IAMGOLD CORPORATION

TECHNICAL REPORT ON EXPANSION OPTIONS AT THE NIOBEC MINE, QUÉBEC, CANADA

NI 43-101 Report

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1 SUMMARY

EXECUTIVE SUMMARY

INTRODUCTION

Roscoe Postle Associates Inc. (RPA) was retained by IAMGOLD Corporation (IAMGOLD), to prepare an independent Technical Report on the Niobec Mine (the Project), near Ville de Saguenay (Chicoutimi), Québec. The purpose of this report is to prepare an updated Mineral Resource estimate and a Preliminary Economic Assessment (PEA) on the viability of an open pit (OP) mining option and production expansion at the Project. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects. RPA visited the property on March 4, 2011 and again from March 21 to 22, 2011.

CONCLUSIONS

It is RPA's opinion that mining the Niobec deposit by open pit mining method may have considerable potential over the current underground method to extend mine life and better maximize the extraction of the resource. The Project base case scenario consists of technical and cost assumptions outlined in this report.

The Project will benefit from the expertise developed in the existing operation, including the processing and converting of niobium ore into ferroniobium over the years and the local and regional infrastructure currently used in the niobium mining/milling/converting activities at the mine site.

The most important risk elements for the Project are the niobium price and, at the present time, the location of waste rock dumps in areas with underlying soils capable of supporting the large tonnage of waste rock. Fluctuations in the niobium price constitute an uncontrollable parameter for which no mitigation measures are proposed. Other elements to which the Project is economically sensitive are controllable and will be addressed in further steps. Waste rock dump issues should be investigated as the Project progresses.

At the PEA level, the open pit scenario is technically feasible and economically viable and should be advanced to the pre-feasibility study level. The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the reserves development, production, and economic forecasts on which this preliminary economic assessment is based will be realized.

Specific conclusions are as follows:

- The combined Measured and Indicated Resources are 458.11 million tonnes grading 0.42% Nb_2O_5 containing 1,927 million kg Nb_2O_5. In addition, there is an Inferred Resource of 336.45 million tonnes grading 0.37% Nb_2O_5 containing 1,240 million kg Nb_2O_5. All these Mineral Resources are constrained by a Whittle open pit shell (6T) at a cut-off grade of 0.20% Nb_2O_5.

- A second economic optimization, which was time constrained, was performed on the 6T pit shell in order to maximize the net present value (NPV) at 8%. The maximum NPV occurs at a cut-off grade of 0.33% Nb_2O_5. This resulted in a total of 370.2 million tonnes of niobium bearing material grading 0.46% Nb_2O_5 within a smaller pit shell, 4T (564 m deep). The proportion of Inferred Resources in the above tonnage that may be potentially mineable via open pit is approximately 20%.

- Ore production will be expanded from the current level of 2.2 million tonnes per year from underground to 10 million tonnes per year from OP. The life of mine (LOM) of the OP scenario will be 42 years, consisting of four years of pre-production and 38 years of production. During the pre-production period, the existing underground mine will contribute, at the current mining rate, an additional 8.8 Mt grading 0.60% Nb_2O_5. At full production, ore will be mined at a rate of 27,400 tpd requiring the excavation of 109,000 tpd waste on average over the Project total duration.

- As part of the analysis of options, RPA and IAMGOLD reviewed an underground block caving scenario as an alternative to an open pit. At this level of study and assumptions, the underground scenario is a viable but slightly less attractive option than the open pit scenario.

- Project components and costs were developed to a ± 40% level of accuracy. Operating costs were estimated in 2011 Canadian dollars while capital costs were mostly estimated in 2011 US dollars.

- Capital expenditures, including 15% contingency and sustaining/ongoing investments, are estimated to be US$1.4 billion (excluding the recovery of

US$66.4 million for working capital and warehouse inventory at the end of the LOM). This amount excludes taxes, duties, and inflation. Initial open pit related capital expenditures will amount to US$830 million; the remaining capital outlay is considered as ongoing investment that will occur during the production period or the operation of the existing underground mine (US$91 million) during the pre-production period.

- The total mine site and ore processing operating cost, for the period ranging from 2011 to 2052 inclusively, is estimated to be C$26.67/t milled. This includes mining, mineral processing and converting, maintenance, administration and human resources costs.

- At peak, manpower is estimated to be 743 individuals for the various administrative units, namely administration, mine, mill, converter, and surface and electrical.

- IAMGOLD and RPA developed the open pit mine plan based on a 3D block model built by RPA. RPA is of the opinion the resource model is conservative and has good potential to expand. The pit shell comprises many blocks that are outside the four main zones, in the mineralized carbonatite where limited to no information is available. A grade of zero has been assigned to those blocks.

- There are no current Mineral Reserves estimated for the open pit mining option at Niobec. Mineral Reserves will be assessed at the pre-feasibility stage of study.

- Open pit mining will be carried out along 564 vertical metres. For the pit size, production requirements, and recommended equipment fleet, RPA considers mining of 12 m benches and 33 m wide ramps, including ditches and safety berms, to be appropriate.

- Ore will be delivered to a gyratory crusher prior to a covered stockpile and then conveyed into the processing plant. The flowsheet is similar to the current plant, comprising crushing, grinding, desliming, flotation (carbonate, pyrochlore, and sulphide), dewatering, leaching, filtration, drying, and packaging. Concentrate is transferred into the converter where the ferroniobium (FeNb final product) is produced. The mill and the converter will be new installations to be constructed to handle the increased production and because the current buildings lie within the open pit footprint. The existing tailings impoundment facility will be expanded to accommodate the additional tonnage.

- Recovery rates are estimated to be 49.6% Nb_2O_5 at the processing plant (head grade dependent) and 97% at the converting stage. FeNb production will total 597 M kg over the LOM.

- Environment mitigation measures will be implemented, with particular emphasis on the open pit and waste rock dumps. Proper water control and treatment will be implemented in order to minimize volumes to release final effluent respecting provincial regulations.

- The conceptual closure plan aims at returning the site in a state compatible with the environment. Particularly, waste rock dumps and tailings pond will be subject to progressive rehabilitation and revegetation measures while the existing pit will naturally flood once operations cease.

- An economic evaluation was carried out by means of pre-tax and after-tax cash flow analysis expressed in 2011 constant US dollars. An exchange rate of US$1.00/C$1.05 was assumed. The average metal price forecast is US$45/kg niobium and was assumed to be constant for the duration of the Project, except for the first year of the LOM (2011) at US$40/kg Nb.

- Sunk costs (exploration and delineation drilling, metallurgical test work, feasibility study costs, Environmental Impact Assessment, etc.) are not included in the Project capital cost or financial evaluation.

- Total revenues generated by the Project are estimated to be US$27.3 billion, and operating cash flow before capital costs total US$10.1 billion. Using a discount rate of 8%, the Project has NPVs of US$3.3 billion pre-tax and US$2.0 billion after-tax.

- The after-tax internal rate of return (IRR) of the Project is 47.5%. The after-tax cumulative cash flow turns positive during the third year of the Project production phase and amounts to US$9.3 billion over the LOM. The capital payback period is 2.2 years.

- Sensitivity calculations were performed on the Project cash flow by applying a ± 20% variance on head grade, mill recovery, niobium price, operating costs, capital expenditures, and C$/US$ exchange rate. The sensitivity analysis shows that the Project is most sensitive to head grade, mill recovery, and niobium price, moderately sensitive to exchange rate and operating costs, and less sensitive to capital costs.

RECOMMENDATIONS

RPA makes the following recommendations:

- Initiate the preparation of pre-feasibility studies for both the block caving and the open pit options in order to compare scenarios at a higher level of definition and accuracy, given both returned quite similar economic results.

- Confirm the waste rock dump location.

- Initiate geotechnical assessments of underlying/surrounding overburden and soils at waste rock dumps, open pit, and tailings pond locations.

- Initiate geomechanical and rock mechanics assessments for open pit wall slope angle and stability, and effects on advancing pit bottom through existing underground openings.

- Initiate an environmental program that will result in the preparation of a document in support of permitting. The program should be comprehensive and include baseline work, acid base accounting, waste rock dump stability, tailings pond stability, open pit wall stability, radon and radiation emission, dust emission, water balance and hydrology, salt contamination of water being exposed to rock in open pit and waste rock dumps and socio-economic studies.

- Initiate negotiations with individuals and governments concerning land purchases.

- Carry out further metallurgical testwork, using representative samples. The program would consist of completing test work to a typical standard for pre-feasibility study, including:
    - Crushing and milling (SAG work index) to properly design and select equipment.
    - Extensive metallurgical testwork to validate the niobium recovery and concentrate quality grade, given the lower head grade anticipated and the sensitivity of final product to impurities.
    - Niobium recovery improvement in investigating modifications to reagent scheme and process flowsheet, in considering previous testing on niobium recovery from the plant tailings and by benchmarking mill results with the laboratory results.
    - Comparative testwork between conventional grinding and classification presently used and new grinding circuit using SAG-ball mill.

- Carry out specific hydrological/hydrogeological studies to refine dewatering needs in the underground mine during pre-production period and in the open pit mine over the LOM.

- Carry out condemnation drilling in advance of mining in areas where waste rock dumps and the tailings storage facility are expected to be located for the operations.

- For the resource database:
    - Remove duplicate sample numbers from the drill database and investigate zero values for Nb_2O_5.
    - Investigate the large number of assays in the database without sample numbers, if possible, by cross referencing drill logs and assay certificates.
    - Study the use of a top cut for Nb_2O_5 assay results and consider adoption for future resource estimations.
    - Initiate the insertion of blanks and independent certified reference materials (CRM) into the sample stream so that they comprise approximately 15% to 20% of the total determinations. RPA also recommends external laboratory check on sample rejects in addition to sample pulps.
    - Investigate the cause of the mild bias observed in pulp duplicates assayed at a second, independent, laboratory.
    - Study the use of equal length composites for resource estimation rather than raw samples.

ECONOMIC ANALYSIS

In this section, the open pit is presented as the base case with comparison to the block caving scenario.

Pre-tax and after-tax Cash Flow Projections have been generated from the LOM production schedule and capital and operating cost estimates, and are summarized in Table 1-1. A summary of the key criteria is provided below.

ECONOMIC CRITERIA

REVENUE

- 10 million tonnes milled per year.
- Nb_2O_5 mill recovery is between 40% and 60%, for a LOM average of 49.6%.
- FeNb converter recovery is 97% (with 69.9% Nb in FeNb).
- Marketing and freight totals US$1.40/kg Nb.
- Exchange rate at US$1.00 = C$1.05 from 2014 on, US$1.00/C$ before 2014.
- Niobium price is US$40.00/kg in 2011 and US$45.00/kg thereafter.
- LOM average unit net value is US$43.58/kg.

COSTS

- Capital cost totals US$1.4 billion, excluding the recovery of US$66.4 million for working capital and warehouse inventory at the end of the LOM.
- Average operating cost over the mine life is US$25.44 per tonne milled.

OTHER

- Pre-production period spans over four years (2011 to 2014).
- During pre-production period, current underground mining continues at 2.2 Mtpa.
- Open pit production period is 38 years, up to 2052.
- LOM stripping ratio is 3.91.

The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the reserves development, production, and economic forecasts on which this preliminary economic assessment is based will be realized.

TABLE 1-1 PRE- AND AFTER-TAX CASH FLOW SUMMARY
IAMGOLD Corp. – Niobec Mine

	Units	Total	2011 to 2014	2015 to 2052
Mine Production				
Waste Mined	Mt	1,481.67	12.00	1,469.67
Ore Mined	Mt	379.03	8.80	370.23
Total Mined	**Mt**	**1,860.70**	**20.80**	**1,839.90**
Waste to Ore ratio		3.91		
Mill Feed				
Ore milled	Mt	379.03	8.80	370.23
Milled ore grade	$\%Nb_2O_5$	0.463	0.603	0.460
Mill recovery	%	49.6	58.3	49.4
Yield	kg Nb_2O_5/t	2.32	3.51	2.30
Concentrate production	M kg Nb_2O_5	880.61	30.93	849.67
Converter recovery	%	97.0%	97.0	97.0
Niobium in FeNb	%	69.9	69.9	69.9
Niobium production	M kg	597.08	20.97	576.10
Revenue				
Niobium production	M kg	597.08	20.97	576.10
Niobium Market Price	US$/kg		43.90	45.00
Gross Revenue	US$ (M)	26,845.45	920.75	25,924.70
Exchange Rate	C$/US$		1.01	1.05
Gross Revenue	C$ (M)	28,153.96	933.02	27,220.94
Marketing and Freight	C$ (M)	875.63	31.62	844.01
Net Revenue	C$ (M)	27,278.33	901.40	26,376.92
Operating Costs				
Mining	C$ (M)	3,483.81	171.99	3,311.82
UG Material Handling Into OP	C$ (M)	7.60	0.00	7.60
Stockpile Reclaiming	C$ (M)	0.00	0.00	0.00
Waste Rock Dump & OP Water Management	C$ (M)	38.00	0.00	38.00
Processing	C$ (M)	3,676.25	118.39	3,557.86
Tailings pond	C$ (M)	0.00	0.00	0.00
General and Administration	C$ (M)	510.36	35.36	475.00
Converter	C$ (M)	2,388.67	84.25	2,304.42
Slag Hauling and Disposal	C$ (M)	4.18	0.00	4.18
Total Operating Cost	**C$ (M)**	**10,108.86**	**409.99**	**9,698.88**
Operating Cost Per Tonne Milled	(C$/t milled)	26.67	46.59	26.20
Operating Margin	**C$ (M)**	**17,169.46**	**491.42**	**16,678.05**

	Units	Total	2011 to 2014	2015 to 2052
Operating Margin	**US$ (M)**	**16,368.33**	**484.47**	**15,883.86**
Capital Costs				
Underground mine	US$ (M)	91.32	91.32	0.00
Open pit mine	US$ (M)	506.08	164.19	341.89
Mill and Crusher	US$ (M)	246.47	197.87	48.60
Converter	US$ (M)	52.50	47.10	5.40
Buildings and Services	US$ (M)	27.55	27.55	0.00
Power distribution	US$ (M)	10.36	10.36	0.00
Tailings and Water management	US$ (M)	97.57	67.57	30.00
Waste rock dump & OP water management	US$ (M)	30.31	30.31	0.00
General and Infrastructure	US$ (M)	21.73	21.73	0.00
Working capital	US$ (M)	0.00	46.92	- 46.92
Warehouse inventory	US$ (M)	0.00	19.43	- 19.43
Owner's cost	US$ (M)	10.30	10.30	0.00
Open pit vs Underground mine	US$ (M)	28.22	23.68	4.54
Progressive rehab. & Mine closure	US$ (M)	47.50	0.00	47.50
Eng. and Const. management	US$ (M)	57.83	56.13	1.70
Contingency	US$ (M)	104.63	104.63	0.00
Total capital costs	**US$ (M)**	**1,332.35**	**919.07**	**413.28**
Pre-Tax Cash Flow				
Cash flow	US$ (M)	15,035.98	-434.60	15,470.58
After-Tax Cash Flow				
Tax Rate (38.6%)				
Tax Payable	US$ (M)	5,778.03	125.14	5,652.89
Cash flow	US$ (M)	9,257.94	-559.74	9,817.69
Project Economics				
Pre-Tax NPV (5%)	US$ (M)	5,364.42		
Pre-Tax NPV (8%)	US$ (M)	3,276.05		
Pre-Tax NPV (10%)	US$ (M)	2,449.95		
Pre-Tax Internal rate of return	%	NA		
Pre-Tax Payback period (years)	Years	1.6		
After-Tax NPV (5%)	US$ (M)	3,251.49		
After-Tax NPV (8%)	US$ (M)	1,952.11		
After-Tax NPV (10%)	US$ (M)	1,437.91		
After-Tax Internal rate of return	%	47.5		
After-Tax Payback period	Years	2.2		

CASH FLOW ANALYSIS

Considering the Project on a stand-alone basis, the undiscounted pre-tax cash flow totals US$15.0 billion over the mine life, and simple payback occurs near the mid-point of 2016 (approximately 19 months from start of OP production). On an after-tax basis, the undiscounted cash flow is US$9.3 billion and the payback occurs at the beginning of 2017 (approximately 26 months from start of open pit production).

The Total Cash Cost is US$16.15 per kg of niobium. The mine life capital unit cost is US$2.35 per kg, for a Total Production Cost of US$18.50 per kg of niobium. Average niobium production during underground and open pit operations is 14.2 million kg per year.

The pre-tax NPV at an 8% discount rate is US$3.3 billion and the after-tax NPV is US$2.0 billion, and the after-tax IRR is 47.5%.

Table 1-2 is a comparison of the economic impact of the 10 Mtpa open pit scenario and the 10 Mtpa block caving scenario to the current LOM plan.

TABLE 1-2 BLOCK CAVING VERSUS OPEN PIT
IAMGOLD Corp. – Niobec Mine

Mining Method	Block Caving	Open Pit
Tonnes processed (millions)[1]	380	380
Strip ratio	–	3.9
Estimated average annual mill throughput (t)	10,000,000	10,000,000
Estimated average annual production (M kg Nb)	13	15
Operating margin after expansion (US$/kg Nb)[2]	28	28
Estimated capital expenditure (US$ M)	1,400	1,400
Initial capital[3]	840	830
Sustaining capital[4]	560	570
Pre-tax NPV 8% (US$ billions)[5]	2.7	3.3
After-tax NPV 8% (US$ billions)[5]	1.6	2
After-tax Project IRR[6]	21%	24%

Notes:
1. Although the resources are estimated down to Block 6, the NPV under both scenarios only assesses the economic impact of the first 42 years.
2. Based on a US$45/kg niobium price. The current margin is US$18/kg based on a US$40/kg niobium price.
3. Includes working capital, warehouse inventory and C$20 M of capital for pre-stripping (in the OP scenario).
4. Includes the sustaining capital for the current operation and the sustaining capital required for the expansion.
5. Assumed niobium prices per kilogram are US$40/kg in 2011 and US$45/kg thereafter as estimated by an independent source.
6. Percentage points above the IRR of the current 2.2 Mtpa scenario.

SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Head Grade
- Mill Recovery
- Niobium Price
- Operating Costs
- Capital Costs
- Exchange Rate

The after-tax NPV (8%) sensitivity over the base case has been calculated for -20% to +20% variations. The sensitivities are shown in Figure 1-1 and Table 1-3.

FIGURE 1-1 NIOBEC OPEN PIT PROJECT AFTER-TAX SENSITIVITY GRAPH

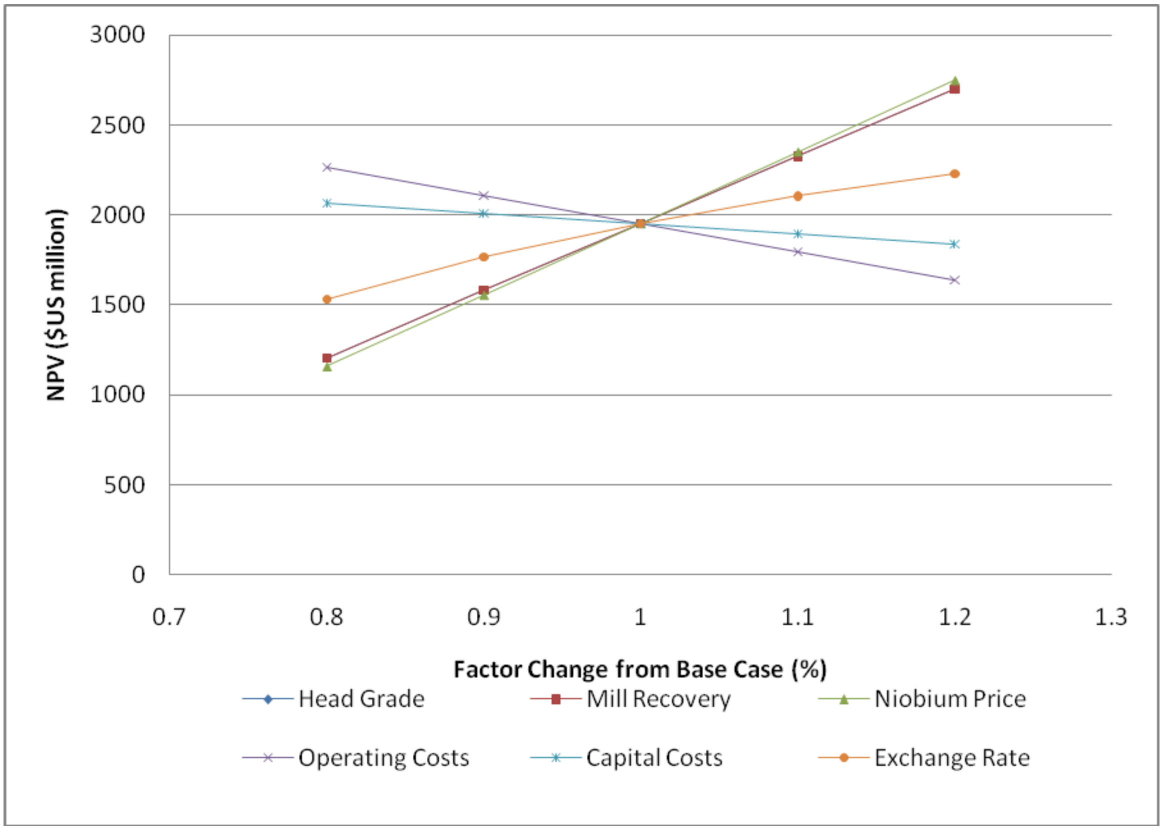


TABLE 1-3 AFTER-TAX SENSITIVITY ANALYSIS
IAMGOLD Corp. – Niobec Mine

Parameter Variables	Units	-20%	-10%	Base	+10%	+20%
Head Grade	% Nb_2O_5	0.37	0.42	0.46	0.51	0.56
Mill Recovery	%	40	45	50	55	60
Niobium Price	US$/kg	36.00	40.50	45.00	49.50	54.00
Operating costs	US$/t milled	20.35	22.90	25.44	27.98	30.53
Capital Costs	US$ billions	1.12	1.26	1.40	1.54	1.68
Exchange Rate	$C/$US	0.84	0.94	1.05	1.15	1.26

NPV 8%	Units	-20%	-10%	Base	+10%	+20%
Head Grade	US$ billions	1.21	1.58	1.95	2.33	2.70
Mill Recovery	US$ billions	1.21	1.58	1.95	2.33	2.70
Niobium Price	US$ billions	1.16	1.56	1.95	2.35	2.75
Operating costs	US$ billions	2.27	2.11	1.95	1.80	1.64
Capital Costs	US$ billions	2.06	2.01	1.95	1.90	1.84
Exchange Rate	$C/$US	1.53	1.77	1.95	2.10	2.23

The sensitivity analysis shows that the Project is most sensitive to head grade, mill recovery and niobium price, moderately sensitive to exchange rate and operating costs, and less sensitive to capital costs.

TECHNICAL SUMMARY

PROPERTY DESCRIPTION AND LOCATION

The Niobec underground (UG) mine is located 25 km north of Ville de Saguenay (Chicoutimi), Québec, in the municipality of Saint-Honoré, in Simard Township, Québec (Figure 4-1). The property is held 100% by Gestion IAMGOLD Québec inc. (IQM), a wholly-owned subsidiary of IAMGOLD. The approximate geographic centre of the property is within National Topographic Series Map reference 22D/11 at longitude 71° 9' 37" west and latitude 48° 31' 44" north. Universal Transverse Mercator (UTM) coordinates for the project centre utilizing projection North American Datum (NAD) 83, Zone 19 are approximately 340,511 m east and 5,377,347 m north. Access to the property is via paved all-weather roads.

LAND TENURE

The Niobec Mine is located on a 2,422.6 ha property comprising two mining leases, Nos. 663 and 706 (with areas of 79.9 ha and 49.5 ha, respectively), and 66 claims totalling 2,293.2 ha. The mining leases have been renewed until 2015 and include surface rights.

There are no outstanding royalty payments on the property. Mineral lease payments are C$2,947.35 per annum.

SITE INFRASTRUCTURE

Currently, the major assets and facilities associated with the Project are:

- The deposit.
- A mine shaft, headframe, access ramp, ventilation raises, maintenance shops, and mobile equipment fleet.
- A coarse ore bin.
- A crushing plant.
- A pyrochlore-to-niobium pentoxide (Nb_2O_5) concentrator.
- A concentrate to ferroniobium (FeNb) converter.
- A paste backfill plant.
- Main ventilation fan.
- A stand-alone assay laboratory.
- Workshops, warehouses, administration buildings, and dry facilities.
- Ample water supply, fire suppression system and sewage treatment.
- Fuel storage and distribution system.
- Main line to Provincial electrical grid, main electrical substation (161 kV), main plant substation, and site distribution network (25 kV).

Access by paved and gravel all-weather roads to the Ville de Saguenay (Chicoutimi) and rail and port infrastructures linking to North American markets.

HISTORY

SOQUEM Inc. (SOQUEM) discovered the Niobec deposit while conducting airborne geophysical surveys to explore for uranium in 1967. In 1970, SOQUEM entered into a joint venture agreement (JVA) with Copperfields Mining Corporation (Copperfields), a predecessor company of Teck Resources (Teck), to explore and develop the niobium deposit.

A joint decision was taken, in 1974, to initiate the development of a 1,500 tpd mine and mill under the management of Teck. The mine was completed and commercial operations started in 1976 with the production of the first niobium pentoxide (Nb_2O_5) concentrate. In 1979, mine production was increased by 30% and mill throughput increased approximately 50% to 2,260 tpd.

As a result of the partial privatization of SOQUEM in 1986, the 50% interest in the Project was transferred to Cambior Inc. (Cambior).

In 1994, the process was expanded to allow conversion to ferroniobium to expand the marketability of the product.

Teck sold its 50% interest to Mazarin Inc. (Mazarin) in 2001. A corporate reorganization of Mazarin resulted in the creation of Sequoia Minerals Inc. (Sequoia) that comprised the industrial minerals segment of Mazarin's holdings. In 2004, Sequoia was acquired by Cambior and in 2006 IAMGOLD and Cambior merged.

GEOLOGY

REGIONAL GEOLOGY

The Saguenay region is underlain by Grenville Province rocks of the Canadian Shield. It is characterized by high-grade metamorphic terranes and deep-level thrust stacks along ductile shear zones. During the Grenvillian Orogeny (1.08 Ga to 0.98 Ga), extensive crustal thickening and tectonic extrusion led to widespread high grade metamorphism. Tectonic extension at the beginning of the Paleozoic incorporated normal faulting, updoming, and igneous alkaline activity and resulted in the formation of the St. Lawrence River Rift system.

The Saint-Honoré alkaline complex (SAC) which hosts the Niobec Mine, is situated along the Saguenay Graben, a 250 km long and 25 km to 40 km wide structure that extends from the St. Lawrence River near Tadoussac to the Lac St.-Jean district. Geology in the vicinity of the SAC comprises anorthosite, syenites, and magnetic diorite gneiss. Shales and limestones found in the vicinity of Saint-Honoré are thought to be the result of a marine transgression during the Ordovician period (about 470 Ma).

PROPERTY GEOLOGY

The SAC, the host to the Niobec Mine, is elliptical in plan view with a north-south trending major axial length of four kilometres and covers approximately 12 km^2 in area. It is comprised of a series of crescent shaped lenses whose age and composition vary with proximity to the core.

The two main niobium zones are subvertical and lenticular in shape. Foliated and often brecciated dolomites and calcites alternate with more massive dolomites that contain red ankerite alteration. The foliated and brecciated unit is host to pyrochlore, the most prevalent niobium mineral, while the massive unit is less mineralized.

MINERAL RESOURCES AND MINERAL RESERVES

RPA conducted an independent update of Mineral Resource estimation, constrained in a Whittle open pit shell. Grades for Nb_2O_5, SiO_2, P_2O_5, and Fe_2O_3 were estimated into blocks using ID^2 weighting. Table 1-4 presents Mineral Resource estimates constrained by Whittle open pit shell at a cut-off grade of 0.2% Nb_2O_5.

TABLE 1-4 RPA INDEPENDENT UPDATED MINERAL RESOURCES – APRIL 1, 2011
IAMGOLD Corp. – Niobec Mine

Classification	Tonnes (000's)	Grade (% Nb_2O_5)	Contained Nb_2O_5 (M kg)
Measured	288,930	0.43	1,242
Indicated	169,180	0.40	685
Measured and Indicated	**458,110**	**0.42**	**1,927**
Inferred	336,445	0.37	1,240

Notes:
1. CIM definitions were followed for Mineral Resources.
2. The Qualified Person for this Mineral Resource estimate is Bernard Salmon, ing.
3. Mineral Resources are estimated at a cut-off grade of 0.20% Nb_2O_5.
4. Mineral Resources are estimated using an average long-term niobium price of US$42 per kg and a US$/C$ exchange rate of 1:1.05.
5. Mineral Resources are constrained by a pit shell to the 2400 level (725 m below surface).
6. Numbers may not add due to rounding.

Grade estimation was conducted using ID^2 in Gemcom GEMS v6.3.0.1 software using uncut original assays employing "hard boundaries" between mineralized domains. Interpolation of grade inside the blocks was done using the following parameters:

- First Pass – Search ellipse dimension of 24.4 m by 18.3 m by 9.1 m.
- Second Pass – Search ellipse dimension of 48.8 m by 36.6 m by 9.1 m.
- Third Pass – Search ellipse dimension of 121.9 m by 91.4 m by 18.2 m.

RPA notes that while Nb_2O_5 is the only material of economic value, other materials, such as SiO_2, Fe_2O_3 and P_2O_5 are also estimated, using the same methodology as Nb_2O_5, because of their importance for blending to maintain the metallurgical recovery of the mill feed.

As the metallurgical recovery of the niobium is variable, each individual sample was coded based on its lithology and mineralization. In the block model, each block is assigned a code using "nearest neighbour" estimation employing the same large search ellipse used for Nb_2O_5 grade estimation.

There are no current Mineral Reserves estimated for the open pit or block caving mining options. Mineral Reserves will be assessed at the pre-feasibility stage of study.

MINING OPERATIONS

The Niobec Mine has been in production since 1976. The present four compartment shaft is 850 m deep and is used for production (ore hoisting) and services (materials and manpower). In addition to the shaft, the mine is serviced by a ramp reaching a depth of 750 m.

Actual production levels are located on the 600, 1000, and 1450 levels and there is further development on the 300, 700, and 1150 levels. Development on production levels is mainly used for ore haulage by trucks to the ore pass. Development is performed using hydraulic jumbos. Ground support is performed to secure the openings using bolters. The broken rock is loaded by load haul dump units (LHD) and hauled by truck to the ore pass. The ore is crushed and hoisted to surface by a skip. Horizontal sill pillars are left between the production levels.

Open stoping has been the main mining method used since mine start-up. Transition to underground paste fill mining method is ongoing. The open stoping method is still used in the upper part of the mine in Block 1 to 3. The average size of the stopes is about 60 m in length, 25 m in width, and 90 m in height. A 25 m pillar is left between the stopes. Secondary extraction of the pillars can be carried out after the complete extraction of the primary stopes. Paste fill mining method is planned in Block 4 to 6.

Three underground mining scenarios were investigated in order to guide future expansion. The first scenario is an updated LOM using the latest 2010 reserve and resource estimates. The second scenario involves the addition of three new mining blocks (7 to 9) using the paste fill mining method. The third scenario involves a change in the mining method approach as the mine would be converted to block caving. This scenario was built using the April 2011 RPA block model.

RPA, in collaboration with IAMGOLD, investigated the potential for open pit mining at the Niobec property. Whittle pit optimization runs were performed based on the following costs and operating parameters:

- Open pit mining: $1.80/t moved
- Milling: $9.50/t milled
- G&A: $1.25/t milled
- Converting: $4.00/kg Nb

- Marketing and freight: $1.41/kg Nb
- Mill recovery: 40% to 60%, depending on Nb_2O_5 head grade
- Converter recovery: 97%
- Niobium in FeNb: 69.9%.

Revenue factors were calculated using the above metallurgical recoveries based on Niobec historical data. The revenue factors were used to generate a net value model which was used to float cones in the Whittle software.

MINE DESIGN

For the purpose of pit optimization, a re-blocking of the block model was done to increase block size to 24.4 m by 24.4 m horizontal by 15.2 m vertical. A pit optimization was run using the previous inputs and a pit slope of 45°. The values of blocks were calculated using a niobium price based on long-term forecasts of $45 per kg and the aforementioned mill and converter recoveries.

The Whittle economic optimization yielded a pit (Pit 6T) which contained the Mineral Resources stated previously in this report. A second economic optimization was performed on the first Whittle result by means of IAMGOLD's Comet Production Scheduler Software in order to maximize the NPV at 8%. A time parameter was introduced into this process to force the schedule to stop in 2052, as was done in the block caving scenario, for comparison purposes. The maximum NPV of this pit optimization process occurred at a cut-off grade of 0.33% Nb_2O_5.

The exercise returned 370 Mt grading 0.46% Nb_2O_5 and corresponds to a smaller pit within Pit 6T, Pit 4T. The proportion of Inferred Resources in the material that may be potentially mineable via open pit is approximately 20%. Waste mining of 1,482 Mt is required, for a strip ratio of 4:1. The pit shell is of circular shape with a 1,830 m diameter and is a maximum of 564 m deep. The crest of the pit encroaches on the current tailings and settling ponds.

For the pit size, production requirements, and recommended equipment fleet, RPA considers mining of 12 m benches and development of 33 m wide ramps, including ditches and safety berms, to be appropriate.

MINING APPROACH THROUGH UNDERGROUND MINE OPENINGS AND EXISTING SURFACE INFRASTRUCTURES

An OP mining method will be used to extract all of the ore in the current mining area. As the pit will be mining through old open stope mine workings, the operating plan includes backfilling of each open stope with crushed limestone or low grade ore when available. During stripping the limestone horizon in the OP, a portion of the waste will be crushed in the pit with a mobile crusher and this material will serve as fill for the UG open stopes of Block 1 via sub-vertical 12 in. to 14 in. diameter drill holes. Sub-economic or low grade ore from Block 1 will be used to fill Block 2 UG openings and same material from Block 2 will be used to fill Block 3 UG openings following the same approach. During stripping overburden and waste, the UG mine will operate until the OP will start producing.

During OP mining, if filled UG stopes are encountered on a bench, the fill material will be reclaimed first and hauled to the waste rock dump or to low grade stock pile or the primary crusher and these openings will serve as first cuts. The OP bench floor elevations for few assigned benches will be designed to fit with drift floor elevations of the UG level.

The pit footprint will eventually reach the existing tailings pond and will be near the future tailings pond. When required, a portion of the existing tailings will be relocated into a new pond and the existing pond will be re-profiled.

The location for waste rock dumps was determined based on the area required (12 km^2 and 76 m high) and with first objectives to minimize the impact on water courses, to have a 100 m buffer distance from Hydro Quebec's high-voltage power lines, and to avoid sterilization of known potential Mineral Resources (REE mineralization north of the open pit). Geotechnical studies and condemnation drilling will have to be carried out at the waste rock dump locations.

Finally, a provisional surface area allows for land purchases for the OP expansion option, particularly for waste rock dumps.

PRE-PRODUCTION SCHEDULE

It is currently estimated that the feasibility studies and permitting will be completed from 2011 through 2013, overlapped and followed by a development and construction phase of two years with increased production starting in the first quarter of 2015.

LIFE OF MINE PRODUCTION SCHEDULE

The mill throughput is scheduled at 10 Mtpa in Year 3 and beyond, ramping up from 5 Mtpa and 7.5 Mtpa in the first and second years of production, respectively.

Twelve million tonnes of waste pre-stripping will be required in Year -1. The transition from UG and OP mines will last four years from Year -4 to -1 (or from 2011 to 2014, as currently planned).

The remaining production from the UG mine combined with the OP expansion project results in a 42 year LOM, up to 2052.

MINE EQUIPMENT

The production schedule requires a fleet capable of moving 55 Mt of material per year, on a 24 hour per day and 365 day per annum schedule.

INFRASTRUCTURE

The OP expansion option is principally a new construction as the current UG operation will be inside the pit limit created by this mining approach. As the current shaft will not be affected until mining activities reach the shaft installation, the existing UG shaft services will remain in place as current underground mining will continue until open pit production start-up.

The water line and water recirculation infrastructure will continue to be used, with the prospect of being expanded. The remainder of the equipment will require demolition. The existing 25 kV line power will be decommissioned in appropriate areas but will keep supplying power to water pumping stations.

With the pit footprint cutting through the Columbium Road near the current mine site, a new road is proposed to reduce traffic flow around the communities. This would permit transport of material to Niobec without having to pass through small villages. The seven kilometre road would be paved.

Building infrastructure is added to the new open pit project to provide appropriate services. Each building is linked up to fire suppression services where required and serviced by a septic field. Potable water is assumed to be provided by the city of Saint-Honoré.

A crushing facility is required to process open pit run-of-mine (ROM) material. A 54” by 75” Metso crusher is proposed to be used with a 50 t/20 t bridge crane. The crane will be installed early to be used during construction. An apron feeder will feed ROM ore via a stockpile conveyor to the ROM coarse ore stockpile of 25,000 t live load capacity. Ore from the covered stockpile will be fed by four apron feeders via another conveyor to the new grinding facility.

As the existing main electrical substation is located in the zone affected by mining, a new substation is proposed just near the new plant. Communications system infrastructure is reproduced in the new design taking advantage of fibre optic links from the 25 kV power network.

This scenario requires a new processing plant as the existing infrastructure will be demolished as a result of open pit operations. Each area has provision for its own or combined heating, ventilation and air conditioning (HVAC) system. The areas affected will be Grinding, Desliming, Pyrite Flotation, Carbonates Flotation, Dewatering, Magnetic Separation, Pyrochlore Flotation, Sulphide Flotation, Leaching and Leach Filtration, Filtration, Drying and Packaging, Tailings Disposal, Converter, Concentrate Storage, and Tailings and Water Management.

RECOVERABILITY

Historically at Niobec, the process recovery has varied for the Nb_2O_5 head grade while the converter recovery has been constant and equal to 97%. The average grade of the potentially mineable material included in the pit shell implies an average mill recovery of 49.6%.

MINERAL PROCESSING

The process utilized at Niobec was first developed from pilot plant programs. The ROM ore is crushed to 100% passing 1 ½ in. and fed to rod mills, ball mills, and classification circuits where the ore is ground to 80% passing 180 µm. The ore is deslimed in two stages of cycloning, and the underflow is sent for conditioning prior to carbonate flotation. The carbonate concentrate is sent to the tails. The carbonate flotation rougher and cleaner tails are cycloned in two stages to change the process water and then sent to the pyrochlore rougher flotation. The rougher concentrate is sent to five stages of cleaner cells. This is followed by pyrite flotation to remove the sulphides leaching with hydrochloric acid to remove phosphorus and then followed by drying to produce a concentrate with less than 0.1% moisture before being converted into ferroniobium. Since 2000, three major expansions have been completed to increase the ferroniobium production.

ORE CHARACTERISTICS

The deposit contains at least two dozen minerals but, at the present time, only two of them are of economic interest. These are pyrochlore $(Na,Ca)Nb_2O_6F$ and columbite $(Fe,Mn)(Nb,Ta)_2O_6$. Pyrochlore itself does not have a rigid chemical composition and contains REE (tantalum, titanium, strontium and zirconium among others) in addition to niobium. Up to eight different varieties of pyrochlore can be found in the deposit. The Fe-enriched pyrochlore and columbite are usually found in altered ore, but are also present in unaltered ore where they are of primary origin.

The variable chemical composition has a major influence on mill production results. A portion of black Fe-enriched pyrochlore and columbite at a certain pH have surface properties different from those of the sodium type of pyrochlore. These two minerals are in fact lost to tailings in the flotation process.

CRUSHING

Primary crushing is done underground with a jaw type crusher before being hoisted to the headframe bin, which has 600 t capacity. A vibrating feeder is located under the headframe bin in order to feed the conveyor to the secondary crusher. The secondary crusher is a gyratory type crusher and its product is stored in four 1,200 t insulated bins.

GRINDING

There are two parallel grinding circuits. Each line is designed with the same equipment specifications. The tertiary crusher screen undersize and tertiary crusher discharge are directly fed to both rod mills. The rod mill discharges on each line is combined with the ball mill discharges before each feeding a 24 way distributor via their mill discharge pump box pumps. Each distributor feeds a set of vibrating screens where the coarse material reports to a screw classifier and the fines that are at 80% passing 180 µm are sent to the desliming circuit.

The softness of the ore, the relatively high specific gravity of the pyrochlore, and the chemistry of the flotation process dictates the configuration of the grinding circuit in order to avoid overgrinding of the pyrochlore crystals.

DESLIMING

The desliming circuit is designed to remove the material that is smaller than seven µm before flotation. This is done in two steps of cyclone classification.

PYRITE FLOTATION

The slurry at 54% solids is first conditioned in two high intensity conditioners in series where copper sulphate and potassium amyl xanthate (PAX) are added. The pyrite flotation is done in two steps including a rougher and cleaner. The cleaning step is made up of 12 cells.

CARBONATE FLOTATION

After the pyrite flotation, the slurry is conditioned with high intensity at 55% solids with an emulsified fatty acid collector and the pH stays close to its natural level. High intensity conditioning is done in two agitated tanks in series. The carbonate flotation has one rougher step and two cleaning steps. Approximately 35% of the weight and 9% of the pyrochlore of the mill feed is floated off in the second cleaner concentrate. The flotation concentrate consists of very fine calcite particles (-50 µm) and medium size apatite.

DEWATERING

After carbonate flotation, the slurry is sent to cyclones. The slurry is first diluted with fresh water to 30% solids. It is then pumped to primary dewatering cyclones. The overflow, at about 2% solids, is pumped to the secondary dewatering cyclones. About

4% of the mill feed by weight and 7% of the pyrochlore are removed in the dewatering cyclones overflow.

MAGNETIC SEPARATION

The dewatering cyclone underflows are sent to magnetic separation. This process removes 2% of the weight and 1% of the pyrochlore.

PYROCHLORE FLOTATION

The non-magnetic material from the magnetic separator is pumped to two high intensity conditioning tanks in series. The slurry is then sent to a bank of flotation cells where the pyrochlore collector is stage added. The rougher concentrate is sent to five stages of cleaning where the pH is gradually reduced to 2.7. Approximately 60% of the pyrochlore is recovered in the concentrate in 0.8% of the weight, while 20% of the pyrochlore is sent to the tails

SULPHIDE FLOTATION

The cleaned pyrochlore concentrate contains about 20 wt% of pyrite, which after the addition of sodium hydroxide is conditioned with silicate to depress the pyrochlore. PAX is then added and a rougher pyrite concentrate is floated.

LEACH FILTERING

The leached product is sent to a belt filter to remove most of the water. The solid cake is mixed with fresh water and copper sulphate for the second sulphide flotation.

SECOND SULPHIDE FLOTATION

The leftover activated sulphides are floated in conventional type cells. The pH is adjusted to 11 with NaOH and PAX is added as collector.

DRYING

The final pyrochlore concentrate is pumped at about 40% solids to a double 4 in. disks filter and sent to a propane countercurrent dryer where the moisture level is reduced to less than 0.1%.

PACKAGING

The dried product is stored in twelve bins. The concentrate is packed into large bags by an automated packing-handling system then transferred to the converter.

CONVERTER

After the pyrochlore concentrate is dried, it is transferred to the converter where the material is transformed into ferroniobium (standard grade). The niobium oxide is converted into FeNb by using an aluminothermic reaction on a batch basis.

PROCESS SELECTION

The expansion to an open pit or block caving mining scenario will necessitate the construction of a new processing plant facility. Major issues will be faced, principally due to the increase in equipment size and the lower mill feed grade. Those issues will be minimized by the utilization of well know equipment and trials done during the last few years. The utilization of the SAG mill and ball mill with conventional cyclones are considered one of the major risks in terms of niobium recovery due to the potential increase in fine particle production and losses at the desliming stage. The grinding circuit, however, will be simpler to operate and easier to perform automatic control strategies. The niobium recovery process will stay the same in terms of metallurgy.

Proposed plant modifications are discussed below.

ORE HANDLING, CRUSHING AND STORAGE

The ore will be delivered to a gyratory crusher before it is sent to the covered stockpile. The ore from the current underground mine will be redirected to the new ore stacker.

GRINDING AND CLASSIFICATION

The grinding circuit will be modified in terms of technology. The ore classification will continue to be done using vibrating screen technology.

DESLIMING

The same type and size of equipment used in the present circuit will be used.

PYRITE FLOTATION

A pyrite circuit will be built based on the current design.

CARBONATE FLOTATION

Two parallel lines of tank cells, each including a conditioner for reagent conditioning, will be used. The first cleaner will consist of eight tank cells and the second of five tank cells.

DEWATERING

The dewatering will be done using two clusters of D10 cyclones followed by eight clusters of D4 and finally two clusters of D2 cyclones

PYROCHLORE FLOTATION

The rougher circuit will be done in 12 tank cells in series. The reagent conditioning will be done in a highly agitated tank. Five cleaner stages will follow. The concentrate will be cleaned using three magnet separators before being sent to the sulphide flotation.

SULPHIDE FLOTATION

The final sulphide flotation will be done using twelve flotation cells.

PHOSPHATE LEACHING

The pyrochlore concentrate from the sulphide flotation will be sent to a 25 m diameter thickener before being sent to four leach tanks in series. The concentrate will be filtered using six belt filters in parallel.

SECOND SULPHIDE FLOTATION

The removal of final sulphide minerals will be done using eight flotation cells.

PACKAGING

The current packaging system built in 2010 appears to have the capacity to support the additional production by adding 12 additional concentrate storage bins.

CONVERTER

The design capacity will be twice the existing converter capacity.

TAILINGS PUMPS

Two tailings lines will be installed, one to provide the coarse material for the tailings dam construction and the second for the disposal of the carbonate and slimes inside the pond.

MILL WATER SUPPLIES AND DISPOSAL

To provide the additional water demand to the mill, the new system that will be built at the Shipshaw River will need to quadruple the current water intake.

OPERATIONS MANAGEMENT

Most of the operating basis will be the same as the current.

OPERATING COSTS

Operating costs have been derived from the actual and historical cost.

REAGENT CONSUMPTIONS AND SUPPLIES

A study performed on the reagents used in the process revealed that at the time of the study no issues were raised on the supplies.

METALLURGICAL TESTING

Metallurgical testwork will need to be done in order to properly design the new processing facilities. Crushing and principally the SAG index will need to be done for proper equipment design and selection. Extensive metallurgical testwork will need to be performed to validate the niobium recovery and concentrate quality grade. Also, niobium recovery improvement will be investigated by modifying the reagent scheme or process flowsheet. Comparative testwork between conventional grinding and classification presently used and new grinding circuit using SAG-Ball mill will need to be done.

ENVIRONMENTAL CONSIDERATIONS

The environmental management system (EMS) for the Niobec Mine is certified under the 2004 revision of the ISO 14001 standard. Niobec successfully passed the ISO 14001 recertification audit in November 2010. Niobec's quality management system is certified ISO 9001:2008 since 1995; it was last recertified in 2009.

In terms of environmental requirements, the Niobec Mine must comply with both Federal and Provincial laws and regulations.

All expansion, construction, and major modification projects in Québec need approval by the Provincial government through the environmental permitting process.

In terms of health and safety requirements, the Niobec Mine must comply with Québec's provincial laws and regulations.

The most recent version of Niobec's closure plan was approved by Ministère des Ressources Naturelles et de la Faune (MRNF) in September 2009. A revised rehabilitation plan has to be submitted to the MRNF by September 2014.

Depending on the nature of the project, a detailed environmental and social impact assessment may be required. The environmental and social impact assessment process is triggered if the production rate of a mineral processing plant is greater than 7,000 tpd or if a mine is opened and operated with a production rate greater than 7,000 tpd. This threshold will be lowered to 3,000 tpa if proposed Mining Act reforms are enacted. The whole assessment, public hearing, and subsequent project analysis by the Provincial government may take up to 15 months and the cost is estimated at C$1 million.

A formal risk assessment for the Project will be conducted through three distinct processes: the environmental and social impact assessment required for the permitting process, the IAMGOLD risk assessment process, and the IAMGOLD Safety in Design study.

For the PEA purposes, the main foreseeable risks and impacts associated with the Project are:

- The OP scenario (1,140 tonnes per hour) would trigger the provincial environmental and social impact assessment and public hearing process

- The existing surface infrastructure would have to be demolished and could be considered as a partial existing mine closure thereby partially activating the Asset Retirement Obligation (ARO) plan.

- Significant land purchase is necessary for the proposed tailings storage facility and waste rock dumps. Depending on how the land purchase negotiation process goes, it may be required to expropriate land.

- New water supply and effluent discharge lines are required. In order to install those new water lines, it may be necessary to expropriate lands located between Niobec and the Shipshaw River. The freshwater supply to the expanded mill and infrastructure would increase significantly. Since mine effluent will be discharged back into the river, additional retention basins and/or a wastewater treatment plant may need to be constructed. Additional septic tanks and leachate fields will be required as well, to account for a potential increase in the discharge of domestic wastewater.

- The hazardous materials and reagents consumption is expected to increase, as well as the hazardous waste and non-hazardous waste generation, energy consumption, and the greenhouse gas emissions.

- Increased health and safety risks will be associated with infrastructure construction activities as well as shaft demolition activities. Opening new surface excavations may lead to a release of radon.

- The open pit itself and the waste rock dumps may constitute a significant environmental and community impact.

These risks would be managed under the Safety Management Plans such as the Health, Safety, Environmental and Community Management Plans and the Closure Plan.

CAPITAL AND OPERATING COST ESTIMATES

The mine capital cost includes mining equipment fleet purchases and pre-stripping and site work related to the open pit. The mine fleet was estimated based on OP operations of a similar scale. The mine services cost covers haul road construction and site work related to open pit and waste rock dump preparation (stripping of vegetation and overburden). Capital expenditures for the underground mine are also required as, during the first four years of the LOM, production will continue from the existing UG mine. The UG capital costs include a new ventilation system and distribution network (existing ventilation raises located within the pit footprint), an increase in pumping capacity to manage additional surface water flowing through the filled open stopes to be eventually found within the open pit, and the first drilling campaign (contracted) to fill open stopes with crushed waste rock.

The ore processing capital cost was estimated by IAMGOLD based on the current process flow sheet and a scale-up of the existing mill and converter facilities. Detailed equipment lists were generated for each area within the ore processing cost item. This capital cost prevails for the block caving and the open pit options as the respective production rates are the same.

Infrastructure costs include general site preparation, construction of on-site roads, and upgrade of a provincial road as main access to the industrial mine site, etc. Also under this cost item are building construction, equipment and furniture, power distribution, fuel storage and distribution, fire protection and laboratory.

Waste rock dump water management costs were provided based on IAMGOLD's approach to tailings water management regarding ditches and effluent water system. The effluent water system also considers open pit mine water. Budget allocation is included here for land purchases at waste rock dump location.

Tailings and process water management costs include an upgrade to the fresh water intake system and pipeline from/to Shipshaw River to the west, as well as effluent water system and tailings pond site preparation and construction. For waste rock dumps, a budget allocation is included for land purchases at the tailings pond location.

Indirect capital costs consist of working capital, warehouse inventory, owner's cost, mill start-up/commissioning, and Engineering, Procurement, Construction Management (EPCM). EPCM costs vary between 15% and 20% of direct capital cost items and also include construction of temporary installations, equipment, tools, travel and lodging, for an average of approximately 13% of total direct capital cost (not considering OP mine fleet).

The mine, mill, and site infrastructure capital costs are summarized in Table 1-5. All costs are in 2011 US$.

TABLE 1-5 CAPITAL COST SUMMARY
IAMGOLD Corp. – Niobec Mine

Area	Cost (US$ M)
Mine	187.9
Mill, Crusher and Converter	245.0
Infrastructure	59.6
Waste rock dump water management	30.3
Tailings and Process water management	67.6
Indirects	134.5
Contingency (≈15% on average)	104.6
Total Pre-Production Capital	**829.5**

Sustaining capital costs are subdivided into two periods. First, during the initial four-year pre-production period (or construction phase), from Year -4 to Year -1 (2011 to 2014 inclusively), costs related to existing underground mine expansion are labelled as sustaining capital. Second, from Year 1 to Year 38 of the production phase (2015 to 2052 inclusively), all capital costs occurring during this period were considered as sustaining capital. Sustaining capital consists of:

- Mine equipment fleet replacement;
- Crusher, mill, and converter buildings maintenance and equipment replacement;
- Relocation of a portion of existing tails, once open pit reach existing tailings pond;
- Drilling holes (contracted) to fill UG openings from the open pit;
- Progressive rehabilitation and mine closure.

Sustaining capital costs are shown in Table 1-6.

TABLE 1-6 CAPITAL COSTS – SUSTAINING
IAMGOLD Corp. – Niobec Mine

Area	Item	Cost (US$ M)
Sustaining	Existing Underground Mine	91.3
	Mine Equipment Fleet	341.9
	Crusher, Mill and Converter	54.0
	Existing Tailings	30.0
	Holes to fill UG openings	4.5
	Progressive Rehabilitation and Mine Closure	47.5
Total Sustaining		**569.2**

The following is excluded from the capital cost estimate:

- Project financing and interest charges
- Escalation during the project
- Permits, fees and process royalties
- Pre-feasibility and Feasibility studies
- Environmental impact studies
- Any additional civil, concrete work due to the adverse soil condition and location
- Taxes
- Import duties and custom fees
- Cost of geotechnical and geomechanical investigations

- Rock mechanics study
- Metallurgical testwork
- Exploration drilling
- Costs of fluctuations in currency exchanges
- Project application and approval expenses

Operating costs for production over the LOM are summarized in Table 1-7. Unit costs are detailed for the whole Project and for the open pit only, because the existing underground mine will still be producing at 2.2 Mtpa during the four-year construction phase of the Project. All costs in this section are in 2011 C$.

TABLE 1-7 LOM AVERAGE UNIT OPERATING COST SUMMARY
IAMGOLD Corp. – Niobec Mine

	UG&OP C$/t milled	UG&OP C$/t moved	UG C$/t milled	OP C$/t milled	OP C$/t moved
Mining	9.31	1.90	19.54	9.07	1.82
Processing	9.70		13.45	9.50	
Converting	6.31		9.57	6.23	
G&A	1.35		4.03	1.25	
Total	**26.67**		**46.59**	**26.05**	

Manpower estimates were based on typical numbers in Canadian open pit operations of a similar scale, and IAMGOLD scaled-up manning based on current data for milling, converting, and G&A. Manpower estimates for the various administrative units are shown in Table 1-8.

TABLE 1-8 MANPOWER SUMMARY
IAMGOLD Corp. – Niobec Mine

Unit	Operation	Maintenance	Supervision and Services	Total
Administration	---	---	46	46
Mine	225	115	20	360
Mill	100	60	20	180
Converter	68	8	5	81
Surface and Electrical	---	65	11	76
Total	**393**	**248**	**102**	**743**

2 INTRODUCTION

Roscoe Postle Associates Inc. (RPA) was retained by IAMGOLD Corporation (IAMGOLD), to prepare an independent Technical Report on the Niobec Mine (the Project), near Ville de Saguenay (Chicoutimi), Québec. The purpose of this report is to prepare an updated Mineral Resource estimate and a Preliminary Economic Assessment (PEA) on the viability of an OP option at the Project. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects. RPA visited the property on March 4, 2011 and again from March 21 to 22, 2011.

IAMGOLD is a Canadian mining company, headquartered in Toronto, Ontario, that produces one million ounces annually from eight gold mines on three continents. IAMGOLD was created as a private company in 1991 when it discovered its first gold deposit, the Sadiola Hill mine (Sadiola) in Mali, Africa. In 1996, IAMGOLD began trading on the Toronto Stock Exchange, Sadiola poured it first gold bar, and exploration commenced in South America. IAMGOLD continued to expand through strategic acquisitions of projects and through the conversion of discretionary assets into gold bullion. IAMGOLD began trading on the American Stock Exchange in 2001 and the New York Stock Exchange in 2005. In 2006, IAMGOLD acquired Cambior Inc. and its 100% ownership of the Project. IAMGOLD has since acquired Orezone Resources in 2009 and declared commercial gold production at its Essakane Project in Burkina Faso, Africa. IAMGOLD has development projects in Canada, Ecuador, and French Guiana, and is actively exploring in Africa and South America. IAMGOLD also holds a 1% royalty in the Diavik diamond property in Canada. A summary of IAMGOLD 2009 production is shown in Table 2-1.

TABLE 2-1 IAMGOLD 2009 PRODUCTION
IAMGOLD Corp. – Niobec Mine

Headings	Operation	Gold Production (000s oz)	Niobium Production (000s kg)
IAMGOLD Operator			
	Rosebel (95%)	392	
	Doyon Division (100%)	109	
	Mupane (100%)	51	
	Essakane (90%)	n/a	
	Niobec (100%)		4,100
Joint Venture			
	Sadiola (41%)	135	
	Yatela (40%)	89	
	Tarkwa (18.9%)	125	
	Damang (18.9%)	38	

Source: IAMGOLD.com

Prior RPA involvement in the Niobec Project includes a reserve audit and review of mining operations that formed part of a technical due diligence conducted in 2001.

This report is considered by RPA to meet the requirements of a PEA as defined in Canadian NI 43-101 regulations. The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the reserves development, production and economic forecasts on which this preliminary assessment is based will be realized.

SOURCES OF INFORMATION

Site visits were carried out by Jacques Gauthier, ing., Manager of Engineering - Québec for RPA, Marc Lavigne, ing., Senior Mining Engineer for RPA, and Barry McDonough, P. Geo., Senior Geologist for RPA.

Discussions were held with personnel from IAMGOLD`:

- Pierre Pelletier, ing., Vice-President, Metallurgy, IAMGOLD

- Réjean Sirois, ing., Manager Mine Geology, IAMGOLD
- Daniel Vallières, ing., Manager, Underground Projects, IAMGOLD
- Philippe Gaultier, ing., Manager Engineering IAMGOLD
- Nathaniel Chouinard, Manager, Mergers and Acquisitions, IAMGOLD
- Alain Grenier, ing., General Manager, Niobec Mine, IAMGOLD
- Steve Thivierge, ing., Geology and Project Superintendent, Niobec Mine, IAMGOLD
- Marianne Blais, ing., Technical Superintendant, Niobec Mine, IAMGOLD
- Jean-Francois Tremblay, P. Geo., Chief Geologist, Niobec Mine, IAMGOLD

The Mineral Resource review and geological aspects of the report were carried out by Bernard Salmon, ing., and Barry McDonough, P. Geo., of RPA and comprise Sections 1 through 15 and 17, and parts of 19 through 21. Mining aspects were carried out by Marc Lavigne, M.Sc., ing., of RPA and Daniel Vallières, ing., of IAMGOLD; they are responsible for parts of Sections 18 through 21. Mineral processing and metallurgical testing were carried out by Pierre Pelletier, ing., of IAMGOLD, who is responsible for Section 16 and parts of 18 through 20. Markets in Section 18 has been prepared by Graham G. Clow, P.Eng., of RPA, who also has overall responsibility for the report

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 21, References.

LIST OF ABBREVIATIONS

Units of measurement used in this report conform to the SI (metric) system. All currency in this report is US dollars (US$) unless otherwise noted.

μ	micron
°C	degree Celsius
°F	degree Fahrenheit
μg	microgram
A	ampere
a	annum
bbl	barrels
Btu	British thermal units
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
d	day
dia.	diameter
dmt	dry metric tonne
dwt	dead-weight ton
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
g/t	gram per tonne
gpm	Imperial gallons per minute
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
hr	hour
ha	hectare
hp	horsepower
in	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km/h	kilometre per hour
km^2	square kilometre
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
L/s	litres per second
m	metre
M	mega (million)
m^2	square metre
m^3	cubic metre
min	minute
MASL	metres above sea level
mm	millimetre
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
m^3/h	cubic metres per hour
opt, oz/st	ounce per short ton
oz	Troy ounce (31.1035g)
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
st	short ton
stpa	short ton per year
stpd	short ton per day
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
tph	metric tonne per hour
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
yd^3	cubic yard
yr	year

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by RPA for IAMGOLD. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to RPA at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by IAMGOLD and other third party sources.

For the purpose of this report, RPA has relied on ownership information provided by IAMGOLD. RPA has not researched property title or mineral rights for the Niobec Mine and expresses no opinion as to the ownership status of the property.

RPA has relied on IAMGOLD for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Niobec Mine.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

The Niobec Mine is located 25 km north of Ville de Saguenay (Chicoutimi), Québec, in the limits of the municipality of Saint-Honoré, in Simard Township, Québec (Figure 4-1). The property is held 100% by Gestion IAMGOLD Québec inc. (IQM), a wholly-owned subsidiary of IAMGOLD. The approximate geographic centre of the property is within National Topographic Series Map reference 22D/11 at longitude 71° 9' 37" west and latitude 48° 31' 44" north. Universal Transverse Mercator (UTM) coordinates for the project centre utilizing projection North American Datum (NAD) 83, Zone 19 are approximately 340,511 m east and 5,377,347 m north. Access to the property is via paved all-weather roads.

LAND TENURE

The Niobec Mine is located on a 2,422.6 ha property comprising two mining leases, Nos. 663 and 706 (with areas of 79.9 ha and 49.5 ha, respectively), and 66 claims totalling 2,293.2 ha. The mining leases have been renewed until 2015 and include surface rights. A list of the active mineral tenures is shown in Table 4-1 and a map showing claims in the vicinity of the mine is shown in Figure 4-2. Mineral lease boundaries have been established by legal survey. The remaining property boundaries have been established by cadastral survey for the mining claims delineated by lots and parcels and by geographical coordinates for the map designated map claims. There are no outstanding royalty payments on the property and mineral lease payments are C$2,947.35 per annum.

RPA is of the opinion that the proposed pit will necessitate the expansion of the mining lease and will require permitting for waste rock and overburden dumps. Permitting for environment will require to be updated.

TABLE 4-1 NIOBEC LAND TENURE
IAMGOLD Corp. – Niobec Mine

NTS Sheet	Tenure Type	Tenure Number	Status	Recording Date	Expiration Date	Area (ha)	Owner (Percentage)
22D11	BM	BM 663	Active	16/01/1975	15/01/2015	79.93	IQM (100%)
22D11	BM	BM 706	Active	05/06/1980	04/06/2015	49.52	IQM (100%)
22D11	CL	CL 2687601	Active	26/10/1967	13/09/2011	20	IQM (100%)
22D11	CL	CL 2687602	Active	26/10/1967	13/09/2011	21.4	IQM (100%)
22D11	CL	CL 2712071	Active	26/10/1967	13/09/2011	40	IQM (100%)
22D11	CL	CL 2712072	Active	26/10/1967	13/09/2011	21.4	IQM (100%)
22D11	CL	CL 2712122	Active	26/10/1967	14/09/2011	40	IQM (100%)
22D11	CL	CL 2713201	Active	26/10/1967	24/09/2011	21.4	IQM (100%)
22D11	CL	CL 2713202	Active	26/10/1967	24/09/2011	21.4	IQM (100%)
22D11	CL	CL 2713212	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713221	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713222	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713231	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713232	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713241	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713242	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713251	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713252	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713362	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713371	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713372	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713442	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713451	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713452	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713461	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713462	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713471	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713472	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713481	Active	26/10/1967	24/09/2011	40	IQM (100%)

NTS Sheet	Tenure Type	Tenure Number	Status	Recording Date	Expiration Date	Area (ha)	Owner (Percentage)
22D11	CL	CL 2713482	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713491	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713492	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713541	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713542	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713551	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713552	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713561	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713562	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713571	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 2713621	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713622	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713631	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713632	Active	26/10/1967	24/09/2011	40	IQM (100%)
22D11	CL	CL 2713641	Active	26/10/1967	25/09/2011	40	IQM (100%)
22D11	CL	CL 5044599	Active	23/11/1989	22/11/2011	20	IQM (100%)
22D11	CDC	CDC 2198143	Active	05/01/2010	04/01/2012	42.4	IQM (100%)
22D11	CDC	CDC 2198144	Active	05/01/2010	04/01/2012	7.41	IQM (100%)
22D11	CDC	CDC 2198145	Active	05/01/2010	04/01/2012	8.42	IQM (100%)
22D11	CDC	CDC 2198146	Active	05/01/2010	04/01/2012	9.4	IQM (100%)
22D11	CDC	CDC 2198147	Active	05/01/2010	04/01/2012	10.4	IQM (100%)
22D11	CDC	CDC 2198148	Active	05/01/2010	04/01/2012	11.63	IQM (100%)
22D11	CDC	CDC 2198149	Active	05/01/2010	04/01/2012	12.34	IQM (100%)
22D11	CDC	CDC 2198150	Active	05/01/2010	04/01/2012	13.34	IQM (100%)
22D11	CDC	CDC 2198151	Active	05/01/2010	04/01/2012	14.31	IQM (100%)
22D11	CDC	CDC 2198152	Active	05/01/2010	04/01/2012	15.29	IQM (100%)
22D11	CDC	CDC 2198153	Active	05/01/2010	04/01/2012	16.27	IQM (100%)
22D11	CDC	CDC 2198154	Active	05/01/2010	04/01/2012	0.54	IQM (100%)
22D11	CDC	CDC 2198155	Active	05/01/2010	04/01/2012	17.26	IQM (100%)
22D11	CDC	CDC 2198156	Active	05/01/2010	04/01/2012	11.14	IQM (100%)
22D11	CDC	CDC 2198157	Active	05/01/2010	04/01/2012	41.07	IQM (100%)

NTS Sheet	Tenure Type	Tenure Number	Status	Recording Date	Expiration Date	Area (ha)	Owner (Percentage)
22D11	CDC	CDC 2198158	Active	05/01/2010	04/01/2012	57.05	IQM (100%)
22D11	CDC	CDC 2198159	Active	05/01/2010	04/01/2012	57.05	IQM (100%)
22D11	CDC	CDC 2198160	Active	05/01/2010	04/01/2012	57.05	IQM (100%)
22D11	CDC	CDC 2198161	Active	05/01/2010	04/01/2012	57.05	IQM (100%)
22D11	CDC	CDC 2198162	Active	05/01/2010	04/01/2012	57.04	IQM (100%)
22D11	CDC	CDC 2198163	Active	05/01/2010	04/01/2012	57.04	IQM (100%)
22D11	CDC	CDC 2198164	Active	05/01/2010	04/01/2012	57.04	IQM (100%)
22D11	CDC	CDC 2198165	Active	05/01/2010	04/01/2012	57.04	IQM (100%)
TOTAL		**68 titles**				**2,422.63**	

Figure 4-1

Iamgold Corporation

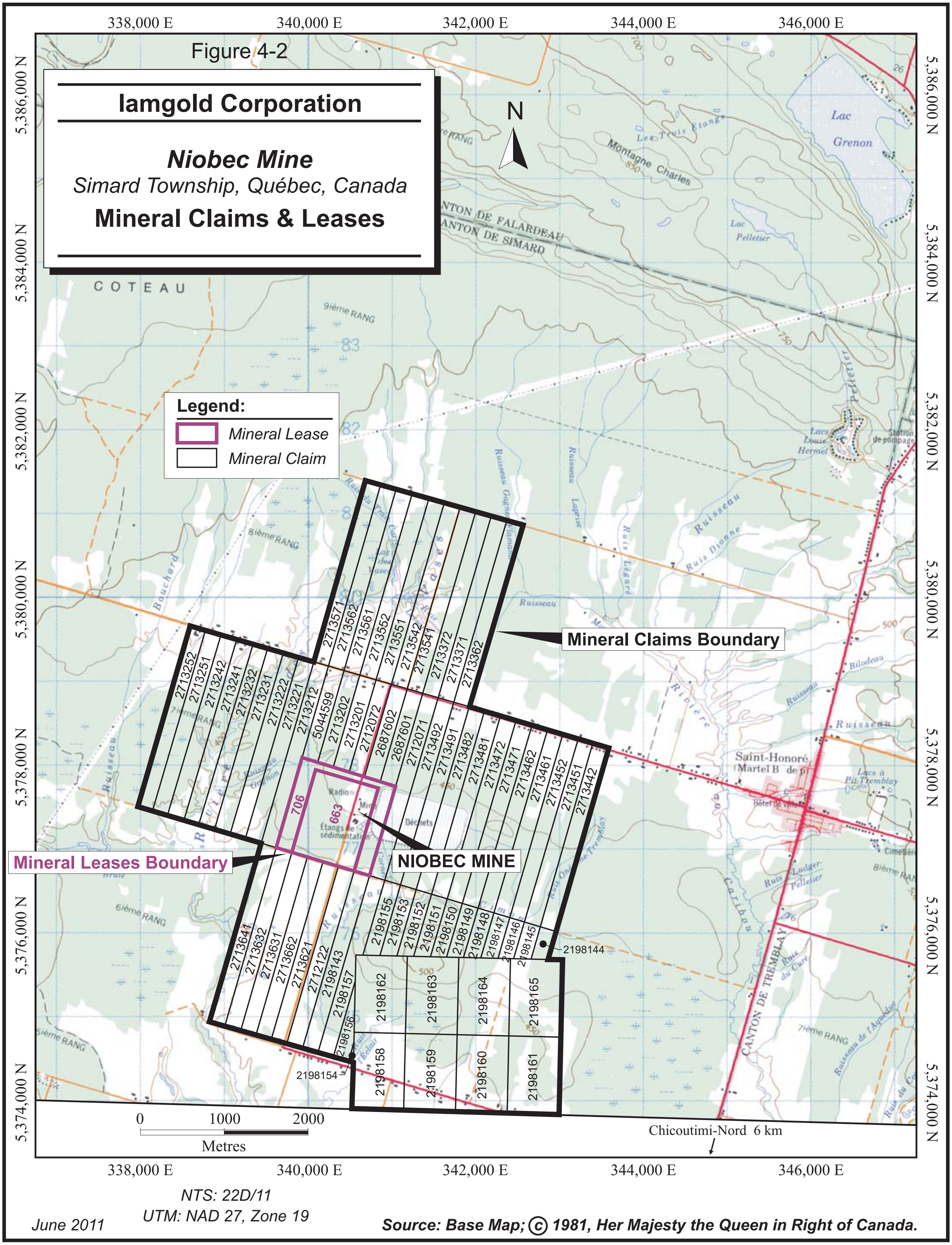
Niobec Mine

Simard Township, Québec, Canada

Location Map

June 2011

Source: Iamgold Corporation, 2011.



June 2011
NTS: 22D/11
UTM: NAD 27, Zone 19
Source: Base Map; © 1981, Her Majesty the Queen in Right of Canada.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

The Project is readily accessible by well established, all-weather paved and gravel roads. It is located in Simard Township, approximately 25 km north of the Ville de Saguenay (Chicoutimi), Québec, and adjacent to the municipality of Saint-Honoré, Québec. Access to the Project from Ville de Saguenay is via Québec Provincial Highway 172, travelling west, to Rue Martel. Bearing north-northeast for approximately nine kilometres, Rue Martel intersects Rue de l'Hôtel-de-Ville. Rue de l'Hôtel-de-Ville is followed west-northwest for approximately five kilometres to where it intersects the gravel Route du Columbium. The mine gate is located approximately 1.5 km south-southwest of this intersection.

CLIMATE

The climate in the Saguenay–Lac-St-Jean region is humid temperate with mild summers. The mean annual temperature for the area is above the freezing point at 2.3°C. Average July temperature is 18.1°C, and average January temperature is -16.1°C.

According to the 1971 to 2000 precipitation data from Environment Canada, the average annual precipitation is 951 mm. Rain precipitation is highest in July, averaging 123 mm of water. Snow precipitation is registered between September and May, but its peak falls on the period between December and March, when its monthly average reaches 63 mm (expressed in mm of water). The prevailing winds are from the West (33% of the time).

LOCAL RESOURCES

The Niobec Mine is close to Ville de Saguenay with a population, according to 2001 census data, of about 170,000. The city is serviced several times a day by regional airlines from Montreal and boasts excellent road, rail, and port infrastructures.

The mine has been in production since 1976 and has well established suppliers of equipment and services from the region. Governmental and health services, schools, and manpower are all available in Ville de Saguenay and other communities in the vicinity. The Project enjoys the support of local communities.

INFRASTRUCTURE

Currently, the major assets and facilities associated with the Project are:

- The orebody.
- A mine shaft, headframe, access ramp, ventilation raises, maintenance shops, and mobile equipment fleet.
- A coarse ore bin.
- A crushing plant.
- A pyrochlore-to-niobium pentoxide (Nb_2O_5) concentrator.
- A concentrate to ferroniobium (FeNb) converter.
- A paste backfill plant.
- Main ventilation fan.
- A stand-alone assay laboratory.
- Workshops, warehouses, administration buildings, and dry facilities.
- Ample water supply, fire suppression system and sewage treatment.
- Fuel storage and distribution system.
- Main line to Provincial electrical grid, main electrical substation (161 kV), main plant substation, and site distribution network (25 kV).
- Access by paved and gravel all-weather roads to the Ville de Saguenay (Chicoutimi) and rail and port infrastructures linking to North American markets.

PHYSIOGRAPHY

The topography is generally flat in the vicinity of the mine with an average altitude of 144 MASL. The vegetation in the Niobec area is mostly forest dominated by coniferous three species such as black spruce, fir tree, and larch tree. There is also some poplar tree and grey pine. The average vegetation density ranges from 40% to 80%. There is no known plant or tree species at risk around the Niobec area.

The fauna found in the Niobec area is typical of the Saguenay–Lac-St-Jean region. Red fox, rabbits, squirrels, black bears, partridges, black crows, Canada geese, and other types of birds and mammals are found in the area. No significant water bodies are found at the Niobec Mine. However, a river, the Rivière-aux-Vases, is located about two kilometres west of Niobec. There is no known animal species at risk around the Niobec area.

6 HISTORY

While conducting airborne geophysical surveys to explore for uranium in 1967, SOQUEM Inc. (SOQUEM) detected coincident circular radiometric and magnetic anomalies. The anomalies, in the vicinity of the deposit, were followed up and led to the discovery of the Saint-Honoré carbonatite complex. Later ground geophysics and diamond drilling by SOQUEM led to the discovery of the Rare-Earth zone and two niobium-bearing zones that lie in the southern part of the complex.

In 1970, SOQUEM entered into a joint venture agreement (JVA) with Copperfields Mining Corporation (Copperfields), a predecessor company of Teck, to explore and develop the niobium deposit (Hatch, 2001).

A shaft was sunk to obtain adequate samples for metallurgical evaluation. After 700 bench scale tests, 11 months of pilot plant operation, and worldwide market research, a joint decision was taken, in 1974, to initiate the development of a 1,500 tpd mine and mill under the management of Teck. The mine was completed and commercial operations started in 1976 with the production of the first niobium pentoxide (Nb_2O_5) concentrate (Belzile, 2009).

In 1978, Niobec Inc. on behalf of SOQUEM, and The Yukon Consolidated Gold Corporation Limited (Yukon) on behalf of Copperfields, signed a JVA based on the previous SOQUEM/Copperfields agreement. The terms of the JVA included:

- Niobec Inc. would sell an undivided 50% interest in certain claims to Yukon.
- Teck Mining Group Limited (TMG) would act as manager of the joint venture project and carry out exploration and develop the niobium deposit.

Simultaneously, Yukon, on behalf of Copperfields, and Niobec Inc., on behalf of SOQUEM, also signed an Operators Agreement and Marketing Operator Agreement with terms that included, among others:

- TMG would become Operator of the Niobec Mine.

- TMG would act as an independent contractor and not as an agent of the participants in the agreement.
- Niobec Inc. would become the Marketing Operator responsible for administering the sales contracts for niobium produced at the Niobec Mine.
- Niobec Inc. would act as an independent contractor and not as an agent of the participants in the agreement.

In 1979, mine production was increased by 30% and mill throughput increased approximately 50% to 2,260 tpd (Hatch, 2001).

As a result of the partial privatization of SOQUEM in 1986, the 50% interest in the Project was transferred to Cambior.

Since 1994, the operators have converted pyrochlore to niobium pentoxide and then into ferroniobium to expand the marketability of their product. On-site infrastructure has been built and the ferroniobium can be delivered in various grain sizes and packaged to meet customer needs.

In 1996, the cut-off grade for production was raised from 0.5% Nb_2O_5 to 0.6% Nb_2O_5 (Hatch, 2001).

Teck sold its 50% interest to Mazarin Inc. (Mazarin) in 2001. A corporate reorganization of Mazarin resulted in the creation of Sequoia Minerals Inc. (Sequoia) that comprised the industrial minerals segment of Mazarin's holdings. In 2004, Sequoia was acquired by Cambior and in 2006 IAMGOLD and Cambior merged.

The Project is currently supplying approximately 7% to 8% of global consumption of niobium. Historical production numbers are provided in Table 6-1.

There was no production on the property prior to SOQUEM's discovery and the mine has seen uninterrupted operation since that time.

TABLE 6-1 NIOBEC HISTORIC PRODUCTION
IAMGOLD Corp. – Niobec Mine

Year	Tonnes	Nb_2O_5 (%)	Metallurgical Recovery (%)
1976	341,639	0.81	52.0
1977	546,255	0.69	66.8
1978	557,613	0.70	65.5
1979	578,232	0.67	65.1
1980	605,170	0.62	65.0
1981	711,763	0.59	67.3
1982	745,126	0.64	67.5
1983	443,155	0.65	62.3
1984	671,840	0.72	60.0
1985	767,688	0.69	60.6
1986	750,590	0.70	66.4
1987	630,851	0.70	63.0
1988	912,228	0.71	60.5
1989	800,775	0.70	62.8
1990	794,239	0.71	60.4
1991	804,778	0.70	60.2
1992	815,269	0.68	59.3
1993	812,190	0.70	59.9
1994	809,009	0.69	59.1
1995	801,726	0.72	57.9
1996	810,269	0.70	58.8
1997	832,001	0.68	57.8
1998	818,745	0.69	58.2
1999	818,017	0.71	58.3
2000	906,741	0.66	54.6
2001	1,103,390	0.71	58.4
2002	1,215,500	0.69	58.3
2003	1,286,156	0.70	54.7
2004	1,334,065	0.71	54.1
2005	1,449,102	0.66	56.7
2006	1,599,072	0.66	58.4
2007	1,618,332	0.65	60.7
2008	1,787,557	0.62	57.9
2009	1,754,947	0.61	58.2
2010	1,863,634	0.61	56.3
Total	**33,097,662**	**0.67**	**59.4**

Source: IAMGOLD, 2011

MINING METHODS

Open stoping has been the only mining method used since the Project started production. While this method is simple and cost effective, it has the disadvantage that the mineralized zones exceed the maximum widths allowed by rock mechanics and consequently, economic mineralization is left in place. This issue is exacerbated at depth where mineralized zones are wider.

Stopes are planned using available diamond drilling data and the average stope size is 61.0 m long by 24.4 m wide by 91.4 m high corresponding to the distance between production and development levels. Pillars of 24.4 m are left between stopes and may be extracted after primary stopes are mined. Occasionally, due to various factors, the secondary extraction of these pillars is impractical or impossible. Mining of the upper two blocks are nearing completion and, with depth, ground mechanics change.

Golder Associates Ltd. (Golder) completed a rock mechanics study in 2007 and recommended higher horizontal pillars and narrower stope dimensions. The wider zones of mineralization, combined with the restrictive mining factors, prompted a review of mining recovery and horizontal pillar recovery.

Also in 2007, Golder Paste Technology Ltd. (PasteTec) carried out a test on the Niobec mine tailings to determine their suitability for use as a cemented UG mine paste backfill. The study concluded that a blended waste stream consisting of 60% pyrochlore, 25% carbonate, and 15% cyclone overflow produced a promising result in the tested areas. Additional testing in 2008 included the introduction of a binder produced from finely ground slag, which was a waste product of the refinery process containing low levels of radiation. Studies concluded that there was no strength advantage to using the material as a cement substitute (Belzile, 2009).

Based on the paste backfill studies and simulations, the recommended size of a stope is 15.2 m by 24.4 m by 91.4 m together with a mining sequence that would allow enough time for curing. Golder has reviewed the results and is in agreement with the studies' conclusions. IAMGOLD's intentions are to use this method for the mining of Block 4 and deeper.

HISTORIC RESOURCE ESTIMATE

Past resource estimates were done for internal purposes and were not disclosed to the public. In 2009, a NI 43-101 compliant Mineral Resource estimate was prepared by IAMGOLD and audited by Belzile Solutions Inc. (BSI). Mineral Resources were estimated by mining block, using an Inverse Distance to the power two (ID^2) interpolation method, and were inclusive of Mineral Reserves. The IAMGOLD Mineral Resource estimate is shown in Table 6-2 and the BSI Mineral Resource estimate, as a check, is shown in Table 6-3.

TABLE 6-2 IAMGOLD 2009 MINERAL RESOURCE ESTIMATE
IAMGOLD Corp. – Niobec Mine

Category	Block No.	Tonnage (Kt)	Nb_2O_5 Grade (%)	Metal Recovery (%)	Yield (kg/t)
Measured	Block 1	1,405	0.51	57.53	2.91
	Block 2	3,203	0.56	57.60	3.20
	Block 3	5,694	0.60	58.73	3.54
	Block 4	1,065	0.64	57.06	3.63
Indicated	Block 4	11,747	0.59	59.92	3.55
	Block 5	385	0.57	56.48	3.24
Total Measured and Indicated		**23,500**	**0.59**	**58.99**	**3.46**
Inferred	Block 4	4,563	0.51	59.56	3.05
	Block 5	12,976	0.56	59.74	3.38
	Block 6	11,238	0.61	59.37	3.63
Total Inferred		**28,777**	**0.58**	**59.53**	**3.42**

Source: Belzile, 2009

TABLE 6-3 BSI 2009 MINERAL RESOURCE ESTIMATE
IAMGOLD Corp. – Niobec Mine

Category	Block No.	Tonnage (Kt)	Nb_2O_5 Grade (%)	Metal Recovery (%)	Yield (kg /t)
Measured	Block 1	1,405	0.50	57.63	2.88
	Block 2	3,194	0.55	57.78	3.18
	Block 3	5,681	0.60	58.98	3.56
	Block 4	1,065	0.64	57.06	3.64
Indicated	Block 4	11,747	0.59	59.93	3.55
	Block 5	385	0.57	56.45	3.23
Total Measured and Indicated		**23,477**	**0.59**	**59.08**	**3.46**
Inferred	Block 4	4,563	0.51	59.56	3.05
	Block 5	12,981	0.57	59.91	3.39
	Block 6	11,238	0.61	59.47	3.64
Total Inferred		**28,783**	**0.58**	**59.68**	**3.43**

Source: Belzile, 2009

BSI reported that resource classification followed "CIM Definition Standards for Mineral Resources and Reserves" (2005) and were classified according to the diamond drilling pattern, the proximity of stoping, and the availability of reconciliation data between models and production. The BSI estimate was not identical to IAMGOLD's, but BSI concluded that the differences were negligible and confirmed that the IAMGOLD Resource Estimate was reliable and repeatable (Belzile, 2009).

Measured Resources were limited to blocks where the diamond drilling pattern is 22.9 m vertically by 15.2 m, with an east-west direction, usually corresponding to areas where final definition drilling had been completed (i.e., mining blocks 1, 2, and 3).

Indicated Resources corresponded to blocks located in an area with a 45.7 m by 30.5 m drilling pattern. This corresponded to the first stage of definition, where exploration drilling had identified the continuity of mineralization (mining block 4).

Inferred Resources corresponded to estimated blocks in areas where exploration drilling had been completed on a 91.4 m by 91.4 m drilling pattern (i.e., mining blocks 5 and 6).

Three different search ellipses were used for grade interpolation for the three categories, depending on the density of information.

Mining factors were applied before the final resource estimation was produced. Stopes were designed and only the blocks inside the stopes were compiled for resource estimation. For mining blocks 1, 2 and 3, the Measured Resources can then be transferred directly into Proven Reserves as shown in Table 6-4.

For blocks 4 and 5, the stope maximum dimensions were 24.4 m by 24.4 m using the cemented paste backfill mining method. In the resource estimation, there was no associated dilution in the modelling of the stopes, although a 5% dilution at zero grade has been added to the resource estimate. As all the Indicated Resources of blocks 4, 5, and 6 were above economic cut-off, they were transferred into Probable Reserves (Belzile, 2009). RPA notes that no resources were estimated for mining blocks below Block 6 in 2009.

TABLE 6-4 IAMGOLD 2009 MINERAL RESERVE ESTIMATE
IAMGOLD Corp. – Niobec Mine

Category	Block No.	Tonnage (Kt)	Nb_2O_5 Grade (%)	Metal Recovery (%)	Yield (kg /t)
Proven	Block 1	1,405	0.51	57.53	2.91
	Block 2	3,203	0.56	57.60	3.2
	Block 3	5,694	0.6	58.73	3.54
	Block 4	1,065	0.64	57.06	3.63
Total Proven		**11,367**	**0.58**	**58.11**	**3.38**
Probable	Block 4	11,747	0.59	59.92	3.55
	Block 5	385	0.57	56.48	3.24
Total Probable		**12,133**	**0.59**	**59.81**	**3.54**
Total Proven + Probable		**23,500**	**0.59**	**58.99**	**3.46**

Source: Belzile, 2009

7 GEOLOGICAL SETTING

REGIONAL GEOLOGY

The Saguenay region is underlain by Grenville Province rocks of the Canadian Shield. It is characterized by high-grade metamorphic terranes and deep-level thrust stacks along ductile shear zones and are analogous to a Himalayan-style collisional orogen. The Grenville Province extends for more than 2,000 km and ranges from 300 km to 600 km wide. During the Grenvillian Orogeny (1.08 Ga to 0.98 Ga) extensive crustal thickening and tectonic extrusion lead to widespread high grade metamorphism. Large anorthosite massifs and coeval batholiths of margerite-charnockite-granite and, frequently, layered mafic intrusives mark periods of post-orogenic emplacement. Post-Grenvillian carbonatite intrusions like the Saint-Honoré and Sept-Îles are host to niobium, and ilmenite and apatite deposits, respectively (Corriveau et al., 2008).

The rocks of the Saguenay area were divided by Dimroth et al. (1981) into three distinct litho-structural units. Unit one constitutes a gneiss complex that is divided in three Groups based on increasing structural complexity from the youngest to the oldest. These are:

- Group I which comprises rocks that have been migmatized and deformed during the Hudsonian Orogeny (1.74 Ga).
- Group II which is composed of post-Hudsonian rocks they may have been emplaced during a periods of non-orogenic felsic magmatism.
- Group III rocks which comprised dykes of granite and amphibolite that generally parallel anorthosite contacts.

The second unit is comprised of anorthosite and charnockite-mangerite batholiths with well preserved igneous structures and textures. Anorthosite emplacement likely commenced during pre-Grenvillian crustal extension and continued through until post-Orogeny but the timing of the final intrusion is uncertain.

Unit three is characterized by calc-alkaline intrusions that cross-cut the host rocks and contains superior amphibolite facies mineralogy. Tectonic extension at the beginning of the Paleozoic incorporated normal faulting, updoming, and igneous alkaline activity, and resulted in the formation of the St. Lawrence River Rift system. The emplacement of this third unit included the intrusion of the SAC, which hosts the Niobec mine, around 650 Ma based on potassium-argon (K-Ar) dating (Belzile, 2009). The SAC is situated along the Saguenay Graben, a 250 km long and 25 km to 40 km wide structure that extends from the St. Lawrence River near Tadoussac to the Lac St.-Jean district (Figure 7-1). Geology in the vicinity of the SAC comprises anorthosite, syenites and magnetic diorite gneiss (Hatch, 2001).

Shales and limestones found in the vicinity of Saint-Honoré are thought to be the result of a marine transgression during the Ordovician period (about 470 Ma). Overlying the deposit are limestone and dolomite of the Trenton group. Up to 75 m thick, this unit is quarried in nearby Saint-Honoré (Belzile, 2009).

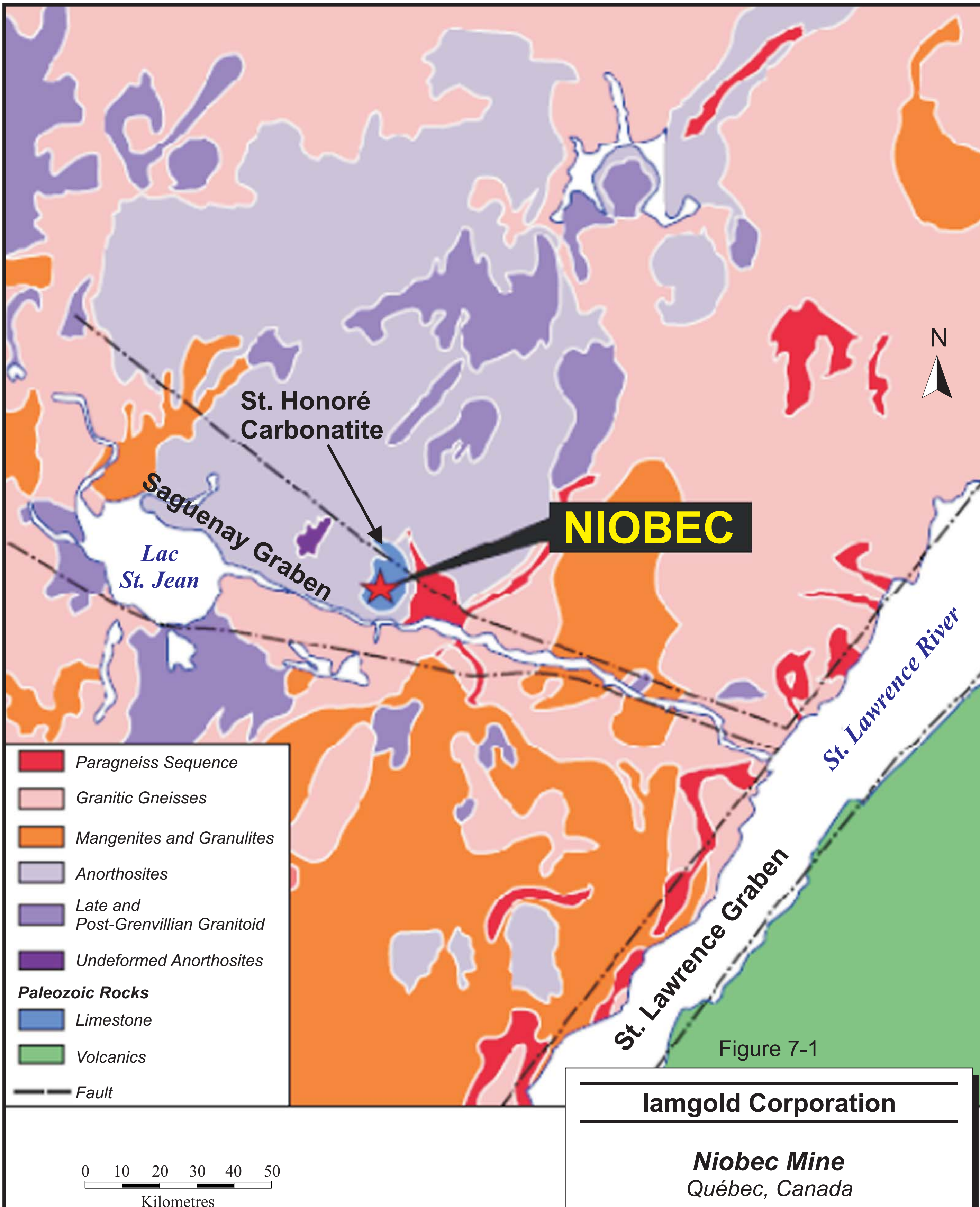


Figure 7-1

Iamgold Corporation

Niobec Mine
Québec, Canada

Regional Geology

June 2011

Source: Iamgold Corporation, 2011.

PROPERTY GEOLOGY

The SAC, the host to the Niobec Mine, is elliptical in plan view with a north-south trending major axial length of four kilometres and covers approximately 12 km^2 in area. It is comprised of a series of crescent shaped lenses whose age and composition vary with proximity to the core. The outer edge is composed of calcite that gets progressively younger inward through dolomite to ferro-carbonatite and is shown in Figure 7-2.

Three main lithologic units are present on the property. These are:

- An elliptical carbonatite core that includes:
  - An excentric core composed of brecciated to massive dolomite and ankerite contains up to 4.5% rare-earth elements (REE) such as cerium, lanthanum, and europium in fine-grained bastnaesite.
  - Ring dykes or cone sheets of barren to low-grade niobium and REE-bearing dolomite.
  - High-grade niobium (greater than 0.4% Nb_2O_5) dolomites and calcites in the southern portion of the core bordered with massive, red, altered dolomite.
  - A ring dyke of phlogopite calcite at the northern extremity of the core.
  - A belt of variable thickness composed of pyroxene calcite at the southern extent of the core.
- A circular outer ring containing feldspathic and feldspathoidal alkaline rocks.
- A triangular body of garnet syenite and cancrinite (a Na-Ca-Al-silicate and carbonate mineral) that lies at the extreme southeast part of the complex.

The two main niobium zones are subvertical and lenticular in shape and vary in lithology. Foliated and often brecciated dolomites and calcites alternate with more massive dolomites that contain red ankerite alteration. The foliated and brecciated unit is host to pyrochlore, the most prevalent niobium mineral, while the massive unit is less mineralized. The lenses measure up to 300 m in length, vary in width from 10 m to 80 m, cover a total area of approximately 600 m by 800 m, and are, generally, oriented concentrically around the core of the pluton. Minor northeast trending structures appear to have cut the lenses, post-mineralization. These mineralized zones are limited to the south by a ring dyke composed of massive fine-grained dolomite containing chlorite, pyrite and magnetite (Hatch, 2001).

The complex is surrounded by magnetite diorite gneisses as well as hypersthene syenite that have been subjected metasomatic alteration from the emplacement of the carbonatites (Belzile, 2009).

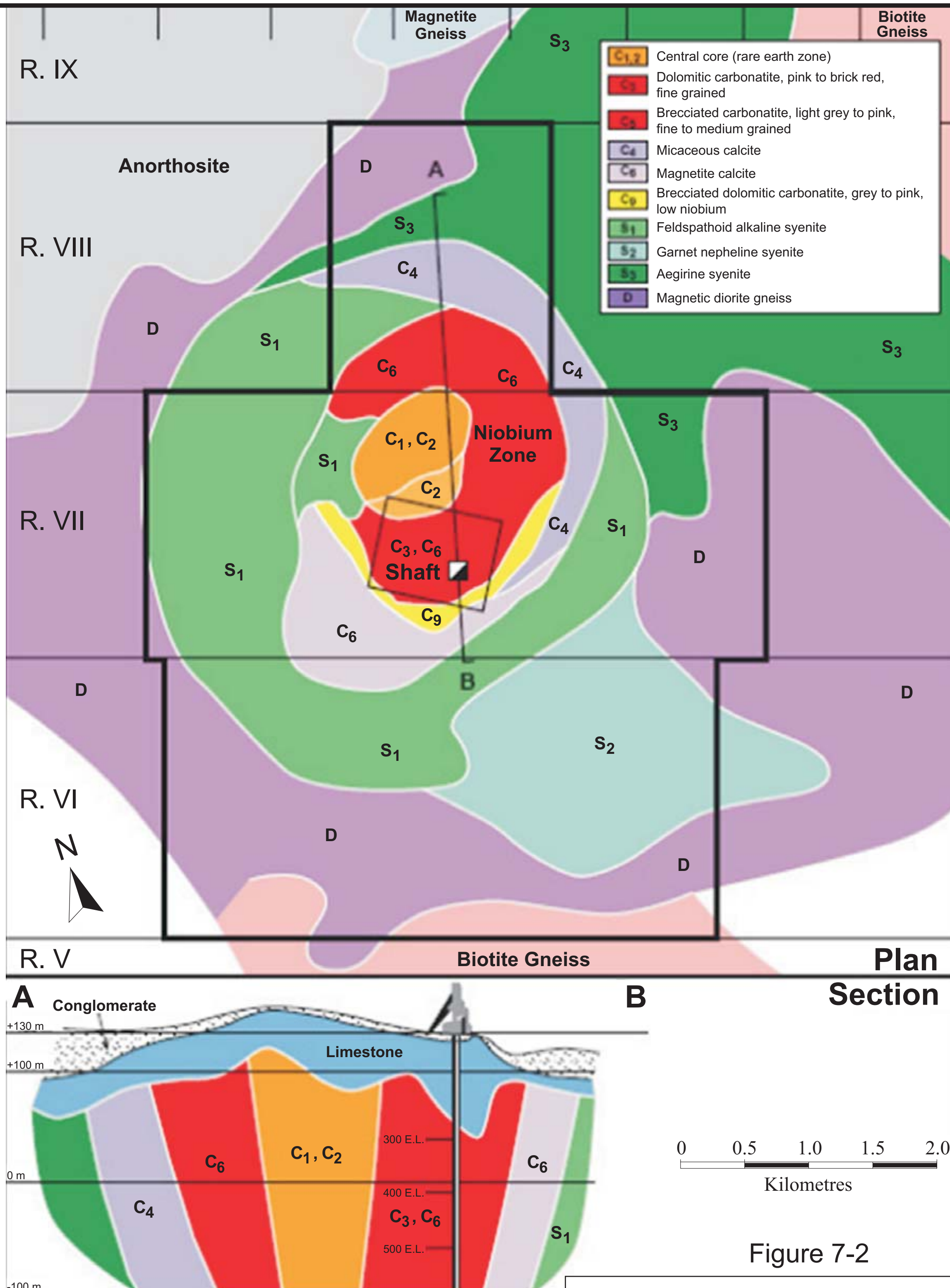


Figure 7-2

Iamgold Corporation

Niobec Mine

Québec, Canada

Property Geology

June 2011

Source: Iamgold Corporation, 2011.

8 DEPOSIT TYPES

A carbonatite is an igneous rock comprising at least fifty percent carbonate minerals. Globally, occurrences are sparse, typically small and widely distributed. Carbonatites usually occur as small plugs within zoned alkalic intrusive complexes, or as dykes, sills, breccias, and veins. Nearly all carbonatite occurrences are intrusives due to the unstable nature of carbonatite lava flows which react quickly in the atmosphere and have been poorly preserved. Carbonatites are almost exclusively associated with continental rift-related tectonic settings. However, they also occur in oceanic crust and have been recognized in compressional fold belts (Belzile, 2009). They are typically associated with under saturated igneous rocks that are nearly peralkaline but any rock type, including intrusives or metamorphic rocks can host these complexes (USGS, 2009).

Pyrochlore, bastnaesite, monazite, baddeleyite, and bornite mineralization are important components in some carbonatites because they carry niobium, REE, and other metals in concentrations high enough for economic extraction.

Carbonatites range in age from Archean to Recent (USGS, 1995). A summary of these is shown in Table 8-1.

TABLE 8-1 GLOBAL CARBONATITE DEPOSITS BY AGE RANGE
IAMGOLD Corp. – Niobec Mine

Age Range (Ma)		No. of Deposits	Global Percentage (%)	Summary Tonnage (Mt)	Percentage of Global Tonnage (%)
From	To				
10.4	100	15	24	4,831	37
101	265	16	26	2,553	22
344	520	6	10	1,295	11
558	680	11	19	1,310	11
1010	1400	7	12	1.038	9
1655	2047	3	5	978	9
no date		2	4	194	1
Total		60	100	11,749	100

Source: USGS, 2009

Constituent carbonate minerals, in order of decreasing abundance, include calcite, dolomite, ankerite, and rarely siderite and magnesite. Carbonatites are greatly enriched in niobium, REE, barium, strontium, phosphorus, and fluorine, and relatively depleted in silicon, aluminum, iron, magnesium, nickel, titanium, sodium, potassium, and chlorine. Sodium- and potassium-rich carbonate minerals have been confirmed at only one locality: the active volcano Oldoinyo Lengai in Tanzania. The extreme differences in mineralogy is thought to be due to the strong fractionation that occurs between the silica and oxide solid phases and the carbonate liquids while the carbonate liquids are separating from their source.

Typical non-carbonate minerals in carbonatites are apatite, magnetite, phlogopite or biotite, clinopyroxene, amphibole, monticellite, perovskite, and rarely olivine or melilite. Secondary minerals produced by alteration of primary magmatic minerals include barite, alkali feldspar, quartz, fluorite, hematite, rutile, pyrite, and chlorite.

Carbonatites yield a variety of mineral commodities, including phosphate, lime, niobium, REE, anatase, fluorite, and copper. Agricultural phosphate fertilizer is the single most valuable product derived from carbonatites; most is obtained from apatite concentrations that develop in lateritic soils during tropical weathering. Lime for agriculture and for cement manufacture is obtained from carbonatites in regions where limestones are lacking (Belzile, 2009).

9 MINERALIZATION

This section is derived from Belzile (2009).

Recognized minerals species at Niobec include carbonates (65%), oxides (magnetite, hematite) (12%), silicates (11%), apatite (10%), sulphides (1%), fluorite, barite and zircon (1% collectively).

Bi-pyramidal niobium minerals, primarily pyrochlore, are disseminated throughout the carbonatite and are generally associated with geological units rich in magnetite, biotite and apatite (typically units C3b, C3c and C3a). Niobium minerals are between 0.2 mm to 0.8 mm in diameter and rarely visible. Contacts are gradational and cannot be visually determined by mineralogy. Diamond drill core assays are the only way to delineate mineralization zones.

The mineralization is defined in terms of percentage of Nb_2O_5, with the mineralized lenses broadly delineated using a cut-off of 0.35% Nb_2O_5 on vertical sections spaced every 15.2 m. Lenses 101 and 102 are localized in the northern portion of the deposit. The mineralization in this area is characterized by hematite alteration that appears to reduce in intensity with depth. Lenses 206 and 208 occur in a more calcite-rich carbonatite with syenite xenoliths.

The economic mineralization is formed of ferrian- and sodic-pyrochlore. Metallurgical recovery, which can vary from 30% to 70%, is greatly influenced by the mineralogical characteristics of the rock type, alteration and type of mineralization.

Mineralized envelopes, oriented north-south, vary between 45.7 m and 182.9 m in width and extend up to 762.0 m in length. Dips are generally vertical or steeply dipping (greater than 70°) to the northwest or northeast. These zones have been defined vertically for at least 731.5 m and remain open at depth. Based on data from the deepest completed drill holes, the grades of mineralization at depth are equivalent to those currently being mined.

Average grades of the large mineralized envelopes are between 0.44% Nb_2O_5 and 0.51% Nb_2O_5. Mine workings are concentrated between the 91.4 m level and 442 m levels, operating simultaneously on three mining blocks. The majority of the Mineral Reserves and the Measured and Indicated Resources are located in the four first mining blocks between the 91.4 m level and 563.9 m level. The bulk of the Mineral Resources classified as Inferred are exclusively found in mining blocks 4 to 8, below the 563.9 m level.

10 EXPLORATION

Exploration at Niobec is concentrated within the carbonatite complex as there is no known occurrence of niobium outside of it. As the mineralization is not visible, exploration is carried out using diamond drilling and all the core is assayed for Nb_2O_5.

The focus of past exploration has been to extend the known mineralization to depth and increase the mine life. To date, the exploration has enjoyed annual success in delineating new Resources and Reserves year as detailed in Table 10-1. Mineralization remains open at depth and there is good future potential to increase resources.

REE EXPLORATION

Original exploration efforts in the area discovered a geological unit on the mine property that hosts an occurrence of REEs such as cerium, lanthanum, neodymium, praseodymium, samarium, dysprosium, and europium. Located approximately one kilometre north of the existing infrastructure it underwent preliminary evaluation in 1985 but was not pursued at the time due to unfavourable market conditions.

In 2010, IAMGOLD began to re-assess the previous work by conducting mineralogical studies on historic core specimen. Results were sufficiently encouraging that IAMGOLD planned a four hole surface diamond drilling program to confirm previous geological information and obtain fresh samples for testing.

At the time of RPA's site inspections in March 2011, the surface diamond drilling program was underway but no work was being conducted at the time. Results are pending and RPA cannot offer any opinion regarding adequacy.

TABLE 10-1 HISTORICAL RESERVES AT NIOBEC MINE
IAMGOLD Corp. – Niobec Mine

Year	Mt	% Nb_2O_5
1976	7.1	0.7
1977	6.3	0.69
1978	6.9	0.69
1979	7.0	0.65
1980	9.4	0.66
1981	11.9	0.66
1982	11.8	0.66
1983	11.4	0.66
1984	10.7	0.66
1985	10.9	0.66
1986	11.1	0.66
1987	10.9	0.66
1988	11.0	0.65
1989	10.8	0.66
1990	10.1	0.66
1991	10.2	0.66
1992	10.3	0.66
1993	9.9	0.66
1994	9.2	0.67
1995	8.4	0.67
1996	11.8	0.73
1997	11.4	0.73
1998	10.5	0.73
1999	10.2	0.73
2000	11.5	0.73
2001	18.1	0.68
2002	23.8	0.65
2003	22.6	0.65
2004	24.3	0.66
2005	21.5	0.67
2006	19.8	0.66
2007	16.4	0.62
2008	23.5	0.59
2009	32.1	0.57
2010	45.7	0.53

Source: IAMGOLD

Notes:

Reserves reported prior to February 1, 2001, pre-date NI 43-101 Standards of Disclosure for Mineral Projects.

11 DRILLING

Most of the drilling at the Project was done by independent contractor Forage Major Kennebec Ltée. (Major Kennebec) of Thetford Mines, Québec, using an air powered drill. In 2010, Major Kennebec was replaced by Boreal Drilling Ltd., another independent drilling contractor, of Val d'Or, Québec, which employs electric drills that offer more range, in terms of drill depth and angle, and the ability to drill BQ-size core.

Diamond drilling is a primary method of exploration at Niobec. Initially, holes are drilled on approximately 91.4 m by 91.4 m spacings that are reduced to 45.7 m by 30.5 m for zone interpretation as results warrant. For stope design and to increase confidence in the location of the mineralization, final drilling is done on 22.9 m by 15.2 m intervals. Most holes were drilled using AQ-size (27 mm dia.) core with some longer holes drilled using BQ-size (36.4 mm dia.) core. Due to the restrictions of the air-powered equipment, inclined holes could not exceed +45° in dip or 91.4 m in length and flat or downward dipping holes could not exceed 198.1 m in length.

Holes are designated based on order of planification and there is no distinction in numbering between exploration and definition drill holes. Hole numbers range from S-1 to S-3556 as of December 31, 2010. RPA notes that there are breaks in sequence of hole numbers so the total number of holes in the database is 3,525.

All holes are designed by the mine's geology department and information such as location, azimuth, dip and special instructions are noted on a drill hole plan. The holes, generally, are drilled on mine section with azimuths of 180° and 360°. Definition holes are used to outline stope blocks based on a designated cut-off that is informed by recovery and dilution inputs. These inputs are based on mining and metallurgical recoveries and measured against mine reconciliations.

Collar are located by the mine's survey department who also delineate the azimuth of the hole with front sights and back sights. The dip of the hole is determined by using a degree rule. Periodic checks are done by the survey department to ensure proper alignment of the drill and dip of the drill head.

Due to the small diameter of the AQ-size drill holes, dip deviation was checked using acid tests at 30.5 m or 45.7 m intervals. In 2010 the use of a Flexit Reflex EZ-AQ orientation tool was implemented to check both azimuth and dip deviations. RPA notes that magnetite is present in the rocks at Niobec and that drill hole orientation readings may not be accurate. IAMGOLD personnel also report that efforts are being made to acquire a gyroscopic orientation tool to ameliorate any magnetism issues that might affect readings. IAMGOLD personnel also report that drill hole intersections are located, where possible, in excavated stopes and the new information is used to update and adjust the drill hole database. In RPA's opinion, because of the density of drilling and the efforts made to correct any errors based on observation of intersections in stopes, any deviation in the drill hole orientation will have negligible impact on the estimation of Mineral Resources.

Once retrieved from the core barrel the core is placed in sequential order in core boxes labelled with the hole number. Each run, usually 3.05 m, is identified by a wood block on which the depth of the hole is marked. At the end of each shift, core boxes are transported to surface by the drill contractor via the mine shaft.

Some, but not all, drill holes are located upon completion by the survey department. RPA is of the opinion that all drill holes should be surveyed. All holes are sealed using an aluminum plug (P 187) and tests are performed to ensure the plugs are firmly set.

IAMGOLD reports that, despite the small core diameter recoveries are very good, exceeding 95%.

RPA inspected an UG drill station and found it to be orderly and well configured for its intended purpose. The drill was not active due to electrical issues but the equipment was found to be industry standard. RPA notes the above industry-standard safety modifications that have been applied to the equipment to protect its operators.

Drill hole planning is based on exploration and production history at Niobec. Past positive reconciliations provide confidence in the geological model and the diamond drilling, as the primary means of defining that model, have well established orientations that reflect the true width and orientation of the mineralization. A summary of all diamond drilling at Niobec to December 2010 is shown in Table 11-1.

RPA found the geological data at Niobec to well organized and competently managed. In the opinion of RPA, diamond drilling is planned and executed in manner that is consistent with industry standards and that information derived from these programs are adequate for use in an estimation of Mineral Resources.

TABLE 11-1 HISTORIC EXPLORATION AND DEFINITION DRILLING AT NIOBEC MINE TO 2010
IAMGOLD Corp. – Niobec Mine

Year	Total Length (ft.)	Total Length (m)
1976	41,059	12,515
1977	39,396	12,008
1978	57,310	17,468
1979	73,913	22,529
1980	85,283	25,994
1981	85,365	26,019
1982	78,317	23,871
1983	36,444	11,108
1984	57,880	17,642
1985	49,116	14,971
1986	58,209	17,742
1987	40,900	12,466
1988	42,226	12,870
1989	31,481	9,595
1990	36,550	11,140
1991	40,481	12,339
1992	38,780	11,820
1993	49,567	15,108
1994	35,280	10,753
1995	24,005	7,317
1996	16,404	5,000
1997	11,456	3,492
1998	39,884	12,157
1999	53,172	16,207
2000	52,012	15,853
2001	45,909	13,993
2002	51,156	15,592
2003	44,946	13,700
2004	32,471	9,897
2005	59,221	18,051

Year	Total Length (ft.)	Total Length (m)
2006	40,288	12,280
2007	40,315	12,288
2008	41,765	12,730
2009	54,859	16,721
2010	56,972	17,365
Total	**1,642,392**	**500,601**

12 SAMPLING METHOD AND APPROACH

DRILL CORE SAMPLING

Estimation of Mineral Resources at the Project are based exclusively on the diamond drill core sampling conducted by the mine's geology personnel.

The core is brought to surface, via the shaft, by the contract diamond drillers and deposited at the core shack. The core boxes are brought into the core shack, opened, and laid out on logging tables by IAMGOLD personnel. It is measured, for core recovery purposes, and then logged for Rock Quality Designation (RQD). RPA notes that RQD measurements were designed for NQ-size (47.6 mm dia.) core and are likely to be underestimated with the smaller diameter core. Notwithstanding its smaller diameter, the AQ-size core recovery at Niobec generally exceeds 80% and should improve with the move to BQ-size core.

Core is then logged by trained geological personnel for lithology, mineralogy, alteration and structure. Percentages of magnetite, hematite, biotite, apatite, pyrite, ankerite and fluorite are routinely noted. Niobium mineralization is very fine-grained and difficult to distinguish. No core photographs were taken prior to 2010 when that procedure has been implemented.

A nominal sample length of 3.05 m has been accepted as the norm. When judged to be geologically appropriate, the sample interval can be shortened. The logging geologist notes start and end of each sample interval and assigns a rock code based on the lithology and mineralogy of that interval. This rock code along with other factors, like the silica and iron contents, which are derived from assaying, influence the predicted metallurgical recovery. These factors, in turn, influence the Mineral Resource estimate.

These data are recorded in Gemcom GEMS Logger, off-the-shelf third-party software. Earlier exploration and definition drilling had these data recorded on paper and later entered into the Logger program. RPA has inspected the Niobec GEMS database and notes some drill holes are missing lithological data. These omissions were brought to the attention of Niobec geology staff and corrected.

Once the intervals have been established by the logging geologist, the whole core is sampled. Core is broken, as necessary, to make sampling more manageable but no reference core remains after selection. Each specimen is placed in a plastic bag along with the tag that bears the unique, assigned sample number. The samples are then delivered to the on-site laboratory by geology personnel. The empty core boxes are returned UG to be used again.

Historically, specific gravity (SG) measurements have been routinely taken at Niobec. In 2010 an additional 53 measurements were added to the dataset. The new data did not cause any change in the accepted values. The density of the rock varies by location within the deposit with Zones 101 and 102 having a value of 2.92 t/m^3 and Zones 206 and 208 have a value of 2.78 t/m^3. The historic data comprises readings from the upper part of the deposit and more SG data will be required for deeper mineralized zones.

RPA inspected the core logging facility and found it to be in good order and adequately configured for its intended purpose. It is, however, quite small and limited in its ability to handle large volumes of core. A new, larger, core logging facility has been designed and will be built to better accommodate increased core volumes. RPA notes that a pre-manufactured SG station, comprising an ultrasonic water bath and electronic balance, is located in the core facility.

RPA agrees that the approach used to plan, locate, log and sample drill holes is reasonable and, in RPA's opinion, gives an adequate representation of the mineralization.

GRADE CONTROL SAMPLING

Grade control at Niobec is based on muck samples. On average, there are six active headings per day. Muck sampling is not done by a geologist but by UG truck drivers at a rate of one sample per shift. Samples are delivered to the shaft and brought to surface by UG personnel. These samples are retrieved by laboratory staff and subject to the same analyses and drill core with results returned within 36 to 48 hours and used for production reconciliation.

Grade control samples were not used in the estimation of Mineral Resources at Niobec.

13 SAMPLE PREPARATION, ANALYSES AND SECURITY

The core is stored in secure wooden boxes and transported from UG to surface, by independent contract personnel, where possession is assumed by IAMGOLD personnel. The core is logged and whole-sampled in the core shack. Samples are then taken, by mine employees, to the on-site assay laboratory for analysis. The minesite is fenced and employs closed circuit video cameras and around the clock security personnel. In the opinion of the RPA, the premises are reasonably secure and the chain of custody protocols employed are adequate.

Sample preparation and assaying are conducted by IAMGOLD personnel at the on-site mine laboratory. The Niobec laboratory is certified ISO 9001 and all working procedures are written in detail. IAMGOLD reports that internal and external audits are regularly performed.

Samples are received, weighed and crushed until 90% of the sample passes -8 mesh (2.38 mm). Samples average approximately four kilograms of which two kilograms are retained and the remainder discarded. From the two kilogram sub-sample a 100 g split is taken and pulverized to 80% passing -325 mesh (44 µm) from which a one gram aliquot is taken for analysis. The remaining crushed sub-sample is retained for metallurgical testing and stored separately.

Prepared samples are dissolved in a borate flux at high temperatures which results in homogeneous bead. This bead is then analyzed using X-ray fluorescence (XRF) for Nb_2O_5, SiO_2, MgO, P_2O_5, Fe_2O_3, CaO, MnO, TiO_2, ZrO_2 and Al_2O_3. While Nb_2O_5 is the only component of economic interest, the other components are used to estimate potential metallurgical recovery and to assist in ore blending to optimize Project economics. These factors, especially metallurgical recovery, impact the estimation of Mineral Resources.

As part of the Niobec laboratory's internal quality control/quality assurance (QA/QC) protocols Certified Reference Materials (CRM) are assayed to confirm the precision and accuracy of determinations. Results are plotted on control charts and reviewed. Where discrepancies are noted actions are taken to ameliorate any issues. Calibration of equipment is done using CRM that have been tested at independent laboratories. External QA/QC protocols are reviewed in Section 13.

RPA inspected the Niobec laboratory and found it to be in good order, adequately configured for its intended use, and staffed by well-trained personnel. In RPA's opinion, sample preparation and analysis method are appropriate for the mineralization and the laboratory QA/QC procedures are adequate. RPA did not inspect the laboratories data management system. Assay results are kept electronically and reported to the Geology department in digital format. Hard copies of assay certificates are available upon request.

14 DATA VERIFICATION

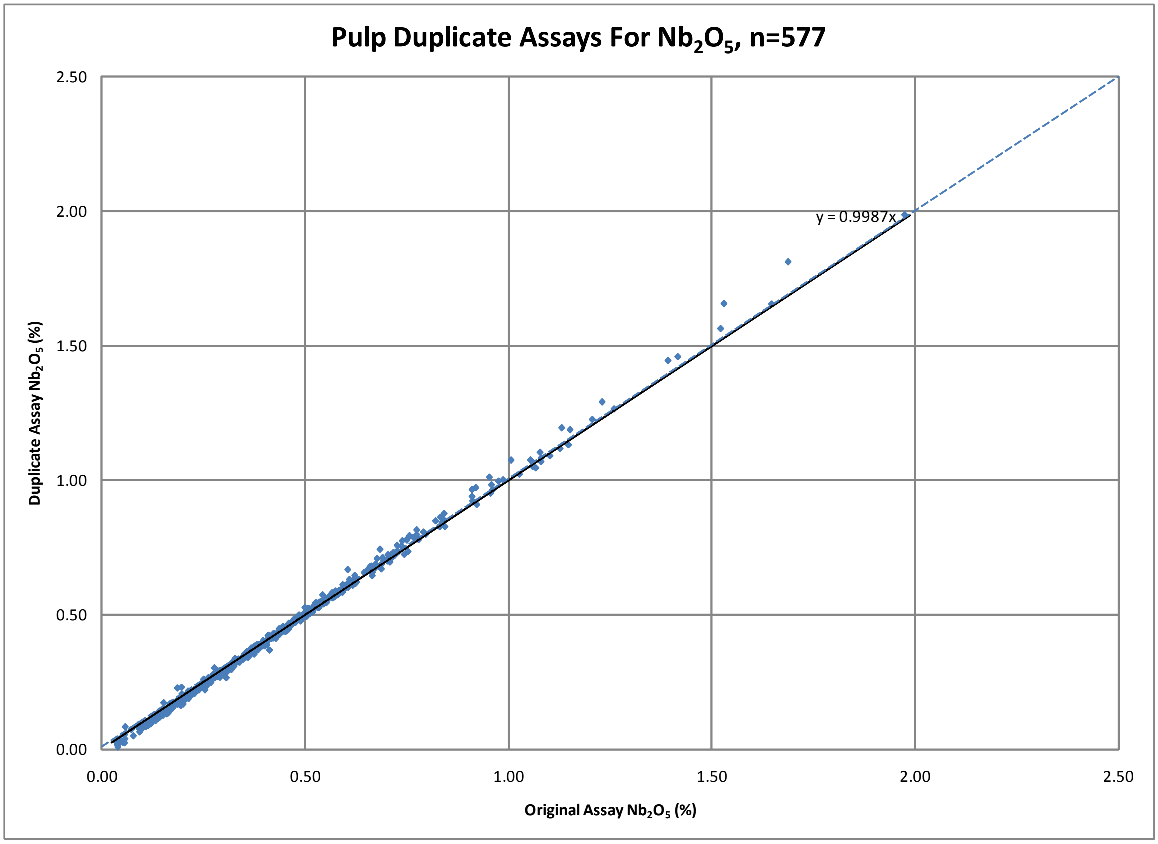
ASSAY QA/QC

The insertion of blanks and independent CRMs into the sample stream is not currently part of the QA/QC protocol at the Project. Since whole-core is sampled, there is no opportunity for core duplicate analyses. QA/QC procedures at Niobec consist of internal checks using Internal Reference Material and running duplicate analyses on second samples derived from both pulps and coarse rejects. These internal pulp and reject duplicate assay checks comprise approximately 20% (10% each) of the determinations conducted at the Niobec laboratory. Results are reviewed using statistical control charts and actions are taken when discrepancies are found in the data.

PULP DUPLICATE ASSAYS

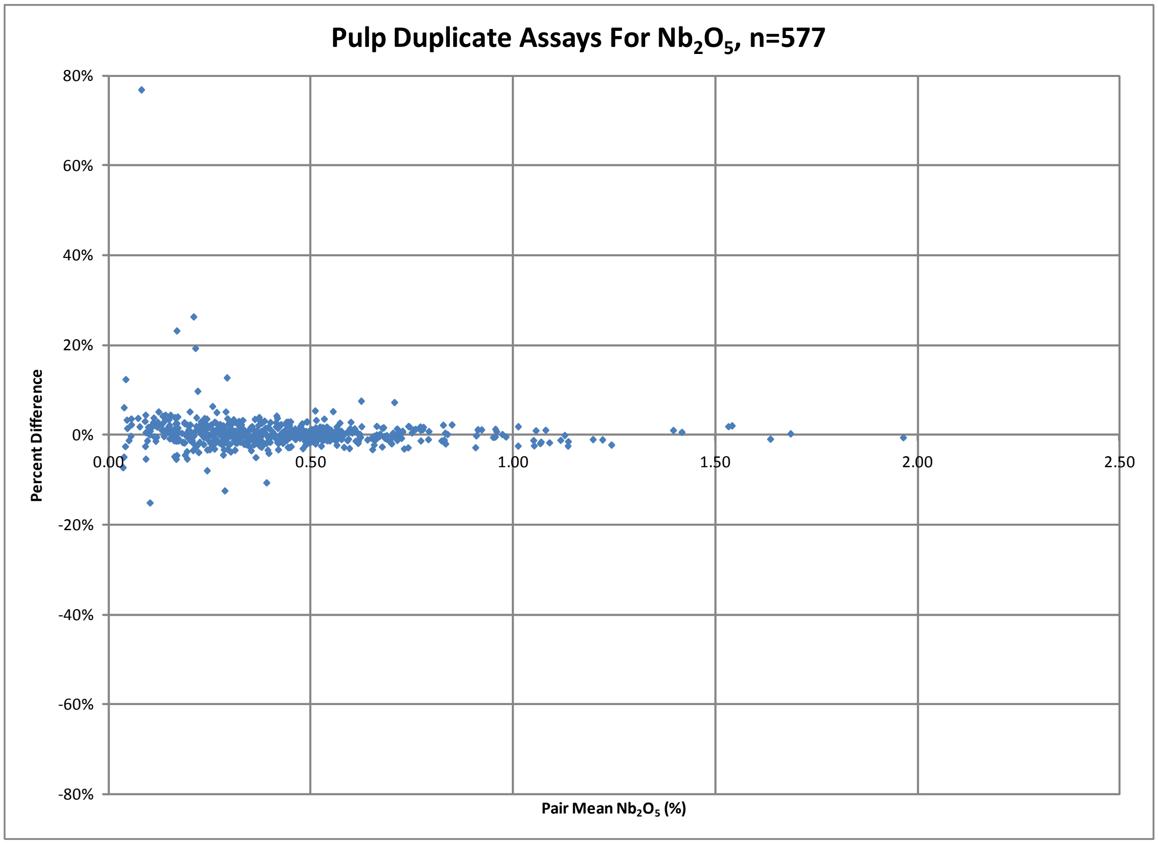
Every tenth submitted sample, i.e., those with a sample numbers ending in “0” (zero), has a duplicate analysis run from a second split from the original pulp. The original and duplicate determinations were plotted on control charts and reviewed. RPA has inspected the results for the 577 duplicate assays done for Nb_2O_5 and concludes that correlation between analyses is excellent. The graphical results are shown in Figure 14-1.

FIGURE 14-1 PULP DUPLICATE ASSAYS



RPA plotted these duplicate results on a relative difference (Thompson-Howarth) plot and inspected the data for indications of bias. No bias was observed. The results are presented in Figure 14-2.

FIGURE 14-2 RELATIVE DIFFERENCE (THOMPSON-HOWARTH) PLOTS FOR PULP DUPLICATE ASSAYS

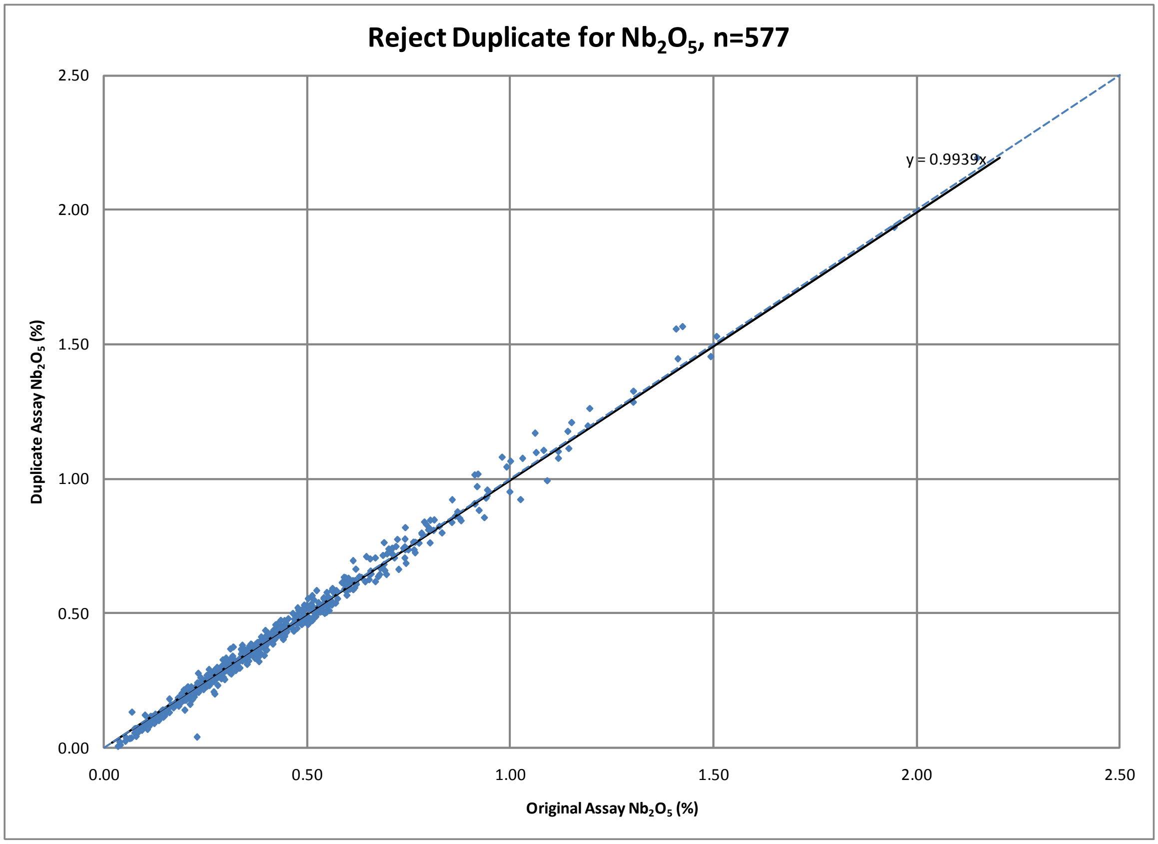


COARSE REJECT DUPLICATE ASSAYS

Every tenth submitted sample, i.e., those with a sample numbers ending in “5” (five), has a duplicate analysis run from a second split from the original coarse reject. These duplicates were taken to help assess a minor change to the sampling protocol that was initiated in 2010. The change resulted in samples terminating at geological boundaries rather that extending across them in an effort to normalize sample lengths to 3.05 m.

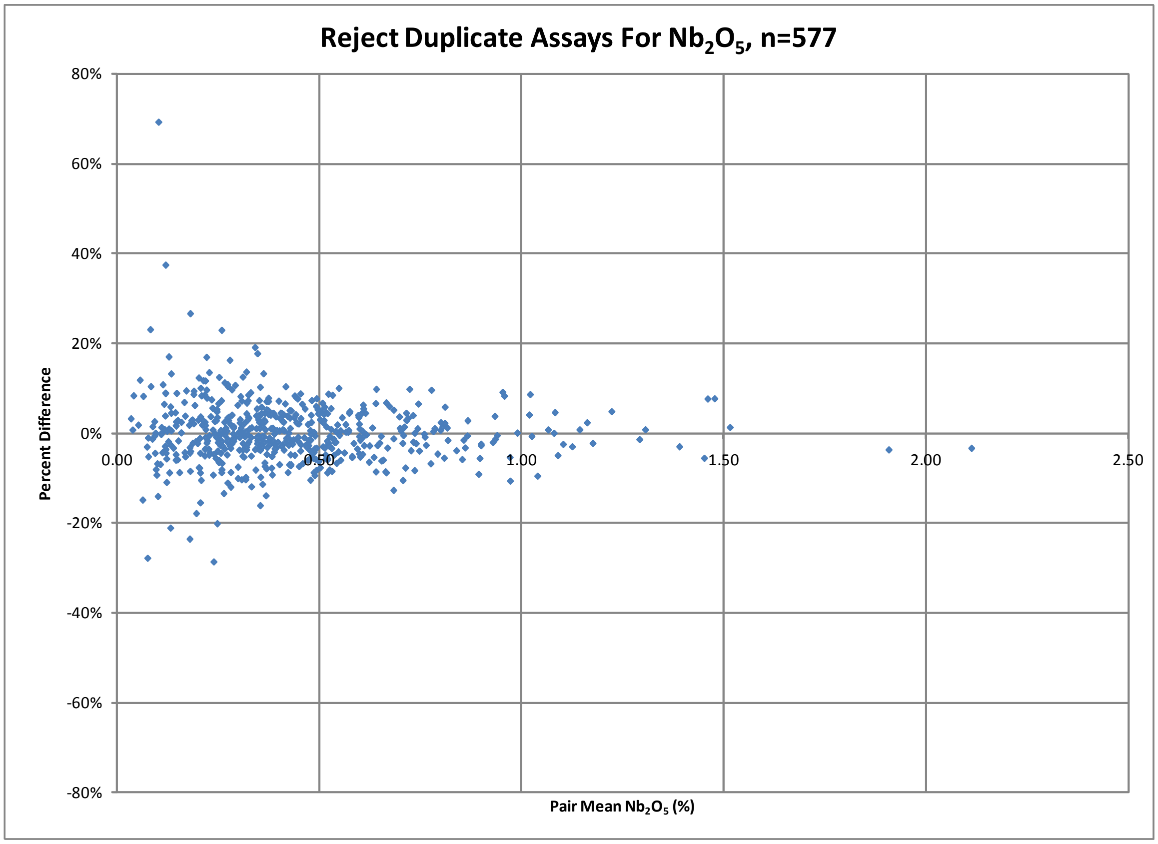
The original and duplicate determinations were plotted on control charts and reviewed. RPA has inspected the results for the 577 duplicate assays done for Nb_2O_5 and concludes that, as with the pulp duplicates, correlation between analyses is excellent. The graphical results are shown in Figure 14-3.

FIGURE 14-3 REJECT DUPLICATE ASSAYS



No bias was observed in the assay data when RPA plotted the duplicate results on a relative difference (Thompson-Howarth) plot. The results are presented in Figure 14-4 and are, predictably, more scattered than pulp duplicate assays.

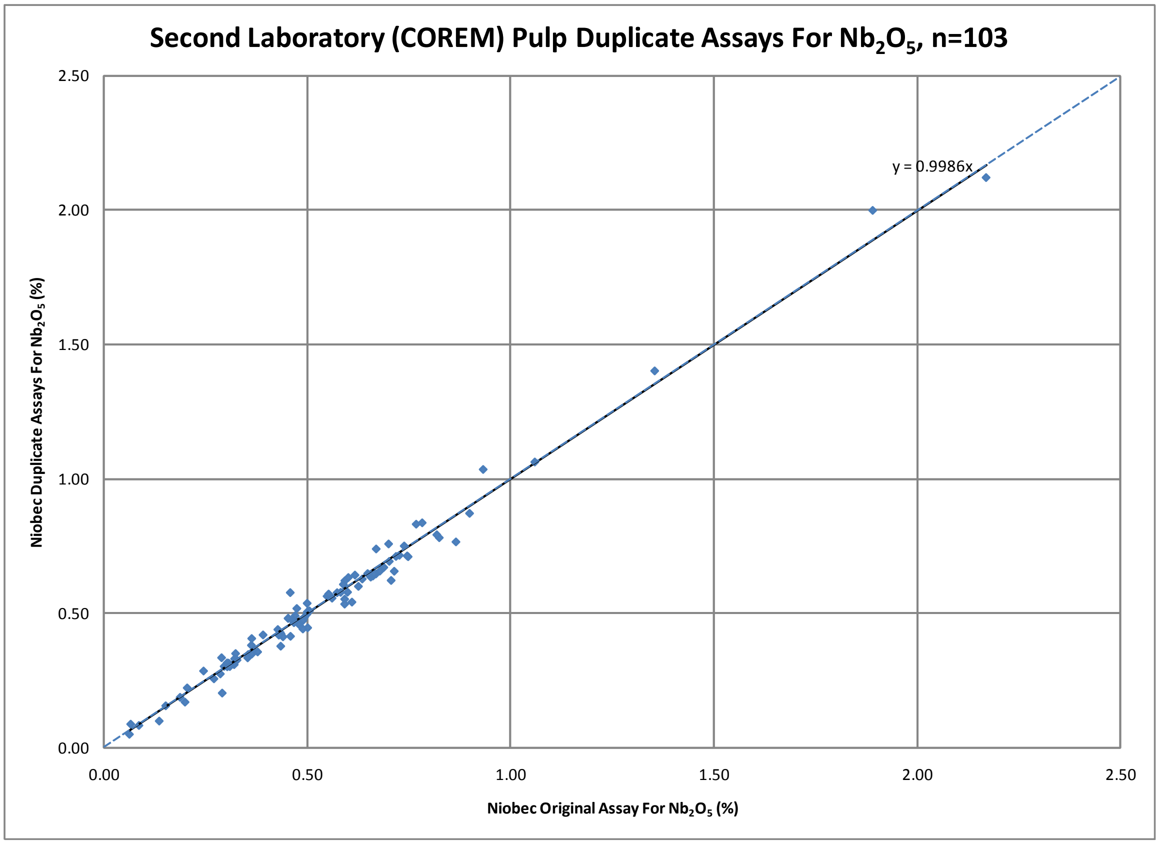
FIGURE 14-4 RELATIVE DIFFERENCE (THOMPSON-HOWARTH) PLOTS FOR REJECT DUPLICATE ASSAYS



INDEPENDENT SECOND LABORATORY PULP DUPLICATE ASSAYS

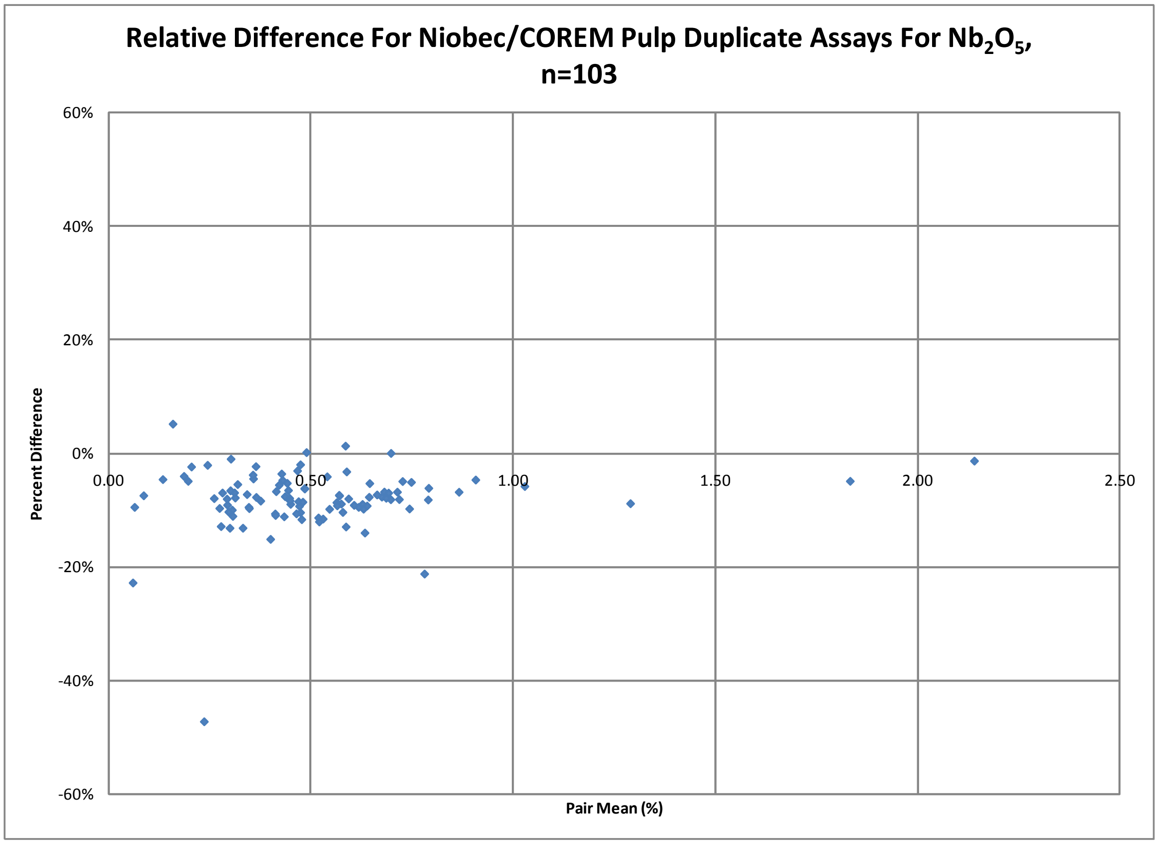
As a check of the Niobec laboratory, 103 pulp duplicates were shipped to COREM Laboratories (COREM) of Ste.-Foy, Québec, an ISO/IEC 17025:2005 accredited laboratory, for analyses. The results of the second analyses, again, showed excellent correlation with the original assays and are shown in Figure 14-5.

FIGURE 14-5 SECOND LABORATORY PULP DUPLICATE ASSAYS



When the relative differences between the two sets of results were plotted by RPA a mild bias was revealed. Original assay results from the Niobec laboratory return, generally, slightly higher grades than those from COREM. RPA notes that the differences are minor and will have no significant impact on the estimation of Mineral Resources. IAMGOLD should, however, investigate the cause of the bias. Results are shown in Figure 14-6.

FIGURE 14-6 RELATIVE DIFFERENCE (THOMPSON-HOWARTH) PLOTS FOR SECOND LABORATORY PULP DUPLICATE ASSAYS



Assay sample reanalysis shows excellent correlation between original and pulp duplicate, and reject duplicate, samples. RPA recommends, however, the introduction of blanks and independently derived CRM into the sample stream to further increase confidence in the assay data. RPA also recommends external laboratory checks on sample rejects in addition to sample pulps.

SAMPLE DATABASE

The data from the Niobec mine resides in a Gemcom GEMS v6.0.1 database *MODĖLE_DEC_2006* that is maintained by IAMGOLD personnel. RPA carried out a number of database validation and verification checks on both historic and new data. RPA notes that a significant part of the data pre-dates NI 43-101 Standards of Disclosure for Mineral Projects. The database contains 3,525 diamond drill holes with a total of 15,436 downhole and collar surveys and 164,177 drill sample records. .

VALIDATION

Gemcom GEMS software contains validation routines that were run on selected tables in the database. No discrepancies were noted in the data required for Mineral Resource estimation and only minor discrepancies were noted in other, less critical, data. These minor discrepancies were corrected prior to the Mineral Resource estimation.

VERIFICATION

The Niobec database contains records for 3,525 drill holes and comprises, primarily, collar, survey, lithologic, assay and RQD data. There is a well established history of production and positive reconciliations between the geological model and mill output at Niobec. In RPA's opinion, a program of spot checking the database is sufficient to establish its adequacy and appropriate for confirming its use in the estimation of Mineral Resources.

Collar, lithology and survey tables were checked against hard copy diamond drill logs and found to be error free. Both early holes, logged on paper, and later holes, logged electronically and printed out, were inspected to confirm that both forms of data input were adequate. RPA notes that drill holes prior to 2007, i.e., those logged on paper and later input manually, do not have associated dates in the in the database.

The assay database comprises 164,177 sample records as to December 31, 2010. RPA inspected this database and notes the following errors:

- A total of 71,307 samples in the database without sample numbers.
- A total of 319 assays with duplicate sample numbers.
- A total of 248 assay intervals in excess of 6.10 m, with 73 of those possessing non-zero values for Nb_2O_5.
- A total of 871 samples with zero value for Nb_2O_5.

Using electronic versions of assay certificates RPA imported 2,006 sample results and compared them against the database results for Nb_2O_5, SiO_2, P_2O_5 and Fe_2O_3 for a total of 8,024 determinations evaluated. One sample assay comprising four determinations was found to be erroneous with values in the database lower than those reported from the laboratory. RPA notes that a large number of samples in the database do not have sample numbers and, as such, could not be verified using assay certificates.

In the opinion of RPA minor errors will have little effect on the global grades and that the database is adequate for use in the estimation of Mineral Resources. RPA notes that the presence of zero grade values in the Nb_2O_5 field of the assay database will result in a more conservative estimation of Mineral Resources and recommends database corrections to improve accuracy.

INDEPENDENT SAMPLING

Since no drilling was taking place during the time of the site inspection, and since drill core is whole-sampled with no reference core retained, RPA was not able to take independent samples for verification. Given the long history of production at Niobec RPA does not consider the lack of independent sampling to be an issue.

UNDERGROUND INSPECTION

An UG inspection was conducted on March 22, 2011 by RPA personnel. Numerous mining levels were visited and open stope mining was observed to be in progress

RPA has conducted verification checks on assays, collars, downhole surveys in addition to validation checks on the database and inspection of QA/QC results. In RPA's opinion the database is reasonably sound and adequate for use in the estimation of Mineral Resources.

15 ADJACENT PROPERTIES

SHIPSHAW PROPERTY

Located seven kilometres from Niobec, DIOS Exploration Inc. (DIOS) has recently discovered a satellite carbonatite complex on its Shipshaw property. Drill testing has identified a circular low magnetic geophysical feature that is coincident with a topographic bedrock low below the overburden. This topographic low is possibly related to a fault zone associated with the Shipshaw River. The airborne geophysical survey that resulted in the discovery of the Niobec Mine did not cover the area underlying the Shipshaw property, but a later, federally sponsored, magnetic survey was done over the DIOS ground.

In June 2010, DIOS announced the results of assays from its drill program that returned grades of 0.053% Nb_2O_5 over 1.5 m with 12% P_2O_5 and 0.487% total REE oxides (excluding yttrium and zirconium) (TREO). Early drill results indicate that the syenite horizon was intersected by drilling within 20 m of surface and contained strontium and high apatite grades (DIOS, 2010a).

DIOS entered into a letter of offer from IAMGOLD to participate in a C$1.2 million private placement in common shares of DIOS at C$0.35 per share. As a consequence, IAMGOLD is granted an exclusive option to enter into an Option and JVA to earn 60% of DIOS' interest in the Shipshaw property within two years (DIOS, 2010b).

CREVIER PROJECT

Located approximately 70 km north-northwest of Saguenay, Québec, the niobium-tantalum mineralized Crevier property comprises 186 contiguous claims covering 10,416.5 ha in Crevier and Lagorce townships. Discovered by SOQUEM in 1975, the property was transferred to Cambior in 1986 as part of the privatization of its assets. IAMGOLD acquired the project in 2006 with its acquisition of Cambior and, in 2008, vended it to Les Minéreux Crevier Inc. (Crevier) for C$500,000 and 2 million shares of Crevier. In 2010 MDN Inc. (MDN) acquired part of Crevier and now controls 72.5% of the company, with IAMGOLD holding the remaining 27.5%.

The igneous alkaline complex, covering approximately 2,500 ha, is found within gneissic Grenville Province rocks along the Waswanipi-Saguenay corridor, a major structural lineament that also hosts the Ste.-Honoré carbonatite. The zone of niobium-tantalum mineralization is located in the southern part of the Crevier alkaline intrusive and is associated with a northwest striking, steeply northeast dipping, porphyritic nepheline syenite dyke. The dyke is approximately 3,000 m long with an average thickness of 20 m and hosts pyrochlore mineralization to a depth of 300 m.

In 2009, SGS Geostat Ltd. (SGS Geostat), using a 0.1% Nb_2O_5 cut-off grade, prepared a NI 43-101 compliant Mineral Resource estimate. SGS Geostat estimated Indicated Mineral Resources to be 25.8 Mt at 0.186% Nb_2O_5 and 199 ppm Ta_2O_5, and Inferred Mineral Resources at 16.9 Mt at 0.162% Nb_2O_5 and 204 ppm Ta_2O_5 (SGS, 2009). A preliminary economic assessment prepared by Met-Chem Canada Inc. of Montreal, Québec, recommended bulk mining the deposit at a rate of 4,000 tpd (Bureau, 2010).

RPA has been unable to verify the information, and that the information is not necessarily indicative of the mineralization on the property that is the subject of the technical report.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

MINERAL PROCESSING

The process utilized at Niobec was first developed from pilot plant programs. Over the years, the process has evolved with the mill operating team. The ROM ore is crushed to 100% passing 1 ½ in. and fed to rod mills, ball mills, and classification circuits where the ore is ground to 80% passing 180 µm. The ore is deslimed in two stages of cycloning, and the underflow is sent for conditioning prior to carbonate flotation. The carbonate concentrate is sent to the tails. The carbonate flotation rougher and cleaner tails are cycloned in two stages to change the process water and then sent to the pyrochlore rougher flotation. The rougher concentrate is sent to five stages of cleaner cells. This is followed by pyrite flotation to remove the sulphides leaching with hydrochloric acid to remove phosphorus and then followed by drying to produce a concentrate with less than 0.1% moisture before being converted into ferroniobium.

Since 2000, three major expansions have been completed to increase the ferroniobium production. The main goals of these expansions have been to maintain Niobec's worldwide market share and to reduce the ore cut-off ratio by having a higher throughput. They have also contributed to the delay of the development of the fourth block, which requires a significant investment and increase in the Mineral Reserve as unit operating costs decrease. Table 16-1 summarizes Niobec's production rate since 1998.

TABLE 16-1 NIOBEC MINE FeNb PRODUCTION (1998 TO 2010)
IAMGOLD Corp. - Niobec Mine

Year	Throughput (tph)	Tonnage (Mt)	Head grade (% Nb_2O_5)	Metallurgical Yield (kg/t)	Production (000's kg Nb_2O_5)	Production (000's kg FeNb)
1998	97.4	817,500	0.69	4.01	3,286	3,300
1999	96.3	818,017	0.71	4.12	3,372	3,464
2000	108.6	906,741	0.66	3.60	3,267	3,263
2001	134	1,103,390	0.71	4.12	4,550	4,548
2002	146.5	1,215,500	0.69	4.05	4,920	5,095
2003	153.5	286,156	0.70	3.81	4,905	5,071
2004	157.8	1,334,065	0.71	3.85	5,131	5,185
2005	173.5	1,449.10	0.66	3.75	5,436	5,674
2006	193.1	1,607,000	0.66	3.84	6,165	6,338
2007	195.5	1,618,330	0.65	3.92	6,337	6,516
2008	213	1,787,560	0.62	3.58	6,400	6,672
2009	210	754,946	0.61	3.55	6,230	6,231
2010	225	1,863,630	0.61	3.46	6,452	6,588

ORE CHARACTERISTICS

The unusual, heterogeneous mineralogy of the deposit makes milling and research a capital intensive undertaking. The deposit contains at least two dozen minerals but, at the present time, only two of them are of economic interest. These are pyrochlore $(Na,Ca)Nb_2O_6F$ and columbite $(Fe,Mn)(Nb,Ta)_2O_6$. They are unequally distributed throughout the carbonate ore. Typical mineral composition of the ore is shown in Table 16-2.

TABLE 16-2 MINERAL COMPOSITION
IAMGOLD Corp. - Niobec Mine

Group	Constituent	Weight Percentage
Carbonates	dolomite	65
	calcite	
	ankerite	
	siderite	
Sulphides	pyrite	0.9
	pyrrhotite	
Oxides	pyrochlore	1.1
	columbite	
	magnetite	
	hematite	1.7
	rutile	
	ilmenite	
Phosphates	apatite	6.8
Silicates	zircon	0.2
	chert	
	biotite	
	chlorite	21
	Na, K, feldspars	
	pyroxenes	
	nepheline	
Others (Minor)	barite	
	fluorite	
	hydrocarbons	
	halite	
	sphalerite	3.3
	parisite	
	bastnaesite	
	monazite	
	silica	

Pyrochlore itself does not have a rigid chemical composition and contains REE (tantalum, titanium, strontium and zirconium among others) in addition to niobium. Up to eight different varieties of pyrochlore can be found in the deposit. The most common variety is the sodium type, characterized by its sodium and calcium content as shown in Table 16-3. The progressive replacement of sodium and calcium by iron and other elements produced an iron enriched type of pyrochlore and columbite.

TABLE 16-3 CHEMICAL COMPOSITION OF MINERALS
IAMGOLD Corp. - Niobec Mine

	Na–Pyrochlore (%)	Fe–Enriched Pyrochlore (%)	Columbite (%)
Nb_2O_5	65.76	68.6	71.08
Ta_2O_5	0.11	N.A.	N.A.
TiO_2	2.75	3.46	3.74
ZrO_2	0.33	N.A.	N.A.
CaO	16.59	11.66	0.62
Na_2O	8.43	4.73	0.46
FeO	0.32	4.96	18.16
MnO	0.04	1.5	2.95
$La2O_3$	0.08	N.A.	N.A.
Nd_2O_3	0.16	N.A.	N.A.
Ce_2O_3	0.34	N.A.	N.A.
UO_2	0.01	N.A.	N.A.
ThO_2	0.23	N.A.	N.A.
TOTAL	**95.15**	**94.91**	**97.01**

The replacement is seldom complete and in fact, pyrochlore and columbite form a continuous series. The Fe-enriched pyrochlore and columbite are usually found in altered ore, but are present in unaltered ore where they are of primary origin.

The variable chemical composition has a major influence on mill production results. A portion of black Fe-enriched pyrochlore and columbite at a certain pH, have surface properties different from those of the sodium type of pyrochlore. These two minerals are in fact lost to tailings in the flotation process.

CURRENT PROCESS DESCRIPTION

The following sections contain a process description for the current processing plant that has a 260 tph capacity. A simplified flowsheet of this process is shown on Figure 16-1.

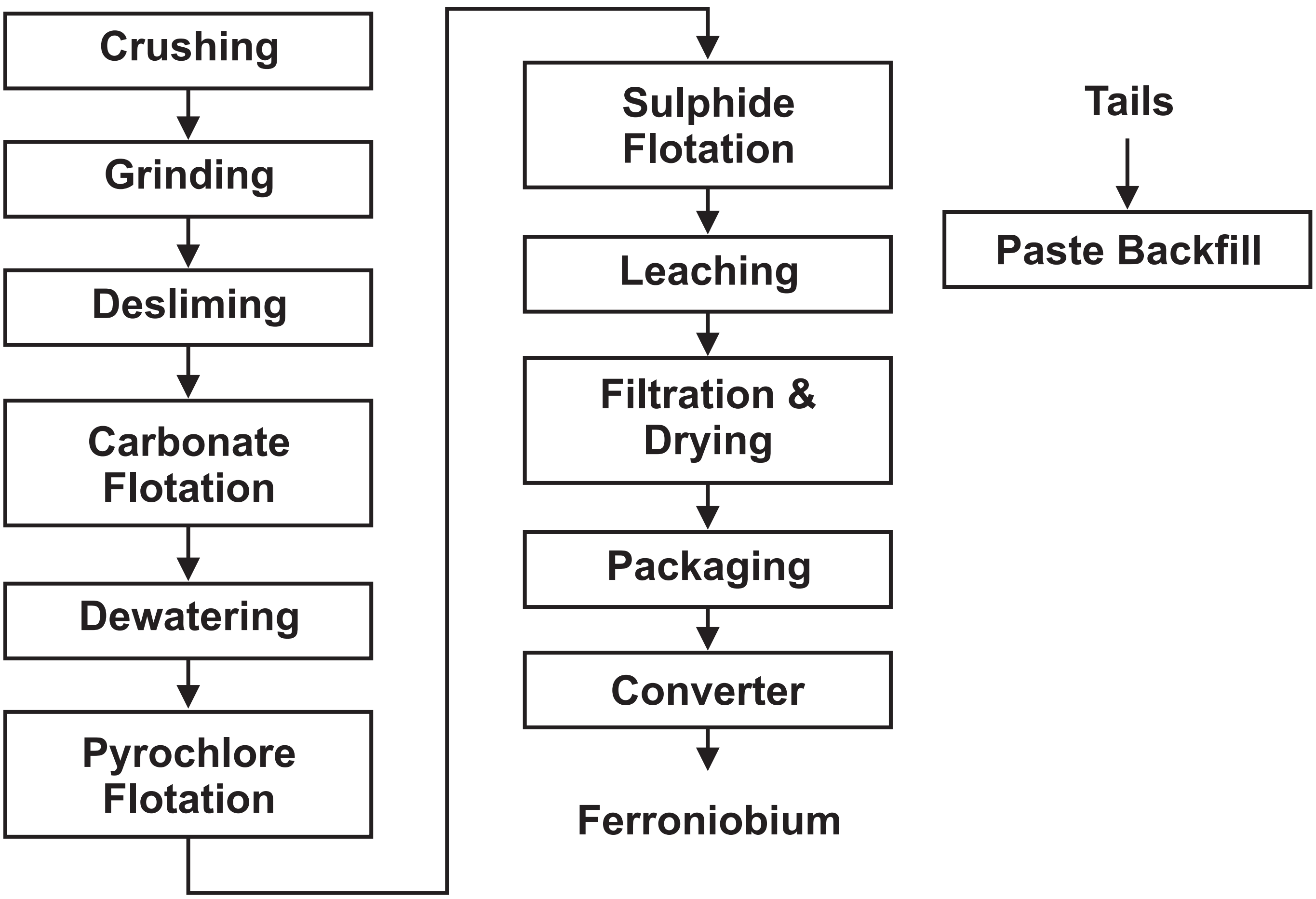


Figure 16-1

Iamgold Corporation

Niobec Mine
Québec, Canada

Simplified Process Flow Sheet

June 2011

Source: Iamgold Corporation, 2011.

CRUSHING

Primary crushing is done underground with a jaw type crusher before being hoisted to the head frame bin that has a capacity of 600 t. A vibrating feeder is located under the head frame bin in order to feed the conveyor before the secondary crusher. The secondary crusher is a gyratory type crusher and its product is stored in four 1,200 t insulated bins. From this point until the secondary desliming step there are two parallel lines. Via conveyors two of the insulated bins are directed to the first line and the two others are directed to the second line. Each bin has two vibrating feeders which are used to control the throughput measured on the belt-scale on the two lines. On each line, the ore is screened in order to send only the coarse product to tertiary crushing. Tertiary crushing consists of a cone crusher on each line. The fine product from the screen and the crushers discharge, which are 100% passing 1½ in., are then conveyed to the two grinding circuits.

GRINDING

As mentioned, there are two parallel grinding circuits. Each line is designed with the same equipment specifications. The tertiary crusher screen under size and tertiary crusher discharge are directly fed to both rod mills. The rod mill discharge on each line is combined with the ball mill discharges before each line is fed into a 24 way distributor via their mill discharge pump box pumps. Each distributor feeds a set of vibrating screens where the coarse reports to a screw classifier and the fines that are at 80% passing 180 microns are sent to the desliming circuit. Both screw classifier fines are sent to a trash screen before reporting to their respective mill discharge pump box. The coarse material from the screw classifiers is sent to the ball mill feed.

The softness of the ore, the relatively high specific gravity of the pyrochlore and the chemistry of the flotation process dictates the configuration of the grinding circuit in order to avoid over grinding of the pyrochlore crystals.

Mine water was initially used for make-up in the grinding circuit. It has a very high salts content of up to 6 g/l and has a deleterious effect on the collection of pyrochlore by amine. Therefore, only reclaimed water is now used in the grinding circuit.

DESLIMING

The desliming circuit is designed to remove the material that is smaller than seven µm before flotation. This is done in two steps of cyclone classification. The first step is done with two sets of parallel cyclones that are fed by the two parallel grinding lines where the slurry has been diluted to 22% solids. The overflow of the primary desliming cyclones at 5% to 7% solids are combined and fed to a second cyclone stage. Overflow from the two stages is sent to the tailings pump box. The cyclones achieve a very efficient sizing by removing nearly all of the –7 µm and nearly none of the plus 15 µm particles. Approximately 9% of the mill feed by weight and 5% of the pyrochlore are removed as slime.

PYRITE FLOTATION

A pyrite flotation is done following the desliming process. The slurry at 54% solids is first conditioned in two high intensity conditioners in series where copper sulphate and PAX are added. The pyrite flotation is done in two steps including a rougher and cleaner. The cleaning step is made up of 12 cells. The tailings are sent to the carbonates flotation circuit and the concentrate is sent to the tailings pump box. Approximately 0.2% of the mill feed by weight and 0.1% of the pyrochlore are removed in the pyrite flotation tailings.

CARBONATE FLOTATION

After the pyrite flotation, the slurry is conditioned with high intensity at 55% solids with an emulsified fatty acid collector. The pH stays close to its natural level at about eight. High intensity conditioning is done in two agitated tanks in series. The carbonate flotation has one rougher step and two cleaning steps. The two cleaning stages do not significantly reduce the total weight of material discarded, however, they do reduce the Nb_2O_5 content from 0.20% in the rougher concentrate to less than 0.10% in the final concentrate. Approximately 35% of the weight and 9% of the pyrochlore of the mill feed is floated off in the second cleaner concentrate. The flotation concentrate consists of very fine calcite particles (-50 µm) and medium size apatite. The carbonate flotation tailings are sent to the dewatering process.

DEWATERING

After carbonate flotation, the slurry is sent to cyclones in an arrangement similar to the desliming circuit for dewatering. The slurry is first diluted with fresh water to 30% solids. It is then pumped to primary dewatering cyclones. The overflow, at about 2%, solids is pumped to the secondary dewatering cyclones. The underflows from the two stages are combined and diluted with fresh water from 64% solids to 40% solids. The second stage overflow at less than 1% solids is sent to the tailings area. About 4% of the mill feed by weight and 7% of the pyrochlore are removed in the dewatering cyclones overflow. The total salt content is reduced about 3.5 times by the process.

MAGNETIC SEPARATION

The dewatering cyclone underflows are sent to magnetic separation. This process removes 2% of the weight and 1% of the pyrochlore.

PYROCHLORE FLOTATION

The non magnetic material is pumped to two high intensity conditioning tanks in series. The slurry is then sent to a bank of flotation cells where the pyrochlore collector is stage added. The rougher concentrate is sent to five stages of cleaning where the pH is gradually reduced to 2.7. Each stage, including the rougher, achieves a concentration ratio of about 1.9. The pyrochlore flotation concentrate is sent to sulphide flotation and the tails are sent to the tailings pump box. Approximately 60% of the pyrochlore is recovered in the concentrate in 0.8% of the weight while 20% of the pyrochlore is sent to the tails. As mentioned, pyrochlore is a sodium calcium fluoroniobate (Na Ca Nb_2O_5 F).

SULPHIDE FLOTATION

The cleaned pyrochlore concentrate contains about 20 wt% pyrite, which after the addition of sodium hydroxide is conditioned with silicate to depress the pyrochlore. PAX is then added and a rougher pyrite concentrate is floated. It is cleaned twice and then pumped back to the grinding circuit to assure that the pyrochlore loss from this flotation is well liberated. About 95% of the pyrite is removed at this stage.

PHOSPHATE LEACHING

The pyrite flotation tailings are sent to a 30 ft. thickener tank prior to leaching. The thickened slurry at 60% solids is then pumped to four leaching tanks where concentrated hydrochloric acid is added. The small amount of apatite present is rapidly dissolved.

LEACH FILTERING

The leached product is sent to a belt filter to remove most of the water. The solid cake is mixed with fresh water and copper sulphate for the second sulphide flotation.

SECOND SULPHIDE FLOTATION

The leftover activated sulphides are floated in conventional type cells. The pH is adjusted to 11 with NaOH and PAX is added as a collector. This concentrate is sent back to the first pyrite flotation circuit.

DRYING

The final pyrochlore concentrate is pumped at about 40% solids to a double 4 in disks filter and sent to a propane counter current dryer where the moisture level is reduced to less than 0.1%. The dryers are equipped with a cyclone for dust collection.

PACKAGING

The dried product is stored in twelve bins. The concentrate is packed into big bags by an automated packing-handling system then transferred to the converter.

CONVERTER

After the pyrochlore concentrate is dried, it is transferred to the converter where the material is transformed into ferroniobium (standard grade). The niobium oxide is converted into FeNb by using an aluminothermic reaction on a batch basis.

The raw materials used in the reaction, apart from the pyrochlore concentrate, include aluminum flakes and chops, a source of iron oxide hematite or mill scale, quick lime (to lower the fusion point of the slag) and sodium nitrate (chemical agent to increase the available energy). Each element in the reaction is calculated to obtain the desired quantity of niobium contained in the FeNb. The total batch size contains a total of 6,000 kg of raw materials to produce 2,200 kg of ferroniobium.

After the raw material is well mixed and added to the reaction vessel (a vertical cylinder), the reactor is placed under a hook and the reaction is started after the fuse, made of barium peroxide, and aluminum powder is ignited. The conversion process serves two basics functions: the reduction of niobium pentoxide (pyrochlore) and iron oxide to ferroniobium metal and the separation of impurity elements from the pyrochlore. The reaction is between the aluminum and the oxide materials. The reaction is strongly exothermic and once it is ignited it cannot be stopped. The aluminothermic reaction is summarized as follow:

$$6\ NaNo_3 + 10\ Al \rightarrow 5\ Al_2O_3 + 3\ Na_2O + 3N_2 + \text{heat} \quad (1)$$

$$Fe_2O_3 + 2\ Al \rightarrow Al_2O_3 + 2\ Fe + \text{heat} \quad (2)$$

$$3\ Nb_2O_5 + 10\ Al \rightarrow 5\ Al_2O_3 + 6Nb + \text{heat} \quad (3)$$

The iron will combine with the niobium to form ferroniobium metal and the alumina will be collected as slag.

It only takes five minutes to liquefy all material at a temperature of 2,200^{o}C. During the following seven to eight minutes, separation of the alloy and slag occurs. After cooling, the slag is brought back underground while the ferroniobium ingot is sampled and stored. The metal ingots are selected and withdrawn from storage according to their chemical analysis and customer's requirements. They are then crushed and fed to a classification section where the ferroniobium is packed to suit the customer's product size specifications.

PROCESS SELECTION

SELECTION BASIS

The remainder of this section will focus on the open pit expansion option (10 Mtpa production rate) and will summarize information and data concerning the existing operations (current and mining expansion underway prior to this PEA) and the block caving expansion option (also 10 Mtpa production rate) driven by IAMGOLD, for comparison purposes.

A completely new processing plant facility will need to be constructed for either the open pit or block caving scenario. New approaches will need to be adopted with respect to the mill throughput and equipment sizes. Conventional crushing and grinding using rod mill and ball mill will be replaced by Semi autogenous grinding (SAG) mill and ball mill. The ore will be crushed at surface and a primary crushing plant will need to be installed. Inside the grinding circuit, the screw classifier will be replaced by conventional cyclones to minimize the circulating load on the classification screens. Attention will be given to the cyclone design to minimize fine particle recirculation to the mill. The design of the recovery process will be based on scale adjustments from the existing facilities in the majority of the cases where it will be possible. As the current converter will fall inside the open pit foot print, it will be relocated near the new processing plant facility and the capacity will be doubled.

TECHNOLOGY AND RISK ISSUES

Some major issues and risks will be faced, principally due to the huge increase in equipment size and the lower mill feed grade. Those issues will be minimized by the utilization of well know equipment and trials done during the last few years. The utilization of the SAG mill and ball mill with conventional cyclones are considered one of the major risks in terms of niobium recovery due to the potential increase in fine particle production and losses at the desliming stage. The grinding circuit will be simpler to operate and easier to perform automatic control strategies. The niobium recovery process will stay the same in term of metallurgy. In terms of mill feed grade reduction, the risk will come from the increase of the concentration ratio requested to achieve the concentrate grade and the control of the concentrate quality.

The commissioning of an entirely new plant will need to be well planned and will pose the highest risk. Sufficient time allocated for the commissioning will be requested to assure a good transition and to continue to produce ferroniobium to respond to client needs. The commissioning will be more of a transitional and physical replacement. The last 35 years of ore processing experience will help to minimize and accelerate the commissioning time.

DESIGN CRITERIA

For the benefits of the reader, Table 16-4 summarizes the design criteria of all scenarios, including those related to the underground block caving and open pit options.

TABLE 16-4 PROCESS DESIGN CRITERIA
IAMGOLD Corp. - Niobec Mine

Item	Units	Base case	Expansion	New plant UG	New plant OP	Source
Mill average throughput	t/d		9,589	27,397	27,397	calc
Annual tonnage	t/a	2,186,496	3,500,000	10,000,000	10,000,000	DV
Process plant design capacity	t/a	2,186,496	3,574,000	10,183,500	10,183,500	Calc.
Design processing rate	t/h	260	425	1,250	1,250	PP/site
Processing plant availability	%	96	96	93	93	site
Nb_2O_5	%	0.56	0.56	0.42	0.47	RS/site
Nb_2O_5 recovery	%	56	56	47	51	site
Yield	Kg Nb_2O_5/t	3.35	3.35	1.96	2.40	site
Nb_2O_5 produced	Kg/year	7,370,000	11,725,000	19,200,000	24,000,000	Calc.
FeNb recovery	%	97	97	97	97	site
Nb produced	Kg/year	5,000,000	8,000,000	13,200,000	16,800,000	Calc
Fusion by days	Nb	12	14	16 or 11	14	Calc
Ferroniobium plant operation	Hrs/day	12	16	16 or 12	16	PP/site
Ferroniobium plant operation	Days/week	5	7	5 or 7	7	PP/site
Primary crusher	Type	Jaw	Jaw	Jaw	Gyratory	PP
Crusher product size	µm	150,000	150,000	150,000	150,000	PP
Bond ball mill work index	metric	10.0	10.0	10.0	10.0	Site
Secondary crusher	µm	35,000	35,000	None	None	PP
Nominal product size	µm	180	180	180	180	Site
Nominal grinding energy	kWh/t	12	12	12	12	Calc
Desliming size	µm	7	7	7	7	Site
Installed power						
Primary mill	Type	Rod mill	Rod mill	SAG	SAG	PP/SB
Secondary mill	Type	Ball mill	Ball mill	Ball mill	Ball mill	PP/SB

PLANT MODIFICATIONS

ORE HANDLING, CRUSHING AND STORAGE

The ore will be delivered to a gyratory crusher before it is sent to the covered stockpile. The ore from the current underground mine will be redirected to the new ore stacker.

GRINDING AND CLASSIFICATION

The grinding circuit will be modified in terms of technology. Considering the mill throughput, conventional SAG mill and ball mill circuit with cyclone classification will replace the existing rod mill and ball mill technology. The ore classification will continue to be done using vibrating screen technology; this approach will simplify the process flowsheet drastically in terms of operation and maintenance. Otherwise, the hypothesis at this stage is that if the additional niobium losses occur in the fines it will be offset by the lower operating costs and the better control of the grinding stage. Grinding test work will be requested for the SAG mill and grinding circuit design. The grinding circuit will have a SAG mill with two ball mills in a closed circuit. A grinding screen at the discharge of the SAG mill will collect the pebbles at the discharge and those pebbles will be re-circulated to the SAG mill feed. The intent is to maintain low residence time inside the mills and minimize the production of fines. The cyclones used will be designed to minimize the final product size at the cyclone underflow. Overflow from the cyclones will be screened using 180 microns five decks vibrating screens. The screen oversize product will be recirculated to the ball mill. The underflow will go to the desliming stage

DESLIMING

The same type and size of equipment used in the present desliming circuit will be used. The primary cyclone will be 10 inches followed by two inches cyclone size. Four parallel clusters of D10 will be used, each followed by four clusters of D2. Experience and importance of the desliming size drives the decision to stay with same equipment size.

PYRITE FLOTATION

A pyrite circuit will be built based on the current design. Two conditioner tanks will provide the capacity for the reagents prior to the flotation stage. Tank cells in series will be used for the sulphur flotation.

CARBONATE FLOTATION

Two parallel lines of tank cells, each including a conditioner for reagent conditioning, will be used. The first cleaner will consist in eight tank cells and the second of five tank cells.

DEWATERING (WATER CHANGE)

The dewatering will be done using two clusters of D10 cyclones followed by eight cluster of D4 and finally two clusters of D2 cyclones. The underflow from those cyclones will be cleaned by magnetic separation to remove magnetic minerals before being sent to the pyrochlore separation.

PYROCHLORE FLOTATION

The rougher circuit will be done in 12 tank cells in series. The reagent conditioning will be done in a highly agitated tank. The first cleaner will be done using 10 x tank cells in series followed by 8 x tank cells for the second cleaning stage. The third cleaner will be done in 12 x 5 m^3 cells in series followed by 10 x OK3R flotation cells in series for the fourth cleaner stage and 10 x OK 1.5R for the fifth cleaner stage. The concentrate will be cleaned using three magnet separators before being sent to the sulphide flotation.

SULPHIDE FLOTATION

The final sulphide flotation will be done using twelve flotation cells.

PHOSPHATE LEACHING

The pyrochlore concentrate from the sulphide flotation will be sent to a 25 m diameter thickener tank before being sent to four leach tanks in series. The concentrate will be filtered using six belt filters in parallel. The filtered concentrate will be re-pulped and send to the second sulphide flotation step for residual sulphide minerals removal.

SECOND SULPHIDE FLOTATION

The removal of final sulphide minerals will be done using eight flotation cells drying.

PACKAGING

The current packaging system built in 2010 appears to have the capacity to support the additional production by adding 12 additional concentrate storage bins. The enhanced production will be managed by increasing the packaging schedule.

CONVERTER

The design capacity will be twice the existing converter capacity.

TAILINGS PUMPS

Two tailings lines will be installed, one to provide the coarse material for the tailings dam construction and the second for the disposal of the carbonate and slimes inside the pond.

MILL WATER SUPPLIES AND DISPOSAL

To provide the additional water demand to the mill, the new system that will be built at the Shipshaw River will need to quadruple the current water intake. The capacity of the return line for the discharge water will be increased. The recirculating pump at the recirculation pond will need to be replaced. The existing one will be relocated to the mill pond number three in a new pumping station. The main mill water line will be replaced.

REAGENT CONSUMPTIONS AND SUPPLIES

A study performed on the reagents used in the process revealed that at the time of the study no issues were raised on the supplies. Each individual supplier has been contacted.

METALLURGICAL TESTING

ADDITIONAL TESTWORK

Metallurgical testwork will need to be done in order to properly design the new processing facilities. Crushing and principally SAG index will need to be done for proper equipment design and selection. Because of the lower head grade than the historical value and the sensitivity of the niobium concentrate quality to impurities, extensive metallurgical testwork will need to be performed to validate the niobium recovery and concentrate quality grade. Also, niobium recovery improvement will be investigated by modifying the reagent scheme or process flowsheet. Following the laboratory tests and results achieved, pilot plant investigation could be required for the new approach. Comparative testwork between conventional grinding and classification presently used and new grinding circuit using SAG-Ball mill will need to be done.

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

MINERAL RESOURCES

SUMMARY

As part of the PEA on the economic viability of open pit bulk mining at the Project, RPA conducted an independent update of Mineral Resource estimation, constrained in a Whittle open pit shell. Grades for Nb_2O_5, SiO_2, P_2O_5, and Fe_2O_3 were estimated into blocks using ID^2 weighting. Table 17-1 presents Mineral Resource estimates constrained by Whittle open pit shell at a 0.2% Nb_2O_5 cut-off grade.

TABLE 17-1 RPA INDEPENDENT UPDATED MINERAL RESOURCES – APRIL 1, 2011
IAMGOLD Corp. – Niobec Mine

Classification	Tonnes (000's)	Grade (% Nb_2O_5)	Contained Nb_2O_5 (M kg)
Measured	288,930	0.43	1,242
Indicated	169,180	0.40	685
Measured and Indicated	**458,110**	**0.42**	**1,927**
Inferred	336,445	0.37	1,240

Notes:
1. CIM definitions were followed for Mineral Resources.
2. The Qualified Person for this Mineral Resource estimate is Bernard Salmon, ing.
3. Mineral Resources are estimated at a cut-off grade of 0.20% Nb_2O_5.
4. Mineral Resources are estimated using an average long-term niobium price of US$42 per kg and a US$/C$ exchange rate of 1:1.05.
5. Mineral Resources are constrained by a pit shell to the 2400 level (725 m below surface).
6. Numbers may not add due to rounding.

IAMGOLD MINERAL RESOURCE AND RESERVE ESTIMATION

Historically, stope design was integral to resource estimation at Niobec. In the upper three mining blocks, open stopes are limited to a width of approximately 24.4 m. Where mineralized zones exceed 24 m, resources were limited to the volume of the designed stopes. The same methods were applied to the pillars that were not scheduled for recovery. Consequently, historical resource estimates had Measured and Indicated

Mineral Resources identical to Proven and Probable Mineral Reserves. Stope design was also considered in the estimation of Inferred Resources. These restrictions did not apply to the lower mining blocks (4 to 9) that employed paste backfill. All potential stopes, designed at 24.4 m by 24.4 m by 91.4 m or 15.2 m by 24.4 m by 91.4 m, within the mineralized zones could now be included in the resources provided the average grade of the volume, based on block model estimation, exceeded the economic cut-off.

The adoption of bulk mining methods will remove mining size restrictions, result in less ore loss, and allow for a lower economic cut-off grade, which reflects mining efficiencies realized through this extraction method.

Historically, IAMGOLD estimated Mineral Resources at the Project but did not publicly disclose the results. In 2009, IAMGOLD produced a Mineral Resource estimate that was audited and confirmed by BSI. A later internal IAMGOLD estimate was done as of December 31, 2010, employing updated data and is shown in Table 17-2.

2009 IAMGOLD RESOURCE ESTIMATE

In 2009, BSI completed an audit of the IAMGOLD resource estimate prepared using a block model constrained by wireframe models of the four mineralized zones, surface topography, and the overlying Trenton limestone. Grades were interpolated into blocks using ID^2. Mineral Resources were quoted inclusive of Mineral Reserves and are shown in Table 6-3 of this report. The BSI estimate showed negligible differences with the IAMGOLD resource estimate, which employed the same ID^2 interpolation methodology.

BSI concluded that the IAMGOLD estimate was reliable and repeatable (Belzile, 2009).

2010 IAMGOLD MINERAL RESOURCE ESTIMATE

Currently, the Project is mined using longhole open stopes with cemented backfill. As part of its annual evaluation, IAMGOLD produced estimates of Mineral Resources and Mineral Reserves using updated data from 2010 and employing similar parameters and assumptions used in previous estimates. The results of the IAMGOLD Mineral Resource estimation are presented in Table 17-2.

TABLE 17-2 IAMGOLD 2010 MINERAL RESOURCE ESTIMATE
IAMGOLD Corp. – Niobec Mine

Category	Block No.	Tonnage (Kt)	Nb_2O_5 Grade (%)	Metal Recovery (%)	Yield (kg /t)
Measured	Block 1	891	0.50	57.1	2.87
	Block 2	1,854	0.54	58.5	3.17
	Block 3	3,503	0.59	59.0	3.51
	Block 4	2,592	0.55	54.1	2.97
	Block 5	1,675	0.51	52.4	2.67
	Block 6	6,056	0.53	54.3	2.86
Total Measured		**16,571**	**0.54**	**55.7**	**3.03**
Indicated	Block 4	23,107	0.53	56.9	3.03
	Block 5	3,123	0.50	55.6	2.78
	Block 6	2,915	0.52	57.1	2.98
Total Indicated		**29,145**	**0.53**	**56.8**	**3.00**
Total Measured and Indicated		**45,716**	**0.53**	**56.4**	**3.01**
Inferred	Block 4	9,740	0.48	54.5	2.60
	Block 5	24,753	0.52	56.3	2.94
	Block 6	16,463	0.54	56.5	3.07
	Block 7,8,9	8,715	0.59	55.6	3.26
Total Inferred		**59,672**	**0.53**	**56.0**	**2.97**

Source: IAMGOLD, 2011

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated by IAMGOLD.
3. Mineral Resources are estimated at a cut-off grade of 0.45% Nb_2O_5 for Measured and Indicated Resources and 0.40% Nb_2O_5 for Inferred Resources.
4. Mineral Resources are estimated using an average long-term niobium price of US$25 per kg for Measured and Indicated and C$37.50 per kg for Inferred Resources, and a US$/C$ exchange rate of 1:1.15 for Measured and Indicated Resources and a US$/C$ exchange rate of 1:1.05 for Inferred Resources.
5. Mineral Resources extend 1,306 m below surface.
6. Indicated Mineral Resources are inclusive of Mineral Reserves.
7. Numbers may not add due to rounding.

RPA independently verified the 2010 Mineral Resource estimates by using the same interpolation parameters. In RPA's opinion, IAMGOLD's estimation of Mineral Resources is reliable and repeatable.

2010 IAMGOLD MINERAL RESERVE ESTIMATE

Mineral Reserves are limited to mining blocks 1, 2, 3, 4, 5, and 6. Since mining factors are applied before Mineral Resources are estimated, Measured Resources can be transferred directly into Proven Reserves for blocks 1, 2, and 3.

For blocks 4 to 6, the stopes are designed using the cemented backfill mining method at a maximum stope size of 15.2 m by 24.4 m. A lack of operation experience with the backfill mining method led to an assumption that some of the mineralized material would be diluted by waste despite the generally good ground conditions. Therefore, a total of 5% dilution has been added to the estimate at zero grade. Despite the added dilution, all Indicated Resources in blocks 4, 5, and 6 remain above the economic cut-off and, as such, they are transferred to Probable Reserves.

The 2010 Mineral Reserve estimate is summarized in Table 17-3.

TABLE 17-3 IAMGOLD 2010 MINERAL RESERVE ESTIMATE
IAMGOLD Corp. – Niobec Mine

Category	Block No.	Tonnage (Kt)	Nb_2O_5 Grade (%)	Nb_2O_5 Tonnes (000)	Metal Recovery (%)	Yield (kg /t)
Proven	Block 1	891	0.50	4.5	57.1	2.87
	Block 2	1,854	0.54	10.1	58.5	3.17
	Block 3	3,503	0.59	20.8	59.0	3.51
	Block 4	2,592	0.55	14.2	54.1	2.97
	Block 5	1,675	0.51	8.5	52.4	2.67
	Block 6	6,056	0.53	31.9	54.3	2.86
Total Proven		**16,571**	**0.54**	**90.0**	**55.7**	**3.03**
Probable	Block 4	23,107	0.53	123.0	56.9	3.03
	Block 5	3,123	0.50	15.6	55.6	2.78
	Block 6	2,915	0.52	15.2	57.1	2.98
Total Probable		**29,145**	**0.53**	**153.8**	**56.8**	**3.00**
Total Proven and Probable		**45,716**	**0.53**	**243.8**	**56.4**	**3.01**

Source: IAMGOLD

Notes:
1. CIM definitions were followed for Mineral Resources.
2. These Mineral Reserves were estimated by IAMGOLD.
3. Mineral Resources are estimated at a cut-off grade of 0.45% Nb_2O_5 for Measured and Indicated Resources and 0.40% Nb_2O_5 for Inferred Resources.
4. Mineral Resources are estimated using an average long-term niobium price of US$25 per kg for Measured and Indicated and C$37.50 per kg for Inferred Resources, and a US$/C$ exchange rate of 1:1.15 for Measured and Indicated Resources and a US$/C$ exchange rate of 1:1.05 for Inferred Resources.
5. Mineral Resources approximately 1,000 m below surface.
6. Indicated Mineral Resources are inclusive of Mineral Reserves.
7. Numbers may not add due to rounding.

RPA MINERAL RESOURCE ESTIMATION DATABASE

SAMPLES

The data used for grade interpolations are entirely from diamond drill sampling. The database contains records for 3,525 holes, of which all are located within the resource model area. The number of sampled intervals varies depending on the element. Key components such as Nb_2O_5, SiO_2, P_2O_5, and Fe_2O_3 were assayed in virtually all samples. Determinations for elements other than Nb_2O_5, the key economic component, were done to understand their impacts on niobium recovery and, ultimately, on yield.

The total number of samples within the database is 165,049, with 871 (0.5%) containing a value of zero for Nb_2O_5. Other elements also contain zero assay values to varying degrees. A table of sample statistics used in the estimate is provided below in Table 17-4. RPA notes that the use of samples containing zero assay values in interpolation produces a conservative resource estimate.

TABLE 17-4 STATISTICAL SUMMARY OF SAMPLE ASSAY DATA (NON-ZERO)
IAMGOLD Corp. – Niobec Mine

Element	No. of Samples	Min	Max	Mean	Std Dev
Nb_2O_5 (%)	164,177	0.01	9.70	0.47	0.33
SiO_2 (%)	140,016	0.10	75.00	6.89	9.32
P_2O_5 (%)	139,995	0.01	52.35	2.95	1.92
Fe_2O_3 (%)	140,181	0.10	116.32	7.32	4.94
MnO (%)	16,536	0.10	3.58	0.82	0.33
TiO2 (%)	25,729	0.01	1.80	0.35	0.24
CaO (%)	4,154	1.00	21.80	13.26	3.69

COMPOSITES

Mineral Resource estimates at Niobec are done on raw assay data since the majority of the sample intervals in the database are 3.05 m in length. RPA has inspected the database and confirms that approximately 76.5% of the samples are exactly 3.05 m long.

RPA notes that approximately 3.5% of samples in the database are equal or less than 1.52 m and approximately 1.3% are equal or greater than 4.57 m. Approximately 10.8%

of samples exceeding 4.57 m have zero values for Nb_2O_5, which likely reflects unsampled drill holes that are assigned zero grade for their entire length. For samples with lengths less than or equal to 1.52 m, approximately 3.4% of those values are equal to zero.

In the opinion of RPA, the assay database should be composited for those samples that lie within interpreted mineralized zones as a means of achieving uniform sample support before interpolation.

ESTIMATION DOMAINS

In 2007, lenses of similar characteristics were identified and regrouped into four main zones (Zones 101, 102, 206 and 208) using an approximate modelling cut-off grade of 0.35% Nb_2O_5 (Figures 17-1 and 17-2). This was done to assist in the estimation of Mineral Resources in the lower blocks that were scheduled to be mined and filled using paste backfill. At the upper levels of the mine, since mining was restricted due to lack of backfill support, the mineralized envelopes were modelled to capture the highest grade material (Belzile, 2009).

The mineralized zones extend 500 m to 800 m along strike and approximately 1,000 m at depth. The zones extend to approximately 1,000 m in depth.

RPA used IAMGOLD's mineralization solids for Mineral Resource estimation.

FIGURE 17-1 MINERALIZED ZONES – ESTIMATION DOMAINS

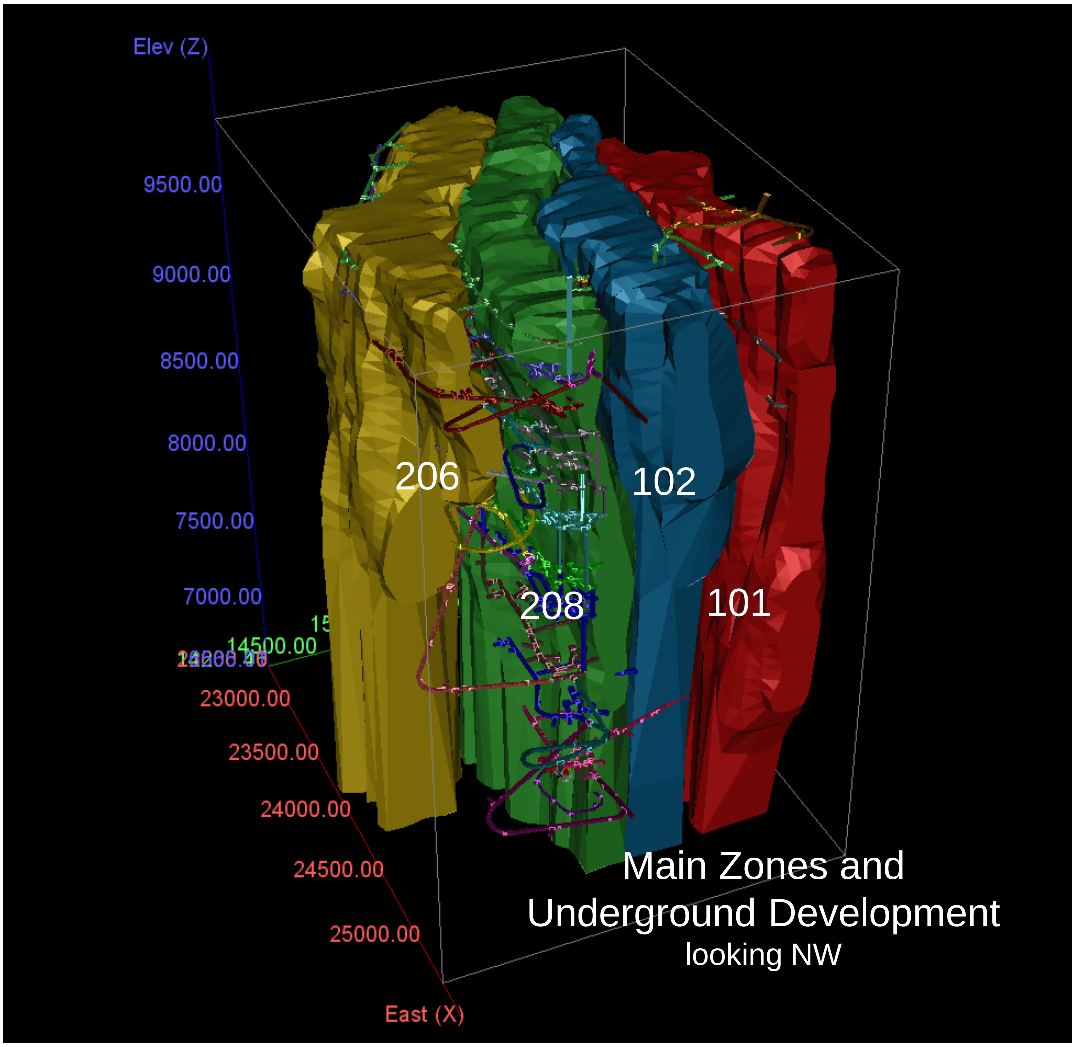
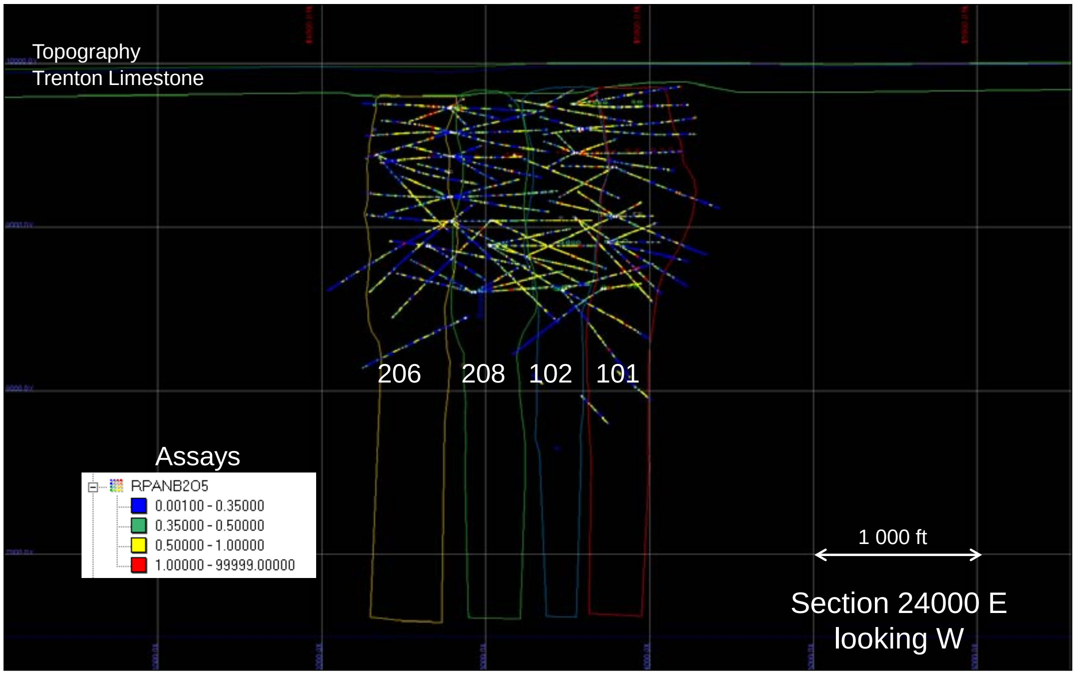


FIGURE 17-2 MINERALIZED ZONES – DIAMOND DRILLING DATA



In addition to the mineralized zones, the surface topography, the interpreted base of the overburden, and Trenton limestone were modelled. Surface and mineralized domain wireframes were updated using the latest drill results. RPA has inspected these solids and judged the interpretation to be reasonable and suitable for use in the estimation of Mineral Resources.

Domain coding of the block model (i.e., rock type model) was based on these wireframe constraints and is presented below in Table 17-5.

TABLE 17-5 BLOCK MODEL CODING
IAMGOLD Corp. – Niobec Mine

Domain Type	Solid or Surface Name	Description	Block Model Code
Surface	Topography_surface_meuble	Topographic Surface - Air	-
Surface	Topography_surface_terrain	Base of Overburden	-
Surface	Pilier_calcaire_block1	Base of Trenton Limestone	-
Envelope	101_GEN_19-06-2008	Mineralized Domain	101
	102_GEN_19-06-2008	Mineralized Domain	102
	206_GEN_04-12-2008	Mineralized Domain	206
	208_GEN_04-12-2008	Mineralized Domain	208
	CARBONAT_50	Surrounding Domain	50
	TRENTON_50	Unmineralized Domain	20
	MINED OUT		60

Descriptive and distribution statistics of the assay results were generated and grouped by mineralized domain and are shown in Table 17-6. The low Coefficient of Variation (CV) values derived from the analyses indicate that high grade values contribute only moderately to the mean grades. Using this data, IAMGOLD has, historically, not used top cuts to mitigate the effects of high grade Nb_2O_5 assays on the overall mean assay grade.

TABLE 17-6 SUMMARY STATISTICS OF ORIGINAL (NON-ZERO) ASSAYS BY MINERALIZED ZONE

IAMGOLD Corp. – Niobec Mine

Zone	Count	Mean Length (m)	Min. Grade (% Nb_2O_5)	Max. Grade (% Nb_2O_5)	Mean Grade (% Nb_2O_5)	Standard Deviation	Variance	CV[1]
101	20,337	2.96	0.01	3.21	0.49	0.29	0.08	0.59
102	32,154	2.96	0.01	7.70	0.51	0.35	0.12	0.68
206	43,940	2.94	0.01	9.70	0.44	0.34	0.11	0.77
208	41,827	2.94	0.01	8.91	0.47	0.31	0.10	0.67

[1] Coefficient of Variation

CAPPING OF HIGH GRADES

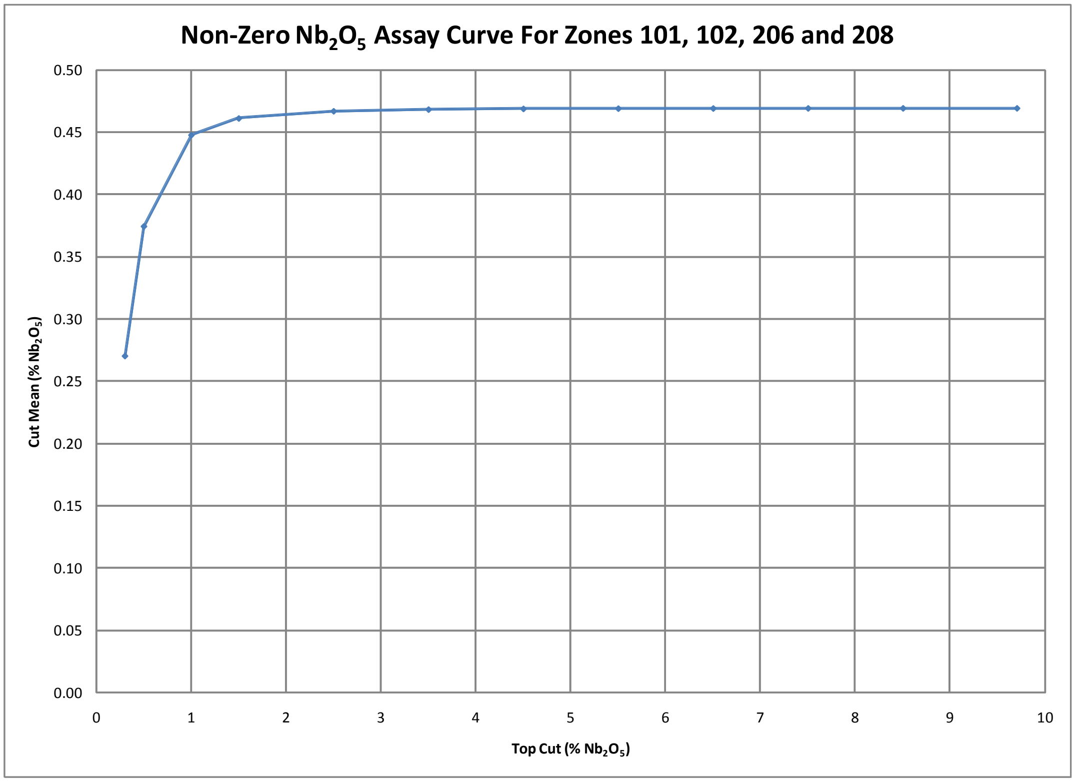
As in the past, no capping of high grades was done for this resource estimation. Reconciliation of past production results against grades predicted from the model supports this practice. RPA carried out an analysis as a check to determine if the use of a top cut is appropriate.

RPA generated "cutting curve" plots of non-zero assay results for Nb_2O_5 for all mineralized domains wherein the samples are subjected to a range of cuts and the "cut mean" is plotted against the top cut value. An example of one of these plots is provided in Figure 17-3.

In RPA's opinion, a reasonable top cut value generally falls along the flatter portion of the curve just before it curves downward toward the y-axis. In the example shown in Figure 17-3, this would be in the range of 2% Nb_2O_5 to 3% Nb_2O_5. RPA conducted additional checks to determine the impact the top cuts will have on the mean, the number of samples affected, and the percentiles of top cuts. A top cut of 2.5% Nb_2O_5, affects only 81 samples, or 0.1% of those within the mineralized domains, and reduces the mean grade 0.5%.

In RPA's opinion, IAMGOLD should study the use of a top cut and consider its adoption for future resource estimations but, overall, they appear to have little impact on global grades.

FIGURE 17-3 EXAMPLE OF CUTTING CURVE



BLOCK MODEL ATTRIBUTES

A series of additional attributes were also incorporated into the block model calculation. These attributes, with description, are shown in Table 17-7.

TABLE 17-7 BLOCK MODEL ATTRIBUTES
IAMGOLD Corp. – Niobec Mine

Attribute Name	Description
Rocky Type	Domain Coding
Density	Specific Gravity
Nb_2O_5	ID^2 Model - % Nb_2O_5
RÉCUPÉRATION	Metallurgical Recovery
RENDEMENT	Yield = (% Nb_2O_5 * Metallurgical Recovery)/10
SiO_2	ID^2 Model - % SiO_2
P_2O_5	ID^2 Model - % P_2O_5
Fe_2O_3	ID^2 Model - % Fe_2O_3
CODELOT	Geological Coding For Metallurgical Recovery

GEOSTATISTICS

VARIOGRAPHY

Variography is not used for resource estimation at Niobec because ID^2 is used for grade interpolation. However, to verify if the search ellipses used for resource estimation and classification were appropriate, studies of vertical and horizontal variograms were performed by RPA. Point areas for mineralized zones, derived from raw Nb_2O_5 assays that were not composited, were used for interpretation. RPA makes the following observations:

- The modelled variograms reveal that a slight anisotropy is developed for the four zones, the major axis being along dip. The semi-major axis, along strike, and the minor axis, across dip, appears similar. Anisotropy ratios are from 1.5 to 2.3 for both major/semi-major and major/minor.

- Variability and ranges in the dip direction appear different from zone to zone. The lowest variability (sill) appears to be in Zone 101 while the highest appears in 102 and 206 zones. Ranges are 15 m to 30 m (40 ft to 100 ft) in 102, 206, and 208 zones while range in Zone 101 is in the order of 50 m to 65 m (150 ft to 200 ft).

- Downhole variography indicates that the nugget effect represents 40% to 50% of the sill. Two structures can be modeled:
    - Structure 1: 8 m to 10 m.
    - Structure 2: 20 m to 30 m.

In RPA's opinion, search ellipsoids used for grade interpolation are appropriate. The variograms are shown in Appendix 1.

BLOCK MODEL GEOMETRY

A block model was constructed (RPA2011_rem) within the MODELE_DEC2006 GEMS v6.3.0.1 project (Figure 17-4). RPA notes that original IAMGOLD's dimensions of the block model were expanded significantly to accommodate the proposed pit shell. Block model parameters are summarized in Table 17-8.

Mined out blocks have been identified by IAMGOLD and a rock code was assigned to these blocks.

FIGURE 17-4 BLOCK MODEL CODIFICATION

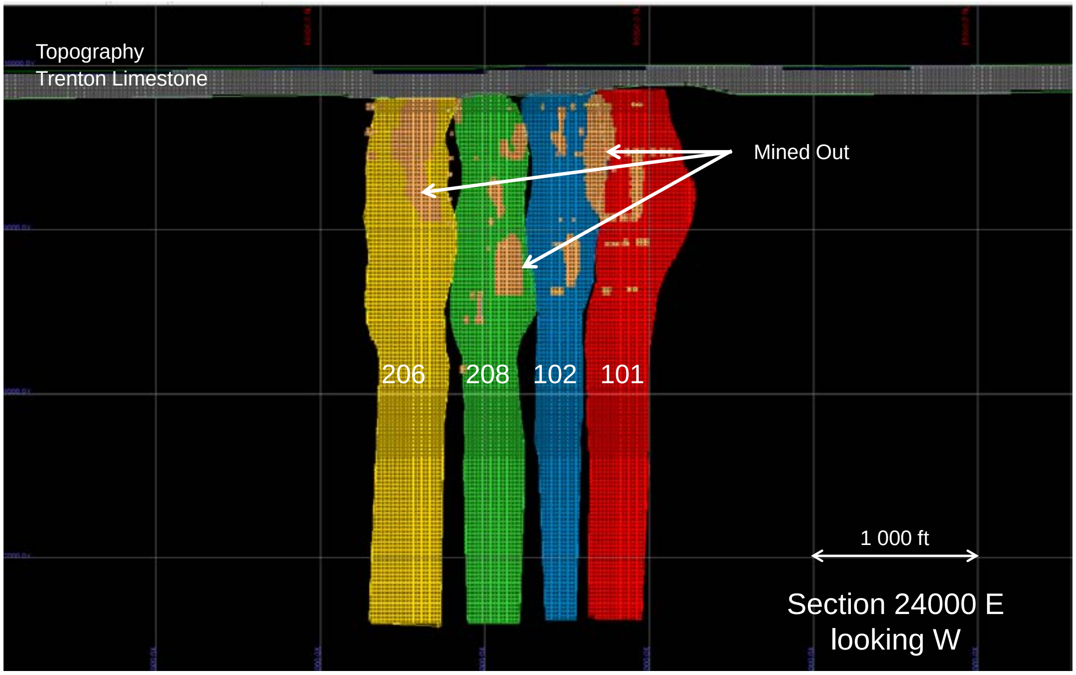


TABLE 17-8 BLOCK MODEL GEOMETRY
IAMGOLD Corp. – Niobec Mine

Origin:	X	20090
	Y	12050
	Z	10000
Rotation:		0
Block Size:	X	20
	Y	10
	Z	25
Columns:		391
Rows:		696
Levels:		140

GRADE ESTIMATION METHODOLOGY

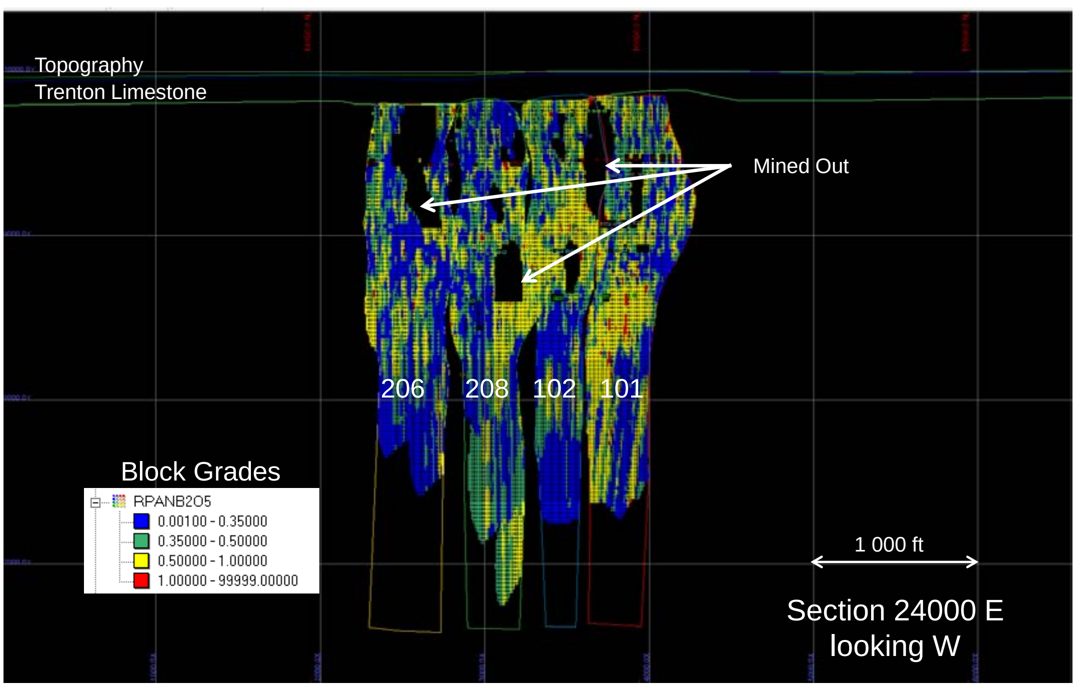
Grade estimation was conducted using ID^2 in Gemcom GEMS v6.3.0.1 software using uncut original assays employing “hard boundaries” between mineralized domains. Interpolation of grade inside the blocks was done using the following parameters:

- First Pass – Search ellipse dimension of 24.4 m by 18.3 m by 9.1 m with a minimum of four and maximum of 12 samples within the search ellipse, and a maximum number of two samples per drill hole.
- Second Pass – Search ellipse dimension of 48.8 m by 36.6 m by 9.1 m with a minimum of four and maximum of 12 samples within the search ellipse, and a maximum number of two samples per drill hole.
- Third Pass – Search ellipse dimension of 121.9 m by 91.4 m by 18.2 m with a minimum of four and maximum of six samples within the search ellipse, and a maximum number of two samples per drill hole.

RPA notes that while Nb_2O_5 is the only material of economic value, other materials, such as SiO_2, Fe_2O_3 and P_2O_5 are also estimated, using the same methodology as Nb_2O_5, because of their importance for blending to maintain the metallurgical recovery of the mill feed.

An example of block model interpolation results is shown on Figure 17-5.

FIGURE 17-5 BLOCK GRADE INTERPOLATION



ESTIMATION OF METALLURGICAL RECOVERY

As the metallurgical recovery of the niobium is variable, each individual sample was coded based on its lithology and mineralization. IAMGOLD has conducted numerous metallurgical tests on different lithologies since production started at Niobec in 1976 and has built up considerable experience and confidence. The expected recovery is an average for each lithological unit and the samples are coded based on these data. A summary of the sample coding and expected metallurgical recoveries for each lithological characteristic are shown in Table 17-9.

In the block model, each block is assigned a code using "nearest neighbour" estimation employing the same large search ellipse used for Nb_2O_5 grade estimation (i.e., 121.9 m by 91.4 m by 18.2 m).

TABLE 17-9 LITHOLOGICAL CHARACTERISTIS AND EXPECTED METALLURGICAL RECOVERIES
IAMGOLD Corp. – Niobec Mine

Code	No. of Metal-lurgical Tests	Description	Recovery (%)
2	41	C3C, CCA (Calcite, Carbonatite)	61.86
4	38	Syenite 50-75% (pink-grey)+C5, C3A, C3C, C3B	49.30
5	98	C3B	60.83
7	87	C3A, C5P, C3B, Hematite	64.04
8	64	(C Dol + C Cal) + Syenite	54.89
15	78	C3NB (white, beige or pinkish)	60.14
17	64	C3NA, C3NB, C3NX (white, beige or pinkish)	58.00
25	52	C3NB (red to brown)	53.84
27	51	C3NA, C3NB, C3NX (red to brown)	52.92
44	18	Chlorite zone (brownish) with Magnetite-Hematite	38.10

ESTIMATION OF RECOVERED METAL CONTENT

In the block model, each block is assigned a recovered metal content, which is defined as the product of the grade (% Nb_2O_5) and the predicted recovery derived from the metallurgical testing. The economic cut-off at Niobec is based on this product (Yield) expressed in kg/t since recovery not only varies between zones but can vary even within stopes. A value is calculated for each block in the block model and is stored in the RENDEMENT attribute. An example of the calculation used to populate this attribute is shown below.

0.60% Nb_2O_5 * 60% Recovery = 0.36% Nb_2O_5 or 3.60 kg Nb_2O_5/t

SPECIFIC GRAVITY

Historically, specific gravity (SG) measurements have been routinely taken at Niobec. In 2010, an additional 53 measurements were added to the dataset. The new data did not cause any change in the accepted values. The density of the rock varies by location within the deposit with Zones 101 and 102 having a value of 2.92 t/m^3 and Zones 206 and 208 have a value of 2.78 t/m^3. The historic data comprises readings from the upper part of the deposit but more SG data will be required for deeper mineralized zones.

CLASSIFICATION

Mineral Resources estimated for the Niobec deposit are classified according to the CIM Definition Standards for Mineral Resources and Reserves (December 11, 2005) and follow the Standards of Disclosure for Mineral Projects as defined in NI 43-101.

RPA has classified Mineral Resources at Niobec according to the search parameters used to inform the block model:

- Measured Resources are limited to blocks which lie within the search ellipsoid dimension of 24.4 m by 18.3 m by 9.1 m. Interpolation indicates that the mean distance of samples to block centres is 14 m.

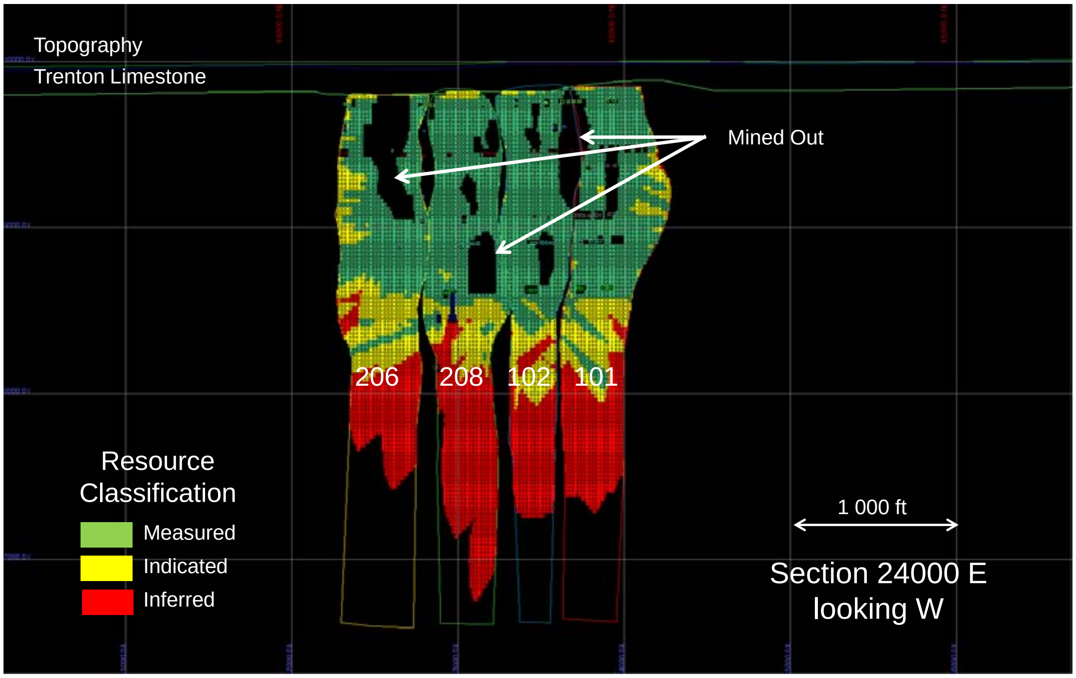
- Indicated Resources are limited to blocks which lie within the search ellipsoid dimensions of 48.8 m by 36.6 m by 9.1 m. Interpolation indicates that the mean distance of samples to block centres is 29 m.

- Inferred Resources are limited to blocks which lie within the search ellipsoid dimensions of 121.9 m by 91.4 m by 18.2 m. Interpolation indicates that the mean distance of samples to block centres is 64 m.

Resource classification at Niobec is supported by the quality and reliability of drilling and sampling data, the distance between sample points (i.e., drilling density), confidence in the geological interpretation of the mineralized body, continuity of geological structures and grade of material within, and a history of positive reconciliations between geological model and mill outputs.

An example of resource classification is shown on Figure 17-6.

FIGURE 17-6 RESOURCE CLASSIFICATION



CALCULATION OF ECONOMIC CUT-OFF GRADE

Mineral Resources, as defined by the CIM Definition Standards for Mineral Resources and Reserves (2005), must have reasonable prospects for economic extraction. A breakeven cut-off grade was calculated for the Niobec Mine using the following assumptions:

- Average long-term niobium price of US$42.00 per kg.
- A US$/C$ exchange rate of 1:1.05.
- An estimated average annual production of 10.0 Mt.
- Average grade of 0.40% Nb_2O_5 for Measured, Indicated and Inferred Resources.
- Metallurgical recovery concentrator of 45.3%.
- Metallurgical recovery convertor of 97.0%.
- Production costs of C$ 26.47/t of ore

These assumptions correspond to scaled-up production and costs.

The breakdown of the production costs is as follows:

- Mining costs of C$10.80/t of ore considering a waste to ore ratio of 5:1.
- Concentrator costs of C$9.50/t of ore.
- Converter costs of C$4.92/t of ore.
- Administration costs of C$1.25/t of ore.

RPA notes that the long term niobium price used in the cut-off calculation is higher than the average of the last three years but lower than the current price.

2010 RPA ESTIMATE (PIT SHELL CONSTRAINED)

As part of the PEA, RPA conducted an independent update of Mineral Resource estimation. Grades for Nb_2O_5, SiO_2, P_2O_5, and Fe_2O_3 were estimated into the blocks using ID^2 weighting. Mineral Resources have been constrained to open pit Whittle shell 6T (refer to Section 18 for details) and are presented in Table 17-1 at the beginning of Section 17.

Figures 17-7 and 17-8 present Mineral Resources vs. Whittle pit shell block model grades and resource classification, respectively.

The independent Mineral Resource estimate conducted by RPA was not identical to the estimate produced by IAMGOLD but, in the opinion of RPA, the differences are negligible and can be attributed to the re-blocking of the model done by IAMGOLD

RPA is of the opinion that the resource model is conservative and has good potential to be expanded. The pit shell includes many blocks that are outside the four main zones, in the mineralized carbonatite where limited to no information is available. A grade of zero has been assigned to those blocks.

FIGURE 17-7 MINERAL RESOURCES (BLOCK GRADES) VS. WHITTLE SHELL

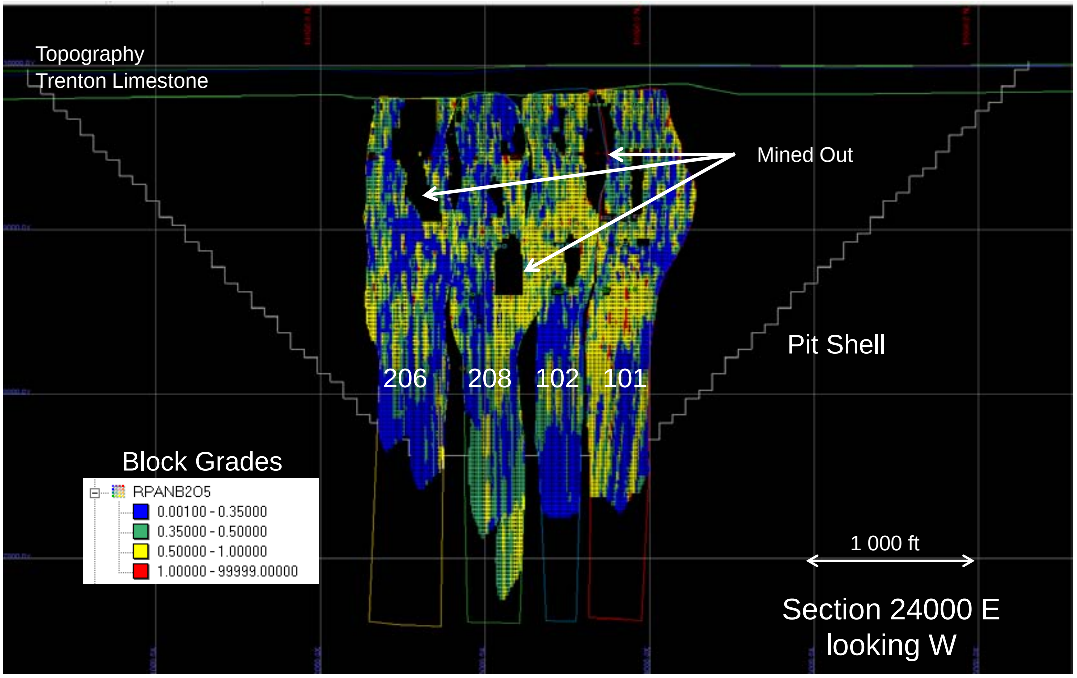
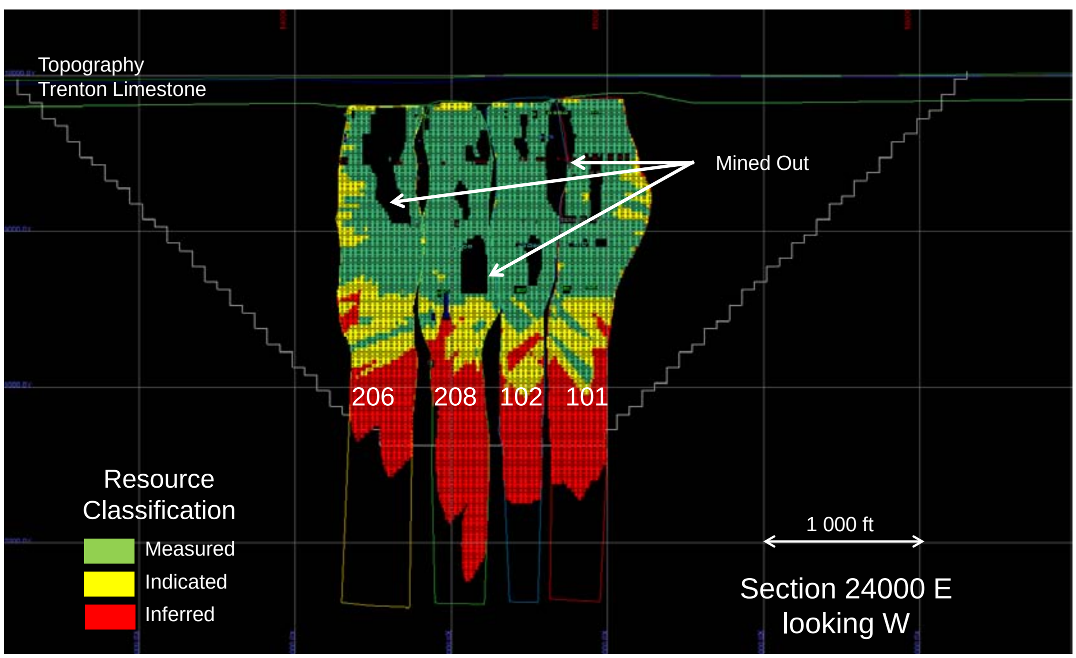


FIGURE 17-8 MINERAL RESOURCES (CLASSIFICATION) VS. WHITTLE SHELL



COMPARISON WITH THE PREVIOUS ESTIMATE

Table 17-10 presents a comparison between the 2010 (UG mining method) and 2011 (OP mining method) Mineral Resource estimates.

TABLE 17-10 COMPARISON OF NIOBEC RESOURCE ESTIMATIONS
IAMGOLD Corp. – Niobec Mine

	Niobec 2010 Resource Estimate[1] UG option			Niobec 2011 Resource Estimate[2] Open Pit option			
	Tonnage (Kt)	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (M kg)	Tonnage (Kt)	Nb_2O_5 Grade (%)	Contained Nb_2O_5 (M kg)	Increase In Contained Nb_2O_5 (%)
Measured	16,571	0.54	90.0	288,930	0.43	1,242.4	1,280
Indicated	29,145	0.53	153.8	169,180	0.40	685.0	345
Measured and Indicated	45,715	0.53	243.8	458,110	0.42	1,927.4	691
Inferred	59,672	0.53	316.3	336,445	0.37	1,240.4	292

Source: IAMGOLD.com

Notes:

1. The December 31, 2010 Mineral Resources extend to 1,036 m below surface.
2. The April 1, 2011 Mineral Resources are constrained in a pit shell, to the 2400 Level (725 m below surface).

MINERAL RESERVES

There are no current Mineral Reserves estimated for the open pit or block caving mining options at Niobec. Mineral Reserves will be assessed at the pre-feasibility stage of study.

18 OTHER RELEVANT DATA AND INFORMATION

RPA and IAMGOLD investigated the potential for open pit and underground (block caving) mining at an expanded rate at Niobec Options were evaluated with ROM ore being processed at 27,400 tpd in a mill and converter plant on site producing FeNb. Infrastructure requirements, road access, and power were investigated. Environmental considerations include the impact of the pit, waste rock dump, and tailings storage.

This section focuses on the OP expansion option and summarizes information and data concerning the existing operations (current and mining expansion under way prior to this PEA) and the block caving expansion option studied concurrently by IAMGOLD, for comparison and ease of understanding purposes.

CURRENT MINING OPERATIONS

The Niobec Mine has been in production since 1976. The present shaft (four compartments) is 850 m deep and is used for production (ore hoisting) and services (materials and manpower). In addition to the shaft, the mine is serviced by a ramp reaching a depth of 750 m

Actual production levels are located on the 600, 1000, and 1450 levels and there is also development on the 300, 700, and 1150 levels (Figure 18-1). Development on production levels are mainly used for ore haulage by trucks to the ore pass. Development is performed using hydraulic jumbos. Ground support is performed to secure the openings using bolters. The broken rock is loaded by LHD and hauled by truck to the ore pass. The ore is crushed and then hoisted to surface by a skip (skip cage configuration hoisting system). Horizontal sill pillars are left between the production levels.

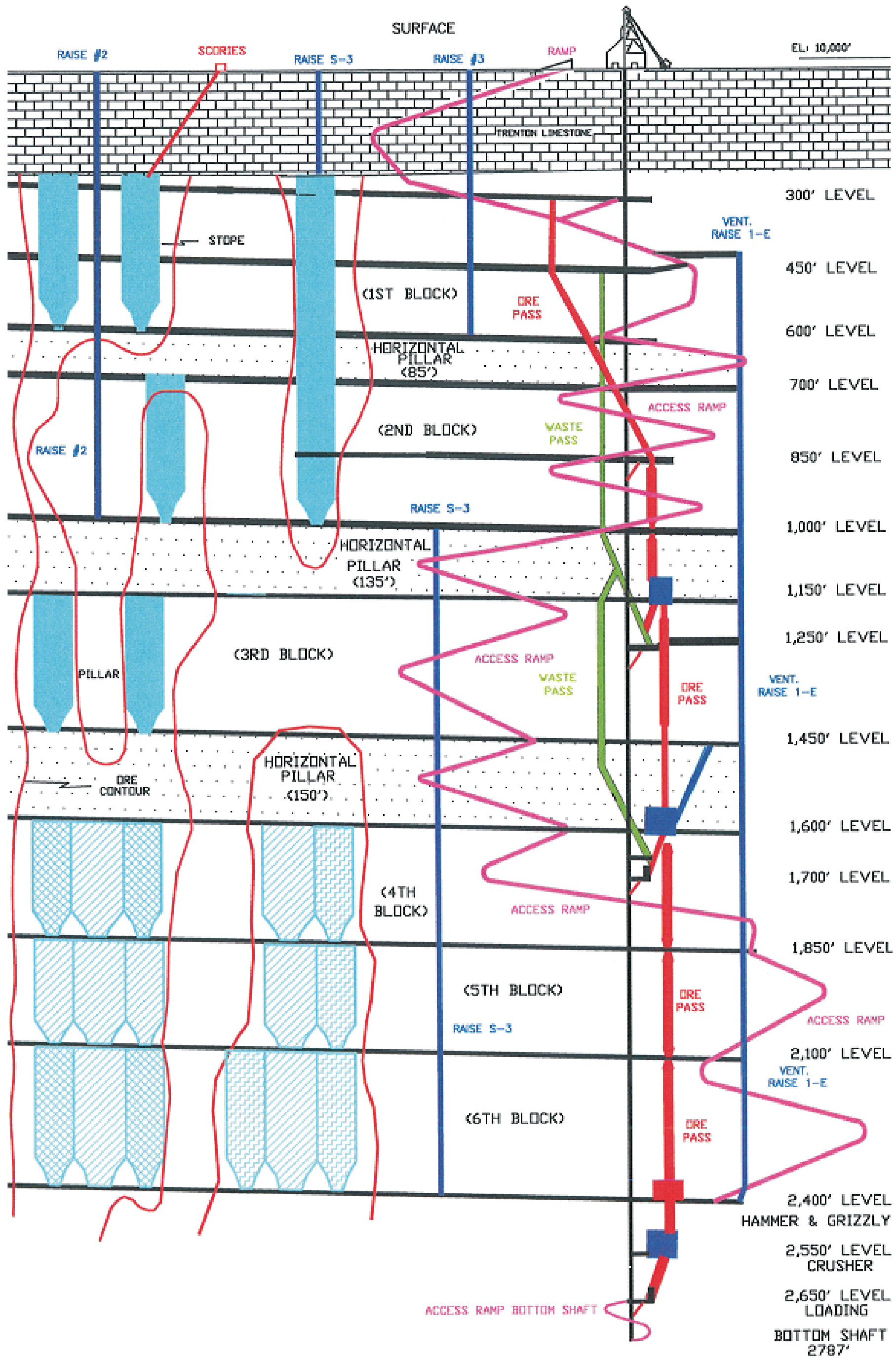


Figure 18-1

Iamgold Corporation

Niobec Mine

Québec, Canada

East - West Section of the Current Underground Mine

June 2011

Source: Iamgold Corporation, 2011.

Open stoping has been the only mining method used since mine start-up. This method is used in the upper part of the mine in block 1 to 3. Stopes are planned and designed based on geological information obtained from diamond drilling. The average size of the stopes is about 60 m in length, 25 m in width and 90 m in height, corresponding to the vertical distance between development and production levels. A 25 m pillar is left between the stopes. Secondary extraction of the pillars can be carried out after the complete extraction of the primary stopes. Occasional mining of the horizontal pillar between two mining blocks is also possible.

Access to a stope is achieved on two levels. On the upper level, secondary parallel drifts separated by a temporary pillar of 4.3 m are excavated within the limits of the stope. On the lower level (production level), draw points are opened at 60° from the transverse drift. They are joined by a drift in the centre of the stope.

Vertical production holes are drilled from the upper secondary. Drilling and blasting of a drop raise (from bottom to top) completes the stope preparation. Production of the stope is achieved with several vertical rings blasted at the same time.

The bulk open stope mining method production cost is the main advantage of the method. At Niobec, this method has been successful as a result of the size of the mineralized zones combined with excellent ground conditions.

A disadvantage of the method is that mineralized zones are wider than the allowable stope width. As a result, a part of the economic mineralization must be left in place as a permanent pillar. Stope design is an important consideration in the Niobec resource estimation. In the upper three mining blocks, open stopes are limited to a width of about 24 m. Where mineralized zones are wider than 24 m, resources must be limited to the volume dictated by stope design; therefore, the remaining resources will not be mined. The same approach applies to pillars that are not planned to be recovered. Mining factors are therefore applied before the final resource estimation. Consequently, historical resource estimates (Measured and Indicated) were the same as reserve estimates (Proven and Probable). Inferred Resources were also estimated taking stope design into account.

The extraction of the top two mining blocks is near completion (less than 5% of current reserves are located in these blocks). Based on a recent rock mechanics study (Golder, 2007), the horizontal pillars will have to be thicker below these depths (45 m between block three and four and 76 m between block four and five). Stope dimensions will also need to be smaller.

Following the Golder recommendations (2007), mining recovery and horizontal pillar recovery were reviewed. The restrictive mining factors described above, combined with the fact that the mineralized zones are wider at depth led to the conclusion that mining extraction could be less than 40% if the current mining method is maintained for blocks 4 to 6.

UNDERGROUND MINING OPTIONS

Three UG mining scenarios were investigated in order to guide future expansion. The first scenario is an updated life of mine (LOM) using the latest 2010 reserve and resource estimates.

The second scenario involves the addition of three new mining blocks (7 to 9) using the paste fill mining method. The second scenario is based on the assumption that the deposit would continue at depth with the same size and metal content. As the second scenario is based on partial or insufficient geological information, the reader should use it as a guide for future strategic decisions. Exploration work combined with detailed optimization studies would be required to confirm the validity of this scenario.

The third scenario involves a change in the mining approach as the mine would be converted to block caving. This scenario was built using the April 2011 RPA block model.

BASE CASE SCENARIO (2.2 Mtpa)

In 2007, a decision was made to deepen the Niobec shaft to 850 m to get access to new mining blocks (4 to 6). As mentioned in the previous section, the mining recovery using open stoping would be as low as 40%. A mining method using cemented fill was selected to allow a much better mining recovery.

Based on the paste backfill studies and simulations, stopes of 15.2 m by 24.4 m by 91.5 m were selected together with a mining sequence that would allow enough curing time. Golder reviewed the work and agreed with the conclusions.

Paste backfill is intended to be used mainly for the mining of block four and lower. As mining is almost completed in blocks one and two, there is currently no plan to backfill these blocks. However, some isolated stopes in the block three could potentially be mined by this method in the future. The recovery of pillar by the paste backfill method was not considered for these blocks.

Stope design restrictions do not apply to the lower mining blocks (four to six) using the paste backfill method. Potential stopes of 24 m X 15 m X 91 m within the mineralized zones are now included in the resources as long as the average grade of the volume (based on the block model) is higher than the economic cut-off. This leads to a substantial resource and reserve increase compared to the open stope mining estimation. The use of a pyramidal mining sequence should be considered for maximal production.

Conversion of the currently used open stoping method to paste fill pyramidal mining method is planned for blocks four to six. As mentioned, a 300 tph backfill plant was required and installed on surface in 2010. The remaining reserves in blocks one to three will be extracted using the open stoping method.

CASE A SCENARIO (3.5 Mtpa)

Case A Scenario involves the addition of three new mining blocks (7 to 9). Blocks 7, 8, and 9 were evaluated using the same mining method as blocks 4, 5, and 6. As such, much of the assumptions are similar to current performances from the Niobec Mine.

Stope drilling and blasting is planned to be similar to current practices. Stopes are 15.2 m wide by 24.4 m long, with sills parallel to the longer edge. Stopes are 91.5 m high, and contain 87,000 t of ore. Drilling and blasting patterns are identical to blocks 4, 5, and 6. For rock stability reasons, a 91.5 m pillar is planned between blocks 6 and 7.

Additional development must be completed to extract blocks 7, 8, and 9. Niobec's current shaft has a depth of 800 m and provides access to blocks one to six. In order to avoid conflicts with current production work and provide additional hoisting capacity, a second shaft is to be sunk 450 m southwest of the present shaft. To facilitate sinking, UG access to the second shaft should be possible via the 2400 level. The UG development work and shaft sinking can begin once the future location of station 2400 is reached.

An east-west longitudinal section view of the mine is shown in Figure 18-2.

CASE B SCENARIO (10 MTPA)

Case B scenario involves a completely different mining method. The mine would be progressively converted to block caving. Block caving mining method is one of the most cost effective UG methods given that its cost could be compared to that achieved with an OP. The deposit needs to satisfy several criteria to be suitable for block caving. In 2011, IAMGOLD engaged Golder to conduct a cavability assessment of the Niobec deposit. In order to guide the reader, an east-west longitudinal section of the Niobec Mine is shown in Figure 18-3.

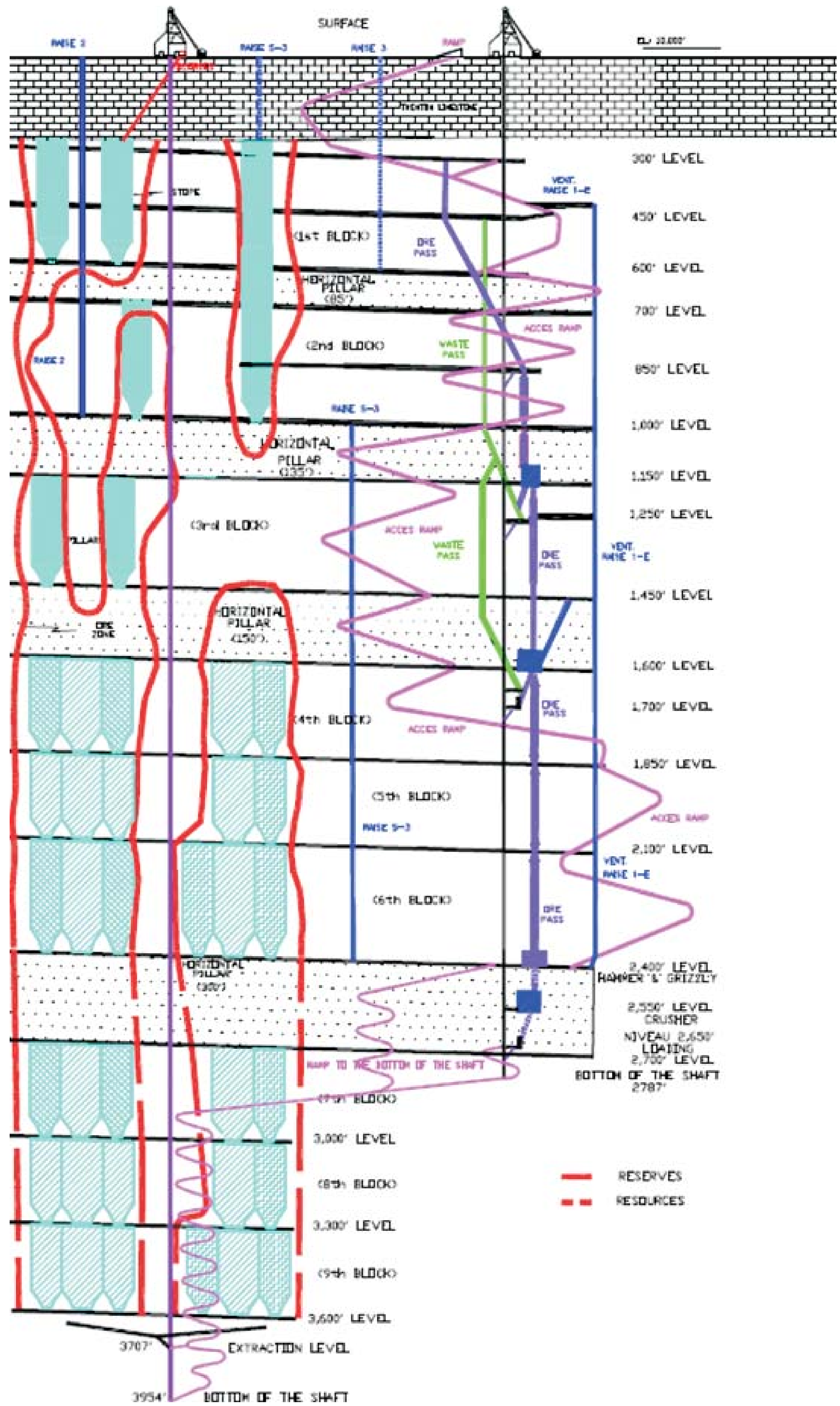


Figure 18-2

Iamgold Corporation

Niobec Mine

Québec, Canada

East - West Section of the Underground Mine in Case A Scenario

June 2011

Source: Iamgold Corporation, 2011.

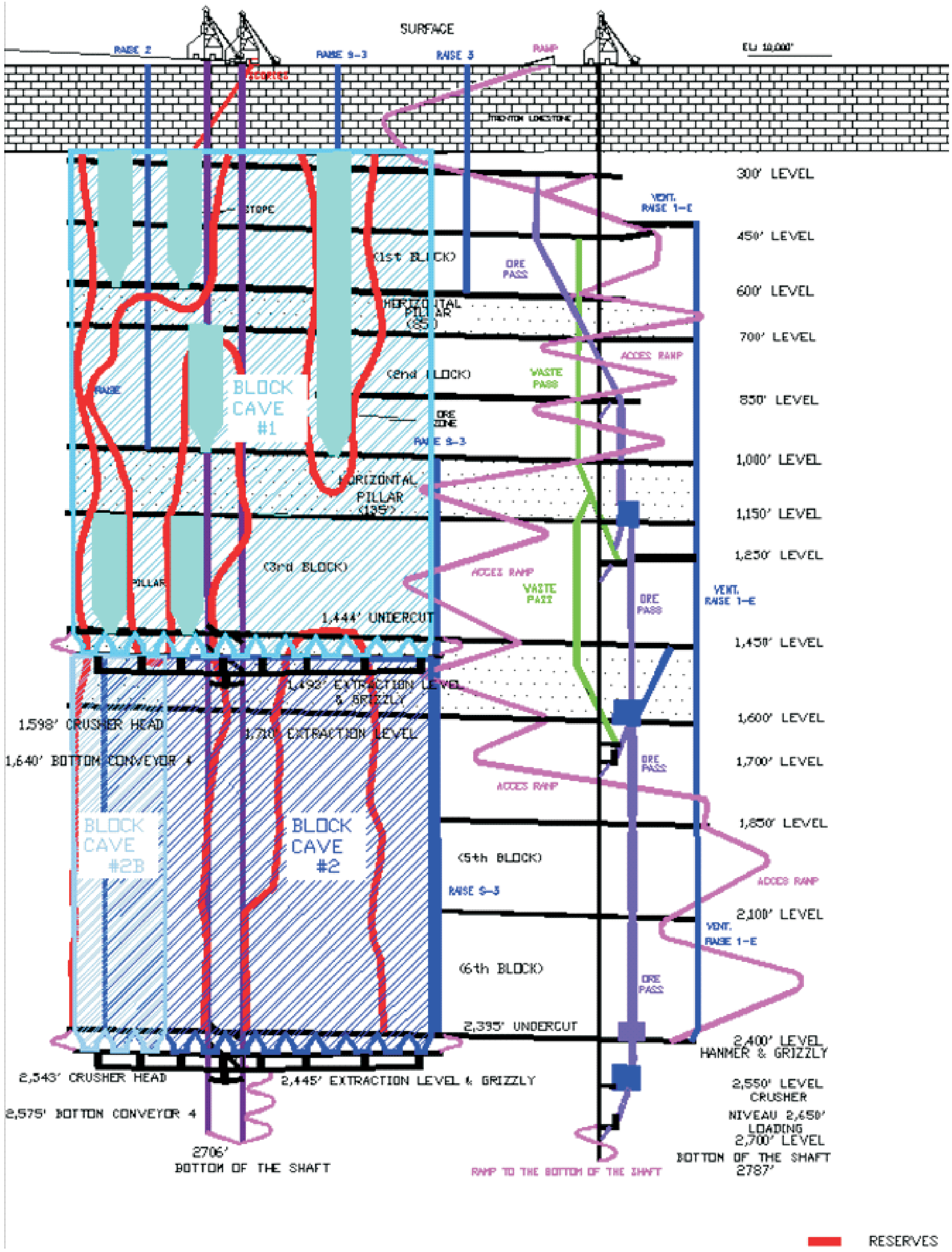


Figure 18-3

Iamgold Corporation

Niobec Mine

Québec, Canada

East - West Section of the Underground Mine in Case B Scenario

June 2011

Source: Iamgold Corporation, 2011.

Block Caving is defined as the process whereby large blocks of ore are undercut and allowed to fall and fracture into smaller pieces that can be loaded and taken to the surface.

Draw points are evenly spaced along the bottom of the orebody and undercut drifts are driven above. From these drifts, miners drill closely spaced holes in a fanning pattern, fracturing the rock. In some cases, explosives can be used to help with the initial fracturation. As said before, the rock above will then rupture under its own weight, falling into the draw points. Once caving begins, the rest of the orebody will continue to collapse and fill the void created, eliminating the need for any additional drilling and blasting.

According to results from Golder's cavability assessment, the required area to ensure caving of the rock mass has to be a 160 m by 160 m. Golder also estimates in this study that a maximum caving height of between 400 m and 500 m would be implemented, after which the draw points will no longer be usable. Consequently, in order to maximize overall mine recovery, two block caving zones are currently planned in existing resources and reserves. The upper block caving would be developed to extract ore up to limestone horizon from 440 m depth level. The lower block caving reaches resources from 730 m depth level up to the upper block caving production level.

The progression rate within the mine is inversely proportional to the overall height being caved. That is, as the height being caved increases, the rate at which draw points are exhausted and need to be replenished diminishes. For the upper block caving level, the caving height of 350 m extracts 24 draw points per year. The lower one, with a height of 250 m, requires 32 draw points per year.

While production continues in the upper parts of the mine, two shafts would be sunk from surface to a depth of 825 m. In order to guarantee the accomplishment of schedule for the upper block caving level, development works such as the drilling levels, production levels and accesses, will have to be finished within the time allowed for shaft sinking. The access for these tasks would be possible by the old shaft. Since the production level will be developed prior and then concurrently with the shafts, block caving can start with the completion of the two shafts. Production using the open stoping method would cease with start up of block caving.

Due to limitations on the size of the conveyors, the mine must operate using both the western and eastern halves simultaneously. Production teams would start from the central haulage accesses, moving towards the extremities as the grade in the draw points becomes too low. This method is similar to panel caving with the fronts moving in an east-west orientated direction.

Use of block caving method in the upper part of the mine (blocks 1, 2, and 3) will allow recovery of the reserves left in place from the open stoping method and mining of new resources and reserves that have been identified in these levels.

OPEN PIT EXPANSION OPTION

MINING METHOD

RPA, in collaboration with IAMGOLD, investigated the potential for OP This option is named the Case C scenario. Open pit possibilities were investigated by floating cone analysis, using Whittle software, run on the resource block model. Pit optimizations indicated that a significant proportion of the resource block model would be economic to mine, using OP methods.

Whittle pit runs were performed based on typical costs for Canadian mining operations of a similar scale, and on IAMGOLD scale-up of current processing/converting, marketing and freight costs as follows:

- Open pit mining: $1.80/t moved
- Milling: $9.50/t milled
- G&A: $1.25/t milled
- Converting: $4.00/kg Nb
- Marketing and freight: $1.41/kg Nb
- Mill recovery: 40% to 60%, depending on Nb_2O_5 head grade

- Converter recovery: 97%
- Niobium in FeNb: 69.9%

Revenue factors were calculated using the above metallurgical recoveries based on Niobec historical data. The revenue factors were used to generate a net value model which was used to float cones in the Whittle software.

MINE DESIGN

For the purpose of pit optimization, a re-blocking of the block model was done to increase block size to 24.4 m by 24.4 m horizontal by 15.2 m vertical (current Niobec coordinates system is Imperial). A pit optimization was run using the previous inputs and a pit slope of 45°. The values of blocks were calculated using a niobium price based on long-term forecasts of $45 per kg and the aforementioned mill and converter recoveries.

In the absence of geotechnical information, pit slope angles were selected based on industry averages. In RPA's opinion, 45° angles may prove conservative, as rock mass is generally competent (stable UG huge open stopes), and steeper slopes have been applied successfully at other mines in the region, such as Inmet's Troilus Mine, which averages 58°. RPA recommends geotechnical testing of representative core samples (located where pit walls are anticipated to be), and, as the Project is advanced, detailed analysis of slope angles is warranted.

Cost inputs for Whittle were based on estimates detailed below, under Operating Costs.

The topographic surface at the Niobec property area is almost flat and the reference surface elevation is approximately 3,050 m. The overburden thickness at the pit location varies from 0.9 m to 5 m, averaging 3 m. For economic purposes in the current PEA, overburden was considered as waste rock.

The Whittle economic optimization yielded a pit (Pit 6T) which contained the Mineral Resource stated previously in this report. A second economic optimization was performed on the first Whittle result by means of IAMGOLD's Comet Production Scheduler Software in order to maximize its net present value (NPV) at 8%. A time parameter was introduced into this process to force the schedule to stop in 2052, as was done in the block caving scenario, for comparison purposes. The maximum NPV of this pit optimization process occurred at a cut-off grade of 0.33% Nb_2O_5.

The exercise returned 370 Mt grading 0.46% Nb_2O_5 and corresponds to a smaller pit within Pit 6T. The proportion of Inferred Resources in the material that may be potentially mineable via open pit is approximately 20%. Waste mining of 1,482 Mt is required, for a strip ratio of 4:1. The pit shell is of circular shape with a 1,830 m diameter and is a maximum of 564 m deep. The crest of the pit encroaches on the current tailings and settling pond (Figure 18-4).

Floating cone analysis does not include individual benches or ramp design. For the pit size, production requirements, and recommended equipment fleet, RPA considers mining of 12 m benches and development of 33 m wide ramps, including ditches and safety berm, to be appropriate. The ramps should be designed at 10% grade to exit the pit towards the northwest and the southwest, in the directions of the mill and the waste rock dumps, respectively.

A section view of the Whittle pit shells looking north is shown in Figure 18-5. The pit shell 4T (bottom at 564 m below surface) is the subject of the PEA in this report. With the exception of pit 6T (host of the Mineral Resource), all other pit shells were used to develop the LOM production schedule. Pit shell numbers correspond to existing UG mining block numbering that will be referred to in the following subsections.

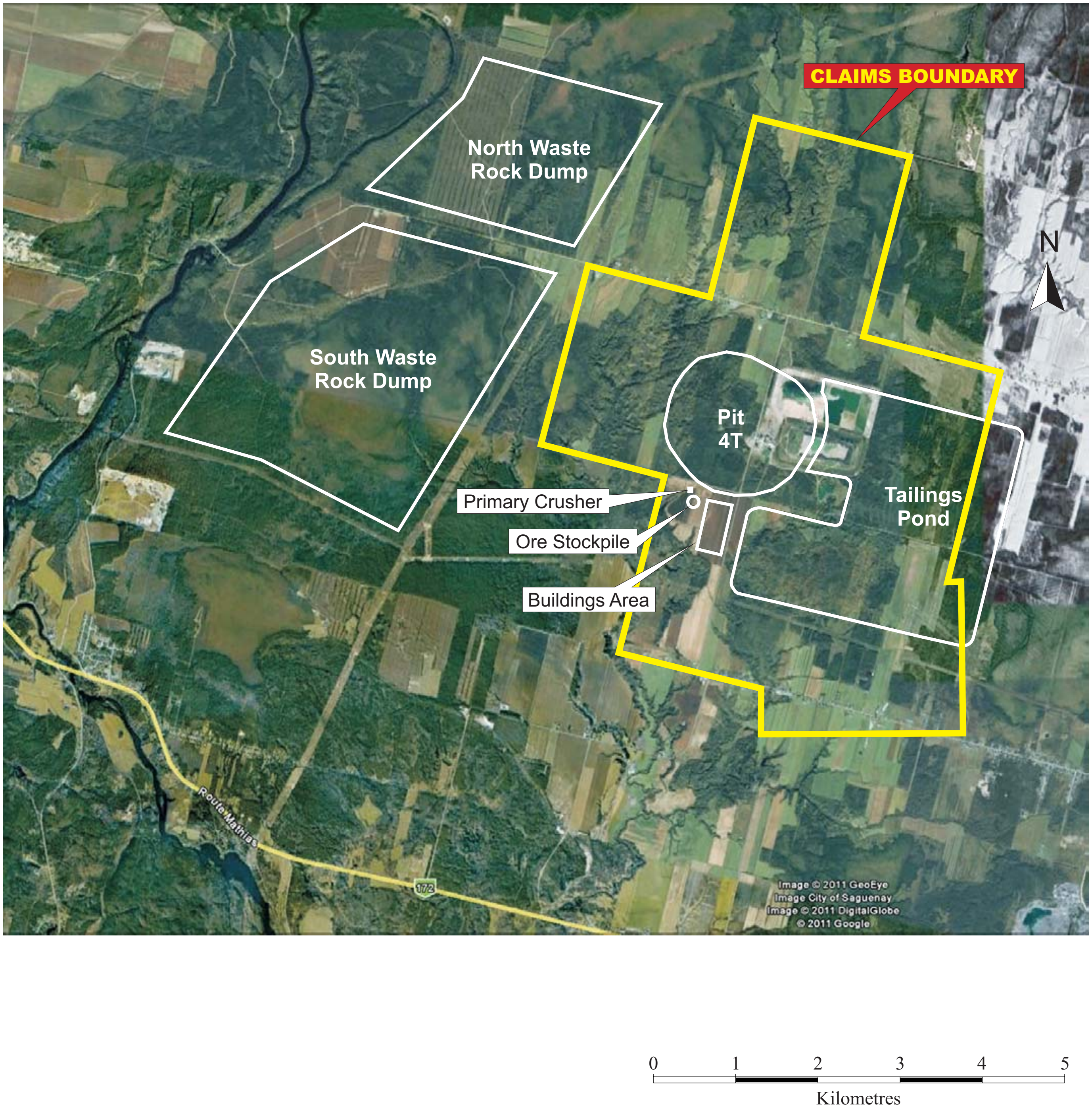


Figure 18-4

Iamgold Corporation

Niobec Mine
Québec, Canada

General Plan View

June 2011

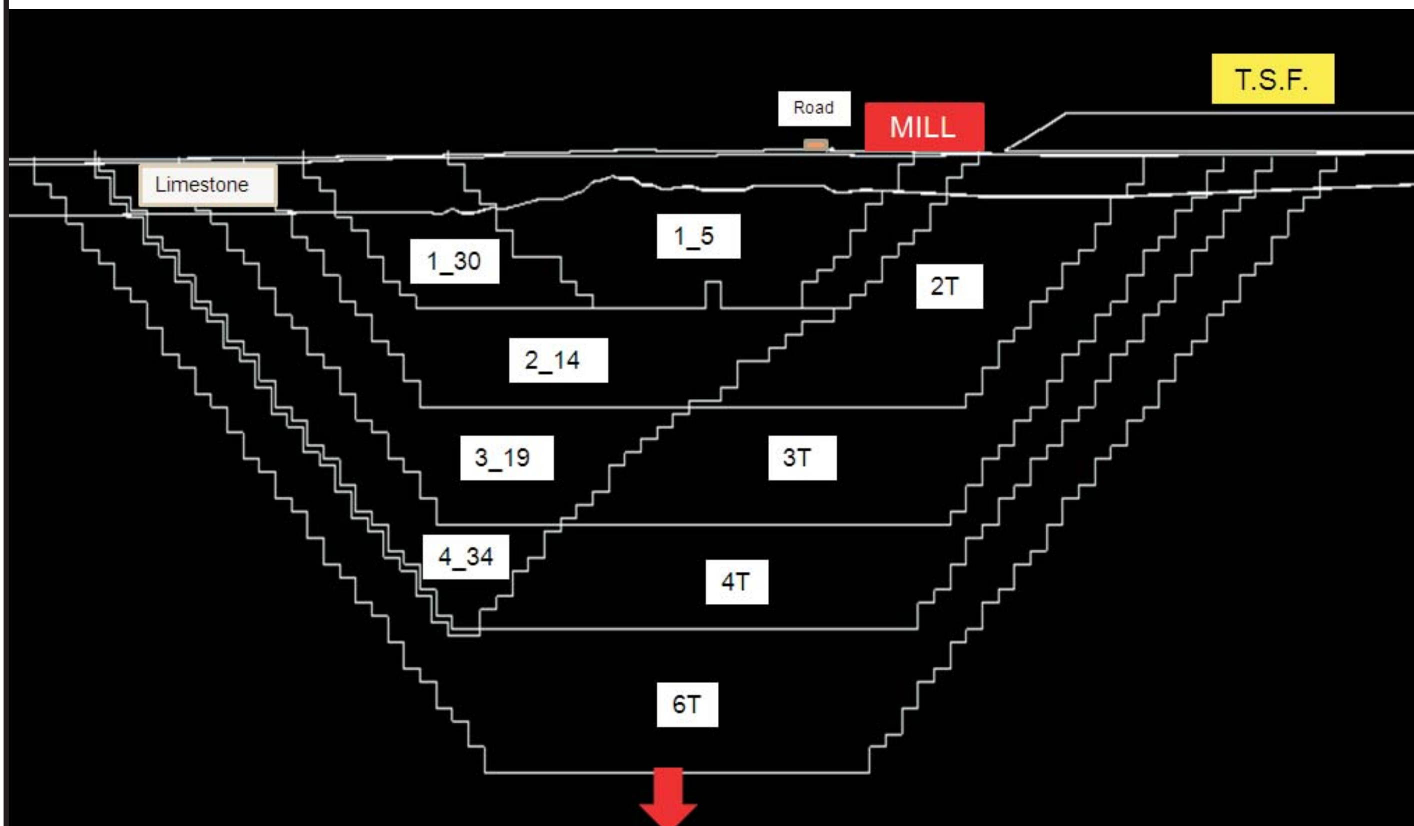


Figure 18-5

Iamgold Corporation

Niobec Mine

Québec, Canada

East - West Section View of Whittle Pit Shells

June 2011

Source: Iamgold Corporation, 2011.

MINING THROUGH UNDERGROUND MINE OPENINGS AND EXISTING SURFACE INFRASTRUCTURE

At Niobec, the use of UG open stope mining in the past left behind mineralized pillars as structural support for the mine.

As the pit will be mining through old open stope mine workings, the operating plan includes backfilling of each open stope with crushed limestone or low grade ore when available. Mining through UG voids without backfilling is common in some large open pits (Super Pit at Kalgoorlie, Western Australia and Dome Mine at Timmins, Ontario) so there remains potential upside to improve mining costs by eliminating the fill.

The vertical layering of current UG mine workings is approximately as follow:

- 0 – 300 ft. (91.4 m) deep: Trenton limestone horizon (waste)
- 300 – 600 ft. (182. 8 m) deep: UG mining Block 1
- 600 – 700 ft. (213.3 m) deep: sill pillar
- 700 – 1,000 ft. (304.8 m) deep: UG mining Block 2
- 1,000 – 1,150 ft. (350.5 m) deep: sill pillar
- 1,150 – 1,450 ft. (441.9 m) deep: UG mining Block 3
- 1,450 – 1,600 ft. (487.7 m) deep: sill pillar
- 1,600 – 1,850 ft. (563.9 m) deep: UG mining Block 4.

During stripping of the OP in the limestone horizon, a portion of the waste will be crushed in the pit with a mobile crusher and this material will serve as fill for the UG open stopes of Block 1 via sub-vertical 12 in to 14 in dia. drill holes. Mining will be scheduled to always have a "buffer" pillar of at least 100 m above UG openings, thus drill holes for filling purposes will be 100 m long at least. Sub-economic or low grade ore from Block 1 will be used to fill Block 2 UG openings and same material from Block 2 will be used to fill Block 3 UG openings following the same approach.

During stripping of the OP in overburden and Trenton waste, the UG mine will operate until OP will start producing. Both operations will manage to have an approximate 243.8

m "buffer" pillar between the bottom of the pit and the UG working places during this transitional period. Therefore, a transitional period at 2.2 Mtpa as current will be followed by the OP production start-up; at that time, UG mine will contribute zero tonnes and be closed. Prior to begin OP development, UG ventilation raise and infrastructures will have to be relocated because currently located inside future pit footprint.

During OP mining, once filled UG stopes are encountered on a bench, the fill material will be reclaimed first and hauled to the waste rock dump (Trenton in Block 1), or hauled to low grade stock pile or the primary crusher (low grade ore in blocks 2 and 3), and these openings will serve as first cuts. Drilling and blasting will then start around them and progress further away. Reclaiming of crushed Trenton into the UG stopes during OP mining will be done very carefully in order to avoid sending Trenton waste to the mill. Trenton limestone is very deleterious material to the process and impacts on the quality of the final product.

The OP bench floor elevations for few assigned benches will be designed to fit with drift floor elevations of UG level (at depth of 300, 450, 600, 700, 850, 1,000, 1,150, 1,300, 1,450 and 1,600 ft). This way, on a bench height basis (12 m), the UG drifts (3.6 m high) should not be an issue because production drill holes will connect with them, be plugged at the bottom, loaded and blasted. If the pillar above drift collapses, the swell should fill the drift before reaching the OP bench floor above.

During the UG to OP transition production period, the filled stopes, once connected into the pit, will potentially drain OP surface water into the UG mine. A provisional increase of the current UG pumping capacity will be considered in the costs estimation.

The buffer pillar approach will have to be validated with geotechnical / geomechanical / rock mechanics studies, as well as the 45° open pit slope angle for scoping design. The OP surface water inflow into the UG mine will both have to be validated as well with hydrology / hydrogeology studies.

The pit footprint will eventually reach the existing tailings pond and be near the future tailings pond. Once required, a portion of the existing tailings will be relocated into new pond and the existing pond will be re-profiled. A safe distance between tailings dam toes and pit crest will have to be determined upon a geotechnical assessment of the overburden underlying/surrounding the tailings ponds at these locations.

The location for waste rock dumps (as shown in Figure 18-4) was determined based on the area required (12 km^2 and 76 m high) and with first objectives to minimize the impact on water courses, to have a 100 m buffer distance from Hydro Quebec's high-voltage power lines and to avoid being over known potential Mineral Resources (REE mineralization north of the open pit). The waste rock dumps and the underlying soils will have to be assessed upon geotechnical studies as well as condemnation drilling at their locations.

Finally, a provisional surface area allows for land purchases for OP expansion option, particularly for waste rock dump footprints.

PRE-PRODUCTION SCHEDULE

It is currently estimated that the feasibility studies and permitting will be completed from 2011 through 2013, overlapped and followed by a development and construction phase of two years with increased production starting in the first quarter of 2015.

LIFE OF MINE PRODUCTION SCHEDULE

The production schedule was generated with the participation of IAMGOLD using Comet Scheduler Software with Whittle optimization results as inputs. The mill throughput was set to 10 Mtpa at Year 3 and beyond, ramping-up from 5 Mtpa and 7.5 Mtpa in the first and second years of production respectively.

Twelve million tonnes of waste pre-stripping will be required in Year -1. The transition from UG and OP mines will last four years from Year -4 to -1 (or from 2011 to 2014, as currently planned).

The UG mine remaining production combined with the OP expansion project resulted in a 42-year LOM, or up to 2052, as shown in Table 18-1.

TABLE 18-1 NIOBEC LIFE OF MINE PRODUCTION SCHEDULE
IAMGOLD Corp. – Niobec Mine

	Units	LOM Total	Year -4 2011	Year -3 2012	Year -2 2013	Year -1 2014	Year 1 2015	Year 2 2016	Year 3 2017	Year 4 2018	Year 5 2019	Year 6 2020	Year 7 2021	Year 8 2022	Year 9 2023	Year 10 2024
Ore Mined	(Mt)	379.03	2.20	2.20	2.20	2.20	5.00	7.49	9.99	10.00	9.99	9.99	9.98	10.00	9.99	9.92
Nb_2O_5 Head Grade	(%)	0.463%	0.550%	0.620%	0.620%	0.620%	0.549%	0.518%	0.489%	0.458%	0.470%	0.480%	0.438%	0.492%	0.503%	0.507%
Waste Mined	(Mt)	1481.67	0.00	0.00	0.00	12.00	49.14	47.51	45.01	44.92	45.01	45.01	44.71	45.00	45.01	45.08
Total Mined	(Mt)	1860.70	2.20	2.20	2.20	14.20	54.14	55.00	55.00	54.92	55.00	55.00	54.69	55.00	55.00	55.00
Waste to Ore ratio		3.91	0.00	0.00	0.00	5.45	9.83	6.34	4.51	4.49	4.51	4.50	4.48	4.50	4.50	4.55

	Units	Year 11 2025	Year 12 2026	Year 13 2027	Year 14 2028	Year 15 2029	Year 16 2030	Year 17 2031	Year 18 2032	Year 19 2033	Year 20 2034	Year 21 2035	Year 22 2036	Year 23 2037	Year 24 2038
Ore Mined	(Mt)	9.49	9.97	9.99	10.00	9.91	9.65	9.95	10.00	9.47	9.98	9.99	9.99	9.98	9.97
Nb_2O_5 Head Grade	(%)	0.524%	0.428%	0.492%	0.500%	0.508%	0.517%	0.508%	0.497%	0.527%	0.446%	0.385%	0.445%	0.431%	0.394%
Waste Mined	(Mt)	45.51	44.33	45.01	45.00	22.09	45.35	44.59	12.00	45.53	44.92	45.01	45.01	45.02	45.03
Total Mined	(Mt)	55.00	54.30	55.00	55.00	32.00	55.00	54.54	22.00	55.00	54.90	55.00	55.00	55.00	55.00
Waste to Ore ratio		4.79	4.45	4.51	4.50	2.23	4.70	4.48	1.20	4.81	4.50	4.51	4.51	4.51	4.52

	Units	Year 25 2039	Year 26 2040	Year 27 2041	Year 28 2042	Year 29 2043	Year 30 2044	Year 31 2045	Year 32 2046	Year 33 2047	Year 34 2048	Year 35 2049	Year 36 2050	Year 37 2051	Year 38 2052
Ore Mined	(Mt)	9.98	9.73	9.99	9.97	9.98	10.00	9.99	10.00	9.94	10.00	10.00	10.00	10.00	9.99
Nb_2O_5 Head Grade	(%)	0.371%	0.322%	0.404%	0.490%	0.470%	0.408%	0.412%	0.446%	0.277%	0.497%	0.493%	0.491%	0.470%	0.477%
Waste Mined	(Mt)	45.02	44.54	45.01	45.03	45.02	45.00	45.01	44.14	45.06	12.00	8.00	7.00	4.00	4.01
Total Mined	(Mt)	55.00	54.27	55.00	55.00	55.00	55.00	55.00	54.14	55.00	22.00	18.00	17.00	14.00	14.00
Waste to Ore ratio		4.51	4.58	4.51	4.52	4.51	4.50	4.51	4.41	4.53	1.20	0.80	0.70	0.40	0.40

MINE EQUIPMENT

The mine equipment fleet, listed in Table 18-2, was selected based upon comparison to operations of similar size and using InfoMine USA Inc. The production schedule (see below) requires a fleet capable of moving 55 Mt of material per year, on a 24 hour per day and 365 day per annum schedule.

TABLE 18-2 MINING FLEET
IAMGOLD Corporation – Niobec Open Pit Project

Type	Quantity
Cable Shovel 13 m^3 (ore)	1
Cable Shovel 35 m^3 (waste)	2
Haul Trucks 218 mt	28
Rotary Drill 38 cm	4
Dozer 305 kW	5
Grader 160 kW	3
Water Truck 75,000 liters	1
Service Truck	12
Bulk Explosive Truck 450 kg/min	2
Fuel truck	1
Light Plant 13 kW	5
Pump 100 kW	6
Pickup truck	24

INFRASTRUCTURE

The OP expansion option is principally a new construction as the current installation will be inside the pit limit created by this mining approach. As the current shaft will not be compromised until mining activities reach the shaft installation, the existing UG shaft services will remain in place as current underground mining will continue until open pit production start-up. For this purpose, part of the existing ventilation surface system and raises will have to be temporarily relocated. The water line from the Shipshaw River is only infrastructure that will be useful beyond this time.

EXISTING INFRASTRUCTURE

The existing shaft will be conserved up until the open pit production start-up and then be demolished. The water line and water recirculation infrastructure will be conserved as per its existing function and also expanded upon. The remainder of the equipment will require demolition. The value of demolition is considered as zero cost, as the value of equipment and scrap metal generated will compensate for the cost work to remove. Special attention will be required to removing equipment that has high radon emission values. The existing 25 kV line power will be decommissioned in appropriate areas to keep supplying power to water pumping stations.

ROADS

With the pit footprint cutting through the Columbium Road nearby the current mine site, a new road is proposed to reduce traffic flow around the communities. The road is shown in Figure 18-6.

For this reason, it has been included in the study to upgrade the road from Highway 172 to the Columbium Road shown by marker A to B in Figure 18-6. This would permit transport of material to Niobec without having to pass through small villages. The seven kilometre road would be paved. Additionally roads and parking facilities as well as terracing of the plant are provided for. A total of four kilometres of gravel road is foreseen. This will be a gravel road for local private traffic and employee parking.

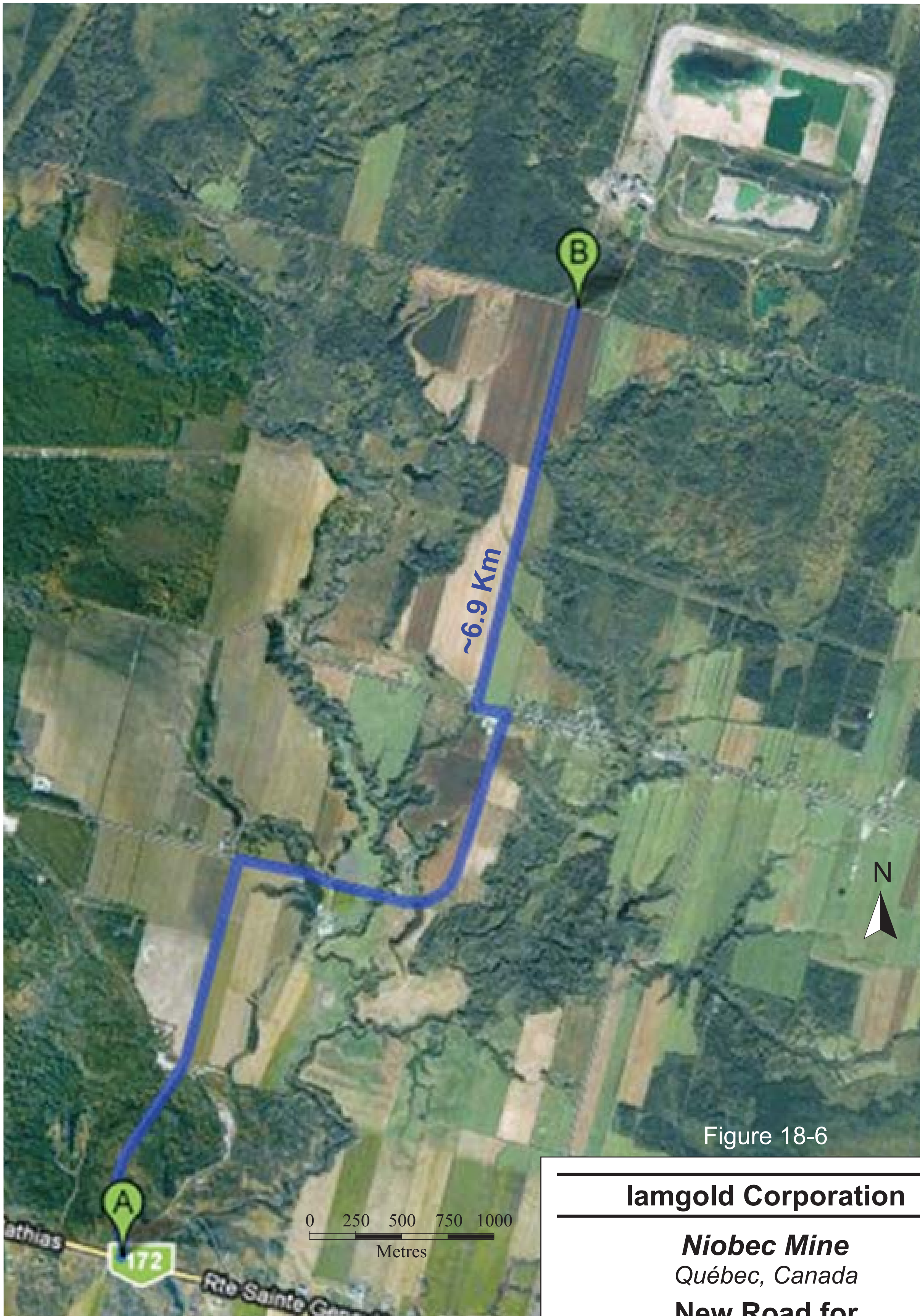


Figure 18-6

Iamgold Corporation

Niobec Mine

Québec, Canada

New Road for Mine Site Access

June 2011

Source: Iamgold Corporation, 2011.

BUILDINGS

The following buildings infrastructure is added to the new open pit project to provide appropriate services: Mine Administration and Service Office, Mill Workshop, Mill Service and Dry Building, Mill Office, and Truck shop and Mine Dry Building.

The Mine Administration and Service Office will be subdivided as per the following: 5,000 m² of offices and 500 m² of warehouse space. Offices are provided with Information Technology (IT) and communications infrastructure. This warehouse will be principally for mill equipment and general supplies.

The Mill Workshop building provides shop layout with central foreman offices for welding and mechanical services as well as small repairs.

The Mill Service and Dry building provides a change house and services building for plant workers for a total of 500 m². This is a typical layout of showers, lockers and sanitary facilities.

The Mill Office provides for 1,000 m² for offices and conference room for mill personnel. Office sizes and layout are similar to the mine offices and provides for IT and communications infrastructure.

Open pit mining operation will require a truck shop for doing maintenance on mobile equipment and change rooms for mine and maintenance workers. These services will be housed into the Truck shop and Mine Dry Building. To allow for mine maintenance planning and execution, attached to the facilities are the mine maintenance offices. This building is divided up into five main areas: the offices, the change rooms, the workshop, the warehouse and the truck shop.

The truck shop offices are laid out over 400 m² comprising of 16 closed offices for the supervisors, engineers, foremen and managers, a larger room for dispatchers and IT, a conference room and a copy room. Leading off from the offices are the men's and women's change room areas. The women's change room has a 10 person capacity while the men's change has a 375 person capacity, and both span an area of 360 m². The workshop is located between the men's change room, the warehouse area and the truck shop, and spans 230 m². It links the truck shop and warehouse, and contains the electrical room, the mechanical room and the tool crib. The workshop has a second floor mezzanine, covering 200 m², with windows overlooking the truck shop. It has a training room for eight people, as well as four private offices for the maintenance staff. There are also two large offices for maintenance coordinators, and smaller room for maintenance general foremen. The warehouse area spans 490 m², and is equipped with a climate control storage area. The truck shop floor is 1,600 m² (approx. 60 m x 26 m) and has an 18-m high roof. The bays are split in two types: preventative maintenance and heavy duty repairs. There is also a 25-t bridge crane spans the length of the entire truck shop.

The wash bay is located near the truck shop and situated in such a way as to provide easy access to all the maintenance bays. The entire area is enclosed by a building envelope with sufficient heating to prevent freezing.

Each building is linked up to fire suppression services where required and serviced by a septic field. Potable water is assumed to be provided by the city of Saint-Honoré.

PRIMARY CRUSHER AND STOCKPILE FACILITY

In this scenario, a crushing facility is required to process rock from open pit run-of-mine (ROM). A 54 by 75 Metso Crusher is foreseen with necessary 50/20 t bridge crane. The crane will be installed early to permit its use during construction. An apron feeder feeds a stockpile conveyor that in turn feeds the ROM coarse ore stockpile of 25,000 t live load capacity. The covered stockpile is drawn using four apron feeders leading onto another conveyor providing feed to the new grinding facility SAG mill feed chute. All conveyors are covered with access from both sides (one side for maintenance and the other side for operation/maintenance).

POWER AND COMMUNICATIONS

As the existing main substation lies in the affected zone of mining, a new substation located just nearby the new plant is proposed. The modification will require extending the current 161 kV line by Hydro Québec and installing two new 60/80 MVA transformers. Redundancy is optimized by being able to tie-in the two switch gears in case one transformer is damaged. The work of constructing the new station will not affect current operations as it is located in a new area. Energization will be the only constraint for the new substation. The estimated total load of the new plant is 80 MW and the design proposed will have no issue in supplying this power.

Communication systems infrastructure is reproduced in the new design taking advantage of fiber optic links from the 25 kV power network. All new infrastructures will have appropriate supply of networking capacity especially in offices where network access is required. Server and outside link services are provided for in all major buildings. As foreseen for all additional plant equipment infrastructure related to communications is the design and pricing, extending out the present PLC infrastructure to the current communication backbone.

FUEL STORAGE AND DISTRIBUTION

A fuel handling system will be provided for the open pit fleet and be equipped with a fire suppression system.

PROCESSING PLANT

This scenario requires a new processing plant as the existing infrastructure will be compromised by the open pit footprint generated from the new mining approach. The major equipment used for processing has been covered into the Mineral processing and Metallurgical testing section earlier in the report, section 16. The following are significant descriptions of the new required plant.

Each area has provision for its own or combined heating, ventilation and air conditioning (HVAC) system. Architecture is appropriately sized for each area. Most areas have bridge cranes foreseen to permit installation of equipment and servicing later upon start-up. There is one central electrical room that services the processing plant.

GRINDING, DESLIMING, PYRITE FLOTATION, CARBONATES FLOTATION, DEWATERING, MAGNETIC SEPARATION, PYROCHLORE FLOTATION, SULPHIDE FLOTATION, LEACHING AND LEACH FILTRATION, FILTRATION, DRYING AND PACKAGING, TAILINGS DISPOSAL, CONVERTER, CONCENTRATE STORAGE

All these items were evaluated and estimated properly. Details of the sizing and arrangements are confidential

TAILINGS AND WATER MANAGEMENT

The increased capacity of storage required with the open pit scenario is approximately 400 million tonnes of mine residue, including existing tailings relocation. The current tailings ponds No. 1 and No. 2, presently have a surface of 2.2 km². SNC-Lavalin Inc. (SNC) was mandated to study the new tailings impoundment scenario earlier this year for a 350-Mt capacity. The design was based on the assumption of a 1.65 t/m³ residue specific gravity and that 100% of the mining residue is stored in the tailings pond with the mining method of this scenario. Despite the difference in capacity, the area at surface was assumed to remain the same, thus SNC's capital cost estimate prevailed. At the end of the LOM, the tailings pond will be higher than estimated resulting in additional dam raising; this has been accounted for into processing operating cost item.

Assuming a design similar to tailings pond No. 2 with decantation of water to a collecting/polishing pond, the new tailings pond will have a surface area 6.5 km² and roughly be 43 m high (38 m high initially estimated by SNC), as shown in Figure 18-7. All lands required for this new impoundment have been estimated at a flat rate and assumed that impediments in acquiring the land. Construction of the ponds will borrow from methods of construction and operation based on the existing No. 1 and No. 2 basins which have proved effective.

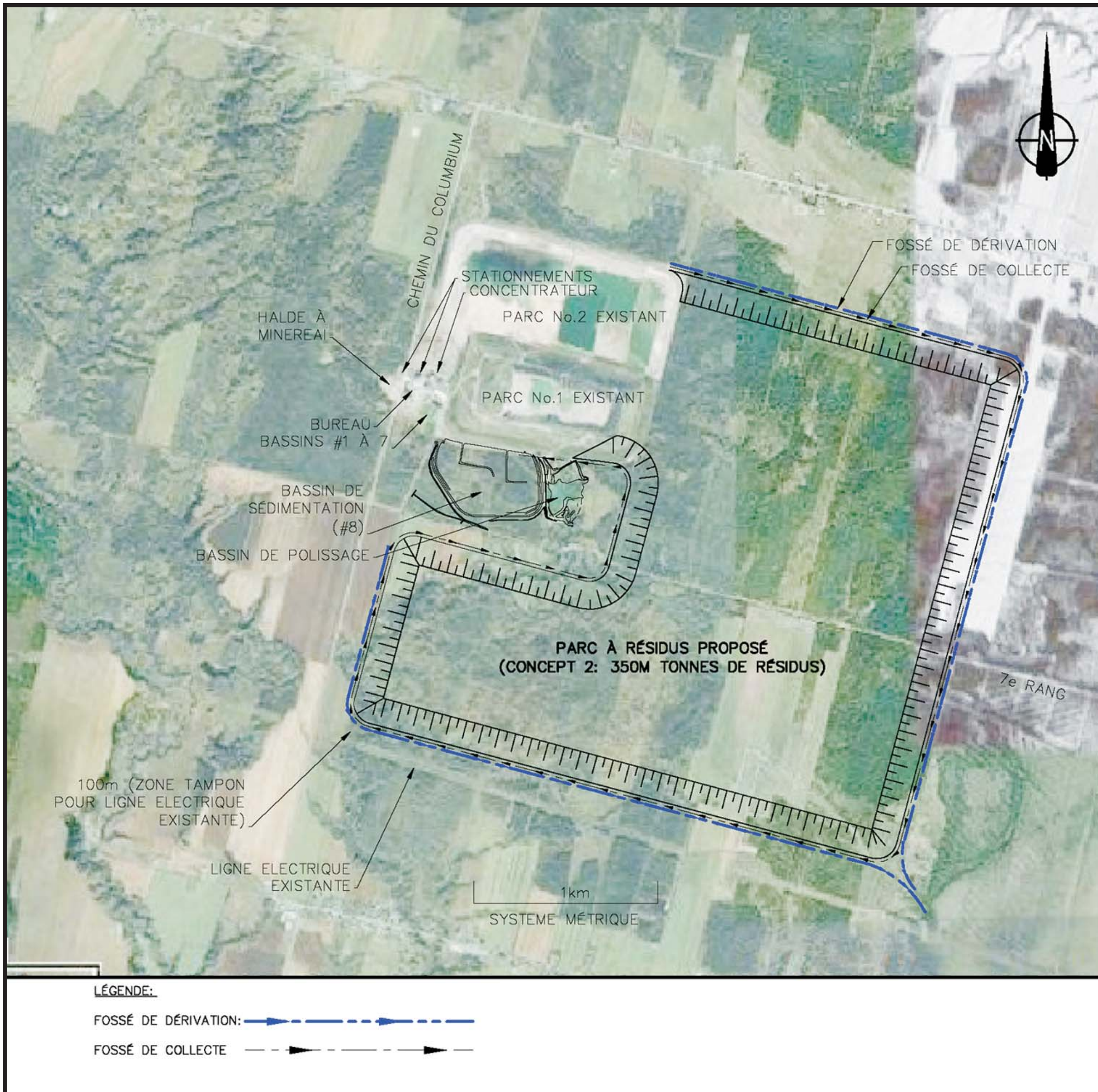


Figure 18-7

Iamgold Corporation

Niobec Mine

Québec, Canada

Tailings and Collecting/Polishing Ponds Facility

June 2011

Source: Iamgold Corporation, 2011.

Water management follows an extension of the existing layout using basin No. 8 as the polishing basin and principal return of water for the mill with overflow directed to effluent water stream pumped to the Shipshaw River.

A new pump station is built beside pump station No. 5 and is expanded to accommodate the increase water require from the plant.

The water provided by the Shipshaw River is augmented by quadrupling its present design capacity feeding basin No. 2 and No. 3. The twin 500 mm high density polyethylene (HDPE) pipe line 12 km long will be implanted alongside the pipelines foreseen to be installed in early 2011. The pumping station at the Shipshaw River will be consequently quadrupled in capacity. The intake affluent and diffusion effluent into the Shipshaw River will be performed in the identical manner of the current concept. The total capacity will be designed at a minimum 54,520 standard cubic metres per day (SCMD) and maximum 65,500 SCMD of fresh water and minimum 62,800 and maximum 80,800 SCMD of effluent water.

As it is assumed that effluent control for suspended solids will remain an important factor for discharge, three new clarifying systems will be installed to cover the increased discharge capacity.

RECOVERABILITY

Historically at Niobec, the process recovery varies with the Nb_2O_5 head grade while the converter recovery is constant and equal to 97%.

Within the mineable ore tonnage delineated in the pit shell under assessment, the average ore grade implies a 49.6% mill recovery on average. Typical historic head grades vary between 0.35% and 0.70% Nb_2O_5 which resulted in respective mill recovery of 40% and 60% at grade limits. On a unit mining block basis, the pit optimization and production schedule were performed using the following equation to correlate Nb_2O_5 grade and process recovery. When Nb_2O_5 grade is below 0.35% or over 0.70%, the mill recovery is therefore the minimum (40%) or the maximum (60%) values achieved in the past.

$$y = -16{,}310\,x^2 + 227.08\,x - 0.1944$$

where: y is mill recovery

x is Nb_2O_5 grade (0.5% Nb_2O_5 being 0.005 in the formulae)

MARKETS

The primary use of niobium is as an alloy to strengthen steel for the automotive, gas pipeline and construction industries. Demand is driven almost entirely by growth in steel production, enhanced by specific increasing demand for high strength steel.

In 2000, world consumption of ferroniobium was 30.9 million kilograms, rising to an estimated 86 million kilograms in 2011. Consumption is mainly split equally between the EU, North America, and China, with a lesser amount to Japan.

Over 90% of world niobium production is in the form of standard grade (~66% niobium) ferroniobium produced from primary deposits in Brazil, Canada, and Australia. Over 95% of the production comes from three producers:

- Companhia Brasileira de Metalurgia e Mineração (CBMM)
- Mineração Catalão de Goias (Catalao), owned by Anglo American
- IAMGOLD Niobec

CBMM has historically been the largest producer by far, generally covering approximately 80% to 85% of the market. Niobec has historically covered 8% to 10% of the market, and Catalao somewhat less.

The ferroniobium market is very secretive, with public and transparent pricing forums non-existent. Commensurate with the demand increase, prices have risen from the area of US$ 15 per kg contained niobium in 2006 to over US$ 40 per kg currently.

For the purposes of this study, RPA has assumed the following for market assumptions:

- Annual consumption at 10% growth for the first four years and then declining to a long term annual growth of 3.5% commencing in 2020 (the latter based on standardized forecasted Compound Annual Growth Rate (CAGR));
- Long term price of US$ 45 per kilogram niobium;
- Niobec forecast annual production at full capacity of 14 million to 18 million kilograms of niobium.

References: World Steel Association, Roskill, IAMGOLD, CBMM Website

CONTRACTS

Niobec's FeNb marketing and sales around the world is contracted to a specialized firm. Transportation cost of FeNb to customers, added to marketing and sales cost over the Project LOM, total C$ 875 million. Details of the agreement are confidential.

ENVIRONMENTAL CONSIDERATIONS

INTRODUCTION

The environmental management system (EMS) for the Niobec mine is certified under the 2004 revision of the ISO 14001 standard. Niobec successfully passed the ISO 14001 recertification audit in November 2010. Niobec's quality management system is certified ISO 9001: 2008 since 1995; it was last recertified in 2009.

Since 2008, samples of Niobec's final effluent are submitted to the Daphnia magna toxicity test. This test, required under provincial regulations, is frequently failed. After many studies, IAMGOLD believes that Niobec's final effluent toxicity issue will be solved by using the Shipshaw River as the source of the freshwater supplies to the mill and discharge the final effluent into this large water body. IAMGOLD are very confident that this option will resolve both the toxicity and freshwater supply issues. Niobec continues to work diligently on obtaining all the authorizations to begin the construction in spring of 2011.

Since 2006, Niobec exceeded the total suspended solids (TSS) monthly average concentration allowed in the MMER (Metal Mining Effluent Regulations and Environmental Effects) during several months each year. A new sedimentation pond, built in 2008 is effective, but not enough to prevent the proliferation of algae in the presence of phosphor, which is the cause of the TSS. On October 28, 2010 IAMGOLD received a directive, in accordance with the Fisheries Act, from the enforcement branch of Environment Canada regarding intermittent exceedances of TSS. Sedimentation fences were installed and an active treatment unit will be used as contingency to achieve consistent compliance. The TSS issue will be solved through the Shipshaw River project and various TSS control measures (i.e., wastewater treatment plant, silt curtains, use of coagulants and ultrasound, etc.). IAMGOLD provided a detailed management plan to Environment Canada without delay and IAMGOLD has committed to take all the necessary measures to be in compliance with the MMER.

LEGAL REQUIREMENTS - HEALTH, SAFETY, ENVIRONMENT AND COMMUNITY

In terms of environmental requirements, Niobec mine must comply with both federal and Québec provincial laws and regulations directives. The main laws Niobec has to comply with are the Canadian Environmental Protection Act (CEPA) and the Environmental Québec Act (EQA). Numerous regulations and directives such as the MMRE and "Directive 019" also apply.

It must be noted that all expansion, construction and major modification projects in Québec need approval by the provincial government through the environmental permitting process. Depending on the nature of the project, many provincial governmental agencies may be involved in the permitting process. Typically, the project approval is done through the delivery of a certificate of authorization emitted by the Québec department of the environment - Minister of Sustainable Development, Environment and Parks (MDDEP). Niobec presently holds numerous certificates. In 2011, the MDDEP will deliver to Niobec a document called "Attestation d'assainissement" which is equivalent to an environmental operation permit delivered in other jurisdictions such as Ontario. This permit will set out the environmental requirements that will apply to Niobec for the next five years. This document was available to the public from November 2010 until January 2011. This permit includes all certificates of approval emitted prior to 2011. For some projects, especially those requiring work in water bodies, the federal government may also be involved in the permitting process.

In terms of health and safety requirements, Niobec mine must comply with Québec's provincial laws and regulations. The main Health and Safety law in Québec is the "Loi sur la santé et la sécurité du travail". Numerous regulations such as the "Règlement sur la santé et la sécurité dans les mines" and the "Règlement sur la santé et la sécurité" also apply. The worker compensation regime in Québec is a no-fault system whereby an employer cannot be prosecuted should a fatality occurs on a work site. However, the newly revised Canadian Criminal Code considers that an employer or its agents (i.e. managers, supervisors, and employees) may be guilty of a criminal offence if it demonstrated that occupational health and safety is not managed in a due diligent manner.

The Québec Mining Act is another important legislation that determines how mines are to be developed, operated and closed. It also allows for mines to expropriate land owners in order to build utilities such as water lines and power lines. Despite this right, the expropriation process can take up to nine to twelve months and involves an approval by the provincial Council of ministers. The Mining Act, and the regulations under it, includes provisions that require mining companies to rehabilitate the areas affected by their activities. The provisions cover extraction activities, exploration activities that require earth work, and mine tailings sites. By law, companies are required to file a site rehabilitation plan and, provide financial guarantees.

Companies carrying out mining activities under the Mining Act must submit a reclamation plan to the Department of Natural Resources of Québec (MRNF). Following consultation with the MDDEP, the MRNF may approve the plan and its implementation schedule. The MRNF may if necessary, request additional research or studies before approving the plan. The closure plan must be submitted for approval to the department before work can begin. The closure plan must be revised every five years, but in certain cases the MRNF may require more frequent revisions.

In 2009, the MRNF made public its Mineral Strategy in which it defines the objectives to be reached and the measures to be taken for the future of the mineral sector in Québec. On December 2, 2009, Bill 79, which seeks to amend the Mining Act, was introduced by Québec's Minister for Natural Resources and Wildlife. Bill 79 proposes several significant amendments to the Mining Act, including the following:

- Under current rules, the financial guarantee is equivalent to 70% of the anticipated cost of rehabilitating accumulation areas. The guarantee must now cover 100% of the anticipated cost of rehabilitating accumulation areas, soil stabilization, stabilizing mine openings and surface pillars, water treatment and roadwork;

- Currently, the schedule of payment for the guarantee is established on the life of the mine basis. The proposed rules will require that the entire guarantee be provided in five annual payments;

- Under the EQA, a public consultation, with an Environmental Impact Assessment (EIA) process and public hearings before the BAPE - Bureau d'audiences publiques en environnement - is required in the case of metal mine with a production capacity over 7,000 tpd. In its Mineral Strategy, the Government announced its intention to reduce the threshold for triggering an EIA process for metal mine to a production capacity over 3,000 tpd;

- The proposed changes will mean that all new mines will be subject to a public consultation process including those for which the EQA does not presently provide a public consultation and in particular metal mine below the capacity threshold mentioned above;

- Urban ban (RPA notes Niobec is not in an urban area).

The Minister announced in February 2011 that Bill 79 was suspended and that there is no plan to introduce a new mining bill for at least a year.

The most recent version of Niobec's closure plan was approved by the MRNF in September 2009. A revised rehabilitation plan has to be submitted to the MRNF by September 2014. A company that expects to use or that is already using an area must provide the MRNF with a financial guarantee once its rehabilitation plan has been approved. The amount of the guarantee must cover 70% of the estimated closure cost and the accumulation areas (i.e. waste rock dumps, tailings pond, etc.). Under the mining reform, the guarantee will be a 100% in the future. Niobec has no financial guarantee in place, based on the life of the mine, but the first installment ($7,280) is scheduled for September 2011.

Once the rehabilitation work has been completed in accordance with the approved plan, and it has been confirmed that there is no future risk of acid mine drainage at the site, the MRNF will issue a certificate stating that the company is released from its obligations. The same certificate will be issued if a third party agrees to take responsibility for rehabilitation.

In terms of community relations, Niobec mine is also complying with the Québec's Civil Code which describes how neighbours interact and how contracts are to be managed.

ENVIRONMENTAL AND SOCIAL IMPACT ASSESSMENT / STATUTORY REQUIREMENTS

Depending on the nature of the project, a detailed environmental and social impact assessment may be required in order to comply with the provincial “Règlement sur l’évaluation et l’examen des impacts sur l’environnement”. At this time, the environmental and social impact assessment process would be triggered if the production rate of a mineral processing plant is greater than 7,000 tpd or if a mine is opened and operated with a production rate greater than 7,000 tpd. As stated above, with the reform, the threshold for triggering an EIA process for metal mine would be lowered at 3,000 tpd. The cornerstone of this process is the environmental and social impact study, which is conducted by a consulting firm, based on the requirement set out by the Québec’s environment department.

The environmental and social impact assessment is typically followed by public hearings conducted by the “Bureau des audiences publiques sur l’environnement” (BAPE). The whole assessment, public hearing and subsequent project analysis by the provincial government may take up to 15 months. The cost for the EIA and the whole assessment is estimated at C$ 1M.

BASELINE STUDIES

No baseline study was conducted at Niobec prior to 1994. In December 1994, Niobec prepared its first mine closure plan for the benefit of the provincial government. This plan contains a baseline study describing the environmental conditions, the surrounding community, and the existing infrastructures.

RISK ASSESSMENT

A formal risk assessment for the Project will be conducted through three distinct processes:

- The environmental and social impact assessment required for the permitting process;
- The IAMGOLD risk assessment process;
- The IAMGOLD Safety in Design study.

All likely health, safety, environmental and community risks and impacts and risk management programs would be identified during the course of those processes.

The main foreseeable risks and impacts associated with the Project are briefly described in the following paragraphs.

HEALTH AND SAFETY RISK ASSESSMENT

The health and safety risk assessment will be conducted through the IAMGOLD risk assessment process and the Safety in Design study.

ENVIRONMENTAL AND COMMUNITY RISK ASSESSMENT

The environmental and community (social) risk assessment will be conducted through the environmental and social impact assessment required by the provincial permitting process and obviously through the IAMGOLD risk assessment process.

POTENTIAL HEALTH, SAFETY, ENVIRONMENTAL AND COMMUNITY IMPACTS

A preliminary risk assessment cession was held with the Niobec Staff in order to identify the major health, safety, environmental and community impacts. For the PEA purposes, the main foreseeable risks and impacts associated with the Project have been identified and are briefly described in the following paragraphs.

Increased health and safety risks will be associated with infrastructure construction activities as well as infrastructure demolition activities. Some areas of the processing plant may exceed the radioactivity regulation and require a particular dismantling and disposition plan. Opening new surface excavations may lead to a release of radon. These risks will be mitigated by the existing health and safety management programs.

The OP scenario (1,140 tph) would trigger the provincial environmental and social impact assessment and public hearing process as the final daily production rate would be much higher than the 7,000 tpd threshold.

Due to the open pit foot print, the existing surface infrastructure would have to be demolished. This scenario could be considered a partial existing mine closure thereby partially activating the Asset Retirement Obligation (ARO) plan.

As new tailings storage facility (approximately 400 Mt capacity) is required, very significant land purchase is necessary. Assuming a design similar to tailings pond No. 2 with decantation of water to a collecting/polishing pond, the new tailings pond will have a surface area 6.5 km² and roughly be 43 m high (Figure 18-7). All lands required for this new impoundment have been estimated at a flat rate and assumed no impediments in acquiring the land. Construction of the tailings pond will borrow from methods of construction and operation based on the existing No. 1 and No. 2 tailings ponds which have proved effective. The waste rock dump also has to be expanded: a surface area of 12 km² and a height of approximately 76 m will be required to accommodate the planned material.

Depending on how the land purchase negotiation process goes, it may be required to expropriate some land owners, which is unsuitable. The surrounding community may be concerned with the expanding mine footprint and infringement on actual or potential agricultural land. A good communication and community relation plan will be a key to success of this process.

New water supply and effluent discharge lines are required. In order to install those new water lines, it may be necessary to expropriate few lands owners whose lots are located between Niobec and the Shipshaw River. The freshwater supply to the expanded mill and infrastructure would increase significantly. The future freshwater source, the Shipshaw River, can easily supply the required amount of water for the production. As the mine effluent is discharged back into this river, a net positive impact is expected in terms of water balance, meaning the amount of water returned to the river will be equal or greater than the amount pumped from the river. In order to allow efficient control of the total suspended solids load in the final effluent, additional retention basins and/or a wastewater treatment plant may need to be constructed. Additional septic tanks and leachate fields will be required as well to account for a potential increase in the discharge of domestic wastewater. At this point in time, no change in effluent discharge point is expected.

The hazardous materials and reagents consumption is expected to increase as the hazardous waste (i.e., waste oil, grease and lubricants, etc.) and non-hazardous waste generation (i.e. cardboard, paper, plastic, etc.). The same holds true for the energy consumption and the greenhouse gas emissions.

Increased health and safety risks will be associated with infrastructure construction activities as well as shaft demolition activities. Opening new surface excavations may lead to a release of radon. These risks will be mitigated by the existing health and safety management programs.

In order to minimize the community impact associated with material transport to the mine site (potential for hazmat spill, noise, dust, etc.), an opportunity exists to build a new access road leading from a major road (i.e., boulevard Martel or boulevard Ste. Genevieve) directly to the mine site, away from residential areas.

The open pit itself and the waste rock dumps may constitute a significant environmental and community impact. It may be a safety and/or visual issues issue as well as the open pit and waste rock dump area would need to be secured. A safety management plan would need to be developed to account for these new risks.

MANAGEMENT PLANS

As discussed above, the overall management program at Niobec is ISO 9001:2008 certified. The health, safety, environmental and community management plans and programs are described below.

HEALTH, SAFETY, ENVIRONMENTAL AND COMMUNITY MANAGEMENT PLANS

Niobec's environmental management program is ISO 14001:2004 certified. This program contains numerous policies and procedures dealing with anything from biomedical waste management to spill response. The environmental management program is audited on a yearly basis, both internally by environmental professionals from other IAMGOLD operation and externally by certified auditors.

Niobec's water management plan is part of the overall environmental management plan. Currently, the freshwater required for the milling process is pumped from an aquifer located eight km east of the mine site. However, about 85% of the water consumed at the mill is reclaimed from the tailings pond. Due to water demand increase, Niobec is presently building a pumping stations and water lines to get its freshwater from the Shipshaw River located 10 km west from the mine. Once, the project is completed, Niobec will reduce its pumping rate in the aquifer by 90%, which is a very positive

community benefit as the aquifer is also used by two municipalities as drinking water supply.
The final effluent of the mine site is presently discharged into the Cimon creek, which runs about seven km before joining up with the Aux-Vases River. This river reaches the Saguenay River six km further south. The final effluent average flow rate is 8,500 m³/d and is composed of two sub-effluents, the tailings sub-effluent and the mine dewatering sub-effluent. The tailings sub-effluent typically accounts for 80% of the final effluent flow. The water is considered non-toxic but still very hard. It contains traces of phosphorous, which may cause occasional algae bloom (leading to an increase TSS load). The mine water is also very hard water and typically fails the acute toxicity Daphnia magna test due to its saline nature. Both the algae bloom and toxicity issues will be resolved in 2011 through the Shipshaw River project and various TSS control measures (i.e. wastewater treatment plant, silt curtains, use of coagulants and ultrasound, etc.). It should be noted that once the Shipshaw River project is completed in 2011, the final effluent will be discharged into this river, thereby guaranteeing a positive water balance in this ecosystem.

Niobec' health and safety management program is composed of two distinct programs and numerous associated procedures:

- Safety prevention program; and
- Occupational health program.

Those programs, or parts of those programs, are audited yearly by safety professionals from other mine sites in Québec, through the Québec Mining Association audit program. Niobec is presently developing a health and safety management program, based on the OHSAS 18001 and the CSA Z1000 recognized standards. The plan is to fully implement the health and safety management program by September 2012.

Various significant aspects of health, safety, environmental and community management programs are audited internally every year through audit protocols set out by the Canadian Mining Association towards Sustainable Mining (TSM) program.

CLOSURE PLAN

In 1994, Niobec presented its first closure plan to the provincial government. This plan was approved by the MRNF. The most recent version of Niobec's closure plan was approved by the MRNF in September 2009. Also, as per IAMGOLD's policies, Niobec prepares every year an Asset Retirement Obligation (ARO) plan. This plan describes in details how the mine site is to be reclaimed, cleaned up, secured, and revegetated once the mine closes. In the 2010 version of the ARO plan, it was estimated that it would cost C$ 7.7 million to totally reclaim and restore the Niobec site.

GENERAL ENVIRONMENTAL SITE AND REGIONAL CONDITIONS

COMMUNITY

Niobec is located in the Saguenay-Lac-St-Jean region in Central Québec. It is a highly industrial region where many aluminum smelters and associated facilities are installed. They are owned and operated by Rio Tinto Alcan. The lumber and paper milling industry is also very present in this area. Niobec is the only active mine in the region.

The region is accessible through a deep-waterway, the Saguenay River which is linked to the St. Lawrence-Great Lakes waterways, a well developed road network, a railroad network and many airports, the largest one being located the Bagotville-Saguenay Airport (located at the Canadian Air Force base Bagotville). The Saint-Honoré airport is located about eight km East of Niobec. In 2006, more than 275,000 people lived in the Saguenay-Lac-St-Jean region.

More precisely, Niobec is located in the municipality of Saint-Honoré de Chicoutimi. About 5,000 people lives in this municipality. Niobec is the largest industry and employer in Saint-Honoré. Around Niobec, the population density is low with the closest house being located about 1.5 km north of the mine site. The surrounding land is mostly used for agriculture (mostly potato, canola and blueberries) but most of the land is still natural or replanted forest.

In general, the relations between Niobec and the municipality and the community in general are excellent. Niobec receives on average one complaint per year, due to dust generated by wind erosion on the tailings storage facilities. Since 2010, Niobec is equipped with a vehicle that controls the dust using water. In 2010, no complaint was received.

The closest aboriginal reserve, Mashteuiash, is located about 100 km west of Niobec.

CLIMATE

The climate in the Saguenay-Lac-St-Jean region is humid temperate with mild summers. The annual average temperature in the region is 2.3^{o}C with monthly average ranging from -16.1^{o}C in January to 18.1^{o}C in July. On average, the total precipitations are 950.8 mm per year, including 341.6 cm falling in form of snow.

FLORA AND FAUNA

The vegetation in the Niobec area is mostly forest dominated by coniferous three species such as black spruce, fir tree and larch tree. There is also some poplar tree and grey pine. The average vegetation density ranges from 40 to 80%. There is no known plant or tree species at risk around the Niobec area.

The fauna found in the Niobec area is typical of the Saguenay-Lac-St-Jean region. Red fox, rabbits, squirrels, black bears, partridges, black crows, Canada geese and other types of birds and mammals are found in the area. No significant water bodies are found at the mine Niobec. However, a river, the “Rivière-aux-Vases” is located about two kilometres west of Niobec. There is no known animal species at risk around the Niobec area.

ARCHAEOLOGICAL SITES

There is no archaeological site identified around the Niobec mine or the area of influence of the Project

TAXES

RPA has relied on IAMGOLD for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Niobec Open Pit.

CAPITAL AND OPERATING COST ESTIMATES – OPEN PIT

PRE-PRODUCTION CAPITAL

The mine, mill and site infrastructure costs are summarized in Table 18-3. All costs in this section are in 2011 US$.

TABLE 18-3 CAPITAL COST SUMMARY
IAMGOLD Corp. – Niobec Mine

Area	Cost (US$ M)
Mine	187.9
Mill, Crusher and Converter	245.0
Infrastructure	59.6
Waste rock dump water management	30.3
Tailings and Process water management	67.6
Indirects	134.5
Contingency (≈15% on average)	104.6
Total Pre-Production Capital	**829.5**

Mine capital cost includes mining equipment fleet purchases and pre-stripping and site work related to the open pit. The mine fleet was estimated based on OP operations of a similar scale. Most equipment costs were based on costs published by InfoMine USA Inc. Costs in 2009 US$ were escalated by 5% to convert them to 2011 US$. Mine services cost covers haul roads construction and site work related to open pit and waste rock dump preparation (stripping of vegetation and overburden). In addition, a provision was made for the first 12 Mt of waste rock stripping at the unit mining cost per tonne. In this particular project, capital expenditures were also required under the underground mine cost item for work enabled by the interaction of the open pit with the existing mine, namely a new ventilation system and distribution network (existing ventilation raises located into the pit footprint), an increase in pumping capacity to manage additional surface water flowing through the filled open stopes eventually connected with the open pit and the first drilling campaign of holes (contracted) to fill open stopes with crushed waste rock. Open pit and UG mine capital costs are shown in the Table 18-4.

TABLE 18-4 CAPITAL COSTS – MINE
IAMGOLD Corp.– Niobec Mine

Area	Item	Cost (US$ M)
Mine	Mobile Equipment	138.4
	Site Work	5.2
	Underground mine	23.7
	Pre-stripping	20.6
Total Mine		**187.9**

Ore processing capital cost was estimated by IAMGOLD based on the current process flow sheet and a scale-up of the existing mill and converter facilities. Detailed equipment lists were generated for each area within the ore processing cost item. Since annual production rates for the block caving and the open pit options are the same, the following cost details are relevant for both scenarios. Ore processing capital costs are shown in Table 18-5.

TABLE 18-5 CAPITAL COSTS – ORE PROCESSING
IAMGOLD Corp. – Niobec Mine

Area	Item	Cost (UD$ M)
Pre-Mill	Ore Handling to Crushing	4.7
	Primary Crusher	12.9
Mill	Ore Handling	9.7
	Grinding	46.9
	Desliming	7.8
	Flotation Pyrite	16.4
	Flotation Carbonates	12.7
	Dewatering	7.8
	Magnetic Separator	1.0
	Flotation Pyrochlore	24.0
	Flotation Sulphide	13.5
	Leaching	2.1
	Leaching filtration	4.8
	Filtration	0.9
	Drying	1.8
	Packaging	10.8
	Tailings	7.0
	On stream analyzer	0.3
	Reagents	7.6
	Reagents Storage	2.6
	Air services	1.2
	Mill Office	1.4
Converter	Fusion	43.7
	Concentrate Storage	3.4
Total Ore Processing		**245.0**

Infrastructure cost was estimated by IAMGOLD and includes general site preparation, construction of on-site roads, upgrade of a provincial road as main access to the industrial mine site and others. Also under this cost item are buildings construction, equipment and furniture, power distribution, fuel storage and distribution, fire protection and laboratory. Infrastructure capital costs are shown in Table 18-6.

TABLE 18-6 CAPITAL COSTS - INFRASTRUCTURE
IAMGOLD Corp. – Niobec Mine

Area	Item	Cost (US$ M)
Infrastructure	General	21.7
	Electrical Line 161 kV	0.3
	Main Substation 161 kV	6.8
	Plant Main Substation	1.4
	Distribution Network 25 kV	1.9
	Mine Administration, Service and Dry Building	7.8
	Mill Workshop	2.6
	Fuel Storage and Distribution	2.0
	Mine Equipment Shop	9.9
	Fire Protection	1.4
	Mill Service and Dry Building	0.6
	Laboratory	3.2
Total Infrastructure		**59.6**

Waste rock dump water management cost was provisioned based on IAMGOLD's approach to tailings water management regarding ditches and effluent water system. Effluent water system also considers open pit mine water. Budget allocation is included here for land purchases at waste rock dump location. Waste rock dump water management capital costs are shown in Table 18-7.

TABLE 18-7 CAPITAL COSTS – WASTE ROCK DUMP WATER MANAGEMENT
IAMGOLD Corp. – Niobec Mine

Area	Item	Cost (US$ M)
Waste rock dump water management	Effluent Water System and Land Purchase	29.1
	Ditches Excavation	1.2
Total		**30.3**

Tailings and process water management cost covers upgrade to fresh water intake system and pipeline from/to Shipshaw River to the west, effluent water system and tailings pond site preparation and construction. As for waste rock dump, a budget allocation is included here for land purchases at tailings pond location. Tailings and process water management capital costs are shown in Table 18-8.

TABLE 18-8 CAPITAL COSTS – TAILINGS AND PROCESS WATER MANAGEMENT
IAMGOLD Corp. – Niobec Mine

Area	Item	Cost (US$ M)
Tailings and Process water management	Fresh Water System	14.1
	Pipeline	13.9
	Effluent Water System	12.5
	Tailings and land purchase	27.1
Total		**67.6**

Indirect capital cost consists of working capital, warehouse inventory, owner's cost, mill start-up/commissioning and Engineering, Procurement, Construction Management (EPCM). Working capital is equivalent to a 2-month FeNb marketing and freight and total operating cost (detailed further in this report), while warehouse inventory was estimated to be 5% of mine, mill, crusher and converter direct capital costs. During the construction phase, owner's cost represents approximately 85% of a one-year total General and Administrative (G&A) operating cost (detailed further in this report) which consists of usual cost items such as administration, wages and salaries, office supplies, consultants cost, insurance, taxes, etc. Finally, EPCM cost varies between 15% and 20% of direct capital cost items and also includes construction temporary installations, equipment, tools, travel and lodging, for an average of almost 13% of total direct capital cost (not considering OP mine fleet). Indirect capital costs are shown in Table 18-9.

TABLE 18-9 CAPITAL COSTS - INDIRECTS
IAMGOLD Corp. – Niobec Mine

Area	Item	Cost (US$ M)
Indirects	EPCM	56.1
	Working Capital[1]	46.9
	Warehouse Inventory[1]	19.5
	Owner's cost	10.3
	Mill Start-up and Commissioning	1.7
Total Indirects		**134.5**

Note:
1. Amounts totalling US$ 66.4 million fully recovered in the last year of the LOM for cash flow analysis purposes.

A contingency of almost 15% on average was applied to direct capital costs for a total of US$ 104.6 M.

SUSTAINING CAPITAL

Sustaining capital costs are subdivided into two periods. First, during the initial 4-year pre-production period (or construction phase), from Year -4 to Year -1 (2011 to 2014 inclusively), costs related to existing underground mine expansion were labeled as sustaining capital. All are quoted in 2011 US$.

Second, from Year 1 to Year 38 of the production phase (2015 to 2052 inclusively), all capital costs occurring during this period were considered as sustaining capital. It consists of:

- Mine equipment fleet replacement;
- Crusher, mill and converter buildings maintenance and equipment replacement;
- Relocation of a portion of existing tails, once open pit reach existing tailings pond;
- Drilling campaigns of holes (contracted) to fill UG openings from open pit;
- Progressive rehabilitation and mine closure.

Sustaining capital costs are shown in Table 18-10.

TABLE 18-10 CAPITAL COSTS – SUSTAINING
IAMGOLD Corp. – Niobec Mine

Area	Item	Cost (US$ M)
Sustaining	Existing Underground Mine	91.3
	Mine Equipment Fleet	341.9
	Crusher, Mill and Converter	54.0
	Existing Tailings	30.0
	Holes to fill UG openings	4.5
	Progressive Rehabilitation and Mine Closure	47.5
Total Sustaining		**569.2**

EXCLUSIONS FROM PRE-PRODUCTION AND SUSTAINING CAPITAL

The following is excluded from the capital cost estimate:

- Project financing and interest charges;
- Escalation during the project;
- Permits, fees and process royalties;
- Pre-feasibility and Feasibility studies;
- Environmental impact studies;

- Any additional civil, concrete work due to the adverse soil condition and location;
- Taxes;
- Import duties and custom fees;
- Cost of geotechnical and geomechanical investigations;
- Rock mechanics study;
- Metallurgical testwork;
- Exploration drilling;
- Costs of fluctuations in currency exchanges;
- Project application and approval expenses.

OPERATING COSTS

Unit costs for production for the LOM are summarized in Table 18-11. Unit costs are detailed for the whole Project and for the OP project only because the existing UG mine will still be producing at 2.2 Mtpa during the four-year construction phase of the Project. All costs in this section are in 2011 C$.

TABLE 18-11 LOM AVERAGE UNIT OPERATING COST SUMMARY
IAMGOLD Corp. – Niobec Mine

	UG&OP C$/t milled	UG&OP C$/t moved	UG C$/t milled	OP C$/t milled	OP C$/t moved
Mining	9.31	1.90	19.54	9.07	1.82
Processing	9.70		13.45	9.50	
Converting	6.31		9.57	6.23	
G&A	1.35		4.03	1.25	
Total	**26.67**		**46.59**	**26.05**	

Details of mine operating costs for OP production only are shown in Table 18-12.

TABLE 18-12 MINE OPERATING COST DETAIL
IAMGOLD Corporation – Niobec Open Pit Project

	C$/t milled	C$/t moved
Mining	8.95	1.80
UG Material Handling into Open Pit	0.02	0.00
Waste rock dump & OP water management	0.10	0.02
Total Mine	**9.07**	**1.82**

The unit operating cost for open pit mining was estimated based on open pit operations of a similar scale. Annual allocations of C$ 200,000 and C$1,000,000 were made respectively to handle UG mine rock bolts, cables, steels and other materials once encountered in the open pit and for treatment of mine water and waste rock dump runoff water.

Operating costs have been derived from the actual and historical cost. Adjustments were done taking into account the tonnage increase, additional manpower and work schedule changes. Most of the costs are variable and will be the same on a cost per tonne basis independently of the scenario throughput. Details of mill operating costs considering open pit production only, fully provided by IAMGOLD are shown in Table 18-13.

TABLE 18-13 MILL OPERATING COST DETAIL
IAMGOLD Corp. – Niobec Mine

	C$/t milled
Processing	8.87
Laboratory	0.51
Environment and Tailings dam raising	0.12
Total Mill	**9.50**

For confidentiality reasons, the above unit costs will not be more detailed. Summarily, the processing unit cost covers manpower for operation, for maintenance and for services, and also includes training, power, reagents, heating, material and services, and contractor services.

The converting cost was estimated by IAMGOLD to C$ 4.00/kg Nb or C$ 6.23/t milled, including slag hauling and disposal into the waste rock dump. As for the processing cost, the converter operating cost accounts for the same cost item and no more details will be revealed for the same reasons.

The G&A unit cost was estimated by IAMGOLD and totalled C$ 12.5 M per year or C$ 1.25/t milled (basis 10 Mtpa). This is equivalent to a 40% increase compare with current annual budget of C$8.9 M. Administrational costs and site services costs comprise the G&A. Salary for administrational, engineering, geology, supervision and surface maintenance staff accounts for a large percentage of administrational costs; the remaining is for office supplies, consultants, insurance and taxes. Site services include water supply, sewage treatment and waste disposal.

MANPOWER

Manpower estimates were based on typical numbers in Canadian OP operations of a similar scale, and IAMGOLD scaled-up manning based on current data for milling, converting and G&A. Most of the operating basis will be the same as the current. The grinding circuit will return to a single circuit compared to the current circuit. The new design with bigger equipment will optimize the efficiency in term of operations management. In this scenario, it was anticipated that the operation of the 10-Mtpa plant will require an equivalent of a 50% increase in manpower compared to the existing level for the mill and 80% for the converter based on niobium production.

Manpower estimates for the various administrative units are shown in Table 18-14 while representative salaries are shown in Table 18-15.

TABLE 18-14 MANPOWER SUMMARY
IAMGOLD Corp. – Niobec Mine

Unit	Operation	Maintenance	Supervision and Services	Total
Administration	---	---	46	46
Mine	225	115	20	360
Mill	100	60	20	180
Converter	68	8	5	81
Surface and Electrical	---	65	11	76
Total	**393**	**248**	**102**	**743**

TABLE 18-15 REPRESENTATIVE SALARIES
IAMGOLD Corp. – Niobec Mine

Position	Annual Salary ($C)
Mine Manager	190,000
Superintendents	140,000
Foremen	115,000
Planning Engineers	90,000
Electricians	85,000
Mechanics	80,000
Mill Operator	75,000
Equipment Operator	75,000

ECONOMIC ANALYSIS

In this section, the open pit is presented as the base case with comparison to the block caving scenario.

Pre-tax and after-tax Cash Flow Projections have been generated from the LOM production schedule and capital and operating cost estimates, and are summarized in Table 18-16. A summary of the key criteria is provided below.

ECONOMIC CRITERIA

REVENUE

- 10 million tonnes milled per year.
- Nb_2O_5 mill recovery is between 40% and 60%, for a LOM average of 49.6%.
- FeNb converter recovery is 97% (with 69.9% Nb in FeNb).
- Marketing and freight totals US$1.40/kg Nb.
- Exchange rate at US$1.00 = C$1.05 from 2014 on, US$1.00/C$ before 2014.
- Niobium price is US$40.00/kg in 2011 and US$45.00/kg thereafter.
- LOM average unit net value is US$43.58/kg.

COSTS

- Capital cost totals US$1.4 billion, excluding the recovery of US$66.4 million for working capital and warehouse inventory at the end of the LOM.
- Average operating cost over the mine life is US$25.44 per tonne milled.

OTHER

- Pre-production period spans over four years (2011 to 2014).
- During pre-production period, current underground mining continues at 2.2 Mtpa.
- Open pit production period is 38 years, up to 2052.

- LOM stripping ratio is 3.91.

The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the reserves development, production, and economic forecasts on which this preliminary economic assessment is based will be realized.

TABLE 18-16 PRE- AND AFTER-TAX CASH FLOWS SUMMARY
IAMGOLD Corp. – Niobec Mine

	Units	Total	2011 to 2014	2015 to 2052
Mine Production				
Waste Mined	Mt	1,481.67	12.00	1,469.67
Ore Mined	Mt	379.03	8.80	370.23
Total Mined	**Mt**	**1,860.70**	**20.80**	**1,839.90**
Waste to Ore ratio		3.91		
Mill Feed				
Ore milled	Mt	379.03	8.80	370.23
Milled ore grade	%Nb_2O_5	0.463	0.603	0.460
Mill recovery	%	49.6	58.3	49.4
Yield	kg Nb_2O_5/t	2.32	3.51	2.30
Concentrate production	M kg Nb_2O_5	880.61	30.93	849.67
Converter recovery	%	97.0%	97.0	97.0
Niobium in FeNb	%	69.9	69.9	69.9
Niobium production	M kg	597.08	20.97	576.10
Revenue				
Niobium production	M kg	597.08	20.97	576.10
Niobium Market Price	US$/kg		43.90	45.00
Gross Revenue	US$ (M)	26,845.45	920.75	25,924.70
Exchange Rate	C$/US$		1.01	1.05
Gross Revenue	C$ (M)	28,153.96	933.02	27,220.94
Marketing and Freight	C$ (M)	875.63	31.62	844.01
Net Revenue	C$ (M)	27,278.33	901.40	26,376.92
Operating Costs				
Mining	C$ (M)	3,483.81	171.99	3,311.82
UG Material Handling Into OP	C$ (M)	7.60	0.00	7.60
Stockpile Reclaiming	C$ (M)	0.00	0.00	0.00

	Units	Total	2011 to 2014	2015 to 2052
Waste Rock Dump & OP Water Management	C$ (M)	38.00	0.00	38.00
Processing	C$ (M)	3,676.25	118.39	3,557.86
Tailings pond	C$ (M)	0.00	0.00	0.00
General and Administration	C$ (M)	510.36	35.36	475.00
Converter	C$ (M)	2,388.67	84.25	2,304.42
Slag Hauling and Disposal	C$ (M)	4.18	0.00	4.18
Total Operating Cost	**C$ (M)**	**10,108.86**	**409.99**	**9,698.88**
Operating Cost Per Tonne Milled	(C$/t milled)	26.67	46.59	26.20
Operating Margin	**C$ (M)**	**17,169.46**	**491.42**	**16,678.05**
Operating Margin	**US$ (M)**	**16,368.33**	**484.47**	**15,883.86**
Capital Costs				
Underground mine	US$ (M)	91.32	91.32	0.00
Open pit mine	US$ (M)	506.08	164.19	341.89
Mill and Crusher	US$ (M)	246.47	197.87	48.60
Converter	US$ (M)	52.50	47.10	5.40
Buildings and Services	US$ (M)	27.55	27.55	0.00
Power distribution	US$ (M)	10.36	10.36	0.00
Tailings and Water management	US$ (M)	97.57	67.57	30.00
Waste rock dump & OP water management	US$ (M)	30.31	30.31	0.00
General and Infrastructure	US$ (M)	21.73	21.73	0.00
Working capital	US$ (M)	0.00	46.92	- 46.92
Warehouse inventory	US$ (M)	0.00	19.43	- 19.43
Owner's cost	US$ (M)	10.30	10.30	0.00
Open pit vs Underground mine	US$ (M)	28.22	23.68	4.54
Progressive rehab. & Mine closure	US$ (M)	47.50	0.00	47.50
Eng. and Const. management	US$ (M)	57.83	56.13	1.70
Contingency	US$ (M)	104.63	104.63	0.00
Total capital costs	**US$ (M)**	**1,332.35**	**919.07**	**413.28**
Pre-Tax Cash Flow				
Cash flow	US$ (M)	15,035.98	-434.60	15,470.58
After-Tax Cash Flow				
Tax Rate (38.6%)				
Tax Payable	US$ (M)	5,778.03	125.14	5,652.89
Cash flow	US$ (M)	9,257.94	-559.74	9,817.69
Project Economics				
Pre-Tax NPV (5%)	US$ (M)	5,364.42		
Pre-Tax NPV (8%)	US$ (M)	3,276.05		

	Units	Total	2011 to 2014	2015 to 2052
Pre-Tax NPV (10%)	US$ (M)	2,449.95		
Pre-Tax Internal rate of return	%	NA		
Pre-Tax Payback period (years)	Years	1.6		
After-Tax NPV (5%)	US$ (M)	3,251.49		
After-Tax NPV (8%)	US$ (M)	1,952.11		
After-Tax NPV (10%)	US$ (M)	1,437.91		
After-Tax Internal rate of return	%	47.5		
After-Tax Payback period	Years	2.2		

CASH FLOW ANALYSIS

Considering the Project on a stand-alone basis, the undiscounted pre-tax cash flow totals US$15.0 billion over the mine life, and simple payback occurs near the mid-point of 2016 (approximately 19 months from start of OP production). On an after-tax basis, the undiscounted cash flow is US$9.3 billion and the payback occurs at the beginning of 2017 (approximately 26 months from start of open pit production).

The Total Cash Cost is US$16.15 per kg of niobium. The mine life capital unit cost is US$2.35 per kg, for a Total Production Cost of US$18.50 per kg of niobium. Average niobium production during underground and open pit operations is 14.2 million kg per year.

The pre-tax NPV at an 8% discount rate is US$3.3 billion and the after-tax NPV is US$2.0 billion, and the after-tax internal rate of return (IRR) is 47.5%.

Table 1-2 is a comparison of the economic impact of the 10 Mtpa open pit scenario and the 10 Mtpa block caving scenario to the current LOM plan.

TABLE 18-17 BLOCK CAVING VERSUS OPEN PIT
IAMGOLD Corp. – Niobec Mine

Mining Method	Block Caving	Open Pit
Tonnes processed (millions)[1]	380	380
Strip ratio	–	3.9
Estimated average annual mill throughput (t)	10,000,000	10,000,000
Estimated average annual production (M kg Nb)	13	15
Operating margin after expansion (US$/kg Nb)[2]	28	28
Estimated capital expenditure (US$ M)	1,400	1,400
Initial capital[3]	840	830
Sustaining capital[4]	560	570
Pre-tax NPV 8% (US$ billions)[5]	2.7	3.3
After-tax NPV 8% (US$ billions)[5]	1.6	2
After-tax project IRR[6]	21%	24%

Notes:
1. Although the resources are calculated down to Block 6, the NPV under both scenarios only assesses the economic impact of the first 42 years.
2. Based on a US$ 45/kg niobium price. The current margin is US$ 18/kg based on a US$ 40/kg niobium price.
3. Includes working capital, warehouse inventory and C$ 20 M of capital for pre-stripping (in the OP scenario).
4. Includes the sustaining capital for the current operation and the sustaining capital required for the expansion.
5. Assumed niobium prices per kilogram are US$ 40/kg in 2011 and US$ 45/kg thereafter as estimated by an independent source to RPA.
6. Points of percentage above the IRR of the current 2.2-Mtpa scenario.

SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Head Grade;
- Mill Recovery;
- Niobium Price;
- Operating Costs;
- Capital Costs;
- Exchange Rate.

The after-tax NPV (8%) sensitivity over the base case has been calculated for -20% to +20% variations. The sensitivities are shown in Figure 18-8 and Table 18-18.

FIGURE 18-8 NIOBEC OPEN PIT PROJECT AFTER-TAX SENSITIVITY GRAPH

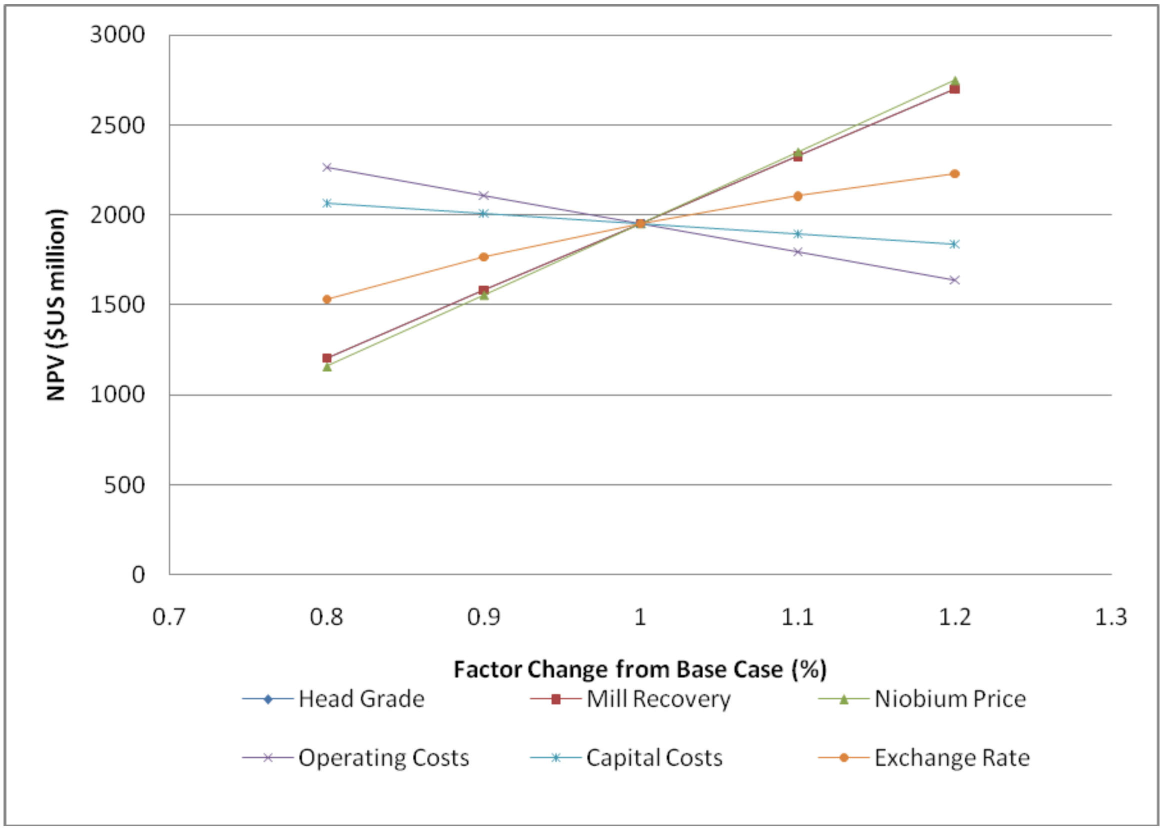


TABLE 18-18 AFTER-TAX SENSITIVITY ANALYSIS
IAMGOLD Corp. – Niobec Mine

Parameter Variables	Units	-20%	-10%	Base	+10%	+20%
Head Grade	% Nb_2O_5	0.37	0.42	0.46	0.51	0.56
Mill Recovery	%	40	45	50	55	60
Niobium Price	US$/kg	36.00	40.50	45.00	49.50	54.00
Operating costs	US$/t milled	20.35	22.90	25.44	27.98	30.53
Capital Costs	US$ billions	1.12	1.26	1.40	1.54	1.68
Exchange Rate	$C/$US	0.84	0.94	1.05	1.15	1.26

NPV 8%	Units	-20%	-10%	Base	+10%	+20%
Head Grade	US$ billions	1.21	1.58	1.95	2.33	2.70
Mill Recovery	US$ billions	1.21	1.58	1.95	2.33	2.70
Niobium Price	US$ billions	1.16	1.56	1.95	2.35	2.75
Operating costs	US$ billions	2.27	2.11	1.95	1.80	1.64
Capital Costs	US$ billions	2.06	2.01	1.95	1.90	1.84
Exchange Rate	$C/$US	1.53	1.77	1.95	2.10	2.23

The sensitivity analysis shows that the Project is most sensitive to head grade, mill recovery and niobium price, moderately sensitive to exchange rate and operating costs, and less sensitive to capital costs.

19 INTERPRETATION AND CONCLUSIONS

It is RPA's opinion that mining the Niobec deposit by open pit mining method may have considerable potential over the current underground method to extend mine life and better maximize the extraction of the resource. The Project base case scenario consists of technical and cost assumptions outlined in this report.

The Project will benefit from the expertise developed in the existing operation, including the processing and converting of niobium ore into ferroniobium over the years and the local and regional infrastructure currently used in the niobium mining/milling/converting activities at the mine site.

The most important risk elements for the Project are the niobium price and, at the present time, the location of waste rock dumps in areas with underlying soils capable of supporting the large tonnage of waste rock. Fluctuations in the niobium price constitute an uncontrollable parameter for which no mitigation measures are proposed. Other elements to which the Project is economically sensitive are controllable and will be addressed in further steps. Waste rock dump issues should be investigated as the Project progresses.

At the PEA level, the open pit scenario is technically feasible and economically viable and should be advanced to the pre-feasibility study level. The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the reserves development, production, and economic forecasts on which this preliminary economic assessment is based will be realized.

Specific conclusions are as follows:

- The combined Measured and Indicated Resources are 458.11 million tonnes grading 0.42% Nb_2O_5 containing 1,927 million kg Nb_2O_5. In addition, there is an Inferred Resource of 336.45 million tonnes grading 0.37% Nb_2O_5 containing

1,240 million kg Nb_2O_5. All these Mineral Resources are constrained by a Whittle open pit shell (6T) at a cut-off grade of 0.20% Nb_2O_5.

- A second economic optimization, which was time constrained, was performed on the 6T pit shell in order to maximize the NPV at 8%. The maximum NPV occurs at a cut-off grade of 0.33% Nb_2O_5. This resulted in a total of 370.2 million tonnes of niobium bearing material grading 0.46% Nb_2O_5 within a smaller pit shell, 4T (564 m deep). The proportion of Inferred Resources in the above tonnage that may be potentially mineable via open pit is approximately 20%.

- Ore production will be expanded from the current level of 2.2 million tonnes per year from underground to 10 million tonnes per year from OP. The LOM of the OP scenario will be 42 years, consisting of four years of pre-production and 38 years of production. During the pre-production period, the existing underground mine will contribute, at the current mining rate, an additional 8.8 Mt grading 0.60% Nb_2O_5. At full production, ore will be mined at a rate of 27,400 tpd requiring the excavation of 109,000 tpd waste on average over the Project total duration.

- As part of the analysis of options, RPA and IAMGOLD reviewed an underground block caving scenario as an alternative to an open pit. At this level of study and assumptions, the underground scenario is a viable but slightly less attractive option than the open pit scenario.

- Project components and costs were developed to a ± 40% level of accuracy. Operating costs were estimated in 2011 Canadian dollars while capital costs were mostly estimated in 2011 US dollars.

- Capital expenditures, including 15% contingency and sustaining/ongoing investments, are estimated to be US$1.4 billion (excluding the recovery of US$66.4 million for working capital and warehouse inventory at the end of the LOM). This amount excludes taxes, duties, and inflation. Initial open pit related capital expenditures will amount to US$830 million; the remaining capital outlay is considered as ongoing investment that will occur during the production period or the operation of the existing underground mine (US$91 million) during the pre-production period.

- The total mine site and ore processing operating cost, for the period ranging from 2011 to 2052 inclusively, is estimated to be C$26.67/t milled. This includes mining, mineral processing and converting, maintenance, administration and human resources costs.

- At peak, manpower is estimated to be 743 individuals for the various administrative units, namely administration, mine, mill, converter, and surface and electrical.

- IAMGOLD and RPA developed the open pit mine plan based on a 3D block model built by RPA. RPA is of the opinion the resource model is conservative and has good potential to expand. The pit shell comprises many blocks that are outside the four main zones, in the mineralized carbonatite where limited to no information is available. A grade of zero has been assigned to those blocks.

- There are no current Mineral Reserves estimated for the open pit mining option at Niobec. Mineral Reserves will be assessed at the pre-feasibility stage of study.

- Open pit mining will be carried out along 564 vertical metres. For the pit size, production requirements, and recommended equipment fleet, RPA considers mining of 12 m benches and 33 m wide ramps, including ditches and safety berms, to be appropriate.

- Ore will be delivered to a gyratory crusher prior to a covered stockpile and then conveyed into the processing plant. The flowsheet is similar to the current plant, comprising crushing, grinding, desliming, flotation (carbonate, pyrochlore, and sulphide), dewatering, leaching, filtration, drying, and packaging. Concentrate is transferred into the converter where the ferroniobium (FeNb final product) is produced. The mill and the converter will be new installations to be constructed to handle the increased production and because the current buildings lie within the open pit footprint. The existing tailings impoundment facility will be expanded to accommodate the additional tonnage.

- Recovery rates are estimated to be 49.6% Nb_2O_5 at the processing plant (head grade dependent) and 97% at the converting stage. FeNb production will total 597 M kg over the LOM.

- Environment mitigation measures will be implemented, with particular emphasis on the open pit and waste rock dumps. Proper water control and treatment will be implemented in order to minimize volumes to release final effluent respecting provincial regulations.

- The conceptual closure plan aims at returning the site in a state compatible with the environment. Particularly, waste rock dumps and tailings pond will be subject to progressive rehabilitation and revegetation measures while the existing pit will naturally flood once operations cease.

- An economic evaluation was carried out by means of pre-tax and after-tax cash flow analysis expressed in 2011 constant US dollars. An exchange rate of US$1.00/C$1.05 was assumed. The average metal price forecast is US$45/kg niobium and was assumed to be constant for the duration of the Project, except for the first year of the LOM (2011) at US$40/kg Nb.

- Sunk costs (exploration and delineation drilling, metallurgical test work, feasibility study costs, Environmental Impact Assessment, etc.) are not included in the Project capital cost or financial evaluation.

- Total revenues generated by the Project are estimated to be US$27.3 billion, and operating cash flow before capital costs total US$10.1 billion. Using a discount rate of 8%, the Project has NPVs of US$3.3 billion pre-tax and US$2.0 billion after-tax.

- The after-tax IRR of the Project is 47.5%. The after-tax cumulative cash flow turns positive during the third year of the Project production phase and amounts to US$9.3 billion over the LOM. The capital payback period is 2.2 years.

- Sensitivity calculations were performed on the Project cash flow by applying a ± 20% variance on head grade, mill recovery, niobium price, operating costs, capital expenditures, and C$/US$ exchange rate. The sensitivity analysis shows that the Project is most sensitive to head grade, mill recovery, and niobium price, moderately sensitive to exchange rate and operating costs, and less sensitive to capital costs.

20 RECOMMENDATIONS

RPA makes the following recommendations:

- Initiate the preparation of pre-feasibility studies for both the block caving and the open pit options in order to compare scenarios at a higher level of definition and accuracy, given both returned quite similar economic results.

- Confirm the waste rock dump location.

- Initiate geotechnical assessments of underlying/surrounding overburden and soils at waste rock dumps, open pit, and tailings pond locations.

- Initiate geomechanical and rock mechanics assessments for open pit wall slope angle and stability, and effects on advancing pit bottom through existing underground openings.

- Initiate an environmental program that will result in the preparation of a document in support of permitting. The program should be comprehensive and include baseline work, acid base accounting, waste rock dump stability, tailings pond stability, open pit wall stability, radon and radiation emission, dust emission, water balance and hydrology, salt contamination of water being exposed to rock in open pit and waste rock dumps and socio-economic studies.

- Initiate negotiations with individuals and governments concerning land purchases.

- Carry out further metallurgical testwork, using representative samples. The program would consist of completing test work to a typical standard for pre-feasibility study, including:
    - Crushing and milling (SAG work index) to properly design and select equipment.
    - Extensive metallurgical testwork to validate the niobium recovery and concentrate quality grade, given the lower head grade anticipated and the sensitivity of final product to impurities.
    - Niobium recovery improvement in investigating modifications to reagent scheme and process flowsheet, in considering previous testing on niobium recovery from the plant tailings and by benchmarking mill results with the laboratory results.
    - Comparative testwork between conventional grinding and classification presently used and new grinding circuit using SAG-ball mill.

- Carry out specific hydrological/hydrogeological studies to refine dewatering needs in the underground mine during pre-production period and in the open pit mine over the LOM.

- Carry out condemnation drilling in advance of mining in areas where waste rock dumps and the tailings storage facility are expected to be located for the operations.

- For the resource database:
  - Remove duplicate sample numbers from the drill database and investigate zero values for Nb_2O_5.
  - Investigate the large number of assays in the database without sample numbers, if possible, by cross referencing drill logs and assay certificates.
  - Study the use of a top cut for Nb_2O_5 assay results and consider adoption for future resource estimations.
  - Initiate the insertion of blanks and independent CRMs into the sample stream so that they comprise approximately 15% to 20% of the total determinations. RPA also recommends external laboratory check on sample rejects in addition to sample pulps.
  - Investigate the cause of the mild bias observed in pulp duplicates assayed at a second, independent, laboratory.
  - Study the use of equal length composites for resource estimation rather than raw samples.

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22 DATE AND SIGNATURE PAGE

This report titled "Technical Report on Expansion Options at the Niobec Mine, Québec, Canada" and dated June 17, 2011, was prepared and signed by the following authors:

(Signed & Sealed) "*Graham G. Clow*"

Dated at Toronto, Ontario
June 17, 2011

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Principal Mining Engineer

(Signed & Sealed) "*Bernard Salmon*"

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June 17, 2011

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(Signed & Sealed) "*Daniel Vallières*"

Dated at Longueuil, Quebec
June 17, 2011

Daniel Vallières, ing.
Manager, Underground Projects
IAMGOLD Corp.

23 CERTIFICATE OF QUALIFIED PERSONS

GRAHAM CLOW

I, Graham G. Clow, P.Eng., as an author of this report entitled “Technical Report on Expansion Options at the Niobec Mine, Québec, Canada”, prepared for IAMGOLD Corporation, and dated June 17, 2011, do hereby certify that:

1. I am Principal Mining Engineer with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave., Toronto, ON, M5J 2H7.

2. I am a graduate of Queen’s University, Kingston, Ontario, Canada in 1972 with a Bachelor of Science degree in Geological Engineering and in 1974 with a Bachelor of Science degree in Mining Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 8750507). I have worked as a mining engineer for a total of 35 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements
    - Senior Engineer to Mine Manager at seven Canadian mines and projects
    - Senior person in charge of the construction of two mines in Canada
    - Senior VP Operations in charge of five mining operations, including two in Latin America
    - President of a gold mining company with one mine in Canada
    - President of a gold mining company with one mine in Mexico

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Niobec Mine.

6. I am responsible for overall preparation of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.4 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th day of June, 2011.

(Signed & Sealed) "*Graham G. Clow*"

Graham G. Clow, P. Eng

...

BERNARD SALMON

I, Bernard Salmon, ing., as an author of this report entitled “Technical Report on Expansion Options at the Niobec Mine, Québec, Canada”, prepared for IAMGOLD Corporation and dated June 17, 2011, do hereby certify that:

1. I am Principal Consulting Geological Engineer with Roscoe Postle Associates Inc. (RPA). My office address is Suite. 203, 170 Avenue Principale, Rouyn-Noranda, Québec, J9X 4P7.

2. I am a graduate of École Polytechnique, Montreal, Québec, Canada, in 1982 with a Bachelor of Science (Applied) in Geological Engineering.

3. I am registered as an Engineer in the Province of Québec, member of the Ordre des Ingénieurs du Québec (#36831) and I am designated as a Consulting Geological Engineer. I have worked as a geological engineer for a total of 28 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Mining geologist, Falconbridge Copper Corp., Opemiska Mine, 1982 to 1987.
    - Chief geologist, Minnova Inc., Ansil Mine, 1987-1992
    - Chief-Geologist and Technical Superintendent, Inmet Mining Inc., Troilus Mine, 1992-1997.
    - Chief-Geologist, Aur Resources Inc., Louvicourt Mine, 1997-2005.
    - Consulting Geological Engineer with RPA from 2005 to present.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Niobec property.

6. I am responsible for Section 17, and contributed to Sections 20, and 21 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Part 1.4 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 17th day of June, 2011.

(Signed & Sealed) "*Bernard Salmon*"

Bernard Salmon, Eng.

MARC LAVIGNE

I, Marc Lavigne, Eng., M.Sc., as an author of this report entitled “Technical Report on Expansion Options at the Niobec Mine, Québec, Canada”, prepared for IAMGOLD Corporation and dated June 17, 2011, do hereby certify that:

1. I am Senior Mining Engineer with Roscoe Postle Associates Inc. My office address is Suite 302, 1305 Boulevard Lebourgneuf, Québec, Québec, G2K 2E4.

2. I am a graduate of Université Laval, Québec, Québec, Canada, in 1987 with a B.A.Sc. in Mining Engineering, and in 1991 with a M.Sc. in Geostatistics.

3. I am registered as an Engineer in the Province of Québec, member of the Ordre des Ingénieurs du Québec (Reg.# 99190). I have worked as a mining engineer for a total of 22 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Specialist in geostatistics - project engineer, Roche Ltd Consulting Group, 1989 to 1995;
   - Project Manager, Roche Ltd Consulting Group, 1995 to 2006;
   - Senior Project Manager, Genivar Limited Partnership, 2006 to 2011;
   - Senior Mining Engineer, RPA from 2011 to present.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I visited the Niobec property on March 4, 2011.

6. I am responsible for Section 18 regarding Mining method, Mine design, Mining approach, Schedules and Mine equipment (Open pit option), related Capital and Operating costs and Economic analysis, and contributed to Sections 19, 20 and 21 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.4 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 17th day of June, 2011

(Signed & Sealed) "*Marc Lavigne*"

Marc Lavigne, M.Sc., ing.

BARRY MCDONOUGH

I, Barry McDonough, P.Geo., as an author of this report entitled “Technical Report on Expansion Options at the Niobec Mine, Québec, Canada”, prepared for IAMGOLD Corporation and dated June 17, 2011, do hereby certify that:

1. I am Senior Geologist with Roscoe Postle Associates Inc. My office address is Suite 388, 1130 West Pender Street, Vancouver, British Columbia, Canada V6E 4A4.

2. I am a graduate of McMaster University, Hamilton, Ontario, in 1986 with a B.Sc. degree in Geology.

3. I am registered as a Professional Geoscientist in the Province of British Columbia (Reg.# 30663). I have worked as a geologist for a total of 25 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Exploration Geologist for numerous companies for projects across Canada from 1986 to 1995.
   - Mine Geologist at Eskay Creek Mine, British Columbia for Barrick Gold Corp. from 1995 to 2005.
   - Senior Mine Geologist at CanTung Mine, Northwest Territories for North American Tungsten Corp. from 2005 to 2006.
   - Chief Geologist at Minto Mine, Yukon Territory for Capstone Mining Corp. from 2006 to 2008.
   - Consulting Geologist with RPA, Vancouver from 2008 to present.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Niobec property from March 21 to 22, 2011.

6. I am responsible for Sections 1 to 15, and contributed to Sections 19, 20 and 21 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.4 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 17th day of June, 2011

(Signed & Sealed) "*Barry McDonough*"

Barry McDonough, P. Geo.

PIERRE PELLETIER

I, Pierre Pelletier, as an author of this report entitled "Technical Report on Expansion Options at the Niobec Mine, Québec, Canada", prepared for IAMGOLD Corporation and dated June 17, 2011, do hereby certify that:

1. I am Vice-President, Metallurgy, at IAMGOLD Corporation, 1111 Rue St-Charles W., Suite 750, Tour Est, Longueuil, Québec J4K 5G4.

2. I am a graduate of Laval University, Quebec (B. Sc. A. Eng. Mining in 1982).

3. I am a registered mining engineer in the Province of Quebec (OIQ # 36825) and I am a member of the Canadian Institute of Mining and Metallurgy. I have practiced my professions of mining engineer in mineral processing continuously for the last twenty-nine years in the fields of gold and base mineral processing.

4. 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.

5. I have visited the Niobec Mine on a regular basis over the past 9 years and I have a good understanding of their cost structures and operational context.

6. I am responsible for the Section 16, Section 18 regarding all processing options, processing design, related Capital and Operating costs and Recoverability, and contributed to Sections 19 and 20 of the Technical Report titled: "Technical Report on Expansion Options at the Niobec Mine, Québec, Canada (June2011)".

7. I have received from my employer participation incentive securities ("options") and company shares since 2007.

8. I have read the NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th day of June, 2011.

(Signed & Sealed) "*Pierre Pelletier*"

Pierre Pelletier, ing.

DANIEL VALLIÈRES

I, Daniel Vallières, as an author of this report entitled "Technical Report on Expansion Options at the Niobec Mine, Québec, Canada", prepared for IAMGOLD Corporation and dated June 17, 2011, do hereby certify that:

1. I am Manager, Underground Projects, at IAMGOLD Corporation, 1111 Rue St-Charles W., Suite 750, Tour Est, Longueuil, Québec J4K 5G4.

2. I am a graduate of Laval University, Quebec (B. Eng. Mining) in 1991.

3. I am a registered mining engineer in the Province of Quebec (OIQ # 107203) and I am a member of the Canadian Institute of Mining and Metallurgy. I have practiced my professions of mining engineer continuously for the last Twenty years in the fields of gold and base metal mining.

4. 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.

5. I have made several visits to the Niobec Mine on a regular basis over the past 5 years and I have a good understanding of their cost structures and operational context.

6. I am responsible for Section 18 regarding Current mining operations, Underground mining options, and Infrastructure, related Capital and Operating costs and Environmental considerations (Open pit option), Taxes and Contracts, and contributed to Sections 19 and 20 of the Technical Report titled: "Technical Report on Expansion Options at the Niobec Mine, Québec, Canada (June 2011)".

7. I have received from my employer participation incentive securities ("options") and company shares since 2007.

8. I have read the NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th day of June, 2011

(Signed & Sealed) "*Daniel Vallières*"

Daniel Vallières, Ing.

24 APPENDIX 1

VARIOGRAMS

FIGURE A-1 VARIOGRAM (AZ 90°, DIP -90°)

Point Area: Analyses-Généreuse

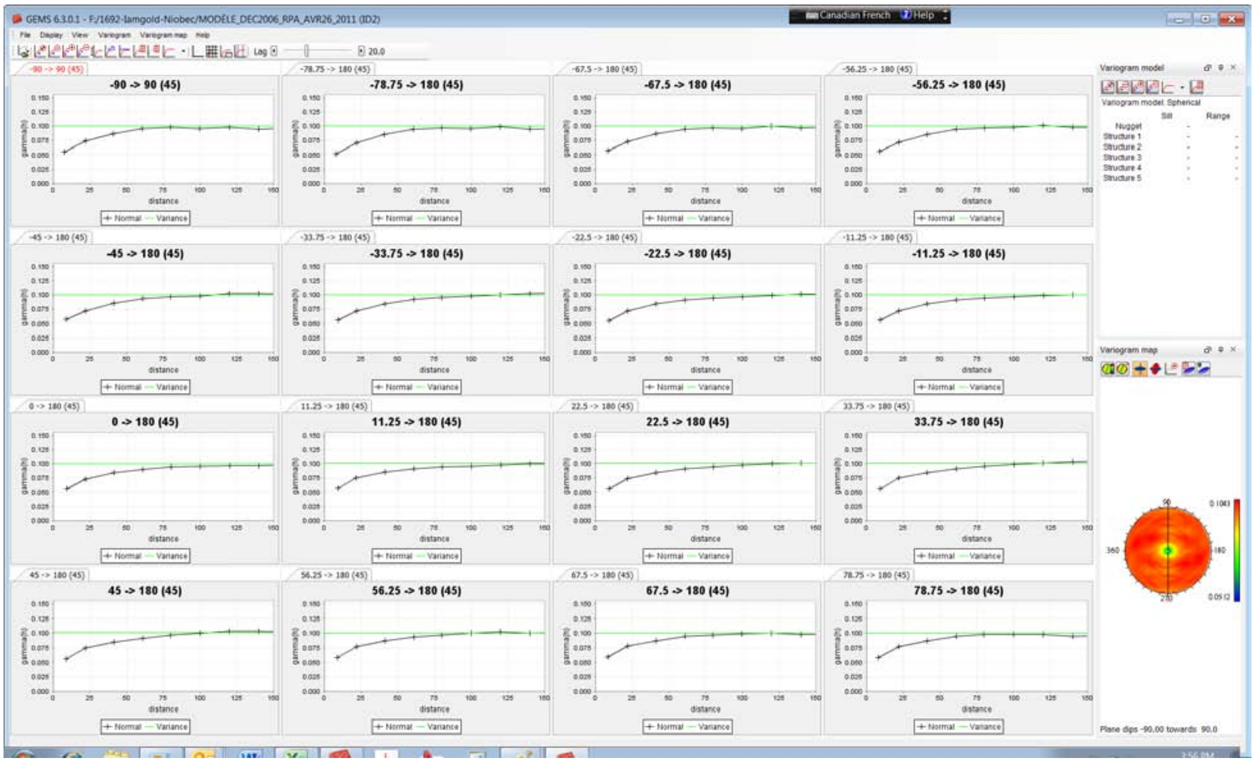

FIGURE A-2 VARIOGRAM (AZ 0°, DIP 0°)

Point Area: Analyses-Généreuse