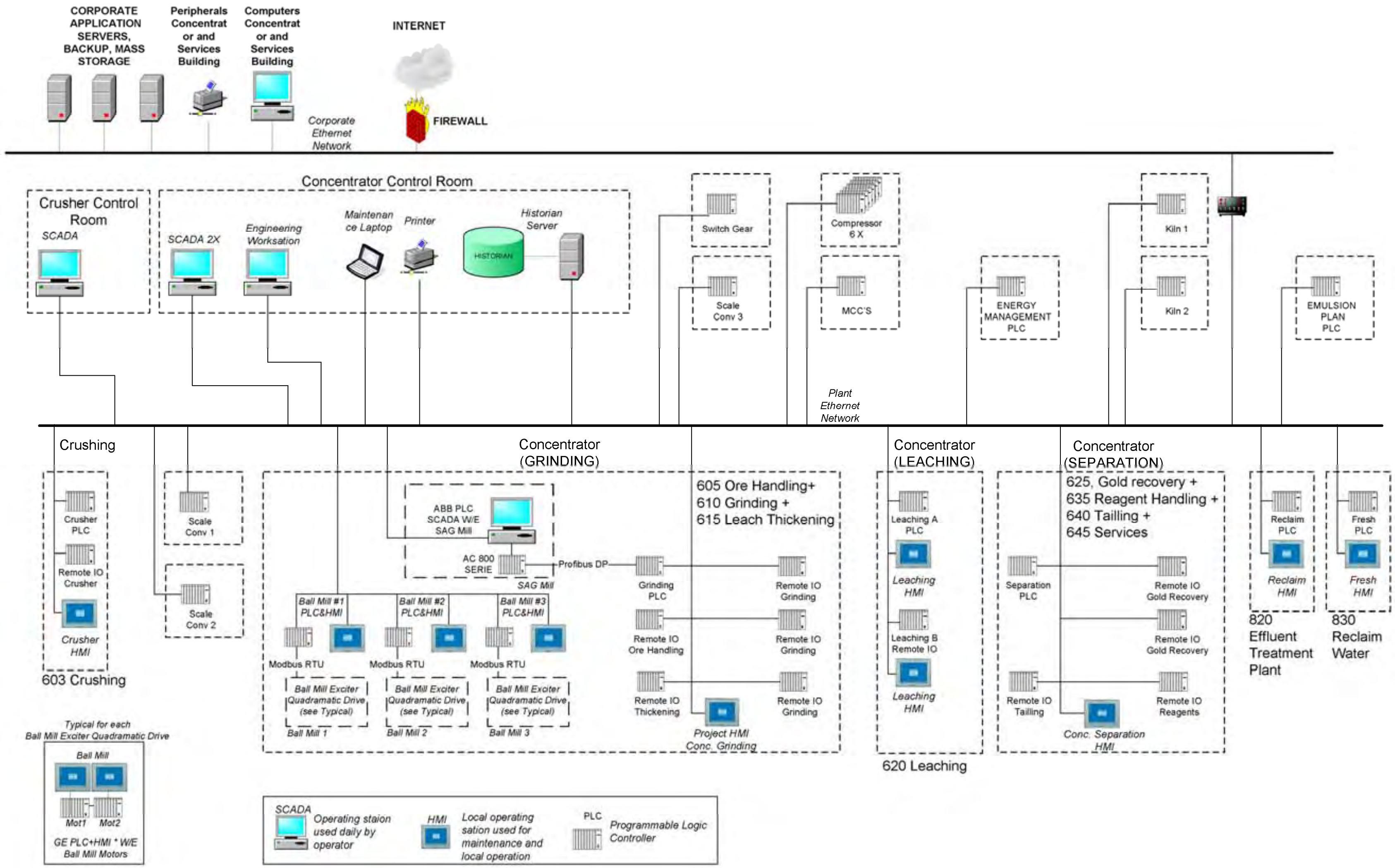
The fiber optic network will be installed across the plant facilities to integrate the different required systems. Fiber optic cables shall be run in cable trays for locations close to the concentrator (crusher, administrative building and garage) while a messenger installed in the various overhead lines will enable the fiber optic to reach the more remote locations (pump houses, tank farm and entrance gate).

Where communication links will share the same communication equipment, a dedicated VLAN will be used for each type of communication link. The network will be configured for packet priority in order to optimize the available bandwidth.

The communication infrastructure will have a 50% bandwidth reserve for future needs.

The global control system architecture is shown in Figure 6.19.

Figure 6.19: Concentrator Global Control System Architecture Network



6.2.3.20 Mill Workshop

The mill workshop is located on the ground floor in the south east corner of the process building. The workshop is 12m x 30m and is serviced with two, 10 t overhead cranes. The shop contains mechanical maintenance equipment such as drills, lathe, burning and welding equipment, pipe shop and specific tools for equipment component repair.

A tool crib for storage of field repair equipment, rigging equipment, etc. is also provided. An office and reference area for equipment manuals and drawings is incorporated into this space for technical reference information required for the shop repairs.

6.2.3.21 Process Mobile Equipment and Tools

The following mobile equipment is provided specifically for operations and maintenance tasks in the crusher and process area:

- 2 – bobcats with fork and bucket attachments
- 2 – 3.5 t fork lift, propane and diesel driven
- 2 – Mobile air compressors
- 1 – 25 t boom truck
- 1 – Loader CAT TH38
- 1 – Loader CAT IT
- 2 – Service trucks
- 2 – Giraffes
- Lot – Tools.

6.2.4 Manpower Requirement

The manpower requirements for the process plant facilities were estimated by sector of activities and are presented, based on an annual 2,080 hours of work per person, in Table 6.10. A total of 90 people are required for the management, operations and maintenance of the process plant including crushing and conveying. In general, the operations personnel will work twelve hour shifts to provide 24 hour coverage. The maintenance crew will normally work on day shift with some sub-groups allocated to emergency repairs and on-demand requirements.

Table 6.10: Milling Manpower Summary

Area/ Position	Roster	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Administration													
Mill Manager	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Mill Manager Secretary	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Sub-Total Administration		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Met Lab													
Chief Metallurgist	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Metallurgist	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Met Technician	Dayshift	3	3	3	3	3	3	3	3	3	3	3	3
Sub-Total Met Lab		**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**
Assay Lab													
Chief Assayer	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Senior Assayer	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Technician	Rotations	3	3	3	3	3	3	3	3	3	3	3	3
Sample preparation	Rotations	8	8	8	8	8	8	8	8	8	8	8	8
Wet Lab	Rotations	8	8	8	8	8	8	8	8	8	8	8	8
Fire Assay	Rotations	4	4	4	4	4	4	4	4	4	4	4	4
Sub-Total Assay Lab		**25**	**25**	**25**	**25**	**25**	**25**	**25**	**25**	**25**	**25**	**25**	**25**
Operations													
Operatons General Foreman	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Production Coordinator	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Trainer	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Foreman	Rotations	2	2	2	2	2	2	2	2	2	2	2	2
Operator	Rotations	12	12	12	12	12	12	12	12	12	12	12	12
Service Operator	Rotations	5	5	5	5	5	5	5	5	5	5	5	5
Crusher Operator	Rotations	6	6	6	6	6	6	6	6	6	6	6	6
Sub-Total Operations		**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**
Mechanical													
Mechanical General Foreman	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Mechanics	Rotations	18	18	18	18	18	18	18	18	18	18	18	18
Sub-Total Mechanical		**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**
Electrical													
Electrical General Foreman	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Electrician	Rotations	8	8	8	8	8	8	8	8	8	8	8	8
Sub-Total Electrical		**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**
PLANNING													
Senior Planner	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Planner	Dayshift	1	1	1	1	1	1	1	1	1	1	1	1
Sub-Total Met Lab		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Grand-Total Mill Department		**90**	**90**	**90**	**90**	**90**	**90**	**90**	**90**	**90**	**90**	**90**	**90**

7.0 TAILINGS AND WATER MANAGEMENT

7.1 Tailings Management

7.1.1 General

The Tailings Management Facility (TMF) design has been developed to accommodate 183 Mt of tailings during the mine life. The tailings from the milling process will be deposited over a period of approximately 10 years at an average production rate of 55,000 tpd. The required tailings storage volume to accommodate the 183 Mt at an assumed settled dry density of 1.5 t/m^3 is 122 Mm^3.

Given the expected environmental issues associated with the TMF, it was decided at the onset of the project to proceed using thickened tailings disposal technology as the preferred tailings disposal approach. This relatively new technique greatly simplifies water management and reduces the overall risks typically associated with conventional tailings management (slurry disposal at lower densities). It is important to mention that the proposed management strategy described in the following sections has been used elsewhere at different scales in various climatic conditions and has been shown to be a robust solution. Among its key advantages, this method provides a better water management facility with non-segregated tailings that exhibit more homogenous properties, reducing the need for confining structures and simplifying the rehabilitation process during site closure.

However, the thickened tailings management method tends to be more labour intensive than other techniques, and requires better planning and closer monitoring. This is clearly the path that the mining industry will follow in the coming years and is an opportunity for Osisko to become a leader in the application of this type of technology in Canada for large scale projects.

7.1.2 Key Design Assumptions

As mentioned, the design efforts for the TMF have been based on the use of thickened tailings technology. Given the high tonnage of tailings to be deposited, this technology will help in minimizing the footprint and improving tailings water management. Thickened tailings do not segregate and are expected to have better overall strength characteristics and behaviour as compared to conventional slurry. The tailings will be thickened to about 68% solids content using high performance thickeners.

With respect to geochemical characteristics, it has been assumed based on early results from the environmental test program that tailings produced by the Canadian Malartic project will not be acid generating and will not be prone to leaching. These assumptions are supported by some early results showing that the vast majority of the tailings produced do not exhibit these problems. A minor fraction of the tailings is nevertheless classified in the ”grey zone” and will require further testing as prescribed by Directive 019 of the MSDEP to ensure they will not be considered as potential acid generating material. It is important to mention that these early assumptions are consistent with what is known and observed from the geochemical behaviour of tailings deposited in the area from prior mining activities. The current environmental issues with tailings in the area originate primarily from tailings from custom-milled ore as a result of old mines outside the Malartic area that were known to be polymetallic deposits with complex mineralogy (e.g. Bousquet, Doyon, etc). It should also be noted that a detailed characterization program has been initiated, aiming at properly identifying the geochemical characteristics of the different materials that will be produced at the future mine. Depending on the outcome of these tests some modifications to the tailings and waste rock management strategy may be needed. It is expected that formal reports will be available by the end of Q1 2009.

In terms of the cyanide consumption used in the process, it has been assumed that by using thickened tailings, the recovery of cyanide at the plant will be maximized. With respect to the tailings stream, cyanide will be mostly destroyed (well over 80%) at the plant using the well established CombinOx® process which is an SO_2/Air and peroxide based technology designed to reduce cyanide to residual levels in the tailings pore water. The proposed approach to manage cyanide will aim at being consistent with the new cyanide management code currently being adopted by the gold industry.

As a result of the fact that the tailings will not be characterized by acid generating potential or leaching problems and that cyanide will be destroyed to residual levels in the pore water, no formal confinement system, e.g. lining system using geomembranes is required for the tailings area. This assumption is a key element for the project. It is believed that a technical case could be constructed, if need be, in order to support this hypothesis by considering the expected benign geochemical characteristics and the intrinsic low permeability nature of the tailings, as well as the expected low exfiltration rates. It is important to highlight that using thickened tailings placed at 68% solids content will result in virtually no excess water release from the tailings on a regular basis. The sources of run-off will only be from precipitation and snow melt.

7.1.3 Site Selection

A review of the available zones in the surrounding area of the proposed mine site was done to identify sectors that could provide the required capacity for tailings deposition. Given the expected volumes of tailings to be produced and the current land use in the area which is well developed, it was identified that finding zones that meet the required capacity without significant impacts to the current land use would be a challenge.

A site location was originally considered by Osisko, West of the road known as the Lac Mourier Road, a few kilometres away from the proposed plant location and immediately upstream of the existing East Malartic tailings pond. This site which offers very advantageous characteristics in terms of topographical confinement is currently characterized by little land use and would have been able to provide the required capacity. Upon review, it was nevertheless felt at an early stage that there would be significant environmental benefits to take advantage of the existing East Malartic tailings pond footprint as the basis of the future tailings basin instead of developing a new area on previously unused lands. An aerial view of the site is presented in Figure 7.1.

Figure 7.1: Current Canadian Malartic Site

The East Malartic tailings pond site has a complex history and is currently the property of the Government of Québec as an “orphan site” following the bankruptcy of McWatters in the early 2000s. This site contains tailings that are known to have high acid generation potential in certain sectors. Given the events surrounding the disappearance of McWatters, the rehabilitation of the East Malartic tailings area has not been completed and currently the site requires close monitoring.

Therefore, using the footprint of East Malartic as a basis for the future Canadian Malartic tailings area would provide the following advantages:

- It would allow the rehabilitation of one of the most important orphan sites in Québec by the placement of non-acid generating tailings on top of acid generating material.
- It would minimize the footprint of zones impacted by tailings placement in the area of Malartic by concentrating all tailings in one location.
- It would offer a location close to the mine site and at an adequate distance from the town.
- It would not infringe on non impacted watersheds.
- It would allow better water quality control.
- It would be in line with the sustainable development policy of Osisko and the Québec Government.

The TMF footprint including the waste rock dump will entirely cover the present existing cells and ponds of the East Malartic tailings facility, including the old sedimentation and polishing ponds. This approach will allow rehabilitation of the existing facility which contains acid generating material in some areas.

Given the location of the open pit as well as the availability of space, it was decided to locate the waste rock dump on the site of the East Malartic tailings facility, so as to minimize further land impact, and to minimize haulage distance. The location for the waste rock dump was identified as the one between the TMF and the open pit. Tailings and waste rock will then be managed in this one location for the mine life.

7.1.4 Delivery System

The thickened tailings will be delivered to the TMF at approximately 68% solids by weight. The deposition plan that has been developed requires that tailings be deposited from different central points, either along an axis or on top of a cone, to create a series of cones that will overlap progressively and will be characterized by gentle slopes. The whole area will be developed by sectors using a pre-defined placement sequence.

Tailings will be discharged by multiple ends of pipe discharge points to reduce energy at placement and promote uniformity in layer thickness. Initially, discharged points will be located at the highest elevations of the original topography allowing discharge to occur in all directions. Discharge points will later be raised on top of the deposited tailings or relocated directly on the created slopes for the development of new sectors. This method of placement will produce beaches that are relatively uniform and of conical shape. From experience, it is recognized that beaches tend to flatten as distance from the discharge point increases.

The tailings will be contained at the boundaries of the TMF by starter perimeter berms built with non acid generating waste rock. These berms and peripheral ditches will allow the collection and management of the run-off water that would otherwise accumulate at the surface of the tailings.

7.1.5 Conceptual details

The tailings deposition plan was developed using the Walace software as well as the Surpac plate-form as visualizing tool. Table 7.1 presents the tonnage and volumes that have been modeled.

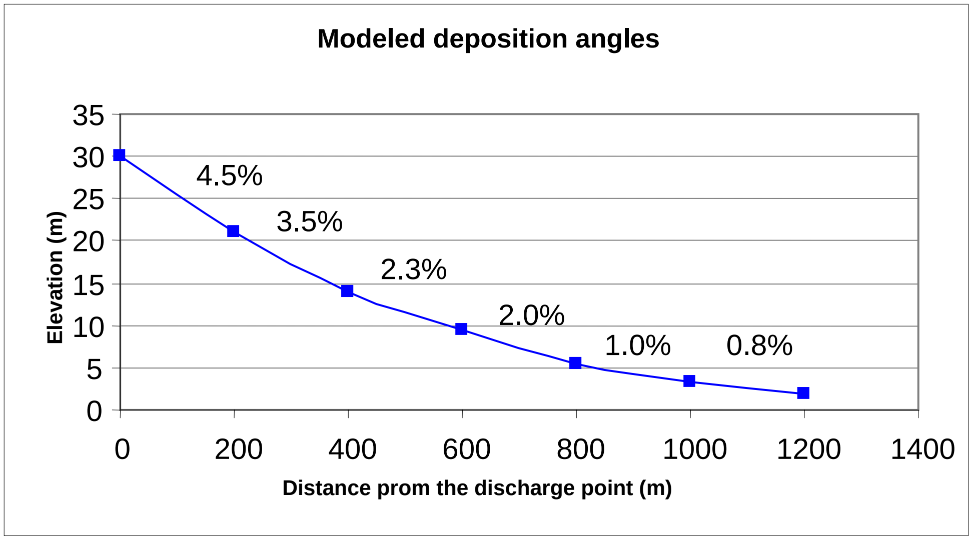
Table 7.1: Conceptual parameters

Tailings production	
Daily production	55,000 tonnes
Tailings physical properties	
% solids (by weight)	68%
Average dry density	1.5 t/m^3
In situ void ratio	0.85
Specific Gravity	2.8
Saturated water content (by weight)	30%

7.1.6 Beach Development

Thickened tailings are expected to settle to varying angles depending on the distance from the discharge points. The following final profile displayed in the graph of tailings deposition profile (Figure 7.2), has been used to model the filling scheme. This profile has been interpreted from actual deposition of tailings at 60% solids content. It is to be noted that there is no project with similar deposition rates and tonnage in Canada operated at 68% solids content. The profile presented in the graph of Figure 7.1 represents the best available approximation of tailings behaviour and is believed to be conservative. The current plan is to proceed to full scale pilot testing at the site at the beginning of the operations in order to adjust these parameters and to update the modelling of the deposition plan.

Figure 7.2: Graph of Tailings Deposition Profile



The development of the tailings pond is based on the upstream raise method. It is currently planned to proceed with the construction of a series of small berms on the tailings beaches approximately 4.0 m high to provide some lateral confinement to the tailings and to limit unnecessary lateral spreading. Berms will be constructed with waste rock to achieve an overall slope of 10H:1V.

The deposition plan has been modeled so that the final topography of the tailings facilities will closely match that of the natural environment. Figure 7.3 presents the final modelled configuration of the tailings facilities. As previously stated, it is important to note that tailings and waste rock will be managed in parallel, according to a common management plan. The final elevation of the designed tailings is expected to be between 349.0 m and 365.0 m representing a maximum height gain of approximately 38 m above the initial ground. Table 7.2 presents a summary of the tailings and berm elevations:

Table 7.2: Summary of Modeled Deposition Quantities and Elevations

Year	Tailings Cumulative Quantities (x 1 000 m^3)	Tailings	
		Maximum Tailings Elevation, (m)	Confinement Berm Elevation (m)
1	13,383	350	333
3	40,149	358	333
5	66,915	364	341
10	120,447	365	349

Currently the plan is to cover all external slopes in areas that have reached the final configuration, such as the sectors downstream of each raise, with a 5 m layer of mine waste rock, and a vegetated cover on top of the waste rock. This approach will allow testing different revegetation methods and will allow progressive closure of the facility. It is planned that approximately 65% of the surface of the tailings will be capped with a 5 m thick layer of waste rock and revegetated by the end of mine operations. At the end of the mine production, the rest of the surface will also be capped with a 5 m thick layer of waste rock and the revegetation will also be completed.

From a geotechnical point of view, these overall gentle slopes should provide very favourable long term safety factors against failure. After placement of the upper layer of waste rock for capping, it is expected that the resulting stack will meet acceptable safety factors and should behave in a very robust way under long term conditions.

7.1.7 Geotechnical Investigation for the TMF Area and the New Polishing Pond

An extensive geotechnical investigation has been completed in order to confirm foundation conditions for the New Polishing Pond, the tailings deposition area, the waste rock pile as well as the mill complex. The global investigation program for all these elements of the TMF consisted of the following:

- A total of 43 boreholes have been drilled at key locations such as the planned dykes for the New Polishing Pond, the perimeter of the tailings deposition area and the mill complex. Soil samples from the boreholes have been sent to the laboratory for further testing. A total of 11 piezometers have also been installed in the boreholes in order to monitor water levels and to, eventually, sample the groundwater. In situ vane testing has been performed in areas where clayey soils have been encountered in order to obtain continuous sheer strength profiles.

- Selected samples have been submitted for tests in order to determine some key characteristics such as grain size distribution curves, Atterberg limits and consolidation rates.
- A total of 14 Cone Penetration Tests (CPT) have been performed mainly in areas with existing tailings and soft clays. CPT is an effective in situ testing method used to interpret and confirm the stratigraphy as well as some resistance characteristics for fine grained materials.
- An extensive borrow search consisting of aerial photo interpretation, test pitting and sampling has been performed. The main goal for the borrow search was to find possible borrow sources for low permeability materials as well as sand and gravel at reasonable distances from the different projected structures.

Taking into account the results from this investigation, the design for the New Polishing Pond and the TMF has been adjusted.

7.1.8 Overall Stability of the Tailings Impoundment

Stability assessment of the tailings stack was carried out in order to confirm some key design elements such as spacing of the berms, need for additional stabilisation and overall stability of the tailings facility. The analyses were carried out for the final modelled configuration of the tailings facility.

The stability analyses were based on the following assumptions:

- The analysis was carried out at what is believed to be the critical cross-section of the site. This cross-section is associated with the existing dyke 5 and the weak clay foundations found underneath the existing tailings pond. Also, the New Polishing Pond, located at the toe of dyke 5 should possibly contribute to a higher water table in the sector.
- The position of the water table in the tailings mass at this stage has been considered approximate and will require further analyses and field monitoring. 2D seepage analyses, using the commercial software SEEP/W, have been performed in order to establish the possible position of the water table for the final tailings configuration. However, setting up boundary conditions for this analysis is a challenge. The stabilized position of the water table was assumed to be as high as at the 2/3rd elevation of the total height of the tailings stack which is believed to be a conservative assumption. According to the modeling results a seepage face might develop at the bottom of the stack upstream of the dyke 5 in the tailings pond itself. Seepage is expected to be very limited and will be collected in the tailings pond. The actual position of the water table will need to be verified regularly in the operational phase of the stack (eg. piezometer measurements) in order to confirm this hypothesis.

- The position of the water table depends also on the hydraulic conductivity of the tailings. Testing on the thickened tailings in order to further determine their physical characteristics will continue into in the detailed phase of the project in order to improve the overall filling scheme. For the modeling typical values from the software library have been used.
- The foundation of the tailings stack in the selected location for the analyses is characterized by the presence of weak clays. Resistance parameters measured for the design phase of dyke 5 and confirmed by CPT testing results during the recent field investigation have been used in the modeling. It is expected that these soil layers as well as the East Malartic tailings and the new Canadian Malartic tailings will undergo a significant consolidation with time; however, it is difficult to quantify these consolidation rates at the present time. Simple relationships to assess the consolidation of the clay layer could have been used, but it was preferred at this stage to produce a more conservative analysis with no strain gain. Therefore, completing the analyses for this study with parameters that do not take into account further consolidation is believed to be a conservative approach. Consolidation should contribute to an increase in resistance parameters to improve overall stability and decrease of the hydraulic conductivity.
- The Canadian Malartic tailings potential for liquefaction assessment will be required in the detailed design phase of the project. Some testing is recommended when an appreciable sample of tailings becomes available. Thickened tailings are less susceptible to liquefaction than tailings deposited as slurry. However, given the high production rate, the Canadian Malartic tailings are expected to be deposited at a very high rate and the time allowed for the dissipation of the pore water pressure might be very short. In order to adequately manage this aspect, the detailed deposition sequence will take into account the results of the additional studies in order to allow sufficient periods of rest before putting in place the subsequent layers of tailings. Further testing and modeling typically takes place at the beginning of the operations and would include consolidation testing with Rowe cells and vibrating table testing, as well as field tests (typically CPT) in order to be able to determine the potential for liquefaction of the tailings mass.

Stability analyses were carried out in static and pseudo-static conditions using a bedrock acceleration coefficient of 0.054 g corresponding to the PGA (peak ground acceleration) with probability of of 10% in 50 years (1 in 476 years). This acceleration is believed to be adequate at this stage. Analyses were also carried out for the entire tailings site including dyke 5 in particular given the fact that the upstream condition for this structure is expected to change by putting in place the Canadian Malartic tailings. All calculated factors of safety meet current design practices at this stage of the design work.

One stability analysis, assuming progressive closure consisting of the installation of an additional rock cover, was also carried out. This configuration was studied in the most critical pseudo-static condition.

The results demonstrate that the rock cover does not have a negative impact on the overall stability and may be gradually put in place.

7.1.9 Waste Requirements for TMF Raises

Table 7.3 below shows the required waste rock material needed for the development of the tailings beach throughout the life of the mine. A total volume of 3.2 Mm^3 of waste rock is needed to construct the needed berms on the side of the tailings beach. These volumes have been calculated for berms with a height of 4 m and a horizontal width of 5 m. Progressive closure will require approximately 12 Mm^3 during operation.

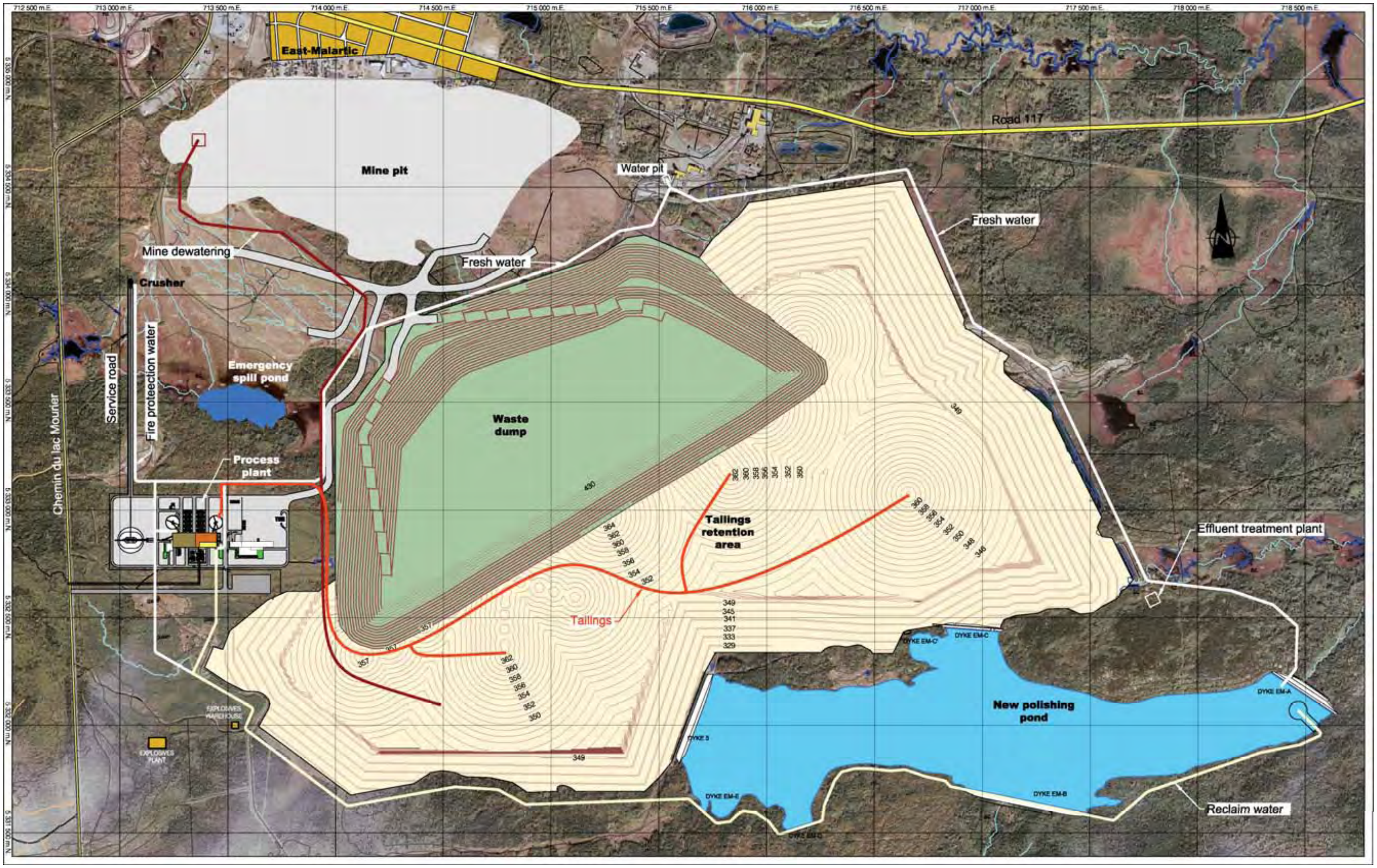
Table 7.3: Waste Rock Requirements for Tailing Pond Construction and Progressive Closure

End of Year	Berm Volume, (m^3)	Progressive Closure (5 m cover Layer), (m^3)
1	295,000	1,122,000
3	420,000	1,608,000
5	550,000	2,110,000
10	1,895,000	7,275,000
Total	**3,160,000**	**12,115,000**

7.2 Waste Rock Management

The waste rock will be managed in conjunction with the tailings, however, some waste rock will be generated a year before the start of the operation. Figure 7.3 presents the modeled final configuration of waste rock and tailings.

Figure 7.3: Final Configuration of Tailings and Waste Rock and Water Networks



The design of the waste rock pile has been developed to accommodate approximately 326 Mt of waste rock that will be mechanically placed and is expected to require a total storage volume of 161 Mm^3. The waste pile has been located north-west of the TMF and is planned to be used as confinement for the tailings as well. The final waste pile elevation has been modeled and it is expected to be in the order of 430 m. This final elevation takes into account the fact that the tailings will be covered with 5 m of waste rock during progressive closure and at the end of the operations and that waste rock will be used to construct all confinement berms.

The waste rock pile design and modeling assume that the waste rock, like the tailings, is not expected to present environmental issues and will make a good construction and cover material. Similarly to the tailings, tests are under way to properly characterize this material. It is expected that if portions of the waste rock to be produced were to present either acid generation potential or leaching issues, they would be managed separately in a proper way to minimize their impact on the environment.

The waste rock pile will have overall slopes of 3H:1V with benches. Table 7.4 presents a summary of the results of the developed design for the waste rock pile.

Table 7.4: Summary: Waste Rock Pile Development and Elevations

Waste Rock		
Year	**Cumulative Volumes (m^3)**	**Waste Rock Pile Elevations, (m)**
1	20,016,832	350
3	66,143,564	383
5	108,313,861	412
10 and closure	161,782,673	430.0 (427.0 as reduced height taking into account the progressive closure)

7.3 Water Management

Water management at the site is composed of different elements. Each of these elements has a specific role and impact on the overall management of the site. These elements are summarised as follows and more detailed description of each of them can be found in the following sections:

- New Polishing Pond: this is the largest structure in the water management system located south of the existing sedimentation and polishing ponds. The pond will accumulate water from different sources and will constitute the water reserve for the mill operations.

- Surface water collection system and management: the water from different external watersheds will be routed to the New Polishing Pond. The system includes areas for water accumulation and ditches allowing collecting the available surface runoff water in the New Polishing Pond.
- Underground pumping for mine dewatering: water coming from the underground pumping system will be accumulated in the New Polishing Pond as well. Modeling including establishing pumping rates and volumes was completed in the hydrogeological study.

7.3.1 Water Balance

A detailed annual water balance analyses was completed for the site and the site water balance flow diagram is presented in Figure 7.4. The water balance addressed two cases as follows:

- Case 1 – Startup: For startup, a total of 6 Mm^3 of water is required, which can be collected over two years.
- Case 2 – Operations: Once in operation, the mill will require approximately 9.45 Mm^3 of water annually. A minimum of 2.84 Mm^3/yr (7,700 m^3/day) is expected to be available from underground water pumping (based on modeling results), with the remainder from other water sources such as surface runoff.

For the Start-up case the following sources have been considered:

- Runoff from the New Polishing Pond local watershed – this water will accumulate directly in the new polishing pond.
- Runoff from the existing tailings, sedimentation and polishing ponds - currently water from the tailings pond is siphon transferred to the existing sedimentation pond. In the future that water will be transferred to the new polishing pond either through an open channel or by pumping. A separate pumping station will be required for water transfer from the existing polishing pond to the new polishing pond.
- Water from the underground mine – this water is currently pumped at an annual rate of 1.4 Mm^3 and will be discharged directly into the new polishing pond.

Acording to the preliminary water balance, these sources should allow accumulating the required 6 Mm^3 in 2 years according to the regional runoff averages.

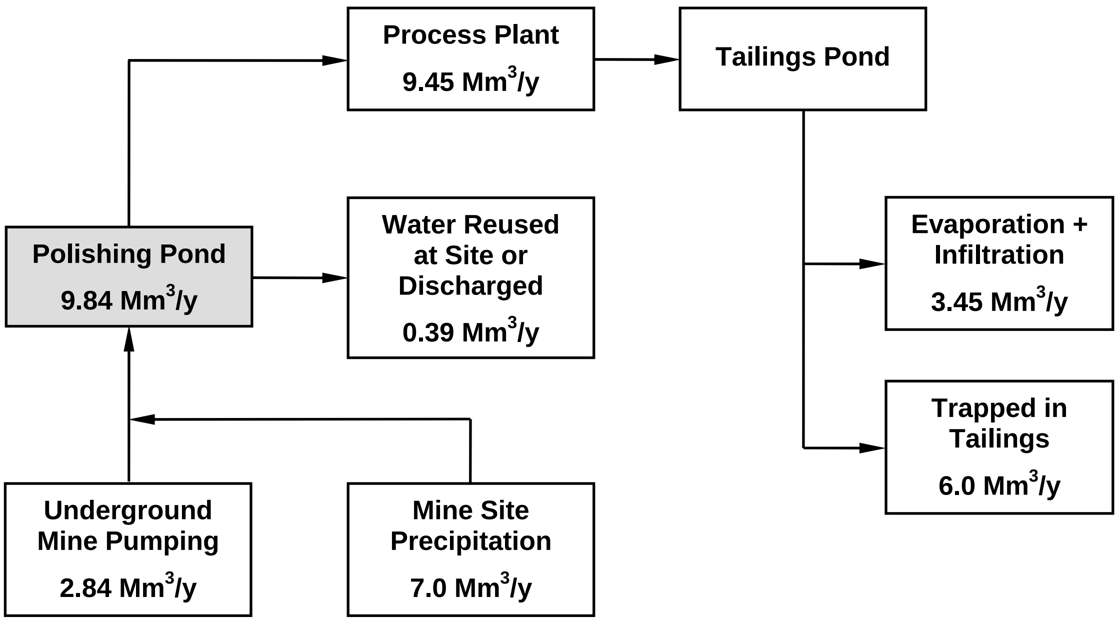
For the Operational case it is considered at the present time that the following main sources of water are available for the operations of the mill:

- Runoff from local watersheds: these watersheds include the New Polishing Pond local watershed as well as runoff for the tailings facility footprint.
- Runoffs from two external watersheds upstream of the East Malartic site namely the North and South diversions.
- Water coming from the dewatering of the open pit and underground mine as well as runoff from the footprint of the open pit itself.
- Water from the Malartic creek located north of the property.
- Thickened tailings are not expected to release significant quantities of water. No input from the tailings was considered in the water balance at this stage.

In conclusion, water balance calculations indicate that assuming the local watersheds produce somewhat comparable runoff to those in the region, there will be sufficient water to fill the new polishing pond for start-up. Also, considering that a sufficient underground water pumping rate, that has been estimated at 2.84 Mm^3/year, is maintained there will be available water to sustain the operations. Some of the conclusions of the water balance calculations are as follows:

- In the event of dry years there might be a risk of insufficient water availability for operations. It is expected that the milling operations will be using the 2.84 Mm^3 available every year from the underground mine carefully such that about 1 Mm^3 can easily become available to cover for these dry periods
- Once the tailings deposition has sufficiently advanced there is a much greater chance of providing all required operational water from the local watershed and man-made structures.
- The uncertainty in this assessment will be reduced as more information is gained through the flow measurement program that has been initiated for the local watershed.
- Typically, by August or September the likelihood of a water shortage would become clear providing several months to respond to an impending water shortage. Storage volumes are always lowest in late winter (February through April), since surface flows during the winter are negligible. This shortage will be controlled with the underground pumping rates. Periods of very low storage pond levels typically occur in years when runoff is insufficient to fill the pond during spring runoff.

Figure 7.4: Site Water Balance

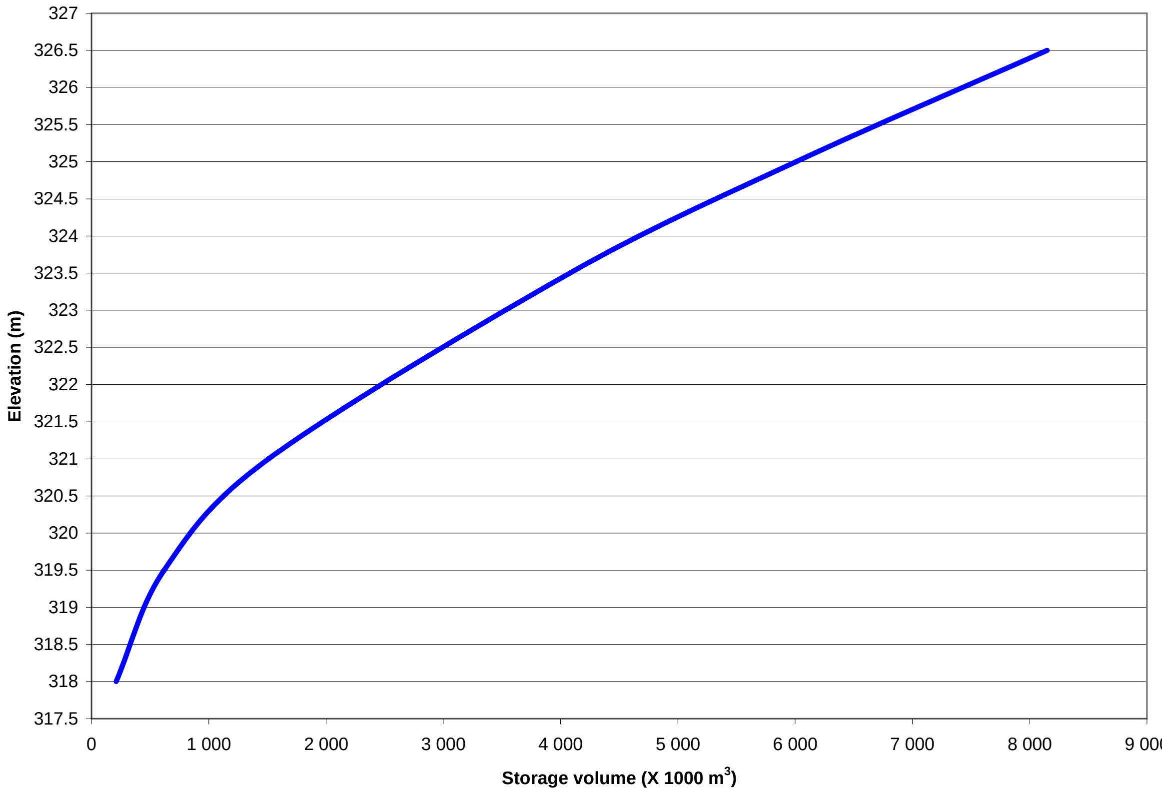


7.3.2 New Polishing Pond Design

The first element of the water management system of the Canadian Malartic project consists of the construction of a New Polishing Pond that will be located south of the existing sedimentation and polishing ponds of the East Malartic property (Figure 7.1). The New Polishing Pond is also the main water management system item concerning the closure of the orphan site as planned by the Québec Government, the owner of the environmental responsibility for this site.

The construction of the New Polishing Pond is required for three main purposes that are the closure of the orphan site, the accumulation of water for the start-up of the milling operations and the water storage facility during operations. The New Polishing Pond will have an approximate capacity of 6 Mm^3. This capacity was determined in order to meet the initial requirement for the start up and is limited by the existing topographical conditions. The pond will be confined by five dykes with crest elevation at 326.5 m and will be operated at a maximum water level of 325 m. The capacity curve for the water pond is presented in the graph below (Figure 7.5).

Figure 7.5: Graph Showing Capacity Curve – New Polishing Pond



The design effort for the New Polishing Pond, including the field investigation, was carried out starting in the fall of 2007. A short summary of the design activities as well as typical cross sections for all proposed dykes was reported. The New Polishing Pond will be confined by five low permeability dykes. The proposed cross-sections for each structure vary given its role, type of low permeability material to be used and foundation conditions. Overall the design consists of a low permeability core with downstream filters and appropriate rock fill. Two of the structures, namely dykes EM-A and EM-B (Figure 7.3) will consist of clayey core equipped with one transition layer and one filter layer. Dyke EM-A will be constructed on a rock foundation which will require preparation. Dyke EM-B will be constructed on a clayey foundation and a provision was made for the eventual expected settlements to occur at its location.

Dykes EM-C, EM-D and EM-E are more simple structures consisting of till core and one filter layer. The design of dyke EM-C was also adjusted to take into account the fact that, with time, this dyke will also become part of the confining berm for the tailings pond and that seepage flow might change direction.

One emergency spillway is planned for the first years of the life of the New Polishing Pond allowing keeping the final discharge point at the location of the existing polishing pond. Latter in the process the emergency spillway will be moved to dyke EM-A.

The design of the New Polishing Pond was advanced to the detailed stage as it is a key element in the closure of the East Malartic tailings site. A detailed borrow search was completed in the spring/summer 2008 with the aim of identifying viable sources for low permeability, sand and gravel materials. The search and subsequent laboratory analyses allowed completing the design effort and the design of the dykes was adjusted to what is believed to be the best strategy taking into account the available resources. A complete report on the New Polishing Pond including the details of the borrow search was issued in August 2008.

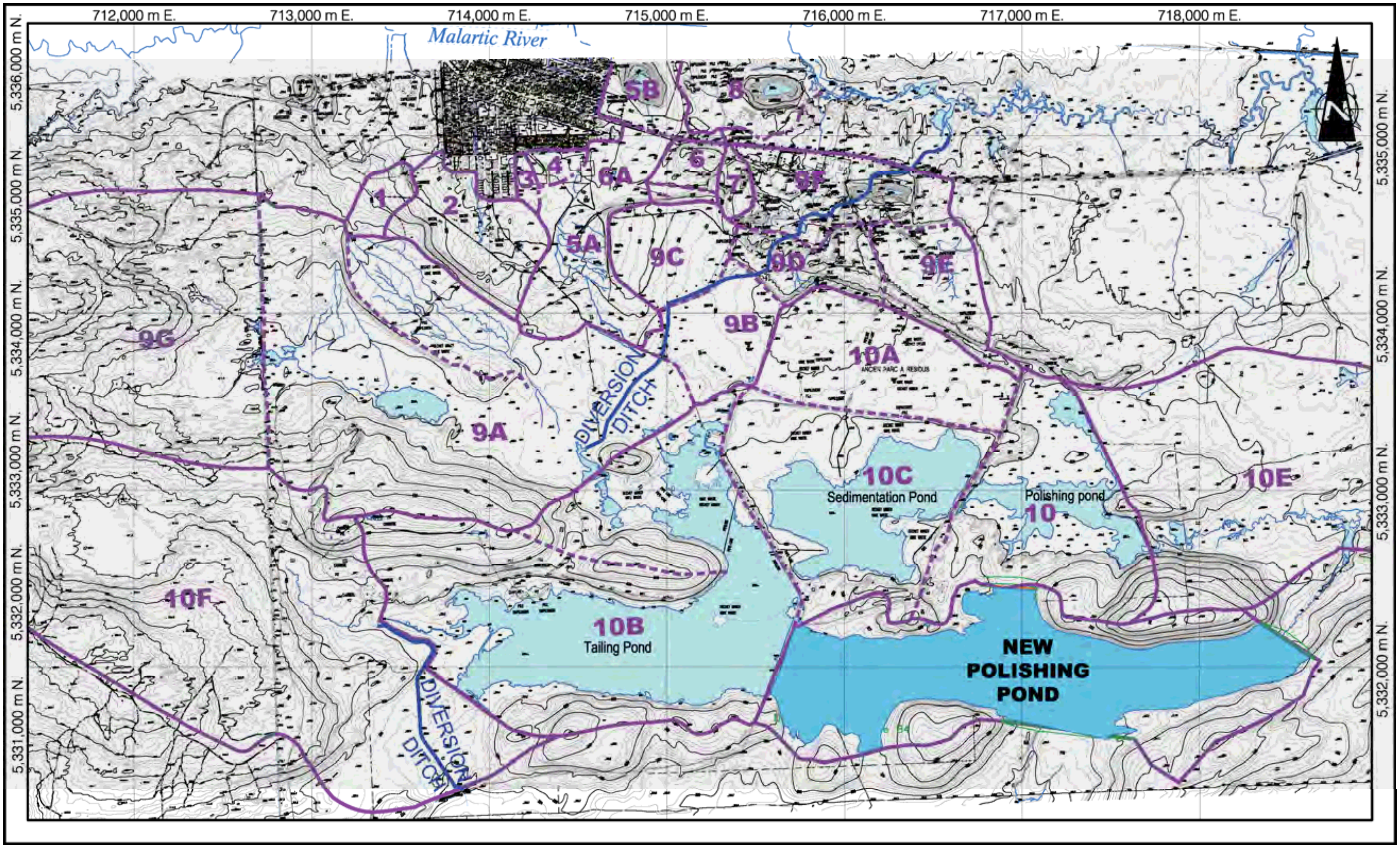
7.3.3 Surface Water Management

The water balance analyses concluded that there might be a certain risk of insufficient water for operation under pre-development conditions (before the existing tailings ponds are covered). Therefore, the aim of the water collection system is to maximize the collection and storage of runoff from local watersheds.

The proposed water collection system is comprised of two major ditches and a collection pond (Figure 7.6). The proposed North Storage Pond will collect runoff from the catchment area north of the site (sub-watershed 9G). The North Storage Pond will be used to supply water to the plant site and the excess water will be pumped to the proposed South Ditch. The proposed North Ditch acts as an overflow structure to prevent the North Storage Pond from inundating the conveyor belt. In the first phase of the mine plan the North Ditch will convey overflows to the existing North Diversion Ditch. The South Ditch will convey runoff from the sub-catchment area 10F associated with the external watershed that was previously associated with the South diversion and pumped flows from the North Storage Pond into the New Polishing Pond, which will be the major water supply pond for the mill.

The emergency spill pond is located north of the mill complex in a natural depression (Figure 7.3). This pond serves as a containment for emergency spills from the process and heavy mobile equipment maintenance areas. A permanent pumping station will be installed at the emergency spill pond to pump any contained spills back to the mill.

Figure 7.6: Watershed Locations



Legend: ______ Watershed Limits.

Estimates of flood water flows for sub-catchments 9G and 10F were required for preliminary design of the surface water collection and drainage system. While the collection and storage of data from the Environment Canada regional flow monitoring system provides adequate information for the preliminary estimation of annual runoff, the monitored catchments are significantly larger than the catchments at the East Malartic site and cannot be directly used to estimate local watershed flood flows. Computer modeling was therefore used to provide preliminary flood flow estimates. Hourly precipitation and temperature data for Val d'Or from 1961 to 1995 obtained from Environment Canada were used as inputs in the QHM computer model. QHM is a Windows-based continuous watershed quantity and quality simulation model that can simulate surface runoff, base flow, winter runoff, soil freeze-thaw and snowmelt processes. Model parameters were selected from the QHM software manual and from professional experience with similar projects. A field program will be implemented in 2009 to collect flow data for local catchments in and around the East Malartic site. This program will provide information for further calibration and validation of the QHM model and will be used in the future to fine-tune the design of the water collection system.

8.0 INFRASTRUCTURE AND SUPPORT FACILITIES

8.1 Electrical and Communication

8.1.1 120 kV Electrical Transmission Line

The electrical power for the Canadian Malartic project will be supplied from the existing Hydro-Québec 120 kV Cadillac main substation. A new 120 kV electrical transmission line approximately 19 km long will be built. Power demand for the entire project is expected to be 85.3 megawatts including all mill and mine support facilities.

The 120 kV electrical transmission line will be of single circuit type consisting of one circuit of three conductors. Most of the towers will be wooden pole structures, while angle and anchor supports will be galvanized steel towers. The average distance between towers would be 200 m. The width of the right of way for the deforestation of the new transmission line will be 46 m.

In October 2008, Hydro-Québec completed their preliminary assessment for the construction of the new transmission line. Hydro-Québec has confirmed that the construction of the line will be completed for September, 2010. The power line will be the property of Hydro-Québec who will be in charge of the engineering, the construction and the commissioning.

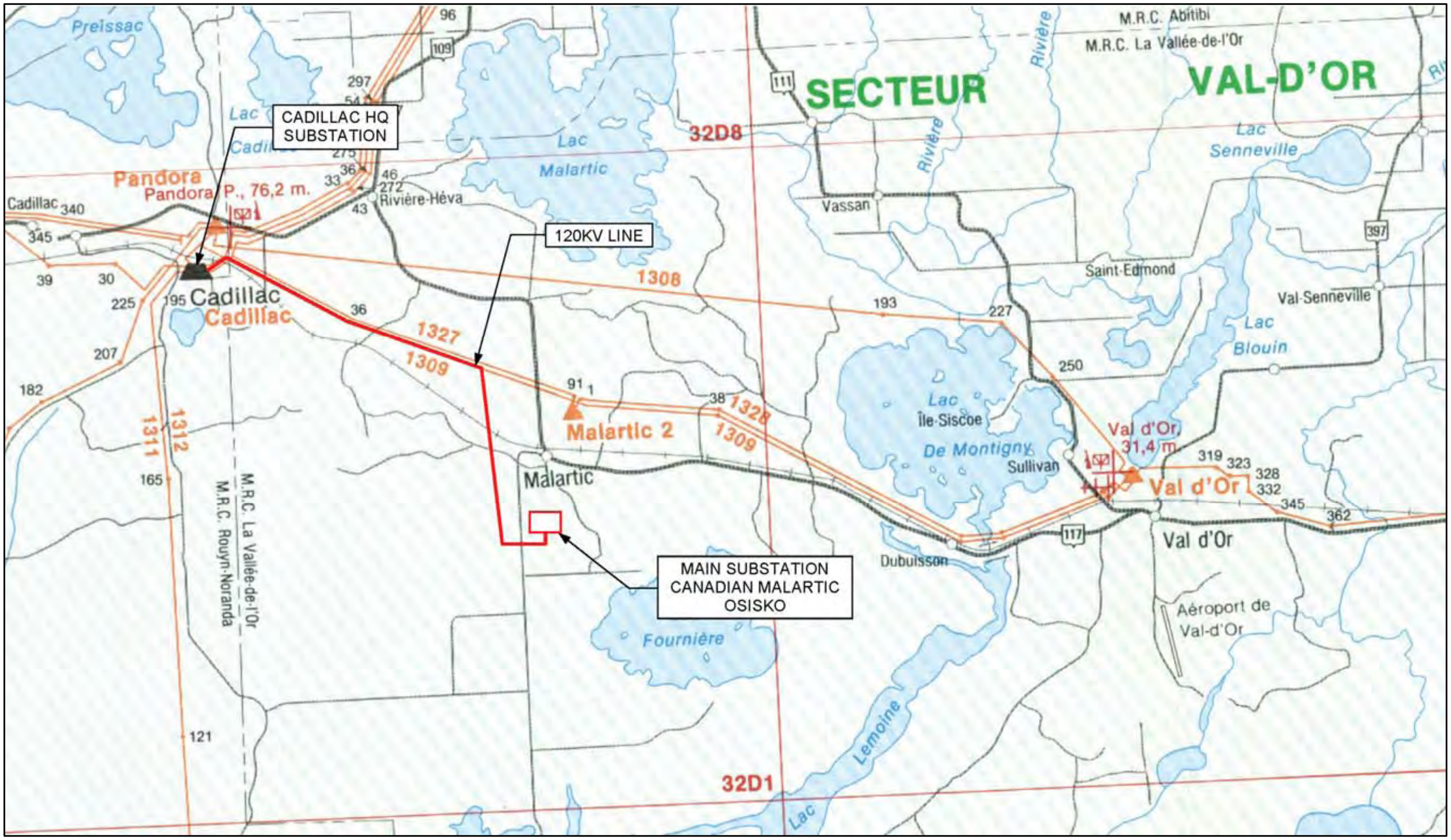
Figure 8.1 shows the proposed 120 kV electrical transmission line location.

8.1.2 Main Substation

The plant main substation will be located next to the process electrical room, close to the biggest loads, the SAG mill and ball mill motors.

The main substation includes all the equipment required by Hydro-Québec to connect the new Canadian Malartic complex to the local grid. Because of the long term delivery, a purchase order has already been awarded for the three 120-13.8 kV power transformers each with a capacity of 42/56/70 MVA. Loads will be distributed to each of the three transformers at an average of 28 MW each. If a transformer failure occurs or if maintenance of one of the transformers is required, the load will be redistributed to the two available transformers in order to guarantee continuity of production.

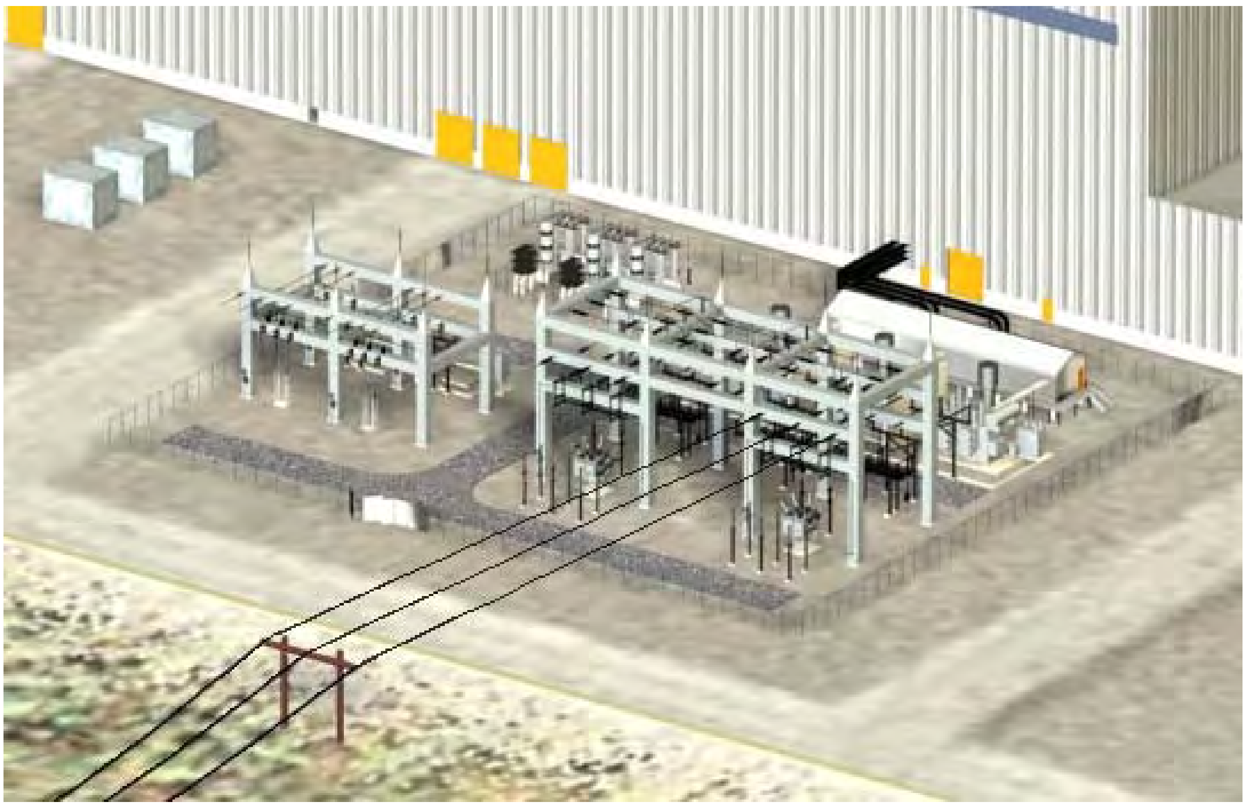
Figure 8.1: 120 kV Electrical Transmission Line Routing



The main substation includes a ''prefab'' building for the 13.8 kV distribution to the entire site. Major loads are fed from this building, including the concentrator service loop, the SAG and ball mill motors, the mine loop, the 5 kV transformers, etc.

Figure 8.2 illustrates the main substation in isometric view.

Figure 8.2: Main Substation - Isometric View



8.1.3 Site Power Distribution

One large electrical room located in the process plant has been included for process and service loads. This electrical room includes transformers and distribution equipment for the 5 kV and 600 V loads in two different areas.

Adjacent to this electrical room, located on the ground floor, are three smaller electrical rooms, one for each ball mill where the excitation and control system will be installed for each set of ball mill motors. There are two smaller distribution electrical rooms, one at each end of the concentrator, to feed all the small loads located in those areas.

For the mine loop, two other 25 kV transformers have been included for the electrical power feed to the open pit (shovels, pumping station, lighting, etc.). Local secondary substations consisting of pad mounted and mobile sub-units have been considered for the open pit operation.

A 13 km long 25 kV electrical distribution network covering the site is also required to feed the following areas:

- Crushing plant
- Reclaim water pumping system
- Effluent treatment plant
- Fresh water pumping system
- Explosives plant.

8.1.4 Emergency Generators

Three emergency generators of different power and voltage have been included and are planned to be installed at strategic locations to feed the loads requiring power during a power outage or during maintenance on the 120 kV line. Emergency power will be required for a minimum amount of building lighting and heating as well as selected process equipment requiring electrical power at all times.

One emergency generator of 1.25 MW at 4.16 kV will be installed close to the concentrator to keep the leach tank agitators in service.

A second unit of 800 kW at 600 Volts has been included to maintain the services of the thickener rake drive mechanism, the reagent mixing tank agitators, the gold room equipment and to keep some sump pumps running.

A third unit of 250 kW at 600 Volts will be installed close to the crusher building to secure this area during an electrical outage or maintenance.

8.1.5 Communication Systems

The plant area telephone network will be connected by optical fiber to the Malartic telephone exchange and will be used for telephone, fax, computer by IP telephone, etc.

The radio phone system will include a main repeater station including the main antenna which will be located close to the administrative office area and a second antenna will be located at the mine pit. This network will have several channels each dedicated to the following departments:

- Mine operations
- Mine Maintenance
- Process plant operations
- Process plant maintenance
- General services (warehouse/surface maintenance)
- Site security.

8.2 Public Road

A 2 km road widening program will be executed on the northern portion of the existing Lake Mourier public road in order to give safe access to employees and suppliers during mine operations.

Osisko is presently in discussion with the Québec Ministry of Transport to generate an execution plan. The work will consist mainly of enlarging the shoulder in both directions with gravel and prepare the approach to the mine site entrance.

8.3 Green Wall (linear park)

A buffer zone of 150 meters wide will be developed along the northern limit of the open pit to mitigate the impacts of the mining activities on the Malartic citizens.

Inside this buffer zone, a landscaped ridge will be built mainly made of rock and topsoil from the pre-stripping mining works.

This landscaped ridge will be 15 m high where the residents' concentration is higher and 5 to 6 m high in non-resident sectors. A security fence will be erected at the bottom of the ridge on the urban side in order to restrict access to the property. Soil and organic matter will cover the surface and then different types of vegetation will be planted such as shrubs trees and grasses.

8.4 Site Roads

On-site roads will be constructed to allow access to the following areas:

- Process plant
- Crushing plant
- Administration/Mining building
- New polishing pond area
- Effluent treatment plant
- Explosives plant.

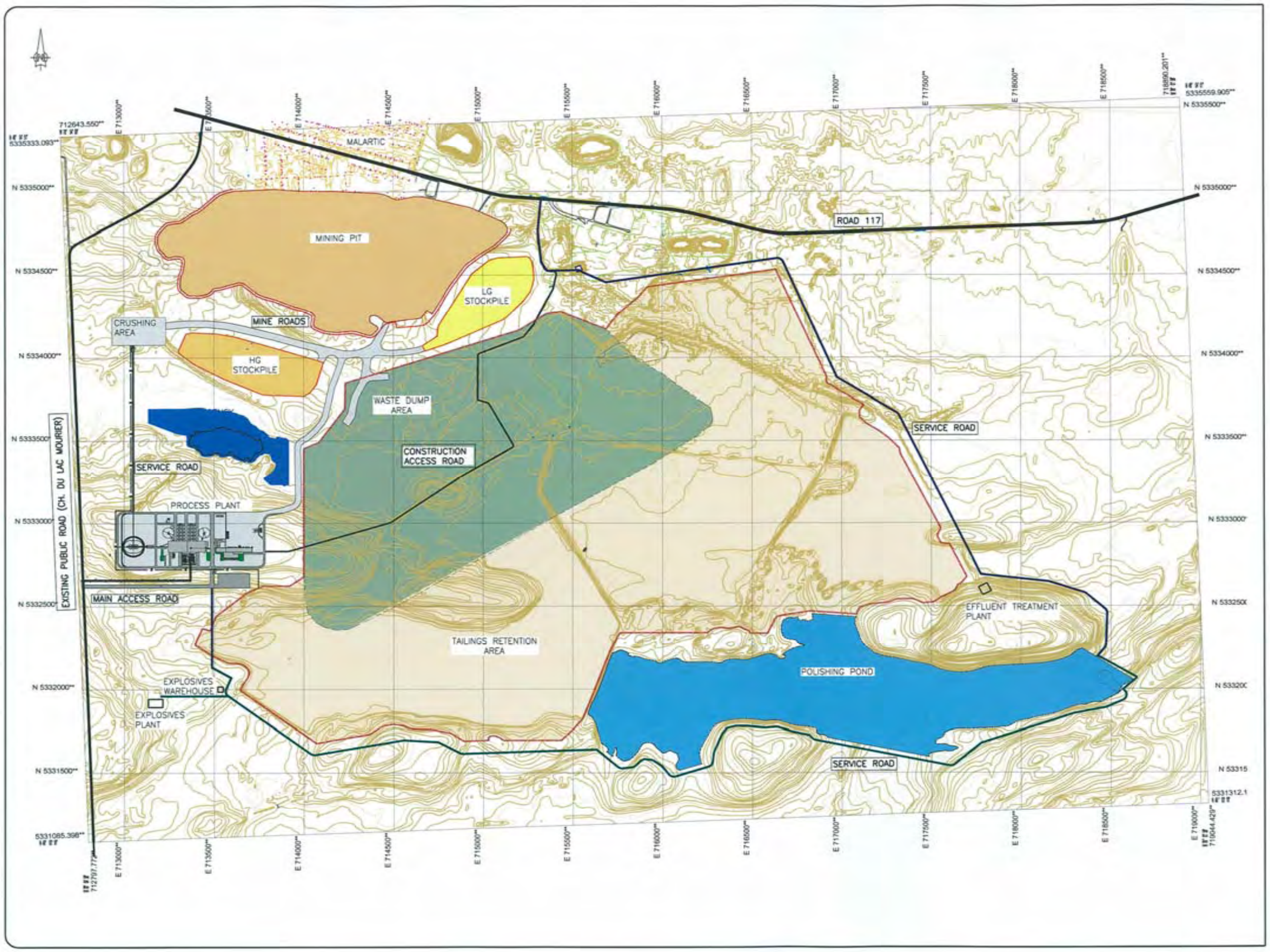
On average, the roads will be 10 meters wide and constructed with granular material from the site borrow pit. In total, approximately 12 km of road length will be constructed for the project. Osisko has already purchased a construction equipment fleet and borrow pit access roads are presently under construction.

During the construction period, the heavy traffic will use a temporary bypass road on site in order to eliminate equipment and merchandise transportation through the town of Malartic.

In the pre-production phase, the heavy mine mobile equipment will be used to construct the mine access road, including the waste dump and the tailings area.

Figure 8.3 shows the overall on-site road locations.

Figure 8.3: On-Site Road Locations



8.5 Main Security Control Gate

A control gate building will be erected at the main entrance of the site to supervise the personnel entrance and merchandise transportation. An automatic control gate system will be installed.

The security personnel will ensure visitors' registration and individual safety equipment distribution.

Furthermore, a fire protection system panel and a surveillance camera will be connected to the main control gate.

A parking lot of 400 vehicles will be built in front of the main control gate.

8.6 Administration/Warehouse Building

The administration/warehouse building will have two floors, each of which has 1,300 m^2 of space. On the ground floor, there will be a warehouse with shelving to maximize the storage space. The warehouse will have an interior door access to communicate with the mine heavy equipment maintenance shop. Furthermore, an outside warehouse facility will be erected with two mega-dome buildings of 800 m^2 each.

On the first floor of the administration building, there will be the following departments or services:

- Management
- General Administration, Accounting
- Geology
- Engineering
- General Services, Surface Support
- Environment, Health and Safety
- Human Resources
- Information Technology, Communication.

This building has been designed with a light structural steel frame with steel siding, flat insulated roofing and conventional interior finishing. The administration/warehouse building will be protected by a fire protection sprinkler system.

8.7 Mine Office/Truck Shop Building

Mine operations and maintenance personnel will be assigned to a building that is joined to the truck shop facilities. The mine office will be 3,000 m^2 and will include the following services:

- Mine Maintenance Staff Office
- Mine Maintenance Personnel Dispatch Center
- Contractor Offices
- Data Room
- Medical Center
- Training and Conference Room
- Mine Maintenance Lunch Room/Dry.

The capacity of the mine dry will accommodate 300 workers for the mine operations and maintenance department. A women's dry room has been planned. This building will also be protected by a fire protection sprinkler system.

The truck shop facilities will be 4,000 m^2 and will be used for the maintenance of heavy mining equipment. A total of 10 bays have been retained for the design and are planned as follows:

- One dedicated wash bay (equipped with a specialized heavy mining equipment washing system and water/oil separator)
- One dedicated welding bay
- Three preventive maintenance bays (lube bay)
- Two major repair and tire handling bays
- Two mine support equipment maintenance bays
- One mechanical shop.

This building has been designed with a light structural steel frame with steel siding, flat insulated roofing and conventional interior finishing.

This building is equipped with two overhead cranes for the following specific uses:

- 30 t/10 t capacity for heavy mining equipment maintenance
- 10 t capacity for the mechanical shop.

A tool crib, lube/oil distribution system and air compressor will also be installed in the truck shop.

Figure 8.4 shows the administration/mining building panoramic view.

8.8 Water/Sewage Infrastructure

Osisko will connect to the town of Malartic sewage and potable water systems. The sewage will be collected at the mine site and pumped in a 200 mm diameter buried high density polyethylene (HDPE) line to connect to the municipal grid located in the industrial park. The potable water will also be connected to the municipal potable water network. The 150 mm potable water line will be buried and will follow the routing of the sewage line.

Figure 8.4 : Administration/Mining Building

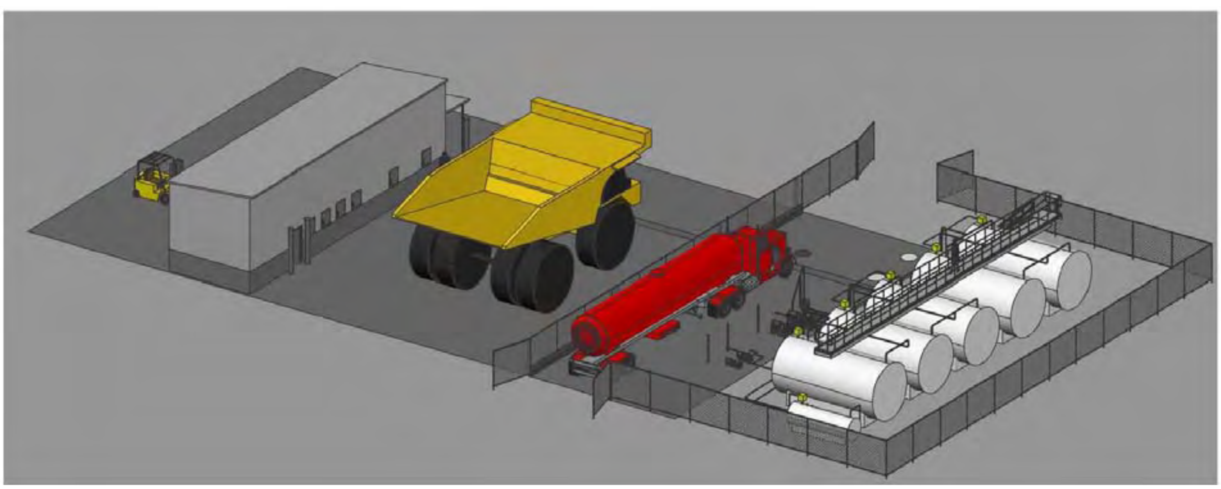
8.9 Fuel Storage Facilities

The fuel storage facilities are designed to have 250,000 litres of storage capacity and are located northeast of the truck shop. The system will include the following items:

- Five 50,000 litres capacity fuel ISO tank
- One 10,000 litres capacity gasoline ISO tank
- One concrete pad equipped with a transfer pump system to unload supplier fuel trucks
- Two fuel distribution points with electronic control card access systems for heavy mining and support equipment built on a concrete pad
- One gasoline distribution point with electronic control card access system for small vehicles built on a concrete pad
- Lubricant product storage facilities for all mining equipment equipped with fire protection sprinkler systems.

Figure 8.5 illustrates the fuel storage facility.

Figure 8.5: Fuel Storage Facility



8.10 Monitoring/Weather Station

A weather station will be installed and connected to the Osisko environmental department monitoring system in order to register climate data and to produce historical surveys.

From the meteorological station, the following information will be available on a daily basis:

- Temperature
- Air humidity
- Wind speed
- Wind orientation
- Atmospheric pressure
- Precipitation.

8.11 General Services and Administration

8.11.1 General

The General Services and Administration (G&A) group includes all personnel relating to senior management, accounting, payroll, human resources, surface support, health and safety, environment and telecommunication. All the departments and sectors not directly related to operations are included in the G&A.

The procurement and logistics group is responsible for sourcing suppliers, organizing transportation of all the supplies and goods, managing site inventory, negotiating prices and delivery conditions.

The surface support group manages the loading and unloading of trailers, and movement of material. The surface support group also takes care of the open pit and tailings pipeline pumping systems maintenance and operations.

The human resources department is responsible for all aspects pertaining to employee fringe benefits, recruitment, discipline management and labour agreements.

The administration sector includes senior management, accounting, public communication and community relations personnel.

IT and telecom are responsible for computer maintenance, networking and all communication systems as well as all radio maintenance, and more important the mine dispatch system.

The sustainability group is responsible for setting up and maintaining the highest standards in health and safety, and environmental matters. The various projects all in line with the ISO 14001 certification are of the responsibility of this group.

8.11.2 Manpower

The labor associated to the G&A is comprised of 77 (Table 8.1) employees required to execute administrative and support tasks; most of this workforce relates to the support of the operations: surface support, telecommunications, purchasing and warehousing being the sectors with the larger groups. In general, the management and administrative staff will work on a 40-hour per week basis, day shift only. Security and warehousing will most likely work a 12-hour shift schedule covering 24 hours per day as part of the support to the operations.

Table 8.1 : Services Manpower Summary

Area/ Position	Roster	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
ADMINISTRATION											
MANAGEMENT											
General Manager	Dayshift	1	1	1	1	1	1	1	1	1	1
Assistant	Dayshift	1	1	1	1	1	1	1	1	1	1
Sub-Total Management		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
ACCOUNTING											
Chief Accountant	Dayshift	1	1	1	1	1	1	1	1	1	1
Accountant/Bookkeeper	Dayshift	3	3	3	3	3	3	3	3	3	3
Sub-Total Accounting		**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
ENVIRONMENT											
Superintendent	Dayshift	1	1	1	1	1	1	1	1	1	1
Technician	Dayshift	2	2	2	2	2	2	2	2	2	2
Sub-Total Environment		**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**
PUBLIC COMMUNICATION											
Manager	Dayshift	1	1	1	1	1	1	1	1	1	1
Assistant	Dayshift	1	1	1	1	1	1	1	1	1	1
Sub-Total Public Communication		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
HUMAN RESOURCES AND INDUSTRIAL RELATIONS											
HR and Superintendent	Dayshift	1	1	1	1	1	1	1	1	1	1
HR Coordinator and agent	Dayshift	6	6	6	6	6	6	6	6	6	6
Sub-Total HR		**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**
HEALTH AND SAFETY											
H&S Superintendant	Dayshift	1	1	1	1	1	1	1	1	1	1
Preventionist	Dayshift	3	3	3	3	3	3	3	3	3	3
Nurse	Dayshift	1	1	1	1	1	1	1	1	1	1
Sub-Total H&S		**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**
MANAGEMENT											
General Services Manager	Dayshift	1	1	1	1	1	1	1	1	1	1
Sub-Total Management		**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
TELECOMMUNICATION											
Telecommunication Superintendent	Dayshift	1	1	1	1	1	1	1	1	1	1
Telecommunication Coordinator/Tech	Dayshift	7	7	7	7	7	7	7	7	7	7
Sub-Total Telecommunication		**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**
PURCHASING											
Purchasing Superintendent	Dayshift	1	1	1	1	1	1	1	1	1	1
Purchaser/Clerk	Dayshift	5	5	5	5	5	5	5	5	5	5
Sub-Total Purchasing		**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**
WAREHOUSING											
Warehouse Supervisor	Dayshift	4	4	4	4	4	4	4	4	4	4
Warehouse Clerk	Rotations	10	10	10	10	10	10	10	10	10	10
Sub-Total Warehousing		**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**
SURFACE SUPPORT											
Surface Support Superintendent	Dayshift	1	1	1	1	1	1	1	1	1	1
Foreman	Dayshift	2	2	2	2	2	2	2	2	2	2
Hoisting Equipment Operator	Dayshift	1	1	1	1	1	1	1	1	1	1
Instrumentation Technician/Electrician	Dayshift	2	2	2	2	2	2	2	2	2	2
Surface Support Mechanic/Welder	Dayshift	2	2	2	2	2	2	2	2	2	2
Lineman	Dayshift	2	2	2	2	2	2	2	2	2	2
Operator/Carpenter	Dayshift	6	6	6	6	6	6	6	6	6	6
Pomp Man	Dayshift	3	3	3	3	3	3	3	3	3	3
Sub-Total Surface Support		**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**
SECURITY											
Security Coordinator	Dayshift	1	1	1	1	1	1	1	1	1	1
Security Agent	Rotations	5	5	5	5	5	5	5	5	5	5
Sub-Total Security		**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**
Grand-Total		**77**	**77**	**77**	**77**	**77**	**77**	**77**	**77**	**77**	**77**

8.12 Organizational Structure and Operating Policies

8.12.1 General

The workforce requirements have been evaluated considering the various phases of the project. As an example the pre-production year workforce mostly includes administration and services supporting activities such as site preparation, as well as the creation of a small technical group allowing for the project progression. During operations the majority of employees that will be hired to work at the Canadian Malartic project will come from Malartic and the surrounding cities and towns of the Abitibi-Timiskaming region.

8.12.2 Operations Workforce

The workforce requirements for Canadian Malartic operations over the proposed mine life will average 464 employees. A summary organization chart of the operations workforce is presented in Figure 8.6.

The management team of the Canadian Malartic operations will consist of regional employees. Local employees are expected to have responsible positions in the Canadian Malartic organization as the project will grow and are expected to progress further on within the company. The project is very well positioned along highway 117 (Transcanadian) just south of Malartic. The minesite is in close proximity to the three major commercial/residential centers in the Abitibi-Timiskaming region, namely Val-D'Or, Rouyn-Noranda and Amos.

8.12.3 Project Policies

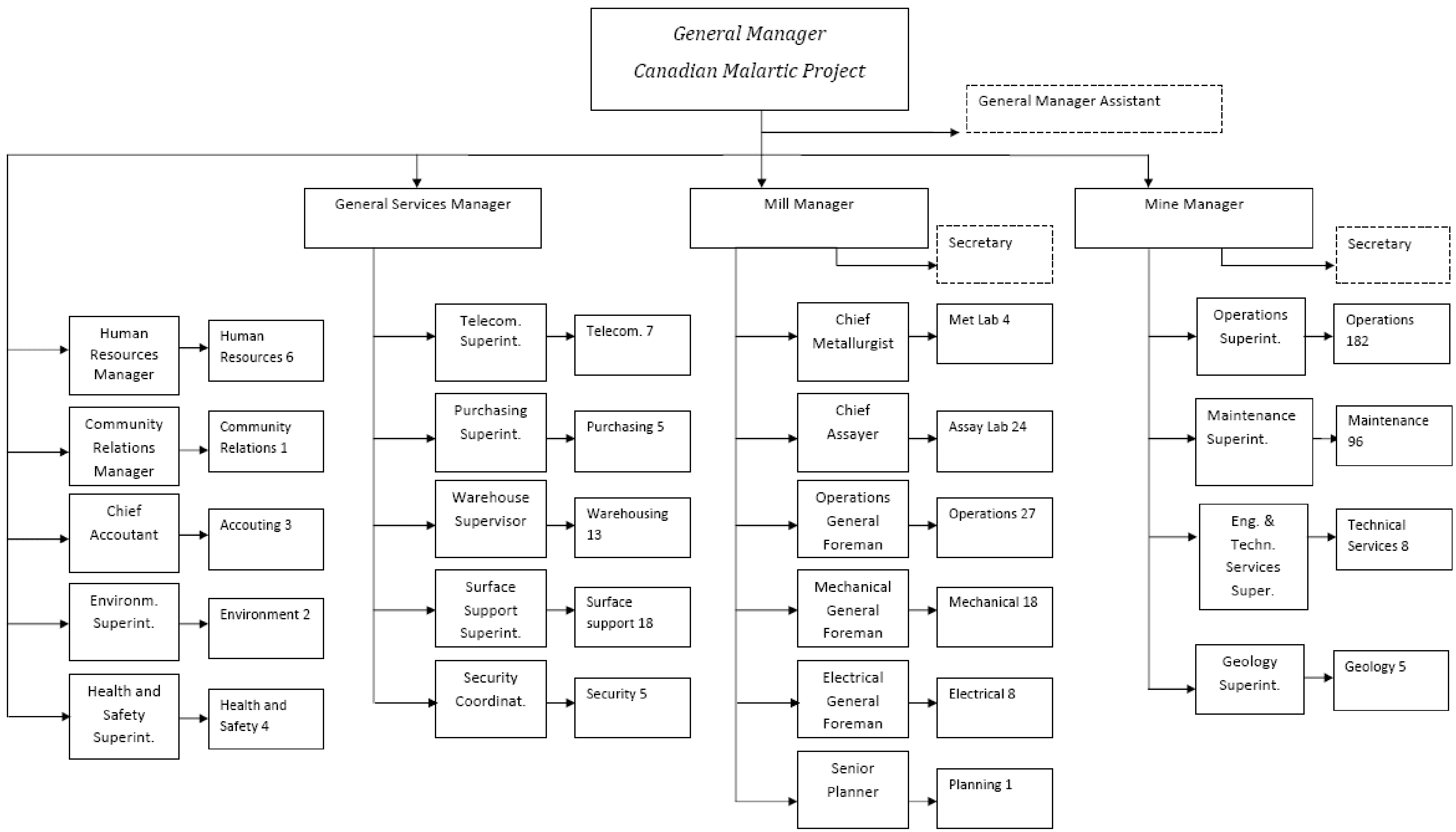
Osisko has already elaborated and put in place operating policies for the project. Efforts have been made to ensure that the Canadian Malartic project is developed in accordance with the interests of both the community and the employees. The following sections outline Osisko's most valued policies in environment, health and safety, human resources and community relations.

8.12.3.1 Environmental Policy

Environment is one of Osisko's top priorities. All of the company's activities focus on minimizing environmental impacts and Osisko works in close collaboration with the community and governmental authorities regarding environmental matters. Management is committed to developing activities in a responsible way by using innovative approaches, all in a sustainable development perspective. Striving for continuous improvement, Osisko will aim at achieving ISO 14001 standards of environmental management.

In order to achieve high environmental standards, employees are expected to play a significant and active role by insuring that their activities are developed in accordance with company policies and respect for the environment.

Figure 8.6: Summary Organization Chart



8.12.3.2 Health and Safety Policy

Osisko is commited to protecting its employees and also the health and safety of the surrounding communities. Procedures have been put in place to insure that each employee receives the required training and information regarding a safe and healthy execution of his job. It is also a top priority to make every workplace safe and to encourage employees to report any irregular or hazardous situations. Continuous improvement is emphasized and all health and safety issues are discussed making sure that employees play their active role in order to create a safe working environment.

8.12.3.3 Human Resources Policy

Osisko promotes the establishment of a positive working environment, encouraging employee's participation by sharing their concerns and promoting a positive and constructive dialog. To insure employee development within the company, training and continuous education will be implemented. Moreover, human resources policies include competitive wage conditions and a complete fringe benefits package in order to attract and retain the best workforce possible.

8.12.3.4 Community Relations Policy

The Osisko community relations team has insured since the project inception that the local and regional population was kept well informed about the project evolution. Public meetings were and are still held periodically on matters of concern such as the relocation program and the environmental impact assessment study. A constructive dialog has always been encouraged with the community. Osisko has a policy of being transparent in all its actions.

The corporate philosophy for Canadian Malartic is to promote procurement from local and regional suppliers thereby increasing the economic return to Québec.

Moreover, the company has created and put into force a sustainable development fund, the ''Fonds Essor Malartic Osisko'' (FEMO) encouraging personal participation in the development and well being of the community of Malartic. The FEMO is investing in projects that improve the quality of life for the population of Malartic and also the region.

9.0 COMMUNITY RESETTLEMENT

9.1 Neighbouring Community and Adjacent Urban Properties

The Canadian Malartic property is located in the vicinity and directly underneath the southern portion of the town of Malartic. Historically a mining and forestry town, Malartic numbers about 3,500 residents. The area influenced by the Canadian Malartic project open pit mine includes approximately 700 residents who will be directly affected and another 500 residents of the town who may be indirectly affected.

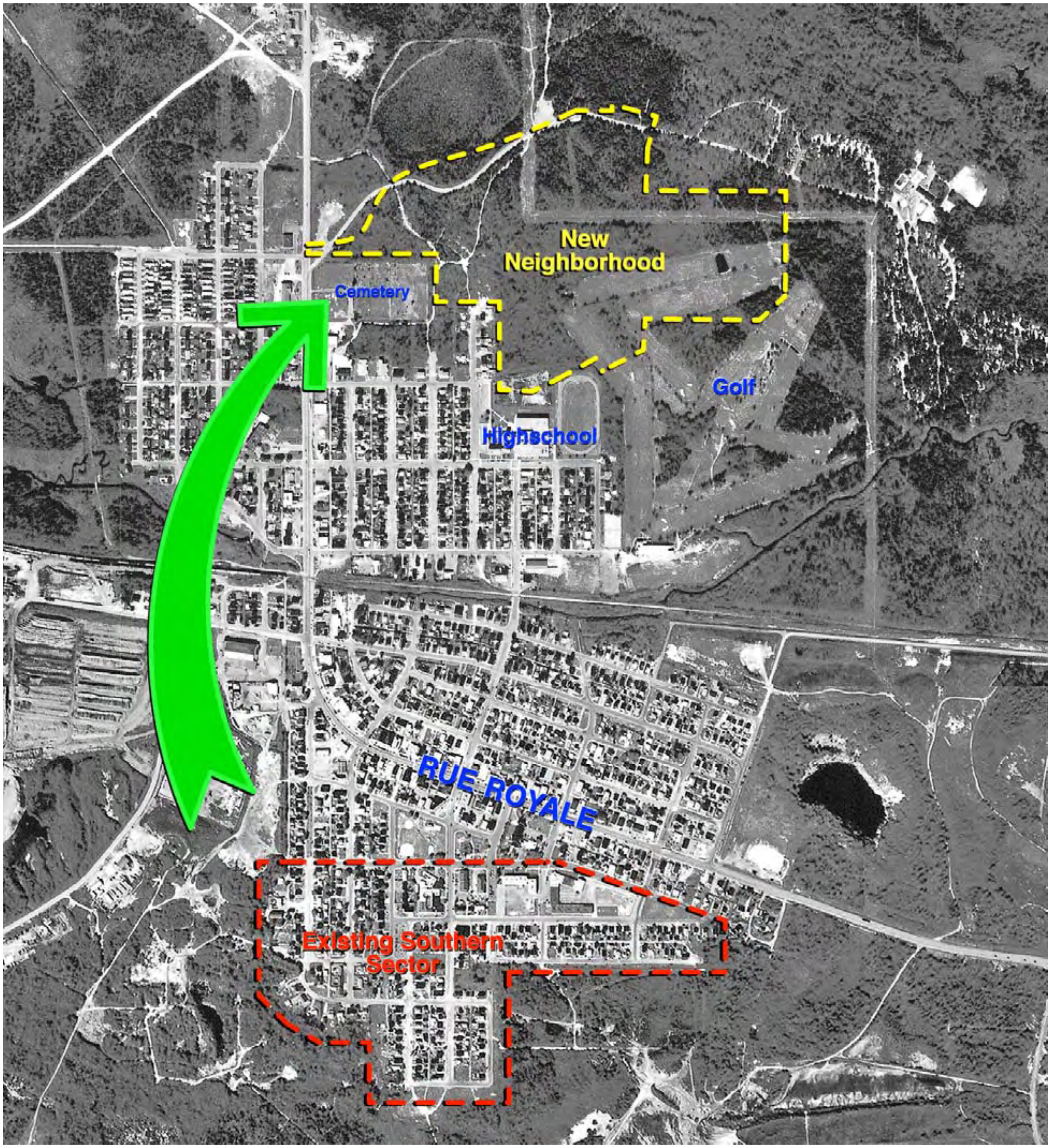
The original definition of the impacted area (March 2006) was restricted to the area south of rue Frontenac, a block to the south of the dashed line indicated in red (Figure 9.1). This area did not include the institutional buildings of rue de la Paix. In November 2006, Osisko announced to the institutional building owners that, due to positive exploration results, the company would propose to build new facilities for them in the northern part of town.

The most recent revision of the area of impact was in June, 2007 when the residences along the eastern end of rue de la Paix were included in the resettlement at their request, to ensure that their front doors would not open onto the Canadian Malartic property.

Community Development (CD) work has progressed since 2006 with the mission to provide Osisko and the town of Malartic with a sustainable development initiative for the relocation of a total of 205 residential properties and 5 institutional buildings from the vicinity of the planned open pit. The CD program mission can be defined as the information, education and involvement of the community on one hand, and the planning as well as the relocation and construction of real estate assets on the other, to create the New Malartic.

Osisko and the town of malartic founded a Community Consultation Group (CCG) bringing together the company, the town of Malartic and representatives of the residents as a consulting body to direct communications, to address requests and complaints, to draft the relocation process and review urban planning relocation designs. The CCG provided support for the effective communication necessary to enable the operations of the exploration team to allow, for example to receive the necessary approval from the population to allow the exploration drilling to proceed in the urban area of Malartic during 2006 and 2007. The CCG is currently active, providing a forum for the opinions and questions of residents, meeting bi-weekly, with every second meeting being open to the public.

Figure 9.1: Impacted Area of the Canadian Malartic Project



9.2 Golf Course

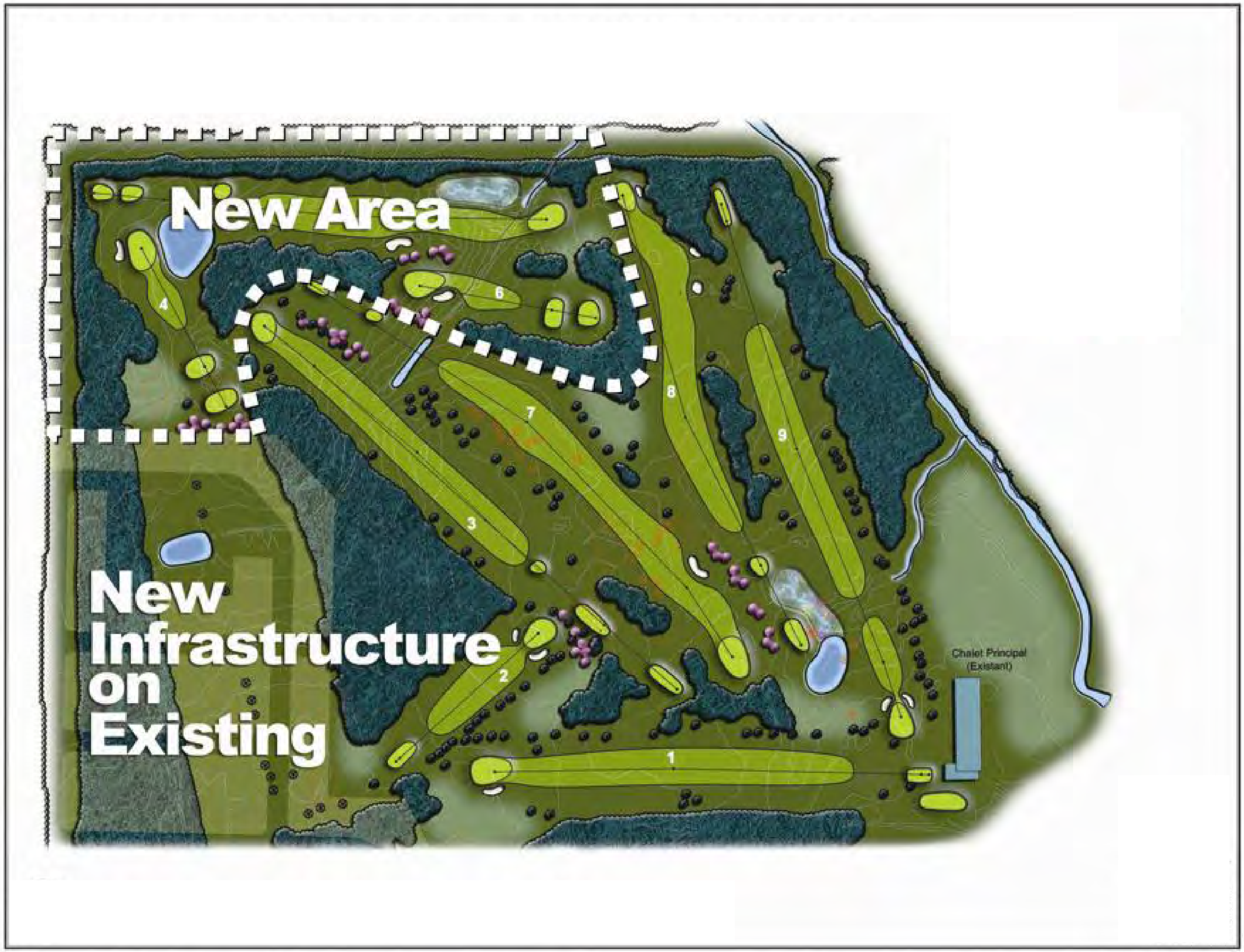
To allow for construction of the new neighbourhood adjacent to the existing urban area (Figure 9.1) an area of approximately 10 hectares was acquired from the Malartic Golf Club to anchor the southern portion of the new neighbourhood. This area houses a little more than half the 190 developed resettlement lots.

Achieving this acquisition required the relocation of three existing holes on the course and the modification of three other holes. Holes 2, 3, and 4 were therefore relocated to the eastern edge of the property owned by the golf course (Figure 9.2).

Up to date features such as filtration ponds, to diminish the presence of chemical fertilizers in the effluent, were included in the design to minimize the effect on the environment. An existing water feature for the golf course was also introduced into the new neighbourhood, allowing for the creation of a park. At the end of October 2008, 90% of the construction was completed for this work.

The agreement with the Malartic Golf Club includes an option for the acquisition in the future by Osisko of further portions of the Club’s real estate holdings as required.

Figure 9.2: Re-configured Golf Course



9.3 Municipal Infrastructure

The residential urban design team bringing together Osisko, the town of Malartic, the CCG, urban designers and civil engineers directed the analysis of potential relocation sites around Malartic, allowing for the ultimate selection in 2007, by City Council and residents. The expansion area selected is located to the North and East of the existing urban area.

Analysis and preliminary urban design towards the selection of a new site was begun in 2006. Thirteen (13) separate areas of expansion around the town of Malartic were studied and rated for their potential as a relocation site. One area was chosen which links to the existing street grid and has the greatest expansion possibilities for the future, potentially 400 lots (Figure 9.3).

Detailed urban design proceeded in this area, which was conceived to suit the existing southern sector lots and their different characteristics. Lot sizing, orientation, location with respect to green space and proximity to existing neighbours were of great importance for most, while architectural homogeneity was least important.

Infrastructure construction of the “demonstration street” of 23 new lots began in the fall 2007. Construction of the 190 new lots has continued since that time, and is slated for completion in 2008 with road surfacing to be completed over the next 12 months. The relocation of homes follows the completion of the infrastructure closely, according to the lot selection of the homeowners.

Figure 9.3: New Malartic Neighbourhood

9.4 Institutional Buildings

The Institutional design teams have been reviewing client needs, selecting suitable sites for the new construction and completing preliminary and final designs for the institutional buildings.

Five existing buildings are being replaced with six new buildings and additional social housing is being constructed to replace apartment units to be demolished. The institutions and their replacements, located in the vicinity of the new neighbourhood, are as follows (Figure 9.4):

- École St-Martin, a 350 student elementary school, to be replaced by a 400 student school, ‘’École des Explorateurs’’, anchoring the new neighbourhood.

- École Renaud, including Centre Le Trait d’Union, an 80 student adult learning center, a 350 seat auditorium and Community group offices, to be replaced by two new structures, a 104 student Adult Learning Center and a Culture and Recreation Center adjacent to the existing high school.

- CHSLD Villa St-Martin, a 57 bed long term care hospital center to be replaced by a 60 bed facility adjacent to the existing Malartic hospital.

- CPE Bambin et Câlin, an 80 infant and children daycare will be housed in a purpose-built facility at the geographical center of town, to maintain a proximity to its client base.

- Résidence Germain Paquette, a 20 unit elderly social housing complex will be rebuilt in proximity to other social housing, one block west of the rue des Pins entry into the new neighbourhood.

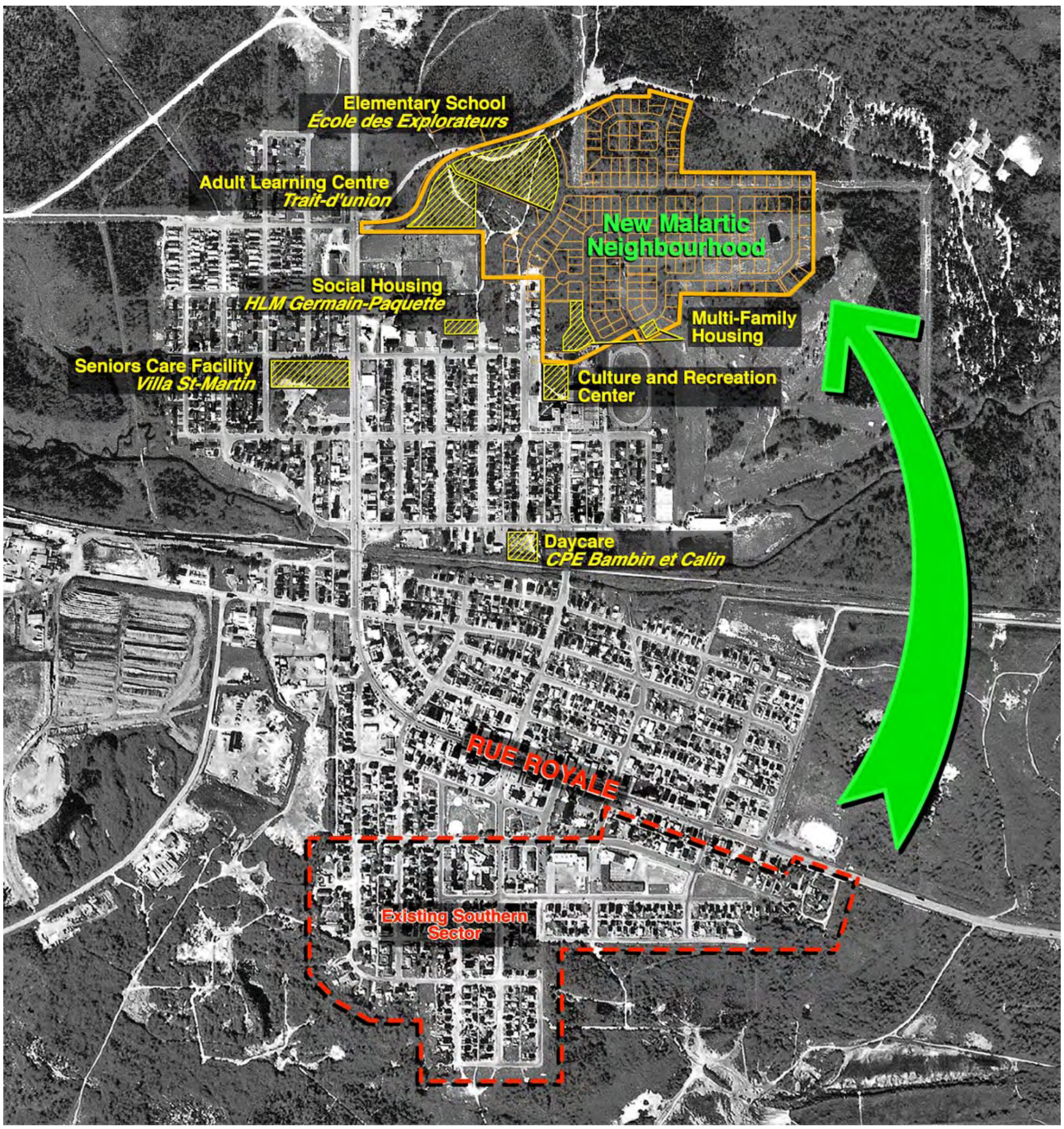
- Immeubles Roc d’Or will be a newly constructed institutional complex of seven buildings containing four living units each. These will replace apartment units which need to be demolished in the southern sector.

The replacement of some of the buildings provided opportunities for a new urban definition, along with efficient use of these assets. This is a welcome change for existing institutions, some of which had been placed in reused buildings not specifically suited for them. The new institutions are one of the key benefits for the residents from the Canadian Malartic project.

In urban terms, a concentration of civic buildings is being created along 4th avenue and its extension Dargis-Ménard avenue, along with the northern entry to the neighbourhood along the Chemin du Camping. With the relocation of the southern neighbourhood and the construction of these amenities, the center of gravity of the town is moving northward.

The relocation of the southern sector should enhance the market value of the residents as the sector was affected by ground instability and potential closure and thereby reducing the market value of the homes.

Figure 9.4: Integration of the New Neighbourhood Into the Existing Town



9.4.1 Elementary School – École des Explorateurs

Anchoring the new neighbourhood, the elementary school is located on a bluff at the corner of Dargis-Ménard Avenue and Chemin du Camping, in the northwest corner. Currently under construction, the building is slated for occupancy in September 2009.

The School to be replaced is the property of the regional school board, the “Commission Scolaire de l’Or et des Bois” (CSOB). It was built in 1957, renovated in 1970 and is home to 305 students in 2008. The New School, representing nearly 5,000 m^2 of area, is to be constructed along up-to-date Ministry of Education, Sports and Leisure (MELS) norms and standards with a capacity of approximately 400 students. It houses 3 kindergarten classrooms, 14 primary classrooms and 1 multipurpose room.

The L-shaped structure defines the corner, with 2-storey common spaces at the hub and classroom wings on either side, 1-storey for kindergarten and 2 storeys for primary (Figure 9.5). The hub is defined by a simple box steel structure, while the classroom wings expose their engineered wood shed roof structures. The classroom wings are clad in composite concrete panels and the hub is clad in concrete block and metal panels.

The existing St-Martin School has been classified as a ‘’Brundland Green School’’ and it is likely that the new school will be designed and constructed based on the same classification.

Figure 9.5: Elementary School - École des Explorateurs

9.4.2 Adult Learning Facility - Trait D'Union Center

The Trait-d'Union Centre is located at the northwest edge of the new neighbourhood on the Chemin du Camping, to the west of the primary school. It holds a prominent position as the gateway to the new sector along its north access road. Currently under construction, the building is slated for occupancy in September 2009.

The Centre is currently housed in the 1950's Renaud school, which was transformed for community use by the school board when its definition as a traditional school was no longer needed. The Centre is constituted of 9 multi-purpose classrooms and is designed with a global capacity of 104 students, or "50 full-time equivalent" students by Ministry of Education, Sports and Leisure (MELS) norms and standards (Figure 9.6).

The steel and concrete structure is arranged with a 2-storey classroom block along the street and a one-storey administrative and service block along the forest side. A glazed spine joins the 2 blocks and allows for the introduction of natural light into the core. With an exposed concrete base, the building is clad in metal panels in majority, with the spine clad in composite wood. Geothermal heating and cooling is being introduced in this building in partnership with the CSOB.

Figure 9.6: Adult Learning Facility - Trait D'Union Center

9.4.3 Culture and Recreation Center

This facility, an extension of the existing Le Tremplin High School, is located directly to the south of the new neighbourhood, at the corner of Rue des Pins and 4th Avenue, on a parcel of land owned by the CSOB.

Renaud school in the southern sector housed the non-educational components of this construction along with the adult learning facility, Centre le Trait d'Union. This new amenity is to be shared by the students of the high school and the community at large, and be co-owned by the CSOB and the town of Malartic. It is slated to become a focus of activities in the newly developed northern sector.

The approximately 1,750 m^2. building houses a 300-seat multi-configuration auditorium, 12 multi-purpose rooms for the use of community organizations and all ancillary spaces. The 2-storey steel structure has a brick and glass envelope and uses the original high school materials and colours (Figure 9.7). Its functions are arranged around the multi-use auditorium with the remaining spaces in a U-shape around it.

Figure 9.7: Culture and Recreation Center

9.4.4 Long Term Care Hospital Center - CHSLD ST-MARTIN

CHSLD St-Martin, a 60-bed Long-Term Care Hospital Center is to be constructed adjacent to the Malartic General Hospital, on land owned by the institution. Along with an ongoing renovation and expansion of the General Hospital, it will create a new Malartic Health Centre, located on rue Royale, 4 blocks west of the start of the new neighbourhood.

The facility, to be reconstructed in line with Ministry of Health (MSSS) standards for “Quality Living Environments for People in Long-Term Care Hospital Center” will replace La Villa St-Martin, a 57-bed adapted facility built in 1970. This facility is owned and operated by the Regional Health Centre (CSSSVO). The 5,700 m^2 construction will accommodate 60 patients divided in four 15-room treatment units with their own living and dining quarters along with treatment and administration facilities. A Day Centre will also be built to welcome outpatient services. The Centre is equipped with full kitchen facilities for residents and staff, and convivial spaces family and friends.

Directly to the north of the existing hospital, the CHSLD is linked to this facility to allow for movement of staff. The 2-storey steel structure, designed to maintain the greatest number of residents at Ground level, is clad in glass and brick (Figure 9.8). Care has been taken to integrate the 2 adjoining structures internally and externally by the team of architects and engineers involved on both projects.

Figure 9.8: Long Term Care Hospital - CHSLD St-Martin

9.4.5 Daycare Center - Bambin et Câlin

CPE Bambin et Câlin– an 80-infant and children Childcare Center facility – is to be reconstructed in the existing Malartic urban area on rue Harricana at the end of 4th Avenue on land acquired from the Town of Malartic. By choosing this location over the new neighbourhood, the CPE wished to maintain a close proximity to its main user group. Currently under construction, the building is slated for occupancy in August 2009.

This new purpose-built facility for the CPE will replace a transformed school administration building from the 1960's. The CPE is a private facility managed by a Board of Directors, elected from amongst parents and staff, and supported by provincial subsidies. Operations and facilities are reviewed and approved along Ministry of Families standards.

The daycare will accommodate 64 children, and 16 infants in a 2-storey structure of 1,700 m^2 overall (Figure 9.9). Apart from common and service spaces, the area is divided into separate playrooms for 8 children and an educator, with the infants receiving twice the care. The indoor gymnasium and outdoor play space are important to this facility, and their design required special care.

The building is made up of a steel and concrete structure clad with composite wood and resin panels. The definition of volumes and panelized skin allowed for the playful expression of building blocks and colours to provide the CPE with an identity.

Figure 9.9: Daycare Center - CPE Bambin et Câlin

9.4.6 HLM - Germain Paquette Residence

Social Housing – 20 units for the elderly– are to be constructed along Québec Housing Society standards in the existing urban area, east of the new neighbourhood entrance at Rue des Pins.

Adjacent to two existing facilities managed by the Municipal Housing Authority, this new building will help reinforce a sense of community for all HLM residents. The natural setting will also provide a peaceful and accessible environment for the residents.

Replicating the facility in the southern sector, this residence consists of eighteen 1-bedroom units and two 2-bedroom units, of standard dimensions, with individual balconies or decks and communal spaces (Figure 9.10).

Figure 9.10: Social Housing for the Elderly - HLM Germain Paquette Residence

9.4.7 Social Housing - Osisko Subsidized Multi-Family Housing

Subsidized housing – 28 units – are to be constructed along market standards to replace apartment units acquired by Osisko which cannot be economically relocated. The first residents of these 2-bedroom apartments will be relocated tenants. These apartments are presently under construction and occupation is scheduled for April 2009.

On a large lot at the corner of Rue des Pins and Miquelon street, the first six of these buildings are under construction (Figure 9.11). The second cluster, a block away, is to be anchored by one building and expanded with up to three more by a not-for-profit housing corporation.

Osisko has come to an agreement with a local not-for-profit housing corporation, Immeubles Roc d'Or, to purchase and take over management of the pre-fabricated structures when installation is completed.

Figure 9.11: Subsidized Apartment Buildings

9.5 House Relocation

The relocation of the southern neighbourhood of Malartic involves 205 individual properties. As described previously, the creation of a new neighbourhood to the north of the town represented the main option for homeowners, while a purchase offer by Osisko was the other.

Relocation to the new neighbourhood for the residents involves close coordination with Osisko. Negotiations have been ongoing since June 2007 to allow for the selection of lots by the home owners. Specific lots have been selected and finalized for relocation by nearly 120 owners.

Most of the remaining owners chose to accept the purchase offer from the corporation. In October, 2007, a purchase offer by the company was made public, for all owners desiring to sell their property instead of having it relocated. This option has been accepted by more than a third of the homeowners, while the remaining owners are tending towards this choice.

The corporation elected to relocate some of the better-quality homes it acquired to the new neighbourhood. These will be put back on the market to provide some choices in home acquisitions for residents, existing and new. New serviced lots will also be put on the market for new home construction. The establishment of the south neighbourhood relocation is planned as follows:

- Approximately 150 homes to be relocated (60 relocated by 10/2008):
  - Approximately 120 are resident-owned
  - 30 Osisko acquisitions.
- 77 homes acquired or purchase agreements signed. Negotiations with the eight (8) remaining households are on-going:

Construction of infrastructure in the new neighbourhood is well advanced, making available 100 serviced lots and planned completion of the residential lots in 2008, with final street surfacing to proceed in 2009.

9.6 South Neighbourhood Rehabilitation

The relocation of the south Malartic neighbourhood will necessitate the rehabilitation of all public services in the sector to allow for the removal of some infrastructure and addition of specific components to enable the municipal network to function properly.

A thorough analysis of the different networks has begun in October 2008, as the effect of a concentrated change affecting 20% of properties in town needs to be calculated and solutions designed and built.

A number of infrastructure components will be removed and returned to the town of Malartic, while public utilities will be mandated to remove their networks. The networks to remain are to be evaluated with their different owners to ensure service to residents is maintained and possibly made better with the elimination of old components.

Due to the increased traffic load during the life of the mine, a new traffic light configuration will be implemented on the main street of the town of Malartic.

10.0 ENVIRONMENTAL CONSIDERATIONS

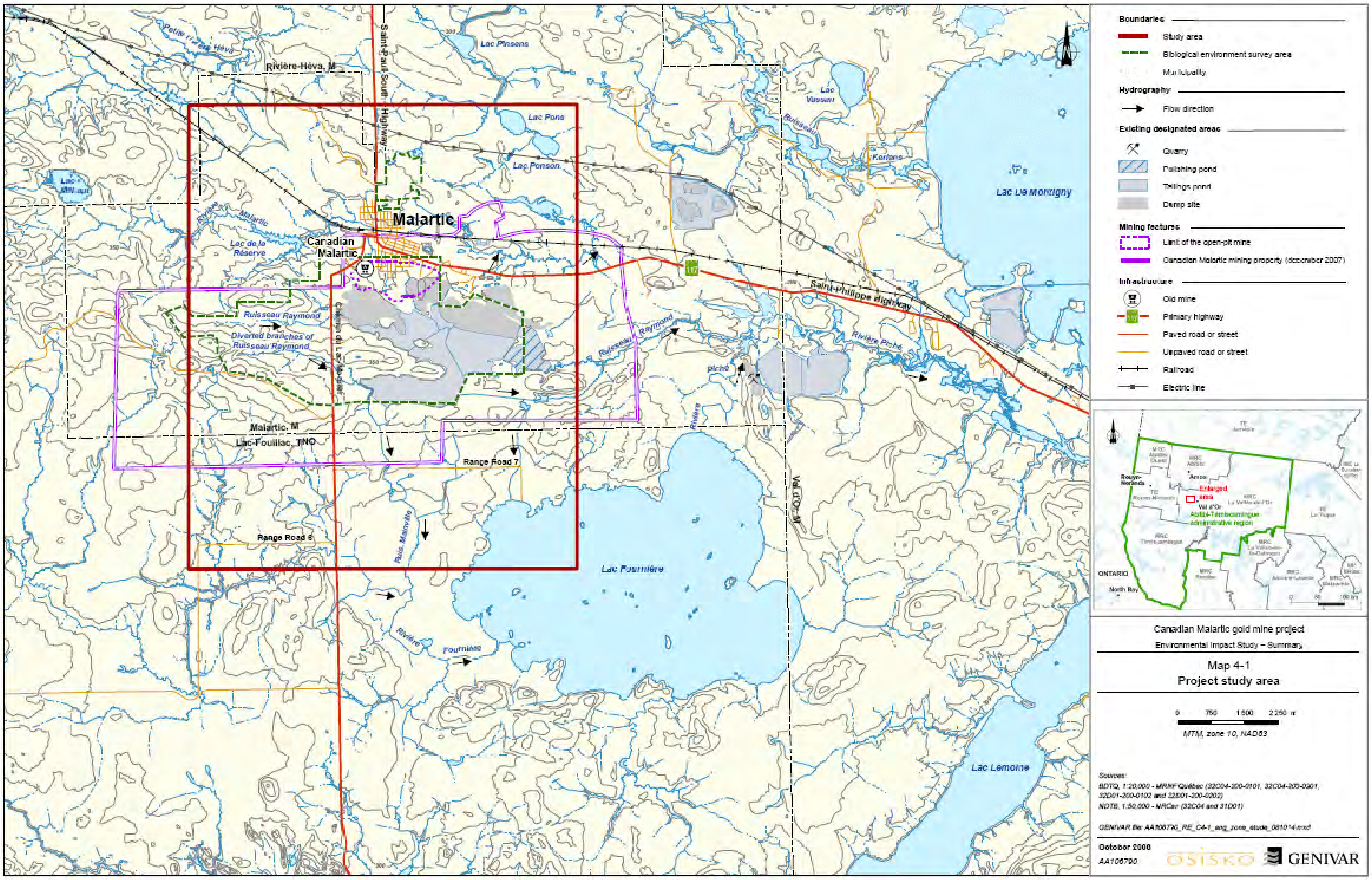
10.1 Project Impacts, Mitigation and Monitoring

10.1.1 Study Area

The main components of the project (open pit mine, process plant, tailings pond and waste rock dump) will be located within the urban and peri-urban perimeter of the Town of Malartic. Some of the proposed plans may lead to disagreements about the impact on the quality of life for residents, or the project's effect on the biological and physical attributes of the area. Moreover, the region has a long history of mining and industrial activities, and the numerous infrastructures left behind could have negative effects on the environment, notably on water quality. Among the many examples of old infrastructure are a tailings pond, settling and polishing ponds, and a mining complex that includes an ore processing plant, namely the East Malartic mine site.

The study covers an area of approximately 14 km^2 (Figure 10.1). The study area was defined by taking into account these aspects, as well as the expected effects on local and regional populations, and on living organisms inhabiting the land and waterways. The boundaries of the study area are just beyond Road no. 6 (6^e Rang; Lac-Fouillac) to the south, northward beyond the Malartic town limit, westward to the boundary between the Rivière-Héva and Malartic municipalities, and eastward towards the edge of the Canadian Malartic property.

Figure 10.1: Project Study Area Map



10.2 Management and Closure of the Tailings Pond and Waste Rock Dump

Over the course of the mine life, it will be necessary to dispose of and manage 122 Mm^3 of tailings, and 161 Mm^3 of waste rock. The storage sites for both the tailings and the waste rock will be located over the old East Malartic tailings pond southeast of the planned pit. Together, these storage areas will cover 760 ha of land (the tailings will be deposited in several cells, thus allowing for continuous rehabilitation of the storage sites). The first cell will be ready in 2011, and filled by the end of 2013. Rehabilitation work is scheduled to begin in 2014. Using this technique, it is estimated that 65% of the site will have been rehabilitated by the time mining operations cease.

The advantages of adopting a continuous rehabilitation program include lowering the environmental impact over the course of the project as required by the government for rehabilitation work. This requires an investment which will be spent in a phased manner over the life of the project.

Osisko and the provincial government have agreed to proceed with the closure of the old tailings disposal site at East Malartic to create the space needed for disposing the waste and tailings from the proposed Canadian Malartic project. The work is included in the construction budget for the processing plant and will be carried out from June 2008 to December 2010.

The capital required for the Canadian Malartic project closure plan is estimated at $45 M, including a 3-year post-rehabilitation monitoring program that will bring the total lifespan for the waste and tailings management and storage closure plan to 12 years (starting in 2011 and ending in 2023). The main aspects of this plan are summarized below:

- Management of approximately 283 Mm^3 of material.
- Tailings from the process plant will be thickened to a density of 68% solids thereby minimizing the water flowing into the tailings area and reclaimed from the new polishing pond.
- Tailings and waste rock will not generate acid.
- Waste rock and tailings will be disposed of on the old tailings pond at the East Malartic mine site, which represents the most beneficial scenario for the project (use of previously disturbed land, less environmental impact, closure of an abandoned site using new mine waste) and tailings.

- Possibility of carrying out continual rehabilitation at the waste rock area and tailings management facility.

- Work related to the closure of the tailings management facility will be completed in autumn 2020.

- Post-closure follow-up work will most likely end in December 2023.

- At the end of mining operations the pit will fill with ground water over a period of time and its boundaries secured and landscaped.

10.3 Study of Water Quality, Sediments and Benthic Invertebrate Community

10.3.1 Study of Fauna

A wildlife survey was conducted to evaluate the different species present in the environmental surroundings of the proposed project. The fish captured in waterways during the wildlife survey belonged to 14 different species, none of which are endangered. Six are of interest for sport fishing, notably walleye, sauger, yellow perch and northern pike. During this survey, fish belonging to two of the waterways studied, Lac Fournière and Rivière Piché, had a markedly diverse collection of species. Lac Fournière is more vulnerable to potential disruption because there is no upstream migration of fish. Possible spawning grounds were identified, one of which would be for yellow perch in Rivière Piché, whereas other habitats that would be favourable for northern pike reproduction were found in Rivière Fournière.

Due to the harsh climate in the region surrounding Malartic, the common garter snake is the only grass snake species observed during the survey. Host green frogs and northern spring peepers, as well as wood frogs and American toads were found in the forest. Several marshes were also located in close proximity to the project site where several amphibian species were observed as well as 55 species of birds (mostly sparrows and waterfowl). The landform depressions created by former mining operations serve as interesting habitats for waterfowl and waders (shorebirds), particularly the smaller pits to the west. Bigger mammals were also observed during the surveys. These included black bears, moose, wolves, Canadian lynx, red foxes and the American marten with the main carnivores represented by the later 3 species. Other small animals, both carnivores and herbivores, were also present in the surveyed area.

None of the wildlife species observed during the survey are designated as threatened, vulnerable or at risk. Present numbers would also suggest that such designations are unlikely in the near future.

The primary impacts to wildlife that could be incurred as a result of the project include loss of habitat, noise disturbances, and the risk related to contamination of the environment during the construction and operation phases. The residual impacts will be low to very low however because Osisko will avoid development of the area west of Chemin du Lac-Mourier, will dispose their tailings on top of the restored East Malartic tailings pond, will undertake continuous restoration of the disturbed sites, and will apply modern and specially adapted mitigation measures (such as using a top soil cover blended with compost, planting indigenous species, etc). Biological diversity is low in the region, and critical habitats will not be touched. In fact, the creation of a polishing pond during the closure work at the old East Malartic mine site should improve environmental conditions and as a result would have a positive effect on fish populations. During the closure phase, the reclamation of the area by wildlife and the creation of new habitats will represent a positive impact.

10.4 Water and Sediments

10.4.1 Hydrologic Regime

The hydrographic network of the study area consists of small meandering, slow-flowing streams. The final effluent of the old East Malartic site discharges into Ruisseau Raymond and Rivière Piché. Rivière Piché represents the outflow of Lac Fournière, and its hydrologic regime is influenced by variations in the lake's water level. Ruisseau Mainville runs into Lac Fournière and receives a flow of natural water from the southern diversion ditch in the western part of the mine site. Rivière Malartic is a stream which passes through the town of Malartic in the northern part of the study area. This river receives water from sources north of the mine site.

The hydrologic regime of the project area is not natural as a result of the presence of the old mine site, and is instead determined by the existing water management infrastructure. The MRNF's water management plan, however, will secure the site. The Canadian Malartic project was also designed to limit any impacts on the hydrologic regime by building new mining infrastructure almost directly on top of the footprint that will be left behind after the MRNF finishes closing the old East Malartic mine site. Moreover, the Canadian Malartic project will re-use the MRNF's drainage and water management infrastructure.

The impact on the hydrologic regime is considered low for all three phases of the project (construction, operations and closure phases). It is expected that modifications to the topography, the surface area of watersheds, and the soil cover will all affect the rate of surface run-off, infiltration and evapotranspiration.

By the time the project enters the construction phase, most land preparation and access road work will have already been completed as part of the MRNF's closure work at the old East Malartic mine. The new infrastructure for the Canadian Malartic project will mostly overlie land that was previously disturbed or that became disturbed during the MRNF's closure work. The only natural land that will be disturbed by the construction is at the proposed sites for the processing plant and the open pit mine. Soil compaction and the removal of vegetative cover around these areas will reduce the amount of infiltration and increase the amount of surface run-off. Modifications to the hydrologic regime, however, are expected to be local and minor.

The hydrologic conditions during the operation phase, should be comparable to those of the construction phase with only the build-up of the tailings pond dykes having an additional influence on the hydrologic regime. A risk of a spillover along the southern edge of the MRNF-restored tailings pond does exist, however, any spillover would be contained within the boundaries of the Ruisseau Raymond watershed.

Revegetation will be a focus of the closure phase. Vegetative cover will help retain some rainfall and meltwater, the rest of which will become involved in run-off, infiltration and evapotranspiration processes. Surface run-off over the entire site will continue to drain into the polishing pond, which will be modified to retain any particles accumulated during mining as well as promoting the settling of new sediments carried into the pond by run-off. Water quality will be verified periodically, and if necessary, the water will be treated until the directive 019 guideline criteria are satisfied, then the treatment will be terminated. Outflow will be guided to Ruisseau Raymond, thus minimizing downstream disturbances to the water network.

10.4.2 Quality of Surface Water and Sediments

Surface water in the study zone is characterized by slight cloudiness, coolness (even in summer), and a slightly acidic pH. It is well oxygenated and its overall productivity is relatively high. It is also well mineralized and slightly hard.

The impact on the quality of surface water and sediments is considered low for all three phases of the project. During construction, there is a potential risk of contamination by accidental spills of petroleum products as well as other hazardous materials used during this phase. The MRNF's planned drainage system for the old East Malartic mine site will allow surface water to be collected from watersheds that overlap the proposed infrastructure sites. The MRNF's network will be re-used and completed during the course of the Canadian Malartic project, and will allow all the water in the project area to be diverted to the new polishing pond. After sedimentation occurs (settling), most of this water will be reintroduced into the processing plant as process make-up water. Any excess water from the new polishing pond will be treated (Effluent Treatment Plant) to ensure that the quality of the discharged water will conform to the applicable norms.

Impacts during the mining phase are mainly related to the risks of contamination by water containing suspended or airborne particles, leached metals or secondary products generated by blasting as well as by accidental spills or mishandling of reagents. The water in the polishing pond represents a potential impact. Nevertheless, the quality of any downstream waterways that receive effluent will be only slightly modified given the emphasis placed on recirculation and the presence of control infrastructure within the site.

During site closure, the placement of top soil cover will have positive effects on the processes of run-off, evapotranspiration and infiltration. The physical properties of the thickened tailings will also ensure lower infiltration rates and limit the contact between surface water and waste rock and tailings. Erosion and transportation of suspended matter will be largely limited by the presence of the vegetative cover. The dismantling of the processing plant and related infrastructure, combined with the placement of a vegetative cover, should improve overall water quality and render it comparable to natural conditions.

10.4.3 Groundwater Quality and Levels

Groundwater quality in the study area was assessed using a dozen observation wells during two sampling programs. In terms of heavy metals, copper, lead, nickel and zinc contents exceeded the MSDEP's applicable criteria for consumption, re-entry into surface water, and infiltration into potable water systems. It should be noted that no petroleum hydrocarbons or cyanides were detected in any of the samples. In addition to the observation wells, more than twenty homes were visited in 2007 and 2008 to verify the quality of drinking water. Results indicate that residential drinking water respects the criteria outlined in the Regulation Respecting the Drinking Water Quality and the Environmental Quality Act (Québec).

During the construction phase, there is a potential risk of groundwater contamination due to the potential infiltration of contaminated surface water (i.e., surface water affected by secondary products and accidental spills). It is expected, however, that any impacts would be minor. Moreover, water quality should not be any different compared to the current situation in which groundwater has already been affected by earlier mining activities, especially considering that controls and monitoring procedures today are much more rigorous than those in the past. In addition, the low permeability of soils will limit the infiltration of any contaminated surface water. The impact of a lower water table caused by pumping of the old underground workings will be comparable to the current situation since the dewatering process will be conducted in the same way as it was in the past.

Potential impacts to water quality during the operations phase are associated with the same risks as those during the construction phase, however, with an additional risk of contamination by leached metals from the ore, waste rock and tailings. Dewatering operations in the open pit will aid in limiting the flow of contaminants into subsurface waters. The pumped water will be sent to the new polishing pond and then reintroduced as processing water in the process plant. In the case of drawdown caused by pumping from the pit, monitoring of groundwater levels will provide a window of opportunity in which to react and put into operation a contingency plan. The degree of disturbance to the level and quality of groundwater is expected to be moderate considering the amount of drawdown expected for the work area.

During the site closure, pumping from the open pit will stop, thus creating a moderate adjustment to groundwater levels and water quality as drawdown and its related effects will gradually be reduced.

10.5 The Study of Climate and Hydrology

The climate in the project area is characterized by long, cold winters and relatively short summers. Total precipitation reaches 914 mm. Winds generally blow from the south or southwest from June through January and are mainly from the northwest from February through May. Evaporation amounts to 652 mm per year, most of which occurs during the summer season when the water balance experiences an average deficit.

The major waterways potentially affected by the project are Rivière Malartic to the north and Rivière Piché; the surface areas for their respective watersheds are 28.5 km^2 and 194.8 km^2. Based on hydrologic studies, the current watershed has the potential to supply the water needed for the processing plant.

10.6 Ambient Air Quality

The results of the ambient air study conducted as part of the Canadian Malartic Environmental Impact Assessment indicate that the air in the Town of Malartic is of very good quality and representative of a rural town with few active industrial activities.

During construction, an increase in the amount of dust and other airborne contaminants is expected. Construction of the plant and related facilities, in addition to the transportation of materials, will only have a minor impact on ambient air quality as a result of the distance between the construction sites and residential areas. In contrast, however, a more significant impact is expected during work on the green zone (the landscaped ridge and linear park) due to the movement of trucks, the dumping of material and the use of machinery near residences. Despite the application of mitigation measures (such as the use of a high wooden fence to hide the work, dust control, avoiding night shift, etc), the anticipated impact may be moderate for those residences closest to the development zone.

During operations, concentrations of dust and other airborne contaminants are expected to be higher than the current levels. To evaluate the potential impact, the study took into account the reduction in airborne particles generated by the closure of the old East Malartic mine. At the time of the study, this site generated significant quantities of particles. Once the Canadian Malartic project is underway, however, it will be covered by thickened tailings, which should aid in reducing air pollution.

Results have indicated that the level of fine particulate matter in the atmosphere may possibly exceed the limits outlined in the ‘’Projet de règlement sur l’assainissement de l’atmosphère’’ (PRAA; Québec’s project on air quality regulations). It should be noted, however, that the applicable norm is not yet in effect, and after the fourth year of mining, the level of fine particulate matter will be reduced and should be reduced to acceptable levels. In a worst case scenario, for the years covered by modeling, the criterion is only exceeded less than 1% of the time, and this only occurs at the southern limit of the Malartic urban centre. Other than this exception, the PRAA is respected at all times. None of the established criteria or norms for potential airborne trace metals will be exceeded. Considering the collective results of the study, the impact on ambient air quality is expected to be low.

The impact on ambient air quality during the closure phase of the mine is expected to be similar to that of the construction phase, but to a lesser degree. One area of note, however, is that the work related to the restoration of the open pit will take place near some residences. Nevertheless, the landscaped ridge behind the linear park will reduce the inconveniences associated with this phase of the project and should result in only minor impacts.

10.7 Background Noise and Vibrations

10.7.1 Background Noise

An analysis of background noise in the study area focused on two main areas: the urban part of Malartic and in the vicinity of Road no. 7 (7^e Rang), south of Malartic. The stations recorded minimum sound levels of 42 to 51 dBA during the day and 21 to 43 dBA at night. The main source of noise was traffic, particularly on Highway 117. The other identified sources were mechanical in origin (local road and air traffic, ATVs), natural (rustling leaves, songbirds, etc.) or the result of human activity (property maintenance). In order to meet the noise criteria outlined in the MSDEP's Directive 019 for the mining industry, the maximum noise contribution of future mining activities, as recorded at the same stations, should not exceed 45 to 51 dBA during the day or 40 to 41 dBA at night.

The projected impact from the project to background noise is considered low during all phases of the project, due mostly to the way in which the project was designed and as a result of the mitigating measures that will be implemented. Furthermore, traffic conditions will change significantly at the intersection at the Royale/Lasalle/Lac Mourier intersection. In addition, a permanent linear park, which will be built between the mine site and the urban centre, will act as a noise reduction barrier and a buffer zone.

During construction, the project's impact on background noise will be mainly related to site preparation work and the construction of the ore processing plant and related facilities. The simulation conducted for the plant construction phase indicated that projected noise levels at the nearest residences will remain below the MSDEP's prescribed day and night guidelines with respect to noise from a construction site. It should be noted that the area likely to generate the most noise (the process plant) will be located approximately 2 km from the nearest residences, while the crusher will be 1 km away. In fact, the increase in noise caused by the plant's construction will not even be perceptible in most cases. Only the most eastern residences (5 or 6) on Chemin du Lac-Mourier are expected to experience a slight increase in noise compared to current background levels.

During the operations phase, the main noise disturbances are expected to be caused by mining operations and an increase in traffic through the town. However, the maximum sound levels established by the MSDEP will be respected at all times. The increases in noise during this phase will be barely perceptible in most cases but again only the 5 or 6 most eastern residences on Road no. 7 (7^e Rang) will notice an increase in noise.

During the closure phase of the mine, the expected noise impact will be similar to that of the construction and operation phases.

10.7.2 Vibrations

The construction phase of the project is expected to cause an increase in vibrations, mainly due to site preparation work and the construction of the ore processing plant and related facilities. The impact should nonetheless be very low given that blasting will take place more than 2 km from the nearest sensitive areas. There may be other sources of noticeable vibrations, for example from the use of percussion equipment or explosives in additional pits, but it is not yet possible to determine their effect with certainty given the current state of advancement of the project. The borrow pits needed for the closure of the East Malartic mine will continue to be used and none of these zones are situated near residential areas.

A moderate impact is expected with respect to vibrations during the operations phase of the project. The impact will mainly concern damage to structures (due to vibrations and air overpressures) and disturbances to the residents in the part of town just north of the mine pit. Added to this is the safety risk associated with possible fly rock and other debris from the pit during blasting.

People's perception of the vibrations may attain the level of "disturbing" or "unpleasant", even though the levels respect the criteria laid out in the MSDEP's Directive 019 for the mining industry. If the air overpressures that accompany vibrations during blasting exceed 115 dB, people may experience short periods of disturbance. It should be noted, however, that the blasts will only last around 3 seconds and that there will typically be only one per day, or two at most on particular occasions at the beginning of mining operations. When weather conditions are not favourable blasting, will be postponed. Finally, the risk of breakage caused by rock chips (fly rock) during blasting will be eliminated by proper design and planning, as well as the use of blast mats.

Considering all the protective measures put into place, the Malartic living environment will not be at risk, nor will the nearest residences. The anticipated risk with respect to structural damage would be very low.

The impact related to vibrations during the closure phase of the mine will be similar to the period during construction but without the inconveniences caused by blasting because it will not be needed. Activities generating vibrations will once again be focused at the plant site, quite far from the nearest residential areas. As a mitigation measure, traffic management will limit the number of heavy loads in the Malartic urban centre. A very low level of impact is anticipated for this phase.

10.8 Vegetation and Wetlands

The area covered by the biological survey can be broken down as follows: 52% land vegetation, 9% wetlands, 33% waterways, and 6% unnatural (modified by humans). Black spruce and aspen stands are the most important vegetative cover in terms of surface area. Other types of vegetation are fallow zones, mixed forest, white birch, other softwoods and fir, in addition to a logging area with protection of regeneration. The wetlands are distinguished by alders, ponds, marshes, wooded bogs and peat-bogs. Note, however, that the natural environment at the Osisko site has likely been affected by past mining operations, particularly along the east side of Chemin du Lac Mourier.

None of the plant species present in the surveyed area are considered threatened, vulnerable or at risk. . Furthermore, the study area as a whole does not have the potential to harbour any plant species at risk.

The initial project was to take place on both sides of the Chemin du Lac-Mourier. The risk of infringing upon valued ecosystem components west of the road (forested areas, wetlands, watersheds, etc.) has led Osisko to participate in the closure of the old East Malartic mine and to superimpose its tailings pond on top of the restored tailings dump at the old mine site. Osisko has chosen to utilize a technology that allows disposed tailings to be piled higher rather than over a wider area. These two factors will significantly reduce the impacts on vegetation by (considerably) limiting the area affected by waste disposal. In addition, rehabilitation work can be conducted continuously, without compromising the operations.

The construction phase of the project will cause a loss of terrestrial vegetation and wetlands, bringing with it a potential risk of contamination. However, none of the plant groupings or species are particularly notable in the area. Also, because the integrity of the environment is not at stake, the impact is considered to be low. By the end of mining operations, more than 65% of the surface area will have been recolonized. With regards to wetlands, if species are lost, new species will arise in their place to compensate.

Positive results are expected during the mining and closure phases of the project. Revegetation efforts during mining and the recovery of vegetation by the end of operations will represent gains in terms of the area's plant coverage.

10.9 Soils

Human activities, mainly related to mining activities, affect virtually the entire eastern half of the study area. These activities have significantly modified the countryside over several decades, and have produced considerable quantities of tailings. Some of the tailings are contaminated and stored in tailings ponds. Other non-contaminated tailings were stockpiled or used as backfill material.

During the construction phase, impacts related to the area's soils will be very low and would be primarily related to the potential risk of contamination by accidental spills of hazardous products or by airborne particulate matter blown from trucks carrying aggregate material. The potential risks will be minimized by applying mitigation measures that focus on prevention through regular equipment controls and the addition of emergency features that provide rapid intervention in the case of accidental spills. The potential for a major spill from a reservoir is virtually non-existent, due in most part to confinement systems and the double-walled design of some reservoirs. Furthermore, potential spills from machinery would be limited in volume, and an emergency plan would be rapidly deployed, thus minimizing the extent of any contamination.

Any potential impact on soils during the operations phase would be related to the potential risk of contamination by leached metals from the ore, by accidental spills, or by airborne particulate matter blown from trucks carrying aggregate material. However, the new mining facilities are to be built on land that was previously disturbed (old tailings, settling and polishing ponds) and which will be progressively restored, thus limiting the infiltration of contaminants into previously undisturbed soils. Furthermore, re-using the drainage infrastructure that is to be built by the MRNF's during the closure of the old East Malartic mine will help control surface run-off water. The application of mitigation measures (such as fueling the truck at the fuel station on a concrete pad instead of fueling in the mine pit from a tanker, good truck maintenance practice, frequent environmental site inspection, the use of water trucks to spray roads, low speed limit on roads, etc.) will aid in minimizing the potential for soil contamination from spills and airborne particulates. These measures will therefore ensure that any impact on the soils will be minor.

Once the mine site closes, the potential impact will be comparable to that during the operations phase – thus low – until the time of final rehabilitation. After the site is restored, the potential impact will be virtually zero since contaminated soils will have been removed from the site during restoration. The tailings pond and waste dump will be covered by a layer of vegetation that will limit the amount of surface run-off and minimize the dispersion of fine particles that could contaminate the soils.

10.10 Permitting

For the eventual environmental permitting needs of the project, Osisko's consultants initiated a baseline environmental data collection study in June 2007. This process lasted until February 2008. The results of the study were published in the Environmental Impact Assessment study (EIA), which was completed September 3rd 2008. The following is a summary of the Environmental Assessment process as well as the proposed activities until the start up:

- June 2007 to February 2008, environmental baseline data collection **(Completed)**
- February 2008 to September 2008 Environmental Impact Assessment (EIA) **(Completed)**
- September 3rd, 2008 submittal of the EIA to the Québec Ministry of Sustainable Development, Environment and Parks (MSDEP) **(Completed)**
- Review of the EIA by the Government - September to December 2008 **(in Progress)**
- Acceptance of the EIA by MSDEP - December 2008
- Public hearings January 2009 to May 2009
- Approval to proceed (Order in Council) - May 2009
- Certificates of authorizations - June 2009.

Note that the Canadian Malartic project (mining, processing) is subject to the EIA approval. The houses and institutions relocation, street infrastructure, East Malartic tailings pond site closure, power line, pre-stripping and land preparation (named satellite projects) are not part of the EIA. Those sub-projects have received their permits or their permitting is in progress according to schedule.

11.0 COMMUNITY CONSIDERATIONS

11.1 Communication and Consultation in the Region

Since the project was first announced, various communication and consultation activities have taken place within the community and with the representatives from the town of Malartic and also with regional representatives. These activities can be grouped into three distinct themes: communication activities organized by Osisko, those organized by the Community Consultation Group (CCG), and consultations and surveys conducted within the context of the Environmental Impact Assessment study.

11.1.1 Osisko's Communication Activities

Osisko has conducted numerous activities in the project area with the objectives of informing people and gaining insight about their concerns and expectations regarding the project. The main activities have been:

- Meetings with elected officials and representatives of the area's organizations
- Public presentations within the community
- Meetings with residents that will be relocated by the project
- Meetings with suppliers
- Presentations given to the various chambers of commerce in the region.

Osisko has also implemented measures for disseminating information about the project, including:

- The Osisko Community Relations Centre in downtown Malartic
- The Osisko internet site, which is largely dedicated to the project
- An Osisko column, called "Osisko Vous Informe", published twice a month in local newspapers
- The CCG website

- An information brochure about the project and various information bulletins and announcements in the local media.

In addition, Osisko created the ‘’Fonds Essor Malartic Osisko’’ (FEMO), a not-for-profit sustainable development fund for the town of Malartic and the region. The fund will finance economic development and community initiatives that will improve the quality of life for Malartic residents. Scholarships and other investments in education will also have an important impact.

11.1.2 Community Consultation Group

The Malartic Community Consultation Group constitutes a link between the residents of Malartic, the Town of Malartic and Osisko. It is formed by representatives from all three of these groups. The role of the group is to develop avenues of communication and to respond to questions from Malartic residents about the project to ensure the well being of their quality of life. Some of the activities currently being delt with by this organization are the relocation of residents from the southern part of town as well as managing requests and complaints related to various aspects of the project.

11.1.3 Consultations and Surveys of the Area Within the Context of the Environmental Assessment

The environmental assessment of the project involved the consultation of organizations in the area from June 2007 to March 2008. In all, 35 interviews took place with representatives of various ministries, regional and local organizations, recreational and tourism organizations, and private businesses. In addition, from autumn 2007 to winter 2008, all residents and commercial enterprises in Malartic, as well as the residents of Lac-Fouillac who live near the proposed mine site, were surveyed using questionnaires. The objective was to gather concerns and opinions about the project and to determine the psychological and social effects that could arise as the project advances.

11.2 Main Concerns Related to the Canadian Malartic Project

The project is expected to generate significant employment and economic benefits for Malartic. Some of these include the arrival of new residents, the opening of new commercial and industrial enterprises as well as new services that will improve the tax base for the town. As for existing businesses, Malartic’s economic growth will allow them to consolidate their activities, promote expansion and diversify their services. However, some citizens do have reservations about this point, most notably related to the proximity of the project to Val-d'Or and the mobility of the regional workforce. An increase in Malartic property values and the creation of new jobs are nonetheless expected. Many participants believe that the subcontracting needs of the project should be met locally as much as possible. However, this may be influenced by the limited amount of qualified workers in the immediate areas, especially given the current high demand in the mining sector. One particular area of concern for the residents and businesses of Malartic is to ensure that the continued economic diversification of Malartic is not significantly hampered by the project in order to ensure a sustainable economy after the mine closure.

On a social level, some of the participants felt that one of the most likely benefits of the project would be the retention of current Malartic residents, whereas others predicted the departure of a number of residents due to environmental inconveniences. Some see the arrival of new residents as an opportunity to make the community more dynamic as well as a chance to improve the social fabric and living conditions. Others, however, are concerned about the potential shortage of housing, possible rent increases, and deterioration in the quality of life due to the potential noise and dust.

Participants also expressed their concerns about the potential risks associated with mining. These included the risks associated with blasting, increased traffic on the municipal road network, and the possible contamination of underground water sources by hazardous materials and mining waste.

11.3 Survey of the Region

The results of a survey conducted in late 2007 revealed that in terms of social acceptability, the project received a very strong show of support from both citizens (84% in favour) and the business community (96% in favour). The risks of conflict and division within the community are thus considered extremely low.

Mining will have minimum affect on the regional social fabric. The risk is low due to the fact that most jobs will be filled by people from Malartic and the Abitibi region. Despite the high number of jobs created, it is expected that the Malartic population will stabilize or increase only moderately. The reason is the close proximity of neighbouring urban centres, such as Val-d'Or, Amos and Rouyn-Noranda from where the workers will commute to the Canadian Malartic work site.

It is very unlikely that the eventual mine closure will lead to significant social divisions, and will thus have a low impact on social cohesion. Efforts to diversify the economy and prepare the community for closure of the mine site will ease the social impacts during this stage.

11.3.1 Population's Attachment to their Community

The construction and operation phases of the project should have positive effects on people's attachment to their surroundings. The project will allow Malartic to revitalize its economy, a process that was already started when the project was first announced. The survey of Malartic merchants revealed, for example, that 37% plan to expand their business, and 18% are doing it because of the project. In terms of demographics, the downward trend experienced from 1991 to 2006 (16%) will be either stopped or reversed, which will breathe new life into the local institutions and schools. For the town, the extra revenues from taxes will help improve infrastructure and services. Osisko will rebuild, at their own expense, the institutional buildings that need to be relocated from the south end of town. All of these factors should improve Malartic's image in the eyes of its citizens and the regional population, thus increasing the attachment that the people of Malartic feel for their community.

One stage of the project which may disturb the population is during the construction of the linear park. However, this disturbance will only be temporary, and should not detract from the positive overall feelings of residents towards the project or impacts to the surrounding landscape. Rather, the design of the landscape ridge took into account the opinions expressed at many consultations with the general public.

A small impact on people's attachment to their living environment is expected during mine closure because it could lead to a deterioration of economic conditions and may prompt residents to leave town if the continued efforts towards economic diversification prove unsuccessful. The community would thus lose some vitality with the loss of residents, relatives and friends. Nevertheless, the existence of the FEMO (Fonds Essor Malartic Osisko), a sustainable development fund for the town of Malartic makes it extremely plausible that the community will succeed in significantly diversifying its economy. The length of the operation will allow for the promotion of a plan to minimize the impact of the closure.

11.4 Archeological and Heritage Study

The project will not have any impact on the established heritage of Malartic or on the archaeological heritage of the area. Construction work will avoid zones of moderate archaeological potential that were identified within the framework of the environmental impact study. This area comprises the banks of a stream on the west side of Lac Mourier Road in Lac Fouillac. In the case that evident cultural elements are discovered, Osisko will advise the Ministère de la Culture, des Communications et de la Condition feminine (Québec's ministry of culture, communications and women's issues).

11.5 Land Planning and Management

The study area mainly covers the municipality of Malartic, and to a lesser extent, Lac Fouillac and Rivière Héva. In 2006, the Malartic population was 3,640 and the Rivière Héva population was 1,056. Only 91 people inhabited the Lac Fouillac territory.

The Malartic urban environment is entirely contained within the project study area. The land consists primarily of private properties. Outside of the urban zone, most of the land is public and monitored by the MRNF. Included in this area are several large private properties held by Osisko.

Mining facilities and access roads that will be part of the Canadian Malartic project will be developed on public land with occupational rights granted by the MRNF or on private land acquired by Osisko. The development of the open-pit mine requires that the southern Malartic neighbourhood be relocated to the northeastern end of the town. In this area, most of the land belonging to private owners has already been acquired by the company.

The construction and closure phases of the project should have no foreseeable impact on land planning and development in the region. In fact, the project's components will not influence the MRNF's plans for public lands in the study area. Furthermore, the Vallée-de-l'Or MRC's planning and development strategy (schéma d'aménagement et de développement), and Malartic's urban plan, zoning regulations and municipal emergency plan, will be or have already been modified to fit with the proposed mining operations and related activities.

During the operations, the project's impact on land planning and development will be positive. The project will favour many of the directions outlined by the MRC's (Municipalité Régionale de Comté) strategy and Malartic's urban plan, including the recognition of the area as one of the province's centres of excellence for mining. In addition, the project will allow for the rehabilitation of the tailings pond at the old East Malartic mine, and will resolve the safety problem related to zones of land subsidence in the southern part of Malartic. To date, land subsidence and old dump sites for mine waste, both of which are products of past mining operations, have created land use problems in the area. The rehabilitation of the East Malartic tailings pond and the relocation of the southern Malartic neighbourhood will thus constitute a real improvement over the current situation.

11.6 Land Use

There are no foreseen negative impacts for land use in the region. The only foreseeable impacts are either positive, or insignificant.

11.6.1 Residential, Commercial, Institutional and Industrial Land Use

The project requires the relocation of some 205 private residential buildings and a dozen municipal lots in the southern Malartic neighbourhood. The loss of this residential sector will be compensated by the development of a new neighbourhood northeast of the current urban centre. This resettlement program will also create more space to accommodate future residential development.

According to the last available estimate, the relocation of residents from the southern neighbourhood will involve moving approximately 150 residences including 21 apartments, with 77 owners opting to sell their residence to Osisko. For residents moving to the new neighbourhood, Osisko will prepare 190 residential lots and will progressively prepare foundations for 150 residences from now until 2009. In addition, Osisko will construct 7 multi-family units for a total of 28 apartments to replace those that cannot be moved.

New municipal infrastructure for the water and sewage collection systems will also be constructed to serve the new neighbourhood. The infrastructure is designed to accommodate 210 additional lots to satisfy future needs for residential space, which will allow the town to proceed with residential development at relatively low cost.

The southern Malartic neighbourhood includes public service institutions (elementary school, adult education centre, long-term care facility and retirement home, community centre and auditorium). Osisko will construct new buildings for these institutions, which will benefit the community.

11.7 Landscape Components

The project will bring many changes to the actual landscape of the southern part of Malartic (Figures 11.1 and 11.2). The construction of the project's basic components will modify the landscape (processing plant, tailings pond, open pit, etc.), however other components will help minimize the impact on the landscape (linear park, wooded buffer zones, etc.).

During the construction phase, visual impacts will not be significant because most of the activities will be far from any observers. Construction work on the landscaped ridge, however, will be visible to residents in the immediate vicinity.

During operations and closure activities, the project's impacts will vary from negligible to moderate according to the type of landscape: industrial, built, recreational or forested. In terms of industrial landscape, the project's impact will be low. The waste rock dump, which will attain 97 meters in height (elevation of 427 m) around the 9th year of mining, will be visible from the road. To minimize its visual impact, Osisko plans to revegetate the dump site and build a wooded buffer zone. With the waste rock dump revegetated, its impact will evolve from moderate to low as the plants grow. Once the wooded area of the buffer zone is established, the waste pile will not be visible.

The impact of the mine's presence on the landscape will be minimized. The construction of the linear park will provide a visual barrier between the town site and the mining operation. The waste rock dump, however, will be visible from the west end of Rue Royale. It will remain visible even when revegetated. For visitors to the camping and golf course, the visual impact of the waste rock dump is minimized by the linear park. Progressive build-up of the dump site will result in a temporary loss of forest.

Figure 11.1: Project Components

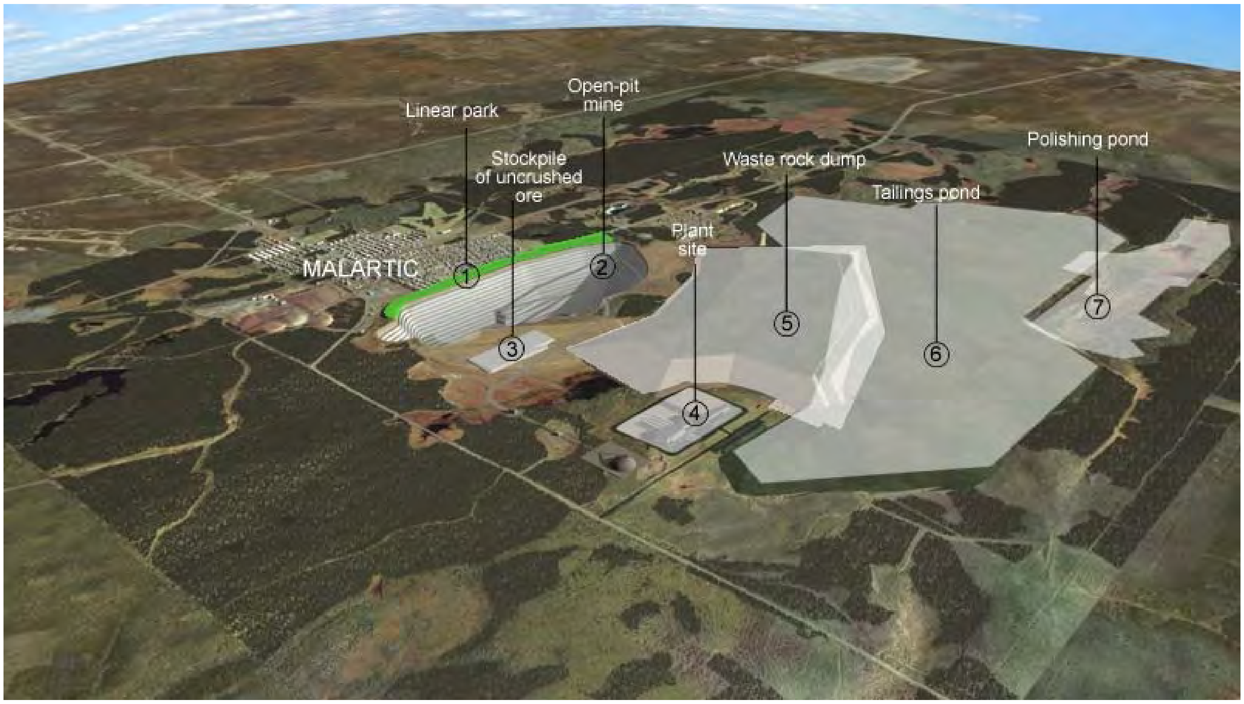
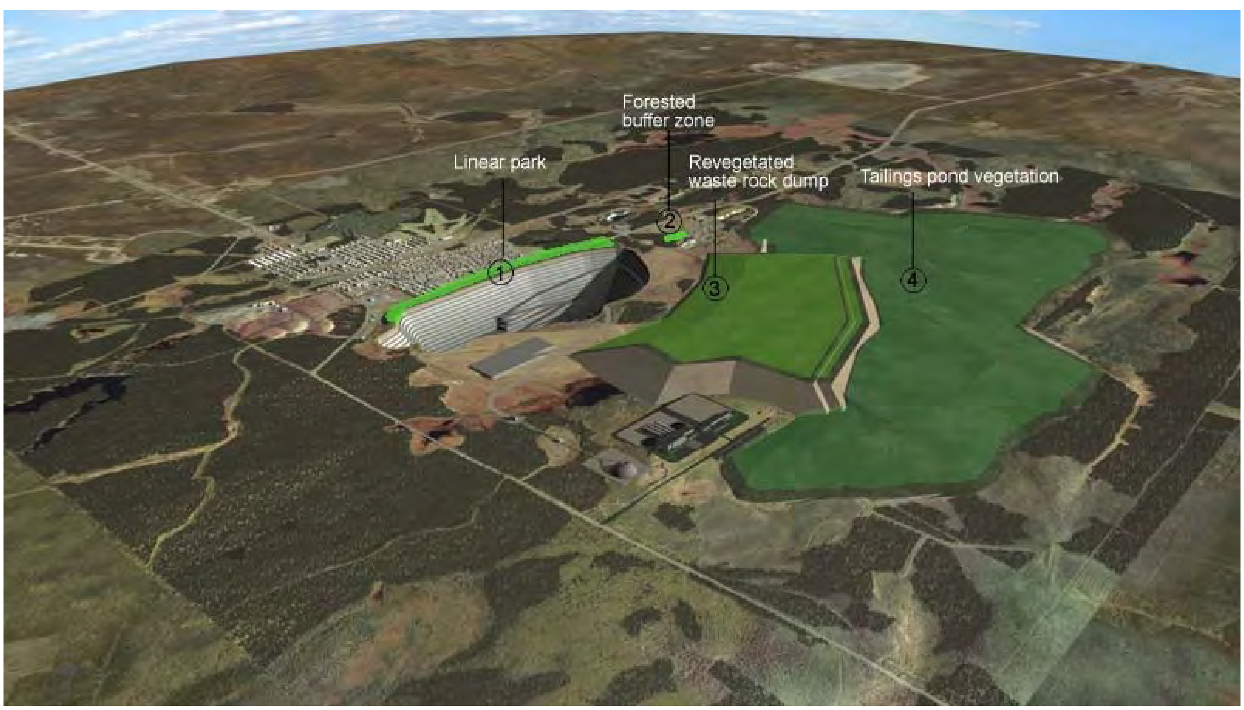


Figure 11.2: Proposed Mitigation Measures



11.8 Quality of Life

11.8.1 Physical Well-Being of the Population

The potential impacts on the physical well-being of the population relate to noise, vibrations and dust emissions. During construction, a moderate impact is expected because the air quality may be affected. Although dust emission levels will be, for the most part, limited by the application of mitigation measures, residents closest to the linear park may be inconvenienced during its construction. As for noise levels and vibrations, standards set out in the MSDEP's Directive 019 for the mining industry will be strictly followed and the large distance between the worksite and the urban setting will reduce the risk of disturbance for residents.

Impacts on the physical well-being of the population during mining operations are deemed moderate and will be mainly associated with vibrations caused by blasting. The vibrations may be felt beyond the immediate vicinity of the open pit, although blasts will only last an average of three seconds daily. In terms of air quality, the results of simulations show only a minor impact during mining operations due to the fact that mine tailings from the former East Malartic mine will be covered by thickened tailings from the Canadian Malartic project, so the new waste disposal area will be much less likely to produce dust. Moreover, the use of dust suppressants, the presence of the linear park and various other measures will contribute to minimizing problems for residents. As for noise levels, they may be somewhat higher than the current background levels, mainly along streets near the open pit and along the eastern part of Road no. 7 (7^e rang). It should be noted, however, that noise thresholds established by the MSDEP will be strictly followed, both during the day and at night.

The community relations activities and ongoing communication program implemented by Osisko will be maintained throughout the mining phase of the project in order to receive feedback from the citizens and adjust operations as needed. A phone line that has been made available for the population to contact Osisko and the establishment of a monitoring committee are two other measures that will help minimize the project's impacts on the physical well-being of the population.

Similarity for the construction phase, dismantling of facilities during the closure phase is likely to generate some noise disturbance as well as dust particle emissions. The impact will be minor, however, since mitigation measures similar to those of the construction phase will be applied.

11.8.2 Perceived Hazards to the Health and Psychological Well-Being of the Population

The absence of significant risks of contamination during the construction phase means that people should not perceive any hazards to their health and psychological well-being. No impact concerning perceived hazards is therefore expected.

During operations a minor impact on health and psychological well-being is expected. Many of the project's features will limit the amount of dust and noise, particularly the distance between residences and the open pit, the presence of a linear park, and the barrier represented by stores on Rue Royale. The community relations service in downtown Malartic, the ongoing communication program and the phone line available to the population are likely to help maintain residents' trust in Osisko and its assessment of the project's environmental impacts. This situation should minimize any concerns regarding potential health hazards associated with the project.

During mine closure, very few psychosocial reactions such as stress, anxiety, insomnia or change of habits are expected in the population. A small number of residents will have worries, but these should be fairly minor for the most part. If citizens believe that Osisko's management of the site is efficient and respectful of the environment, then it is believed that negative reactions from the population should be minimized.

11.8.3 Community Services

Project impacts on community services will be positive during the construction phase. Construction of the mining facilities will generate significant economic activity, allowing Malartic merchants to improve or at least maintain their current level of service, and allowing other suppliers to establish themselves in Malartic. This situation will help improve the quality of life for Malartic citizens.

During the mining phase, the economic spin-offs and probable increase in the population will bring about the creation of new commercial and community services and improve the services currently offered. In addition, the municipality's real estate tax base will increase. Additional income will be available to improve municipal infrastructure and services and accordingly the quality of life for the citizens. The population will also benefit from the improved services provided by the new public institutional buildings built by Osisko as part of the resettlement program for the southern neighbourhood.

A reduction in community services due to economic slowdown is expected, however, during the closure phase of the project. The significance of this impact on the quality of life is deemed moderate and will vary according to the proportion of local jobs dependant on the Canadian Malartic project. Economic diversification efforts undertaken by the municipality, with the help of the FEMO fund, will help reduce the impact of the mine closure on the local economy.

11.8.4 Economic Impact

The construction and operational phases of the project will lead to improved economic returns for the population of Malartic due to the creation of jobs, higher salaries and higher real estate prices. The job market is expected to improve, and unemployment and part-time employment should potentially decrease. Many jobs will also be created by the commercial revitalization brought about by the project. The value of homes has risen since the announcement of the project, and this upward trend should continue during the construction and mining phases, which will increase the net worth of most homeowners in Malartic. This situation will improve the economic well being of households, thereby improving quality of life. There is however a possibility for lower or fixed-income residents to experience economic difficulties due to probable rent increases. The vacancy rate for rental units is already very low in the region. For some tenants in Malartic, rent increases, should they occur, may add to their economic difficulties.

A moderate impact is expected during the closure phase. This impact will be the result of a reduction and potential deterioration of the job market, local economy, demographics or the value of homes. This situation would reduce the economic value of households, their consumption levels, and thus their quality of life. This could lead to social problems. Offsetting this, rent prices could decrease, which would benefit lower or fixed-income residents. As is the case for community services, the significance of this impact will depend on the economic diversification of Malartic at that time.

11.8.5 Workforce Employability

No significant impact on workforce employability is expected during the construction phase of the project. The vast majority of workers involved in constructing the mining facilities will be governed by the regulations of the Construction Commission of Québec. Most of the work to be performed is conventional and the need for additional training will be minimal for the workers from Malartic or the surrounding region.

An improvement in workforce employability for Malartic and the Abitibi-Timiskaming region is expected during the operations phase, resulting in a positive impact. It is expected that many employees will come from various industry sectors, particularly the forestry sector. Osisko plans to offer training programs that will teach mining and ore processing procedures and safety measures applicable to future operations. This will improve the workers' employability given their newly acquired skills and experience. Moreover, job opportunities related to the Osisko project, the booming mining sector and attractive salaries will hopefully convince many young people in Malartic and the surrounding region to pursue their studies. In addition, the stronger economic health of Malartic will have a positive impact on the school success rate as well as on the employability of newcomers to the job market.

11.8.6 Health Hazards to the Population

An assessment of the potential risks of the project on human health was conducted as a sector study and the highlights discussed in the main report of the project environmental impact study. Substances likely to be released into the environment or to come into contact with the population through various means were taken into consideration. Based on the results of this assessment, the project poses no threat to the health of the surrounding population in terms of airborne metal emissions.

A low impact on employability is expected when the mine shuts down. The cessation of mining activities will involve work that requires minimal additional training. The departure of the employer will put an end to the training programs it provided, although training opportunities might be available elsewhere in the region. However, employees will leave the project with a significant level of training, skills and expertise. This expertise will be in demand for other mining projects or other industrial activities. Finally, with the departure of a major employer such as Osisko, the FEMO fund could be used to create projects to reduce the impact of the mine closure.

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12.0 PROJECT ORGANIZATION AND EXECUTION

12.1 Project Organization

Engineering/Procurement

All project phases including detailed engineering, procurement, pre-production and construction activities are under the direction of the Osisko Vice President, Engineering and Construction.

Permitting and project financing activities will be supported by Osisko Sustainable Development and Financial sectors respectively.

Osisko has an internal experienced development team and will be in charge of project management functions for the Canadian Malartic project. The team consists of highly experienced individuals with knowledge of the local construction conditions and contractors. They have successfully managed projects in difficult conditions and in remote environments from the engineering and planning stages through construction to plant commissioning and operations.

The Osisko technical group will supervise the project detailed engineering. During the feasibility study phase, process plant, tailings/water management and infrastructure detailed engineering have already been initiated by the engineering firm including the procurement process.

Engineering firms are responsible for the following procurement functions:

- Technical specification
- Bid request
- Addenda
- Reception of bids
- Technical and economical evaluations
- Short list meetings
- Purchase order requisition preparation
- Drawing management and approval
- Reception and coordination of vendor maintenance and operational documents.

The Osisko technical team is responsible for the following procurement functions:

- Final negotiation
- Contract award
- Purchase order release
- Progressive payment
- Shop visits
- Site logistics.

At the beginning of the feasibility study, Osisko has retained the professional services of architects and engineers for the institutional buildings and housing relocation projects. Most of these mandates including construction site supervision are already well advanced.

Due to the complexity of major process equipment transportation, Osisko has retained the services of a specialized company in international logistics services.

Construction Management

Osisko will provide project construction management services under the direction of a General Construction Manager. The Construction Management Team (CMT) will include the following services:

- Site supervision
- Project cost control
- Scheduling
- Reporting
- Health and safety
- Site procurement and logistics.

It is recognized that an effective health and safety program during the project is an absolute necessity. The success of the construction safety program is contingent upon its enforcement at all stages of the project, including design, construction planning, construction execution, start-up and commissioning.

The CMT will receive technical support from vendor's representatives who will assist in most of the major process and mechanical equipment installations.

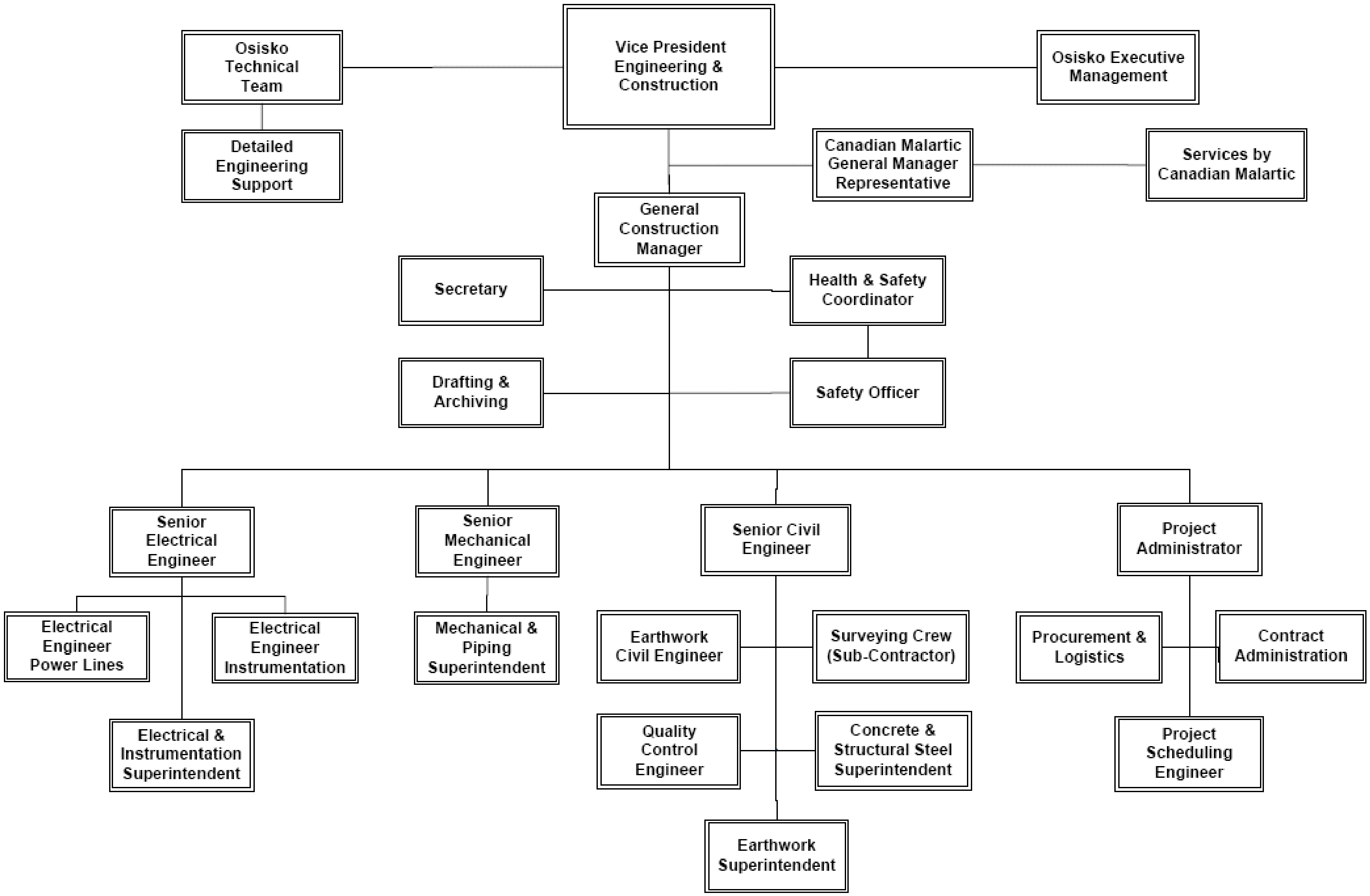
The CMT will also follow the Canadian Malartic procedures and work methods to ensure the protection of the environment. Furthermore, the CMT will work closely with each department of the operations group to ensure proper installation and functional results. During the construction phase, personnel from operations will be integrated into the construction team as coordinators or supervisors.

The Canadian Malartic operations group will support the CMT for the following services during the construction phase:

- Staff payroll
- Accounting support
- IT support
- Site security
- Public relations
- Environmental and permitting
- Medical and first aid
- Mass backfill work (with heavy mining mobile equipment)
- Site logistics.

Figure 12.1 shows the general project CMT organization chart:

Figure 12.1: General Project Construction Management Team Organization Chart



12.2 Execution Plan

Osisko decided to initiate the neighbourhood relocation program before the approval of the Canadian Malartic project to proceed. Timing of this strategic decision was necessary in order to involve and gain support of the local community in the project at an early stage and to be able to complete the relocation before the start of mine pre-production works.

To execute the neighbourhood relocation program, Osisko decided on the following strategies:

- A lump sum contract for the infrastructure works for the new neighbourhood
- House relocation executed by the CMT with sub-contractors
- All institutional buildings under individual turn key contract packages.

For the mine complex, the CMT decided on the following plan based on their extensive experience in large scale mining projects:

- Preparation of the borrow pit and quarry including site access for future site construction works
- Execution of civil preparation works to facilitate the future installation of the mobile concrete batch plant
- Installation of a temporary construction facility immediately following approval to proceed
- Implementation of the following contract strategies:
  - Osisko mining group will execute all mass excavation and backfill
  - Secondary earthwork will be sub-contracted in various packages
  - Lump sum contracts for concrete divided into major process areas
  - Lump sum contracts for structural steel and architectural works divided into major process areas
  - Equipment installation (mechanical, piping, electrical) will be mainly executed within a cost plus type contract. This execution strategy takes full advantage of the experience of the CMT.

12.3 Project Schedule

The overall project schedule is based on a 22-month construction and commissioning period starting in June 2009 and completion of commissioning in March 2011. The construction schedule's most critical milestone is the environmental permit approval. Any delay of the issuing of the environmental permit will cause a delay in the start-up of construction activities. Commercial production is scheduled to start in April 2011.

The project schedule is not affected by any long delivery equipment. Long delivery equipment such as grinding mills, crushers and main substation transformers have been ordered during the Feasibility Study phase and will be delivered to site well in advance of mechanical and electrical erection.

Table 12.1 enumerates the most critical project milestones:

Table 12.1: Project Milestones

Activity	Start Date	Completion Date
Pre-Construction Phase		
Environmental Impact Assessment	September 2007	September 2008
Feasibility Study	April 2008	December 2008
Detailed Engineering	June 2008	September 2009
Permitting	September 2008	May 2009
Financing	Q4 2008	Q1 2009
Project Decision	-	Q1 2009
Construction Phase		
Housing Relocation	October 2007	December 2009
Institutional Buildings	October 2008	March 2010
Temporary Construction Facilities	June 2009	September 2009
120kV Transmission Line	June 2009	September 2010
Polishing Pond	June 2009	October 2010
Mining Services Building	June 2009	December 2009
Mill Area – Concrete Foundation	August 2009	March 2010
Leach Tank Erection	August 2009	November 2010
Mine Pre-Stripping	October 2009	March 2011
Mill Building Erection	October 2009	April 2010
Mill Equipment Installation	May 2010	February 2011
Mill Commissioning	-	March 2011
Mill Commercial Production	-	April 2011

The underlying philosophy for the schedule is to mobilize and start construction activities immediately after the issuance of the environmental permit. Most of the concrete foundations have to be poured before winter 2009-2010. Some specific concrete foundation work will continue into the first quarter 2010 and will require specific cold weather protection measures. In order to attain this target, the detailed engineering for the process building and the grinding equipment foundation must be completed at the end of the first quarter 2009.

The project main access, borrow pit and quarry preparation will be done before the issuance of the environmental permit.

Equipment installation is scheduled to begin in the second quarter 2010 after structural steel completion and overhead crane availability.

The 120 kV transmission line is critical to supply power for the testing and commissioning of the process equipment. The construction works of the 120 kV electrical transmission line are under Hydro-Québec management.

Leach tank construction will be scheduled during the summer of 2009 and the summer of 2010 to optimize the productivity.

Truck shop facility construction is scheduled to start in June 2009 to be completed in time for the heavy mining equipment assembly starting in September 2009.

Figure 12.2 shows the Summary Schedule.

Figure 12.2: Summary Schedule (page 1 of 3)

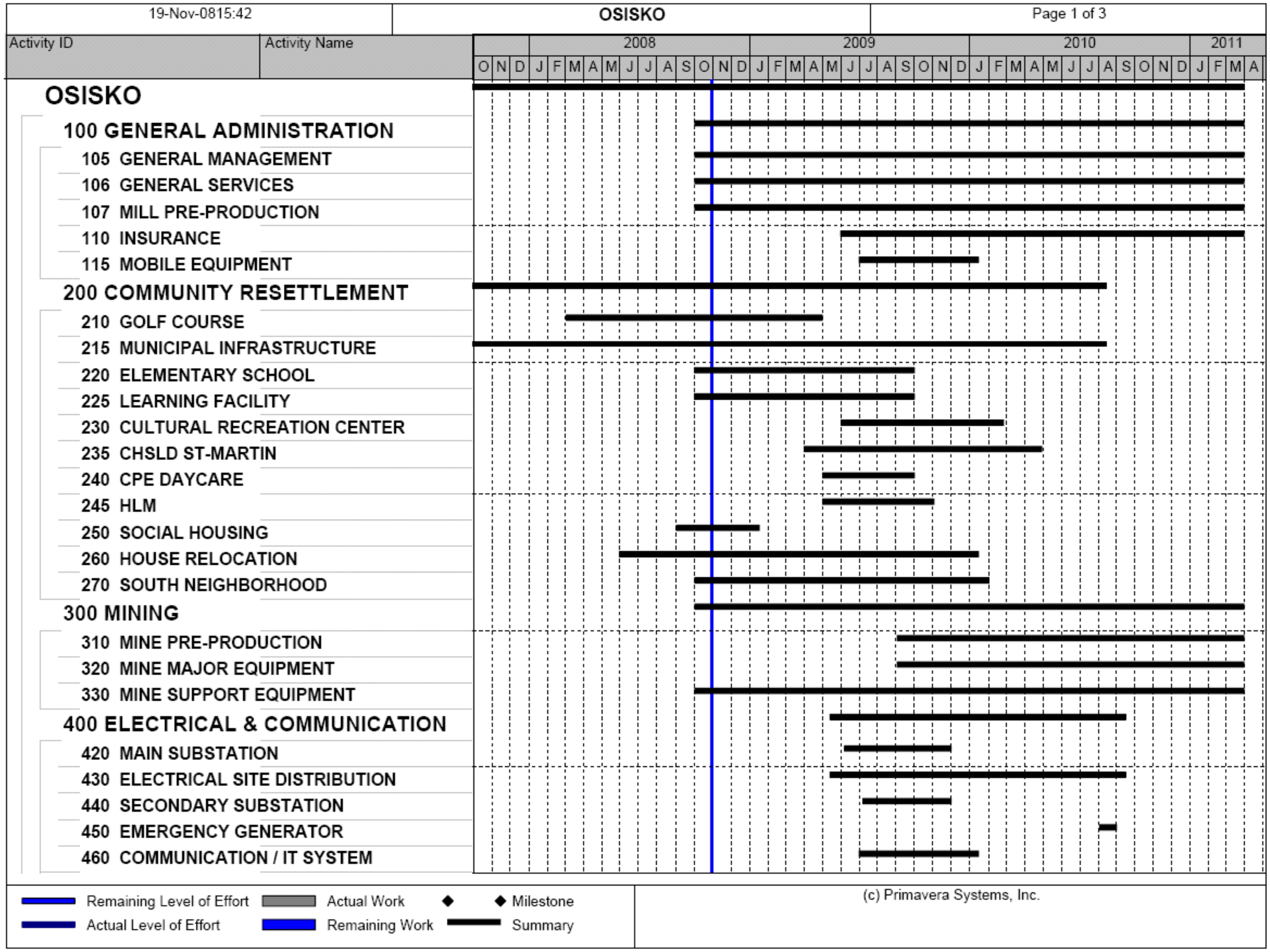


Figure 12.2: Summary Schedule (page 2 of 3)

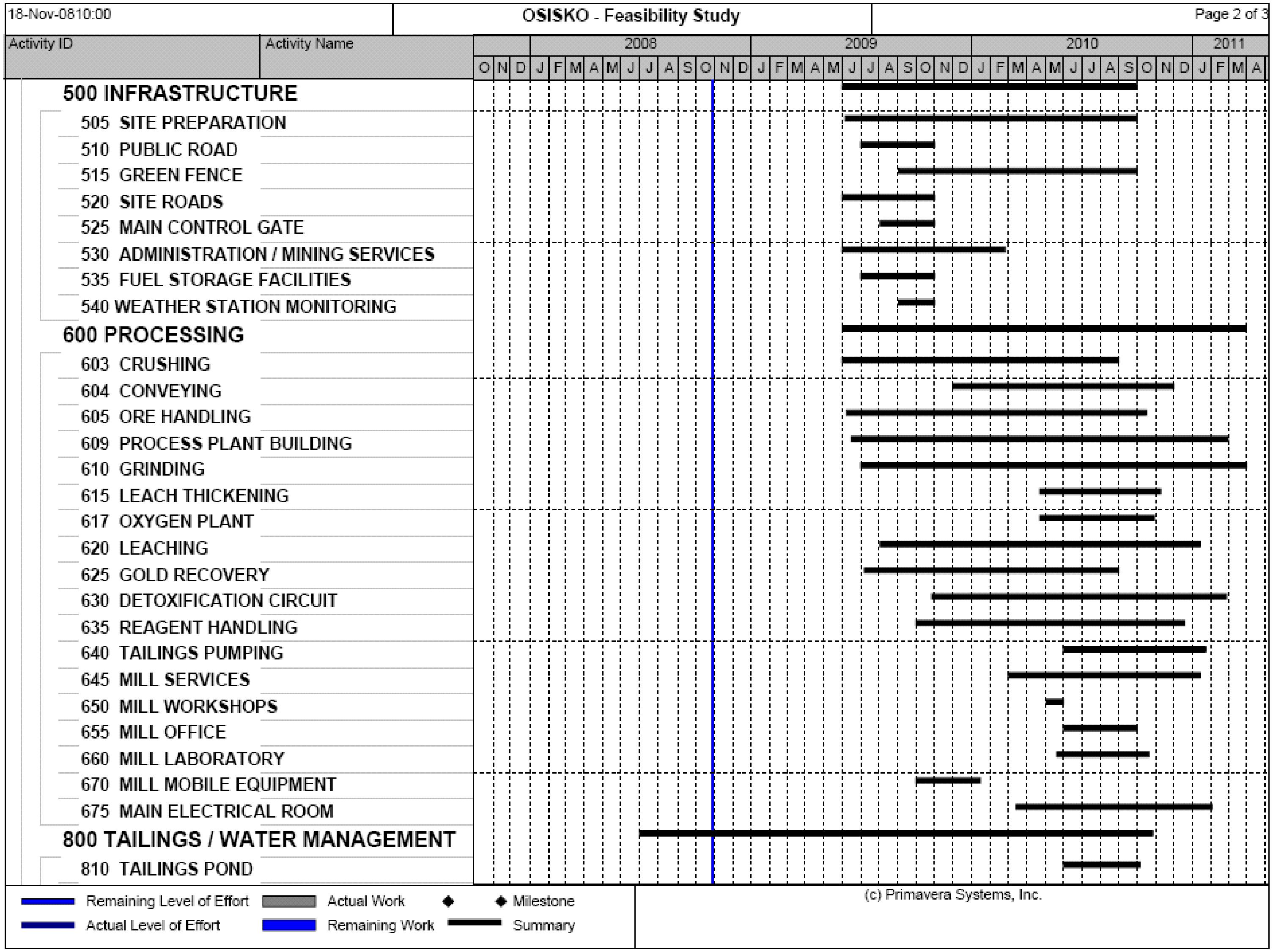
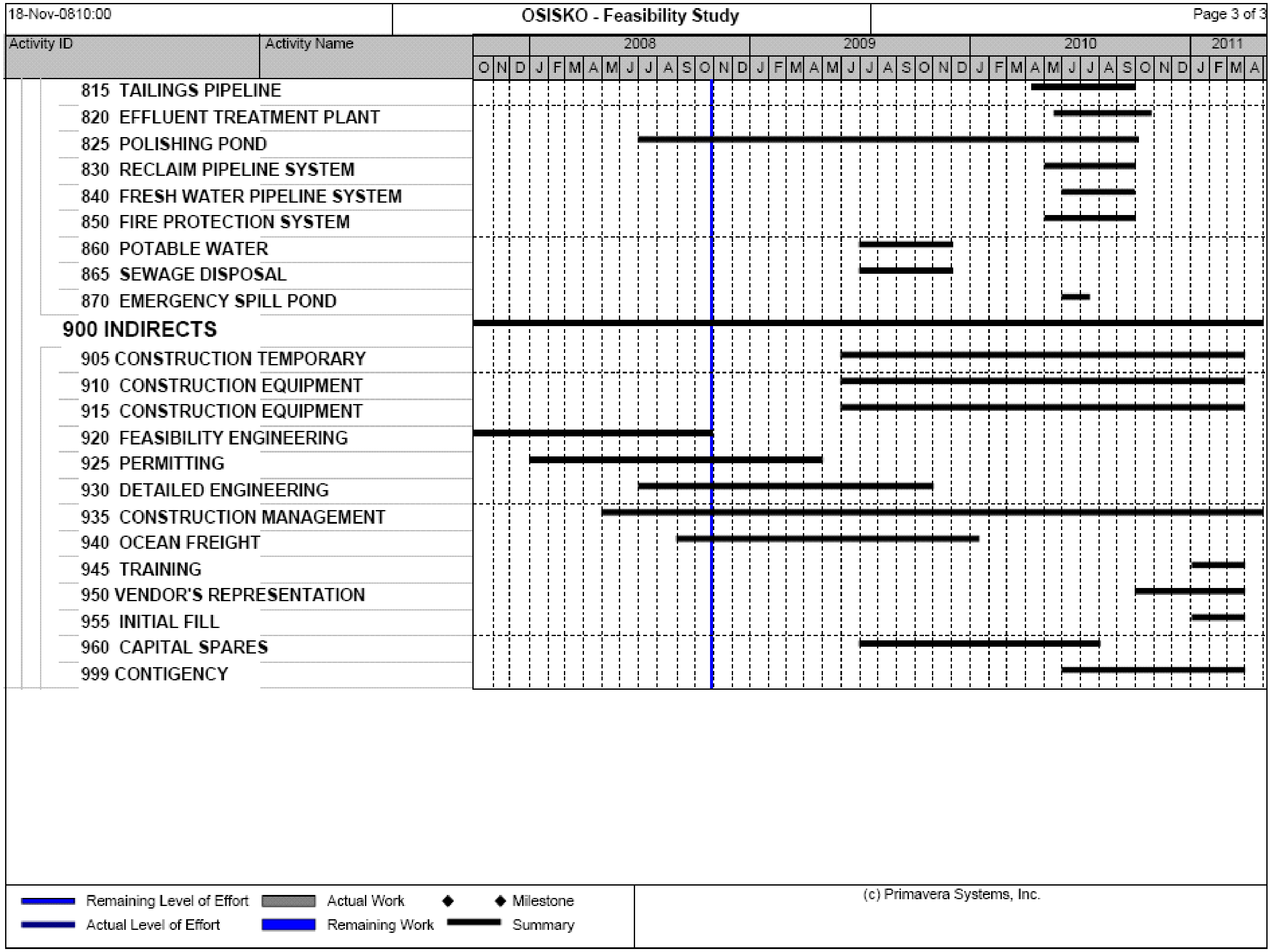


Figure 12.2: Summary Schedule (page 3 of 3)



12.4 Construction Labour

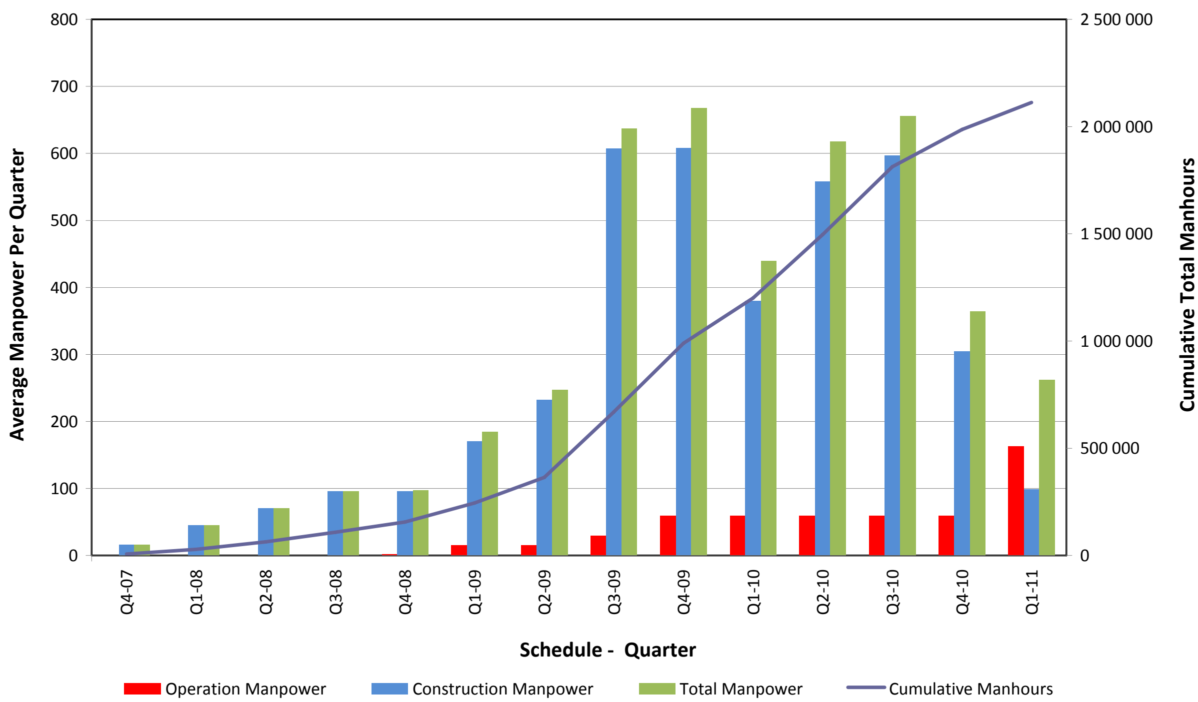
The total construction labour, including pre-production works to be executed by the Canadian Malartic operations group, is estimated at 2.1 million manhours.

A peak of approximately 600 construction workers on site is expected.

The manpower schedule is based on a 50-hour workweek for earthworks and concrete. For all other disciplines, the work/week is based on 40 hours.

Figure 12.3 shows the cumulative manpower distribution per quarter for the total project.

Figure 12.3: Manpower Distribution per Quarter



13.0 CAPITAL COST ESTIMATE

13.1 Summary

The capital cost estimate for the Canadian Malartic project is $788.9 M based on costs compiled in the 3rd quarter of 2008 in United States dollars. The exchange rate used for conversion from Canadian to United States dollars was CDN$1.18 = US$1.00.

The estimate includes no allowance for escalation in prices or fluctuations in currency exchange rates.

The capital cost summary is presented in Table 13.1 and reflects a level of accuracy of ± 10%.

Table 13.1: Capital Cost Summary

DESCRIPTION	CAPITAL COST ($M)
General Administration	14.5
Community Resettlement	87.0
Mining	136.7
Electrical & Communication	19.5
Infrastructure	29.7
Processing	348.0
Tailing and Water Management	15.3
Indirects	72.7
Total:	**723.3**
Contingency	65.6
Grand Total:	**788.9**

13.2 Basis of Estimate

13.2.1 General

At the time of estimate, the project engineering had progressed to an advanced stage including partial completion of detailed engineering. Procurement was completed for major and secondary process equipment in order to advance the detailed engineering and to secure pricing and delivery dates.

Firm prices with no escalation were obtained through the early procurement of the above process equipment as well as all the mining production and service equipment.

For all other equipment required for the mine and processing plant area detailed budget quotations were received. No equipment was estimated based on historical information from other projects.

Structural and civil works design was sufficiently advanced to obtain accurate estimates of quantities for these two major commodities. Unit prices were applied to these quantities based on actual unit prices used on recent projects in the region and on discussions with local contractors.

Labour rates and the availability of labour were discussed with local contractors who are now or have been recently very active on projects in the Abitibi area. The results from these discussions are reflected in the development of labour rates as presented below and in the installation man-hour estimates.

The neighborhood relocation program is approximately 60% committed at the present time.

13.2.2 Quantity Estimates

The quantities for earthwork, concrete, structural steel and architecture works were estimated by the design group based on engineering drawings for these disciplines.

Mechanical bulk quantities were taken off from existing drawings and sketches provided by engineering. The mechanical quantities and unit prices were reviewed and confirmed by a competent fabricator with experience in shop design and fabrication of this type of equipment.

Piping quantity take offs were based on completed piping diagrams for both process and water systems. The piping diagrams were considered to be developed to the level of basic engineering and will serve as a basis for making final P&IDs.

Electrical and Instrumentation bulk material quantities are based on drawings and lists provided by engineering. These quantities were based on preliminary P&IDs, electrical single line diagrams and on detailed general arrangement drawings for all process facilities.

Unit pricing depends on both labour rates, expressed as crew rates, and on construction productivity and on the price of bulk construction commodities such as concrete, steel, cables/cable tray, etc. In order to minimize the error in estimating the site construction works a detailed investigation of labour crew rates, productivity and commodity pricing was carried out as described below.

13.2.3 Labour Rates

The hourly labour rates have been based on rates established by the Association de la Construction du Québec (ACQ) industrial sector in the date of April 27th, 2008.

A heavy industry premium allowance and an overtime allowance, which is usually from 1 to 2 hours per day, depending on the trade, was fixed and constitutes the average direct hourly rate.

In addition to the direct hourly rate, indirect fees are applied such as:

- Mobilization/demobilization costs of the contractor
- Profit and administration
- Small tools
- Individual safety protection equipment
- Construction mobile equipment (except cranes over 45 t capacity)
- Crew supervision labour
- Subsistance allowance for housing and transportation.

Based on the above criteria, the following table presents the crew labour rates that were applied in the capital cost estimate.

Table 13.2: Crew Labour Rates

Discipline	Crew Rate ($/h)
Earthwork	129.66
Concrete	88.98
Structural steel	111.02
Mechanical	97.46
HVAC	86.44
Piping	91.53
Electrical and instrumentation	86.44
Architectural	97.46

13.2.4 Labour Productivity

Installation man-hour estimates were based on using composite crews at the rates shown above. The appropriate productivity rates were used and are considered specific for the Abitibi region.

Local contractors who are presently executing work or have recently completed projects in the Abitibi region were consulted on labour productivity and on labour effort required for the major installation tasks. The mix of local versus non-local labour used for the estimate was based on the advice of local contractors.

13.2.5 Bulk Material Costs

The unit price for the supply and placement of concrete has been determined in consultation with local contractors. Costs per cubic meter of concrete, in-place, were determined for 6 categories of concrete: lean concrete, building foundations, equipment foundations, structural concrete, slabs on grade and elevated slabs.

Structural steel is divided in 3 distinct categories: primary steel, secondary steel and auxiliary steel. Unit costs of materials are based on shop fabrication and shop painting, while installation prices include field erection and fitting.

All the piping costs are provided by suppliers.

More than 75% of the electrical costs are firm price quotes.

All the instrumentation costs are provided by suppliers.

13.3 Estimate Exclusions

The costs related to the following items are excluded from the capital cost estimate:

- Cost of remediation of hazardous material
- Sales taxes
- Escalation of equipment and bulk material costs after the 3rd quarter of 2008.
- Weather delays

- Lost time due to civil unrest or strikes
- Force majeure issues
- Issues beyond the control of the Owner.

13.4 Detailed Capital Cost Summary

A capital cost summary is presented in Table 13.3.

Table 13.3: Capital Cost Estimate Summary

AREA	DESCRIPTION	EXPENDED TO DATE ($M)	COMMITTED ($M)	CAPITAL COST ($M)
100	**GENERAL ADMINISTRATION DESCRIPTION**			
105	General Management	0	0	4.2
106	General Services	0	0	4.1
107	Mill Pre-Production	0	0	2.3
110	Insurance	0	0	1.4
115	Mobile Equipment	0	0	2.5
	Sub-total:	**0**	**0**	**14.5**
200	**COMMUNITY RESETTLEMENT**			
210	Golf Course	1.0	0.4	1.4
215	Municipal Infrastructure	5.8	3.1	13.7
220	Elementary School	0.7	10.4	12.9
225	Adult Learning Facility	0.1	3.8	5.1
230	Culture and Recreation Center	0.1	0.3	5.9
235	Long Term Care Hospital Center	0.2	0.7	14.9
240	Daycare Center	0.4	2.7	4.0
245	HLM	0	2.0	3.1
250	Social Housing	0.5	1.9	2.1
260	House Relocation	13.7	1.9	20.8
270	South Neighbourhood Rehabilitation	0	0	3.0
	Sub-total:	**22.5**	**27.2**	**87.0**
300	**MINING**			
310	Mine Pre-production	0	0	36.6
320	Mine Major Equipment	0.0	77.3	85.7
330	Mine Support Equipment	4.7	1.4	14.4
	Sub-total:	**4.7**	**78.7**	**136.7**
400	**ELECTRICAL AND COMMUNICATION**			
410	120kV Transmission Line	0	0	0
420	Main Sub-station	0.4	3.8	11.3
430	Electrical Site Distribution	0	0	1.7
440	Secondary Sub-station	0	0	4.1
450	Emergency Generator	0	0	0.8
460	Communication/IT System	0	0	1.6
	Sub-total:	**0.4**	**3.8**	**19.5**
500	**INFRASTRUCTURE**			
505	Site Preparation	0	0	4.1
510	Public Road	0	0	1.9
515	Green Fence	0	0	3.7
520	Site Roads	0	0	1.4
525	Main Control Gate	0	0	0.4
530	Administration/Mining Services	0	0	16.2
535	Fuel Storage Facilities	0	0	1.9
540	Weather Station/Monitoring	0	0	0.3
	Sub-total:	**0**	**0**	**29.7**

Table 13.3: Capital Cost Estimate Summary (continued)

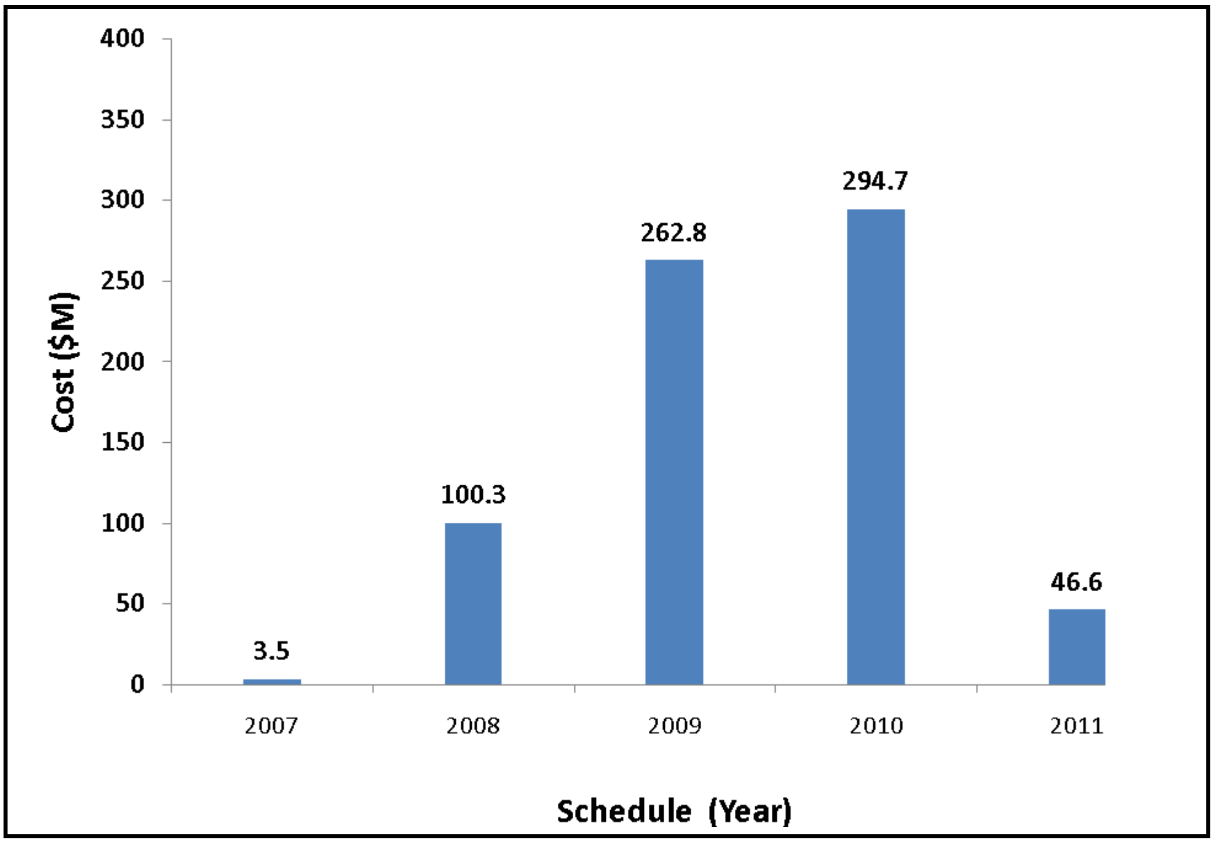
AREA	DESCRIPTION	EXPENDED TO DATE ($M)	COMMITTED ($M)	CAPITAL COST ($M)
600	**PROCESSING**			
603	Crushing	2.8	2.9	21.9
604	Conveying	0	3.6	8.9
605	Ore Handling	0.5	2.7	14.3
609	Process Plant Building	0	5.0	54.8
610	Grinding	37.1	44.7	130.7
615	Leach Thickening	0	0	9.6
617	Oxygen Plant	0	0	0.5
620	Leaching	2.1	2.4	31.7
625	Gold Recovery	0	21.3	29.1
630	Detoxification Circuit	0	0	5.3
635	Reagent Handling and Distribution	0	0	3.4
640	Tailings Pumping	0	0	15.1
645	Mill Services	0	0	7.5
650	Mill Workshops	0	0	0.8
655	Mill Office	0	0	1.3
660	Mill Laboratory	0	0	2.4
670	Mill Mobile Equipment	0	0	1.9
675	Main Electrical Room	0	0	8.6
	Sub-total:	**42.5**	**81.1**	**348.0**
800	**TAILINGS AND WATER MANAGEMENT**			
810	Tailings Pond	0	0	0.6
815	Tailings Pipeline	0	0	0.7
820	Effluent Treatment Plant	0	0	1.4
825	Polishing Pond	1.1	1.8	3.8
830	Reclaim Pipeline System	0	0	2.5
840	Fresh Water Pipeline System	0	0	2.2
850	Fire Protection System	0	0	3.0
860	Potable Water	0	0	0.4
865	Sewage Disposal	0	0	0.5
870	Emergency Spill Pond	0	0	0.3
	Sub-total:	**1.1**	**1.8**	**15.3**
900	**INDIRECTS**			
905	Construction Temporary Facilities	0	0	8.6
910	Construction Equipment/Tools	0	0	1.6
915	Construction Equipment/Maintenance	0	0	1.2
920	Feasibility Engineering	6.8	0	7.0
925	Permitting	0.1	0	0.5
930	Detailed Engineering	1.1	9.4	12.8
935	Construction Management	2.0	0	20.5
940	Ocean Freight	0.8	0	10.7
945	Training	0	0	0.5
950	Vendor's Representation	0	0	0.3
955	Initial Fill	0	0	5.6
960	Capital spares	0	2.0	3.4
	Sub-total:	**10.8**	**11.4**	**72.7**
	Total:	**82.0**	**204.0**	**723.3**
	CONTINGENCY			**65.6**
	GRAND TOTAL:	**82.0**	**204.0**	**788.9**

Osisko has entered into commitments on many long-lead items, construction contracts and fixed price quotations for an estimated cost of $286 M. No contingency has accordingly been applied to those amounts. A contingency provision of 15% has been estimated on the remaining costs.

Total capital cost includes expended to date and committed amounts as of September 30, 2008.

The capital costs are disbursed over a period of approximately 40 months from Q4 2007 to Q1 2011. The capital cost schedule is presented in Figure 13.1 below.

Figure 13.1: Capital Cost Disbursements per Year



The capital disbursements include mining equipment leasing costs up to the start of commissioning. A detailed cash flow distribution by project sub-area is presented in Table 13.4.

Table 13.4: Pre-production Cash Flow Distribution (k$ Per Quarter)

DESCRIPTION	Q4-07	Q1-08	Q2-08	Q3-08	Q4-08	Q1-09	Q2-09	Q3-09	Q4-09	Q1-10	Q2-10	Q3-10	Q4-10	Q1-11	TOTAL
100	- $	- $	- $	- $	117 $	628 $	691 $	2 065 $	2 065 $	819 $	819 $	819 $	819 $	5 677 $	14 519 $
105 General Management	- $	- $	- $	- $	117 $	117 $	117 $	117 $	117 $	117 $	117 $	117 $	117 $	3 167 $	4 222 $
106 General Services	- $	- $	- $	- $	- $	452 $	452 $	452 $	452 $	452 $	452 $	452 $	452 $	452 $	4 070 $
107 Mill pre-production	- $	- $	- $	- $	- $	58 $	58 $	58 $	58 $	58 $	58 $	58 $	58 $	1 868 $	2 335 $
110 Insurance	- $	- $	- $	- $	- $	- $	64 $	191 $	191 $	191 $	191 $	191 $	191 $	191 $	1 398 $
115 Mill mobile equipment	- $	- $	- $	- $	- $	- $	- $	1 247 $	1 247 $	- $	- $	- $	- $	- $	2 493 $
200	1 877 $	4 752 $	5 802 $	6 292 $	5 846 $	11 557 $	11 948 $	15 634 $	15 032 $	5 437 $	1 881 $	906 $	- $	- $	86 964 $
210 Golf Course	- $	44 $	265 $	579 $	451 $	43 $	- $	- $	- $	- $	- $	- $	- $	- $	1 382 $
215 Municipal Infrastructure	1 877 $	1 877 $	2 191 $	2 220 $	2 220 $	230 $	626 $	906 $	- $	- $	626 $	906 $	- $	- $	13 680 $
220 Elementary School	- $	140 $	475 $	475 $	- $	3 061 $	2 938 $	2 913 $	2 913 $	- $	- $	- $	- $	- $	12 916 $
225 Learning Facility	- $	21 $	63 $	101 $	101 $	1 301 $	1 206 $	1 206 $	1 143 $	- $	- $	- $	- $	- $	5 141 $
230 Cultural Recreation Center	- $	- $	- $	- $	- $	102 $	1 195 $	1 743 $	1 743 $	1 134 $	- $	- $	- $	- $	5 918 $
235 CHSLD ST-Martin	- $	48 $	144 $	144 $	144 $	144 $	1 647 $	3 851 $	3 765 $	3 765 $	1 255 $	- $	- $	- $	14 907 $
240 DayCare	- $	21 $	62 $	62 $	62 $	1 210 $	865 $	865 $	865 $	- $	- $	- $	- $	- $	4 012 $
245 HLM	- $	- $	- $	14 $	28 $	1 078 $	870 $	817 $	272 $	- $	- $	- $	- $	- $	3 079 $
250 Social Housing	- $	- $	- $	97 $	193 $	1 786 $	- $	- $	- $	- $	- $	- $	- $	- $	2 076 $
260 House Relocation	- $	2 601 $	2 601 $	2 601 $	2 601 $	2 601 $	2 601 $	2 601 $	2 601 $	- $	- $	- $	- $	- $	20 811 $
270 South Neighborhood	- $	- $	- $	- $	47 $	- $	- $	730 $	1 729 $	538 $	- $	- $	- $	- $	3 043 $
300	- $	- $	- $	- $	217 $	217 $	217 $	2 142 $	8 163 $	8 163 $	9 147 $	9 147 $	9 147 $	9 147 $	55 705 $
310 Mine Pre-Production	- $	- $	- $	- $	- $	- $	- $	1 926 $	5 777 $	5 777 $	5 777 $	5 777 $	5 777 $	5 777 $	36 589 $
320 Mine Major Equipment	- $	- $	- $	- $	- $	- $	- $	- $	2 169 $	2 169 $	3 153 $	3 153 $	3 153 $	3 153 $	16 950 $
330 Mine Support Equipment	- $	- $	- $	- $	217 $	217 $	217 $	217 $	217 $	217 $	217 $	217 $	217 $	217 $	2 167 $
400	- $	- $	- $	- $	- $	- $	1 519 $	9 041 $	6 790 $	- $	499 $	1 631 $	- $	- $	19 480 $
420 Main Substation	- $	- $	- $	- $	- $	- $	1 006 $	5 978 $	4 348 $	- $	- $	- $	- $	- $	11 331 $
430 Electrical Site Distribution	- $	- $	- $	- $	- $	- $	333 $	- $	- $	- $	499 $	831 $	- $	- $	1 663 $
440 Secondary Substation	- $	- $	- $	- $	- $	- $	171 $	2 273 $	1 653 $	- $	- $	- $	- $	- $	4 098 $
450 Emergency Generator	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	800 $	- $	- $	800 $
460 Communication / IT System	- $	- $	- $	- $	- $	- $	9 $	790 $	790 $	- $	- $	- $	- $	- $	1 589 $
500	- $	- $	- $	- $	- $	- $	1 281 $	9 637 $	11 876 $	3 383 $	1 309 $	2 198 $	- $	- $	29 684 $
505 Site Preparation	- $	- $	- $	- $	- $	- $	429 $	1 350 $	460 $	- $	460 $	1 350 $	- $	- $	4 050 $
510 Public Road	- $	- $	- $	- $	- $	- $	373 $	1 119 $	373 $	- $	- $	- $	- $	- $	1 864 $
515 Green Fence	- $	- $	- $	- $	- $	- $	- $	283 $	848 $	848 $	848 $	848 $	- $	- $	3 677 $
520 Site Road	- $	- $	- $	- $	- $	- $	280 $	839 $	280 $	- $	- $	- $	- $	- $	1 398 $
525 Main Control Gate	- $	- $	- $	- $	- $	- $	- $	272 $	93 $	- $	- $	- $	- $	- $	365 $
530 Administration/Mining services	- $	- $	- $	- $	- $	- $	199 $	5 007 $	8 432 $	2 535 $	- $	- $	- $	- $	16 173 $
535 Fuel Storage Facilities	- $	- $	- $	- $	- $	- $	- $	628 $	1 249 $	- $	- $	- $	- $	- $	1 877 $
540 Weather Station / Monitoring	- $	- $	- $	- $	- $	- $	- $	140 $	140 $	- $	- $	- $	- $	- $	280 $

Table 13.4: Pre-production Cash Flow Distribution (k$ Per Quarter) (continued)

600	- $	16 265 $	16 265 $	17 470 $	8 099 $	12 493 $	24 103 $	39 499 $	42 687 $	44 868 $	45 568 $	46 469 $	28 105 $	6 101 $	347 992 $
603 - Crusher	- $	- $	- $	1 204 $	1 611 $	1 611 $	2 200 $	4 398 $	5 420 $	2 383 $	1 329 $	1 404 $	370 $	- $	21 931 $
604 - Conveyor	- $	- $	- $	- $	- $	- $	- $	744 $	1 331 $	1 759 $	3 189 $	1 189 $	719 $	- $	8 932 $
605 - Ore Handling	- $	- $	- $	- $	- $	193 $	695 $	4 081 $	2 101 $	579 $	3 190 $	3 000 $	452 $	- $	14 291 $
609 - Concentrator building	- $	- $	- $	- $	- $	4 201 $	4 347 $	6 202 $	5 505 $	15 915 $	11 326 $	6 152 $	703 $	469 $	54 821 $
610 - Grinding	- $	16 265 $	16 265 $	16 265 $	6 488 $	6 488 $	6 488 $	11 413 $	12 922 $	7 225 $	6 626 $	13 135 $	9 034 $	2 102 $	130 715 $
615 - Pre-leach thickener	- $	- $	- $	- $	- $	- $	608 $	608 $	608 $	608 $	2 289 $	2 893 $	1 972 $	- $	9 584 $
617 - Oxygen plamt	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	458 $	- $	- $	- $	458 $
620 - Leaching	- $	- $	- $	- $	- $	- $	2 284 $	3 643 $	3 643 $	2 284 $	4 377 $	7 303 $	5 883 $	2 284 $	31 701 $
625 - Gold Recovery	- $	- $	- $	- $	- $	- $	4 999 $	4 993 $	5 156 $	7 333 $	3 697 $	2 963 $	- $	- $	29 141 $
630 - Detoxification	- $	- $	- $	- $	- $	- $	533 $	533 $	533 $	533 $	2 042 $	- $	443 $	665 $	5 283 $
635 - Reagent preparation	- $	- $	- $	- $	- $	- $	468 $	468 $	648 $	468 $	- $	382 $	929 $	- $	3 363 $
640 - Tailing	- $	- $	- $	- $	- $	- $	1 164 $	1 164 $	1 164 $	1 164 $	1 354 $	4 422 $	4 383 $	317 $	15 134 $
645 - Mill Services	- $	- $	- $	- $	- $	- $	317 $	317 $	317 $	699 $	1 837 $	1 785 $	2 252 $	- $	7 525 $
650 - Mill Workshop	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	634 $	204 $	- $	- $	839 $
655 - Mill Offices	- $	- $	- $	- $	- $	- $	- $	- $	419 $	242 $	169 $	508 $	- $	- $	1 339 $
660 - Mill Laboratory	- $	- $	- $	- $	- $	- $	- $	- $	- $	1 602 $	316 $	331 $	169 $	- $	2 417 $
670 - Mill Mobile Equipment	- $	- $	- $	- $	- $	- $	- $	935 $	981 $	- $	- $	- $	- $	- $	1 916 $
675 - Main Electrical Substation	- $	- $	- $	- $	- $	- $	- $	- $	1 939 $	2 072 $	2 735 $	795 $	795 $	265 $	8 601 $
800	- $	- $	- $	668 $	668 $	- $	668 $	1 275 $	869 $	311 $	3 738 $	6 975 $	89 $	- $	15 260 $
810 - Tailing Pond	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	144 $	433 $	- $	- $	577 $
815 - Tailing Pipeline	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	286 $	425 $	- $	- $	711 $
820 - Effluent Treatment plant	- $	- $	- $	- $	- $	- $	- $	104 $	311 $	311 $	318 $	266 $	89 $	- $	1 398 $
825 - Polishing Pond	- $	- $	- $	668 $	668 $	- $	668 $	668 $	223 $	- $	223 $	668 $	- $	- $	3 785 $
830 - Reclaim Pipeline System	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	717 $	1 753 $	- $	- $	2 470 $
840 - Fresh water pipeline system	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	726 $	1 432 $	- $	- $	2 157 $
850 - Fire Protection System	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	1 215 $	1 823 $	- $	- $	3 038 $
860 - Potable Water	- $	- $	- $	- $	- $	- $	- $	224 $	149 $	- $	- $	- $	- $	- $	373 $
865 - Sewage Disposal	- $	- $	- $	- $	- $	- $	- $	280 $	186 $	- $	- $	- $	- $	- $	466 $
870 - Emergency Spill Pond	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	109 $	177 $	- $	- $	285 $
900	1 604 $	1 698 $	2 868 $	6 510 $	6 780 $	6 635 $	7 090 $	8 882 $	8 074 $	2 930 $	10 571 $	23 728 $	25 202 $	25 682 $	138 253 $
905 - Construction Temporary Facility	- $	- $	- $	- $	- $	- $	393 $	1 178 $	1 178 $	1 178 $	1 178 $	1 178 $	1 178 $	1 178 $	8 638 $
910 - Construction Equipment / Tools	- $	- $	- $	- $	- $	- $	71 $	212 $	212 $	212 $	212 $	212 $	212 $	212 $	1 556 $
915 - Construction Equipment Maintenance	- $	- $	- $	- $	- $	- $	54 $	163 $	163 $	163 $	163 $	163 $	163 $	163 $	1 194 $
920 - Feasibility Engineering	1 604 $	1 604 $	1 604 $	1 604 $	535 $	- $	- $	- $	- $	- $	- $	- $	- $	- $	6 950 $
925 - Permitting	- $	94 $	94 $	94 $	94 $	94 $	31 $	- $	- $	- $	- $	- $	- $	- $	500 $
930 - Detailled Engineering	- $	- $	- $	2 387 $	2 387 $	2 411 $	2 411 $	2 411 $	804 $	- $	- $	- $	- $	- $	12 810 $
935 - Construction Management	- $	- $	1 170 $	1 756 $	1 756 $	2 122 $	2 122 $	2 122 $	2 922 $	589 $	1 389 $	1 389 $	1 389 $	1 756 $	20 483 $
940 - Ocean Freight	- $	- $	- $	670 $	2 009 $	2 009 $	2 009 $	2 009 $	2 009 $	- $	- $	- $	- $	- $	10 713 $
945 - Training	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	545 $	545 $
950 - Vendor's Representation	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	136 $	136 $	273 $
955 - Initial Fill	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	5 591 $	5 591 $
960 - Capital Spares	- $	- $	- $	- $	- $	- $	- $	787 $	787 $	787 $	787 $	262 $	- $	- $	3 411 $
999 - Contingency	- $	- $	- $	- $	- $	- $	- $	- $	- $	- $	6 841 $	20 523 $	22 123 $	16 101 $	65 590 $
Total cost per quarter	3 481 $	22 715 $	24 935 $	30 940 $	21 727 $	31 530 $	47 517 $	88 174 $	95 556 $	65 909 $	73 531 $	91 874 $	63 361 $	46 607 $	707 857 $
Total cost per year	3 481 $	100 316 $				262 777 $				294 675 $				754 465 $	

14.0 OPERATING COST ESTIMATE

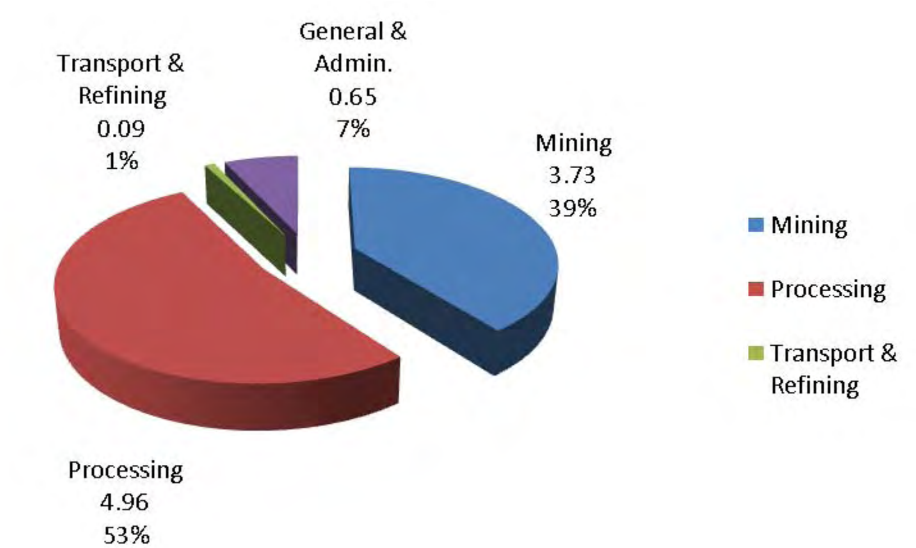
14.1 Operating Cost Summary

Operating costs are summarized in Table 14.1 and Figure 14.1 below and are based on a total of 183.3 Mt of ore milled over the mine life with an average of 20 Mt/y processed.

Table 14.1: Operating Cost Summary

Item	Average Annual Cost ($/y)	Unit Cost ($/t Milled)
Mining	74,890,756	3.73
Processing	99,572,028	4.96
Transport & Refining	1,736,655	0.09
General & Admin.	13,048,721	0.65
Total	**189,248,161**	**9.43**

Figure 14.1: Operating Cost Summary



The operating cost estimate includes labour, maintenance parts, consumables, freight, etc.

The unit prices for the various consumables, mainly grinding media and chemical reagents, have been obtained from reputable vendors presently supplying to the Abitibi region.

<u>Diesel</u>

The fuel cost has been assumed at $0.63/l including the refining margin, transportation and Federal taxes. Over the mine life a total of close to 210 Ml will be utilized. The fuel cost represents less than 8% of the total operating cost.

<u>Energy</u>

Table 14.2 shows that the total energy costs are assumed to be $1.31/t milled. This is an average rate based on a cost for the estimated electrical power demand of 78,000 kW and the cost for energy consumption as prescribed by Hydro-Québec and inclusive of all relevant taxes at $0.0373/kWh. The total energy costs include the average annual natural gas consumption estimated at 4,191,579 m^3/y. The natural gas portion is mostly used for HVAC and some specific process usage such as carbon regeneration kilns and carbon stripping circuit boilers. Natural gas represents approximately 5% of total energy cost and its unit costs are assumed to be $0.3118/$m^3$ based on discussions with Gaz Metro, the regional distributor, and including all relevant taxes.

Table 14.2: Total Energy Cost

Item	Average Annual Energy Cost ($/y)	Energy unit Cost ($/t Milled)
Energy - Natural Gaz	1,307,044	0.07
Energy - Electricity	25,084,145	1.25
Energy - total	**26,391,189**	**1.31**

<u>Labour</u>

Labour costs were estimated based on existing labour rates and benefits for the mining industry in the Abitibi area. In order to estimate the total manpower cost on the mine life basis, the project has been divided in three distinct departments that are:

- Mining (average of 297 employees with 46 staff, 251 hourly)
- Mill (90 employees with 21 staff, 69 hourly)
- General Services and Administration (77 employees with 55 staff, 22 hourly)
- TOTAL workforce (464 employees with 122 staff, 342 hourly).

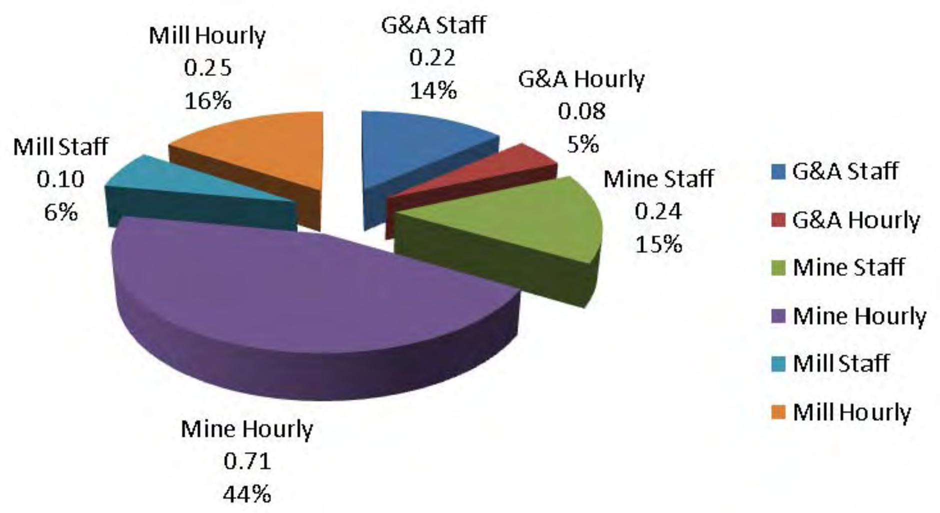
The annual and hourly salaries have been estimated according to studies on management and staff salaries in the mining industry and to scale hourly salaries in comparative mining companies. For the three areas of activities, the total cost has been obtained by calculating a benefit average cost of 36% and 43% for staff and hourly labour. The benefits include regular government benefits as well as social benefits provided by Osisko. The annual working hours have been estimated at 2080 and 2190 for dayshift and rotations schedule.

The average cost per annum is at $31,804,990 or $1.58/t milled (Table 14.3 and Figure 14.2) which represents 17 % of the total project yearly average operating cost. Staff salaries represent 35% of the total labour compared to hourly labour at 65%. Mine department labour cost represents 60% of the total manpower cost.

Table 14.3: Total Manpower Cost

Item	Average Annual Manpower Cost ($/y)	Manpower Unit Cost ($/t Milled)
G&A Staff	4,326,253	0.22
G&A Hourly	1,533,568	0.08
Mine Staff	4,752,619	0.24
Mine Hourly	14,161,773	0.71
Mill Staff	2,044,866	0.10
Mill Hourly	4,985,910	0.25
Total	**31,804,990**	**1.58**

Figure 14.2: Department Labour cost



14.2 Mine Operating Costs

The average manpower requirement for the mine operations is 297 employees. The detailed manpower requirements are presented on an annual basis in Table 5.35 in Section 5.0, Mining.

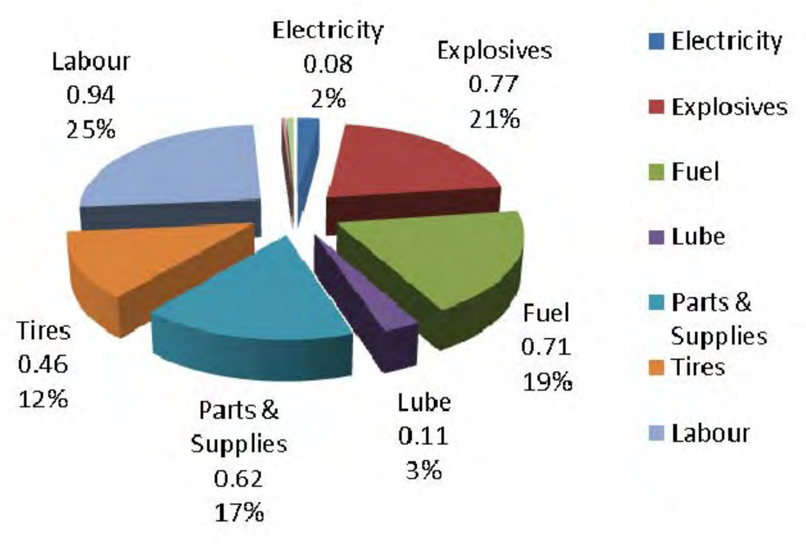
The average life of mine operating cost is $1.41/t mined or $3.73/t milled including rehandling cost. The total operating cost by category for the mine is presented in Table 14.4 below.

Table 14.4: Mine Operating Cost Summary

Item	Average Annual Mining Cost ($/y)	Average Mining Unit Cost ($/t Mined)	Average Mining Unit Cost ($/t Milled)
Electricity	1,638	0.03	0.08
Explosives	15,447	0.29	0.77
Fuel	14,308	0.27	0.71
Lube	2,139	0.04	0.11
Parts & Supplies	12,351	0.23	0.62
Tires	9,239	0.17	0.46
Labour	18,914	0.36	0.94
Training	289	0.01	0.01
Services	564	0.03	0.03
Total	**74,891**	**1.41**	**3.73**

Figure 14.3 presents the distribution of mining costs by category. Labour is the dominant cost center representing 25% of operating costs followed by explosives (21%), fuel (19%), parts & supplies (17%) and tires (12%).

Figure 14.3: Distribution of Mining Costs by Category



A detailed mining operating cost budget was constructed from basic cost elements such as detailed wage and salary costs, consumable prices, fuel prices and productivities. The fuel price assumed for estimating mining costs is $0.63/litre derived from a long term price projection for a crude oil price of $70.00/bbl and an exchange rate of 1.18 CDN$/US$. The major equipment and the principal support equipment hourly operating costs are presented in Table 14.5. Tire costs are deemed to reflect current market prices.

The average mining cost over the operations period is $1.41/tonne mined with the annual amounts and units costs per period presented in Table 14.6. This mining cost includes re-handled tonnes from stockpiles. Table 14.7 presents the fuel consumption included in the operating costs. Average fuel consumption during operations is 0.42 litres per tonne moved. Annual fuel consumption peaks at around 27.9 M litres in 2017 with a mine life average of 22.9 Ml.

Table 14.5: Mine Equipment Hourly Operating Costs

Major Equipment ($/hr)

Equipment Type	Hydraulic Shovel	Hydraulic Shovel	Front-End Loader	Rigid Truck	Track Dozer	Motor Grader	Wheel Dozer	Blasthole Drill	Blasthole Drill
Operating Cost Item	**O&K RH340-B**	**O&K RH200**	**CAT 994F HL**	**CAT 793F**	**CAT D10**	**CAT 16M**	**CAT 834**	**DTH 5.5"**	**DTH 8.5"**
Operating labour	35.85	35.85	34.74	32.52	33.64	33.64	33.64	34.74	34.74
Maintenance labour	73.92	66.53	36.21	28.87	21.28	16.20	17.83	17.22	23.99
Maintenance parts & overhaul	107.68	77.49	30.62	24.40	21.66	11.74	12.93	12.48	17.39
Special Items (GET, Drill tools...)	98.43	91.40	84.37	-	24.72	1.66	19.83	85.49	132.68
Fuel	-	180.98	90.00	91.31	37.50	15.62	30.12	44.00	54.41
Lubes	46.35	25.32	15.50	8.82	1.84	1.41	2.52	9.57	12.37
Electricity	50.34	-	-	-	-	-	-	-	-
Tires	-	-	43.51	82.81	-	5.61	25.67	-	-
Total	**412.57**	**477.58**	**334.94**	**268.72**	**140.63**	**85.87**	**142.53**	**203.50**	**275.58**

Support Equipment ($/hr)

Equipment Type	Hydraulic Excavator	Front-End Loader	Art. Dump Truck	Track Dozer	Motor Grader	Boom truck	Water Tanker	Low Boy & Tractor	Maint. Service Truck	Fuel & Lube Truck
Operating Cost Item	**CAT 345**	**CAT 980**	**CAT 740**	**CAT D6**	**CAT 14M**	**25ft Boom**	**76k L Tank**	**150T Low Boy**	**Maint. Truck**	**Fuel Truck**
Operating labour	33.64	34.74	32.52	33.64	33.64	28.96	32.52	-	-	-
Maintenance labour	13.34	11.30	8.86	9.48	11.45	3.33	16.77	55.10	1.74	11.01
Maintenance parts & overhaul	9.66	9.55	4.99	6.02	8.30	2.69	10.63	38.88	1.26	6.56
Special Items (GET, Drill tools...)	7.48	0.94	-	12.98	1.38	-	-	-	-	-
Fuel	25.31	20.56	17.44	16.06	13.12	6.01	35.19	58.12	6.01	16.05
Lubes	3.46	5.21	3.67	2.56	1.26	0.92	6.17	16.57	0.52	4.11
Tires	-	7.75	11.02	-	3.31	0.51	22.88	56.95	0.51	6.61
Total	**92.88**	**90.06**	**78.48**	**80.74**	**72.46**	**42.41**	**124.16**	**225.62**	**10.04**	**44.35**

Table 14.6: Operating Costs by Activity

Activity	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Costs (k$)												
Drilling	-	8,033	11,522	10,903	10,294	9,919	9,126	8,315	7,028	4,907	713	80,759
Blasting	-	12,893	19,251	18,680	18,191	17,730	16,510	15,223	13,139	9,757	1,584	142,958
Loading	-	4,656	8,398	8,260	8,480	8,395	7,544	6,659	5,113	3,092	543	61,139
Hauling	-	12,630	23,049	25,701	27,736	32,149	33,747	35,084	33,700	23,775	3,395	250,965
Ancillary	-	4,375	5,848	5,832	5,832	5,832	5,749	5,733	5,733	4,928	824	50,687
Support Equip.	-	1,617	2,184	2,184	2,184	2,184	2,184	2,184	2,184	2,184	363	19,455
Lighting & Dewatering	-	536	715	715	715	715	715	715	715	715	119	6,374
Re-handling	-	-	601	1,728	-	-	-	549	-	1,240	2,298	6,417
Geology	-	1,393	1,868	1,821	1,808	1,778	1,742	1,715	1,661	1,597	207	15,591
Operations	-	1,608	2,133	2,133	2,133	2,133	2,133	2,133	2,133	2,133	311	18,982
Maintenance	-	1,919	2,558	2,558	2,558	2,558	2,558	2,558	2,558	2,558	417	22,799
Engineering	-	660	879	879	879	879	879	879	879	879	99	7,791
Total	**-**	**50,321**	**79,007**	**81,392**	**80,809**	**84,272**	**82,886**	**81,748**	**74,843**	**57,765**	**10,873**	**683,916**
Unit Costs ($/t mined)												
Drilling	-	0.21	0.18	0.17	0.16	0.15	0.15	0.15	0.16	0.17	0.19	0.17
Blasting	-	0.34	0.30	0.29	0.28	0.28	0.28	0.28	0.29	0.34	0.42	0.29
Loading	-	0.12	0.13	0.13	0.13	0.13	0.13	0.12	0.11	0.11	0.14	0.13
Hauling	-	0.33	0.36	0.40	0.43	0.50	0.57	0.65	0.75	0.82	0.90	0.52
Ancillary	-	0.11	0.09	0.09	0.09	0.09	0.10	0.11	0.13	0.17	0.22	0.10
Support Equip.	-	0.04	0.03	0.03	0.03	0.03	0.04	0.04	0.05	0.08	0.10	0.04
Lighting & Dewatering	-	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.02	0.02	0.03	0.01
Re-handling	-	-	0.01	0.03	-	-	-	0.01	-	0.04	0.61	0.01
Geology	-	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.04	0.06	0.05	0.03
Mine Oper.	-	0.04	0.03	0.03	0.03	0.03	0.04	0.04	0.05	0.07	0.08	0.04
Mine Maint.	-	0.05	0.04	0.04	0.04	0.04	0.04	0.05	0.06	0.09	0.11	0.05
Engineering	-	0.02	0.01	0.01	0.01	0.01	0.01	0.02	0.02	0.03	0.03	0.02
Total Cost ($/t mined)	**-**	**1.31**	**1.24**	**1.27**	**1.26**	**1.32**	**1.39**	**1.52**	**1.67**	**2.00**	**2.88**	**1.41**
Mill Feed (k tonnes)		**15,056**	**20,075**	**20,075**	**20,075**	**20,075**	**20,075**	**20,075**	**20,075**	**20,075**	**7,672**	**183,329**
Total Cost ($/t milled)		**3.34**	**3.94**	**4.05**	**4.03**	**4.20**	**4.13**	**4.07**	**3.73**	**2.88**	**1.42**	**3.73**

Table 14.7: Fuel Consumption (Operations Period)

Fuel Consumption	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Fuel (kL)		11,650	20,912	22,933	23,421	26,258	26,849	27,857	25,715	19,270	4,202	209,066
Litres/tonne moved		0.30	0.32	0.34	0.37	0.41	0.45	0.51	0.58	0.61	0.48	0.42

14.3 Mill Operating Cost

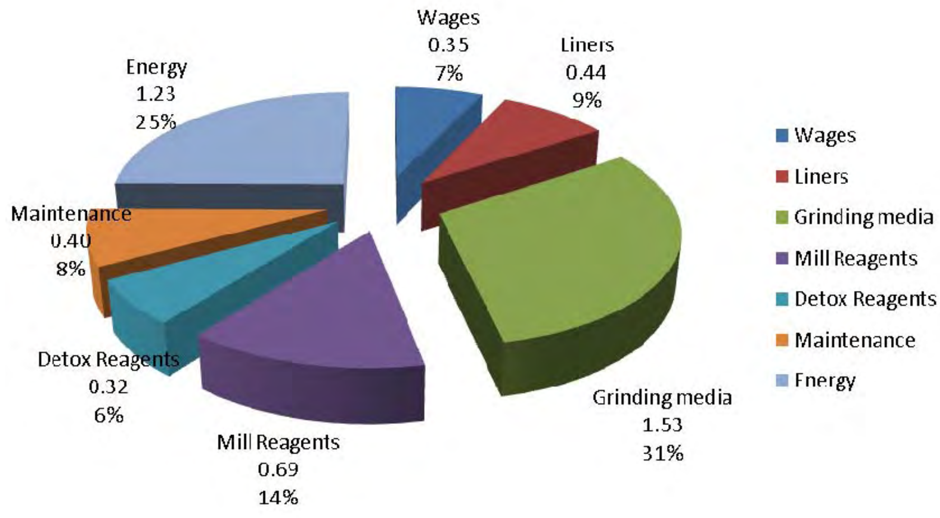
The mill operating cost, based on the life of mine, is averaging $4.96/t milled. The operating cost summary is presented in Table 14.8 and Figure 14.4 below.

Table 14.8: Mill Operating Cost Summary

Item	Average Annual Milling Cost ($/y)	Milling Unit Cost ($/t Milled)
Wages	7,030,776	0.35
Liners	8,802,273	0.44
Grinding media	30,726,193	1.53
Mill Reagents	13,825,172	0.69
Detox Reagents	6,404,897	0.32
Maintenance	8,030,000	0.40
Energy	24,752,717	1.23
Total	**99,572,028**	**4.96**

Figure 14.4 shows the proportion of the main items of the mill operating cost. Grinding media at 31%, energy at 25% and total reagents at 20% are the dominant costs in the mill operation.

Figure 14.4: Mill Operating Cost Summary



14.3.1 Manpower

The manpower requirement for the operations of the mill is 90 employees. The detailed manpower requirements are presented in Table 6.10 in Section 6.0 Mineral Processing and Metallurgical Testing. Wages including salary burden and benefits for the process plant are based on competitive employment conditions as compared to other mining operations of the Abitibi area. At $0.35/t milled the manpower cost represents 7% of the total mill operating cost. Depending on their respective sectors of work, employees will have various working schedules including weekends and night shifts. Premiums for working weekends and night shifts were taken into account in the cost estimation. An allowance was made for overtime hours worked for the maintenance labour.

14.3.2 Consumables

The process plant operating cost for consumables represents the main portion of the total process plant operating cost. It includes liners for crushers and mills, grinding media and the numerous reagents. Those costs were developed using consumption information derived from testwork as well as from best operating practice in the Abitibi area. All pricing information on the liners, grinding media and various reagents was received from suppliers. At $2.98/t milled, those costs represent close to 60% of the mill operating cost.

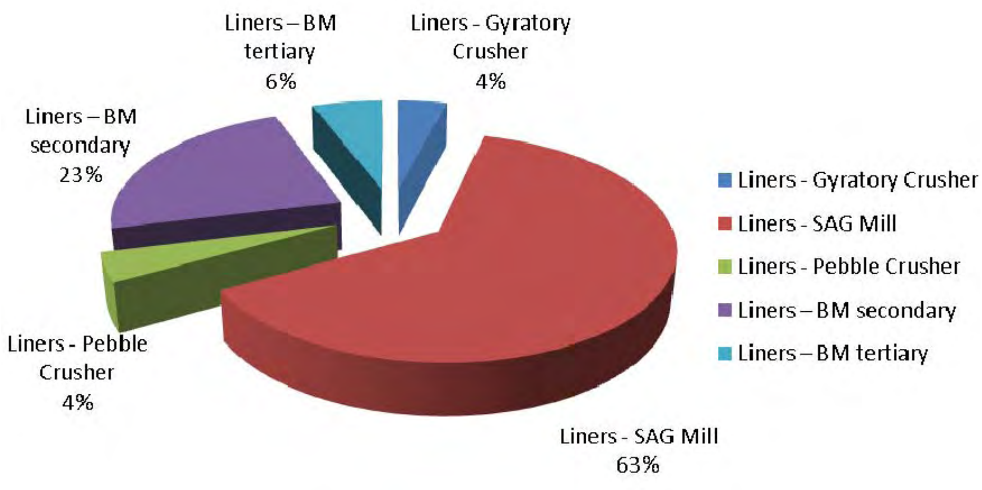
The process plant costs for consumables are presented in Table 14.9 below.

Table 14.9: Process Plant Detailed Consumables Cost

Item	Unit Cost ($/t)	Unit Consumption (kg/t)	Average Annual Total Cost ($)	Operating Cost ($/t Milled)
Liners - Gyratory Crusher	-	-	[redacted]	[redacted]
Liners - SAG Mill	-	-	[redacted]	[redacted]
Liners - Pebble Crusher	-	-	[redacted]	[redacted]
Liners – BM secondary	-	-	[redacted]	[redacted]
Liners – BM tertiary	-	-	[redacted]	[redacted]
Sub-total liners	**-**	**-**	**8,802,273**	**0.44**
Grinding Media – SAG	[redacted]	[redacted]	[redacted]	[redacted]
Grinding Media - BM secondary	[redacted]	[redacted]	[redacted]	[redacted]
Grinding Media - BM tertiary	[redacted]	[redacted]	[redacted]	[redacted]
Sub-total grinding media			**30,726,193**	**1.53**
Sodium Cyanide	[redacted]	[redacted]	[redacted]	[redacted]
Quick Lime	[redacted]	[redacted]	[redacted]	[redacted]
Flocculant	[redacted]	[redacted]	[redacted]	[redacted]
Activated Carbon	[redacted]	[redacted]	[redacted]	[redacted]
Anti-scaling	[redacted]	[redacted]	[redacted]	[redacted]
Oxygen	[redacted]	[redacted]	[redacted]	[redacted]
Caustic Soda	[redacted]	[redacted]	[redacted]	[redacted]
Nitric Acid	[redacted]	[redacted]	[redacted]	[redacted]
Sub-total process reagents			**13,825,172**	**0.69**
Flocculant	[redacted]	[redacted]	[redacted]	[redacted]
Quick Lime	[redacted]	[redacted]	[redacted]	[redacted]
SO_2 Liquid	[redacted]	[redacted]	[redacted]	[redacted]
Copper Sulphate	[redacted]	[redacted]	[redacted]	[redacted]
Hydrogen Peroxide	[redacted]	[redacted]	[redacted]	[redacted]
Sub-total detox reagents	**-**	**-**	**6,404,897**	**0.32**
Total	**-**	**-**	**59,758,535**	**2.98**

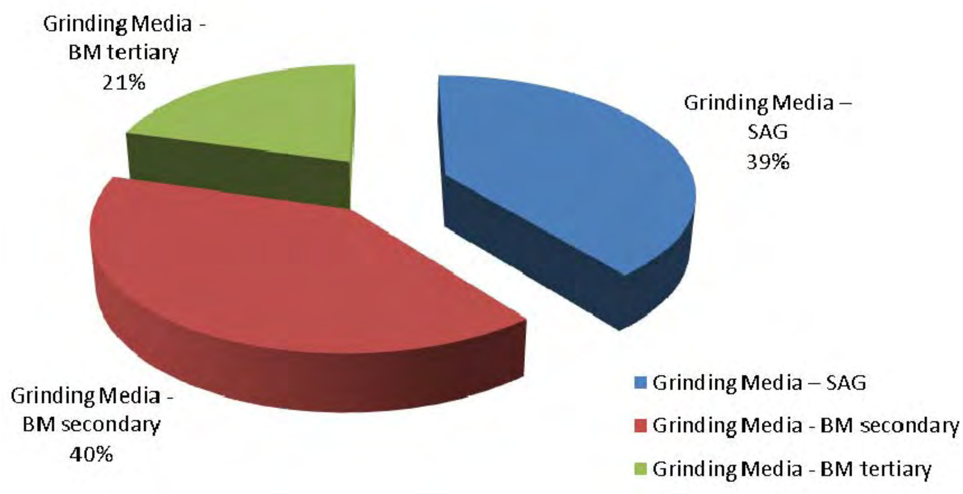
At $0.44/t milled the costs for liner replacements represent 9% of the total mill operating cost. The first gyratory crusher mantle and concave liner set, except for the lower concaves, will be made of Manganese steel. This option is considered to be a conservative start-up solution and minimizes the risk of cracking the concaves during the commissioning period. Lower concaves will be of alloy steel to maximize their life. The SAG mill liners cost is representing 64% of the liner costs, at $0.28/t milled. The liner costs were estimated based on grinding work index testwork and best operating practice for similar ore types. The first set of ball mill liners will be in rubber to minimize initial capital expenditures and allow a smoother commissioning period. Liner costs are based on competitive quotations from reputable liner manufacturers.

Figure 14.5: Crusher and Grinding Mill Liner Costs



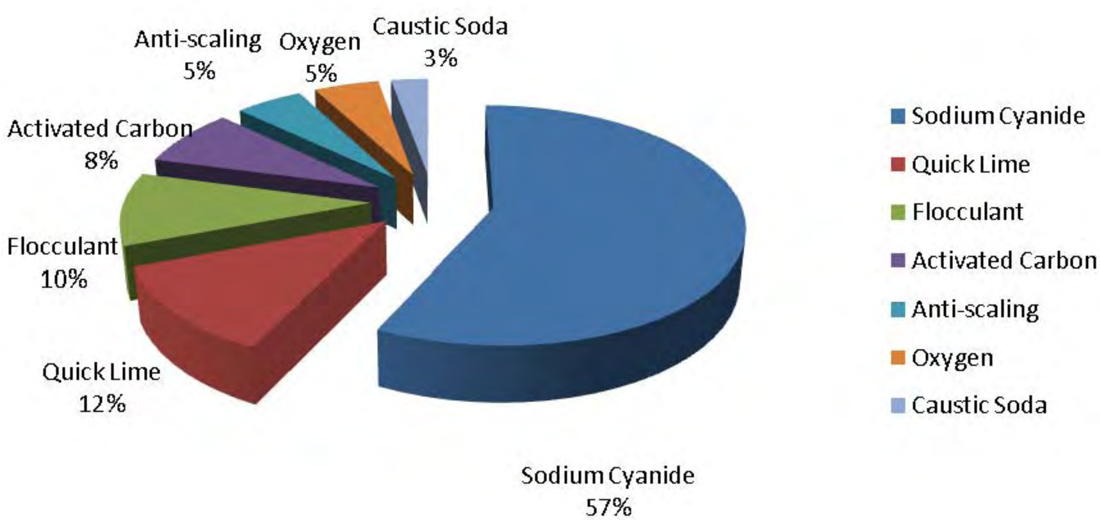
Grinding media cost is based on consumption estimated from the testwork and from experience in processing similar ore types. Grinding media represents a significant portion of the variable cost at $1.53/t milled. A great deal of effort will be required to maintain optimum grinding operating conditions. The stabilization of the grinding circuit will be the key to minimizing media consumption and also to maximize grinding efficiency thus thoughput. The cost of the grinding media was based on information obtained from suppliers. The scrap metal surcharge used in the media unit cost was based on reducing the current market price by 40%. This is based on current long term metal price projections.

Figure 14.6: Grinding Media Costs



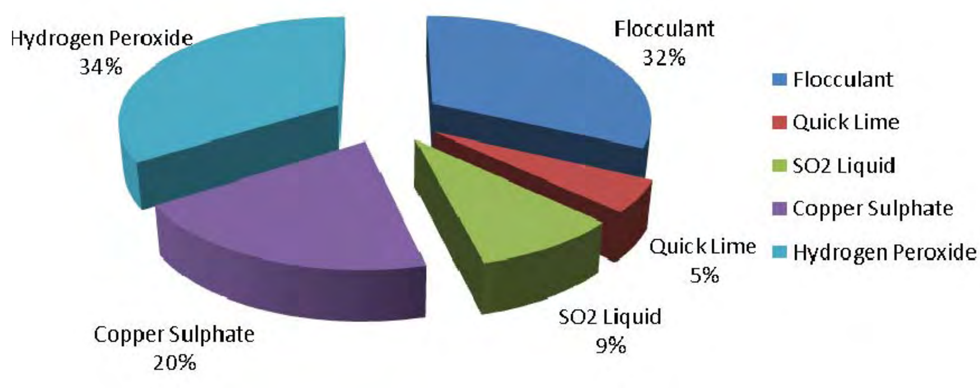
Cyanide and other reagent consumption estimates are based on exhaustive testing and on general experience in the industry for similar ore types.The main process reagent is cyanide which represents close to 60% of the cost of the process reagents.

Figure 14.7: Process Reagent Costs



The detoxification of the tailings represents a significant operating cost to comply with environmental requirements. The reagents associated with the CombinOx® process mainly involve liquid SO_2, hydrogen peroxyde and copper sulfate. The detoxification of the slurry will cost $0.32/t milled.

Figure 14.8: Tailings Detoxification Costs



14.3.3 <u>Maintenance Parts</u>

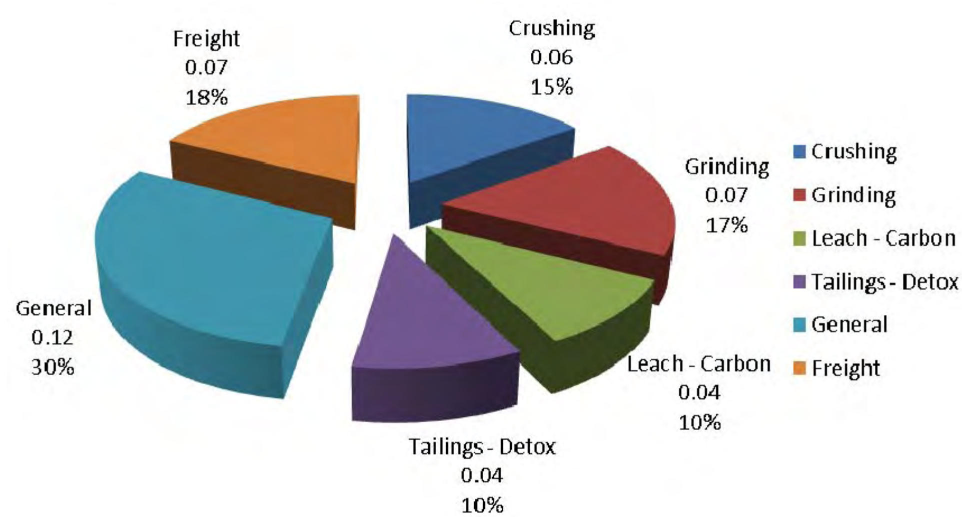
The maintenance parts cost for the process plant are based on operating experience from other high tonnage process plants with similar equipment.

Maintenance costs were determined to be $0.40/t milled or 8% of process plant total operating cost (Table 14.10). This cost includes mainly replacement parts relating to the maintenance of the various milling equipment such as conveyors, pumps, screens and so on. Figure 14.9 presents the main items included in the maintenance costs.

Table 14.10: Maintenance Parts Cost including Freight Charges

Item/Area	Average Annual Total Cost ($)	Operating Cost ($/t Milled)
Crushing	1,204,500	0.06
Grinding	1,405,250	0.07
Leach - Carbon	803,000	0.04
Tailings - Detox	803,000	0.04
General	2,409,000	0.12
Freight	1,405,250	0.07
Total	**8,030,000**	**0.40**

Figure 14.9: Maintenance Parts Cost



14.3.4 Energy

14.3.4.1 Electrical Energy

The electrical energy cost is calculated on the basis of the current Hydro-Québec "L" rate. The MRNF has recommended the authorization for Hydro-Québec to provide Osisko with a 85 MW block of power at Hydro-Québec's general high power tarif.

Electrical energy consumption is estimated at an average of 672,909,558 kWh/y. The main usage is in the process plant with approximately 95% of the available power used. The split between mine and process plant was estimated based on installed total power for each sector. The main energy consumer in the process plant is the grinding circuit with 55,225 kW installed only for the grinding mills. That circuit represents approximately 70% of the electrical energy consumed in the entire site. The electrical unit cost is $0.0373/kWh. This unit cost includes the base utility energy rate and the cost of electrical power demand.

The subscribed electrical demand power is 78,000 kW, with a peak demand of 85,300 kW during winter.

14.3.4.2 Natural Gas Energy

The natural gas energy cost is based on the October 2008 Gaz Métro rate. Natural gas consumption is estimated at 4,191,579 m^3/y, including all combustion inefficiency losses. Approximately 80% of the natural gas is used for process heating in the kilns and boiler and the remaining 20% is used for building heating. The natural gas unit cost is $0.3118/$m^3$. The maximum natural gas demand is 683 m^3/hr and occurs during winter, due to building heating loads.

14.3.4.3 Combined Energy

In order to add natural gas to electrical costs the natural gas is converted to its electrical equivalent using 10.525 m^3/kWh. The average cost of the total energy usage inclusive of all taxes is calculated at 0.03680/kWh with an average of 712,025,953 kWh equivalent per year. Approximately 3% of this energy is used for building heating, ventilation, air conditioning and lighting services and is subject to government taxes.

The combined total energy cost, including electricity, natural gas and applicable taxes is $1.31/t milled for a total of $26,391,188 per year. The power unit cost is very competitive when compared to other similar power consuming operations worldwide. The low power cost is a positive factor in favor of the project.

14.4 General Services and Administration

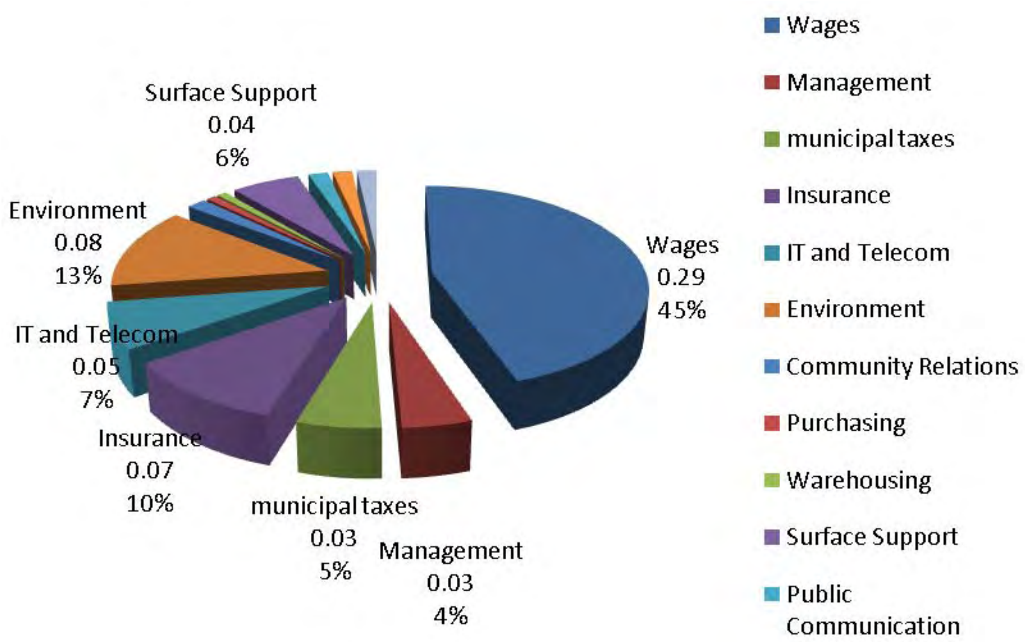
The General Services and Administration cost is estimated at $0.65/t milled. It includes the cost of all personnel relating to management, administration and surface support. In addition, this cost includes the supplies needed to maintain the activities related to the various non-operating support departments such as accounting, environment, health & safety and human resources. Also included are any taxes, legal and consulting fees and insurance costs.

The cost of General Services and Administration is presented in Table 14.11 below.

Table 14.11: General Services and Administration Cost

Item	Average Annual G&A Cost ($/y)	G&A unit Cost ($/t milled)
Wages	5,859,821	0.29
Management	576,600	0.03
municipal taxes	697,500	0.03
Insurance	1,395,000	0.07
IT and Telecom	948,600	0.05
Environment	1,674,000	0.08
Community Relations	223,200	0.01
Purchasing	111,600	0.01
Warehousing	111,600	0.01
Surface Support	781,200	0.04
Public Communication	223,200	0.01
Human Resources	223,200	0.01
Health and Safety	223,200	0.01
Total	**13,048,721**	**0.65**

Figure 14.10: General Service and Administration Costs



The labour cost accounts for the 77 employees required for management, administration and surface support tasks. The majority of this workforce relates to support operations activities: surface support, telecommunication, purchasing and warehousing, environment and so on. In general, the management and administrative staff will work 40 hours per week on day shift. Security and warehousing personnel will work a 12-hour shift per day to support the 24 hours per day operations requirements. The labour cost represents 45% of the General Services and Administration cost.

The costs for the computers, networking and all communication systems as well as radio maintenance, and the mine dispatch system is included in the IT and telecom services.

Environmental costs are comprised of various projects which all are in accordance with the ISO14001 certification as well as the environmental department activities including reclamation testworks.

The surface support cost includes building repairs, the numerous site roads and yards maintenance and, the organization and execution of the open pit pump moves. The thickened tailings system management is also the responsibility of this group.

15.0 FINANCIAL ANALYSIS

The following section presents all the elements of the financial model starting with the financial parameter assumptions, metal production and revenues, royalty agreements, operating costs, capital costs, sustaining capital, salvage value, working capital, closure and reclamation costs, financing, taxation and net project cash flow.

The financial results are presented on a CAPEX to completion basis from October 1, 2008, which represents the total project costs less the expenditures to that date. Also, the CAPEX to completion excludes lease payments under the CAT Finance facility for the mine mobile equipment, payable during the operational phase. In addition, the total project basis is also presented with the assumption that all prior expenditures are incurred in the first nine months of 2008.

The financial analysis is based on an annual production plan for the life of the project.

15.1 Financial Assumptions

The important financial assumptions influencing the economics of the project include the following parameters:

- Gold price in US$775/oz
- Silver price in US$10/oz
- Canadian dollar exchange rate (CDN$1.18:US$1.00)
- Oil price in US$70/bbl used to derive the diesel price
- Power cost in CDN$0.043/kWh.

15.1.1 Gold Price

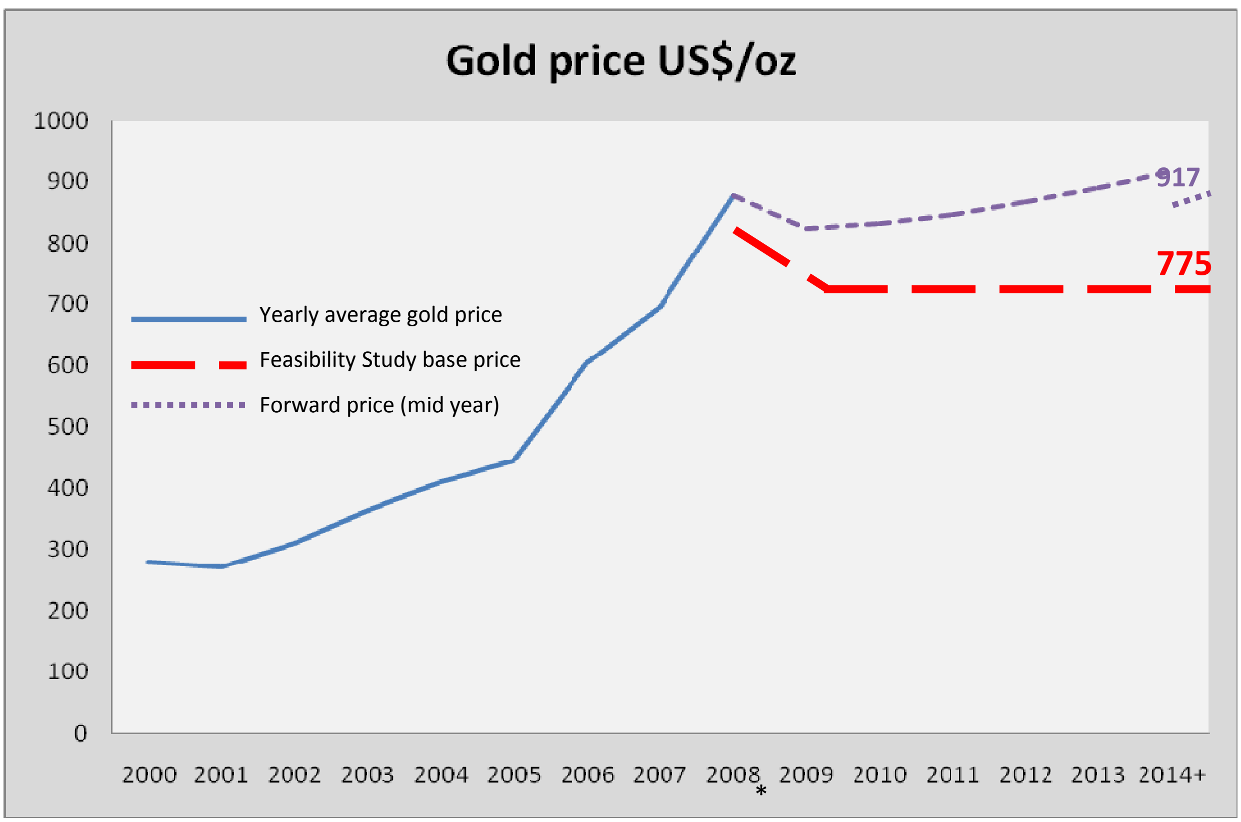
The gold market has been on an upward trend since 2001, reaching a record level in March 2008 of $1,011/ounce. Gold is traded on public markets and during the past few years, new financial products have been introduced to facilitate the accessibility of gold as an investment vehicle.

The base case financial model utilizes a gold price of $775 per ounce. This price level is assumed based on the maintenance of continued favourable market conditions which include:

- Continued weakness in the US dollar currency due to on-going current deficits and the level of indebtness of the US Government.
- Increased demand for gold mainly driven from the investing public.
- Continued orderly Central Banks' sales, the largest holders of above ground gold reserves.
- Declining annual gold production profile based on the low rate of discovery of new deposits and increased permitting risks.
- Continued geopolitical events creating uncertainty and enhancing the safe haven aspect of gold.
- Continued de-hedging thereby impacting supply/demand in the gold market.

The price level used is within financial and mining analysts' long-term forecast prices and forward selling curves. The gold price trend is shown in Figure 15.1.

Figure 15.1: Gold Price Trend



15.1.2 Silver Price

Metallurgical test work and historical data from the original Canadian Malartic mine have shown that the mineralization contains silver. Silver assays were not conducted on a systematic basis and are not modelled in the geologic block model of the deposit. As a result, silver credits were not used to estimate revenues in the Whittle optimization process. The estimated silver production is described in further detail in section 15.2.2. The base case silver price used to estimate by-product silver credits is $10/oz.

15.1.3 Canadian Dollar Exchange Rate

A significant portion of the capital and operating costs are denominated in Canadian dollars. These include:

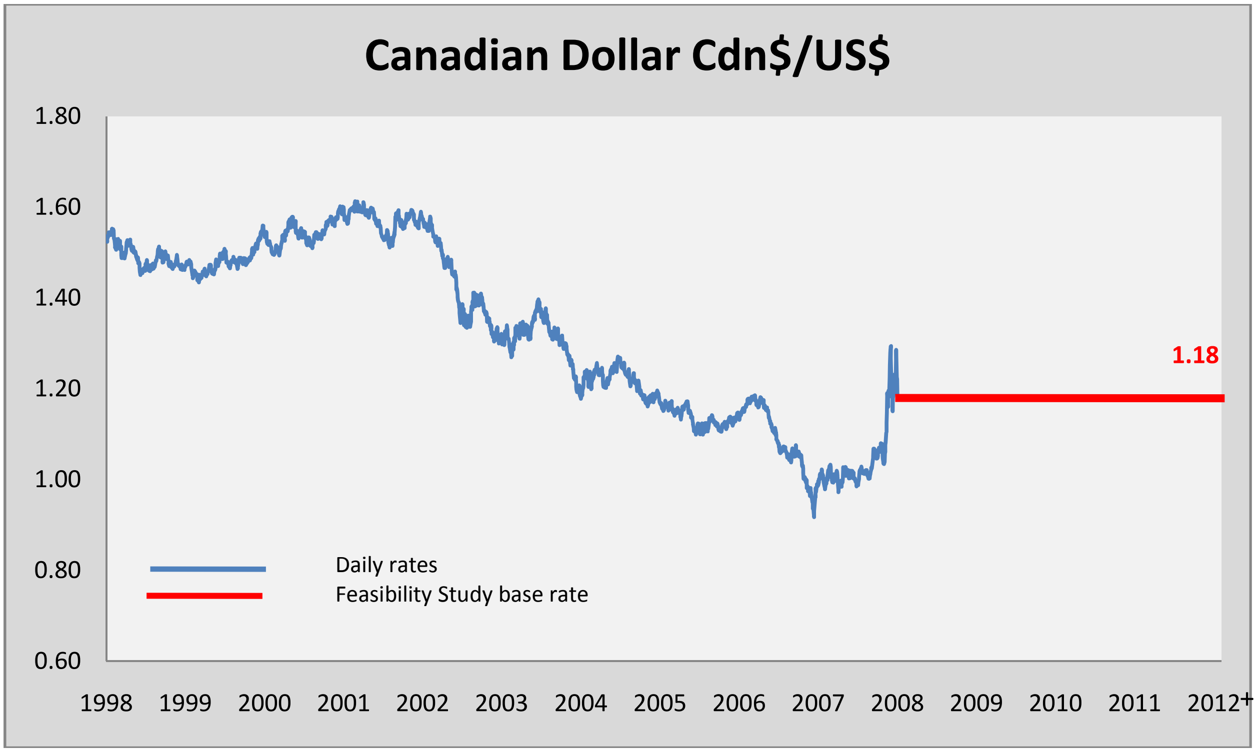
- Wages and salaries
- Hydro-electrical power
- Contractor costs
- Services costs
- Material costs
- Corporate income taxes (federal and provincial), Québec mining duties and municipal taxes.

It is estimated that some 60% of the capital costs are denominated in Canadian currency.

For the operations, the Canadian denominated costs are estimated at 56% of the total operating costs. All mining duties and taxes are denominated in Canadian dollars.

Figure 15.2 provides the exchange rate over the past ten years.

Figure 15.2: Canadian Dollar Exchange Rate

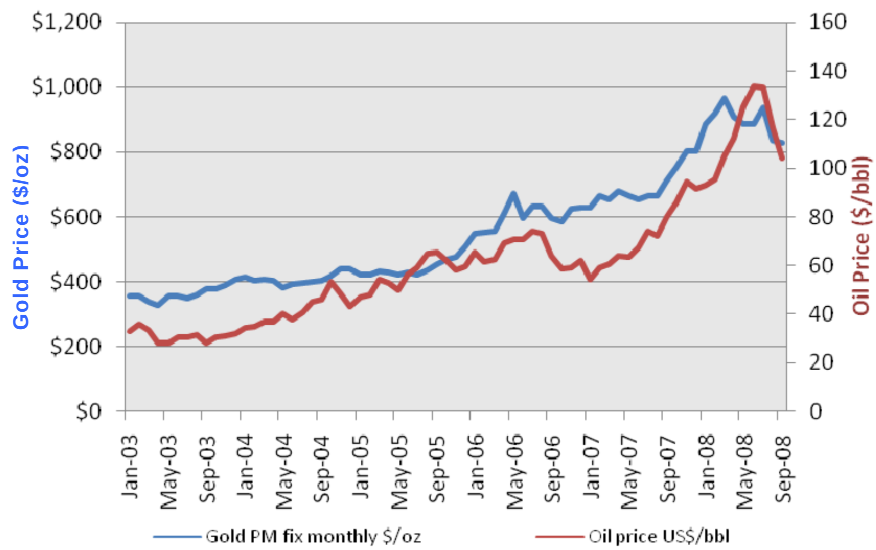


For the study purpose, an exchange rate of CDN$1.18 to US$1.00 has been utilized.

15.1.4 Oil Price

There is a strong relationship between the gold price and oil price in the market place as illustrated in Figure 15.3. A crude oil price of $70/bbl was used to estimate a delivered diesel price.

Figure 15.3: Gold and Oil Price History (Jan. 03 to Sept. 08)



The diesel price used for the project is based on the following assumptions:

• Crude oil price (US$/bbl)	70.00
• Crude oil price (CDN$/litre)	0.52
• Refining margin	15%
• Transportation (CDN$/litre)	0.10
• Diesel price excluding taxes (CDN$/litre)	0.70
• Diesel price incl. tax (CDN$/litre)	0.74
• Diesel price incl. tax (US$/litre)	0.62

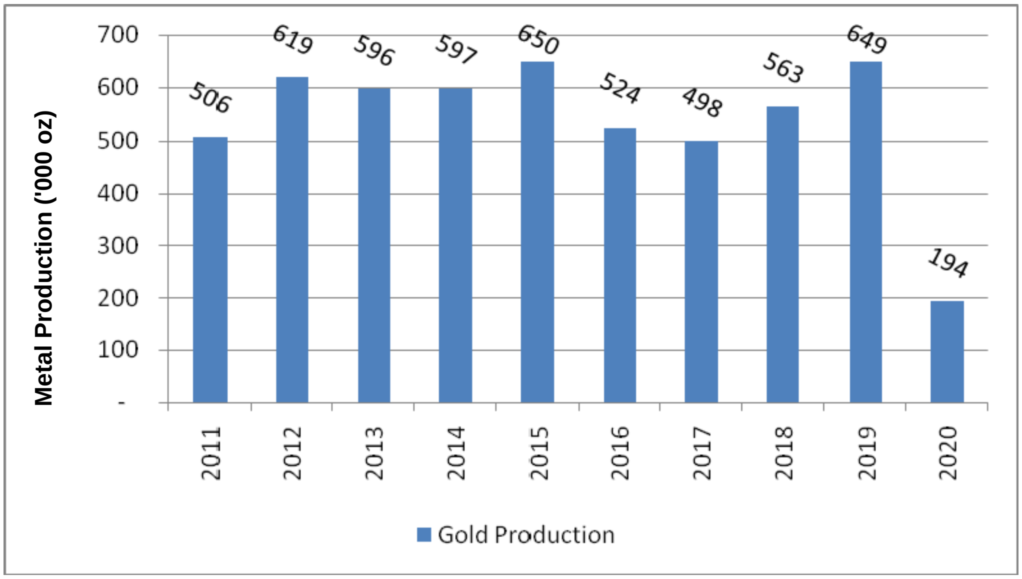
Provincial and federal sales taxes have not been included as they are assumed to be reimbursable.

15.2 Metal Production and Revenues

15.2.1 Gold Production

Total gold production over the project life is 5.4M ounces for an average annual production of 591 koz. The gold production per year is presented in Figure 15.3. The first year of production (2011) is a partial year of 9 months.

Figure 15.4: Annual Gold Production



15.2.2 Silver Production

Silver is present in the mineralization and the presence of silver in the ore was estimated during the metallurgical test work program. Silver content was estimated by establishing a Silver/Gold ratio with the same recovery zones in the mine established for gold.

The silver grade is related to the gold grade with the following equation: Ag = 0.638 + (0.987 x Au). The recovery of silver for the north, west and east sectors is estimated with the following equation: 57.55 + (7.983 x Ag grade) with a maximum of 85%. The estimated silver recovery from the south sector is 36.21+ (11.77 Ag grade) with a maximum of 70%. An average Silver/Gold ratio of 1.58 was determined for the deposit. The average silver recovery is estimated at 69.3%. These values were used to calculate silver production and silver credits.

The total life of mine silver production is 6.88 Moz for an average annual production of 753 koz.

Figure 15.5: Annual Silver Production

15.2.3 Revenues

Total gross gold revenue of $4,183 M is estimated over the project life at a gold price of $775/oz. Total silver revenues, or credits, amount to $68.8 M. The silver credits represent 1.7% of total gross revenues. The silver credits represent $0.38/t milled or $12.75 per recovered ounce of gold. Total gross revenues are $4,251 M (Table 15.1).

Table 15.1: Gross Revenues

Revenues	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Gold											
Gold production (koz)	506	619	596	597	650	524	498	563	649	194	**5,397**
Gold in circuit (koz)	(8)	-	-	-	-	-	-	-	-	8	**-**
Saleable gold (koz)	498	619	596	597	650	524	498	563	649	202	**5,397**
Gold price ($/oz)	775	775	775	775	775	775	775	775	775	775	
Gold revenue ($k)	386,198	479,927	462,201	462,897	503,491	406,439	386,159	436,425	502,600	156,261	**4,182,598**
Silver											
Silver production (koz)	633	785	762	759	823	678	643	713	827	257	**6,880**
Silver in circuit (koz)	(13)	-	-	-	-	-	-	-	-	13	**-**
Saleable silver (koz)	621	785	762	759	823	678	643	713	827	270	**6,880**
Silver price ($/oz)	10	10	10	10	10	10	10	10	10	10	
Silver revenue (k$)	6,208	7,852	7,617	7,593	8,233	6,776	6,430	7,126	8,266	2,697	**68,798**
Total revenue (k$)	392,406	487,778	469,818	470,490	511,724	413,216	392,589	443,551	510,867	158,958	**4,251,396**

15.3 Royalties

The Canadian Malartic property is subject to two royalty agreements, which are not overlapping. Under a royalty agreement with Royal Gold, a 3% net smelter return royalty is payable. The agreement includes a right to buy-down the royalty by 1.5% for CDN$1.5 million. The study assumes that Osisko will exercise this right at commercial production and a residual 1.5% NSR will be payable on certain claims. The total undiscounted value of the remaining 1.5% royalty is estimated at $47.2 million at the base case of $775/ounce. The royalty is based on the recoverability of 4.0 million ounces from these claims.

An additional 660,000 ounces recoverable are subject to a 2.5% gross overriding royalty agreement with a private holder. The total undiscounted value of this royalty is estimated at $12.8 million.

The overall royalty payments are estimated at 1.4% of gross revenues.

Table 15.2: Royalty Payments

Royalties	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
1.5% NSR to Royal Gold											
- Ore Tonnes (kT)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
- In-situ Gold (oz)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
- Recovered Gold (oz)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
2.5% GOR to a Private Party											
- Ore Tonnes (kT)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
- In-situ Gold (oz)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
- Recovered Gold (oz)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
Royalty Payments											
1.5% Royal Gold NSR (k$)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
2.5% Private Party GOR (k$)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
Total Royalty Payments (k$)	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]	[redacted]
% Royalty Payment	1.4	1.4	1.6	1.4	1.2	1.3	1.6	1.4	1.5	1.4	1.4

15.4 Operating Costs

Operating costs include mining, processing, transportation and refining of gold, and general and administrative services. Detailed operating cost budgets have been constructed from basic cost elements based on detailed wage scales, consumable prices, fuel prices and productivities. Operating cost information is presented in Section 14 of this study.

The transportation and refining costs used in the financial model take into account the estimated low purity of the dore bars due to the high silver content. The transportation and refining costs are based on the following terms and assumptions, which have been derived from quotations from refiners.

- Bullion with 55% gold purity
- Gold payout of 99.935%
- Silver payout of 99.0%
- Refining cost of CDN$1.25/oz (weight of bullion)
- Transportation and insurance cost of CDN$0.45/oz.

The transportation and refining cost per ounce of gold is estimated at $2.81/oz. The silver refining cost is estimated at $0.10/oz of silver. The average transportation and refining cost (including silver) is $2.94/oz of gold produced.

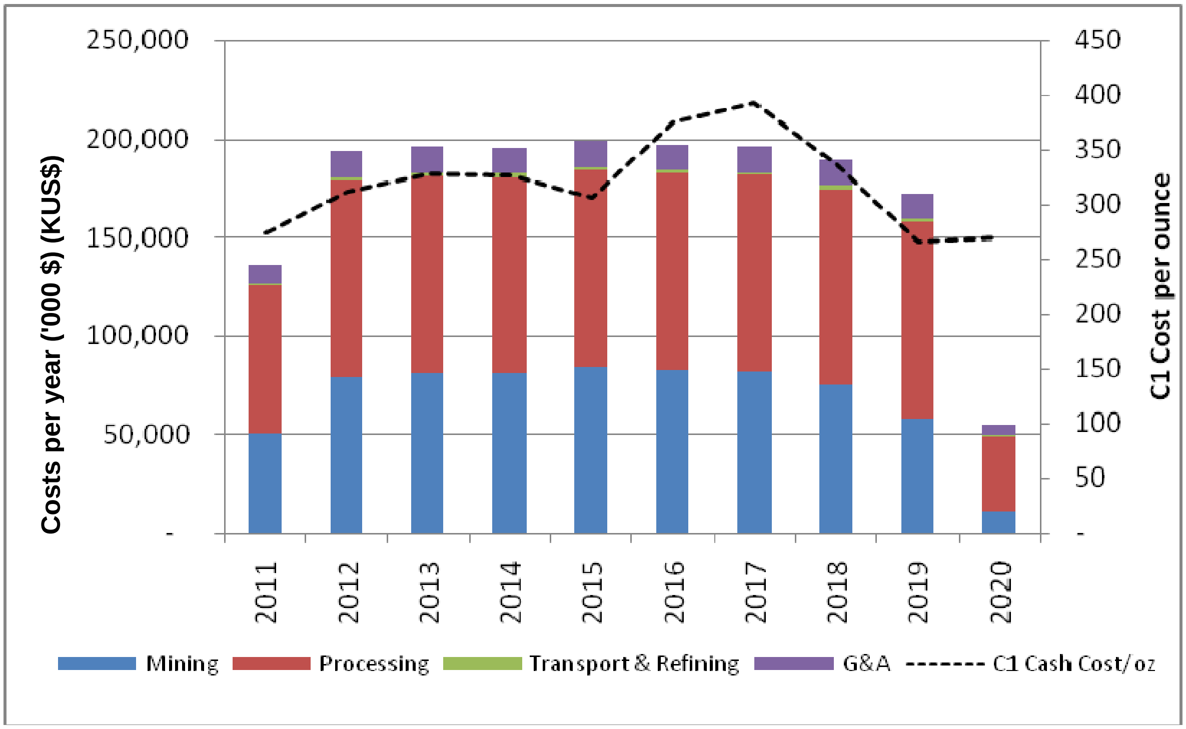
Table 15.3: Operating Cost Summary

Category	Total Mine Life Costs (k$)	Operating Costs ($/t milled)	Operating Costs ($/oz Au)
Mining	683,916	3.73	126.72
Processing	909,310	4.96	168.49
Transp. & Refining	15,859	0.09	2.94
G&A	119,164	0.65	22.08
Total (C1 Costs)	**1,728,249**	**9.43**	**320.23**
Royalties	59,942	-	11.11
Silver by-product credit	(68,798)	-	(12.75)
Total (C2 Costs)	**1,719,393**	**-**	**318.59**

Figure 15.6 presents the operating costs by year and the C1 cash costs per ounce. The operating costs for 2011 is from April to December or three quarters of a year. The cash costs are lower for the first 5 years with an average cost of $310.53/oz Au versus the average for the project of $320.23/oz Au. Cash

costs are higher in 2016 and 2017 due to the processing of lower grade ore. The average C2 cash costs including royalties and net of silver by-product credits is $318.59/oz Au over the project life as presented in Table 15.3.

Figure 15.6: Operating Costs



15.5 Capital Expenditures

The capital expenditures include initial capital and sustaining capital to be spent after commencement of commercial operations. The initial capital expenditures on a going forward basis are assumed to begin October 1st 2008.

15.5.1 Initial Capital

Total initial capital required is estimated at $788.9 M which includes $82.0 M in past expenditures (sunk costs) and includes a contingency of $65.6 M (Table 15.4). Past expenditures include outlays for the purchase of major mill equipment, relocation costs and detailed engineering. In the CAPEX to completion, most of the major mining equipment has already been negotiated with amounts already known with certainty.

The mining fleet purchased from Hewitt Equipment (Cat dealer) has been financed through CAT Finance with a capital lease. The amounts in Table 15.4 represent the values to be financed in those periods.

Table 15.4: Initial Capital Cost Summary

Initial Capital	**2008**	**2009**	**2010**	**2011**	**Total**
Mining equipment (k$)	5,401	58,755	35,978	-	**100,134**
Plant and infrastructure (k$)	21,566	252,089	259,051	37,429	**570,134**
Pre-production (k$)	-	-	19,840	16,749	**36,589**
Past Project Expenditures (k$)	82,015	-	-	-	**82,015**
Total (k$)	**108,982**	**310,844**	**314,869**	**54,177**	**788,872**

15.5.2 Sustaining Capital

The sustaining capital cost budget includes the purchase of additional units for the mobile equipment fleet and some replacement units. A significant portion (61%) of the mobile equipment sustaining capital relates to the purchase of additional haulage trucks as the tonnage increases the second year of operation and the haulage cycle gets progressively longer as the pit deepens over the years.

Tailings dam berm construction is also budgeted to allow for the raising and staging of thickened tailings deposition.

Total sustaining capital required over the project life is estimated at $94.7 M with the majority required for the purchase of mining equipment ($58.1 M) as presented in Table 15.5.

Table 15.5: Sustaining Capital Cost Summary

Sustaining Capital	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**Total**
Mining equipment (k$)	17,178	8,641	9,381	4,071	10,965	6,279	890	210	460	-	**58,075**
Plant and infrastructure (k$)	-	-	2,000	-	2,000	-	2,000	-	-	-	**6,000**
Pre-production (k$)	-	-	-	-	-	-	-	-	-	-	**-**
Tailings (k$)	-	2,238	2,238	2,935	2,935	4,735	4,735	4,735	4,735	1,295	**30,581**
Total (k$)	**17,178**	**10,879**	**13,619**	**7,006**	**15,900**	**11,014**	**7,625**	**4,945**	**5,195**	**1,295**	**94,656**

15.6 Salvage Values

Salvage values have been estimated for the major mine mobile equipment and major equipment items of the process plant (Table 15.6). Initial major mining equipment purchased for the project will have about 55,000 hours of operation at the end of the project and less for additional units purchased during operations. The estimated salvage value of the mining fleet was estimated by Hewitt (CAT dealer) and the plant equipment by BBA.

The salvage value of the mine mobile fleet includes residual values for the haul trucks, shovels, track dozers and wheel dozer. The salvage value of mining equipment estimated at $26.7 M represents 24% of the purchase cost of the equipment.

The process plant salvage value consists of the estimated residual value for the gyratory crusher, SAG and ball mills, SAG and ball mill motors, pebble crusher, thickener drive and rake mechanism, slurry pumps and conveyors. The total residual value of plant and infrastructure equipment estimated at $33 M represents 32% of the purchase price.

Table 15.6: Salvage Value

Salvage Value	2020	2021	Total
Mining equipment (k$)	26,687	-	**26,687**
Plant and infrastructure (k$)	-	33,197	**33,197**
Pre-production (k$)	-	-	**-**
Tailings (k$)	-	-	**-**
Total (k$)	**26,687**	**33,197**	**59,884**

15.7 Working Capital

Working capital is required to finance the supplies in inventory. Given that the project is easily accessible and close to suppliers the working capital requirements are relatively low given the size of the operation.

Production inventory is accounted for by subtracting gold build up in the circuit to estimate the saleable metal. It is estimated that 8 thousand ounces of gold and 13 thousand ounces of silver will be built-in inventory and recovered only at mill decommissioning.

A supplies inventory representing 2 months of consumption has been estimated and represents an investment of about $24 M. It is estimated that 1 month of supplies will be payable.

Accounts receivable representing 1 week of gold production will be in transit or at the refinery waiting to be sold will have to be financed with working capital. Accounts receivable will fluctuate between $8 M and $10 M.

A cash account of $5 M is planned as an additional provision to the working capital requirement.

An initial $22.7 M working capital requirement has been estimated and required in 2010 to purchase consumables prior to commercial production. This does not include initial fill requirements which are included in the capital budget.

15.8 Reclamation and Closure Costs

Reclamation and closure costs have been estimated to rehabilitate the tailings facility and waste dump, vegetate the surrounding area, dismantle the plant and associated infrastructures and perform environmental monitoring for a period of 15 years. The reclamation and closure cost is estimated at $45 M and is composed of the following:

• Tailings facility and waste dump ($M)	33.44
• Polishing pond closure ($M)	0.50
• Tree planting ($M)	2.67
• Dismantling of complex($M)	5.00
• Environmental monitoring ($M)	2.63
• Other ($M)	0.77
Total	**45.01 M**

15.9 Project Financing

The net capital investment of the project on a CAPEX to completion basis needs to be funded from various sources including cash resources, equity and/or debt. Osisko has entered into an agreement with CAT Finance to fund the acquisition of the mining mobile equipment fleet through a capital lease facility of US$83.25 M at an assumed interest rate of 7.25%.

In the financial model, it is assumed that $83.25 M will be financed as a lease buyback with an assumed effective interest rate of 7.25% to calculate interest payments. Arranging fees and commitment fees have also been taken into consideration in the financial model. Lease payments are made on a monthly basis and begin once the machine is commissioned according to the lease schedule with a duration of 60 months (or five years). During the pre-production period, $19.2 M in capital lease payments will have to be made. Therefore, the net financing of this facility is $64.1 M

The Project required funding is as follows:

Table 15.7: Required Project Funding

		Costs ($M)
Total Project Capital Costs		**788.9**
Less:		
Investment to September 30, 2008	82.0	
Net Capital Lease Financing	64.1	(146.1)
CAPEX to be funded		**642.8**
Working Capital		22.7
Lease Financing Fees		8.7
TOTAL		**674.2**

15.10 Taxation

The project is subject to three levels of taxation including provincial mining duties, provincial income tax and federal income tax. Initial tax pools available at year 2007 were considered in the model for estimating taxes. The available federal tax loss carryforwards and tax pools at year 2007 are as follows:

- Canadian Exploration Expense (CEE) (CDN$M) 26.28
- Canadian Development Expense (CDE) (CDN$M) 17.71
- Capital Cost Allowance pools (CCA) (CDN$M) 4.41
- Non-capital losses (CDN$M) 11.42

Provincial tax pools were assumed to be identical. These tax pools were converted at the assumed exchange rate of 1.18 to convert them into US$ in the financial model.

15.10.1 Québec Mining Duties

The Québec mining duties rate is 12% of taxable income. The taxable income is calculated by subtracting the closure and reclamation costs and various allowances from the EBITDA (Earnings Before Interest, Taxes, Depreciation and Amortization).

After all the depreciation pools have been used, a processing allowance is applied before arriving at the taxable income. This processing allowance is the minimum of 8% of the cost of the processing facility and 65% of income before the processing allowance, thereby reducing the effective Québec Mining Duties rate to 4.2%.

The total mining duties to be paid are estimated at $67.8 M over the project life.

15.10.2 Provincial Income Taxes

The provincial income taxes have been estimated from taxable income calculated by subtracting the various tax depreciation pools from the EBITDA.

This analysis has included a loss carry forward of CDN$11.42 M (or US$ 9.68 M).

The depreciation rates of the asset pools are identical to the federal income tax pools except for the Canadian Development Expense (CDE) pool where 100% can be claimed in calculating the provincial taxable income versus 30% at the federal level.

The applicable provincial tax rate is constant over time at 11.9%.

Cash taxes paid to the province are estimated at $156.2 M over the project life.

15.10.3 Federal Income Taxes

The federal income taxes have been estimated from taxable income calculated by taking into account the federal tax pools and their depreciation rates. The applicable federal tax rate is currently 19% with rates declining over time as follows:

- 2009: 19%
- 2010: 18%
- 2011: 16.5%

- 2012: 15%.

Cash taxes paid to federal government are estimated at $198.3 M over the project life.

15.11 Financial Results

15.11.1 Project Cash Flow

The project cash flows are presented on a total project basis and on a CAPEX to completion basis in Table 15.8 and Table 15.9 respectively. It should be noted that there is no impact on the tax calculation for these two cash flow presentations. The cash flow results are summarized in Table 15.7.

Table 15.8: Net Cash Flow Summary

Net Cash Flow	Total Project ($ M)	CAPEX to Completion ($ M)
Before Tax CF (NPV 0%)	1,573	1,655
Before Tax CF (NPV 5%)	920	1,001
After Tax CF (NPV 0%)	1,151	1,233
After Tax CF (NPV 5%)	650	731

On a total project basis the undiscounted after tax cash flow is $1,151 M.

15.11.2 Internal Rate of Return

- On a total project basis the before tax internal rate of return (IRR) is 24.1% and 20.5% after taxes.
- On a CAPEX to completion basis the before tax IRR is 28.8% and 25.1% after taxes.
- The IRR rates include limited leverage relating to the $83.3 million CAT Finance facility.

15.11.3 Payback Period

- The payback period on a total project basis for a capital cost of $788.9 M is 36 months.
- Payback period on a CAPEX to completion basis for a capital cost of $642.8 M is 32 months.

Table 15.9: Canadian Malartic Base Case Cash Flows – Total Project Basis (k$)

Project Cash Flows	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	Total
Revenue	-	-	-	392,406	487,778	469,818	470,490	511,724	413,216	392,589	443,551	510,867	158,958	-	-	**4,251,396**
Total Cash Costs (C1)	-	-	-	(136,250)	(193,447)	(195,766)	(195,185)	(198,801)	(197,049)	(195,833)	(189,118)	(172,292)	(54,509)	-	-	**(1,728,249)**
Royalties	-	-	-	(5,378)	(6,607)	(7,290)	(6,752)	(6,153)	(5,435)	(6,287)	(6,146)	(7,700)	(2,194)	-	-	**(59,942)**
Depreciation	-	-	-	(74,426)	(93,864)	(92,296)	(93,571)	(105,124)	(87,232)	(84,867)	(97,884)	(116,689)	(37,574)	-	-	**(883,528)**
Accrued Closure Costs	-	-	-	(3,700)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(1,885)	-	-	**(45,050)**
EBIT	**-**	**-**	**-**	**172,653**	**188,926**	**169,534**	**170,049**	**196,712**	**118,567**	**100,668**	**145,469**	**209,252**	**62,796**	**-**	**-**	**1,534,627**
add Depreciation	-	-	-	74,426	93,864	92,296	93,571	105,124	87,232	84,867	97,884	116,689	37,574	-	-	**883,528**
add Closure Costs	-	-	-	-	-	-	-	-	(5,270)	(5,270)	(5,270)	(5,270)	(5,270)	(8,500)	(10,200)	**(45,050)**
less Accrued Closure Costs	-	-	-	3,700	4,933	4,933	4,933	4,933	4,933	4,933	4,933	4,933	1,885	-	-	**45,050**
add Debt Funding	5,401	53,549	24,299	-	-	-	-	-	-	-	-	-	-	-	-	**83,250**
less Debt Repayments	(217)	(3,035)	(12,494)	(13,478)	(13,478)	(14,330)	(20,617)	(5,601)	-	-	-	-	-	-	-	**(83,250)**
less Interest & Other Payments	(87)	(2,505)	(4,907)	(4,469)	(3,553)	(2,568)	(1,560)	(184)	-	-	-	-	-	-	-	**(19,832)**
less Gold loan	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
less Royalty Purchase	-	-	(1,500)	-	-	-	-	-	-	-	-	-	-	-	-	**(1,500)**
less Capital Expenditures	(108,982)	(310,844)	(314,869)	(71,356)	(10,879)	(13,619)	(7,006)	(15,900)	(11,014)	(7,625)	(4,945)	(5,195)	(1,295)	-	-	**(883,528)**
add Salvage Value	-	-	-	-	-	-	-	-	-	-	-	-	26,687	33,197	-	**59,884**
less Changes in Working Cap.	-	-	(22,982)	(2,913)	167	32	(1,063)	2,015	495	(416)	48	13,536	11,082	-	-	**-**
Before Tax Cash Flow	**(103,884)**	**(262,835)**	**(332,453)**	**158,564**	**259,981**	**236,278**	**238,307**	**287,099**	**194,943**	**177,157**	**238,119**	**333,946**	**133,459**	**24,697**	**(10,200)**	**1,573,179**
Disc. Before Tax CF (5%)	**(102,628)**	**(247,292)**	**(297,898)**	**135,317**	**211,273**	**182,867**	**175,655**	**201,542**	**130,315**	**112,786**	**144,379**	**192,839**	**73,387**	**12,934**	**(5,087)**	**920,388**
Before-tax IRR	**24.1%**															
less Mining Duties	-	-	-	-	-	-	(11,016)	(12,475)	(8,395)	(7,722)	(10,257)	(13,781)	(4,111)	-	-	**(67,757)**
less Federal Income Taxes	-	-	-	-	-	-	(18,642)	(41,070)	(26,415)	(23,698)	(32,146)	(44,292)	(12,061)	-	-	**(198,323)**
less Provincial Income Taxes	-	-	-	-	-	-	(13,185)	(32,640)	(21,057)	(18,800)	(25,514)	(35,016)	(9,942)	-	-	**(156,154)**
less Provincial Capital Tax	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
After Tax Cash Flow	**(103,884)**	**(262,835)**	**(332,453)**	**158,564**	**259,981**	**236,278**	**195,464**	**200,914**	**139,076**	**126,938**	**170,203**	**240,857**	**107,345**	**24,697**	**(10,200)**	**1,150,945**
Disc. After Tax CF (5%)	**(102,628)**	**(247,292)**	**(297,898)**	**135,317**	**211,273**	**182,867**	**144,075**	**141,041**	**92,969**	**80,814**	**103,199**	**139,084**	**59,027**	**12,934**	**(5,087)**	**649,695**
After-tax IRR	**20.5%**															

Table 15.10: Canadian Malartic Base Case Cash Flows – CAPEX to Completion (k$)

Project Cash Flows	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	Total
Revenue	-	-	-	392,406	487,778	469,818	470,490	511,724	413,216	392,589	443,551	510,867	158,958	-	-	4,251,396
Total Cash Costs (C1)	-	-	-	(136,250)	(193,447)	(195,766)	(195,185)	(198,801)	(197,049)	(195,833)	(189,118)	(172,292)	(54,509)	-	-	(1,728,249)
Royalties	-	-	-	(5,378)	(6,607)	(7,290)	(6,752)	(6,153)	(5,435)	(6,287)	(6,146)	(7,700)	(2,194)	-	-	(59,942)
Depreciation	-	-	-	(66,853)	(84,454)	(83,233)	(84,494)	(95,252)	(79,262)	(77,295)	(89,327)	(106,834)	(34,510)	-	-	(801,513)
Accrued Closure Costs	-	-	-	(3,700)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(4,933)	(1,885)	-	-	(45,050)
EBIT	-	-	-	180,226	198,337	178,597	179,126	206,585	126,537	108,240	154,027	219,108	65,860	-	-	1,616,642
add Depreciation	-	-	-	66,853	84,454	83,233	84,494	95,252	79,262	77,295	89,327	106,834	34,510	-	-	801,513
add Closure Costs	-	-	-	-	-	-	-	-	(5,270)	(5,270)	(5,270)	(5,270)	(5,270)	(8,500)	(10,200)	(45,050)
less Accrued Closure Costs	-	-	-	3,700	4,933	4,933	4,933	4,933	4,933	4,933	4,933	4,933	1,885	-	-	45,050
add Debt Funding	5,401	53,549	24,299	-	-	-	-	-	-	-	-	-	-	-	-	83,250
less Debt Repayments	(217)	(3,035)	(12,494)	(13,478)	(13,478)	(14,330)	(20,617)	(5,601)	-	-	-	-	-	-	-	(83,250)
less Interest & Other Payments	(87)	(2,505)	(4,907)	(4,469)	(3,553)	(2,568)	(1,560)	(184)	-	-	-	-	-	-	-	(19,832)
less Gold loan	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
less Royalty Purchase	-	-	(1,500)	-	-	-	-	-	-	-	-	-	-	-	-	(1,500)
less Capital Expenditures	(26,967)	(310,844)	(314,869)	(71,356)	(10,879)	(13,619)	(7,006)	(15,900)	(11,014)	(7,625)	(4,945)	(5,195)	(1,295)	-	-	(801,513)
add Salvage Value	-	-	-	-	-	-	-	-	-	-	-	-	26,687	33,197	-	59,884
less Changes in Working Cap.	-	-	(22,982)	(2,913)	167	32	(1,063)	2,015	495	(416)	48	13,536	11,082	-	-	-
Before Tax Cash Flow	(21,869)	(262,835)	(332,453)	158,564	259,981	236,278	238,307	287,099	194,943	177,157	238,119	333,946	133,459	24,697	(10,200)	1,655,194
Disc. Before Tax CF (5%)	(21,605)	(247,292)	(297,898)	135,317	211,273	182,867	175,655	201,542	130,315	112,786	144,379	192,839	73,387	12,934	(5,087)	1,001,411
Before-tax IRR	28.8%															
less Mining Duties	-	-	-	-	-	-	(11,016)	(12,475)	(8,395)	(7,722)	(10,257)	(13,781)	(4,111)	-	-	(67,757)
less Federal Income Taxes	-	-	-	-	-	-	(18,642)	(41,070)	(26,415)	(23,698)	(32,146)	(44,292)	(12,061)	-	-	(198,323)
less Provincial Income Taxes	-	-	-	-	-	-	(13,185)	(32,640)	(21,057)	(18,800)	(25,514)	(35,016)	(9,942)	-	-	(156,154)
less Provincial Capital Tax	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
After Tax Cash Flow	(21,869)	(262,835)	(332,453)	158,564	259,981	236,278	195,464	200,914	139,076	126,938	170,203	240,857	107,345	24,697	(10,200)	1,232,960
Disc. After Tax CF (5%)	(21,605)	(247,292)	(297,898)	135,317	211,273	182,867	144,075	141,041	92,969	80,814	103,199	139,084	59,027	12,934	(5,087)	730,718
After-tax IRR	25.1%															

15.12 Sensitivities and Analyses

Sensitivities were performed on the following key market driven variables:

- Gold price
- Silver price
- Canadian dollar exchange rate
- Crude oil price.

The project driven parameters evaluated include:

- Direct cash costs
- Process recovery
- Initial capital expenditures.

Figure 15.7 presents the after-tax IRR sensitivity and Figure 15.9 the NPV 5% for the various parameters studied for the total project basis. Figure 15.8 and Figure 15.10 present results for the same variations but on a CAPEX to completion basis that excludes past expenditures. The values on a total project basis and a CAPEX to completion basis are presented in Table 15.11 through Table 15.17.

The internal rate of return on a CAPEX to completion basis is 4.6% higher than on a total project basis.

The results indicate that the project financial return is most sensitive to gold price and gold recovery (revenues) and less sensitive to initial CAPEX, direct cash cost and exchange rate. The price of silver and crude oil have almost no effect on the project financial return.

Figure 15.7: After-Tax IRR Sensitivity – Total Project Basis

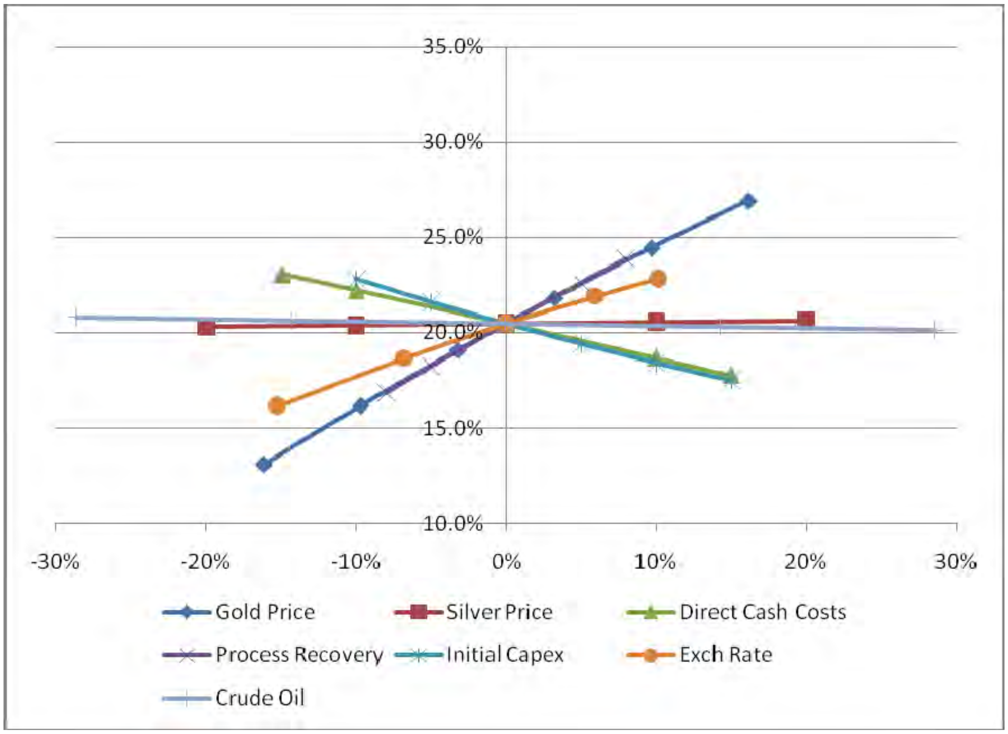


Figure 15.8: After-Tax IRR Sensitivity – CAPEX to Completion Basis

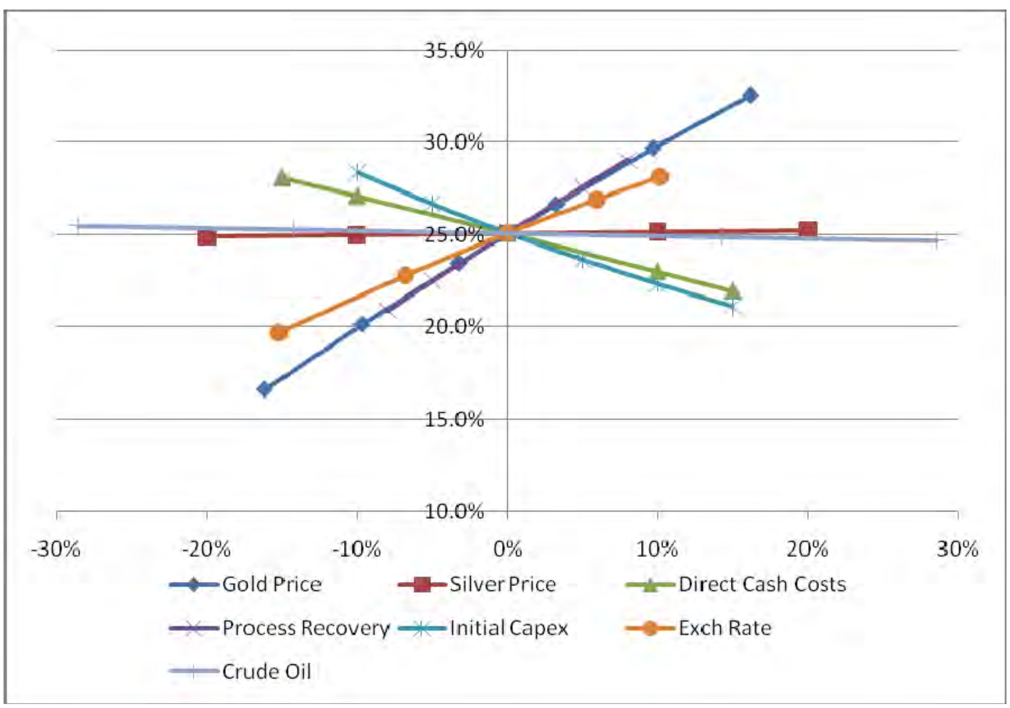


Figure 15.9: After-tax NPV 5% Sensitivity – Total Project Basis

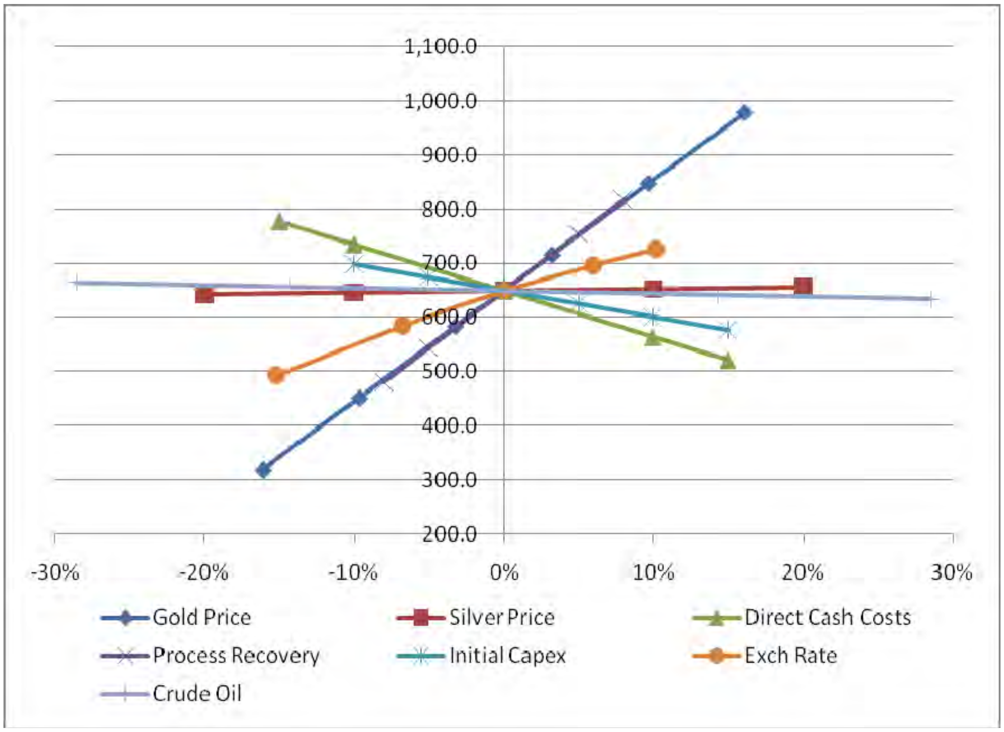


Figure 15.10: After-tax NPV 5% Sensitivity – CAPEX to Completion Basis

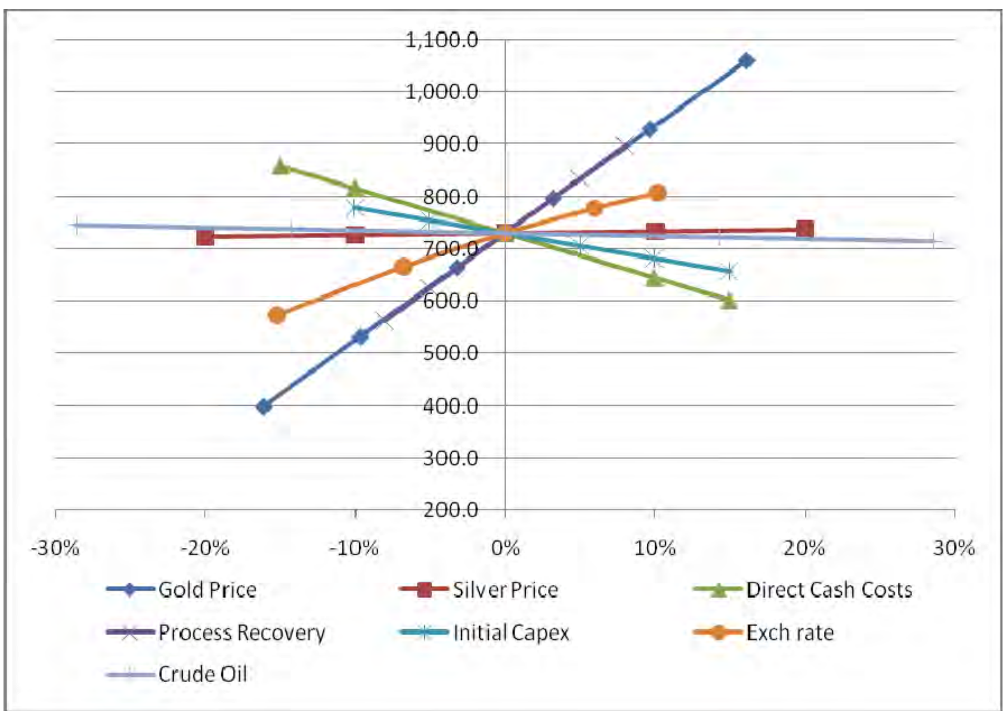


Table 15.11: Gold Price Sensitivity

	Total Project Basis							CAPEX to Completion Basis						
% Var.	Gold Price ($/oz)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)	Gold Price ($/oz)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
-16%	650	13.1%	318.4	686.2	15.4%	455.3	908.7	650	16.6%	399.4	768.2	19.0%	536.3	990.7
-10%	700	16.2%	451.4	872.1	19.0%	641.3	1,174.5	700	20.1%	532.4	954.1	23.1%	722.4	1,256.5
-3%	750	19.1%	583.7	1,058.0	22.4%	827.4	1,440.3	750	23.5%	664.8	1,140.0	27.0%	908.4	1,522.3
0%	775	20.5%	649.7	1,150.9	24.1%	920.4	1,573.2	775	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
3%	800	21.8%	715.6	1,243.9	25.7%	1,013.4	1,706.1	800	26.7%	796.6	1,325.9	30.7%	1,094.4	1,788.1
10%	850	24.5%	847.3	1,429.8	28.7%	1,199.4	1,971.9	850	29.7%	928.4	1,511.8	34.2%	1,280.5	2,053.9
16%	900	26.9%	978.4	1,615.7	31.6%	1,385.5	2,237.7	900	32.6%	1,059.4	1,697.7	37.5%	1,466.5	2,319.7

Table 15.12: Silver Price Sensitivity

	Total Project Basis							CAPEX to Completion Basis						
% Var.	Silver Price ($/oz)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)	Silver Price ($/oz)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
-20%	8	20.3%	642.9	1,141.4	23.9%	910.9	1,559.6	8	24.9%	724.0	1,223.4	28.6%	991.9	1,641.6
-10%	9	20.4%	646.3	1,146.2	24.0%	915.6	1,566.4	9	25.0%	727.3	1,228.2	28.7%	996.6	1,648.4
0%	10	20.5%	649.7	1,150.9	24.1%	920.4	1,573.2	10	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
10%	11	20.5%	653.1	1,155.7	24.2%	925.1	1,580.0	11	25.2%	734.1	1,237.7	28.9%	1,006.2	1,662.0
20%	12	20.6%	656.5	1,160.5	24.2%	929.9	1,586.8	12	25.2%	737.5	1,242.5	29.0%	1,010.9	1,668.8

Table 15.13: Direct Cash Cost Sensitivity

% Var.	Total Project Basis: Cash Costs ($/oz)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)	CAPEX to Completion Basis: Cash Costs ($/oz)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
-15%	272	23.1%	778.3	1,332.5	27.1%	1,101.7	1,832.4	272	28.1%	859.3	1,414.5	32.3%	1,182.8	1,914.4
-10%	288	22.2%	735.4	1,272.0	26.1%	1,041.3	1,746.0	288	27.1%	816.4	1,354.0	31.2%	1,122.3	1,828.0
0%	320	20.5%	649.7	1,150.9	24.1%	920.4	1,573.2	320	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
10%	352	18.7%	563.9	1,029.9	22.0%	799.5	1,400.4	352	23.0%	644.9	1,111.9	26.4%	880.5	1,482.4
15%	368	17.8%	521.0	969.4	20.9%	739.0	1,313.9	368	22.0%	602.0	1,051.4	25.2%	820.1	1,396.0

Table 15.14: Process Recovery Sensitivity

% Var.	Total Project Basis: Recovery	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)	CAPEX to Completion Basis: Recovery	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
-8%	0.790	16.8%	481.0	913.7	19.8%	683.1	1,234.3	0.790	20.9%	562.0	995.7	24.0%	764.1	1,316.4
-5%	0.816	18.2%	544.4	1,002.6	21.5%	772.1	1,361.4	0.816	22.5%	625.4	1,084.7	25.8%	853.1	1,443.4
0%	0.859	20.5%	649.7	1,150.9	24.1%	920.4	1,573.2	0.859	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
5%	0.902	22.6%	754.9	1,299.3	26.6%	1,068.7	1,785.0	0.902	27.6%	835.9	1,381.3	31.7%	1,149.7	1,867.0
8%	0.928	23.9%	818.0	1,388.2	28.0%	1,157.7	1,912.0	0.928	29.1%	899.1	1,470.3	33.4%	1,238.7	1,994.0

Table 15.15: Initial Capital Expenditures Sensitivity

	Total Project Basis							CAPEX to Completion Basis						
% Var.	Initial CAPEX ($ M))	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)	CAPEX to Completion ($ M)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
-10%	718.19	22.8%	698.7	1,200.4	26.7%	985.1	1,643.9	578.61	28.4%	779.8	1,282.5	32.5%	1,066.2	1,725.9
-5%	753.53	21.6%	674.2	1,175.7	25.4%	952.8	1,608.5	610.76	26.7%	755.2	1,257.7	30.6%	1,033.8	1,690.5
0%	788.87	20.5%	649.7	1,150.9	24.1%	920.4	1,573.2	642.90	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
5%	824.22	19.4%	625.1	1,126.2	22.9%	888.0	1,537.8	675.05	23.6%	706.2	1,208.2	27.2%	969.0	1,619.9
10%	859.56	18.4%	600.6	1,101.4	21.7%	855.6	1,502.5	707.19	22.3%	681.6	1,183.5	25.7%	936.7	1,584.5
15%	894.90	17.5%	576.0	1,076.7	20.7%	823.3	1,467.2	739.34	21.1%	657.0	1,158.7	24.4%	904.3	1,549.2

Table 15.16: Exchange Rate Sensitivity

	Total Project Basis							CAPEX to Completion Basis						
% Var.	Exchange Rate (C$/US$)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)	Exchange Rate (C$/US$)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
-15%	1.00	16.1%	492.6	952.3	19.0%	702.2	1,286.1	1.00	19.7%	573.6	1,034.3	22.6%	783.3	1,368.1
-7%	1.10	18.7%	585.5	1,069.5	21.9%	831.0	1,455.4	1.10	22.8%	666.6	1,151.5	26.2%	912.0	1,537.4
0%	1.18	20.5%	649.7	1,150.9	24.1%	920.4	1,573.2	1.18	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
6%	1.25	21.9%	696.8	1,210.5	25.7%	986.0	1,659.3	1.25	26.9%	777.8	1,292.5	30.9%	1,067.1	1,741.3
10%	1.30	22.8%	727.0	1,248.6	26.8%	1,028.1	1,714.5	1.30	28.2%	808.0	1,330.7	32.3%	1,109.1	1,796.5

Table 15.17: Crude Oil Sensitivity

	Total Project Basis							CAPEX to Completion Basis						
% Var.	Crude Oil ($/bbl)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)	Crude Oil ($/bbl)	After-tax IRR	After-tax NPV 5% ($ M)	After-tax NPV 0% ($ M)	Pre-tax IRR	Pre-tax NPV 5% ($ M)	Pre-tax NPV 0% ($ M)
-29%	50.00	20.8%	665.4	1,172.9	24.5%	942.5	1,604.6	50.00	25.5%	746.4	1,254.9	29.3%	1,023.5	1,686.6
-14%	60.00	20.6%	657.5	1,161.9	24.3%	931.4	1,588.9	60.00	25.3%	738.6	1,243.9	29.1%	1,012.5	1,670.9
0%	70.00	20.5%	649.7	1,150.9	24.1%	920.4	1,573.2	70.00	25.1%	730.7	1,233.0	28.8%	1,001.4	1,655.2
14%	80.00	20.3%	641.8	1,140.0	23.9%	909.3	1,557.5	80.00	24.9%	722.9	1,222.0	28.6%	990.4	1,639.5
29%	90.00	20.1%	634.0	1,129.0	23.7%	898.3	1,541.8	90.00	24.7%	715.0	1,211.0	28.4%	979.3	1,623.8

16.0 CONCLUSIONS AND RECOMMENDATIONS

The pit design contains a proven and probable reserve of 183.3 Mt of ore at a head grade of 1.07 g/t Au for an in-situ gold content of 6.28 Moz. Given the important quantities of low grade mineralization and a favorable waste to ore strip ratio (1.78:1), it was determined that a high production rate of 55,000 tpd would result in a financially viable and robust project.

The results of the Feasibility Study indicate that the Canadian Malartic project warrants proceeding to implementation stage of the project. It does not present significant technical difficulties and economic indicators resulting from this study are very positive. Capital expenditures (CAPEX) are estimated at $788.9 M or $146 per recoverable ounce placing the Canadian Malartic project within current industry norms as one of the best undeveloped gold projects in the world.

Based on the CAPEX to completion scenario ($642.9 M), the internal pre-tax rate of rate (IRR) is estimated at 28.8% and the pre-tax NPV (discount 5%) is estimated at $1,001 M. Results of the Feasibility Study demonstrate that the Canadian Malartic project is robust in the current market.

BBA concludes that Osisko should advance the project to the final stage of implementation including the following recommendations:

- Secure the project financing
- Complete the relocation program by the end of 2009
- Secure required permits and authorizations from Government and regulatory agencies
- Continue the detail engineering program
- Move to the construction phase following environmental permitting.

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18.0 QUALIFICATION CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

I, David Runnels, Eng., do hereby certify that:

1. I am currently employed as Project Manager – Mining and Metallurgy in the consulting firm:
   BBA Inc.
   630, boul. René-Lévesque Ouest, Suite 2500
   Montreal, Quebec
   Canada, H3B 1S6

2. I graduated from Queen's University, Kingston, Ontario, Canada with a B.Sc. in Metallurgy in 1971.

3. I am in good standing as a member of the Order of Engineers of Québec (#22450) and I am a member of the Canadian Institute of Mining.

4. I have practiced my profession continuously since my graduation from university.

5. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that as a result of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I have worked in a mining operation for a period of 6 years and in the engineering consulting field for 31 years.

6. I have visited the Canadian Malartic site in October, 2007.

7. I am responsible for the coordination of the complete Technical Report and for the preparation of Sections 1, 2.0 to 2.3.4, 6, 8, 9, 12 , 13, 14, 15, 16, 17, and 18 of the Technical Report entitled *"Feasibility Study – Canadian Malartic Project"*, dated December 2008.

8. As of the date of this certificate I am not aware of any changes in fact or circumstances with respect to the subject matter of this report which materially affects the content of the report or the conclusions reached.

9. I am independent of the issuer in accordance with Section 1.4 of National Instrument 43-101.

10. I have read national Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Prepared in Montréal, Québec, December 4th, 2008

Signed
David Runnels, Eng.

G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON (QP)

I, Louis-Pierre Gignac, do hereby certify that:

1. I am currently employed as a Consulting Mining Engineer by:

   G Mining Services Inc.
   8250 Racine
   Brossard, Québec
   Canada J4X 1T8

2. I graduated from McGill University, Canada with a B.Sc. In Mining Engineering in 1999, and from École Polytechnique de Montréal, Canada with a M.Sc.A. in Industrial Engineering in 2002.

3. I am in good standing as a member of the Order of Engineers of Québec (#132995) and I am a member of the Canadian Institute of Mining.

4. I have worked in the mining industry continuously since my graduation from university.

5. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that as a result of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I have been involved in mining operations, engineering and financial evaluation for 8 years, including pit optimization, surface mine design, mineral reserve calculations and mine scheduling. This has included work at Rosebel Gold Mines NV, the Camp Caiman Project and work on other pre-feasibility and feasibility studies.

6. I visited the Canadian Malartic Project on November 7, 2007.

7. I am responsible for the preparation of Sections 5.0 to 5.8 and 15.0 for the report titled "43-101 Technical Report for the Feasibility Study - Canadian Malartic" dated December 2008.

8. I am independent of the issuer in accordance with Section 1.4 of National Instrument 43-101.

9. I have read national Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I also certify that as of the date of this certificate, to the best of my knowledge, information and belief that I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.

11. I consent to the filing of the Technical Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 3rd day of December 2008.

Louis Pierre Gignac

Signed

Louis-Pierre Gignac, Eng.

micon INTERNATIONAL LIMITED | mineral industry consultants

CERTIFICATE

B. TERRENCE HENNESSEY

As the author of a portion of this report on certain mineral properties of Osisko Mining Corporation in the Malartic area of northwestern Québec, Canada, I, B. Terrence Hennessey, P.Geo., 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
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   tel. (416) 362-5135
   fax (416) 362-5763
   e-mail thennessey@micon-international.com;

2. I hold the following academic qualifications:

B.Sc. (Geology)	McMaster University	1978

3. I am a registered Professional Geoscientist with the Association of Professional Geoscientists of Ontario (membership # 0038) with temporary registration in the Province of Quebec for this assignment; as well, I am a member in good standing of several other technical associations and societies, including:

   The Australasian Institute of Mining and Metallurgy (Member)
   The Canadian Institute of Mining, Metallurgy and Petroleum (Member)

4. I have worked as a geologist in the minerals industry for over 29 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 7 years as an exploration geologist looking for iron ore, gold, base metal and tin deposits, more than 11 years as a mine geologist in both open pit and underground mines and 11 years as a consulting geologist working in precious, ferrous and base metals as well as industrial minerals;

6. I visited the Canadian Malartic project site during the period July 15 to 16, 2008 to review the results of exploration at site;

7. I am responsible for the preparation of Sections 2.4, 3.0 to 3.5.5 and 3.6 to 3.6.7 (portions) of the Technical Report titled "Canadian Malartic Project Feasibility Study" and dated December, 2008;

8. I am independent of the parties involved in the transaction for which this report is required, as defined in Section 1.4 of NI 43-101;

9. I have had no prior involvement with the mineral properties in question;

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 Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading;

Dated this 3rd day of December, 2008

"B. Terrence Hennessey" {signed and sealed}

B. Terrence Hennessey, P.Geo.

CERTIFICATE

ELZÉAR BELZILE

As the author of a portion of this report on certain mineral properties of Osisko Mining Corporation in the Malartic area of north-western Québec, Canada, I, Elzéar Belzile, Professionnal Engineer of the Province of Quebec, do hereby certify that:

1 I reside at 399, Montée du Sourire, Rouyn-Noranda, Quebec, J9X 5L2

2 I am an independent mining consultant (Belzile Solutions Inc.) and carried out this assignment for :

Micon International Limited
Suite 900, 390 Bay Street
Toronto, Ontario
M5H 2Y2

3 I hold the following academic qualifications:

B. SC. (Génie géologique) Laval University (Qc) 1983

4 I am a registered Professionnal Engineer with Ordre des Ingénieurs du Québec (membership # 43790): as well, I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum

5 I have worked as an engineer since my graduation in exploration and mining geology. Over the last 25 years, I have completed numerous resource estimations for precious and base metal deposits.

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 3 years as an exploration geologist looking for gold and base metal deposits, more than 16 years as a mine geologist in both open pit and underground mines and 6 years as Manager, Mining Geology for Cambior (2002-06) Inc and Iamgold Corporation (2006-08). I am independent consultant since February 2008.

7 I visited the Canadian Malartic Project in February 14, 2008 and in July 15 and 16, 2008

8 I am responsible for the preparation of sections 3.6 (portions) and 4.0 of the report titled “ Canadian Malartic Project, Feasibility study” and dated December, 2008;

9 I am independent of the parties involved in the transaction for which this report is required, as defined in Section 1.4 of NI 43-101;

10 I am co-author of a technical report titled "An updated mineral resource estimate for the Canadian Malartic Project, Malartic, Quebec" dated October 23, 2008

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, sections 3.6 and 4.0 of the This Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 3rd day of December, 2008 in Rouyn-Noranda,

"*Elzéar Belzile" signed and sealed*

Elzéar Belzile, Ing (OIQ #43790)

CERTIFICATE OF QUALIFIED PERSON

I, Michel R. Julien, ing., Ph.D., do hereby certify that :

1. I am currently employed as a Principal, Regional Director, in the consulting firm:
   Golder Associés Limitée (GAL)
   9200 Boulevard de L'Acadie
   Montréal, Québec
   Canada H4N 2T2

2. I graduated from École Polytechnique with a B.Sc. in mining Engineering in 1986, a M.Sc. in Geo-Engineering from University of Minnesota in 1988 and a Ph.D. from École Polyetchnique in Mineral Engineering from Civil Geological and Mining Department in 1999.

3. I am in good standing as a member of the Order of Engineers of Québec (42119) and of the Association of Engineers and Geoscientists of Newfoundland and Labrador (03033). I am also member of several technical associations and have been author and co-author of several publications in the field of mine waste management and applied mechanics.

4. I have been working for Golder Associés since 1990. I left for about 4 months GAL to accept a Professor position at École Polytechnique de Montréal in 2001.

5. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that as a result of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 7.0 of the report entitled "43-101 Technical Report for the Feasibility Study – Canadian Malartic" (the Report) based on my knowledge and extensive experience on environmental and geotechnical issues related to mining operations worldwide. I have visited the property and have been involved in mine waste management projects in Malartic area for over 18 years and therefore I very well understand the context, history and challenges associated with the development of this site at Malartic.

7. As of the date of this certificate I am not aware of any changes or circumstances with respect to the subject matter of the Report which materially affects its content or the conclusion reached.

8. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

9. I have read national Instrument 43-101 and Form 43-101F1, and the Report has been prepared in compliance with that instrument and form.

10. I consent to the filing of the Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Prepared in Montréal, Québec, December 3, 2008

Michel R. Julien, ing., Ph.D.

GENIVAR

CERTIFICATE OF QUALIFIED PERSON (QP)

I, André-Martin Bouchard, Professional Civil / Environmental Engineer, do hereby certify that:

1. I am currently employed as Environmental Director by:

   GENIVAR sec
   1600, René Lévesque blvd., 16th floor
   Montréal, Québec
   Canada
   H3H 1P9

2. I graduated from École Polytechnique de Montréal, Canada with a Bachelor in Civil Engineering with a major specialization in Environment and Water Resources in 1993.

3. I am in good standing as a member of the Order of Engineers of Québec (Nº 112394).

4. I have worked as an Environmental Engineer continuously since my graduation from university in 1993.

5. I have read the definition of «qualified person» set out in the National Instrument 43-101 (NI 43-101) and certify that as a result of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a «qualified person» for the purposes of NI 43-101. I have been involved in environmental consultation, engineering and permitting processes for 16 years, including mining projects.

6. The team of professionals that I supervise visited the Canadian Malartic Project site on several occasions during the course of our mandate with Osisko, but I have not visited the site myself.

7. I was responsible for the preparation of Sections 10.0 and 11.0 for the report titled « 43-101 Technical Report for the Feasibility Study - Canadian Malartic » for the Osisko Canadian Malartic Project, dated December 2008.

8. I am independent of the issuer in accordance with section 1.4 of National Instrument 43-101.

9. I have read national Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I also certify that as of the date of this certificate, to the best of my knowledge, information and belief, that I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.

11. I consent to the filing of the Technical Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 3rd of December 2008

Signed

André-Martin Bouchard, P. Eng.