GENIVAR

ABCOURT-BARVUE PROJECT
BARRAUTE, QUEBEC

TECHNICAL FEASIBILITY STUDY REPORT ON THE ABCOURT-BARVUE DEPOSIT

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TECHNICAL FEASIBILITY STUDY REPORT ON THE ABCOURT-BARVUE DEPOSIT

Prepared for

ABCOURT MINES INC.

By

GENIVAR LIMITED PARTNERSHIP
And
BUMIGEME INC.

NI 43-101 and NI 43-101F1
Technical Feasibility Study Report
GENIVAR Report No. Q106150

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February 15, 2007
Q106150

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To be cited as:

GENIVAR 2007. *Abcourt-Barvue project. Barraute, Quebec. Technical feasibility study report on the Abcourt Barvue deposit.*. Report by GENIVAR and BUMIGEME to Abcourt Mines Inc. 167 p. and appendices.

IMPORTANT NOTICE

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

NOTE TO THE READER

The sections 4 to 18 of MRB & Associates' Resources evaluation report for the Abcourt-Barvue project, 2006, prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, and filed as a technical report with Canadian Securities Regulatory Authorities on May 31, 2006, have been summarized to become sections 2 and 3 of this report.

SUMMARY

Abcourt Mines Inc. (Abcourt) plans to develop a mine-mill complex with underground and open pit operations on its Abcourt-Barvue property near Barraute in the Abitibi region, Quebec, Canada. This property has already been in production by an open pit and an underground mine during two different periods. The project will consist of a series of three open pits, namely Barvue (existing pit expansion), Abcourt East and Abcourt West, and the development of an underground mine to produce silver-gold ingots and a zinc-silver concentrates.

The in-situ mineral resource model of the whole deposit was updated by MRB & Associates in May 2006 according to the guides set fourth in National Instrument 43-101. The new resources evaluation was based on price conditions of 10.00 US$ per ounce for silver and 1.00 US$ per pound for zinc, an exchange rate of 0.85 US$/C$ and all geological information gathered to date.

The undiluted measured and indicated mineral resources stand at 7 018 969 tonnes grading 61.19 g/t Ag and 3.33 % Zn, with cut-off grades of 2.40, 2.55 and 3.20 % zinc equivalent (one ounce of silver is equivalent to 0.65 lb of zinc) for Abcourt open pit, Barvue open pit and underground mine respectively. The measured resources are 6 515 863 tonnes at 58.32 g/t Ag and 3.33 % Zn and the indicated resources are 503 106 tonnes at 98.35 g/t Ag and 3.44 % Zn. The undiluted inferred resources total 1 505 687 tonnes grading 120.53 g/t Ag and 2.98 % Zn.

The measured resources are all located between surface and a maximum vertical depth of 160 m but with the majority of the measured material being located between the surface and –75 m from surface. The indicated resources are located along the fringe of the measured resources. The inferred resources are presently known to reach a maximum vertical depth of 360 m below surface. The Abcourt-Barvue mineralization remains untested laterally and at depth.

Following the optimized open pit and underground mine designs, more than 80 % of the measured resources is planned to be mined by open pits to a maximum depth of 166 m. The remaining of the measured resources and a portion of the indicated resources are amenable to production using the underground Avoca cut-and-fill method, later in the 10-year production schedule, with three main declines as access to a maximum depth of 200 m.

The conversion of mineral resources into reserves takes into account dilution and losses occurring during mining operations. Different dilution factors were used for the Abcourt-Barvue project depending on the mining method and ore body configuration.

The dilution factors were 5 % in Barvue pit, 10 % in Abcourt pits, 20 % and 10 % for underground development in ore and production stopes respectively. As a result, the diluted mineral reserves are as follows:

Mineral Reserves Statement (including dilution)

Mining method	Classification	Tonnage (t)	Grade Ag (g/t)	Grade Zn (%)	Grade Zn EQ (%)
Open pit	Proven Mineral Reserves	5 338 731	44.79	3.15	4.09
Underground	Proven Mineral Reserves	1 169 662	105.19	2.87	5.06
	Probable Mineral Reserves	315 139	101.61	3.23	5.35
	Total Underground	1 484 801	104.43	2.95	5.12
Open pit and Underground	Proven Mineral Reserves	6 508 393	55.64	3.10	4.26
	Probable Mineral Reserves	315 139	101.61	3.23	5.35
	Total	6 823 532	57.76	3.11	4.31
Open pit	Proven Marginal Ore	1 151 502	17.65	1.58	1.95

Note: Abcourt pits' cut-off : 2.4 % Zn Eq; Barvue pit's cut-off : 2.55 % Zn Eq; underground stopes' cut-off : 3.2 % Zn Eq.

From these ore reserves, mainly in the proven category, 6.446 Mt grading 54.96 g/t Ag, 3.11 % Zn and 0.138 g/t Au (based on historical production data) will be mined and milled during the first 10-year production period at a milling rate of 1 800 tonnes per day. Of this tonnage, 5.339 Mt will be extracted from open pits and 1.107 Mt from the underground mine. Total waste to be extracted from the open pits amounts to 34.3 Mt including the marginal ore, for a waste to ore ratio of 6.42:1 which is reduced to 5.7:1 after preproduction stripping.

Moreover, after the first 10-year production period, the remaining underground proven and probable reserves, the stockpiled marginal ore and additional underground resources with the potential to become mineral reserves stand at about 2 Mt grading 47.88 g/t Ag and 2.43 % Zn for a zinc-equivalent grade of 3.43 %. This represents slightly more than 3 additional years of production at the same milling rate, which could be extended further with inferred resources after additional exploration and development.

Mining will be carried out using a conventional open pit method with 10 %-gradient ramps for almost 83 % of the scheduled tonnage. Drilling will be performed by conventional production drills. Blasting operations will use an emulsion-ANFO (ammonium-nitrate fuel oil) and a down-hole delay initiation system. Hydraulic backhoe shovel and a front-end loader will be used to load rigid haulage trucks of 62-t capacity. The production equipment will be supported by bulldozers, a grader, a hydraulic shovel mounted with either a rock-breaker or a scaling bar (pit wall securing) and a water truck (adapted from a 30-t articulated truck already owned by Abcourt) for dust control.

For the remaining 17 % of scheduled production tonnage, production will come from underground stopes (Avoca cut-and-fill method) for years 6 to 10. Underground mining will be carried out using three main declines from open pit bottoms, cross-cuts into the ore zone and lateral development on both sides to the limits of the lenses. Production will start at the bottom of the proven and probable ore blocks with sub-levels developed at 15 to 20-metre vertical intervals. All mining will be trackless. Jumbos will be used for declines, access cross-cuts and sub-level development. In the Avoca stopes, the ore will be removed with remote controlled 5 or 2-cubic-yard scoops. The benches will be drilled with long hole machines. The main declines will be driven with a minus 15 to 17 % slope. Blasting operations will use ANFO explosives with a gel cartridge to facilitate the detonation both for development and production purposes. Trucks with 30-t loads will tram the ore to the surface and will come back with a load of waste rock to be used as backfill if needed. Fresh air for underground mine ventilation purposes will be injected through ventilation raises and the existing shaft after its rehabilitation. The stopes will be backfilled with waste rock and paste or cemented backfill. The backfill will consist of mill tailings and/or pyrite concentrate.

The process plant was sized and cost for a throughput rate of 1 800 t/day. After an initial build-up, the plant throughput is scheduled at 650 000 t/a. The processing plant will operate 365 days per year. An ore mixing procedure will take place on a dedicated area close to the crusher to facilitate blending of the crusher feed by the other front-end loader of the equipment fleet. Mixing will prevent sharp grade variations in the mill and provide better operating results. The crushed product will be discharged via a surge bin to the SAG mill.

Cyanidation and flotation will be parts of a two-step ore treatment process. First, the ore will pass through the cyanidation circuit to recover most of the silver and a small amount of gold in Ag-Au ingots.

The cyanidation circuit consists of two cyanidation stages with drum filters and repulpers between each stage, followed by a conventional Merrill Crowe Ag-Au recovery system. Then, the filter-press cake will be calcined prior to be smelted and refined in an electric arc furnace to produce the Ag-Au ingots. After washing and aeration, rejects from cyanidation will be directed to a flotation circuit to recover zinc and the remaining silver.

The zinc flotation will be performed into a conventional rougher-scavenger-cleaner circuit from which Zn-Ag concentrates will be produced. The rejects will be directed to the pyrite flotation circuit to minimize the bulk of residues with acid-generating potential in the tailings pond. The pyrite concentrate represents 12 to 15 % of the total tailings and will be disposed of in a high-security and impermeable sulphide cell located in the tailings pond. The desulphurized tailings containing 0.24 % S are then considered as non-acid generating or neutral material. The pyrite concentrate cell will eventually be closed during the production period (pyrite concentrate going underground as stope backfill material) and covered with neutral tailings material, thus facilitating the final reclamation of the area.

By carrying out optimization tests, the authors consider that it will be possible to increase the metallurgical results of the Abcourt concentrator over the estimated numbers stated in this report. The main components of this program would be wet grinding to reduce NaCN consumption, the application of more appropriate cyanidation parameters to raise the Ag recovery from 70 % to 78 % and the selection of a better frother in the Zn-Ag flotation to increase the concentrate Zn grade from 54.5 % to 56 %.

It is assumed that ingots will be delivered to a precious metals refinery in Canada while Zn-Ag concentrate will be trucked, railed and shipped by boat to a custom smelter.

The existing installations on site will be used if needed, namely service building, warehouse, power line, fuel storage and refuelling station, roads, parking area, telephone line, artesian well, sewage treatment facilities, ditches and ponds. With only few minor improvement and reshaping, the service building will comprise the engineering and mine department office, the mine dry, the mine warehouse and a mechanical workshop for light vehicles and equipment of limited size. The process plant will be erected on the foundations of the previous mill and the repair shop for mining equipment will be constructed on an existing concrete slab near the service building. A water treatment plant is required mainly for the treatment of mine water to precipitate metals in solution; it will be connected to a settling pond via a ditch. Industrial water for mineral processing facility will be supplied from pit dewatering, mine water and tailings pond.

The objective of the environmental management program is to meet or surpass the standards and regulations. Thus, environmental criteria have already been applied and taken into account to minimize the environmental impact of the project. Actually, several measures were implemented at this stage of project development and are as follows:

- the integration of pyrite flotation in mineral processing to lower the sulphur content of the tailings;
- the installation of a high-security and impermeable sulphide cell to manage the pyrite concentrate in the tailings pond;
- the excavation of ditches to manage cleaned water separately from water affected by ore and marginal ore stockpiles;
- the construction of water treatment facilities and a settling pond for of mine water and ore stockpiles runoff treatment prior to their discharge into the environment;
- the application of water on mining roads for dust suppression;
- the supply of industrial water from existing on-site ponds for mine needs and from mine pumped water and tailings pond for mill requirements; no water intake installed in natural water course;
- the implementation of progressive rehabilitation programs (revegetation and stabilisation) for waste dumps, overburden pile and tailings pond;
- the establishment of a conceptual closure and rehabilitation plan including post-closure activities (monitoring, maintenance and remedial work).

The preproduction capital expenditure required for the Abcourt-Barvue project is estimated at 67.88 M$ in third-quarter 2006 Canadian dollars, excluding taxes and duties, while the working capital amounts to 3.38 M$.

Capital cost estimate summary

Item	$'000
Mine equipment & preproduction stripping, etc	19 812
Process plant (EPCM & contingencies included)	44 275
Infrastructure	1 862
Owner's costs (including warehouse inventory)	1 930
Preproduction Total	**67 879**
Working capital	**3 376**

On-going investments of 24.42 M$ for additional equipment purchase, overburden stripping, marginal ore mining and stockpiling, dam raising of tailings pond and progressive rehabilitation are anticipated over the first 10-year life of the project.

A major economic improvement to the project would be the reduction in capital costs realized by the purchase of used equipment for the process plant. That could come from one of two existing non-operating mills in the Abitibi region. Also, the use of former mill foundations could represent another saving. The authors strongly recommend the purchase of good used equipment for the mill and to maximize the use of existing foundations.

Operating costs vary from year to year and depend on tonnages of rock mined, haul distances and mining methods (open pit average cost at 2.34 $/t vs underground average cost at 20.93 $/t, both including pumping and water treatment). The first 10year costs and average unit operating costs are summarized as follows:

Operating cost estimate summary

Item	First 10-year costs	Unit costs	
	Total (M$)	Mined ($/t)	Milled ($/t)
Mining	98.60	2.77	15.30
Processing	87.42	2.45	13.46
G&A	14.30	0.40	2.22
Royalties	1.16	0.03	0.18
Total operating	**201.48**	**5.65**	**31.16**

Pre-tax and after-tax models have been developed for the Abcourt-Barvue project. The following analyses have been carried out:

- six scenarios have been developed to see the impact of changing metal prices for zinc, silver and gold and the exchange rate (C$/US$) on the economics of the project;
- the cash flow for various scenarios have been modelled on the assumptions of 40 % equity and 60 % debt at 7 % per year fixed interest rate;
- the cash flow for various scenarios have also been modelled with 100 % equity financing, with no debt, interest expense nor capital repayment;
- a base case scenario was developed on the assumption of decreasing metal prices during the project schedule;
- a sensitivity analysis on the base case was performed in order to examine the effect of variation of key parameters on the economic analysis results. The sensitivity to changes in total revenue (NSR value of ore) and capital and operating expenditures has been calculated from the cash flow statement.

The results and economic parameters of the cash flow analysis for the six scenarios modelled which were performed on constant metal prices and exchange rate over the 10-year project schedule, are tabulated in the following table.

On an initial capital cost of 67.88 M$ and the 60 %/40 % debt to equity ratio, the base case scenario of the project has payback periods of 4 years (pre-tax) and 5 years (after-tax) and generates net cash flows of 154 M$ pre-tax and 108 M$ after-tax. The revenue was calculated from the following variable metal prices and exchange rate:

Cash flow analysis results.

	Scénario 1	Scénario 2	Scénario 3	Base Case	Scénario 4	Scénario 5	Scénario 6
Au ($US/oz)	370	450	545	560	600	625	650
Ag ($US/oz)	4.80	8.00	9.50	9.54	10.00	11.75	13.50
Zn ($US/lb)	0.40	0.60	1.00	1.15	1.40	1.80	2.20
Exchange rate ($CAN/$US)	1.33	1.22	1.16	1.15	1.12	1.12	1.12
Total Revenue (N.S.R.)	**188 845 900 $**	**272 423 400 $**	**392 827 900 $**	**433 537 500 $**	**497 561 000 $**	**627 638 300 $**	**757 774 100 $**
Total Capital Expenditures	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**
Total Operating Expenditures	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**
FINANCING METHOD : 60% DEBT, 40% EQUITY							
Financial Valuation before taxes							
Return on equity (ROE)	N/A	N/A	20.32%	39.66%	45.29%	79.36%	116.28%
NPV (5%)	(78 576 740 $)	(14 671 746 $)	77 574 445 $	114 704 209 $	157 859 241 $	257 477 065 $	357 139 396 $
IRR of the project	N/A	N/A	14.76%	25.00%	29.45%	46.22%	62.45%
Payback period (years)	NO PAY BACK	NO PAY BACK	6.78	4.00	3.40	2.16	1.60
Financial Valuation after taxes							
Taxes	0 $	0 $	(30 140 282 $)	(45 783 696 $)	(71 722 817 $)	(123 573 555 $)	(175 860 274 $)
Return on equity (ROE)	N/A	N/A	16.23%	33.12%	38.08%	68.15%	98.69%
NPV (5%)	(78 576 740 $)	(14 671 746 $)	57 835 865 $	82 575 331 $	108 335 000 $	169 382 050 $	229 458 990 $
IRR of the project	N/A	N/A	11.59%	20.02%	23.79%	37.66%	50.33%
Payback period (years)	NO PAY BACK	NO PAY BACK	8.63	5.00	4.20	2.66	1.99
FINANCING METHOD : 100% EQUITY							
IRR	N/A	N/A	17.04%	27.54%	31.68%	48.27%	64.18%
NPV (5%) (1)	(94 988 095 $)	(34 126 197 $)	53 727 318 $	89 088 999 $	130 189 030 $	225 063 147 $	319 979 653 $
EBITDA (2)	(11 979 584 $)	71 597 916 $	192 002 416 $	232 712 016 $	296 735 516 $	426 812 816 $	556 948 616 $
Taxes	0 $	0 $	(34 435 405 $)	(50 078 820 $)	(76 017 941 $)	(127 868 679 $)	(180 155 397 $)
Bare-Bones Valuation (3)	(11 979 584 $)	71 597 916 $	157 567 011 $	182 633 197 $	220 717 576 $	298 944 138 $	376 793 219 $

1 Non inflation discount rate: (7 % interest rate - 2 % inflation rate) = 5 %.

2 EBITDA: earnings before interests, taxes, depreciation and amortization and as per definition, without considering any investment.

3 Bare-Bones Valuation: cash flow evaluations based on constant metal prices, constant dollars, no inflation, no debt, no interest, on a project basis and after tax.

- zinc: from 1.36 to 0.85, for an average price of 1.15 US$/lb;
- silver: from 9.95 to 8.94, for an average price of 9.54 US$/oz;
- gold: from 595 to 510, for an average price of 560 US$/oz;
- exchange rate: from 1.13 to 1.19, for an average rate of 1.15 C$/US$.

The project has IRR of 25 % and 20 % and NPV values at a 5 % discount rate (7 % interest rate - 2 % inflation rate) of 115 M$ and 83 M$, both pre-tax and after-tax respectively. Furthermore, a cash flow calculation was performed with current metal prices and exchange rate as per February 8, 2007 (Zn at 1.36 US$/lb, Ag at 13.63 US$/oz, Au at 657 US$/oz and 1.1853 C$/US$), kept constant all over the 10-year production schedule, and the above numbers give an IRR of 30.1 % and a NPV of 135 M$ on an after-tax basis.

Sensitivity calculations were performed on the base case project cash flow by applying a range of variation of ± 25 % against net revenue and capital and operating expenditures both on a pre-tax and an after-tax basis. The project is highly sensitive to changes in total revenue, based on the NSR value of the ore thus metal prices, and moderately sensitive to the operating expenditures and to the capital expenditures. Therefore, higher metal prices would have a tremendous positive effect on the project (see scenarios 4, 5 and 6 in table above). Inversely, lower metal prices would have a negative effect on the viability of the project. If this happens, it could be mitigated by increasing the zinc-equivalent cut-off grade during the low metal prices period to increase zinc and silver grades fed to the mill.

At constant metal prices and exchange rate over the 10-year schedule, the break even point of the 60 % loan / 40 % equity scenario occurs at 0.70 US$/lb Zn, 8.42 US$/oz Ag, 475 US$/oz Au and 1.20 C$/US$, while the 100 %-equity scenario presents a break even Zn price of 0.66 US$/lb, both considering capital expenditures. On a 100%-equity basis without considering any investment (EBITDA: earnings before interests, taxes, depreciation and amortization), the breakeven point occurs at a zinc price of 0.43 US$/lb.

The Abcourt-Barvue project contains an economic mineral reserve and, based on the cash flow analysis using realistic long-term metal prices, it is recommended to proceed with the development of a mine through permit applications, detailed engineering and construction of an 1 800 tpd mine-mill complex and subsequently produce Ag-Au ingots and Zn-Ag concentrates for sale.

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1. INTRODUCTION AND TERMS OF REFERENCE

1.1 Project description

GENIVAR Limited Partnership (GENIVAR) and Bumigeme Inc. (BUMIGEME) have been retained by Mr Renaud Hinse, President of Abcourt Mines Inc (Abcourt), to prepare a technical feasibility study concerning the Abcourt-Barvue project, a deposit lying on the Abcourt-Barvue property and comprising zinc-silver mineralization with associated gold. The deposit has potential for development as open pit and underground mining operations. Geological exploration and diamond drilling programs have been carried out, resulting in a revised mineral resource estimate. This resource estimate, together with geotechnical information, has been used in the development of open pit and underground mine designs and production schedule. All the information available from previous milling tests and milling operations is well reported by BUMIGEME, and Roche Ltd 1999. Additional metallurgical testwork has been done to provide parameters for the design of a crushing, milling, cyanidation and flotation concentrator for the production of saleable silver-gold ingots, zinc-silver concentrates, a pyrite concentrate and some non-acid generating tailings.

The Abcourt-Barvue project (Figure 1.1.1) is located in the municipality of Barraute, in the Abitibi area (north-western Quebec, Canada), 37 km east from the town of Amos and 56 km north from the mining community of Val-d'Or.

For the following sections through mineral resource estimate, GENIVAR make use of the documentation already publicly filed by MRB & Associates (MRB) in its Resources evaluation report for the Abcourt-Barvue project of May 2006 and by Innovexplo in its Resources validation report of March 2005.

1.2 Project ownership

The Abcourt-Barvue property is wholly owned by Abcourt and all claims are in good standing. The property is subject to some royalty payments which will be described in section 2.2.

1.3 History and work carried out

The history of the Abcourt-Barvue property began with the discovery of silver and zinc mineralization in 1950 when a geological survey of the Quebec Department of Mines, under the supervision of Dr. W.W. Weber, discovered zinc mineralization in range VII of Barraute Township (Cornwall. 1955).

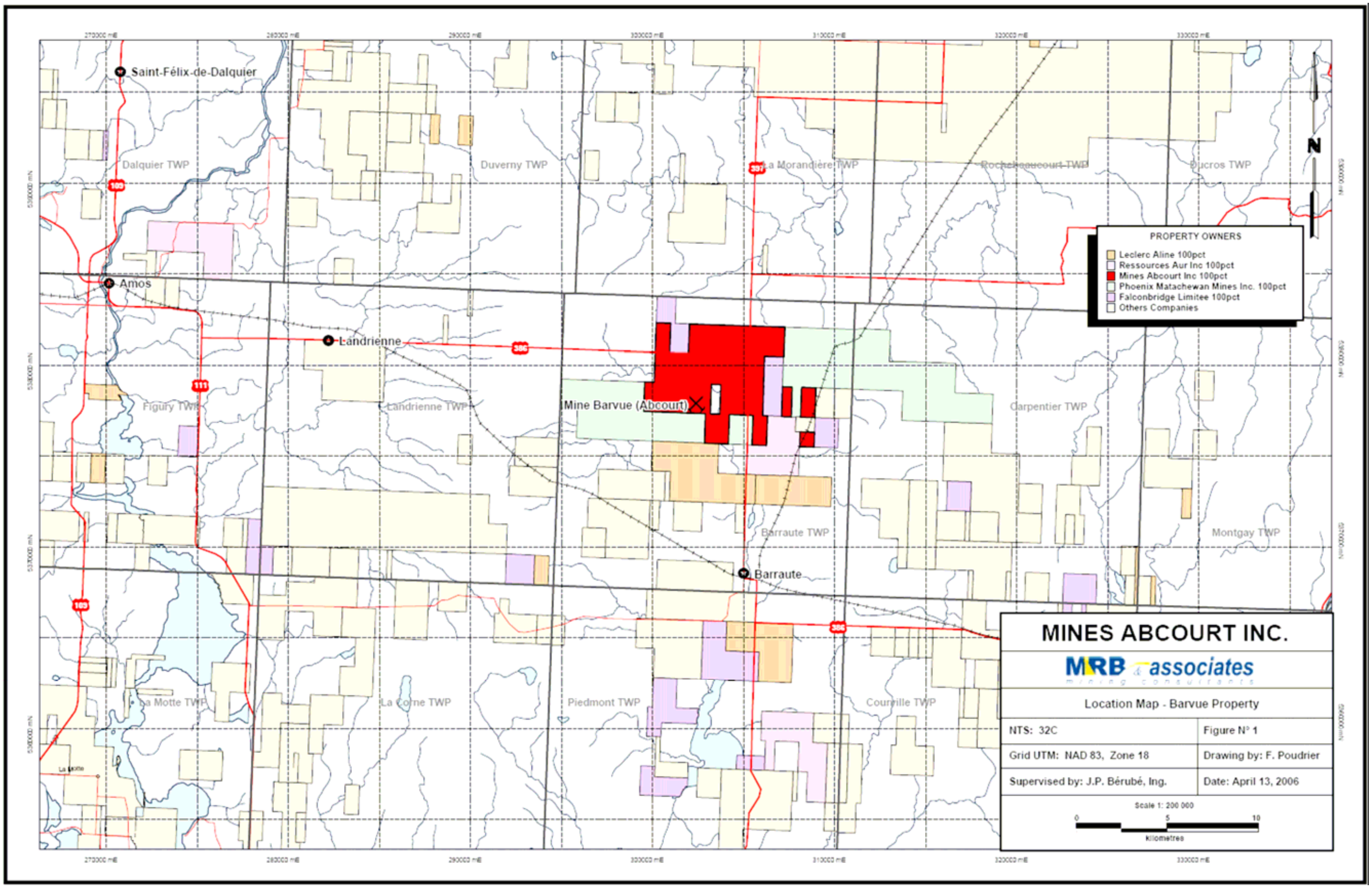

Figure 1.1.1 Location map (Source: MRB's Resources evaluation report, May 2006).

This discovery initiated a widespread prospecting and staking rush in the Barraute-Fiedmont area. Numerous sulphide boulders were located in Fiedmont Township over an area that extends northward from the shores of Lac Fiedmont to the boundary line between Fiedmont and Barraute Townships. During this period, prospector Gérald Leclerc was doing surface trenching on the former Consolidated Pershcourt and Barvue properties where he found a zinc-silver showing grading 3.62 % Zn and 188.7 g/t Ag over 6.7 metres (Vachon, 1994).

After the initial discovery, Golden Manitou defined with a limited amount of surface drilling, a mineral resources of 15 400 000 t grading 3.26 % Zn and 39.0 g/t Ag (Vachon, 1994). Subsequently, the Abcourt-Barvue deposit was in production during two periods: between 1952 and 1957 by Barvue Mines Limited and between 1985 and 1990 by Abcourt. The tonnages produced in the past are: 5 002 190 metric tonnes grading 38.74 g/t Ag and 2.98 % Zn (5 514 000 short tons grading 1.13 oz/st Ag and 2.98 % Zn, between 1952 and 1957) from the Barvue's open pit and 632 319 metric tonnes grading 131.65 g/t Ag and 5.04 % Zn (697 016 short tons grading 3.84 oz/st Ag and 5.04 % Zn, between 1985 and 1990) from underground production.

According to the MRNFP, over 1 325 diamond drill holes have been drilled in the Barraute Township by past owners and Abcourt on the Abcourt-Barvue property. A summary of Abcourt-Barvue's history is listed below.

1950 : Discovery of zinc mineralization on the Barvue claims by Gérald Leclerc. Best trench sampling result; 3.62 % Zn and 188.5 g/t Ag over 6.70 metres.

1950-51 : Surface exploration program by Pershcourt Goldfields Ltd. (Pershcourt) totalling 36 holes and 9 240 metres (P-01 to P36 series). The mineralized zone was followed over 900 metres to the west of the Barvue property.

Golden Manitou drilled 100 holes on the Barvue property (BVS-01 to BVS-100 series) and delineated a zinc-silver deposit over 760 metres up to 31 metres wide and down to 210 metres in depth. Resources were estimated at 17 Mt grading 3.26 % Zn and 39 g/t Ag. Barvue Mines Ltd (Barvue) was subsequently created.

1952: Pershcourt carried out an underground exploration program totalling 17 holes and 375 metres (U1-1 to U1-4, U2-1 to U2-13 series) on the Abcourt property. A three compartment shaft was sunk to a depth of 170 metres and 225 metres of drifting were excavated on two levels before work was stopped due to weakening zinc prices.

Barvue operated its open pit from 1952 to 1957 for a total production of 5 514 000 short tons grading 1.13 oz/st Ag and 2.98 % Zn. The open pit was mined over a length of 825 metres, a width of 150 metres and a depth of 75 metres.

1957: Barvue excavated a decline between the 76 m and the 152 m levels with 15 m spaced sub-levels. Underground work was stopped the same year due to falling zinc prices.

1968-69: Pershcourt and Frebert Mines Ltd conducted a surface drilling program on the Abcourt property totalling 54 holes (FS-63 to FS-66 and P-37 to P-86 series).

1971: Merging of Pershcourt and Abitibi Silver Mining Corp., successor of Frebert, to become Abcourt Metals Inc. The company signed an agreement with Jamieson Mines Ltd for a surface drilling program totalling seven holes and 1 144 metres (A-01 to A-07 series). An evaluation study was also performed.

1974: Norex (Noranda) optioned the Barvue claims, dewatered the underground workings, conducted a feasibility study and purchased the property.

1975: Rayrock Mines Ltd and Ashland Oil Canada Ltd performed a joint drilling program totalling 18 holes and 5 069 metres (RA-01 to RA-18) on the Abcourt property. They also conducted metallurgical tests.

1977: Norex carried out magnetic and electromagnetic surveys on the Barvue property, drilled two holes and sampled some parts of the sub-levels.

1980: The control of Abcourt Metals Inc was taken over by "Fonds Miniers Hinse" and became Les Mines d'Argent Abcourt Inc (Abcourt). The company carried out a 19-hole exploration program totalling 1 299 metres (80-01 to 80-19 series) along with metallurgical tests.

1981: Norex studied the possibility to make a joint venture with Abcourt.

1983-84: Abcourt purchased the Barvue property from Norex and proceeded with the dewatering and rehabilitation of the mine. Apart from one (1) hole drilled on Lamontagne claims (east of Barvue), Abcourt drilled:

- 128 surface holes totalling 9 678 metres (83-20 to 83-139 and 84-140 to 84-147 series);
- 31 surface holes totalling 2 824 metres (84-148 to 84-178 series); and
- 69 underground holes totalling 3 037 metres (84-ST-01 to 84-ST-59, 84-ST-99 to 84-ST-104 and 84-ST-125 to 84-ST-132 series).

Mining reserves were confirmed by underground drilling.

1985-86: Abcourt carried out a surface drilling program totalling nine (9) holes and 1 399 metres (85-01 to 85-09 series) and an extensive underground drilling program totalling 215 holes and 6 778 metres (85-ST-01 to 85-ST-215 series) in the Abcourt Area.

1986-89: The Barvue mine was connected to the Abcourt shaft with an internal ramp. The mine produced 697 016 short tons grading 3.85 oz/st Ag and 5.04 % Zn between 1985 and 1990. An total of 204 ounces of gold was recovered with the silver concentrate and paid for.

1990-1993: Norex optioned the Abcourt-Barvue property excluding the mineralized zone to a depth of 300 m and carried out an extensive exploration program. Apart from the geological, pedo-geochemical, magnetic, electromagnetic, gravimetric and geo-electrochemical surveys conducted over the areas, Norex carried out the following deep diamond drilling programs:

- 1991: Three (3) holes totalling 1 324 metres (AB-91-01 to AB-91-03 series);
- 1992: Two (2) holes totalling 852 metres (AB-92-04 to AB-92-05 series);
- 1993: Four (4) holes totalling 2 468 metres (AB-93-06 to AB-93-09 series).

1997-2005: Abcourt remained active on its property and proceeded with several drilling programs:

- 1997: One (1) 167 metres hole (AB97-38) on Lot 31, Range 7;
- 1998: Ten (10) BQ holes totalling 2 140 metres (98-1 to 98-10 series);
- 1999: Two (2) NQ holes totalling 285 metres (AB-99-01 to AB-99-02 series);
- 2003: Ten (10) holes totalling 530 metres (AB-03-01 to AB-03-10 series);
- 2004: Twenty-four (24) holes totalling 1 169 metres (AB-04-01 to AB-04-24 series);
- 2005: Forty-six (46) holes totalling 5 879 metres (AB05-01 to AB05-46 series).

1.4 Preparation of the technical feasibility study

In October 2005, Abcourt initiated the study reported herein using a combination of Abcourt personnel, GENIVAR and other specialist consultants. In particular, a report entitled Stability analysis of the Abcourt-Barvue pit extension, part 1, December 2002 and part 2, September 2003 by François Charette, Eng., M.Sc. was used.

GENIVAR and BUMIGEME have been retained by Abcourt to prepare this technical feasibility study based on the establishment, on the Abcourt-Barvue property, of open pit and underground mining operations with on-site production of saleable silver-gold ingots and zinc-silver concentrates in the process plant.

Abcourt, GENIVAR and BUMIGEME conducted this study in conjunction with a number of other consultants responsible for specific technical areas as shown hereafter.

Mineral resources estimate, mine plan, mine equipment and mine facilities	MRB and Associates and GESCAD GENIVAR Limited Partnership
Open pit geotechnical studies	François Charette, Eng., M.Sc.
Metallurgical testing	Laboratoires LTM and Bumigeme Inc
Process engineering	Bumigeme Inc.
Mine rock and tailings disposal	Spinofex (Yves Gagnon, Eng., M.Sc.) GENIVAR Limited Partnership
Environmental management plan and permitting geotechnical studies	Spinofex, Golder Associés Ltée GENIVAR Limited Partnership
Infrastructures and plant design, capital expenditures and operating costs	Bumigeme Inc GENIVAR Limited Partnership
Economic evaluation	GENIVAR Limited Partnership

MRB, GESCAD, François Charette, Laboratoires LTM and Spinofex were retained directly by Abcourt for the purposes of this feasibility study.

Site visit has been done on October 31, 2005 by Marc Lavigne, Eng, M.Sc of GENIVAR in the company of Renaud Hinse of Abcourt.

1.5 Responsibility

The mineral resource estimate described herein is sourced from MRB's Technical Report, May 31, 2006 under the responsibility of Jean-Pierre Bérubé, P.Eng. The open pit designs and production schedules, as well as the capital and operating cost estimates for the open pit mining aspects of the project, were prepared by Marc Lavigne, Eng. M.Sc., of GENIVAR. Underground mining related activities were prepared by GENIVAR with the assistance of Abcourt and are based on Roche's Technical Validation Report, 1999; cost figures were updated to 2006. BUMIGEME prepared the process plant and related infrastructures designs and the associated capital and operating cost estimates under the direction of Florent Baril, Eng. Spinofex and GENIVAR were responsible for the design and costing of mill tailings pond based on Golder Associés Ltée 2007 report (Golder) and Roche's 1999 report, while François Charette, Eng., M.Sc. was responsible of geotechnical work in relation to the open pits. GENIVAR and Abcourt worked on the metal prices, NSR values and cash flow projections.

1.6 Basis of the technical feasibility study

1.6.1 Technical and economic parameters

Ore will be mined at a rate of 1 800 t/d mainly from open pits with overlapping underground mine production at the end of the 10-year period under study. The mining operation will use an excavator/loader truck configuration, hauling ore directly to a mixing bed for the open pit mining and use trackless equipment (scoop truck configuration) via main declines and cross-cuts for the Avoca underground mining method. All concentrates produced by the on-site processing plant will be shipped to a custom smelter/refiner in North America or overseas for treatment. The silver-gold ingots will be shipped to a precious metal refiner.

Concentrate sales terms used are based on published rates and indicative quotes.

All capital and operating costs were estimated and economic evaluations were conducted using Canadian dollars of third-quarter-2006 value. Economic analysis was carried out by means of discounted cash flow analysis expressed in constant dollar terms. Table 1.6.1 provides the principal criteria for the Abcourt-Barvue project.

An analysis for different metal prices and exchange rates has been performed to compare zinc market price with the project production cost for zinc. The production cost includes mining, milling, royalties, concentrate transportation to Antwerp, Belgium, and smelting. The results are concisely tabulated in Table 1.6.2. These calculations have been done without taking into account the capital investment and on a before interest and taxes basis, and are commonly known as EBITDA (earnings before interests, taxes, depreciation and amortization and as per definition, without considering any investment). Following this exercise, the break even point for the Abcourt-Barvue project occurs at a zinc price of 0.43 US$/lb.

1.7 Disclaimer

GENIVAR and BUMIGEME have reviewed and analyzed data provided by Abcourt, its consultants and previous operators of the property, and has drawn its own conclusions therefrom, also based on its direct field examination. GENIVAR and BUMIGEME have not carried out any exploration work, drilled any holes or proceeded with any sampling and assaying. However, the zinc-silver bearing mineralization is visible in the local rock observed in core. GENIVAR used the model developed by MRB in its mineral resources evaluation to undertake estimation of open pit and underground mine ore's qualities and quantities.

Table 1.6.1 Principal criteria of the project for the first 10 years of production.

Item	Unit	Quantity
First 10-year production		
Ore from open pit (OP)	Tonnes	5 338 731
Waste from open pit (low grade ore incl.)	Tonnes	34 298 191
Total Rock	**Tonnes**	**39 636 922**
Waste to Ore ratio	(t/t)	6.42
Ore from underground (UG)	Tonnes	1 107 269
Total Ore (open pit and underground)	**Tonnes**	**6 446 000**
Diluted average zinc grade (OP&UG)	%	3.11
Diluted average silver grade (OP&UG)	Grams per tonne	54.96
Diluted average zinc equiv. (OP&UG)	%	4.26
Diluted average gold grade (OP&UG)	Grams per tonne	0.138
Contained zinc (OP&UG)	'000 pounds	441 957
Contained silver (OP&UG)	'000 troy ounces	11 390
Contained gold (OP&UG)	'000 troy ounces	29
Plant recovery		
Zinc in Zn-Ag concentrate (flotation)	%	96.0
Silver in Zn-Ag concentrate (flotation)	%	18.0
Silver in ingot (cyanidation)	%	70.0
Gold in ingot (cyanidation)	%	90.0
Metal grades in Zn-Ag concentrate		
Zinc in Zn-Ag concentrate (flotation)	%	54.5
Silver in Zn-Ag concentrate (flotation)	Grams per tonne	181
Annual production (payable metal)		
Average annual production, ore milled	Tonnes	650 000
Average annual production, Zn in concentrate	'000 pounds	35 760
Average annual production, Ag in concentrate	'000 troy ounces	68.5
Average annual production, Ag in ingot	'000 troy ounces	796
Average annual production, Au in ingot	'000 troy ounces	2.5
Capital costs (contingency and EPCM incl.)		
Mining	'000 C$	19 812
Milling	'000 C$	44 275
Infrastructure	'000 C$	1 862
Owner's costs	'000 C$	1 930
Total initial capital	**'000 C$**	**67 879**
Working capital	**'000 C$**	**3 376**
On-going investment (rehabilitation included)	**'000 C$**	**24 418**

Table 1.6.2 Economic parameters.

Au price (US$/oz)	370	450	545	600	625	650
Ag price (US$/oz)	4.80	8.00	9.50	10.00	11.75	13.50
Zn price (US$/lb)	**0.40**	**0.60**	**1.00**	**1.40**	**1.80**	**2.20**
Exchange rate (C$/US$)	1.33	1.22	1.16	1.12	1.12	1.12
Production cost (US$/lb Zn)	0.557	0.650	0.794	0.934	1.054	1.175
Precious metals credit (US$/lb Zn)	0.133	0.213	0.254	0.270	0.312	0.356
Net cost (US$/lb Zn)	**0.424**	**0.437**	**0.540**	**0.664**	**0.742**	**0.819**

The various agreements under which Abcourt holds title to the mineral lands for this project have not been investigated or confirmed by GENIVAR nor BUMIGEME. However, the validity of the claim status has been verified by MRB independent consultant and MRB stated that all claims are in good standing.

The results and opinions outlined in this report are dependent on the aforementioned information being current, accurate and complete as of the date of this report and it has been assumed that no information has been withheld which would impact the conclusions or recommendations made herein.

All currency amounts are stated in Canadian or US dollars, as specified, with costs typically expressed in Canadian dollars and commodity prices in US dollars. Silver and gold grades may be expressed in grams per metric tonne (g/t) or troy ounces per short ton (oz/st) while zinc grade is expressed in percent (%).

Data from different sources were used in the production of this report. GENIVAR and BUMIGEME generally used the metric system: thousands are expressed by a space and decimals are separated by a point. Abbreviations and conversion factors used are as follows:

°C	Degree Celsius	oz	Troy ounce
g	Gram	oz/st	Ounce per short ton
ha	Hectare	g/t	Gram per metric ton
kg	Kilogram	ppb	Part per billion
km	Kilometre	ppm	Part per million
m	Metre	st	Short ton
mm	Millimetre	t	Metric tonne
'	Foot	"	Inch
lb	Pound	W	Watt
kW	Kilowatt	V	Volt
kV	Kilovolt	WI	Work Index
S.G.	Specific Gravity	C$	Canadian Dollar
US$	American Dollar	H	Horizontal
V	Vertical	m^2	Square metre
m^3	Cubic metre	M	Million
a	Year	tpa	Metric tonne per year
cm^3	Cubic centimetre	d	Day
OP	Open pit	UG	Underground mine
G&A	General and administration	cm/s	Centimetre per second
g/l	Gram per litre	mg/l	Milligram per litre
amp	Ampere		

1 inch	= 25.4 mm	1 mm	= 0.3937 inch
1 foot	= 0.305 m	1 m	= 3.28083 foot
1 mile	= 1.609 km	1 km	= 0.6214 mile
1 acre	= 0.405 ha	1 ha	= 2.471 acre
1 acre	= 4046.825 m^2	1 ha	= 0.01 km^2
1 oz	= 31.103 g	1 g	= 0.3215 0z
1 g/st	= 34.36 g/t	g/t	= 0.0291 oz/st
1 short ton	= 0,907 t	t	= 1.102 ton (short)

2. PROPERTY DESCRIPTION AND LOCATION

2.1 Location

The Abcourt-Barvue property is located in the Abitibi area (north-western Quebec, Canada, NTS 32C12), in the municipality of Barraute in the Barraute Township. The Abcourt-Barvue deposit is located on a past producing mine site with several readily useable infrastructures.

The Abcourt-Barvue property is the result of merging of two (2) previously separate properties: Abcourt and Barvue. It is located 37 km east from the town of Amos and 56 km north from the mining community of Val-d'Or. Access to the property from Val-d'Or is by highway 397 and by highway 386 from Amos to Barraute (Figure 1.1.1).

2.2 Claim status

The Abcourt-Barvue property consists in two (2) mining concessions (MC) and seventy-seven (77) contiguous mining claims (CL). The mineral holdings cover an area of 3 174.71 hectares in Barraute Township (see Figure 2.2.1). Abcourt also owns three (3) isolated blocks totalling eight (8) claims and 277.18 hectares located on Range VI, lots 52 to 54 and Range VII, lots 48, 49, 52 to 54. These last claims are surrounding the Bartec gold deposit currently owned by Ontex Resources Ltd. Finally, Abcourt is also 100 % owner of the surface rights of parts of Lots 24 to 34 in Range VII for approximately 250 ha. All claims are 100 % owned by Abcourt and they are in good standing.

The property is subject to some royalty payments. The details of these rights are discussed in Innovexplo's Technical Report (2005) and summarized in Table 2.2.1.

- Claim groups obtained from Mr. Jean-Guy Barrette and Mr. Jack Stoch, which comprise the north half part of lots 27 to 32 (Range VIII) are subject to a royalty of 1 % NSR. No ore has been found on this property until now;
- Manitou-Barvue Mines Limited (now Terratech Resources Inc., (Terratech) has the right to a standard royalty of $0.25/st of ore extracted from the original Barvue property. The royalty is payable on MC 390, lot 27 in range VII, half south part of lot 32 in range VII, and the north half part of lots 33 and 34 in range VI.

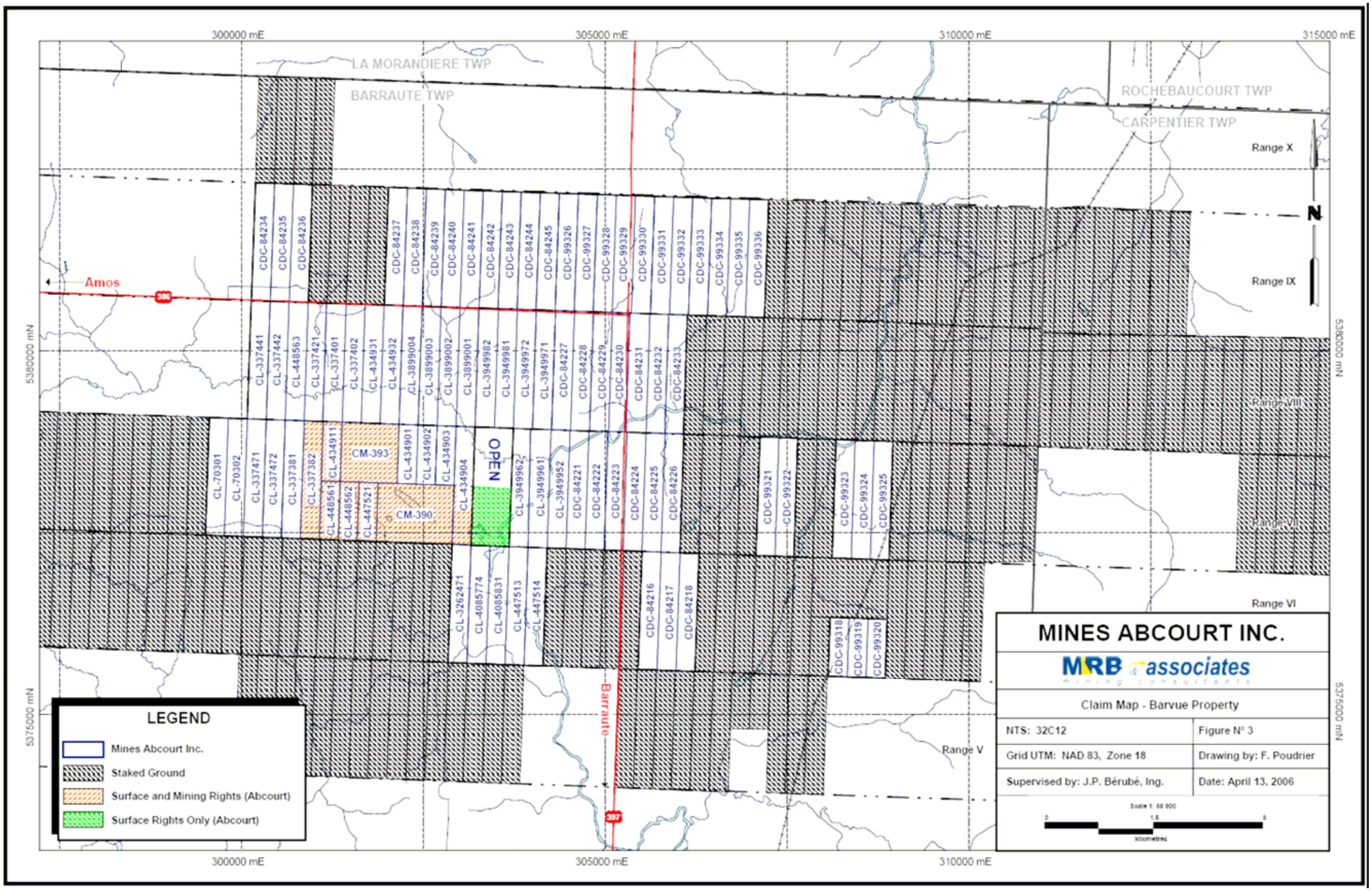

Figure 2.2.1 Claim map (Source: MRB's Resources evaluation report, May 2006).

Table 2.2.1 Detailed royalties by claim on the Abcourt-Barvue property.

Claim title	Area	Ownership	Royalties
CL 0434904	40.0	100 % Abcourt	½ south: $0.25/st (Terratech)
CL 0434931	40.0	100 % Abcourt	½ north: 1 % NSR (Barrette and Stoch)
CL 0434932	40.0	100 % Abcourt	½ north: 1 % NSR (Barrette and Stoch)
CL 0447521	20.0	100 % Abcourt	$0.25/st (Terratech)
CL 3899001	40.0	100 % Abcourt	½ north: 1 % NSR (Barrette and Stoch)
CL 3899002	40.0	100 % Abcourt	½ north: 1 % NSR (Barrette and Stoch)
CL 3899003	40.0	100 % Abcourt	½ north: 1 % NSR (Barrette and Stoch)
CL 3899004	40.0	100 % Abcourt	½ north: 1 % NSR (Barrette and Stoch)
CL 4085774	40.0	100 % Abcourt	½ north: $0.25/st (Terratech)
CL 4085831	40.0	100 % Abcourt	½ north: $0.25/st (Terratech)
MC 390	80.94	100 % Abcourt	$0.25/st (Terratech)
MC 393	60.7	100 % Abcourt	None

2.3 Environmental liabilities

There are no environmental liabilities related to the Abcourt-Barvue property. The mining concession related to the old Barvue tailings pond was returned to the Crown and is not included in the Abcourt-Barvue property. Therefore, there is no liability related to this tailings pond for Abcourt.

2.4 Accessibility, infrastructures and local resources

The property is easily reached all year round from Val-d'Or via highway 397 and by range road 6-7 to the mine site, about 5.5 km north of Barraute (Province of Quebec).

Electricity is available on the Abcourt-Barvue property from a power line that supplies the mine site.

In 1990, with the falling price of silver and zinc, the Abcourt-Barvue mine was shut down after five years of underground production. The site is still well provided with infrastructures and mine equipment. The proximity of an active mining centre such as Val-d'Or guarantees the availability of material and human resources for exploration and mining.

The Abcourt-Barvue project is located on an existing mine site having readily useable infrastructures. It also includes several pieces of equipment for underground mining such as Volvo and Caterpillar trucks, jumbo drills, scoops, pumps, ventilators, compressors, etc.

2.5 Physiography and climate

The area is relatively flat and lies within the great "Clay Belt" of northern Ontario and Quebec. The surface is a plateau-like grayish clay-covered plain, in places pierced by ridges of rock and glacial debris or dissected by streams. In some places, this clay-covered plain is interrupted by small rocky islands and rounded ridges or long sinuous eskers of sand and gravel. The eskers have in general a north-south orientation. The average elevation is 35 metres above sea level with a maximum at 100 metres.

This region is characterized by a continental climate with winter temperature lows in the -10°C to -35°C range with an average snow cover of 83 cm and summer temperature highs in the range of 10°C to 22°C with 115 mm of rain. Access to water is available at the Laflamme river, which runs north through the property.

3. GEOLOGY AND MINERAL RESOURCES

The following descriptions of the regional and local geological settings for the Abcourt-Barvue project together with discussions and/or statements about mineralization, exploration, drilling and mineral resources were given by Mr Jean-Pierre Bérubé, P.Eng., in the Technical Report entitled "NI 43-101 Resources evaluation report for the Abcourt-Barvue project, Barraute Township, Quebec, NTS 32C12, for Abcourt Mines Inc.", dated May 31, 2006 and filed with SEDAR on June 1, 2006. GENIVAR can not guarantee the accuracy of the comments made in MRB's report. The author did not proceed with any drill hole database validation nor any sample re-assaying tests. While exercising all reasonable diligence in checking and confirming it, GENIVAR has relied upon data presented by MRB in formulating its opinion.

"The Abcourt-Barvue area is located within the Abitibi geological Subprovince, a typical granite-greenstone terrane located in the south-eastern part of the Superior province of the Canadian Shield. With an 85 000 km^2 surface, the Abitibi belt is the largest greenstone belt of the world (Card. 1990) and also one of the richest mining areas (Hodgson and Hamilton. 1989; Poulsen *et al.*, 1992). The Abitibi greenstone Subprovince extends approximately 700 km from the Kapuskasing Structural Zone in north-central Ontario eastward to the Grenville Front in the south-central Opatica gneiss and plutonic terrane, while to the south it is bounded by the Bellecombe sequence of metasediments."

3.1 Regional geology

"The Abitibi Subprovince is divided in a "Northern Volcanic Zone" and a younger "South Volcanic Zone" (Ludden *et al.*, 1986; Chown *et al.*, 1992; Mueller *et al.*, 1996). The Porcupine-Destor fault zone (PDF) is interpreted to be the limit dividing these two terranes. The Northern Volcanic Zone is interpreted as an older diffuse volcanic arc, 2730-2710 Ma and the Southern Volcanic Zone is interpreted as a younger arc segments, 2705-2698 Ma (Mueller *et al.*, 1996). In the region of the properties, the east branch of this main fault, which reaches the Timmins mining area to the west, is named the Porcupine-Destor-Manneville tectonic zone (see Figure 3.1.1). The eastern extension of the Manneville tectonic zone is bounded by the NNE trending Laflamme River fault. Nevertheless, this limit could probably be extended to the east by the Courville tectonic zone which also constitutes the limit between the Landrienne Group and the Héva Group. The Abitibi greenstone belt is interpreted to result from island arc volcanism and is composed of volcanic and sedimentary sequences.

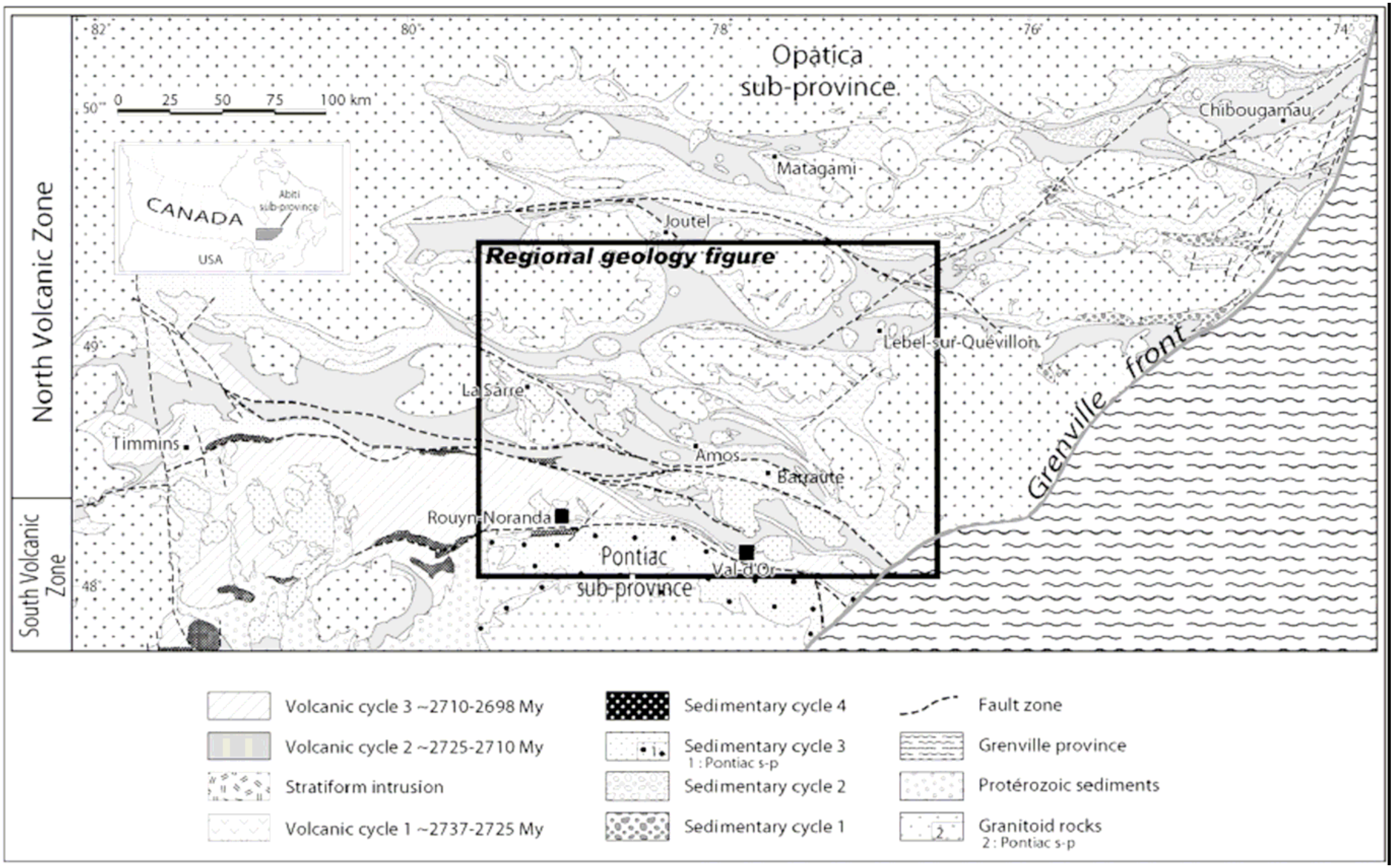

Figure 3.1.1 Regional sedimentary and volcanic sequences map of the Abitibi Subprovince (Source: MRB's Resources evaluation report, May 2006).

“The Abcourt-Barvue area includes different tholeiitic and calc-alkalic volcanic sequences. The property sits in the south-central part of the "Northern Volcanic Zone", in the Harricana Group (see Figure 3.1.1). From north to south, the stratigraphic pile can be described as follows; (1) Héva and Dubuisson Formations; (2) Landrienne Group (host of the Vendôme VHMS cluster); (3) Figuery Group (calc-alkalic in the northern part host of the Abcourt-Barvue volcanogenic deposit cluster and tholeiitic in its southern part) and; (4) Amos Group (tholeiitic). These Groups and Formations could be summarized as follows:

“(1) The Héva Formation's thickness varies from 3 to 5 km with a regional trend similar to the other formations at EW to ESE-WNW over more then 100 km. This formation is poorly exposed on surface. It is made up of felsic volcanics and andesitic to basaltic interflows which vary from massive to pillow lavas and local variolitic flows. The felsic volcanics are represented by brecciated dacitic to rhyolitic flows and thin pyroclastic layers which lie in the central part of the Vendôme property.

“(2) The Landrienne Group is mainly composed of tholeiitic massive to pillowed basalts outcropping in the southern part of the Abcourt-Barvue property (Labbé, 1995). Thin rhyolitic flows are inter-layered within this formation. The contact between the Landrienne and the Figuery Group is difficult to trace as it is coincident with an east-west structure, the Abcourt deformation corridor, which is a two kilometre wide shear zone in a felsic agglomerate horizon (Labbé, 1995).

“(3) The Figuery Group is an EW, 3.5 to 7 kilometre wide geological unit bounded to the north and the south by regional faults. It is mainly composed of massive to pillowed andesitic flows and minor flow breccias. It also includes units of ash tuffs, lapillis tuffs and blocks tuffs associated with decametric dacitic interlayers. The geochemical affinity of this group is tholeiitic to the south, and calc-alkalic close to the Abcourt deposit (Guay, 1998). Graphitic shales are also present and they can reach thicknesses up to 300 metres. These sediments are locally rich in nodular pyrite. The lateral extensions of these units can be traced by the airborne EM-Input survey carried out by Aerodat Ltd in 1990.

“(4) The Amos Group is composed of pillowed basalts of tholeiitic affinity. They are locally weakly amygdular and rarely brecciated. Some conformable sills and unconformable dykes are contemporaneous to the volcanics and vary in composition from peridotites to pegmatites. Previous field work carried out by Guay in 1998 shows that the contact with the Figuery Group is marked by a 250 to 400 metres wide graphitic sedimentary unit.

"The regional trend is E-W and regional faulting is E-W and NNE-SSW. Tectonism has been related to the Kenorean orogeny, which deformed the volcanic sequences. Regional metamorphism is at the greenschist facies. The amphibolite facies is reached at the margin of late to post-tectonic pluton (e.g. hornsfels). Several major shear zones also cut the area in an ENE-WSW direction. They are cut by a NNE-SSW late faulting system. The main events related to the regional tectonic history could be summarized as follows:

"1) A pre-Kenorean phase which produced the large regional E-W faults like the Larder-Lake-Cadillac and Destor-Porcupine-Manneville faults and probably the more local Barvue fault;
2) The Kenorean phase where E-W folding appears and granitic to gabbroic intrusives took place and;
3) Local extensions which produced N-S faulting like the Laflamme fault."

3.2 Local geology

"On the Abcourt-Barvue property, the volcanic sequence is represented by the Figuery Group which is bounded to the north by the Amos Group and to the south by the Landrienne Group. The Abcourt-Barvue deposit is located at the contact between the Figuery Group and the Landrienne Group. The area is affected by the Chevalier river anticline (Labbé, 1995).

"The deposit is hosted in a volcanoclastic sequence characterize by tuffs and agglomerates usually strongly carbonatized (ankerite. siderite and calcite) and sericitized. At Abcourt-Barvue, the Zn-Ag mineralization is located close to a major E-W shear zone (Barvue deformation corridor). This deformation zone consists of a talc/sericite tuffaceous horizon with intense carbonatization and sericitization. Chlorite is usually restricted to intermediate to mafic volcanics rocks (Vachon, 1994).

"The Abcourt-Barvue mineralized horizon strikes E-W in the western portion of the property. In the Abcourt shaft area, the mineralized horizon changes its strike from E-W to SE-NW in the Barvue portion of the deposit. On the property, the units have steep (75°) dips to the north with a well developed E-W regional schistosity. The stratigraphic tops for these units have been documented in the field as being to the north.

"Favorable felsic volcanoclastic rocks and graphitic shale units have been documented on the property along the Abcourt-Barvue trend but also elsewhere on the property such as in its northern portion (i.e. North zone area). Potential of the Abcourt-Barvue trend extends at depth but also towards the west and to the south-east through the former Bar-Manitou zone.

“On the Abcourt-Barvue property, magnetic anomalies located on the north-west and south-west sides of a granodioritic pluton, with some gravimetric anomalies and several electro-magnetic conductors (EM) indicate a potential for mineralization. In this area, pyrrhotite has been documented in diamond drill holes.

“Ankerite and fuchsite have been recognized on surface in the south-western portion of the property. These minerals are found in close association with gold occurrences in the area (e.g. Swanson) and indicates that the Abcourt-Barvue also has a significant gold potential.”

“Abcourt-Barvue deposit area geological units:

“A marker tuff unit allows good stratigraphic correlation in the Abcourt-Barvue deposit area. The marker tuff horizon varies in thickness between 1 to 5 m and it is composed of bedded, light pale brown, quartz-carbonate millimetric beds to dark grey to black millimetric beds essentially composed of quartz with fibro-radial texture. This unit forms a hard and siliceous unit. In the thicker part of this marker unit and essentially in the east part of the deposit, the marker tuff contains quartz pebbles. The siliceous, finely bedded marker tuff unit contains silver and zinc values associated with dark layers usually in the upper part of the unit.

“The geological units found in the Abcourt-Barvue deposit area and located north of the marker tuff unit are summarized as follows (from north to south):

“1) Medium to dark grey graphitic tuff, varying from massive to bedded with sericite-quartz-graphite and argillite beds with 1 to 5 % disseminated pyrite;
2) Green massive andesite with interlayers of grey tuff which varies from massive to finely bedded sericite and quartz-carbonate, which could host a graphitic horizon;
3) Pale grey, millimetric felsic tuff beds with green-brown intermediate tuff beds hosted in the central part of a strong deformation zone (locally with chalcopyrite veinlets);
4) Mixed zone of a fine grained, massive, green tuff and a beige finely grained and bedded tuff;
5) Carbonatized, sericitized and deformed coarse volcanoclastic rocks unit evolving from stratified agglomerate to coarser agglomerates at top of the unit;
6) Zn-Ag bearing, dark coloured, sub-rounded quartz-carbonate felsic agglomerate conformable with the marker tuff unit (Alain Vachon, 1994).

“South and adjacent to the marker tuff unit and within the Abcourt-Barvue deposit, we can identify the following geological units (from north to south):

"1) Grey tuff and agglomerate schist (Gs) host of a major deformation zone. This unit is characterized by sericite and contains grey chert clasts. This unit is limited to the north by a fault gouge and could be derived from the adjacent agglomerate and tuff units (Alain Vachon, 1994);
2) Massive green-grey tuff to agglomerate at the top of the unit;
3) Gabbroic sills.

"In the Abcourt-Barvue deposit area late N045° to N090° trending vertical faults are displacing the mineralization from 6 to 15 metres apart. Finally, a NNE trending fault caused an apparent horizontal displacement of more than 300 metres to the south at the eastern limit of the Barvue pit."

3.3 Deposit types

"The Abcourt-Barvue and the entire Barraute area is hosting a wide range of mineralized deposits; an asbestos chrysotile deposit (e.g. Canadian Bolduc mine), Ni-Cu-PGE occurrences (e.g. Consolidated Mogador), Volcanic-Hosted Massive Sulphide (VHMS) deposits (e.g. Vendôme deposit), syenite-associated disseminated gold deposits (e.g. Swanson), related Cu-Mo-Au porphyry occurrences (e.g. Michaud no.1 and no.2), Mo-Bi and Li-Be deposits associated with S-type granitoids (e.g. Quebec Lithium and Molybdenite Corporation mines) and orogenic lode gold deposits (e.g. Bartec).

"The succession through time of different ore deposit environments may indicate crustal thickening, from typical near-surface volcanogenic mineralized hydrothermal systems (2715-2700 My), to deep seated porphyry-style and syenite-associated disseminated gold systems (2682-2672 My), and deeper settings for orogenic lode gold deposits (< 2670 My) mineralized systems. This evolution through time is explained by the paleotectonic evolution of the Abitibi greenstone belt.

"The Héva formation hosts a volcanic massive sulphide deposit known as the Vendôme deposit. Small gold shear zone deposits occur in the immediate vicinity. A nickel-copper showing (Mogador-Vendôme, Ni) was also discovered S-W of the Vendôme deposit in the Héva formation and in contact with an intrusive.

"The Abcourt-Barvue silver-zinc deposit is classified as disseminated volcanogenic sulphide deposit. It shares numerous similarities with the Mattabi-type volcanogenic sulphide deposit. The Zn and Ag sulphides mineralization is composed of disseminated and bedded sphalerite and pyrite found in close association with felsic volcanoclastic rocks.

“Exploration tools and guidelines for the prospecting and delineation of volcanogenic sulphide mineralization are well documented and usually involve a multi-disciplinary approach (e.g. geophysics, geochemistry, and volcanology).”

“The Abcourt-Barvue deposit is a "Zn-Ag volcanogenic disseminated sulphide deposit" associated with felsic pyroclastic rocks. Also referred as tuff/agglomerate, marker tuff and grey schist in log books these rocks are strongly altered in carbonates, sericite and silica with minor pyrophyllite and chlorite. Mineralization has probably been formed close to a local basin, near surface, in association with felsic volcanism. The mineralization has possibly been remobilized later due to regional tectonism, the Kenorean orogeny.

“The Barvue deposit was the most important zinc producer of the province of Quebec in the 1950’s. From 1952 to 1957, 5 002 190 metric tonnes grading 38.74 g/t Ag and 2.98 % Zn (5 514 000 short tons grading 1.13 oz/st Ag and 2.98 % Zn) were extracted. Ore was extracted by an open pit mining operation. During the second production period from 1985 to 1990, ore was extracted by underground mining. This production period yielded 632 319 metric tonnes grading 131.65 g/t Ag and 5.04 % Zn or 697 016 short tons grading 3.84 oz/st Ag and 5.04 % Zn (Source: MRNFP, SIGEOM web site).”

3.4 Mineralization

“The Abcourt-Barvue mineralized zone has a thickness ranging from 2 to 30 metres. The deposit has a known east-west strike length of 2.2 km (Figure 3.4.1). “Mineralization has been delineated by diamond drilling to a maximum vertical depth of 425 m below the surface. Mineralized horizons are dipping at 75° to the north (Figure 3.4.2).

“Zinc and silver are the main metallic constituents of the Abcourt-Barvue deposit and these metals are associated with sphalerite, native silver and argentite. The presence of some gold and copper has been observed. Gold is usually associated with silver and copper occurs mostly at the eastern limit of the Barvue deposit.

“The ore mineralization which is mostly made up of disseminated and bedded sphalerite and pyrite (up to 10 % of the rock) is described as:

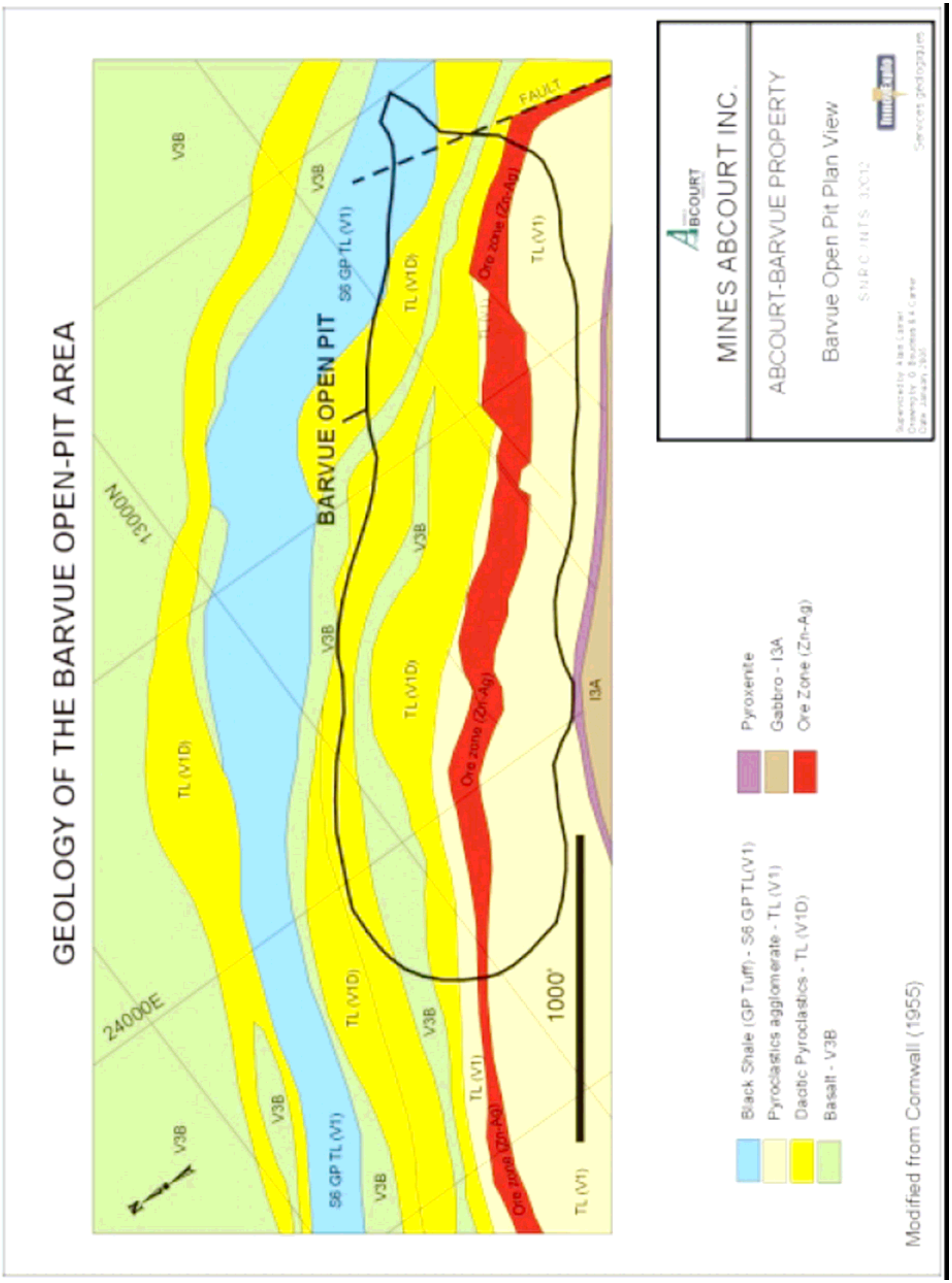

Figure 3.4.1 Geological plan view of the Barvue open pit area (Source: MRB's Resources evaluation report, May 2006).

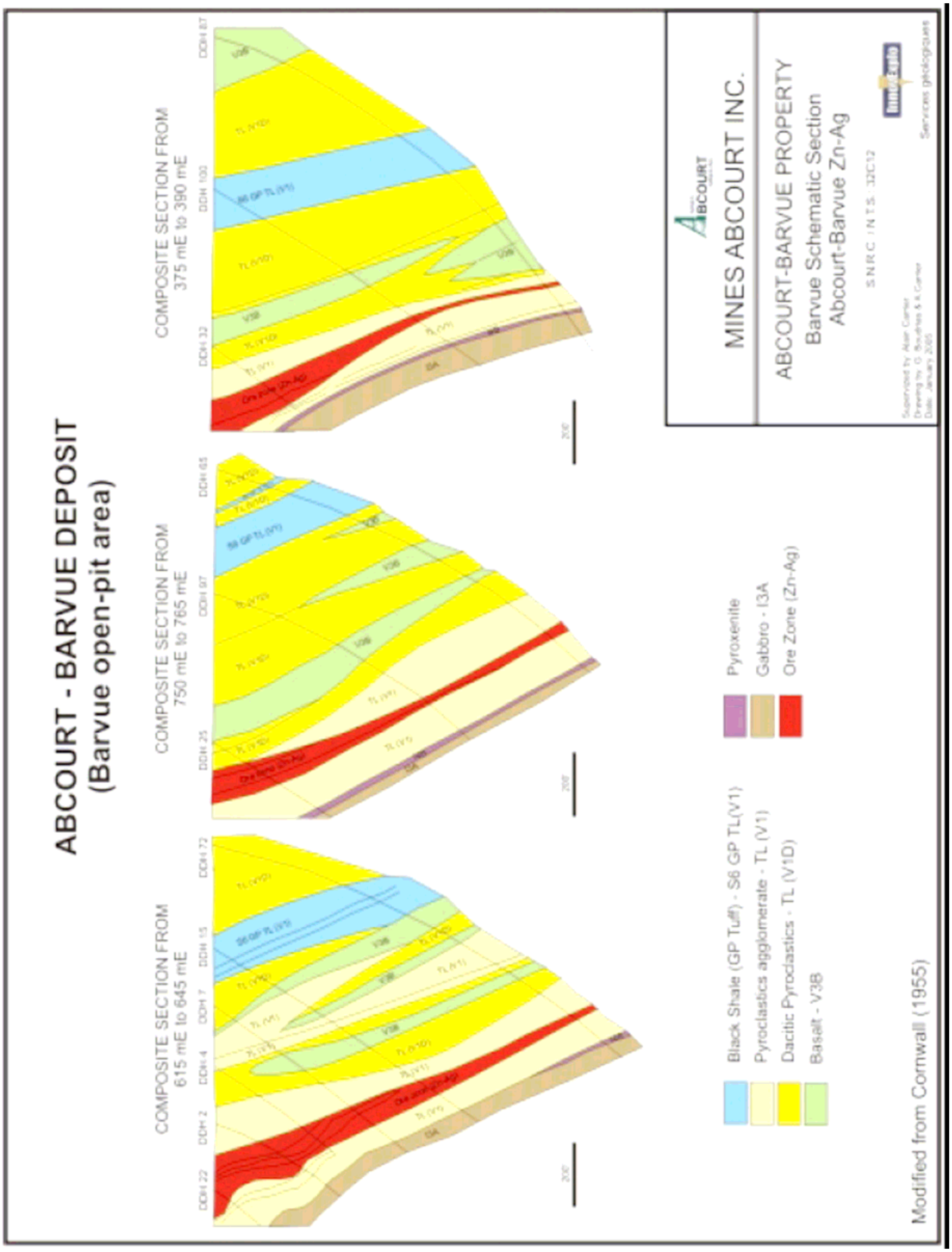

Figure 3.4.2 Schematic cross-sections of the Barvue open pit area (Source: MRB's Resources evaluation report, May 2006).

- iron-poor, honey brown and dark colour sphalerite ($ZnFeS_2$), finely disseminated in the tuff agglomerate horizons concentrated generally to the north of the marker bed, with additional mineralization in the marker bed and in the shear zone;
- 1-2 % disseminated pyrite (some encompassed within the sphalerite);
- minor galena (PbS), chalcopyrite ($CuFeS_2$), native silver (Ag) and proustite (Ag_3AsS_3).

"Thickness of the mineralized horizon tends to increase eastward, most of the sulphides being located in the Barvue area. Most of time, we can observe a zinc enrichment in both footwall and hanging walls of the mineralized zone which are generally separated by lower grade marginal material. These two sub-parallel zinc-enriched units are currently known over a total length of 2.2 km to a vertical depth of 600 metres (Hole AB-92-05) in the mine sector (Vachon. 1994).

"The gangue minerals include siderite, quartz, chlorite, chloritoïd, sericite, illite and rhodocrosite (Cornwall, 1955).

"Previous Abcourt's mining experience has demonstrated that it is possible to apply good grade control during day to day operations with a well trained geological technician (Hinse, 2000) considering that each of the marker beds have their own "mines terminology" and they are well documented."

3.5 Exploration

"Since 1980, Abcourt has carried out almost all the exploration work done on the Abcourt-Barvue property. The main result from these successive exploration programs, in conjunction with a high silver price, was the reopening of the mine in 1985. In 1990, lower silver prices caused the mine closure.

"Between 1990 and 1993, NORANDA (NOREX option) carried out electro-magnetic, magnetic and gravimetric geophysical surveys, a pedo-geochemical survey and more than 4 644 metres of diamond drilling on the Abcourt-Barvue property. Some gravimetric anomalies and several electro-magnetic (EM) conductors compiled by NORANDA are still unexplained and represent potential drilling targets.

"From 1997 to the last drilling program in 2005, all the exploration work has been carried out by Abcourt with the objective of increasing and improving the quality of the resources likely to be exploited by open pit or underground mining to a depth of 150 metres. A total of 10 170 metres of diamond drilling has been completed since 1997, from which an amount of 5 879.2 metres was carried out in 2005 (Section 3.6).

“According to Franklin (2001), the Barraute Area has a very good potential for Volcanogenic Massive Sulphide (VMS type) deposits for three reasons; 1. The area has compositions of volcanic rocks similar to those of highly productive camps; 2. Known deposits are smaller than the norm, meaning that mid to giant size deposits still may be found; 3. The area is under-represented in tonnes of ore as compared to "mature" mining districts such as Val-d’Or, Matagami and Detour.”

3.6 Drilling

“Following recommendations made by Innovexplo, surface holes were drilled by Abcourt in 2005 to increase the tonnage and improve the quality of the resources likely to be exploited by open pit or by underground mining from surface to a depth of 150 metres. Consequently, Abcourt drilled forty-six (46) BQ size holes (AB05-01 to 46 series) totalling 5 879.2 metres. Of that number, 3 200 metres were drilled on the Abcourt Area and 2 679.2 metres on the Barvue Area. This work was performed by Forage Nordic Inc. from Val-d’Or. Most of the holes were drilled near surface to better define the limits of the open pits and to increase the confidence level in the mineral resources.

“The geological description of the diamond drill holes includes the identification of the main lithologies, alteration types and details on textures as well as structures. Drilling was completed according to the standard practice in the industry. Surface drill holes were surveyed for deviation and plotted accordingly. Diamond drill holes show minor deviations. The core recovery was very good with the exception of hole AB05-19, drilled on section 5100 E, and hole AB05-20, drilled on section 5130 E, where 0.4 to 2.0 metres intervals were lost in probable faulted zones. A 0.4 metre mineralized section was also lost in hole AB05-23.

“Among the most significant results obtained in the west segment of the Abcourt area, hole AB05-38 confirmed a 30-metre displacement of the main mineralization to the south. Striations observed on fault planes in the mine when it was in operation, indicate that the displacements produced by faults have a vertical component much greater than those observed in the horizontal plane. Hence, hole AB05-38 indicates that the good values found on section 4995 E may be found towards the west at depths of 300 metres or more.

“On the Barvue’s east extension, holes 12, 13 and 14 located a mineralized segment of the ore zone displaced to the south. The 155 m deep intersection cut in hole AB05-14 (45.8 g/t Ag and 4.40 % Zn over a core length of 4.5 m) is very significant. In fact, at that depth, the nearest intersection is located at a distance of about 150 metres in a north-westerly direction. Consequently, this hole indicates an important extension of the mineralized zone, completely unexplored as of today.”

3.6.1 Sampling method and approach

“Sample lengths vary from 0.16 m to 2.20 m but generally are 1.50 m long. The length of each sample was determined by the physical characteristics of the geology, mineralization and alteration of the core. Composite interval are obtained from sample results and calculated over a minimum horizontal width of 1.52 m (5.0’) in mineral resources estimate.

“During the 2005 drilling program (holes AB05-01 to AB05-46), each mineralized intersection of the BQ size core was saw split. Half of the core was sent to Techni-Lab laboratories in Ste-Germaine-de-Boulé for silver and zinc determination, rarely for copper. A total of 720 samples were selected, identified, bagged and shipped to Techni-Lab.”

3.6.2 Sample preparation, assaying and security

Core logging and sampling was done by Abcourt personnel at the Abcourt-Barvue mine site. Samples were brought directly to the laboratories by the Abcourt staff.

A review of the sampling procedure done by MRB concludes to the accuracy of the practice which is summarized as follows: the samples came from BQ size core which was split in half. Half of the core was kept in core boxes at the Abcourt’s mine site, as witness for future crosschecks or verifications, and the other half was sent to the laboratory for assayings. Original assay certificates are kept by Abcourt.

“Since the Abcourt underground mine closure in 1990, samples from exploration drilling programs were assayed by Techni-Lab S.G.B. Abitibi Inc laboratories at Ste-Germaine-de-Boulé, Quebec. Check samples (pulps and/or rejects) were also assayed at SGS Minerals of Rouyn-Noranda and ALS-Chemex of Val-d’Or.

“Techni-Lab is using the "AA spectroscopy" method for zinc, silver, copper and gold determination. The samples are first dried and crushed at minus 1/8 inch. A 200 to 300 grams sample is then weighted and pulverized at minus 200 meshes (80 %).

“A one (1) to two (2) grams mass is used for analysis by spectroscopy. The principle is to attack the pulverized and dried sample with nitric and chlorhydric acids. The detection limit of this analytical method is 0.5 ppm.

"The Techni-Lab S.G.B. Abitibi Inc. in Ste-Germaine-de-Boulé follows a strict QA/QC program which includes mineralized standards, blank and field duplicates for each batch of twenty-four (24) samples. Each set comprises one blank in the first third, one double in the second third and one standard in the last third. The position of each blank is incremented of one position, from one set to another and it returns to the beginning of the set after the eighth set.

"The blank is used to detect possible contaminations usually introduced by improper cleaning methods between each sample preparation. The pulp duplicate is used to determine the repeatability of the method and the standard is used to monitor the accuracy of the measurements.

"There are three (3) standards used for gold:
1) *Rocklab's ppb standards;*
2) *Rocklab's g/t standards;*
3) *CANMET certified standards, for gold.*

"There are two (2) standards used for all other metals:
1) *In-house, Techni-Lab standards for metals;*
2) *CANMET certified standards for metals.*

"The verification of the standards is made each month on a set of twenty-four (24) samples assayed for gold and other metals. Standards are made in calculating the average of all values without the highest and lowest values of the lot. The certified standard's recovery rate have to be higher than 90 % or else, a revision of this standard or the measuring equipment should be done until reaching a margin of error of less than 10 %.

"The calibration curve should have a coefficient of correlation of 0,995 or else, new standard solutions should be used or a revision of the measuring equipment should be done.

"Internal assay checks and quality control procedures are routinely performed by Techni-Lab which has an ISO/CEI 25 Certification."

3.6.3 Data verification by MRB

During its NI 43-101 technical report process, MRB reviewed all data available and Mr Jean-Pierre Bérubé (MRB) visited the Abcourt-Barvue property. The site visit included core review of borehole AB05-02. The following activities was also performed:

- validation of the drill hole database;
- definition of the size of mineralized blocks derived from Abcourt's geological interpretation based on past production and field work;
- block category allocation depending on density of information (old pit outline, underground muck and chip sampling, underground definition drilling and near surface diamond drill intersections.

Furthermore, fourteen (14) samples of borehole AB05-02, for a total continuous length of 21 metres, were sent lately by MRB to Laboratoire Expert Inc. (Expert) in Rouyn-Noranda to be re-assayed for silver and zinc. The results were sent directly to MRB's office in Val-d'Or.

MRB observed that, on the average, Expert's silver and zinc values are respectively 2.2 % and 0.2 % lower than Techni-Lab's original values over the whole length of the 21 m intersection resulting in overall values being quite the same between the two labs.

For a particular interval, high silver and zinc values are respectively 8 and 10 % higher than Techni-Lab's original values. It seems that high grade values were under evaluated by Techni-Lab (or over estimated by Expert) giving a favourable level of confidence in the estimation of the resources calculated by Abcourt. It is noteworthy that the reconciliation of historical underground production from 1985 to 1990 demonstrates that the recovered silver grade was better than expected but the zinc grade was lower than estimated.

MRB explains the difference in the zinc grade by a combination of factors:

- unreported losses in the mill as the ore was custom milled by Noranda at Matagami and because the flotation circuit and concentrate handling equipment was not set-up to report any loss to Abcourt;
- a grade too high given to the dilution material (1.73 % Zn);
- a relatively high population of uncorrected high zinc assays (over 10 % Zn) in the ore reserve calculations;
- an incomplete extraction of the high grade material during the mining operation.

Note : In its own mill, Abcourt will not have any unaccounted for losses and in the current resource calculations, the number of high (over 10 % Zn) zinc assays is minimal. In addition, the zinc grade given to the dilution material is only 0,80 % Zn in pits and 0,00 % Zn in underground stopes. With these corrections, we don't expect any difference in zinc grade in the future.

3.7 Mineral resources

Results from MRB's resources validation and mineral resources updating of the Abcourt-Barvue Zn-Ag deposit are presented in this section, according to the CIM Standing Committee on Reserves Definitions (CIM Definition Standards, Nov. 1, 2004):

"A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that is has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

The MRB's resources estimate of the Abcourt-Barvue Zn-Ag deposit are based on previous evaluation works done by Abcourt, Roche ltd (1999), Gescad (2004) and Innovexplo (2005). MRB has either: (i) validated and re-classified or (ii) re-calculated resources. These estimates were based on all results made available by Abcourt.

The recommendations of CIM Standing Committee on Reserves Definitions (CIM Definition Standards, Nov. 1, 2004) were used in defining the resource classification. Thus, in accordance with MRB's statement, results of the resource estimate are compliant to regulation of the National Instrument 43-101 and its addendum 43-101F.

The resource estimate was calculated along the entire length of the Abcourt-Barvue deposit which is 2 170 metres long to a maximum vertical depth of –400 metres from surface.

3.7.1 Geological interpretation

"The geological interpretation made on cross-sections by Abcourt and Gescad (2006) conveniently traces the outline of the Abcourt-Barvue mineralization, is concordant with the different volcanic units and fits well with the regional geological interpretation. Lateral and at-depth geological continuity of the mineralization is confirmed by the Barvue open pit grade control maps, geological mapping of underground workings and by numerous drill holes intersects on 15 m spacing cross-sections. Due to the continuous nature of the mineralization and the simple geometry, differences in hole-to-hole correlation do not appear to be a possible source of discrepancies. The character of the Abcourt-Barvue deposit allows us to make a reliable estimate of the size, tonnage and grade of the zinc-silver mineralization.

"No data located outside the 2 170 m east-west strike of the deposit and below a vertical depth of 400 metres were incorporated in the resources estimate. The geological interpretation can however easily be extended laterally and at depth as further exploration drilling could lead to substantial findings."

3.7.2 Resource estimation methodology

The polygonal method was used in previous estimation work completed by Abcourt, Roche (1999), Gescad (2004) and Innovexplo (2005). Both the Abcourt and Barvue portions of the deposit were entirely reviewed, re-calculated and classified into measured, indicated and inferred mineral resources by MRB. Detailed calculations for all the estimated blocks are provided and illustrated on the longitudinal views in MRB's technical report (2006).

Polygons have been traced using two different approaches: (i) on cross-section for measured and indicated mineral resources and (ii) on longitudinal views for inferred mineral resources.

"Measured and indicated resources blocks were delimited on cross-section using the parameters defined on the following page (see section 3.7.3) in areas characterized by a greater density of geological data (ex: fringes of open pit or underground workings, or areas with regular drill holes spacing of 15 m to a maximum of 30 m). The mid-distance rules have been applied on sections and laterally between drill hole intercepts for defining the resources blocks. Grade of the estimated blocks were obtained from weighted averages of drill hole intercepts or from underground sampling when available. Areas of influence for each polygon were obtained on sections from Autocad drafting tools. Lateral influences of the polygons were defined on the longitudinal view using the mid-distance rule. Usually 7.5 m away and 7.5 m towards for measured resources obtained from sections drilled on a 15 m spacing and 15.0 m away and 15.0 m towards for indicated resources from sections drilled on 30 m spacing. The influence of one drill hole intersection on a cross-section mainly represents approximately 15 m (down dip) for measured resources and 30 m (down dip) for the indicated resources. Some exceptions were made for the measured resources when it was clear that adjacent sections were showing a good continuity over a tighter pattern of drilling. In those cases, the maximum down dip extension of the mineralized intersections may have been extended up to 30 metres. MRB also used the metal factor (grade x thickness) for each of the intersections located in the west side of the deposit (Abcourt) to see if there was any "weakness" in the distribution of the zinc-silver mineralization on the longitudinal view.

“Inferred mineral resources were obtained in a second step from polygons drafted on the longitudinal view in areas characterized by fewer drill hole intercepts (ex: drill holes spacing of 60 m and more). The area covered by the measured and indicated resources was transferred on the longitudinal view and was used as a boundary, limiting the area of influence of the inferred blocks. A 70 m distance from drill holes intercepts was used until reaching a mid-distance between drill hole intercepts or reaching the limit defined by the measured and indicated resources. The area covered by each inferred polygon was calculated with the Gemcom and Autocad softwares. The weighted averages were calculated from drill hole intercepts while horizontal widths combined with longitudinal areas were used to calculate the volume of each resource block.

“The in-situ and undiluted mineral resources estimate are presented with a cut-off grade (Ag expressed in Zn equivalent + % Zn) of 2.40 % Zn for the Abcourt open pit, 2.55 % Zn for the Barvue open pit and finally, 3.20 % Zn for the underground operations. The preliminary open pit outlines were provided by Abcourt and the resources estimate was established within these limits.”

3.7.3 Resources categories definition

Resources categories for the Abcourt-Barvue resources estimate follow the recommendations of the CIM Standing Committee on Reserves Definitions:

"Mineral resources are sub-divided, in order of increasing geological confidence into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource."

Measured mineral resource

According to CIM Definition Standards, the definition of a measured resource is:

"A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity."

At Abcourt-Barvue, the nature and character of the geology and the grade continuity of the Zn-Ag volcanogenic mineralization is sufficiently confirmed by closely spaced drill holes. For the measured resource category, MRB has used the following parameters:

- maximum distance of fifteen (15) m or until reaching a mid-distance (hole to hole distance measured on section from mid-point to mid-point of the mineralized intervals);
- lateral influence of blocs defined on a longitudinal view;
- polygonal method on cross-sections;
- resources blocks are drawn on cross-sections only.

Indicated mineral resource

According to CIM Definition Standards, the definition of an indicated resource is:

"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed."

For the indicated resource category, MRB has used the following parameters:

- maximum distance of thirty (30) m or until reaching a mid-distance (hole to hole distance measured on sections from mid-point to mid-point of the mineralized intervals);
- lateral influence of blocs defined on a longitudinal view;
- polygonal method on cross-sections;
- resources blocks are drawn on cross-sections only.

Inferred mineral resource

According to CIM Definition Standards, the definition of an inferred resource is:

"An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity.

The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes."

For the inferred resource category, MRB has used the following parameters:

- maximum distance of seventy (70) m or until reaching a mid-distance (hole to hole distance measured on a longitudinal view from mid-point to mid-point of the intervals);
- polygons defined on a longitudinal view are limited by the area covered by the indicated and measured resources blocks (previously delimited from cross-sections and reported on the longitudinal view).

It is of MRB's opinion that inferred resources are only indicative of areas having the potential to be upgraded to the indicated or measured categories depending on the amount of information made available by subsequent surface and/or underground works. Considering the lower level of confidence given by this category, the inferred resources will not be used by the author (GENIVAR) to evaluate reserves.

3.7.4 Specific gravity

The specific gravity previously used by Abcourt in resources estimate and during the underground mining operations (between 1985 and 1990) was 3.0 g/cm^3. This specific gravity has been obtained from tests done on representative core samples coming from the mineralized zones of both the Abcourt and Barvue portions of the deposit.

Fixed density measured on samples	*Abcourt (1984)*	*3.02*
	Barvue	*2.94*
	Average used	***3.00***

Moreover, the results from flotation tests conducted at the Novicourt mine's laboratory on representative bulk samples of the Abcourt and Barvue silver/zinc ore bodies revealed that specific gravity tests confirmed the factors used in the calculations of the resources estimation made in March 2005 by Innovexplo (Abcourt's Press Release dated January 31, 2005). MRB also agrees on the specific gravity of 3.0 g/cm^3 used by Innovexplo (called fixed density by them) and MRB's calculations were made using the same number.

3.7.5 Minimum width

"A minimum horizontal width of 1.52 m (5.0') was used for defining the minimum mineralized intercepts (drill holes or underground sampling) by using the grade of the adjacent material when assayed or a value of zero when not assayed. Mineralized intervals at Abcourt usually reach horizontal widths of 4 to 5 m. These intervals are much wider in the Barvue Area."

3.7.6 Zinc grade equivalence (EQ) as estimated by Abcourt

The Abcourt-Barvue's mineralization is polymetallic with silver and zinc being the most abundant elements. So, silver values are expressed in zinc equivalent to obtain a zinc grade equivalence (identified by EQ). For the mineral deposit, the grade equivalent is the sum of the zinc (%) and silver (g/t) contents of each mineralized intersection according to the following equation:

$$\text{Grade equivalence} = ((\text{g/t Ag} / 31.105) \times 0.65) + \%\ \text{Zn}$$

The silver value (in g/t) divided by 31.105 gives the silver value expressed in oz/t. This silver value has to be multiplied by a silver/zinc ratio (0.65) determined by calculations which take into account several parameters:

- estimated zinc and silver grades from resources evaluation;
- milling processes used and expected metal recoveries;
- rate of concentration for zinc;
- silver grade in zinc concentrate;
- NSR for one tonne of zinc concentrate at a price of 1.00 US$/lb for zinc, 10.00 US$/oz for Ag and a rate of exchange of 1 C$ for 0.85 US$;
- smelting, transportation and other costs.

We can then summarize all those calculations as follows:

- Abcourt's open pit:
  The NSR (Net Smelter Return) for 1 % zinc = 12.83 US$ and the NSR for 1 oz/t Ag = 8.29 US$. So, 1 oz/t Ag = 0.65 % Zn.
- Barvue's open pit:
  The NSR for 1 % zinc = 12.86 US$ and the NSR for 1 oz/t Ag = 8.08 US$. So, 1 oz/t Ag = 0.63 % Zn.

- Underground operations:
  The NSR for 1 % zinc = 12.83 US$ and the NSR for 1 oz/t Ag = 8.29 US$. So, 1 oz/t Ag = 0.65 % Zn.

Consequently, the zinc grade equivalence for each ounce of silver was set at 0.65 % Zn. It means that an intersection grading 100 g/t Ag and 2 % zinc has a zinc percentage equivalent of 4.09 % Zn, i.e.: ((100 g/t Ag / 31.105) x 0.65) + 2 = 4.09.

GENIVAR, as well as MRB, agrees with Abcourt on the calculation procedures and numbers used to set this silver to zinc ratio.

3.7.7 Cut-off grades as estimated by Abcourt

The cut-off grades were conservatively estimated by Abcourt by using a price of 1.00 US$ per pound for zinc, 10.00 US$ per ounce for silver, an exchange rate of 0.85 US$ for each Canadian dollar and taking into account the average grade of different parts of the ore body, the planned milling process and estimated costs. Abcourt has established the cut-off grades of each zone as follows:

- Abcourt's open pit: 2.40 % zinc equivalent;
- Barvue open pit: 2.55 % zinc equivalent;
- Underground: 3.20 % zinc equivalent.

During production, these cut-off grades will be adjusted to take into account the prevailing zinc and silver prices. Once again GENIVAR, as well as MRB, concurs with Abcourt on the calculation procedures and numbers used to set these cut-off grades.

3.7.8 Cutting value of high grade assays

An exhaustive statistical analysis was conducted by Innovexplo in 2005 and their observations, comments and conclusions are reported in their integrity hereafter.

The Abcourt-Barvue deposit is characterized by some high grade silver values (mainly in the Abcourt zone), requiring that a high grade assay cutting value be established in order to reduce the risks of overestimating the total silver content of the deposit. The high grade assay cutting value parameters have been established by Abcourt. A high grade cutting value of 514.5 g/t silver (15 oz/t Ag) was used by Abcourt during mining operations (between 1985 and 1990) and in previous resources estimate. For zinc, no cutting value was used.

A LOG Normal Probability Plot indicates that the zinc assays have sharp breaks in slope at seemingly key grade levels. From 0 to 0.1 % Zn a steady slope is probably indicative of the waste population of Zn samples. A low grade ore population is probably represented by the steady slope from 0.1 % to approximately 1.5 % Zn. The distinct change in slope from 1.5-10 % Zn is undoubtedly indicative of the ore population. The break in slope at +10 % is indicative of a high grade nugget population. However, the relatively low coefficient of variation (1.2) for the data set is indicative that the high grade nugget population is not large.

A review of the basic statistics for Zn for the two individual data sets demonstrates significant differences in mean, median, standard deviation and coefficient of variation. The Barvue data tends to have higher grades with mean and median grades greater than 2 and 3 times the mean and median grades of the Abcourt data set. A larger standard deviation also exists in the Barvue data set. However, the coefficient of variation is lower at 1.02 indicating the high grade nugget population is small. In general, the Barvue data contains more ore grade values in the typical ore grade range than the Abcourt data but it also has more of the high grade nugget populations than Abcourt.

In the case of Zn, the high grade population is not considered to be as significant as the coefficient of variation is near 1. As such, using the recommended indicated threshold cutting value of 10 % Zn or not applying a cutting factor at all is immaterial for the resources calculations. However, similar to the Ag results, the Zn populations demonstrate high means in comparison to their median values. A sensitivity analysis of the resources estimate should therefore include similar variations in the Zn cutting grade used.

A LOG Normal Probability Plot for silver shows a quite different trend than zinc. The graphic displays much lesser changes in slope than zinc. The changes in slope are more pronounced for the entire data set than for the Abcourt subset alone. Closer examination of the results from Barvue and Abcourt separately clearly demonstrates that these changes in slope are far more significant in the Barvue Ag subset and may be indicative of a zoning or change in geological controls similar to what was found for the zinc results. These grade level breaks for Ag are well demonstrated on the Probability Plot for the Barvue data subset. A steady slope up to 3.5 g/t Ag is indicative of waste and the sharp change in slope from 3.5 to approximately 20-30 g/t Ag demonstrates the likelihood of a low grade halo of mineralization. Another relatively sharp change in slope from 30 g/t Ag to +1 000 g/t Ag marks the geological boundary for ore grade mineralization such as would occur with semi-massive to massive sulphides or veins. The sharp break in slope above the 1 000 g/t Ag marker visible on each of the individual data sets and the entire Abcourt-Barvue data set is indicative of a high grade nugget population. The mean, standard deviation and co-efficient of variation is very high for the entire silver population. The high coefficient of variations indicates a very large high grade nugget population. The low median values compared to the means indicate that the high grade samples strongly influence the data set and are an important component of the ore.

Further evaluation of this upper high grade silver population would help to determine an appropriate value. A cut-off value anywhere below 1 000 g/t Ag (as applied in Abcourt's and MRB's calculations) is not supported by the plots and it would seem that a cut-off of 1 000 g/t Ag should be more appropriate. In fact, high grade silver values were cut to 1 000 g/t Ag (30 oz/st Ag) under the management of Manitou-Barvue Mines Ltd on Barvue's open pit. Abcourt used an even more conservative cut-off grade of 514.5 g/t Ag for this mineral resources evaluation.

3.7.9 Resources estimate summary

The measured and indicated resources for the Abcourt-Barvue deposit before dilution, as estimated by MRB, amount to 7 018 969 tonnes grading 61.19 g/t Ag and 3.33 % Zn. The measured resources are 6 515 863 tonnes at 58.32 g/t Ag and 3.33 % Zn and the indicated resources are 503 106 tonnes at 98.35 g/t Ag and 3.44 % Zn.

The inferred resources before dilution total 1 505 687 tonnes grading 120.53 g/t Ag and 2.98 % Zn.

The resources estimate are undiluted and in situ and based on a cut-off grade of 2.4 EQ for the Abcourt open pit, 2.55 EQ for the Barvue open pit and 3.2 EQ for all the underground resources. At 10.00 US$/oz Ag and 1.00 US$/lb Zn, these cut-off are roughly corresponding to a net smelter return of 30 US$/t for open pit ore and 41 US$/t for underground resources.

Table 3.7.1 gives more details about resources summary while Figure 3.7.1 is the schematic longitudinal view of the Abcourt-Barvue deposit showing the measured, indicated and inferred resources.

Table 3.7.1 Abcourt-Barvue deposit resources estimate (Source: MRB's Technical Report, 2006).

Abcourt-Barvue mineral resources estimate – Summary (undiluted)

	Tonnage (t)	Ag (g/t)	Zn (%)	Oz of Ag	Tonnes of Zn
Measured resource	6 515 863	58.32	3.33	12 217 971	216 677
Indicated resource	503 106	98.35	3.44	1 590 843	17 284
Measured & Indicated resources	7 018 969	61.19	3.33	13 808 814	233 961
Inferred resources	1 505 687	120.53	2.98	5 834 727	44 874

Note: These amounts include in situ and undiluted resources for open-pit and underground operations.

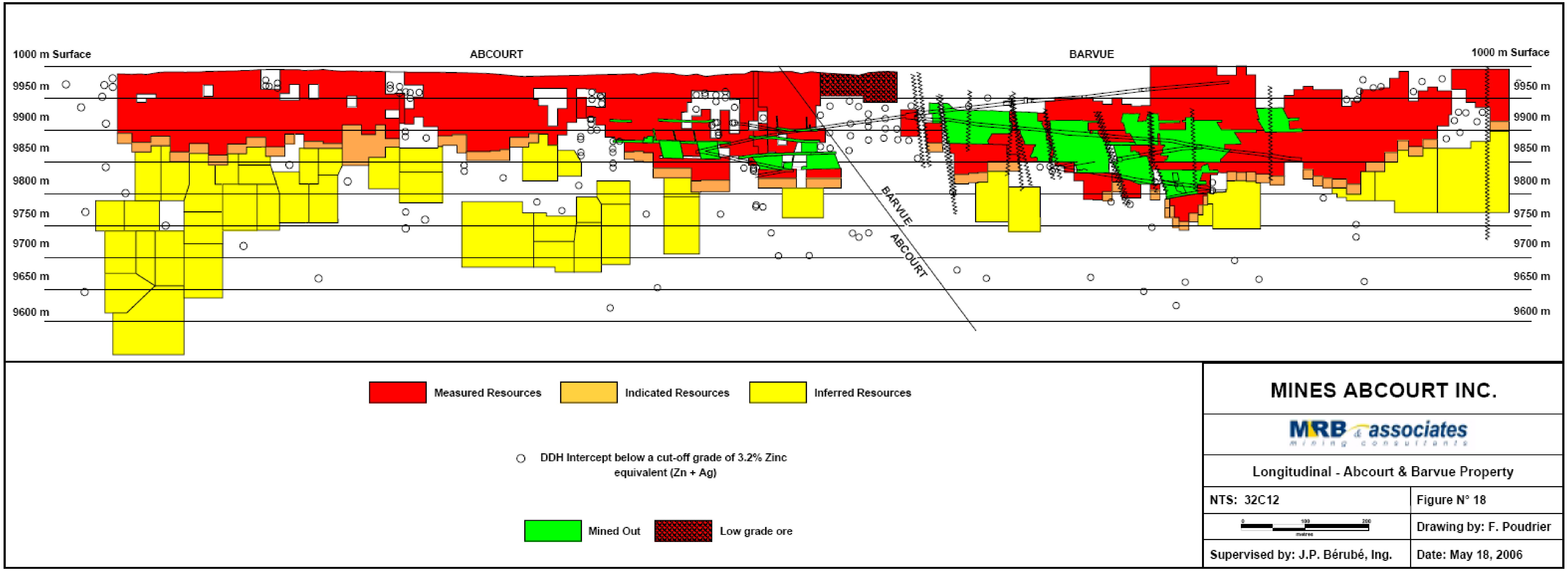

Figure 3.7.1 Schematic longitudinal view of the Abcourt-Barvue deposit (Source: MRB's Technical Report, 2006).

The next tables (Table 3.7.2 to Table 3.7.6) contain tonnage and grade of mineralization by area and resource category and same details for the low grade ore.

Table 3.7.2 Abcourt area resources estimate using a cut-off grade of 2.4 % Zn equivalent for open pit sector and 3.2 % Zn equivalent for underground sector (Source: MRB's Technical Report, 2006).

Abcourt Area Resources (EQ = Ag in Zn equiv. + % Zn)						
	Tonnage (t)	Ag (g/t)	Zn (%)	Oz of Silver	Lbs of Zinc	EQ
Open pit (cut-off grade of 2.4)						
Measured resource	1 009 151	81,52	2,87	2 644 954	63 862 607	4,57
Indicated resource	0	0,00	0,00	0	0	0,00
Open pit total	**1 009 151**	**81,52**	**2,87**	**2 644 954**	**63 862 607**	**4,57**
Underground (cut-off grade of 3.2)						
Measured resource	1 047 878	114,31	3,12	3 851 170	72 089 815	5,51
Indicated resource	232 214	111,53	3,43	832 679	17 562 693	5,76
Underground total	**1 280 092**	**113,81**	**3,18**	**4 683 849**	**89 652 508**	**5,55**
Measured and indicated resources	**2 289 243**	**99,57**	**3,04**	**7 328 803**	**153 515 115**	**5,12**

Table 3.7.3 Barvue area resources estimate using a cut-off grade of 2.55 % Zn equivalent for open pit sector and 3.2 % Zn equivalent for underground sector (Source: MRB's Technical Report, 2006).

Barvue Area Resources (EQ = Ag in Zn equiv. + % Zn)						
	Tonnage (t)	Ag (g/t)	Zn (%)	Oz of Silver	Lbs of Zinc	EQ
Open pit (cut-off grade of 2.55)						
Measured resource	4 116 978	38,11	3,41	5 044 466	309 557 634	4,21
Indicated resource	0	0,00	0,00	0	0	0,00
Open pit total	**4 116 978**	**38,11**	**3,41**	**5 044 466**	**309 557 634**	**4,21**
Underground (cut-off grade of 3.2)						
Measured resource	341 856	61,63	4,28	677 381	32 262 318	5,57
Indicated resource	270 892	87,05	3,44	758 163	20 547 700	5,26
Underground total	**612 748**	**72,87**	**3,91**	**1 435 544**	**52 810 018**	**5,43**
Measured and indicated resources	**4 729 726**	**42,61**	**3,47**	**6 480 010**	**362 367 652**	**4,37**

Table 3.7.4 Total resources estimate for the open pits and the underground sectors (Source: MRB's Technical Report, 2006).

Total Resources (EQ = Ag in Zn equiv. + % Zn)

	Tonnage (t)	Ag (g/t)	Zn (%)	Oz of Silver	Lbs of Zinc	EQ
Open pit (cut-off grade of 2.4 and 2.55)						
Measured resource - Undiluted	5 126 129	46,66	3,30	7 689 420	373 420 242	4,28
Indicated resource - Undiluted	0	0,00	0,00	0	0	0,00
Open pit total	**5 126 129**	**46,66**	**3,30**	**7 689 420**	**373 420 242**	**4,28**
Underground (cut-off grade of 3.2)						
Measured resource	1 389 734	101,35	3,41	4 528 551	104 352 133	5,52
Indicated resource	503 106	98,34	3,44	1 590 843	38 110 393	5,49
Underground total	**1 892 840**	**100,55**	**3,41**	**6 119 393**	**142 462 526**	**5,51**
Measured and indicated resources	**7 018 969**	**61,17**	**3,33**	**13 808 814**	**515 882 768**	**4,61**

Table 3.7.5 Abcourt-Barvue inferred resource estimate with a cut-off grade of 3.2 % Zn equivalent (Source: MRB's Technical Report, 2006).

Abcourt-Barvue Inferred Resources (UG only) (cut-off grade of 3.2 (Ag in Zn equiv. + % Zn))

	Tonnage (t)	Ag (g/t)	Zn (%)	Oz of Silver	Lbs of Zinc	EQ
Abcourt	1 144 406	143,15	2,56	5 267 071	64 599 430	5,55
Barvue	361 281	48,87	4,31	567 656	34 334 520	5,33
Total inferred	**1 505 687**	**120,53**	**2,98**	**5 834 727**	**98 933 950**	**5,50**

Table 3.7.6 Internal marginal material (low grade ore with cut-off between 1.5 and 2.4 and less than 1.5 % Zn equivalent) within the open pit limits (Source: MRB's Technical Report, 2006).

Abcourt-Barvue open pit internal marginal material (per zone with cut-off grades between 1.5 and 2.4 and < 1.5))

	Tonnage (t)	Ag (g/t)	Zn (%)	Oz of Silver	Lbs of Zinc	EQ
Cut-off 1.5 to 2.4						
Abcourt	153 951	30,76	1,48	152 258	5 034 221	2,13
Barvue	963 723	16,95	1,75	525 318	37 123 911	2,10
Total	**1 117 674**	**18,86**	**1,71**	**677 576**	**42 158 132**	**2,10**
Cut-off < 1.5						
Abcourt	49 675	8,81	0,71	14 063	774 401	0,89
Barvue	173 975	14,03	0,98	78 494	3 751 753	1,27
Total	**223 650**	**12,87**	**0,92**	**92 556**	**4 526 154**	**1,19**

4. MINING AND MINERAL RESERVES

The Abcourt-Barvue measured and indicated resources along the mineralized structure span a distance of 2 230 m (1 080 m west of the shaft in an E-W direction, and 1 150 m east of the shaft in a S49°E direction). The dip is 75° to 90° to the north.

In Barvue, the ore lays in the south wall of the old pit, at its eastern extremity and below the current pit floor (76 m deep) to a maximum depth of 240 m. The mining proposal for Barvue includes deepening and expanding the pit to the east and at depth to 166 m from surface and underground mining of the remaining measured and indicated resources at depth.

In the Abcourt sector it is proposed to mine the upper part of the ore body by open pit to a maximum depth of 72 m. Mining under the pit will proceed with underground methods using three declines for access and trackless equipment to a depth of 150 to 200 m where Avoca cut-and-fill stopes will be developed.

Assuming prices of 1.00 US$ per pound for zinc and 10.00 US$ per ounce for silver, an exchange rate of 0.85 US$ per 1.00 C$ and taking into account the average grade of different parts of the ore body, the planned milling process and estimated costs, cut-off grades were established as follows (section 3.7.7):

- Abcourt open pits: 2.40 % zinc equivalent;
- Barvue open pit: 2.55 % zinc equivalent;
- Underground: 3.20 % zinc equivalent.

4.1 Open pit mining

Mining will be carried out using conventional open pits method. Drilling will be performed by conventional production drills. Blasting operations will use an emulsion-ANFO (ammonium-nitrate fuel oil) and a down-hole delay initiation system. Hydraulic backhoe shovels and a front-end loader will be used to load rigid haulage trucks of 62-t capacity.

For most of the open pit tonnages (ore and waste), haulage trucks will exit from Abcourt and Barvue pits onto a common haul road approximately 0.5 km west of the plant site. Waste rock will be hauled to two (2) different storage areas, one located 0.9 km from the exit of the Barvue pit on its north side and the other located 0.2 km from the Abcourt pit exit to the south of the Abcourt pit.

Open pit mining will occur at a maximum annual production rate of 6 Mt of rock per year of which 650 000 tonnes (1 800 tpd) will be ore and the remaining 5 350 000 tonnes will be marginal ore and waste rock. More details are given on the following pages.

4.1.1 Pit slope analysis

In the reports entitled "Stability analysis of the Abcourt-Barvue pit extension, Part 1 – Review analysis of structural data, December 2002, and Part 2 – Analysis of structural strength and stability, September 2003", François Charette, Eng, M.Sc., made the following conclusions. Summary of work performed for each study are also presented hereafter.

1st "The objectives of the study is to assess the stability of the open cast excavations. Several field surveys have been conducted on the property to assess the mining potential and the physical feasibility of the mining operation. Most interesting data for the purposes of the present report are the oriented diamond drill holes survey and the field mappings."

"The most prominent structures on the Abcourt-Barvue property are the N-S faults and the major E-W fault. In addition, shear structures oriented roughly NNE are also very common. Joint sets with NNW orientation constitute the second most common joint orientation. From the Abcourt (to the West) to the Barvue (to the East) zones, part of the joint network seems to slightly rotate clockwise with most of the network preserving its relative joint sets arrangement."

2nd "This report presents the results of back analysis of the existing slopes on the Abcourt-Barvue property as well as forward stability analysis of the planned open cast excavation on the same property. Mechanical strength properties used were inferred from structural analysis performed in a previous report (Part 1) submitted to Abcourt Mines Inc. Safety factors were calculated for typical sections all along the planned excavations and potential modes of instability were analyzed and discussed."

"The stability of the Abcourt-Barvue site showed that it is possible to deepen the Barvue pit at a 60° slope and more. Similarly, the excavation of the Abcourt pit at a slope of 60° is also possible. Almost all possible instabilities (plane, wedge and toppling failure potential) were found in conjunction with fault zones and those areas will need more attention during the excavation phase" but "their influence on the feasibility is considered minor".

> "The study also showed that values of factor of safety are high enough to allow steeper slope in many locations. So locally, slope angle could be increased safely, based on the experience acquired during the operation. Pit walls should be kept well drained to maintain maximum stability, maximize slope angle and recovery of ore and minimize waste extraction."

Thus 60° wall slopes will be used by GENIVAR in the designs of the Abcourt and Barvue open pits.

4.1.2 Pit water inflow

The majority of inflow into the open pits will result from the drainage of groundwater and from precipitation directly into the pits. Almost no contribution will come from surface runoffs given the relatively flat topography in the surrounding pits area and collecting ditches at the periphery of the pits.

During the 1983-1990 production period, the average water inflow was estimated at 190 imperial gallons per minute or 1 250 m^3/d. Water inflow rates in the Abcourt pits will probably account for an additional 50 imperial gallons per minute or 330 m^3/d.

Further evaluation will be required in order to determine the optimum pumping and water storage capacity for each pit to handle anticipated precipitation events.

4.1.3 Net smelter return values

For a Zn-Ag deposit such as at Abcourt-Barvue project, the NSR value of the resource is used to determine the net value of each mining block (or polygon). The NSR values were calculated during the establishment of zinc grade equivalence to transfer Ag grade into Zn grade. During the calculation procedures, a number of assumptions and parameters were utilized such as estimated Zn and Ag grades from resources evaluation, expected metal recoveries, rates of concentration and smelting, transportation and other fees (see sections 3.7.6 and 3.7.9)

In summary, based on a price of 1.00 US$ per pound for zinc, 10.00 US$ per ounce for silver and a rate of exchange US$/C$ of 0.85, the NSR values for the Abcourt and Barvue open pits are respectively 12.83 and 12.86 US$/t of ore per one percent of zinc equivalent. For pit design purposes, the minimum NSR values required to make a mineralized polygon economically mineable are 30.79 US$/t (cut-off grade of 2.40 % Zn EQ) and 32.79 US$/t (cut-off grade of 2.55 % Zn EQ) for Abcourt and Barvue open pit ore respectively. The higher cut-off grade for the Barvue open pit is needed because of an anticipated higher waste to ore ratio.

4.1.4 Model preparation

Geological interpretation done by Abcourt, Gescad, Innovexplo and updated by MRB on a 15-metre vertical section pattern is the basis of the model for the open pit design.

Polygons on each cross-section (measured resources) were transposed in plane at mid-bench elevation and extended mid-distance between adjacent cross-sections (7.5 metres apart). Result for each open pit mining bench is shown in polygons of mineralization with Zn and Ag grades derived from estimated grades of cross-section polygons intersected all along the strike of the Abcourt-Barvue deposit.

Furthermore, the Gs geological unit interpreted on cross-sections, which hosts a major deformation zone, was also transposed in plane. These limits will be used during the open pit design process to avoid permanent pit wall location in this geological unit for stability reasons.

4.1.5 Open pit design

On a 10-year life span, the Abcourt-Barvue deposit is amenable to development as an open pit mine for more than 80 % of its tonnage of measured resource.

In the creation of toe and crest lines with Surpac's pit design tool utilities, the following parameters were used to design the open pits:

- Average mineralization dip: 75° to 90° to the north;
- Bench height:
  - Barvue pit : ± 17 m (in two cuts in ore);
  - Abcourt pits : 16 m (in two cuts in ore).
- Individual bench slope: 70° to 90°;
- Inter-ramp slope angle: 60°;
- Berm width: 7.5 – 11.0 m;
- Berm width to overburden slope: 6 m;
- Maximum depth of Barvue pit: 166 m;
- Maximum depth of Abcourt pits: 72 m;
- Ramp width:
  - Barvue and East Abcourt pits: 8 m and 16 m;
  - West Abcourt pit: 12 m and 19 m.

• Ramp gradient:	10-12 %;
• Density of ore:	3.0 t/m³;
• Density of waste:	2.8 t/m³;
• Slope in overburden:	2.5H:1V.

The specific gravity or density of waste of 2.8 t/m³ is based on past production figures from open pit mining in Barvue.

Referring to the Quebec's regulation respecting occupational health and safety in mines (S-2.1, r.19.1):

"45. Haulage roads used by motorized vehicles in an open-pit mine shall:
(1) be edged by a pile of fill or a ridge where vehicles could fall more than 3 metres (9,8 ft.). The pile of fill or the ridge shall have a height equal to at least the radius of the largest wheel of any vehicle travelling on the road. A pile of fill or a ridge is also required along the edge of dumps;
(2) be maintained by clearing or scarifying or by spreading an abrasive substance, so as to keep a non-skid surface."

"45.1. In addition to the standards prescribed in section 45, haulage roads:
(1) constructed from 1 April 1993 and used by motorized vehicles in an open-pit mine shall have a width at least equal to:
  (a) one and one-half times the width of the widest vehicles if they are single-track roads;
  (b) two and one-half times the width of the vehicles if they are 2-way roads;
(2) constructed in an open-pit mine at which operations begins from 1 April 1993 and used by motorized vehicles shall have a width at least equal to:
  (a) twice the width of the widest vehicles if they are single-track roads;
  (b) three times the width of the vehicles if they are 2-way roads."

Following this regulation, ramps in the Barvue pits and the most easterly Abcourt pit which are an expansion of the existing pit in production before April 1st 1993, will be 1.5 (one-way) or 2.5 (two-way) times the width of the widest vehicle. Ramps in the remaining Abcourt pit to the west (new pit in production after April 1st 1993) will be 2 (one-way) and 3 (two-way) times the width of the widest vehicles.

Main pit ramps were designed at a gradient of 10 % and to respect regulation, ramp widths of 16 m and 19 m in Barvue/East Abcourt and West Abcourt pits respectively were established to facilitate two-way truck traffic. Final pit bottoms access ramps were designed at a gradient of 12 % and widths of 8 m and 12 m, again in Barvue/East Abcourt and West Abcourt pits respectively, to accommodate one-way traffic. This configuration results in an overall slope angle ranging between 45° and 55°.

For the Barvue pit, bench floor elevations were established to fit with the floor of existing underground works (drifts, cross-cuts, sub-level developments) that will be mined during open pit production phase. Accordingly, bench heights vary between 13 and 19 metres.

All along the footwall of the deposit, special attention was given to the Gs geological unit immediately south of Zn-Ag bearing mineralization. In this case, individual bench wall was moved further south during the design to ensure that part of the final pit wall won't lie in this unstable deformation zone.

Thickness of overburden is between 2 to 6 metres around the existing Barvue pit, thus over the Barvue pit expansion, and about 10 metres over Abcourt pits. The overburden slope (2.5H:1V) will be stabilized by a layer of rock.

Consideration was given at all times in the design process to issues regarding existing topography, haulage roads and mine rock storage areas and to provide adequate operational space for production equipment. For each pit, several designs were performed to maximize ore recovery and to minimize the waste to ore ratio. The final pits outlines are shown in Figure 4.1.1. A typical cross-section of the Barvue open pit is illustrated in Figure 4.1.2.

Part of the ore planned to come from the East Abcourt pit will be subject to royalties because it is located east of the limit separating the Barvue from the Abcourt part of the property.

4.2 Underground mining

4.2.1 Review of previous underground work

In 1951, after a surface exploration program consisting of 36 diamond drill holes, a decision was taken to proceed with an underground exploration program on the Abcourt property. A three-compartment shaft was sunk to a depth of 170 m and 225 m of drifts were excavated on the 91 m and 137 m levels. In November 1952, all work was suspended due to weakening zinc prices.

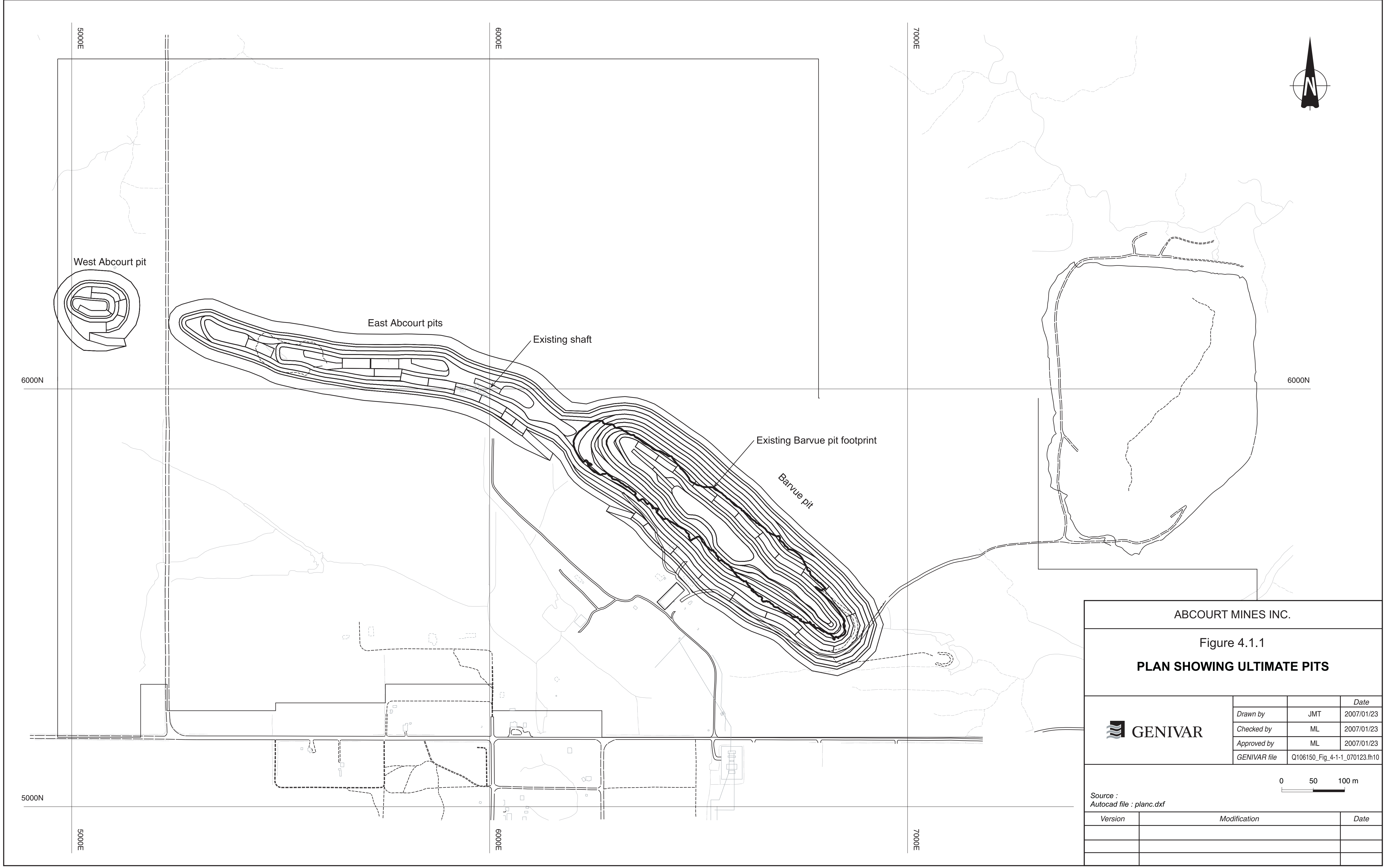

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Figure 4.1.1

PLAN SHOWING ULTIMATE PITS

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Figure 4.1.2

TYPICAL CROSS-SECTION OF THE BARVUE OPEN PIT

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In 1957, after the extraction of 5M short tons of ore by open pit, Barvue Mines Limited started preparation work for underground mining on the Barvue property by driving a decline between the 76 m and the 152 m levels and excavating sub-levels at 15 m intervals. Due to falling zinc prices, the operation was suspended later that year.

In 1980, the control of Abcourt was taken over by "Fonds miniers Hinse". In 1983, Abcourt bought the Barvue property, unflooded it and rehabilitated it. Additional drilling was done and mining reserves were confirmed. With an investment of 20 M$, the Barvue mine was then brought back into production, by underground mining. The ore was hauled by trucks to Matagami to be processed in the Noranda mill. From 1985 to 1990, approximately 700 000 short tons of ore were mined with an average grade of 3.85 oz/st Ag and 5.04 % Zn. Also, 204 oz of gold were recovered from the silver concentrate. The mine was closed in 1990 due to low metal prices. Since then, the surface buildings and facilities have been kept in good shape and the mining equipment has been rebuilt.

During the latest production period (1985 to 1990), the Barvue mine was connected to the Abcourt shaft with an internal ramp. The shaft was used as an escape way and for ventilation, air, water and electrical services. The Abcourt side of the property was also partly developed with drifts, ramps and sub-levels.

4.2.2 Avoca cut-and-fill method

For the remaining measured resources (not exploited by open pit) and part of the indicated resources, the Abcourt-Barvue ore is found generally in the hanging wall of a marker tuff where ground conditions are good. For this zone, the Avoca cut-and-fill method will be used for the 6th to 10th year of production.

From three main declines, cross-cuts will be driven into the ore zone and lateral development will proceed on both sides to the limits of the lenses. Production will start at the bottom of the indicated resources with sub-levels developed at 15 to 20-metre vertical intervals. All mining will be trackless.

Jumbos will be used for decline, access cross-cut and sub-level development. In the Avoca stopes, the ore will be removed with remote controlled 5 or 2-cubic-yard scoops. The benches will be drilled with long hole machines. The main declines will be driven with a minus 15 to 17 % slope.

Blasting operations will use ANFO explosives with a gel cartridge to facilitate the detonation both for development and production purposes. Trucks with 30-t loads will tram the ore to the surface and will come back with a load of waste rock to be used as backfill if justified. Fresh air for underground mine ventilation purposes will be injected through ventilation raises and the existing shaft after its rehabilitation.

The main declines, haulage drifts and access cross-cuts will be 5.5-m wide x 4.5-m high. The sub-drifts in ore will be 4.5-m high over the full width of the ore zone. Each stope will have an access cross-cut for each sub-level. The 1.5-m x 1.5-m slot raises will be drilled with a long hole machine. The Avoca method produces 100 % recovery of the ore.

The stopes will be backfilled with waste and paste or cemented backfill. The backfill will consist of mill tailings and/or pyrite concentrate.

Figure 4.2.1 is a schematic view of the Avoca mining method with summary description of the steps of a production cycle. Concurrently of production cycle, development crews will be driving additional sub-levels above the top sill. The longitudinal view of the underground mine and the plan view of a typical sub-level development are illustrated in figures 4.2.2 and 4.2.3 respectively.

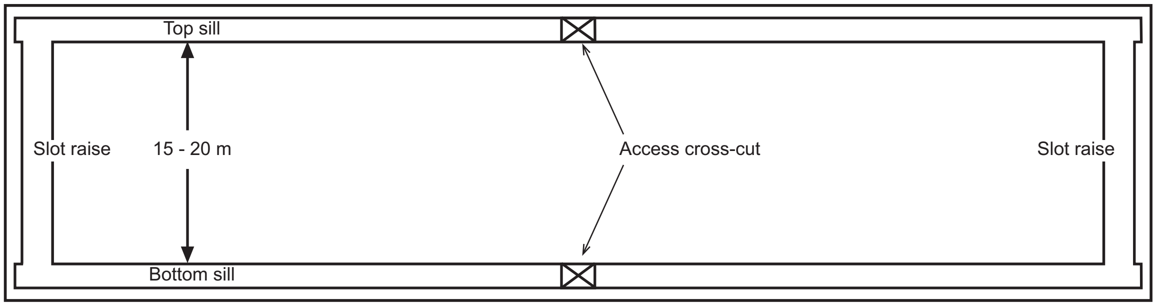
4.2.3 Net smelter return values

As estimated in section 3.7.6 and based on prices of 1.00 US$ per pound for zinc and 10.00 US$ per ounce for silver and a rate of exchange US$/C$ of 0.85, the unit NSR value for the underground mine is 12.83 US$/t of ore per one percent of zinc equivalent.

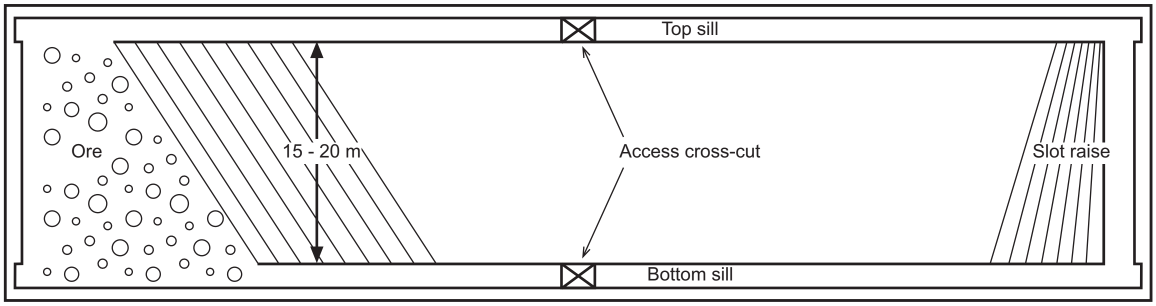
For underground mine design purposes, the minimum NSR value required to make a mineralized block (or polygon) economically mineable is 41.06 US$/t (cut-off grade of 3.20 % Zn EQ).

4.2.4 Model preparation

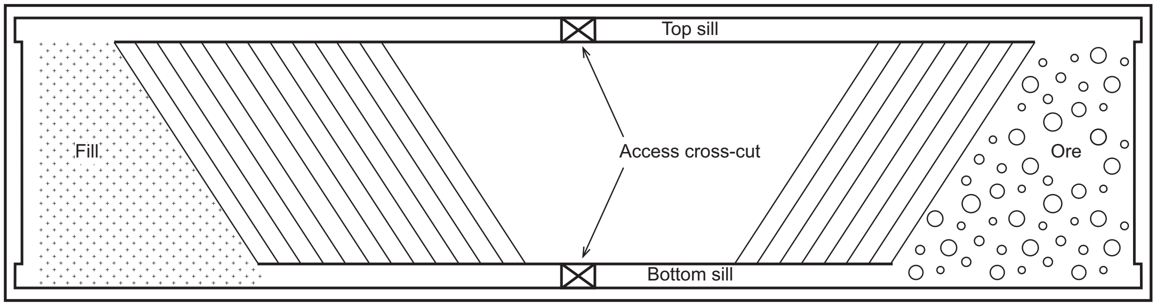
The mineralized model used herein has been described in section 3.7. Polygons were traced on cross-sections for measured (as for open pit) and indicated mineral resources. These polygons were extended mid-distance apart cross-sections and drawn on a longitudinal view with ore tonnages and grades established during the MRB's resource evaluation process. Designed Avoca stopes and required development will both be based on that model.



Step 1 - From access cross-cuts, drive east and west bottom and top sills. Open slot raises.



Step 2 - Drill and blast the west bench over a distance of 8 to 15 meters along strike depending on ground conditions. Long holes may be drilled vertically or inclined. Muck out with remote-controlled scoops.



Step 3 - Blast east end of bench and muck out while filling the west end. Drilling of long holes has to be fitted in the schedule.

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Figure 4.2.1

SCHEMATIC VIEW OF THE AVOCA MINING METHOD

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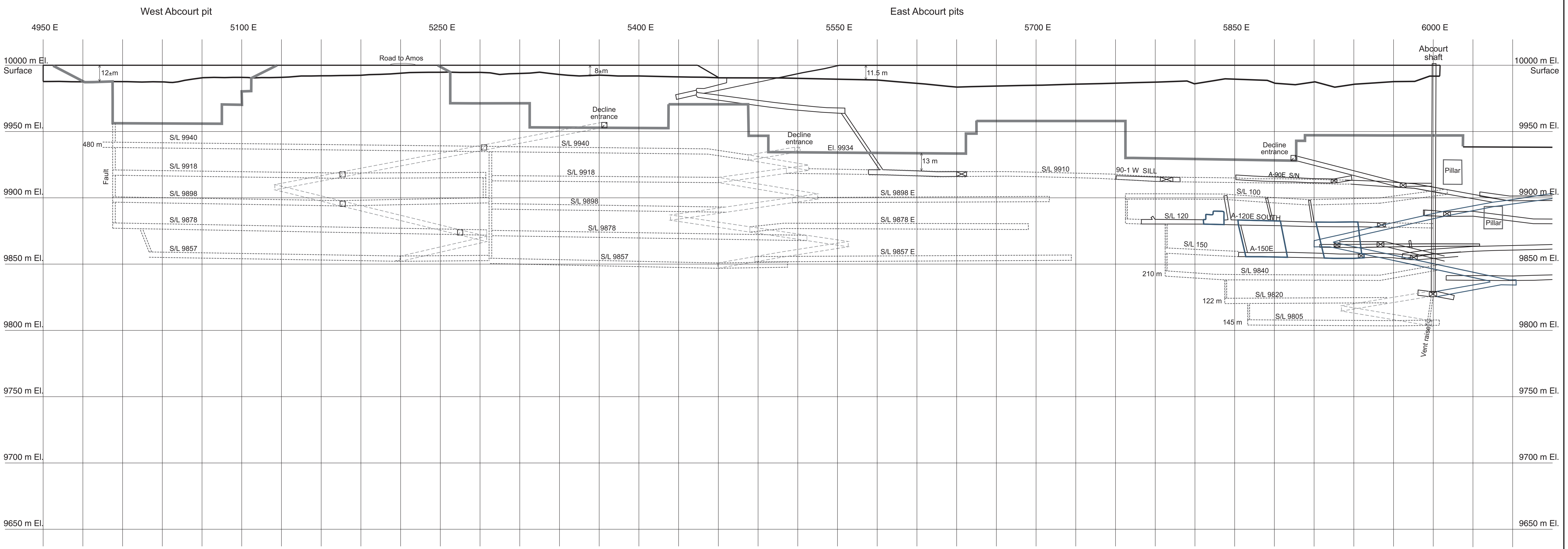

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Figure 4.2.2

LONGITUDINAL VIEW OF THE UNDERGROUND MINE

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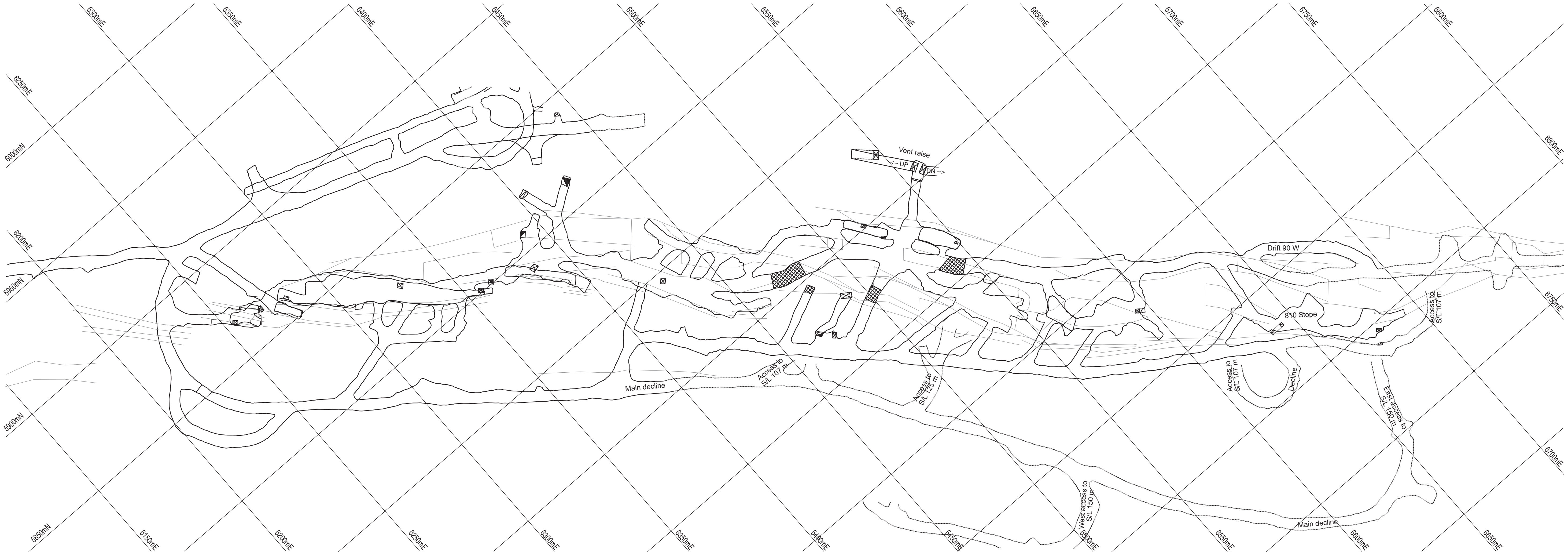

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Figure 4.2.3

PLAN VIEW OF A TYPICAL SUB-LEVEL DEVELOPMENT

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4.2.5 Underground mining proposal for the Abcourt-Barvue ore body

For mining purposes and resource calculations, the ore body was divided into two sectors, that is the Barvue and the Abcourt, the existing Abcourt shaft being the dividing point.

The underground production will come from three Avoca cut-and-fill stopes which will be named «west stope», «central stope» and «east stope», and from mine development. The stopes extend over a distance of about 1 125 metres from 5010E on the Abcourt side to 135E on the Barvue side. All the ore tonnage which is planned to be mined into the aforementioned limits is located west of the limit separating Barvue from Abcourt and is therefore not subject to royalties.

The west and central stopes will start at a depth of about 150 metres and finish at a depth of about 55 metres in pit bottoms. The east stope will start at a depth of about 200 metres and will be completed at an elevation of about 65 metres, also in pit bottoms. Each stope will have two faces in ore, so it will be possible to alternate the drilling, blasting and mucking of ore cycle at one end of the stope with the filling cycle at the other end. The objective is to have a steady flow of muck from each stope.

Development will produce about 5 000 tonnes of ore per month. Once fully developed, each stope will produce about 7 350 tonnes per month for a maximum of 22 000 tonnes of ore per month. In total, the underground mine will have a capacity to produce 27 000 tonnes of ore per month.

4.3 Mineral reserves

4.3.1 Mining dilution

The conversion of mineral resources into reserves takes into account dilution and losses occurring during mining operations. Different dilution factors were used for the Abcourt-Barvue project depending on mining method and ore body configuration.

A dilution factor of 5 % in volume was applied to ore tonnage potentially mined in the Barvue open pit given the large width of the ore body. Diluting material is composed of 50 % marginal ore grading 16.93 g/t Ag and 1.64 % Zn and 50 % waste material grading 9 g/t Ag and 0 % Zn (to be conservative with the zinc production grade).

For the Abcourt ore tonnage potentially mined by open pits, as the width of the mineralization is lower than in the Barvue pit, a 10 % dilution factor in volume was used with diluting grades of 20.15 g/t Ag and 1.38 % Zn for the 50 % portion made of marginal ore and 13 g/t Ag and 0 % Zn for the 50 % waste portion (again to be conservative with the zinc production grade).

In both cases, diluting grades of marginal ore are the average grades of sub-marginal and marginal ore of mineral resources under 2.4 % Zn equivalent for Abcourt, and under 2.55 % Zn equivalent for Barvue.

For waste material, the above values for silver were estimated from drill hole intersections found on both sides of the ore in contiguous rock.

Finally, a dilution factor of 20 % for development and 10 % in stopes was applied on ore tonnage potentially mined by the underground Avoca mining method for which a 3.2 % Zn equivalent cut-off grade was applied. The diluting grades are 31.1 g/t Ag and 0 % Zn. The silver grade was estimated from drill hole intersections found on both sides of the ore in contiguous rock.

4.3.2 Statement of mineral reserves

The mineral reserves were determined by applying the aforementioned cut-off grades for both the open pit and underground mine designs which are 2.40, 2.55 and 3.20 % Zn equivalent for Abcourt pits, Barvue pit and underground mine respectively and depends on estimated Zn and Ag grades from resources evaluation, expected metal recoveries, rates of concentration and smelting, transportation and other fees.

The Zn equivalent grade must exceed these cut-off grades in order to be profitably processed. Material slightly below cut-off grades will be stored on a marginal (low grade) stockpile for processing later, at the end of the mine life or sooner if a mill expansion is eventually justified.

All the material designated as ore in the open pit designs was derived from measured resources while ore in the underground stopes was derived from both measured and indicated resources. **The economic analysis in chapter 13 shows that measured and indicated resources, with the addition of the estimated dilution for different parts of the ore body, should now be considered as proven and probable reserves.**

The mineral reserves estimate including dilution is presented in Table 4.3.1

Table 4.3.1 Mineral reserves statement.

Mining method	Classification	Tonnage (t)	Grade Ag (g/t)	 Zn (%)	 Zn EQ (%)
Open pit	Proven Mineral Reserves	5 338 731	44.79	3.15	4.09
Underground	Proven Mineral Reserves	1 169 662	105.19	2.87	5.06
	Probable Mineral Reserves	315 139	101.61	3.23	5.35
	Total Underground	1 484 801	104.43	2.95	5.12
Open pit and Underground	Proven Mineral Reserves	6 508 393	55.64	3.10	4.26
	Probable Mineral Reserves	315 139	101.61	3.23	5.35
	Total	6 823 532	57.76	3.11	4.31
Open pit	Proven Marginal Ore	1 151 502	17.65	1.58	1.95

Note: Abcourt pits' cut-off : 2.4 % Zn Eq; Barvue pit's cut-off : 2.55 % Zn Eq; underground stopes' cut-off : 3.2 % Zn Eq.

4.4 Mine production schedule

The production schedule was established on a 10-year basis (6 446 000 t) because it was estimated that subsequent years of production have minor influence on the economics of the project. It is noteworthy that there are slightly more than 3 additional years of production at the same milling rate with the remaining underground proven and probable reserves, the underground measured and indicated resources under the Barvue pit and in the Gs zone which can potentially become mineral reserves and the stockpiled proven marginal ore. After year 10, this represents a tonnage of about 2 Mt grading 47.88 g/t Ag and 2.43 % Zn for a zinc-equivalent grade of 3.43 %. This could be extended further with inferred resources after additional exploration and development.

At first, production will come from open pits. Starting in year 6, open pit production will be supplemented by underground stope production. In total, 83 % of production will come from open pits and 17 % will come from underground stopes. Haulage trucks from both the open pit and underground mine will dump directly to the ore mixing area near the crusher.

The average waste to ore ratio for the open pits is 6.42:1 considering the marginal ore as waste. The mining of 3.87 Mt of waste (and marginal ore) and 128 000 t of ore is scheduled as preproduction thus lowering the previous average waste to ore ratio to 5.7:1 for the 10-year production period. The preproduction ore will be stockpiled apart on the marginal ore stockpile area to be reclaimed during the first two years of production.

The waste to ore ratio is falling from 8.85:1 during years 1 to 4 to 3.23:1 during years 5 to 10 when open pit operations will be recovering ore from the deepest benches. In the last years, as the waste to ore ratio is decreasing in the pits and as underground production is phased in, operating shifts in the open pit will be reduced in order to maintain a high productivity rate for the mining fleet. Table 4.4.1 shows the annual production tonnage and grades mined over the 10-year life span of the project. It is worth noting that with high metal prices, the open pits could go deeper.

4.4.1 Open pit production schedule

At full milling rate (650 000 t/a), the open pit ore reserves would be exhausted in about 8 years. For the 10-year project life-span under study, two additional years of production were thus needed. The monthly estimated 27 000-t of ore production rate of the underground mine from both development and production activities was not enough to achieve alone the design feed rate of the mill. Therefore, a combination of open pit and underground mine productions is planned.

From year 6 to 10, the ore production rate from open pits is gradually decreased to 326 000 tpa while, at the same time, the production from the underground mine is gradually increased and reaches 324 000 tpa, at year 8.

The learning curve for the open pit mining operations was fixed at 50 % - 70 % of full designed mining rate during the first two months of pre-stripping at year –1. At year 1, it was estimated that 596 000 t will be milled because of the production ramp-up of the concentrator (2 first months at 50 % of the designed milling rate).

It was determined that most of the 43-101 open pit resources of Abcourt-Barvue deposit with 2.4 and 2.55 % zinc-equivalent cut-off grades for Abcourt and Barvue respectively will be mined out. Therefore, 5 338 731 tonnes of diluted ore and the related waste rock tonnage will be mined during the 10-year production schedule.

4.4.2 Underground production schedule

The 43-101 underground resources of Abcourt-Barvue with a 3.2 % zinc-equivalent cut-off grade are as follows:

	Tonnes	Ag g/t	Zn %
Abcourt			
measured	1 047 878	114.31	3.12
indicated	232 214	111.53	3.43
	1 280 092	113.81	3.18
Barvue			
measured	341 856	61.63	4.28
indicated	270 892	87.05	3.44
	612 748	72.87	3.91
Total	**1 892 840**	**100.56**	**3.42**

Gold: 0,138 g/t = 0.004 oz/t

Table 4.4.1 Overall mine production schedule.

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	**Total**
Open Pit Production												
Waste mined [1] ('000 t)	3 871.5	5 350.0	5 350.0	5 350.0	5 350.0	3 350.0	2 664.0	1 556.9	834.2	354.5	267.1	**34 298.2**
Ore mined ('000 t)	128.5	550.0	567.5	650.0	650.0	650.0	630.0	534.7	326.0	326.0	326.0	**5 338.7**
Rock mined ('000 t)	4 000.0	5 900.0	5 917.5	6 000.0	6 000.0	4 000.0	3 294.0	2 091.6	1 160.2	680.5	593.1	**39 636.9**
To/From ore stockpile ('000 t)	-128.5	46.0	82.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
Ore stockpile inventory ('000 t)	128.5	82.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**0.0**
Waste to ore ratio	30.13	9.73	9.43	8.23	8.23	5.15	4.23	2.91	2.56	1.09	0.82	**6.42**
Underground production												
Ore mined ('000 t)	0.0	0.0	0.0	0.0	0.0	0.0	20.0	115.3	324.0	324.0	324.0	**1 107.3**
Ore milled ('000 t)	0.0	596.0	650.0	650.0	650.0	650.0	650.0	650.0	650.0	650.0	650.0	**6 446.0**
Zinc (% Zn)	0.00	2.73	2.76	2.87	3.37	3.43	3.37	3.26	3.10	3.06	3.12	**3.11**
Silver (g Ag/t)	0.00	62.42	58.10	52.42	37.93	34.34	38.95	57.65	69.17	67.46	71.75	**54.96**
Zinc equivalent (% Zn Eq)	0.00	4.03	3.97	3.97	4.16	4.15	4.18	4.46	4.55	4.47	4.62	**4.26**

1 Including waste rock (33.15 Mt) and marginal ore (1.15 Mt).

This total includes 453 166 tonnes grading 71.23 g/t silver and 3.52 % zinc located from sections 315E to 1185E, under the Barvue pit. These resources will only become available after the completion of mining in the pit in year 10.

Also, there are about 109 582 tonnes of ore grading 71.38 g/t Ag and 4.74 % zinc located in the Gs on sections 5100E to 5280E on the Abcourt side. The Gs is a weak rock and stopes will need good support. Because of the expected difficulties in mining this ore, it is not included in our current production schedule.

In addition, there are 1 505 687 tonnes grading 120.53 g/t silver and 2.98 % zinc of inferred ore not included in the above totals because insufficiently developed to be included in our mining plans.

Here is the schedule of the ore produced by development and by stoping:

Year	Development Tonnes of ore	Stope production Tonnes of ore
6	20 000	---
7	60 000	55 269
8	60 000	264 000
9	60 000	264 000
10	60 000	264 000
Total	**260 000**	**847 269**

At year 7, three months of ore production from stopes are planned. A learning curve for Avoca mining method was supposed for the monthly mining rate estimated at 22 000 tonnes of ore. Therefore, the first month of stopes production was fixed at 50 % of full mining production rate.

For each stoping block, the available tonnage and grades of ore to be extracted either by development or by stoping will be as in Table 4.4.2.

Table 4.4.2 Underground mine schedule.

	Total resources			Development			Available stope tonnage		
	Quantity (t)	Ag (g/t)	Zn (%)	Quantity (t)	Ag (g/t)	Zn (%)	Quantity (t)	Ag (g/t)	Zn (%)
West stope									
(5010E-5280E)	330 575	146,63	2,58	52 000	146,63	2,58	278 575	146,63	2,58
Central stope									
(5295E-5760E	550 428	118,29	2,80	83 000	118,29	2,80	467 428	118,29	2,80
East stope									
(5775E-6000E)	289 507	83,15	4,12	60 000	83,15	4,12	229 507	83,15	4,12
and (15W-135E)	159 582	78,88	4,89	22 000	78,88	4,89	137 582	78,88	4,89
Sub-Total	**1 330 092**	**112,96**	**3,28**	**217 000**	**111,37**	**3,32**	**1 113 092**	**113,27**	**3,28**
Dilution of 20 % in development				43 400	31,10	-----	-----	-----	-----
Dilution of 10 % in stopes				-----	-----	-----	111 309	31,10	-----
Total				**260 400**	**97,99**	**2,77**	**1 224 401**	**105,80**	**2,98**
Needed for production scheduled				260 000	97,99	2,77	847 269	105,80	2,98
Remaining at the end				400	97,99	2,77	377 132	105,80	2,98

4.5 Mining operations

4.5.1 Open pit

Open pit operations will be carried out mainly on three 8-hour shifts, seven days per week and 52 working weeks per year. For the shift change, mine employees will be transported to the drills and loading equipment while truck operators will be relieved at the service building. All equipment will be diesel powered.

4.5.1.1 Drilling and blasting

Drilling and blasting will be required on all ore and waste rock. The waste and the ore will be blasted as separate operations to provide better ore control and limit dilution. One conventional in-the-hole hammer rig will be used to drill 6 or 8-in (152 or 203 mm) diameter blast holes on a 8 m by 8 m pattern approximately or smaller for 6-in holes in waste rock (full bench height). In the ore, one crawler drill and compressor will be used to drill 3-in (76 mm) diameter blast holes on a 1.5 to 2 m by 2 m pattern (half bench height). This drill will also be used both for pre-shearing and secondary blasting.

An explosive supplier will deliver in the open pit mine the emulsion-ANFO needed for production blast holes. The product will have a high velocity of detonation and a good water resistance. The mixture will be prepared at the on-site bulk explosives plant operated by the supplier. To avoid loosing of bulk explosives in cracks or underground openings, a plastic pipe or tube will be inserted in the hole prior to loading. Cartridge gel explosives will be used for pre-shearing and secondary blasting. The design powder factors are 0.25 and 0.27 kg of explosives per tonne of rock for ore and waste respectively, while it is 0.6 kg of explosives per metre drilled for pre-shearing. After mining commences, fragmentation will be evaluated and the drill and blast parameters may be further refined to optimize results.

Two storage magazines will be needed on site, one for cartridge explosives and one for explosive accessories, and both will be installed and rented to Abcourt by the supplier.

4.5.1.2 Load and haul

The waste and the ore will be loaded as separate operations. The main loading equipment will be a 6.8-m^3 front-end loader in waste and a 3.5-m^3 hydraulic backhoe shovel in ore. As back-up to the loader, a second 6.8-m^3 wheel loader (same model as the other) will be available for short time assignment; its main duty will be at surface between the ore mixing area and the crusher. A fleet of up to five 62-t capacity trucks will haul ore to the ore mixing area near the crusher and other mine rock to either waste rock dumps or the marginal ore stockpile.

At the ore mixing area, the mine trucks will dump the ore in individual piles which will be re-handled twice by the assigned front-end loader. First, the loader will placed the ore from the individual piles in a mixing stockpile and will keep occasional boulders apart for later rock-breaking. Second, the stockpile will be reclaimed perpendicularly to the pile-up direction and the ore will be discharged in the crusher. This will ensure a good ore mix at the mill feed.

4.5.1.3 Support equipment

The production equipment will be supported by two bulldozers (240 kW each), one grader (150 kW), one hydraulic shovel (2-m^3 equivalent) mounted with either a rock-breaker or a scaling bar (pit wall securing) and one water truck (adapted from a 30-t articulated truck already owned by Abcourt) for dust suppression.

4.5.1.4 Grade control

Grade control will be performed by a geological technician. Cuttings of blastholes in or near the ore will be sampled and analyzed on site in the mill laboratory. The results will be used to determine which portion is above the cut-off grades established according to prevailing metals prices and should be considered as ore. Blasting limits in ore will be set accordingly.

4.5.2 Underground mine

Underground operations will be carried out mainly on two 8-hour shifts, five days per week and 52 working weeks per year. For the shift changes, mine employees will be transported to the main decline entrances with a minibus. All equipment for underground production will be diesel powered.

4.5.2.1 Drilling and blasting

Drilling and blasting will be required for development and production work. The waste, in minor quantities, will be sourced from development in waste rock (declines, access cross-cuts, etc.) and the ore will be produced from development (cross-cuts and sub-levels) and production stopes. Three jumbos and three long hole machines will be used to drill 3-in (76 mm) diameter blast holes on a 1.5 m by 1.5 to 2 m pattern in Avoca stopes (full bench height). A specific drilling pattern will be optimized for underground development of drifts to maximize stability of excavations.

Cartridge gel explosives and ANFO will be used for both development and production blasting operations. The design powder factors are respectively 1.00 and 0.50 kg of explosives per tonne of rock.

The two storage magazines used for the open pits will be used as well for the underground operations.

4.5.2.2 Load and haul

The waste and the ore will be mucked with three 5 or 2-cubic-yards scoops (one per stope), hauled to loading stations where 30-t underground trucks will be loaded for rock transportation to either the ore mixing area or waste rock disposal areas at surface. During underground production, trucks will come back with a load of waste rock to be used as backfill if needed.

At the ore mixing area, the mine trucks will dump the ore in individual piles which will be re-handled by the assigned front-end loader, in the same manner as the open pit ore (section 4.5.1.2).

4.5.2.3 Grade control

Grade control will be performed by a geological technician. Each sub-level face will be sampled and assayed. Based on the assay results and a visual inspection, the technician will mark the ore limits in the face. He will also use the available diamond drilling information to draw the ore limits in benches. Muck samples will be taken by the muckers.

4.6 Mine support facilities

4.6.1 Logistical support

Pickup trucks will be used by engineering, geology and management personnel for travelling on the site. A land-rover and a minibus, both rented, will be available to travel underground and to transport employees to underground mine entrance respectively. No ambulance is needed as the distance to the nearest hospital is covered by existing ambulance services.

4.6.2 Dewatering

The existing Barvue pit will be dewatered during preproduction phase. Five submersible pumps with all accessories (hoses, pipes, valves, holding valves, couplings, pump cribs) and a telescopic crane to handle the pumps are already owned by Abcourt.

Dewatering of pits and underground mine during production phase will be performed with existing stationary and submersible pumping equipment.

A water treatment plant will be installed on the site, at the southeast limit of the tailings pond, in which lime, a coagulant and a flocculent will be added to precipitate fine particles and zinc in solution.

During the pre-production and production periods, the mine water will be directed to the water treatment plant and discharged into a drainage ditch feeding a settling pond at the northwest limit of the tailings pond, or used in the mill as fresh water.

4.6.3 Mine dry

Changing and washing facilities for both open pit and underground workers are already located in the existing service building on site. Employee parking space is also available on the site.

5. METALLURGY

5.1 Metallurgical history

5.1.1 Milling operations

From 1952 to 1957, Barvue Mines Ltd mined by open pit some 5 500 000 short tons of ore grading on the average 2.98 % Zn and 1.13 oz Ag/st. The ore was processed in the 4 500 st/day capacity concentrator located on the property.

The process consisted in grinding to 68 % -200 mesh. The ground product was pumped to a primary rougher and scavenger flotation circuit. The rougher concentrate was cleaned by flotation to produce a final concentrate while the scavenger concentrate was reground, in a separate circuit to 95 % -200 mesh and returned to the rougher stage. The metallurgical balance of the Barvue Mines concentrator was estimated as follows.

Period		Heads		Concentrate			
From January	To September	Zn	Ag	Zn	Ag	% Recovery	
		%	oz/t	%	oz/t	Zn	Ag
1956	1956	2.93	1.17	59.63	18.70	90.61	73.76
1957	1957	3.33	1.30	59.27	19.36	92.02	81.37

The concentrator also included a zinc concentrate deleading circuit, which was operated intermittently according to the ore lead content.

The above information is contained in documents prepared by A. Stemerowitz, D. A. Livingston, P.Eng., and A. A. Almstrom who were mill superintendents at Barvue (see References).

5.1.2 Custom milling

From 1985 to 1990, Abcourt Mines Inc. extracted 700 000 short tons of ore from an underground operation. This ore was hauled to the Noranda Mines Ltd mill at Matagami and treated in a separate circuit assigned to Abcourt.

The circuit assigned to Abcourt consisted in grinding to a target grind of 80–90 % minus 200 mesh, followed by a silver flotation circuit, the tailings of which were further processed in a Zn-Ag flotation circuit consisting of one rougher stage followed by three cleaning stages. The metallurgical balance of the Abcourt circuit was calculated as follows for the period from May 1985 to June 1990 during which 647 465 st were milled.

	Weight	Assays		% Recovery	
	%	% Zn	oz Ag/t	Zn	Ag
Ag Concentrate	0.67	30.320	264.060	4.04	45.98
Zn-Ag Concentrate	7.75	56.300	19.76	87.09	39.99
Tailings	91.58	0.485	0.587	8.87	14.03
Heads Assays	100.00	5.010	3.830	100.00	100.00

This information is contained in a Note addressed by Mr. Denis Hamel to Mr. Renaud Hinse, president of Abcourt Mines Ltd with copy to Mr. Michel Garon (see References).

BUMIGEME considers this information trustworthy but points out that a higher metal recovery and grade of Zn concentrate would most likely have been obtained through finer grinding and the addition of a scavenger stage in the primary Zn flotation circuit.

5.1.3 Test work and mineralogical studies

Following is a summary of the metallurgical test work and mineralogical studies carried out on samples from the Abcourt and Barvue parts of the Abcourt property. Test work and studies were carried out by qualified persons and BUMIGEME considers that the information contained in their reports is trustworthy.

- July 1, 1953, Denver Equipment Company, Ore Testing Division.

  A 285 lb sample, submitted by D. M. Giachino, Mine Manager of Pershcourt Goldfield Limited, was made up of ore lumps (< 4″) assaying 1.17 oz/t Ag, 0.32 % Pb and 4.95 % Zn. Four flotation tests were made.

  The authors, Clarence Thom, Director of Ore Testing Division and Henry J. Gesler, Manager of Ore Testing Division, concluded "that our preliminary examination of this ore indicated that flotation with fine grinding is the proper method of concentration of the silver, lead and zinc values. The tests include selective flotation to recover separate lead-silver concentrates, zinc concentrates and followed by gravity table concentration to recover pyrite. Selective flotation to recover a combined silver-lead-zinc concentrate and gravity table treatment of this concentrate to recover the silver-lead value was also conducted."

- July 15, 1968, Report of Mineralogical Study by Dr. Guy Perreault.

  A microscopic (mineralgraphic and electronic) and microprobe study was conducted on 6 samples from the Consolidated Pershcourt (Abcourt Section) property. It was concluded that the very fine silver mineralization (2 to 5 microns) occurs mainly as pyrargyrite and freibergite. Native silver was exceptional and tetrahedrite was not found. Sphalerite is coarse and the silver minerals are found in veinlets and inclusions in the sphalerite. Chalcopyrite was also observed associated to the pyrargyrite mineralization.

- November 24, 1973, Mineralogical report for Laszlo Dudas, Mineralogical Consultant, Tucson, Arizona, USA.

  Five samples from holes A-2, A-3, A-5 and A-7 of the Abcourt deposit were examined.

  Mica (mainly muscovite and sericite) is the major gangue mineral. Tetrahedrite was observed in two samples while chalcopyrite blebs occurring as inclusions in sphalerite was observed in all samples. The samples weighting between 60 and 86 grams, Mr. Dudes concluded that the samples "were inadequate for thorough mineralogical investigation".

- January 10, 1974, Lakefield Research of Canada Limited, Progress Report No. 1.

  The objectives of this study were to determine the flotation response of the ore with respect to the selective separation of silver and zinc minerals, as well as the production of a marketable zinc concentrate. Cyanidation tests were to be performed on principle products to determine the solubility of the silver and of silver bearing minerals.

  The sample was prepared from crushed core rejects from holes A-1 to A-5 and A-7. Compositing was based on 200 grams per foot of core length. Hole A-6 was omitted because only pulverized samples were available. Average grade 2.05 % Zn, 0.06 % Pb, 0.02 % Cu and 3.2 oz/t Ag. Six (6) tests were done. They showed that it was possible to produce an Ag-Pb concentrate and a separate zinc concentrate or a combined Ag-Zn concentrate. Cyanidation tests were done on the Ag-Pb concentrate, on the zinc concentrate and on the pyrite concentrate. None were done on the ore.

As the samples were not representative of the ore body and as the tests were essentially characterization tests, the results are not incorporated in our study.

- November 12, 1975, Matagami Lake Mines Limited, Mill Laboratory, by K. Stowe, Laboratory Metallurgist.

  Approximately 90 pounds of uncrushed split core from hole RA-11 (200′ – 263′), assaying 1.33 oz/t Ag and 2.61 % zinc was used for the test. This hole was drilled on the Abcourt property. As this hole was not representative of the whole ore body, we concur with K. V. Konigsman, Mill Superintendent: "I have to repeat myself and emphasize that the results of our report should not be used as a basis for mill design."

- January 5, 1981, Matagami Mill Laboratory – Report by Michel Garon and Camil Prince.

  The test work was realized in a ± 40 kg composite sample which, according to Mr. Renaud Hinse, was prepared with core rejects from the Abcourt part of the Abcourt property. Some of the characteristics of the sample were summarized as follows:

Zn %	Ag g/t	S %	Cu %	Pb %	S.G.	Work Index kWh/st
2.04	163.2	1.86	0.04	0.070	2.84	11.6

Various flotation tests, with and without pre-flotation of a silver concentrate, were conducted at a grind of 93.8 % -200 mesh. The best results obtained were summarized as follows:

Test No.	Product	Weight %	Zn %	Ag g/t	% Recovery Zn	% Recovery Ag
A-M 21	Ag Conc.	4.00	4.72	3 390.4	10.1	80.9
A-M 21	Zn Conc.	3.10	52.56	362.2	86.5	6.7
A-M 18	Zn-Ag Conc.	3.47	51.36	3 623.3	89.9	76.5

Two cyanidation tests in bottle were also conducted on the composite sample, the cyanidation period having been set at 40 hours. Results were summarized as follows:

Test	Grind % -200 mesh	% Extraction Ag	% Extraction Zn
A-M 7	86.2	74.2	8.1
A-M 8	90.7	84.6	13.8

Test A-M 8 was conducted at 10 kg/t concentration of cyanide and has shown a better recovery of 84.6 % Ag.

- May 4, 1981, Matagami Concentrator Laboratory, report by Michel Garon and Susan Beaulieu.

  Out of ±500 kg of split core samples received in bags, the Matagami laboratory chose 235 kg from holes 80-1, 80-1A and 80-2 to prepare two composite samples of the Barvue part of the resources. Some characteristics of composite 1 were summarized as follows:

Zn %	Ag g/t	S %	Cu %	Pb %	S.G.	Work Index kWh/st
2.55	16.50	3.40	0.035	0.070	2.94	10.00

  The heads of composite 2 were calculated at 1.54 % Zn and 13.06 g Ag/t.

  Zn flotation tests, with and without pre-flotation, were conducted at various grinds. The best results were obtained from an Ag pre-flotation followed by Zn flotation test conducted on composite 1 ground to 73 % -200 mesh and which are summarized as follows:

Product	Weight %	Zn %	Ag g/t	% Recovery	
				Zn	Ag
Ag Conc.	2.4	7.19	358.1	6.7	47.0
Zn Conc.	3.6	60.64	175.6	85.6	35.0

  The report notes “that these samples were not representative of the Barvue ore type”.

- April 12, 1984, Matagami Concentrator Laboratory, report by Michel Garon and Yvan Lemieux.

  The laboratory prepared a composite sample of the Abcourt part of the resources through mixing half of the content of 10 bags of drill core rejects identified 83-01 to 83-10.

  The characteristics of the composite were summarized as follows:

Zn %	Ag g/t	S %	Cu %	Pb %	S.G.	Work Index kWh/st
3.5	149.83	2.87	0.037	0.082	3.02	11.3

A series of grinding tests followed by a silver pre-flotation and Zn flotation test was carried out. It was determined that a 93 % -200 mesh grind was necessary to achieve good recoveries. Typical results of a cycle-test were summarized as follows (test AB-23-49):

Product	Weight	Zn	Ag	% Recovery	
	%	%	g/t	Zn	Ag
Ag Conc.	2.6	9.85	4 389.0	7.3	78.1
Zn Conc.	5.7	53.91	239.5	85.8	9.1

- May 9, 1984, Matagami Concentrator Laboratory.

The covering pages, the table of content, the samples origin and the characteristics of the Abcourt sample of the resources are believed to be missing in the copy of this report sent to bumigeme. Some of the main characteristics of the samples were identified as follows:

Composite	Zn %	Ag g/t	S %	Cu %	Pb %	S.G.	Work Index kWh/st
60 % Barvue, 40 % Abcourt	5.65	148.80	4.71	0.04	0.126	2.91	11.2
Abcourt*	3.66	140.89	---	---	---	---	---
Barvue	4.28	147.08	5.88	0.031	0.133	2.88	11.1

* Calculated from results f cycle test AB-26-42.

Results of test work conducted on these composite samples are discussed in Section 4 of the present report.

- April 2001, URSTM, Jean Lelièvre, Eng., M.Sc.

Several tests were done on two different mixtures of Abcourt-Barvue ore to determine if new flotation reagents could improve the recovery of silver and zinc in separate or combined concentrates. Unfortunately, these tests did not show any improvement on the previous tests done by Noranda.

- December 2004, J. Cayouette and J. Bilodeau of the mineralogical laboratory of the Louvicourt Mine.

Representative samples of ore from the Abcourt, Barvue and Vendôme properties were prepared by Abcourt and delivered to the Louvicourt Mine laboratory to test various combinations of ore, i.e. 50 % Abcourt–50 % Barvue, 37.5 % Abcourt-37.5 % Barvue-25.0 % Vendôme, 100 % Vendôme.

The objective was to evaluate the metallurgical results that might be obtained with the treatment process used at the Louvicourt mill, to compare these results with those obtained at Matagami (1985-1990) and those obtained at Barvue (1957) and to float pyrite from the tailings. The desulphurization tests on the tailings and the acid generation potential of the desulphurized tailings were done by URSTM on the 50/50 Abcourt-Barvue and 100 % Vendôme samples.

- 2006, tests done by Mr. Edmond St-Jean, Eng. at LTM Laboratory.

  Several characterization tests were done by Mr. St-Jean to establish the formula that should be used to treat the Abcourt-Barvue ore by cyanidation to recover most of the silver followed by flotation to produce a zinc concentrate (with some silver) and a second flotation to produce non-acid producing tailings.

  Mr. St-Jean is a qualified person.

  We have controlled the tests conducted by Laboratoire LTM and we agree with the results, which have been used to support our study. The LTM test program can be consulted in Appendix 1.

5.2 Characteristics of the Abcourt-Barvue Ore

5.2.1 Geological environment

On the Abcourt-Barvue parts of the property, the mineralized horizon is hosted in a volcanoclastic sequence characterized by tuffs and agglomerates, generally highly carbonatized and sericitized. The regrouped Abcourt-Barvue Zn-Ag disseminated sulphide deposit is located close to a major shear zone consisting of a talc, sericitic tuffaceous horizon, also strongly carbonatized and sericitized.

5.2.2 Mineralization

In the deposit, zinc and silver are the main metallic elements of commercial interest. Zinc occurs as an iron-poor, honey brown and dark sphalerite which is found finely disseminated as well as bedded. Silver occurs as native Ag, argentite and prestite (Ag_3 As S_3) and is closely associated to sphalerite. Pyrite occurs finely disseminated and part of it is encompassed within the sphalerite. Minor amount of galena and chalcopyrite is also found in certain parts of the deposit.

The main gangue minerals comprise siderite, quartz, chlorite, sericite, illite and rhodocrosite.

5.2.3 Mill feed

Based on the geological interpretation and on the estimate of the "in place" mineral resources, carried out by MRB & Associates, GENIVAR developed the mining plan and the corresponding mining schedule shown in Table 5.2.1.

Table 5.2.1 Ore mining schedule for the Abcourt-Barvue mine.

Year	Annual				Cumulative Summary			
	Ore (t)	Grade			Ore	Grade		
		Ag (g/t)	Zn (%)	Zn Eq (%)	(t)	Ag (g/t)	Zn (%)	Zn Eq (%)
-1								
1	596 000	62.42	2.73	4.04	596 000	62.42	2.73	4.04
2	650 000	58.10	2.76	3.97	1 246 000	60.17	2.75	4.00
3	650 000	52.42	2.87	3.97	1 896 000	57.51	2.79	3.99
4	650 000	37.93	3.37	4.17	2 546 000	52.51	2.94	4.04
5	650 000	34.34	3.43	4.15	3 196 000	48.82	3.04	4.06
6	650 000	38.95	3.37	4.18	3 846 000	47.15	3.09	4.08
7	650 000	57.65	3.26	4.46	4 496 000	48.67	3.12	4.13
8	650 000	69.17	3.10	4.55	5 146 000	51.26	3.12	4.19
9	650 000	67.46	3.06	4.47	5 796 000	53.07	3.11	4.22
10	650 000	71.75	3.12	4.62	6 446 000	54.96	3.11	4.26
Total	**6 446 000**	**54.96**	**3.11**	**4.26**				

The above yearly run of mine ore tonnage and grade estimate includes dilution and corresponds to the yearly tonnage and grade of the Mill Feed. The average grade of 54.96 g Ag/t and 3.11 % Zn was retained as the grade of the milled ore in the process design criteria.

5.3 Process development

Milling operations and metallurgical test work carried out on the Abcourt-Barvue property ore since the early part of the 1950's were essentially related to the following two concentration processes:

- Direct flotation of a zinc-silver concentrate;
- Direct flotation of a silver concentrate followed by the flotation of a zinc-silver concentrate.

For the first time, in February 2006, LTM tested a third process which comprises the following operations in series:

- Direct cyanidation to recover the major part of the silver in a "silver brick";
- Processing by flotation the rejects of the cyanidation circuit to recover the zinc, as well as additional silver, in a commercial grade Zn-Ag concentrate;

- Further processing by flotation of the tailings from the Zn-Ag flotation circuit to:
  - reject a tailing assaying less than 0.3 % S to be stored into a conventional tailings pond;
  - float a pyrite concentrate to be stored in a "unit-cell" according to the directives of the Ministry of Environment.

The first 4 tests (AB-07 to AB-09) conducted in February-March 2006, were followed by test AB-10 in May and then tests AB-12, 13 and 14 in August 2006. Results of this process development test work were consigned into a Preliminary Report submitted by LTM in September 2006.

Considering the promising results obtained, Abcourt retained the services of BUMIGEME to evaluate these results as well as to recommend a work program which would have to be carried out to generate any additional data required by BUMIGEME to proceed with the final process and mill design as well as with its estimate of construction and operating costs of the concentrator. BUMIGEME comments, evaluation and recommendations were as follows.

5.3.1 Composite sample

A ±42 kg composite sample made up of material already crushed to ±8 mesh was received by LTM for the purpose of carrying out preliminary process development test work. According to Mr. Hinse, President of Abcourt, the composite sample was made up of material from the Abcourt section (35 %) and the Barvue section (65 %) of the ABCOURT property.

Heads of the composite identified by BUMIGEME as Composite A were calculated from the results of each one of the 7 cyanidation-flotation tests conducted on Composite A and presented in Table 5.3.1.

Table 5.3.1 Calculated heads of composite A.

No	Date	Heads Calculated	
		% Zn*	Ag g/t
07	27-02-06	2.84	104.15
08	25-03-06	2.99	90.29
09	27-03-06	2.96	95.39
10	14-05-06	2.74	98.41
12	25-08-06	2.86	122.81
13	29-08-06	2.85	125.56
14	29-08-06	2.73	106.25

* The calculated % Zn did not take into account the Zn lost (≈ 0.05 % Zn) in the cyanidation solutions which, at the time, were not assayed for Zn.

The important differences between the heads calculated for Ag suggest that Composite A might have been delivered in more than one bag and/or that the 3000g samples used for each test were divided without appropriate pre-mixing and/or appropriate regrinding.

5.3.2 Cyanidation tests

All cyanidation tests were conducted at a grind of about 95 % passing the 200 mesh screen opening which had previously been determined by LTM as the optimum grind. Results obtained are summarized in Table 5.3.2.

Table 5.3.2 Result of cyanidation tests conducted on composite A.

Test No.	pH at the End*	Ca O	NaCN		Calculated Heads	Recovery	
		Added g/t	Added kg/t	Consumed kg/t	Ag g/t	Ag g/t	%Ag Rec.
AB-07	12.77	1 683	10	2.8	104.15	72.59	69.70
AB-08	12.77	1 683	10	2.8	90.29	59.02	65.37
AB-09	n.d.	1 683	10	2.8	95.39	77.57	81.32
AB-10	n.d.	2 000	10	2.8	98.41	79.45	80.73
AB-12	11.63	2 000	5	3.6	122.81	98.01	79.81
AB-13	11.63	2 000	5	1.9	125.56	101.38	80.74
AB-14	11.63	2 000	5	2.1	106.25	84.60	79.62

* pH at the start is reported as having been set at 12.5 in all cases.

Considering that the first two tests were of exploratory nature, the results of the cyanidation tests have demonstrated that over 80 % of the Ag could be recovered by cyanidation from ores grading 95 g Ag/t or higher.

The important variations in the consumption rate of NaCN from tests AB-12 to 13 can hardly be explained. It should be noted however that these 3 tests were conducted nearly 8 months after the preparation of the composite A and that some alteration phenomena might have occurred.

5.3.3 Zinc-silver flotation

5.3.3.1 LTM test work

LTM carried out a zinc flotation test on each one of the rejects from the cyanidation tests. Each reject was filtered and thoroughly washed with at least an equal weight of fresh water. The “washed” filter cake was then repulped at 35 % solids with fresh water. The pulp was agitated and aerated in a flotation cell for 1 hour. Thereafter, a standard flotation test, comprising a rougher stage followed by two cleaning stages, was conducted on the aerated pulp, applying the conditions of operation shown in Table 5.3.3.

Table 5.3.3 Operation data on flotation tests conducted on aerated pulps.

Flotation Step	Solids %	pH	Time in Minutes	Flotation Reagents	
				Name	g/t
#1 Conditioner	35	11	2	CaO	560
				$CuSO_4$	570
Rougher #1			1	MIBC	70
				3477	50
Rougher #2			2	3477	
Rougher #3			3	3477	125
Rougher #4			4	3477	
1st Cleaner	35	12	2	CaO	1 000
2nd Cleaner			2		

The metallurgical balance of each test is shown in Table 5.3.4 together with average metallurgical balance of the six tests which is also illustrated graphically on Figure 5.3.1.

As a mean to estimate the influence of recirculation of the cleaner tails on the % Zn and % Ag recovery, as well as on the grade of the final concentrate, data corresponding to the 2nd cleaner concentrate (Case 1) and 1st cleaner concentrate (Case 2) are summarized in Table 5.3.5.

Table 5.3.4 Metallurgical balances of zinc-pyrite flotation tests conducted by Laboratoire LTM.

		Direct					Cumulative				
	Products	**% Weight**	**% Zn**	**Ag (g/t)**	**% Dist. Zn**	**% Dist. Ag**	**Weight %**	**% Zn**	**Ag (g/t)**	**% Dist. Zn**	**% Dist. Ag**
AB-8	Clean. conc. #2	3.14%	61.90	181.0	65.02%	6.29%	3.14%	61.90	181.00	65.02%	6.29%
	Clean. tail. #2	1.77%	50.34	381.0	29.91%	7.49%	4.91%	57.73	253.26	94.93%	13.78%
	Clean. tail. #1	2.95%	1.89	191.0	1.86%	6.23%	7.86%	36.79	229.91	96.79%	20.01%
	Pyrite conc. #1	1.05%	1.50	304.0	0.52%	3.52%	8.90%	32.64	238.61	97.32%	23.53%
	Pyrite tail	91.10%	0.09	11.0	2.68%	11.10%	100.00%	2.99	31.27	100.00%	34.63%
	Cyanidation			59.03		65.37%			90.29		100.00%
	Calculated heads	100%	2.99	90.29	100%	100%	100.00%				
AB-9	Clean. conc. #2	4.43%	62.64	265.00	93.86%	12.31%	4.43%	62.64	265.00	93.86%	12.31%
	Clean. tail. #2	0.37%	12.30	110.00	1.52%	0.42%	4.80%	58.79	253.16	95.38%	12.73%
	Clean. tail. #1	5.53%	1.54	23.00	2.88%	1.33%	10.33%	28.13	129.91	98.27%	14.07%
	Pyrite conc. #1	6.43%	0.16	42.50	0.35%	2.87%	16.76%	17.40	96.37	98.62%	16.93%
	Pyrite tail	83.24%	0.05	2.00	1.38%	1.75%	100.00%	2.96	17.82	100.00%	18.68%
	Cyanidation			77.57		81.32%			95.39		100.00%
	Calculated heads	100%	2.96	95.39	100%	100%	100.00%				
AB-10	Clean. conc. #2	4.07%	60.04	136.00	89.35%	5.63%	4.07%	60.04	136.00	89.35%	5.63%
	Clean. tail. #2	0.55%	23.17	239.40	4.69%	1.35%	4.63%	55.63	148.37	94.03%	6.98%
	Clean. tail. #1	4.68%	2.43	113.40	4.16%	5.40%	9.31%	28.87	130.78	98.19%	12.38%
	Pyrite conc. #1	0.17%	0.41	58.50	0.03%	0.10%	9.48%	28.36	129.50	98.22%	12.48%
	Pyrite conc. #2	4.38%	0.09	20.80	0.15%	0.93%	13.86%	19.43	95.15	98.36%	13.40%
	Pyrite tail	86.14%	0.05	6.70	1.64%	5.86%	100.00%	2.74	18.96	100.00%	19.27%
	Cyanidation			79.45		80.73%			98.41		100.00%
	Calculated heads	100%	2.74	98.41	100%	100%	100.00%				
AB-12	Clean. conc. #2	3.16%	63.02	221.00	69.64%	5.68%	3.16%	63.02	221.00	69.64%	5.68%
	Clean. tail. #2	0.79%	41.85	273.00	11.54%	1.75%	3.95%	58.79	231.38	81.18%	7.44%
	Clean. tail. #1	4.07%	11.83	158.00	16.85%	5.24%	8.02%	34.95	194.13	98.02%	12.67%
	Pyrite conc. #1	2.58%	0.36	87.00	0.33%	1.83%	10.60%	26.52	168.03	98.35%	14.50%
	Pyrite clean tail #1	5.25%	0.12	8.00	0.22%	0.34%	15.85%	17.77	114.98	98.57%	14.84%
	Pyrite clean tail #2	2.45%	0.26	68.00	0.22%	1.35%	18.30%	15.43	108.70	98.79%	16.20%
	Pyrite tail	81.70%	0.05	6.00	1.43%	3.99%	100.00%	2.86	24.79	100.00%	20.19%
	Cyanidation			98.01		79.81%			122.81		100.00%
	Calculated heads	100%	2.86	122.81	100%	100%	100.00%				

Table 5.3.4 (cont'd) Metallurgical balances of zinc-pyrite flotation tests conducted by Laboratoire LTM.

		Direct					**Cumulative**				
	Products	**% Weight**	**% Zn**	**Ag (g/t)**	**% Dist. Zn**	**% Dist. Ag**	**Weight %**	**% Zn**	**Ag (g/t)**	**% Dist. Zn**	**% Dist. Ag**
AB-13	Clean. conc. #2	2.84%	62.88	137.00	62.84%	3.10%	2.84%	62.88	137.00	62.84%	3.10%
	Clean. tail. #2	0.90%	24.37	195.00	7.74%	1.40%	3.74%	53.60	150.98	70.57%	4.50%
	Clean. tail. #1	7.52%	10.43	158.00	27.57%	9.46%	11.26%	24.78	155.67	98.15%	13.96%
	Pyrite conc. #1	2.43%	0.28	79.00	0.24%	1.53%	13.69%	20.43	142.06	98.39%	15.49%
	Pyrite clean tail #1	3.62%	0.14	17.00	0.18%	0.49%	17.32%	16.19	115.90	98.57%	15.98%
	Pyrite clean tail #2	1.07%	0.21	79.00	0.08%	0.67%	18.39%	15.26	113.75	98.64%	16.66%
	Pyrite tail	81.61%	0.05	4.00	1.43%	2.60%	100.00%	2.85	24.18	100.00%	19.26%
	Cyanidation			101.38		80.74%			125.56		100.00%
	Calculated heads	100%	2.85	125.56	100%	100%	100.00%				
AB-14	Clean. conc. #2	3.35%	60.87	99.00	74.94%	3.12%	3.35%	60.87	99.00	74.94%	3.12%
	Clean. tail. #2	1.16%	27.92	144.00	11.90%	1.57%	4.51%	52.39	110.58	86.84%	4.70%
	Clean. tail. #1	7.20%	4.22	106.00	11.16%	7.19%	11.72%	22.78	107.76	98.01%	11.89%
	Pyrite conc. #1	3.55%	0.22	88.00	0.29%	2.94%	15.27%	17.53	103.17	98.29%	14.83%
	Pyrite clean tail #1	4.90%	0.14	15.00	0.25%	0.69%	20.17%	13.31	81.75	98.55%	15.52%
	Pyrite clean tail #2	0.58%	0.22	70.00	0.05%	0.38%	20.75%	12.94	81.42	98.59%	15.90%
	Pyrite tail	79.25%	0.05	6.00	1.45%	4.48%	100.00%	2.73	21.65	100.00%	20.38%
	Cyanidation			84.60		79.62%			106.25		100.00%
	Calculated heads	100%	2.73	106.25	100%	100%	100.00%				
Average	**Global**	% Weight	% Zn	Ag (g/t)	% Dist. Zn	% Dist. Ag	% Weight	% Zn	Ag (g/t)	% Dist. Zn	% Dist. Ag
	Clean. conc. #2	3.50%	61.83	173.17	75.84	5.61%	3.50%	61.83	173.17	75.84%	5.61%
	Clean. tail. #2	0.92%	34.99	223.73	11.34	1.91%	4.42%	56.22	183.73	87.18%	7.53%
	Clean. tail. #1	5.33%	5.71	124.90	10.66	6.16%	9.75%	28.63	151.60	97.84%	13.68%
	Pyrite conc. #1	2.70%	0.31	109.83	0.30	2.75%	12.45%	22.49	142.54	98.14%	16.43%
	Pyrite conc. #2	0.73%	0.09	20.80	0.02	0.14%	13.18%	21.25	135.79	98.16%	16.57%
	Pyrite clean tail #1	2.30%	0.13	11.11	0.11	0.24%	15.48%	18.11	117.30	98.27%	16.81%
	Pyrite clean tail #2	0.68%	0.24	72.33	0.06	0.46%	16.16%	17.36	115.40	98.32%	17.26%
	Pyrite tail	83.84%	0.06	5.95	1.68	4.62%	100.00%	2.853	23.64	100.00%	21.88%
	Cyanidation			84.38		78.12%			108.01		100.00%
	Calculated heads	100%	2.853	108.01	100.00	100%	100.00%				

Table 5.3.5 Metallurgical data on the 2nd and 1st cleaner concentrates.

	Products	Weight %	Zn %	Ag (g/t)	% Distribution Zn	 Ag
AB-08	2nd Clean.C.	3.14	61.90	181.00	65.02	6.29
	1st Clean. Conc.	4.91	57.73	253.26	94.93	13.78
AB-09	2nd Clean. Conc.	4.43	62.64	265.00	93.86	12.31
	1st Clean. Conc.	4.80	58.79	253.16	95.38	12.73
AB-10	2nd Clean. Conc.	4.07	60.04	136.00	89.35	5.63
	1st Clean. Conc.	4.63	55.63	148.37	94.03	6.98
AB-12	2nd Clean. Conc.	3.16	63.02	221.00	69.64	5.68
	1st Clean. Conc.	3.95	58.79	231.38	81.18	7.44
AB-13	2nd Clean. Conc.	2.84	62.88	137.00	62.84	3.10
	1st Clean. Conc.	3.74	53.60	150.98	70.57	4.50
AB-14	2nd Clean. Conc.	3.35	60.87	99.00	74.94	3.12
	1st Clean. Conc.	4.51	52.39	110.58	86.84	4.70
Average	2nd Clean. Conc.	3.50	61.83	173.17	75.84	5.61
	1st Clean. Conc.	4.42	56.22	183.73	87.18	7.53

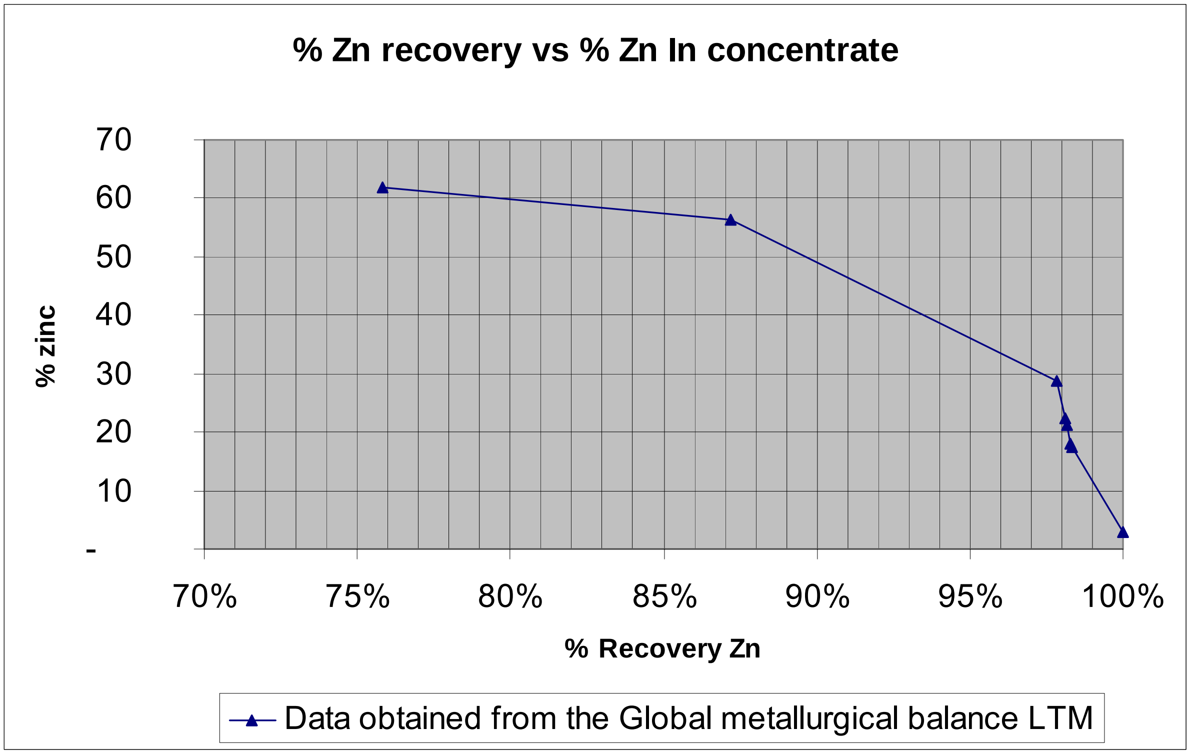


Figure 5.3.1 % Zn recovery vs % Zn in concentrate.

5.3.3.2 Matagami test work

For comparison purposes, Table 5.3.6 presents the results of cycle flotation tests conducted on unleached (not precyanided) ore samples from Abcourt and Barvue parts of the property by the laboratory of Noranda Mines Limited at Matagami (see References).

Table 5.3.6 Summary of the Zn cycle flotation tests on samples from Abcourt and Barvue parts of the property conducted by the Matagami Laboratory.

Test No.	Calculated Heads		Concentrate			
	Zn	Ag	Zn	Ag	% Recovery	
	%	oz/t	%	Oz/t	Zn	Ag
26-33	5.60	4.24	56.14	34.46	94.70	75.80
-40	5.72	4.24	56.60	34.08	94.80	78.30
41	5.43	4.17	52.52	31.00	95.10	78.10
Average (1)	5.54	4.22	55.09	33.18	94.87	77.40
26-42	3.57	3.83	54.04	46.87	94.60	73.60
-44	3.71	4.49	53.48	47.20	95.60	73.20
Average (2)	3.64	4.16	53.76	47.04	95.10	73.40
26-43	6.83	4.22	51.72	27.38	92.90	79.00
-45	6.78	4.68	51.19	29.19	94.80	79.40
Average (3)	6.81	4.45	51.46	28.29	93.85	79.20
Global	**5.33**	**4.28**	**53.43**	**36.17**	**94.61**	**76.67**

Note 1: 60 % Barvue, 40 % Abcourt: Head assayed 5.65 % Zn, 4.33 oz Ag/t.
Note 2: 100 % Abcourt: Head assayed 3.83 oz/t AG, 3.57 % Zn.
Note 3: 100 % Barvue: Head assayed 6.69 % Zn, 4.28 oz Ag/t.

5.3.3.3 Metallurgical balance of the Abcourt process

As for the LTM flotation tests, the Matagami cycle flotation tests included one rougher stage followed by two cleaning stages, but also involved the recirculation of a rougher scavenger concentrate and of the two cleaner tails.

To establish a sound basis for the estimation of the influence of the circulating load on the efficiency of the cyanidation-flotation Abcourt process, BUMIGEME entered into Table 5.3.7 some of the results of the Matagami cycle tests, together with data corresponding to the weighted average of the 2nd cleaner concentrates (Case 1) and of the 1st Cleaner Concentrates (Case 2) of the 6 flotation tests conducted by LTM (c.f. Table 5.3.5).

Table 5.3.7 Reference data on the Matagami and LTM test work.

		Matagami - Cycle Zn Flotation Tests							Matagami	LTM	
		33	40	41	42	43	44	45	Mean	Case 2	Case 1
1st Cycle Concentrate											
	% Wt	9.95	8.40	8.78	5.20	6.49	10.63	10.85	8.61	4.42	3.50
C	% Zn	50.63	57.00	55.41	57.12	55.13	56.52	54.83	55.23	56.22	61.83
A	% Zn Rec.	91.85	83.71	89.63	83.25	96.56	88.05	87.82	88.70	87.18	75.84
1st Cycle Circulating Load (LTM Test – Cleaner Tails)											
	% Wt	4.61	5.80	5.87	5.80		5.97	7.00	5.84	5.33	6.25
	% Zn	5.20	2.92	4.61	7.51		4.56	1.86	4.44	5.71	10.02
E	% Zn Dist	4.39	2.94	5.00	12.26		3.98	1.92	5.08	10.66	22.00
Last Cycle Concentrate											
	% Wt	9.27	10.10	9.91	6.38		12.40	12.65	10.12		
D	% Zn	56.15	56.06	52.52	59.04		51.72	51.19	54.45		
B	% Zn Rec.	95.32	98.99	95.90	96.56		93.98	95.55	96.05		
B/A											
	%	103.80	118.30	107.00	116.00		106.70	108.80	108.5		
D/C											
	%	110.90	98.40	94.80	103.40		91.50	93.40	98.60		
Head Calculated - All Cycles											
	% Zn	5.2	5.72	5.43	3.57	3.71	6.83	6.78	5.32	2.85	2.85
Head Calculated - Last Cycle											
	% Zn	5.5	5.98	5.48	3.43	3.72	6.88		5.40		

The A, C and E factors of the Case 2, being closer then those of case 1 to the corresponding factors of the Matagami tests, were retained to estimate as follows the influence of the circulating load on the overall metallurgical balance of the cyanidation-flotation process.

Estimate of % Zn Recovery

Reference		Matagami Mean	LTM /Case 2		BUMIGEME Estimate
B (1st Cycle) A (last Cycle)	=	$\frac{96.05}{88.70}$	x 87.18	=	94.4 %

Estimate of % Zn in Concentrate

D (Last Cycle) C (1st Cycle)	=	$\frac{54.45}{55.23}$	x 56.22	=	55.49 %

Referring to the results of tests AB-08 (94.93 % Zn recovered in a 57.7 % Zn concentrate) and AB-09 (95.38 % Zn recovered in a 58.79 % Zn concentrate), BUMIGEME retained a value of 94.6 % as its estimate of the % Zn recovery which would be obtained through recirculation of the cleaner tails. BUMIGEME also arrived at the conclusion that the pre-cyanidation of the flotation feed improves the selectivity of the flotation over that of the direct flotation as applied in the Matagami cycle test. Therefore BUMIGEME retained a value of 57.0 % Zn for its estimate of the grade of concentrate which would be produced through recirculating the cleaner tails.

Since the Matagami cycle tests did not include the pre-cyanidation, the results of these tests could not be used to estimate the influence of the circulating load on the Ag recovery. Thus BUMIGEME assumed that 90 % of the silver contained in the 1^{st} Cleaner tail (10.66 %) would be recovered through recirculation resulting into an overall 13.1 % Ag recovery.

Table 5.3.8 presents BUMIGEME's estimated metallurgical balance of the Abcourt process applied to the treatment of Abcourt-Barvue ore having the same characteristics as those of the sample tested by LTM.

Table 5.3.8 Estimated metallurgical balance of the cyanidation zinc flotation process.

Product	Weight	Zn	Ag	% distribution		Ag g/t
	%	%	g/t	Zn	Ag	Recovery
Zn Concentrate	5.11	57.00	146.49	94.60	13.10	7.39
Flotation Tails	94.89	0.18	5.28	5.70	8.90	5.01
Cyanidation			43.99		78.00	43.99
Heads Calc.	100.00	3.08	56.40			

5.3.4 Pyrite flotation

Pyrite flotation tests were conducted on the rougher tails of the Zn circuit. The tests consisted in a 2-minute conditioning stage with $CuSO_4$ followed by a rougher and 2 cleaner flotation stages, using Aerofloat 208 as a collector. BUMIGEME has calculated the metallurgical balance of three tests for which the information contained in the LTM report was sufficiently complete. Results obtained are presented in Table 5.3.9.

Table 5.3.9 Global metallurgical balance of pyrite flotation - Tests AB-12,13,14.

Product	Weight	S	Cumulative	
	%	%	% Weight	% S
Pyrite Tails	90.20	0.48	90.20	0.48
1^{st} Clean Tails	5.10	6.08	95.30	0.08
2^{nd} Clean Tails	1.50	38.19	96.80	1.35
Pyrite Concentrate	3.20	41.20	100.00	2.62
Pyrite Tails+1^{st} Clean Tails	95.30	0.78	95.30	0.78
Pyrite Concentrate+ 2^{nd} Clean Tails	4.70	40.23	100.00	2.62

Considering that, to be classified as non acid generator, the tailings would have to assay less than 0.3 % S, BUMIGEME referred to the results of a flotation test conducted by the URSTM (see References) on the tailings of a zinc flotation test conducted on an unleached 50-50 Abcourt-Barvue ore sample. The conditions and results of this sulfur flotation test are shown in Table 5.3.10.

As it can be seen by floating a pyrite concentrate representing 20.5 % by weight of the zinc flotation tails, the final tails contain less than 0.2 % S. Therefore it was assumed that the same results would be obtained when applied to the "desulphurization" of the zinc flotation tailings, after cyanidation.

Table 5.3.10 Basic data on the USTRM's pyrite flotation test on tailing from a zinc flotation fest - Conducted on a non-cyanidated Abcourt-Barvue ore sample.

		Abcourt (50 %) – Barvue (50 %)
Type d'essai		Désulfuration
Vitesse de rotation		1200
Solide		1000
Pulpe		3226.3
% solide		31 %
pH initial		7.77
pH visée à l'acidification		6
Type de collecteur		Kax-51
Pureté du KAX-51		78.40 %
Dosage collecteur visé (premier dosage)		60
Ajout de collecteur (ml) (premier dosage)		7.5
Dosage collecteur visé (deuxième dosage		40
Ajout de collecteur (ml) (deuxième dosage		5
Dosage collecteur visé (final		100
Concentration en collecteur (%)		1 %
Temps de conditionnement au collecteur		12
Ajout d'eau	5 min	500
	épuisage (10 min)	0
Type de moussant		Cytec F-507
Moussant ajouter avant la flottation		2
Masse du concentré 1 sec		205.7
Masse du concentré 2 sec		99.3
Masse du rejet sec		695
Masse totale sec		1000
Proportion	Concentré 1	21 %
	Concentré 2	10 %
	Rejet	70 %
% soufre	Concentré 1	15 %
	Concentré 2	0 %
	Rejet	0.13 %
Soufre (gramme)	Concentré 1	30.9
	Concentré 2	0.5
	Rejet	0.92
Première flottation	Dosage réel (gft)	58
	Temps de flottation	10
	Rejet #1 (% Soufre)	0.18 %
Deuxième flottation	Dosage réel (gft)	39
	Temps de flottation	10
	Rejet final (% Soufre)	0.13 %
% alimentation (calculé)		3.20 %

5.3.5 Validation test work program

On the basis of its evaluation of the LTM process development test work, BUMIGEME recommended that additional test work be carried out to validate its estimate of the metallurgical efficiency of the cyanidation-flotation process as summarized in Table 5.3.8 above. The work program was to include:

- Preparation of a composite sample representative of the average grade and mineralogical composition of the Abcourt-Barvue mine ore as estimated by GENIVAR;
- Characterization tests on the composite sample such as:
    - cyanidation tests at various NaCN concentrations and cyanidation periods;
    - kinetic flotation tests;
    - LTM conventional cyanidation-flotation tests.
- Pyrite flotation tests to:
    - improve the efficiency of the LTM process;
    - verify the indicated efficiency of the USTRM pyrite flotation process.
- The carrying out of a cyanidation-flotation test which would include 6 Zn-Ag flotation cycles with recirculation of a rougher scavenger concentrate and the two cleaner tails.

5.4 Validation of the abcourt cyanidation-flotation process

5.4.1 Cyanidation-flotation cycle test

As part of the validation test work program recommended by BUMIGEME, LTM carried out a six-cycle cyanidation-flotation test.

5.4.1.1 Sample tested

The cycle test was conducted on a composite sample, identified by BUMIGEME as composite C, representative of the average grade of the Abcourt-Barvue resources (see Table 5.2.1). This 90 kg composite was prepared by Mr. Renaud Hinse, a qualified engineer and president of Abcourt, who delivered to LTM the composite sample contained in two bags, after having taken out a sample from each bag for assay.

Upon its receipt, LTM splitted composite C into 30 x 3 000 g samples which were bagged and identified for future test work. From one 3 000 bag, LTM splitted out 3 x ≈1 000 g samples for head assays purposes. Two samples were handed over to BUMIGEME's representative who brought them to Bourlamaque Assay Office and the third sample was sent by LTM to Techni-Lab. Results of these assays are presented in Table 5.4.1.

Table 5.4.1 Results of assays on composite C.

Sample	Au g/t	Zn %	Ag g/t	Cu %	Pb %
Abcourt	---	2.98	61.8	---	---
Abcourt	---	3.00	61.7	---	---
LTM	<0.06	2.97	57.5	0.108	0.124
LTM	<0.06	3.01	56.7	0.110	0.120
BUMIGEME	0.06	2.96	61.0	0.131	0.124
BUMIGEME	0.05	3.05	60.0	0.135	0.127
Average					

LTM has also determined the size distribution of composite C which is presented in Table 5.4.2.

Table 5.4.2 Size distribution of composite C.

	Weight Retained	Cumulative % Weight	
	%	Retained	Passing
+4 mesh	4.61	4.61	95.39
-4 +10 mesh	20.05	24.66	75.34
-10 +20 mesh	19.26	43.92	56.08
-20 +30 mesh	7.77	51.69	48.31
-30 +48 mesh	11.57	63.26	36.74
-48 mesh	36.74	100.00	0.00
Total	**100.00**		

Considering its coarseness (24.6 % +10 mesh) it would have been preferable to crush composite C to -10 mesh before its splitting in order to minimize the fluctuations of the Ag content in the 3 000 g samples.

5.4.1.2 Description of cycle test

The cycle test comprised the same unit operations as the standard cyanidation-flotation development tests conducted by LTM. The cycle test unit operations were as follows:

Grinding

Six 3 000 g samples were used to carry out the cycle test. Each sample was dry-ground to ≈ 95 % -200 mesh in a 22 cm Ø by 40 cm long rod mill. The mill charge consisted of 5 x 1.8 cm Ø, 11 x 2 cm Ø and 1 x 2.4 cm Ø steel balls.

Cyanidation

For each cycle, a 3 000 g sample was cyanided for 36 hours, at 50 % solids and a 5 kg NaCN/t initial concentration. The cyanidation period completed, the pulp was filtered and washed; the pregnant and washing solutions were measured and assayed. The filter cake was repulped to 50 % solids and aerated during one hour. All these operations were carried out in opened circuit.

Flotation

A Zn-Ag flotation test consisting of a rougher and scavenger stage followed by two cleaning stages was conducted on each aerated pulp (cyanidation tail). The rougher scavenger concentrate and the 1st cleaner tail of each cycle were recirculated to the rougher flotation stage of the next cycle, while the 2nd cleaner tail was recirculated to the 1st cleaning stage of the next cycle.

Pyrite flotation

A one-stage pyrite flotation test was conducted on the Zn-Ag flotation tailing of each cycle test.

5.4.1.3 Results of the cycle cyanidation-flotation test

Results of the cycle test are presented in Table 5.4.3 and summarized in Table 5.4.4, together with the combined metallurgical balance of the 5th and 6th cycles. The detailed evaluation of the results corresponding to each component of the cycle test (cyanidation, Zn-Ag flotation, pyrite flotation) is presented in sections 5.4.3, 5.4.4 and 5.4.5 below.

Table 5.4.3 Metallurgical balance of the cycle test.

						% Distribution			
Products	Weight		Zn	Ag	S	At the Flotation		Global	
	Grams	%	%	g/t	%	Zn	Ag	Zn	Ag
Zn-Ag. Conc.									
1st cycle	137.00	0.76	57.61	191.00	22.88	15.16	8.02	14.90	2.43
2nd cycle	134.00	0.74	60.88	179.70	26.66	15.60	7.34	15.33	2.23
3rd cycle	151.00	0.84	56.02	223.20	31.30	16.29	10.35	16.01	3.14
4th cycle	154.90	0.86	52.75	215.80	29.40	15.71	10.25	15.44	3.10
5th cycle	158.70	0.88	55.23	216.80	29.90	16.83	10.53	16.54	3.20
6th cycle	145.70	0.81	55.97	224.30	31.50	5.70	10.03	15.43	3.05
Total Zn-Ag.conc.	**881.30**	**4.89**	**56.28**	**209.34**	**28.74**	**95.28**	**56.53**	**93.63**	**17.16**
Pyrite Conc.									
1st cycle	411.00	2.28	0.46	41.30	30.50	0.36	5.20	0.36	1.58
2nd cycle	421.00	2.34	0..62	51.90	31.80	0.50	6.71	0.49	2.04
3rd cycle	482.00	2.68	0.43	36.50	30.50	0.40	5.40	0.39	1.64
4th cycle	434.30	2.42	0.29	27.40	28.40	0.24	3.66	0.24	1.11
5th cycle	538.80	2.99	0.36	36.00	24.60	0.37	5.94	0.37	1.80
6th cycle	519.30	2.89	0.39	34.00	27.40	0.39	5.43	0.38	1.65
Total Pyrite conc.	**2 806.40**	**15.60**	**0.42**	**37.54**	**28.66**	**2.27**	**2.34**	**2.23**	**9.82**
Pyrite Tail									
1st cycle	2 220.00	12.34	0.07	2.40	0.22	0.30	1.64	0.9	0.50
2nd cycle	2 372.00	13.18	0.07	2.90	0.25	0.32	2.11	0.31	0.64
3rd cycle	2 397.00	13.31	0.07	2.70	0.20	0.32	1.98	0.32	0.60
4th cycle	2 346.00	13.02	0.07	1.90	0.27	0.32	1.37	0.31	0.41
5th cycle	2 198.00	12.21	0.07	2.30	0.27	0.30	1.55	0.29	0.47
6th cycle	2 442.10	13.56	0.07	1.10	0.23	0.33	0.82	0.32	0.25
Total Pyrite Tail	**13 975.10**	**77.62**	**0.07**	**2.21**	**0.24**	**1.88**	**9.47**	**1.85**	**2.88**
Circ. Load									
2nd Clean Tail	15.70	0.09	1.23	39.60	6.54	0.04	0.20	0.04	0.06
1st Cleaner Tail	201.70	1.12	0.35	16.10	4.41	0.14	1.00	0.13	0.30
Rgh. Scav. Conc.	122.30	0.68	1.69	13.00	8.89	0.40	0.49	0.39	0.15
Total Circ. Load	**339.70**	**1.89**	**0.87**	**16.10**	**6.12**	**0.57**	**1.68**	**0.56**	**0.51**
Flotation Feed	18 002.50		2.8884	18.11	6.19	100.00	100.00	98.28	30.37
Cyanidation Extraction									
1st cycle			0.008	7.07				0.261	11.85
2nd cycle			0.009	7.61				0.300	12.76
3rd cycle			0.008	6.35				0.272	10.65
4th cycle			0.009	6.63				0.306	11.11
5th cycle			0.009	7.22				0.312	12.11
6th cycle			0.008	6.65				0.272	11.15
Total Cyanidation extraction			**0.051**	**41.53**				**1.724**	**69.63**
Heads Calculated	18 000.00	100.00	2.94	59.64				100.00	100.00

Table 5.4.4 Summary of cycle test metallurgical balance.

Products	Weight		Zn	Ag	S	% Distribution	
	g	%	%	g/t	%	Zn	Ag
Zn-Ag Concentrate	881.3	4.89	56.28	209.34	28.74	93.63	17.16
Pyrite Concentrate	2 806.40	15.60	0.42	37.54	28.66	2.24	9.82
Pyrite Tails	13 975.1	77.62	0.07	2.21	0.24	1.85	2.89
Circulating Load	339.7	1.89	0.87	16.10	6.12	0.57	0.51
Flotation Feed	18 002.5		2.89	18.11	6.19	98.30	30.37
Cyanidation Extraction			0.05	41.53		1.70	69.63
Heads Calculated	18 000.0		2.94	59.64			
Combined Metallurgical Balance of the 5th & 6th Cycle							
Zn-Ag Concentrate	304.4	5.07	55.58	220.57	30.71	94.31	18.56
Pyrite Concentrate	1 058.1	17.62	0.37	34.96	25.96	2.21	10.22
Pyrite Tails	4 640.1	77.31	0.07	1.67	0.26	1.81	2.18
Cyanidation Extraction			0.05	41.61		1.67	69.04
Heads Calculated	6 002.6	100.00	2.99	60.27			

The correspondence between the total weight of the feed (6 000 g) and that of the products (6 002.6 g) in the 5th and 6th cycles confirm that equilibrium had been reached. Furthermore, considering the closeness of the results of the overall cycle test to those of the combined 5th and 6th cycles, BUMIGEME considers these results as being trustworthy.

5.4.2 Grinding

The cycle test and all other cyanidation-flotation tests on composite C were conducted at about 95 % -200 mesh grind which had been demonstrated by LTM as being the appropriate grind. This was confirmed by the results of the cycle test as discussed in section 5.4.4.3. Results of the screen analysis are presented in Table 5.4.5.

Table 5.4.5 Screen analysis of Abcourt-Barvue ore samples.

Matagami Laboratory Screen Analysis on an Abcourt-Barvue Ore Sample Ground to 94.2 % -200 mesh.

Screen Opening	Weight %	Cumulative % Weight	
	Retained	Retained	Passing
75	5.77	5.77	94.23
56	1.95	7.72	92.28
40	38.85	46.57	53.43
28	27.95	74.52	25.48
-28	25.48	100.00	

Reference: Matagami report January 5, 1981.

LTM Screen Analysis on Composite C Ground to 95.4 % passing -200 mesh

Screen Opening	Weight %	Cumulative % Weight	
Mesh	Retained	Retained	Passing
65	0.65	0.65	99.35
100	0.27	0.92	99.08
140	1.37	2.29	97.72
200	2.35	4.64	95.36
270	8.05	12.69	87.31
325	3.38	16.07	83.93
400	6.53	22.60	77.40
-400	77.40	100.00	

Based on these results the K_{80} factor of the Abcourt-Barvue ore ground to 95 % -200 mesh is estimated at 43µ (325 mesh).

5.4.3 Cyanidation

5.4.3.1 Cycle tests

Table 5.4.6 presents the results of the six cyanidation tests which were carried out as the first part of the global cyanidation-flotation cycle test. Each test was conducted at 50 % solids, a concentration of 5 kg NaCN/l and 36 hour cyanidation period.

Table 5.4.6 Results of the cyanidation tests as part of the cyanidation-flotation cycle test.

Cycle	NaCN Consumed	Extraction		Ag Calculated Head	Ag Recovery
		Zn	Ag		
No	Kg/t	%	g/t	g/t	%
1	2.35	0.046	42.39	63.46	72.44
2	2.50	0.053	45.66	64.84	72.18
3	2.45	0.048	38.12	56.80	66.44
4	2.25	0.054	39.75	57.59	70.55
5	2.45	0.055	43.33	65.21	68.83
6	2.60	0.048	39.90	51.69	67.18
Average	**2.43**	**0.051**	**41.52**	**59.64***	**69.63***

* As calculated from the Metallurgical Balance (see Table 5.4.3).

Ag extraction

All cyanidation tests were conducted in open circuit and without recirculation of solution which is not feasible at the laboratory level. Based on the results BUMIGEME estimates that cyanidation of the Abcourt-Barvue ore resources, with an estimated grade of 54.96 g Ag/t, 69.6 % (minimum) of the Ag will be extracted.

NaCN consumption

The average consumption rate of 2.43 kg NaCN/t appears relatively high based on similar operations and could be due to the oxidation of the samples during dry grinding. In its operation cost estimates, BUMIGEME has used consumption rates of cyanide of 1 kg/t (refer to Section 6 – Design Criteria) which were obtained from experienced operators in similar operations and from tests conducted by LTM in January 2007 on different Abcourt ore composites of different grades. It is suggested that this hypothesis be verified during the carrying out of the recommended optimization test work program (section 15).

5.4.3.2 Cyanidation characterization tests

Tests on composite B

Before receiving composite C, on which the cycle test was conducted, LTM carried out four cyanidation-flotation tests on a composite, referred to by BUMIGEME as composite B, made up of crushed drill core samples originating from the Abcourt mining property. The average of the calculated heads for each one of the four tests stands at 2.94 % Zn and 67.61 g Ag/t. The four cyanidation tests were conducted at 50 % solids and 5 kg NaCN/t concentration on samples ground to ≈95 % -200 mesh. Results of these cyanidation tests are presented in Table 5.4.7.

Table 5.4.7 Results of the cyanidation tests on composite B.

Test No AB-	NaCN Consumed Kg/t	Cyan. Time Hours	Extraction		Heads Calc. Ag g/t	Recovery % Ag
			Zn %	Ag g/t		
15	3.25	24	0.040	57.78	84.01	69.07
16	2.80	24	0.056	62.78	91.33	69.24
17	2.95	36	0.063	71.23	87.06	81.87
18	3.33	36	0.063	78.65	95.88	82.93

These results showed that cyanidation for 36 hours at a 5 kg NaCN concentration gives a slightly higher recovery than those obtained on Composite A, that is 81.32 % and 80.73 % on samples grading respectively 95.39 g Ag/t and 98.41 g Ag/t, from cyanidation tests AB-09 and AB-10 conducted at a 10 kg NaCN/t concentration and over a 48 hour cyanidation period (Table 5.4.12).

Considering these results, in the wait for the results of characterization cyanidation tests on composite C, it was decided to retain a 5 kg NaCN/t concentration and a 36-hour cyanidation period for the other tests aiming at optimizing the conditions of operation to be retained for the cyanidation-flotation cycle test.

Characterization tests on composite C

LTM conducted four cyanidation characterization tests on composite C used for the cycle test. One 3 kg sample bag and a ±1 016 g sample split out from another 3 kg sample bag were mixed and ground together to ≈ 95 % -200 mesh. The ground product was then divided into three 1 000 g samples and one 1 015 g sample. Each sample was cyanided in a 50 % solid pulp. Results obtained are presented in LTM Report (22/12/06) as shown in Table 5.4.8.

Table 5.4.8 Results of characterization tests on composite C.

	NaCN Concentration	Cyanida-tion Time	NaCN Consumed	Recovery	
				Silver	Zinc
AB-	Kg/t	Hours	Kg/t	%	%
22	5.0	48	3.65	72.84	1.65
23	10.0	48	5.00	74.20	1.83
24	5.0	36	2.85	76.24	1.96
25	10.0	36	4.00	78.72	2.20
Laboratoire LTM inc					22/12/06

The filtered and washed tailings from Tests AB-24 and 25 were put together to realize the Zn-Ag flotation test AB-26, those of cyanidation tests AB-22 and 23 to realize flotation test AB-27. From the data contained in the LTM report, BUMIGEME concluded that the LTM metallurgical balance was calculated as shown in Table 5.4.9.

Table 5.4.9 Calculation of the % Ag recovery from characterization tests as per LTM Report.

	a	b	a+b	a ÷ (a+b)	Extraction
Test no	Cyanidation Extraction Ag g/t	Calculated Heads		Ag Recovery %	% Zn
		To flotation Ag g/t	To cyanidation Ag g/t		
AB-22	51.90	19.35	71.25	72.84	0.050
AB-23	55.71	19.35	75.06	74.20	0.055
AB-24	51.15	15.94	67.09	76.24	0.056
AB-25	58.95	15.94	74.89	78.72	0.063

Based on these results LTM concluded that “the more interesting result is the AB-24 because it has the lowest cyanide consumption and the second best recovery of silver”.

In the course of its evaluation of the LTM Report, when preparing Table 5.4.9, BUMIGEME concluded that in its calculation of the % Ag recovery, LTM started from the hypothesis that the Ag content of each couple of cyanidation tails “AB-22, AB-23” and “AB-24, AB-25” was equal independently of the quantity of Ag extracted by the cyanidation.

BUMIGEME re-calculated the efficiency of the characterization tests starting from the much higher probability that, when a 2 000 g sample ground to 95 % -200 mesh is carefully splitted in two parts, each part will have the same Ag content. Therefore BUMIGEME recalculated the heads of each couple of cyanidation tests; results are shown in Table 5.4.10.

Table 5.4.10 Heads of the characterization cyanidation tests as calculated by BUMIGEME.

Test No	Weight	Cyanide Extraction	Calc. Heads Flotation	Heads Calc. Cyanidation
	g	Ag g/t	Ag g/t	Ag g/t
AB-21	1 000	51.90	19.35	
AB-22	1 000	55.71	19.35	
Feed AB-21-22	2 000	53.80	19.35	73.15
AB-23	1 000	51.15	19.35	
AB-24	1 015	58.95	19.35	
Feed AB-23-24	2 015	55.09	15.94	71.02

Summary of the metallurgical balance of the cyanidation tests

Three other cyanidation tests were conducted on composite C as part of 3 cyanidation-flotation tests. These cyanidation tests were also conducted under the 5 kg NaCN and 36 hours operation parameters. Table 5.4.11 summarizes the results of all cyanidation tests conducted on Composite C.

Table 5.4.11 Summary of the metallurgical balance of cyanidation tests conducted on composite C.

Test no	NaCN Consumed	Silver Extraction	Heads Calculated	Recovery	Zn Extraction
	Kg/t	Ag g/t	Ag g/t	% Ag	% Zn
22	3.65	51.90	73.15	70.95	0.050
23	5.00	55.71	73.15	76.16	0.055
24	2.85	51.15	71.02	72.02	0.056
25	4.00	58.95	71.02	83.00	0.063
29	2.61	40.15	57.86	69.16	0.037
30	2.40	39.48	57.72	68.40	0.038
32	2.55	37.46	56.81	65.84	0.040
Cycle-1	2.35	42.39	63.46	72.44	0.046
Cycle-2	2.50	45.66	64.84	72.18	0.053
Cycle-3	2.45	38.12	56.80	66.44	0.048
Cycle-4	2.25	39.75	57.59	70.55	0.054
Cycle-5	2.45	43.33	65.21	68.83	0.055
Cycle-6	2.60	39.90	51.69	67.18	0.048
Average (1)	2.64	42.46	60.78	69.85	0.048
Average (2)	2.48	41.07	58.82	69.82	0.047
Cycle test	2.43	41.52	59.64	69.72	0.051

Average (1): Weighted average of all tests.
Average (2): Weighted average of all tests with operation parameters of 5 kg NaCN/t and 36 hours.

The above Table 5.4.11 also demonstrates that the metallurgical balance of the cyanidation tests carried out as a component of cycle test is almost identical to that obtained from all the cyanidation tests conducted on Composite C.

Influence of the Ag content

Table 5.4.12 presents results of cyanidation tests conducted on samples having an Ag content varying from 71 to 125 g/t. These results suggest that the leachable (cyanidation) part of the Ag contained in the Abcourt-Barvue ore would be almost a constant with a value of ±80 %.

Table 5.4.12 Influence of the Ag content on the % Ag recovery.

Test no	Composite	Operation NaCN	Parameter Time	Heads Calculated	Ag Recovery
		Kg/t	Hours	Ag g/t	%
AB-25	C	10	36	71.02	83.00
AB-17	B	5	36	71.23	81.87
AB-18	B	5	36	78.65	82.93
AB-09	A	10	48	95.39	81.32
AB-10	A	10	48	98.41	80.73
AB-14	A	5	48	106.25	79.62
AB-12	A	5	48	122.81	79.81
AB-13	A	5	48	125.56	80.74

It should be emphasized however, that the optimum parameters of operation 10 kg NaCN/t and 36 hours which were determined for Composite C (test AB-25) were not tested on any other sample.

5.4.4 Zn-Ag flotation

5.4.4.1 Description of cycle tests

A 6-cycle Zn-Ag flotation test was conducted by LTM to validate and firm up BUMIGEME's October 2006 estimate that the Abcourt concentrator will recover by flotation into a Zn concentrate of 57 % Zn at 96 % recovery of the Zn contained in the cyanidation tails (feed to flotation).

The feed sample to each cycle consisted of the filtered washed and aerated tailings of one of the 6 cyanidation tests each conducted on a 3 000 g sample of Composite C, representative of the Abcourt-Barvue ore resources. This test was conducted along a flowsheet identical to that retained for the Abcourt concentrator as describes in section 6.

The primary scavenger concentrate and the 1st cleaner tails of each cycle were recirculated to the primary flotation stage of the following cycle, while the 2nd cleaner tails were recirculated to the 1st cleaning stage of the following cycle test. The conditions of operation of the Zn-Ag flotation tests are summarized in Table 5.4.13 while the results are presented in Table 5.4.14.

Table 5.4.13 Parameters of the cycle flotation tests conducted on aerated pulps.

Flotation Steps	Solids %	pH	Flotation time (min)	Flotation Reagents	
				Name	Dose (g/t)
#1 Conditioner	35	11	2	CaO	940
				$CuSO_4$	563
Rougher #1			1	MIBC	50
				3477	70
Rougher #2			2	3477	2
Rougher #3			3	3477	2
Rougher #4			4	3477	5
1st Cleaner	35	12	2	CaO	870
2nd Cleaner			2	CaO	675

Table 5.4.14 Metallurgical balance of Zn-Ag flotation cycle tests.

Products	Weight		Zn	Ag	% Distribution	
	Grams	%	%	g/t	Zn	Ag
1st Final Conc.	137.0	0.76	57.61	191.0	15.16	8.02
2nd Final Conc.	134.0	0.74	60.88	179.7	15.60	7.34
3rd Final Conc.	151.0	0.84	56.02	223.2	16.29	10.35
4th Final Conc.	154.9	0.86	52.75	215.8	15.71	10.25
5th Final Conc.	158.7	0.88	55.23	216.8	16.83	10.54
6th Final Conc.	145.7	0.81	55.97	224.3	15.70	10.03
1st Float. Tail.	2631.0	14.62	0.13	8.48	0.66	6.83
2nd Float. Tail.	2793.0	15.52	0.15	10.29	0.82	8.82
3rd Float. Tail.	2879.0	15.99	0.13	8.36	0.72	7.38
4th Float. Tail.	2780.3	15.44	0.11	5.89	0.56	5.02
5th Float. Tail.	2736.8	15.20	0.13	8.93	0.67	7.49
6th Float. Tail.	2961.4	16.45	0.13	6.86	0.72	6.24
6th Sec. Cln. tail.	15.7	0.09	1.23	39.60	0.04	0.20
6th 1 st Cln. tail.	201.7	1.12	0.35	16.10	0.13	1.00
6th Scav. Conc.	122.3	0.68	1.69	13.00	0.39	0.49
Summary						
Final Conc.	881.3	4.89	56.28	209.34	95.29	56.53
Flotation Tails	16781.5	93.22	0.13	8.12	4.15	41.78
Circulating load	339.7	1.89	0.87	16.10	0.56	1.69
Flotation Feed	18002.5	100.00	2.89	18.11	100.00	100.00

5.4.4.2 Metallurgical balance of the 5th & 6th cycle

The combined metallurgical balance of the 5th and 6th cycles, is presented in Table 5.4.15. The fact that the combined weights of the flotation concentrate and tailings (6 002.5 g) is almost identical to the weight of the feed (2 x 3 000 g) to these two cycles confirms that the state of equilibrium had been reached. The metallurgical balance of the 6 cycle test is also very similar to that of the combined 6th and 5th cycles which confirms that the results are trustworthy.

Table 5.4.15 Combined metallurgical balance of the 5th & 6th Cycle Zn-Ag flotation test.

	Weight		Zn		Ag	
	Grams	%	%	% Dist.	g/t	% Dist.
Flot. Conc.	304.40	5.07	55.58	97.59	220.57	60.00
Flot. Tails	5 698.20	94.93	0.13	2.41	7.85	40.00
Heads Calc.	6 002.60	100.00	2.94		18.64	

5.4.4.3 Circulating load and grinding parameters

Each component of the circulating load, corresponding to the 6th and the last cycle, was weighted and the weight distribution calculated as follows:

Table 5.4.16 Distribution of the circulating load.

	Weight			Zn	Ag	S
Component	g	%*	% **	%	g/t	%
2nd Cleaner conc.	15.70	4.62	0.52	1.23	39.60	6.54
1st Cleaner conc.	201.70	59.38	6.72	0.35	16.10	4.41
Rough. Scavenger Conc.	122.30	36.00	4.08	1.69	13.00	8.89
Total Circulating Load	**339.70**	**100.00**	**11.32**	**0.87**	**16.10**	**6.12**
Feed to the 6th Cycle	3 000		100.00		18.11	6.19

* % calculated on the basis of the total weight of the circulating load.

** % calculated on the basis of the weight of the fresh feed to 6th cycle.

The circulating load of the 6th and last cycle corresponds to 11.3 % by weight of the "fresh feed" (3 000 g) to each cycle. This relatively low recirculation rate, retained for the design of the concentrator, confirms that there were no undue build-ups of middlings. Therefore the 95 % -200 mesh grind, applied in the cycle test provides adequate liberation of the zinc and silver minerals and is also retained for the design of the concentrator. Thin sections of various flotation products were prepared but results of their study had not been received when the present report was prepared.

5.4.4.4 Metallurgical balance of the Abcourt Ag cyanidation and Zn-Ag flotation circuits

Table 5.4.17 presents BUMIGEME estimates of the metallurgical balance of the Abcourt Ag cyanidation and Zn-Ag flotation circuits, when processing the Abcourt-Barvue ore grading 3.11 % Zn and 54.96 % Ag/t (see Table 5.2.1).

Table 5.4.17 Estimated metallurgical balance of the Abcourt Ag cyanidation and Zn-Ag flotation circuits.

	Weight	Zn		Ag	
	%	%	% Dist.	g/t	% Dist.
Products					
Flotation Concentrate	5.48	54.50	96.0	181.04	18.0
Flotation Tail	94.52	0.13	4.0	6.97	12.0
Cyanidation Extraction		---	---	38.47	70.0
Heads*	100.00	3.11	100.0	54.96	100.0

BUMIGEME considers that through optimizing the parameters of the cyanidation circuit it might be possible to raise its silver recovery level to ±78 %. Similarly the Zn recovery could probably be raised to over 96 % through using a combination of frother collector which would yield a better froth.

The almost identical results obtained in both cases for the weight of the feed (3 000 g per cycle) vs that of the products as well as for the heads assayed vs the heads calculated demonstrate that the cycle test was well carried out and that the results obtained are trustworthy.

Grade of concentrate

Although the grade of the concentrate from the combined 5th and 6th cycles stands at 55.6 % Zn, BUMIGEME estimates that the Abcourt concentrator could produce a 56 % Zn concentrate. However, BUMIGEME, in order to improve the Zn recovery, retained a concentrate of 54.5 % Zn which is an excellent concentrate for smelters.

Zinc recovery

The 1st, 3rd, 5th and 6th cycles returned tailings at 0.13 % Zn, while 0.15 % Zn and 0.11 % Zn (0.13 % Zn average) were respectively obtained from the 2nd and 4th cycle. Therefore BUMIGEME retained 0.13 % Zn to estimate the efficiency of the Zn-Ag flotation circuit.

Referring to these results and considering the possibility of improving the conditions of operation of the flotation circuit, the Zn content of the tailings could probably be lowered to 0.12 %, and the recovery increased to 96 % Zn.

Silver recovery

The recirculation of the cleaner tails permits to increase the Ag recovery from a level of 48 % in the 1st cycle to over 60 % starting with the 3rd cycle and maintained in the 4th, 5th and 6th cycle. Therefore BUMIGEME estimates that 60 % of the silver contained in the feed to the Zn-Ag flotation circuit, equivalent to 18 % contained in the mill feed, will be recovered in the Abcourt concentrator flotation circuit.

In the cyanidation circuit, a minimum of 70 % of the Ag contained in the Abcourt-Barvue ore will be recovered.

5.4.5 Pyrite flotation

Pyrite flotation characterization tests were conducted on composite C. Each test consisted of a rougher flotation followed by a cleaning stage. These tests returned pyrite tails (mill tailings) assaying over 0.3 % S. Considering that to be classified as "non acid generator" the mill tailings would have to contain less than 0.3 % S, LTM decided to delete the cleaning stage and to add a fourth step of collector addition flotation.

Retaining the conditions shown in Table 5.4.18, LTM conducted a pyrite flotation test on the tailings of each one of the 6 cycle Zn-Ag flotation tests. Referring to the metallurgical balance of the cycle test (Table 5.4.3), one notes that each pyrite flotation test return tailings assaying less than 0.3 % for an average of 0.24 % S as shown in the summary (Table 5.4.19). Therefore BUMIGEME considers that this flotation circuit retained for the Abcourt concentrator will return mill tailings assaying less then 0.3 % and a pyrite concentrate assaying approximately 30 % S and corresponding to about 15 % by weight of the mill feed.

Table 5.4.18 Parameters of the pyrite flotation tests on the tailings of the Zn-Ag cycle flotation tests.

Flotation Stage	ph	Solids %	Period Minute	Flotation Agent Added	
				Type	g/t
Conditioner	6	35	2	H_2SO_4	6000
Conditioner				PAX	630
1st Rougher			4	MIBC	50
2nd Rougher			4	PAX	110
3rd Rougher			4	PAX	110
4th Rougher			4	PAX	110

Table 5.4.19 Summary of the pyrite flotation tests.

	Weight* %	Zn %	Ag %	S %	% Dist. Zn	 Ag
Pyrite Conc.	15.60	0.42	37.54	28.66	2.23	9.82
Pyrite Tail.	77.62	0.07	2.21	0.24	1.85	2.88
Heads Calc.**	93.22	0.13	6.66	5.00	4.08	12.70

* Calculated on the basis of the total feed to the cyanidation-flotation cycle test.

** Heads of the pyrite circuit corresponding to the tailings of the Zn-Ag flotation circuit.

5.5 Data corroboration

5.5.1 Sample preparation

The origin of Composite C, used for the realization of the cycle test is presented in section 5.3.1 together with the method used to split out the 3 000 g samples for test work and the 1 000 g samples for head assays.

5.5.2 Assaying

All products obtained from the LTM test work conducted on samples from the Abcourt property and referred to in this report were assayed by Techni-Lab S.G.B. Abitibi Inc. which was accredited ISO 17025 for environment analysis by the Quebec Ministry of Environment. Techni-Lab also participates on a voluntary basis to the PTP-MAL since 1998. A reference document on Techni-Lab, its standard sample preparation and assaying, quality control and quality assurance procedure is included in Appendix 1. Table 5.4.20 presents results of internal check assays realized on CANMET standard samples parallel to assays of Abcourt-LTM samples. Techni-Lab also realized check-assays on "in house" prepared standards parallel to the assays of Abcourt-LTM samples of cyanidation solutions.

Table 5.4.21 presents results of duplicate assays realized on the same solutions as prepared for the AA. Other check assays were also made starting from the same pulp. With the exception of the Ag assays on sample SFCP AB-34 (52.2 g Ag/t vs 46.1 g Ag/t), These results confirm the reliability of the assay results as reported by Techni-Lab.

Table 5.4.20 Results of Techni-Lab internal verification assays.

	Ag g/t	Zn %	Cu %	Pb %
STD-CCU-1C				
Nov 27 2006	128.9	4.04	0.108	0.351
Nov 27 2006	128.9	4.04	0.110	0.344
Nov 29 2006	128.9	4.04		0.351
Nov 29 2006	128.9	4.04		0.344
Dec 20 2006	128.8	4.03		
Dec 20 2006	129.2	4.05		
Dec 20 2006	128.5			
STD-CZ-3				
Nov 27 2006	45.5	50.75	0.616	0.113
Nov 27 2006	44.3	50.60		
Nov 27 2006	44.4	51.10		
Nov 29 2006	45.2	51.60		0.110
Nov 29 2006	45.3	50.30		0.112
Dec 07 2006	43.7	50.92		
Dec 20 2006	46.5	50.80		
Dec 20 2006	45.5	50.20		
Dec 20 2006	44.4	50.90		

Table 5.4.21 Comparative results of duplicate assays realized by Techni-Lab.

	Duplicate Assays		Original Assays	
	Ag mg/l	Zn mg/l	Ag mg/l	Zn mg/l
Solution-No				
LCL AB-25	24.7	254	24.6	265
LCL AB-32	13.0	123	13.0	122
LCL AB-36	31.8	816	30.8	796
LCL AB-38-A	55.6	832	55.6	848
LCL AB-41	14.0	174	13.9	196
Solids- No				
SFCI AB-27	108.2	32.53	112.8	35.84
SFCI AB-27	104.66	33.80	103.5	32.75
SFNIAZ AB-29	271.3	6.85	274.1	6.86
SFCP AB-34	52.2	0.648	46.1	0.653
SCR AB-37	30.8	2.884	30.8	2.94
SFR AB-44	1.1	0.073	1.1	0.071
SFNIZ AB-44	17.1	0.352	16.1	0.350
SFR AB-43	0.26 % S vs 0.27 % S			

5.5.3 Control

BUMIGEME assigned Mr. Jean-Guy Saint-Jean as its representative at the LTM laboratory, where he supervised the carrying out of all test work conducted on Composite C from the receipt of the composite to the sending of the last product of the cycle test to Techni-Lab. On this basis bumigeme confirms that the LTM test work was correctly conducted according to the procedures and conditions of operations as described in LTM report dated December 22, 2006. Furthermore, at BUMIGEME's request, six samples were returned to Techni-Lab for check assays. Results are presented in Table 5.4.22.

Table 5.4.22 Check assays results.

Sample	Au ppb	Ag ppm	Zn %	S %
SFR AB-42	7	1.7	0.073	0.24
SFCP AB-42	91	28.3	0.265	
SFCZ AB-42	40	205.0	52.650	
SFCP AB-43	69	30.0	0.430	
SFCZ AB-43	58	235.3	51.100	
SFR AB-46	6	17.3	0.086	
SFR AB-47	7	12.3	0.073	
SFR AB-48	14	14.9	0.163	
SFR AB-49	12	9.6	0.124	
SFR AB-50	11	5.3	0.064	
SFR AB-51	22	6.1	0.143	
SFC AB-46	25	515.6	20.020	
SFC AB-47	10	421.3	15.620	
SFC AB-48	20	102.4	23.980	
SFC AB-49	21	104.4	21.120	
SFC AB-50	123	48.0	11.400	
SFC AB-51	221	58.3	24.800	
SFR (Recheck) AB-42	---	---	---	0.22
SFR (Recheck) AB-51	21	6.6	0.180	
SFR (Recheck) AB-47	---	15.40	---	

6. PROCESS DESIGN

6.1 Design criteria

The design criteria used for development of the flowsheet have been based on the test information available and BUMIGEME's in-house experience for similar operations. Process flowsheets are provided in Appendix 2.

6.1.1 General

Annual ore production	650 000	t
Mill head grade	3.11 %	Zn
	54.96 g/t	Ag
Recovery of zinc	96.0 %	Zn
Recovery of silver in cyanidation	70 %	Ag
Distribution of silver in zinc concentrate	18 %	Ag
Operation schedule	365	d
Daily nominal capacity	1 800	t
Operating time	92.0	%
Daily design capacity	1 956	t
Milling rate	81.5	t/h
Ore specific gravity	3.0	t/m^3
Ore bulk density	2.2	t/m^3
Zn concentrate specific gravity	4.25	t/m^3
Zn concentrate bulk density	2.30	t/m^3
Zn concentrate grade	56 %	Zn
Zn concentrate weight recovery	5.25	%

6.1.2 Coarse ore storage

Coarse ore storage bin live capacity	1 800	t
Coarse ore storage bin capacity (in treatment days)	1	d
Ore moisture	3.0	%

6.1.3 Grinding

Fineness of grinding (K_{80})	43	µm
Fineness of grinding (% passing 74 µm)	95	%
SAG mill work index (metric)	12.3	kWh/t
SAG mill feed (SAG F_{80})	125 000	µm
SAG mill product (SAG P_{80})	500	µm

SAG mill energy consumption	5.15	kWh/t
SAG mill discharge solids density	78	%
SAG mill ball charge	8.0	% vol.
SAG mill circulating load	75	%
Ball mill work index (metric)	12.43	kWh/t
Ball mill feed (F_{80})	500	µm
Ball mill product (P_{80})	43	µm
Ball mill energy consumption	12.96	kWh/t
Ball mill circulating load	300	%
Ball mill discharge pulp solids density	75	%

6.1.4 Primary thickening

Unit area	0.7	m^2/td
Feed pulp solids density	35	%
Underflow pulp solids density	50	%

6.1.5 Cyanidation

Retention time	36	hours
Number of stages	2	--
Cyanidation pH	11.5	--
Agitator aeration rate	8.6	m^3/h/m^2
Agitator power provided	0.26	kW/1 000 l
Sodium cyanide concentration	5.0	g/l
Sodium cyanide consumption	1.0	kg/t
Lime consumption	1.0	kg/t

6.1.6 Gold/Silver solution recovery

Number of filtration and washing stages	2	--
Wash ratio per stage	1.5	--
Specific filtration rate	200	kg/m^2/h
Wash efficiency	97.8	%

6.1.7 Gold/Silver solution clarification and precipitation

Pregnant daily solution rate	6 072	m^3/d
Zinc dust consumption	0.05	kg/t
Lead acetate consumption	0.005	kg/t

6.1.8 Flotation circuits

Zinc flotation times (lab locked cycle test x 2)

Aeration time	60	min.
Conditioning time	8	min.
Rougher	12	min.
Scavenger	8	min.
1st Cleaner	4	min.
1st Cleaner – scavenger	4	min.
2nd Cleaner	4	min.

Pyrite flotation times (lab locked cycle test x 2)

Conditioning time	8	min.
Rougher flotation time	32	min.

Flotation pH

Zinc rougher/scavenger circuit	11.0	--
Zinc cleaner circuit	12.0	--
Pyrite flotation	6.0	--

Flotation pulp solids density

In Zn rougher and scavenger circuit	35	%
In Zn 1st Cleaner and scavenger circuit	27 – 29	%
In Zn 2nd cleaner circuit	15 – 20	%
In Pyrite rougher circuit	28	%

6.1.9 Zinc concentrate thickening

Unit area	0.4	m^2/mt/d
Feed pulp solids density	14	%
Underflow pulp solids density	65	%

6.1.10 Zinc concentrate filtration

Filter type	pressure	--
Filtration rate	n/a	kg/m^2/h
Filter cake moisture content	8.0	%

6.1.11 Zinc storage capacity

Zinc concentrate bulk density	2.30	mt/m^3
Concentrate storage capacity (in daily production)	2.0	d

6.1.12 Chemical reagent consumption

Sulfuric acid	Pyrite flotation	4.4 kg/t
MIBC	Zinc and pyrite flotation	0.10 kg/t
Cytec 3477	Zinc Flotation	0.07 kg/t
Copper sulfate	Zinc and pyrite flotation	0.39 kg/t
PAX	Pyrite flotation	0.44 kg/t
Percol 338	Primary and Zinc concentrate thickeners	0.025 kg/t

6.2 Process description

6.2.1 General

The main process steps for treating the Abcourt and Barvue ores are primary crushing and stockpiling, SAG/ball grinding, primary thickening and double stage cyanidation of the ores, gold/silver solution recovery by double stage filtration and washing, gold/silver solution clarification, precipitation of dissolved gold & silver by zinc dust, refining of gold/silver precipitates, aeration of cyanide tails, zinc flotation from aerated cyanide tails, and dewatering of a zinc concentrate by thickening and filtration. There is also a pyrite (or sulphide) concentrate produced as the last stage of differential flotation. This pyrite (or sulphide) concentrate is disposed of in a safe sulphide containment cell.

6.2.2 Crushing and ore stockpiling

Barvue and Abcourt ores are mixed on outside stockpiles located near the crushing facilities. The ore is picked up and carried by a front end loader over a feed hopper equipped with an incline 500 mm square openings grizzly with the undersize dropping into a lined 50 t hopper. Mixing the ore will regulate the grade fed to the processing plant. The ore is extracted from the feed hopper by a reciprocating feeder equipped with a slotted 140 mm openings grizzly section at its discharge end. This feeder discharges in a jaw crusher. The crusher will be a single toggle jaw crusher with a 150 kW motor and a 1 220 mm by 810 mm (48″ x 32″) feed openings. The crusher size was selected based on its ability to handle the expected rock size as determined by the grizzly openings (500 mm). With this criterion, the rated tonnage capacity of the crusher is much higher than the mill throughput design.

The jaw crusher discharges onto a 914 mm transfer belt conveyor. An electromagnet is located at the head of the transfer belt conveyor to remove tramp metal that might otherwise cause belt damage or block the SAG mill feed chute. The transfer belt conveyor discharges onto a 914 mm stacking belt conveyor carrying the ore from the crushing area to the coarse ore storage bin.

The crushing facilities are equipped with a combined dust cyclone and bag house to minimize the dust exhausted to atmosphere. A vertical sump pump will be provided in a sump in the crusher area basement to collect natural groundwater inflows and wash-down water used for cleanup. The sump water will be pumped to the grinding circuit.

A 1 800 t coarse ore bin will provide a buffer of 24 hours of plant operation between the crusher and feed to the SAG mill. The 9 m (29.5′) diameter by 15 m (49′) high coarse ore bin will be covered and space heaters will be installed to reduce air moisture in the winter. The bin walls will be covered with spray-on insulation to prevent freezing.

A sump pump will collect natural inflows and clean-up water. The drain water will be pumped to the grinding circuit.

6.2.3 Grinding

The ore is withdrawn from the coarse ore storage bin by a 1 000 x 5 000 mm apron feeder. The feeder discharges on a 914 mm belt conveyor carrying the ore to a semi-autogenous (SAG) mill 5 490 mm diameter x 2 135 mm long driven by a variable speed 600 kW motor.

The feed rate to the SAG mill is monitored by a belt scale and is controlled by automatic adjustment of the apron feeder speed. The grinding water addition is also controlled to suit the ore feed rate.

The SAG mill discharges onto a single deck vibrating screen. The screen oversize is returned to the SAG mill feed via two 610 mm oversize return belt conveyors in a row, the last one discharging onto the SAG mill feed belt conveyor. The fist oversize return belt conveyor is equipped with a belt magnet to remove metals and the last one with a belt scale. The screen undersize is directed by gravity to the ball mill discharge pump box. The ball mill is 3 960 mm diameter x 5 500 mm long and is driven by a synchronous 1 500 kW motor.

The screen undersize and the ball mill discharge are pumped by one of two variable speed pumps to a cyclone cluster consisting of 4 cyclones of 380 mm diameter (3 in operation and 1 standby). The cyclone underflow is returned by gravity to the ball mill for further grinding. The cyclone overflow is directed by gravity to the primary thickener. There is an inline sampler installed in the cyclone overflow line. This sampler will move a continuous cyanidation feed sample stream to the on stream analyzer. All grinding is done in lime and cyanide solution.

An inching drive, liner handler and jib crane will be provided to facilitate SAG mill liner replacement. An overhead crane will be installed in the grinding bay to facilitate initial installation of the mills and the handling of used and new liner components during operation. Each ball size will have a dedicated ball bin located outside the mill building and designed to accept direct discharge from the delivery trucks. Small delivery buckets, one for each ball size, will be filled using a magnet. A ball-loading chute will be fitted at the feed end of both the primary grinding mills, to receive the bottom-opening ball bucket and dump the balls into the feed stream of the mill.

A sump pump to reclaim spillage and direct it to the SAG mill discharge pump box will service the overall grinding area.

6.2.4 Primary thickening

The dilute cyclone overflow is gravity fed to a 20 meter diameter thickener. A flocculent solution is added to the thickener feed to promote high underflow densities and clean overflow water. The thickener underflow at a density of 50 % solids is pumped to the primary cyanidation circuit. The thickener overflow proceeds by gravity to a surge tank and then is pumped to the pregnant solution tank.

6.2.5 Gold/Silver cyanidation and solution recovery

The primary cyanidation circuit is composed of two 10 000 mm diameter x 10 500 mm high mechanical agitators in series. The discharge from the second agitator is the feed to the primary filtration circuit which is composed of two 3 660 mm x 4 880 mm long drum filters equipped with snap blow discharges. Filter cakes are repulped to 50 % solids with barren solution and pumped to the secondary cyanidation circuit. Primary filtrates are returned to the primary thickener to be recovered as pregnant solution.

The secondary cyanidation circuit is composed of three 10 000 mm diameter x 10 500 mm high mechanical agitators in series. The discharge of the third agitator feeds the secondary filter circuit composed of two 3 660 mm diameter x 4 880 mm long drum filters. Filter cakes from secondary filters are repulped with water and pumped to the zinc flotation aeration tank. Filtrates from secondary filters are pumped to the mill solution tank and become the primary make-up water for the grinding circuit. All stages of filtration employ a ‘flood’ wash of barren solution for washing the soluble precious metals from the solids. The wash displacement ratio on the filters is 1.5. Each twin drum filter station is serviced by a complete vacuum system including filtrate receivers, filtrate pumps and a wet vacuum pump with silencer.

6.2.6 Gold/Silver solution clarification and precipitation

The precious metals are recovered from solution by zinc precipitation. The pregnant solution is first clarified in a 30 bag clarifier using diatomaceous earth as the filter medium then de-aerated in a vacuum Crowe 2 000 mm diameter x 5 000 mm high tower tank. The zinc dust is added to the precipitation pump suction pipe. The precipitation pump pumps the precipitate slurry to one of two pre-coated 914 mm x 914 mm x 10 plate and frame filter presses. The barren solution going out of the in-service filter-press is directed by gravity to the barren tank. The barren solution is used for washes on primary filter station, cyanide solution make up, grinding solution make up and for slurry pump gland water service everywhere excepted in flotation and dewatering areas. A bleed to tailings via the pyrite tails pump box.

6.2.7 Refining of gold/silver precipitates

The filter press cake is melted to buttons with standard fluxes in an electrical arc tilting smelter furnace. The primary slag will be returned to the grinding ball mill feed after hand crushing. Buttons are remelted with additional borax and bullions are poured into 1 000 and 400 oz. molds. The slag from remelting buttons is remelted in the smelter furnace with the next charge of precipitates.

6.2.8 Zinc flotation

The oxidized cyanide pulp from the 60 minute retention aeration tank is pumped by one of two horizontal slurry pumps to the first of two zinc flotation conditioning tanks. Based on the required conditioning time, an active volume of 27 m^3 will be provided via two tanks. The first will be 3 050 mm in diameter and 3 350 mm high and will be used for lime conditioning. The second will be of same dimensions and will be used for activation with copper sulphate. Both tanks will provide 0.3 m of freeboard.

The last conditioning tank feeds by gravity the combined zinc rougher and scavenger flotation bank (5 x 8.5 m^3 cells + 3 x 8.5 m^3 cells). The scavenger rejects are pumped by one of two pumps to the pyrite (or sulphide residual) flotation conditioning tank. An inline sampler is delivering a continuous scavenger tails sample stream to the on stream analyzer. The scavenger concentrate is pumped by an integrated tank pump to the first cell of the rougher bank for refloating and upgrading. The rougher concentrate is pumped by an integrated tank pump to the feed of the first cleaner flotation bank (2 x 1.4 m^3 cells) which is followed by the cleaner scavenger bank (2 x 1.4 m^3 cells). The cleaner scavenger rejects are pumped to the zinc flotation second conditioning tank. The cleaner scavenger concentrate launder exit is piped back to the first cleaner flotation cell feed. The first cleaner rejects are transferred to the first cell of the cleaner scavenger bank by a cell to cell displacement.

The first cleaner concentrate launder exit is piped back to the first cell of the 2nd cleaner bank (2 x 1.4 m^3 cells). The 2nd cleaner rejects are transferred to the first cell of the first cleaner bank by a cell to cell displacement. The 2nd cleaner concentrate is pumped by an integrated tank pump to the Zn concentrate thickener as a final concentrate.

The rougher flotation feed, the 2nd cleaner concentrate and the cleaner scavenger tails are sampled by inline samplers which deliver continuous sample streams to the on stream analyzer.

6.2.9 Pyrite (or residual sulphide) flotation

The 8-minute retention flotation conditioning tank (3 050 x 3 050 mm) feeds by gravity the rougher flotation bank (10 x 8.5 m^3 cells). The rougher rejects are pumped by one of two pumps to the tailings pond. An inline sampler is delivering a continuous rougher tail sample stream to the on stream analyzer. The rougher concentrate (or the sulphide residual concentrate) is pumped by one of two pumps to the safe sulphide containment cell. An inline sampler is delivering a continuous rougher concentrate sample stream to the on stream analyzer.

6.2.10 On-stream analyzer system

An on-stream-analyzer (OSA) system will be installed within the flotation area to provide near real-time grade information to the operators.

A single analyzer head will be installed to provide sequential assay information on 8 streams in the cyanidation, zinc and pyrite flotation circuits. These assays will provide the operators with on-line information of the final concentrate quality as well as the level of metals remaining in final tailings streams. In combination with intermediate stream information, on-line calculations of achieved recoveries in sub-circuits will be available to the operator to both stabilize and optimize the circuits.

The primary element of the OSA system will be slurry samplers, the most common of which will be in line samplers that provide a representative sample of the slurry stream. Another type is the in-line cutter sampler used when there are headroom restrictions or to prevent mixing of two streams that are to be evaluated independently. For both types of samplers, the sample will be discharged into a vertical froth pump for delivery to the on-stream analyzer's two multiplexers. Some samples will be collected solely for metallurgical accounting of the plant performance.
The multiplexers will present one sample at a time to the OSA measuring window. When selected, the sample volume will be reduced in size by the multiplexer then flow continuously for a preset time in front of the OSA measuring window. A wash cycle will be initiated between samples.

The sample rejects will be handled by any of two four-stream de-multiplexers, which will select the appropriate return vertical pump for receiving the rejected samples that are to be reintroduced at the best location within the process. For this purpose, a set of three return pumps will be required: one returns rejects to Zn conditioner, one to pyrite conditioner and another one to the plant tailings pump box.

A full cycle of sample assays will be completed every 20 to 30 minutes.

6.2.11 Zinc concentrate dewatering and storage

The pulp density of the zinc concentrate is increased to about 65 % by a 9.8 m diameter thickener. The concentrate thickener overflow is pumped by a centrifugal horizontal pump to the neutral plant tailings pump box from the thickener overflow tank.

The thickener underflow will be transported with one of two peristaltic pumps (to prevent dilution of the thickened stream with gland seal water) to a surge tank sized to hold from 12 hours of concentrate production at peak capacity. The agitated storage tank will be 3.6 m in diameter and 4.0 m high (inclusive of a 0.3 m freeboard) with a net volume of 38 m^3. A 10 kW motor will power the holding tank agitator. From this surge tank, the concentrate is pumped by one of two pumps to a horizontal pressure filter. The filter cloth area required to handle the peak zinc concentrate production is 40 m^2 spread over 18 plates. The filtrate is pumped to the thickener feed well. The filter cake is dropped out directly by gravity to a storage pad. From the storage pad, the concentrate is picked up by a front end loader and loaded into trucks for shipping.

6.2.12 Process plant effluents treatment

The last process step consists to float a sulphide (mostly pyrite) concentrate and to produce a flotation tails with a low sulphide content (-0.3 % S). This flotation step is conducted at pH of 6.

There will be a milk of lime addition into the plant tails pump box in order to neutralize this stream. The sulphide concentrate will be pumped to a safe containment cell. On a discontinuous basis, the excess acid water from this safe containment cell will be discharged by gravity into a treatment pond. From there, the acid water will be treated with lime in order to neutralize it and to precipitate its content in heavy metals. There is no test data available to design this effluent treatment process.

6.2.13 Reagent handling systems

6.2.13.1 General

The reagent mixing systems will be designed to handle either bulk delivery by tanker truck, large tote bags or drums. The bulk delivery method will minimize handling requirements and result in less waste from packaging products.

Diluted reagents will be distributed to the addition points via pressurized loops, with controlled pressure drop to compensate for actuated valve cycling. A pressure transmitter will be used to send a feedback signal to a control valve on the main distribution line, and actuates its operation to maintain the desired line pressure.

The reagent addition valves will control the delivery from individual hoses, tied to the main distribution line, of the required dosage of each reagent as per the set point entered by the operator through the control system. Undiluted reagents, for which the addition rate is usually only a few ml/min, will be added via variable speed metering pumps, providing a flow rate proportional to their speed.

A preparation system will be provided for all the solid reagents and the reagents requiring dilution prior to distribution, which includes a mixing tank with agitator and a distribution tank.

When the mixing tank is small, the control unit will be mounted above the corresponding distribution tank for transfer by gravity. For the larger floor mounted mixing tanks, a transfer pump will be used. The transfer pumps will not have a spare, but the distribution pumps will be fitted with backup units, to ensure reagent flow to the circuit in the event of a pump failure. All the transfer and distribution pumps on pressurized loop systems will be seal-less pumps, equipped with magnetic drives, to eliminate joints which would need external water or process solution for lubrication. This feature will prevent undue dilution of the reagent stock and spillage on the pump casing from reagent bleed to seal joints, thereby improving safety around this equipment, and isolate it from continued contact with reagent solution.

A sump pump will be provided in the reagent preparation area.

6.2.13.2 Zinc flotation collector (Cytec 3477)

Total consumption of the zinc collector will be 70 g/t of mill feed.

Cytec 3477 will be delivered to the site in 210 kg drums and transferred, as required, by a drum pump to the 2.2 m^3 distribution tank with a capacity for 14 days of consumption. Cytec 3447 will be distributed to the individual addition points by metering pumps. One standby pump will be provided.

6.2.13.3 Pyrite flotation collector (PAX)

The xanthate PAX will be delivered to the site in drums containing 204 kg of reagent. Five drums will be used to prepare one batch of solution, requiring an equivalent mixing tank volume of 9 m^3 which requires a tank size of 2.25 m in diameter by 2.55 m in height. The distribution tank will be sized to hold a 1.5 batch volume of 13.5 m^3 and dimensions of 2.55 m in diameter by 2.85 m in height.

One batch of PAX solution will last 24 hours, on the basis of the peak consumption of 0.63 kg/t anticipated with the design zinc grade of 3 %.

PAX solution will be distributed to addition points through a pressurized loop. All tanks containing PAX solution will be covered and maintained under negative pressure by the scrubber fan.

6.2.13.4 Zinc flotation circuit frother (MIBC)

Total consumption of the MIBC frother will be 100 g/t of mill feed.

MIBC will be delivered to the site in 210 kg drums and transferred, as required, by a drum pump to the 2.2 m^3 distribution tank with a capacity for 10 days of consumption. MIBC will be distributed to the individual addition points by metering pumps. One standby pump will be provided.

6.2.13.5 Quicklime

Total consumption of quicklime (90 % available CaO basis) will average 1.0 kg/t of mill feed for the cyanidation and flotation requirements.

Quicklime slaking and slaked quicklime distribution facilities have been sized based on a maximum daily consumption.

Quicklime will be delivered at -10 mesh by bulk tanker truck and pneumatically unloaded into a 90 t capacity silo equipped with a bin activator.

Quicklime will be metered out of the silo using a rotary valve, discharging into a screw conveyor. The conveyor will lead to an elevated-temperature slaking unit capable of producing 640 kg/h of slaked lime.

The slaker will produce a 25 % $Ca\,(OH)_2$ slurry which will be pumped by a butyl rubber lined slurry pump, since the lime slurry temperature will approach 90°C, to a slaked lime storage tank and diluted to 12 % $Ca\,(OH)_2$. The storage tank will have adequate capacity for one day of design consumption. Slaked lime slurry will be distributed to process points through a pressurized loop using a horizontal slurry pump. The pressurized distribution loop will span all the mill areas where addition points are located. Individual points will be serviced by on/off pinch valves and their opening cycles will be dictated by pH measurements downstream of the addition point.

A sump pump will be provided in the quicklime slaking and will receive a constant supply of water from the reagent scrubber.

6.2.13.6 Sodium cyanide

Consumption of sodium cyanide for cyanidation will be approximately 1.0 kg/t of mill feed. Some extra cyanide losses are expected from excess barren solution to be bleeded off. Sodium cyanide will be delivered in 1 000 kg capacity bulk bags, and transferred as required to the reagent area for mixing in 4 000 kg batches to produce a 20 % solution. The mixed solution will be transferred to a storage tank, with adequate capacity for 32 hours at peak consumption. Sodium cyanide solution will be distributed to process points using a pressurized loop.

The mix tank and storage tank will be covered and maintained under negative pressure by the scrubber fan. The tank dimensions will be 2.8 m in diameter by 3.1 m in height for the mixing tank and 3.2 m in diameter by 3.5 m in height for the distribution tank.

6.2.13.7 Copper sulphate pentahydrate ($CuSO_4.5H_20$)

Copper sulphate, added to activate the sphalerite in the zinc circuit, will be prepared as a 10 % solution to ensure proper dissolution even with cold water. Copper sulphate is a corrosive material and all the reagent preparation system equipment will therefore be specified accordingly: tanks made of reinforced fiberglass polyester (FRP), the hopper of the mixing tank and the agitator and its shaft made of stainless steel, grade 316. The transfer and distribution pumps will also be made of stainless steel.

The reagent will be received in 1 000 kg bulk bags. One bag will be used to prepare one batch of solution, requiring an equivalent mixing tank volume of 9 m^3 which equates to a tank size of 2.25 m in diameter by 2.55 m in height. The distribution tank will be sized to hold a 1.5 batch volume of 13.5 m^3 and dimensions of 2.55 m in diameter by 2.85 m in height.

One batch of copper sulphate will last over 36 hours, on the basis of an average consumption of 0.39 kg/t anticipated with the design zinc grade of 3 %.

The tanks will be vented to the reagent scrubber.

6.2.13.8 Sulfuric acid

The sulfuric acid is used to adjust the pH level for the flotation of pyrite. The sulfuric acid is received in 30-tonne tank trucks. The trucks are unloaded using 30 psi compressed air, into a 45 tonne storage tank located outdoor. The tank is installed into a confinement area to retain any spillage from the storage tank. The acid solution is pumped to the pyrite flotation conditioner feed by a 1.5 kW dosing pump.

6.2.13.9 Flocculant

One flocculant preparation unit, supplied as a complete package by a specialized manufacturer, will be provided. The package will be capable of preparing one 7.5 kg batch of flocculant every four hours, mixing the reagent with water to form a 0.5 % solution that will be transferred in an agitated distribution tank. At the peak consumption, this quantity will be sufficient to last 4 hours.

The flocculant will be pumped to the primary and zinc concentrate thickeners via progressive cavity pumps to preserve the long molecular chain of the polymer intact. In-line dilution will be provided on the distribution lines to achieve a 0.05 % flocculant solution strength.

The flocculant will be supplied in 22.5 kg bags, added to the holding hopper of the mixing system, metered out with a screw feeder then added through a wetting cone by the suction created by a blower.

6.2.14 Water and mechanical service systems

6.2.14.1 Fresh and fire water

The water requirement for fresh water is estimated at 180 m^3/hr. About 135 m^3/h of water is reclaimed from the plant tailings pond. Vertical pumps are used and the water is pumped into a fresh water tank located near the concentrator. The fresh water tank has a 540 m^3 capacity and is 8.9 m in diameter x 9.2 m high with 0.5 m free board. An allowance of 456 m^3 is dedicated to fire protection water which represents a reserve of two (2) hours at 228 m^3/h (96 500 imp. gals).

The inlets of the distribution pumps are connected in the upper parts of the reservoir to maintain the fire water supply at all times.

The seal water required for the slurry pumps is provided by one of two 38 mm x 25 mm pumps.

The fresh water required for the process itself (flotation area) and the reagent mixing is supplied by one of two 75 mm x 50 mm pumps. The wash water required for the pressure filter is provided by one 75 mm x 50 mm pump.

The fire water pumps (diesel, electric and jockey) have been sized to supply fire water for the whole site.

6.2.14.2 Make-up water

Make-up water requirement for the process plant is estimated at 50 m^3/hr (176.4 imp. gals./min.) and will be provided by one of two 100 mm vertical submersible pumps. Mine dewatering will provide all the water needed.

6.2.14.3 Potable water

The potable water supply and distribution system on site has not been sized nor cost herein because it is described in section 7.5.4.

6.2.14.4 Compressed air

The plant air required for the operation of the pressure filter, the on stream analyzer system, the equipment and pneumatic tools, is provided by one 1 900 m^3/h at 710 kPa oil free and water cooled screw type compressor. One air receiver tank is provided prior to the distribution network.

The instrument air is supplied by a 360 m^3/h at 690 kPa oil free, water cooled reciprocating type compressor. An air dryer and a receiver tank are provided prior to the distribution network.

The compressed air required for the grinding mill clutches is provided by a 360 m^3/h at 690 kPa. Two air receivers (one for each mill) are provided to maintain pressure.

6.2.14.5 Flotation air

The air required for flotation is supplied by two 7 000 m^3/h blowers.

6.2.14.6 Chemical reagents scrubber

A reagents scrubber will be provided to clean the fumes and collect the fugitive dust created when emptying bags or drums of dry reagent into the mixing tanks. The scrubber fan will maintain a slightly negative pressure in all of the reagent tanks to reduce the hazard of inhalation of the reagent fumes or dust.

6.2.14.7 Truck scale

There will be no truck scale but the weight of the outgoing zinc concentrate will be measured by a scale on the loader.

7. INFRASTRUCTURE

The technical valuation made by Roche Ltd in 1999 lists the mine facilities and equipment available. Minor repairs have to be done to the buildings and surface installations. This study stated that:

"Buildings and surface installations at the Abcourt-Barvue mine are in good shape. Only minor modifications are required. In particular, the electrical heating system of the shop and storage areas will be replaced by a propane system. Also, the warehouse will be moved to make room for the engineering office and the main 25 000 V electrical line will have to be checked and relocated."

7.1 Plant and site layout

The main criteria considered during the site layout development for the mill and other surface infrastructures were:

- minimum haulage distance from the pits to ore mixing area, waste dumps and marginal ore stockpile;
- maximize occupancy of land owned by Abcourt;
- minimize disturbance on water courses on the property.

The general site area is relatively flat and easily accessible by the existing road to the property. The total site area to be cleared or levelled for construction is about 200 ha of which 50 ha approximately was already used by past mining activities (mainly pit, mine site and waste dump).

The tailings pond, the piles/stockpiles/dumps and the Abcourt pits footprint areas are covered by trees. Although most of the commercial lumber has recently been cut, tree-cutting is required over some of the surface areas. Topsoil thickness varies from 0.5 to 3 metres while overburden thickness ranges from 2 to 10 metres.

7.2 Roads

7.2.1 Access road

The existing access road will be used for the present project and necessitates almost no improvements.

7.2.2 Site roads

Site roads will be required to access different parts of the property, namely the Abcourt pit to the western end of the property, the topsoil pile, the overburden pile, the tailings pond, the waste rock dumps and the exit of the Barvue and the other Abcourt pits. The use of existing site roads will be maximized. The site roads will be constructed with waste rock from the existing waste rock dump and with waste rock coming from Barvue and Abcourt pits' excavation. Inert waste rock will be used for this purpose.

7.2.3 Site gate and fencing

The main gate will be located on the existing access road right beside the existing fence. It will consist of an access-code operated gate completed with a phone booth.

Fencing will be restricted to the main gate, the main sub-station, the propane tanks and the explosive storage areas. Only authorized employees will have access to the explosive magazine area which will be enclosed by a fence with a locked gate.

7.2.4 Parking area

The existing parking area will be used.

7.3 Crushing facility and processing plant layout

The surface site plan has been designed to maximize the use of the existing surface infrastructures, namely in this case the foundations of the former mill. The concentrator and the crushing facility have been located accordingly (Appendix 3).

7.3.1 Crusher

The primary crusher area is of a compact design. The site for the crusher installation was selected to fit with the present concentrator design on existing mill foundations and the location of the ore mixing area needed to regularise head grades into the process.

7.3.2 Concentrator

The concentrator building is a conventional ore processing type building (Appendix 3). The mill footprint is 3 100 m^2. The concentrator building houses the grinding, cyanidation, refining, flotation (zinc and pyrite), thickening, reagent and filtration areas, the compressors, the electrical room, the concentrate loadout (designed for a front-end loader and a drive through for concentrate trucks) as well as tailings pumps and pump boxes.

Offices, lunchroom and washrooms are provided for on the second floor of the men drying and storage.

7.3.3 Concentrate weighing system

Loading of the concentrate trucks will be done by the contractor retained for concentrate transportation. The loader will have to be equipped with a scale for weighing purposes. In this manner, no truck scale will be required on site.

7.4 Service building

The existing service building is in good condition. Only minor improvement and reshaping will be required for the present project. In particular, the electrical heating system of the shop and storage areas will be replaced by a propane system. Also, the warehouse will be moved to make room for the engineering office. The service building will contain the mine department office (staff and technical service), the mine dry, the mine warehouse and a mechanical workshop for light vehicles, drills and other equipment of limited size. Heavy equipment such as trucks, front-end loaders and hydraulic shovel will receive maintenance and repair services in a new shop on an existing concrete pad near the service building. This new shop will only accommodate one machine at a time.

All new open pit equipment will be bought with a maintenance contract by the suppliers. The maintenance and normal repair costs are included in equipment hourly operating cost provided by suppliers.

7.5 Services

7.5.1 Fuel storage and refuelling station

This installation exists and is in good condition. It is adjacent to the new mill location.

7.5.2 Explosives storage and plant

The explosive supplier will erect and operate a mixing plant for bulk explosive emulsion on the property for both open pits and underground production blast holes.

Cartridge explosives for pre-shearing and secondary blasting and explosive accessories will be stored in separate magazines (two) provided and installed by the supplier under a rental contract.

7.5.3 Water systems

The on-site water system network includes: the fresh water intake system, the reclaim water system, the tailing system, the mine dewatering system, the mill water supply system, the underground mine water supply system and the water treatment plant.

The current on-site network of drainage ditches and ponds is well developed but some portions will be modified and/or relocated to fit the forthcoming needs of the mine-mill complex. However, the use of the existing network will be maximized for the present project in order to minimize the environmental impacts.

The service building is connected to an artesian well that was and is currently used to provide fresh water to sustain the needs for on-site employees and staff. Its capacity will be checked in terms of the future requirements.

The reclaim water and tailings systems will be developed between the mill and the tailings pond for tailings transportation and water recirculation to the mill.

The mine dewatering system will provide fresh water to the mill or will be connected to the water treatment plant for water treatment prior to its discharge into a drainage ditch feeding the settling pond at the northwest limit of the tailings pond (see figure 8.1.1).

The mill water for initial start-up will be supplied from open pit mine water. Once in production, process water in the mill will be supplied from the tailings pond and, if needed, from open pit and/or underground mine.

The industrial water needed in the underground mine, when development begins at year 6 of production schedule, will be supplied from the existing settling pond located west of the mill as done in the past.

7.5.4 Potable water

The artesian well presently servicing the on-site service building is only to provide fresh water for all needs but drinking water. The latter will be provided by bottled water.

7.5.5 Heating, ventilation and air conditioning of surface installations

Exhaust fans will be installed for each building to ensure proper air change and temperature control in the summer. In most cases, propane heaters will be used to heat the buildings.

7.5.6 Underground mine services

During the last production period (1985 to 1990), the Barvue mine was connected to the Abcourt shaft with an internal ramp. The shaft was used for ventilation and for air, water and electrical services

Before any pre-production development of underground mine starts, existing ancillary services, removed from the top of the existing Abcourt shaft to allow mining of the upper part of the ore zone by open pit, will have to be re-installed. They will be relocated right beside the entrances of the three main underground declines, located at the bottoms of the East Abcourt pit, after surface mining is completed. This includes industrial water lines, mine water lines, compressed air lines, electrical lines, electrical substations, the main ventilation system and the air heating system by propane direct fired units. Also, permanent pumping stations will have to be installed at the bottom of stopes. Most of this equipment and accessories/supplies are already owned by Abcourt and are in good condition.

7.6 Electrical power

Due to past mining activities on the property, a 25 kV electrical line with transformers and electrical distribution network are still available and operational on site. The existing service building is currently supplied with hydro power. The main 25 kV electrical line will have to be checked and relocated where required.

The power demand for the plant is estimated to 5.1 MW as per the equipment list. The estimate assumes an utilisation factor of 0.65.

The main electrical room, located near the main loads, will contain the following equipment: four (4) power transformers including protections and wiring; six (6) breakers 600 volts 400-800 amp; six (6) Motor Control Centres (MCC) for the concentrator motors; enclosure cubicle with the main breaker, four (4) switchgears, bus and wiring.

7.7 Waste management

7.7.1 Sanitary waste water

A sanitary waste water system already exists on the mining property and is operational. It consists of a collection network of underground piping, a concrete septic tank located near the service building and a drainfield. The sanitary water from the mill building will be connected to the existing sanitary waste water system.

7.7.2 Domestic garbage disposal

Domestic garbage will be handled by a local contractor and transferred to the municipal disposal site.

7.7.3 Other wastes

Metal scrap will be sold to a scrap dealer and industrial waste/hazardous materials will be managed by an approved contractor.

7.8 Telecommunications and computers

7.8.1 Telecommunications

The service building is already connected to a Telebec telephone line. On-site reticulation will be needed to connect the mill building. Portable phones and a few individual lines will be available. This would allow employees to have access to telephone and internet services.

7.8.2 Computers

The computer systems of the mine will include a network platform and Windows XP-based PC's. The software that will be available on-site would comprise an industry standard office applications package including accounting and stores, and technical systems for mine planning and production control.

8. OVERBURDEN, MINE ROCK AND TAILINGS MANAGEMENT

Once again, the main criteria considered during the design of all of the accumulation piles were minimum haulage distance, maximum occupancy of the mining property and minimum encroachment and disturbance on water courses. Figure 8.1.1 presents the general arrangement of all the surface infrastructures.

8.1 Topsoil and overdurden piles

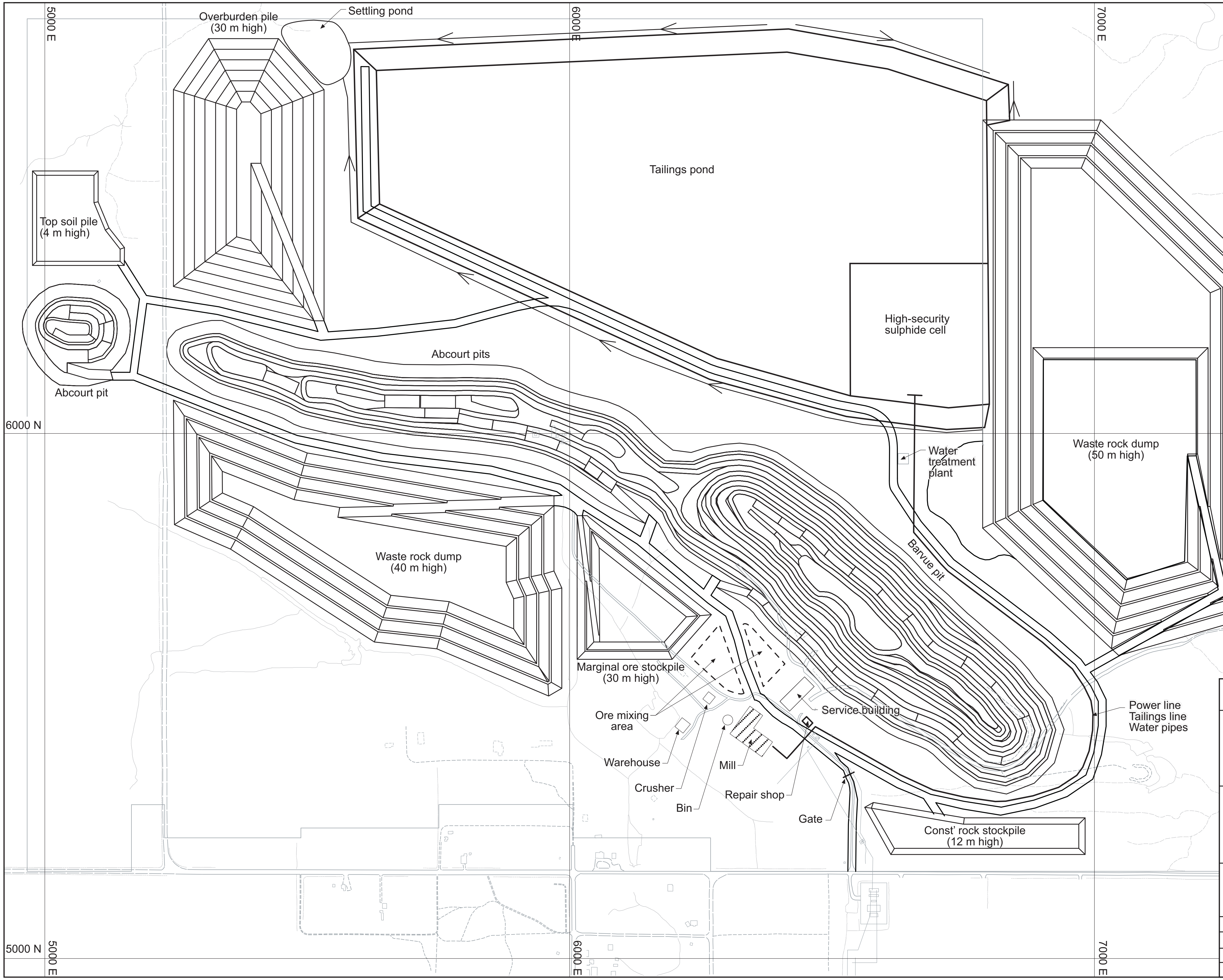
Topsoil and overburden piles will be contiguous at the north-western end of the Abcourt-Barvue property. The topsoil will originate from the stripping of the open pit footprints and the underlying southeast portion of the tailings pond (under the sulphide impermeable cell), for a total of approximately 200 000 m³ (0.5-m average thickness over pits and 2.5-m thickness under the sulphide cell). The overburden will only originate from the open pit footprints stripping for a total of about 1.85 Mm³ (9.5-m and 4.5-m average thicknesses for Abcourt and Barvue pits respectively).

The topsoil storage area has a capacity of about 200 000 m³ of material. The topsoil pile is planned to have a maximum height of 9 m and to be developed in 3-m lifts without setbacks or berms. The topsoil will be dumped at the angle of repose at three different elevations to achieve an overall angle of 26.5°(2H:1V). A swell factor of 20 % was used in designing the topsoil pile which will be progressively and completely reclaimed for on-going rehabilitation and post-production closure activities.

The adjacent overburden storage area has a capacity of 2 Mm³ of material. The overburden pile was designed with a swell factor of 10 % to a maximum height of 30 m and a 10 %-gradient ramp. The pile will be developed in 5-m lifts and once again without setbacks. The overburden will be dumped at six different elevations and at the angle of repose which will be flattened down with a bulldozer (one lift at a time) to achieve an overall angle of 14° (4H:1V), for stability and rehabilitation purposes.

8.2 Marginal ore stockpile

The marginal ore is mineralized rock coming from open pits with grade lower than 2.55 % Zn equivalent for the Barvue pit and 2.4 % Zn equivalent for the Abcourt pits. Over the 10-year life span of the project, a total of 1.15 Mt of marginal ore will be produced at average grades of 17.65 g/t Ag and 1.58 % Zn or 1.95 % Zn equivalent, and stockpiled. For the waste to ore ratio during production, the marginal ore is considered as waste but for environmental management purposes, it is considered as ore.



ABCOURT MINES INC.

Figure 8.1.1

General arrangement

GENIVAR

		Date
Drawn by	JMT	2007/02/14
Checked by	ML	2007/02/14
Approved by	ML	2007/02/14
GENIVAR file	Q106150_Fig_8-1-1_070214.fh10	

0 50 100 m

Source :
Autocad file : planc.dxf

Version	Modification	Date

Q106150

The marginal ore storage area has a capacity of 1.7 Mt giving an additional capacity if required. The marginal ore stockpile was designed with a swell factor of 60 % to a maximum height of 30 m and a 12 %-gradient ramp. The pile will be developed in 15-m lifts with 3-m setbacks. Typical angle of repose for such material varies from 30° to 50°. This material will be dumped at the angle of repose (±45°) at two different elevations to achieve an overall angle of 42° (1.1H:1V) including setbacks.

8.3 Waste rock dumps and stockpile

The waste rock is mineral material coming from open pits and without Zn-Ag bearing mineralization. It is assumed that waste rock of the Abcourt-Barvue project is non-acid generating or inert. It will be utilized for construction purposes at surface (pads, roads, dikes, embankments, fill) for this reason and as backfill in underground stopes when required. During the preproduction and the 10-year production phases, 33.15 Mt will be excavated from open pits. Waste rock will be disposed of onto three different storage areas.

Two waste rock dumps and a construction rock stockpile will be built for the project requirements. The construction rock stockpile will be composed of competent waste rock (gabbroic sill and marker tuff in this case) with the best physical characteristics for construction purposes. The waste dumps will contain the remainder of waste rock.

The construction rock storage area has a 500 000-t capacity. The construction rock stockpile was designed with a swell factor of 60 % to a maximum height of 12 m and a 12 %-gradient ramp. Once again, typical angle of repose for such material varies from 45° to 50°. Waste rock will be dumped at the angle of repose at one elevation to achieve an overall angle of 45° (1H:1V).

The waste rock dumps will be the largest on-site rock accumulation piles of the Abcourt-Barvue project. Specific design criteria were used for physical stability purposes. Both waste rock dumps were designed with a swell factor of 60 % to maximum heights of 40 m (west dump) and 50 m (east dump) and 10 %-gradient ramps. Once again, typical angle of repose for such material is about 45°. This time, an inter-berms typical slope of 30° (1.75H:1V) was retained to facilitate rehabilitation.

As designed, the total capacity of both waste rock dumps together is 32.65 Mt. The east waste rock dump with a capacity of 23.3 Mt will only contain waste rock from the Barvue pit while the west dump with a 9.35-Mt capacity will be composed of both Barvue and Abcourt pits waste rock. This material will be placed at a 30°-slope between consecutive 6.5-m setbacks and the waste rock dumps will be developed in 10-m lifts to achieve an overall angle of 23.5° (2.3H:1V).

8.4 Tailings pond

8.4.1 Location and description of tailings pond site

An evaluation of potential sites to locate the tailings pond was done by Golder in 2006 at the request of Abcourt's management (Golder, 2007). The proposed site is the only one out of five under study that respects the appropriate discriminating criteria following a previous analysis (Golder, 1999) and numerous meetings and discussions with Golder's and Government representatives.

The selected site complies with Federal, Provincial and Municipal by-laws. It is compliant namely with agricultural zoning by-laws and respects all Barraute Municipality's by-laws that were recently revised.

The tailings pond site covers an area of about 60 hectares located in range VII of Barraute Township, in the northern part of lots 26 to 30. Such an area is justified by safety factors involving low dikes or dams taking into account the relatively soft nature of the soil and the absence of constraints in the selected area. Specifically, the tailings pond area and limits are ruled by the actual surface rights, the local topography, the open pit proximity and its proposed extent, the underlying overburden nature, the distance to existing roads and buildings and the optimal distance to the milling complex.

The tailings pond covers an area averaging about 1 155 m east-west per 515 m north-south. Its total area is 59.4 ha and its perimeter is 3.3 km. The safety margins between the dikes' crest and the open pit future limits are of at least 130 m.

The existing site is a poorly drained plateau located at an elevation between 10 000 and 10 002 m. This zone is corresponding to a local high point resulting in a poorly developed drainage system both to the south and to the north. The slopes are locally a little steeper to the northwest and to the northeast where higher dikes will be needed. Most of the area is covered with a relatively thin layer of black earth and is generally forested although most of the commercial lumber was recently cut. A small north-south access road crosses the western portion of the proposed tailings pond leading towards a small former sand pit. Two strips of higher ground are visible to the northeast where some rock outcrops are seen.

8.4.2 Geotechnical and hydrogeological tests

At the Abcourt's request, Golder performed a series of geotechnical holes and tests on site in October 2006. A total of 8 holes were then drilled along the proposed axis of the dikes and in the center of the future tailings pond. The detailed results are stated in Golder's report; the depth of the holes varies between 3 and 17 metres. Samples were taken for further testing and the stratigraphy was described by Golder's on-site geologist.

Instrumentation was installed in 6 of the 8 holes to monitor the hydrogeological characteristics of the area to be covered by the tailings pond. On-site piezometers measurements were taken to evaluate the horizontal permeability of the actual soils.

A series of different tests were performed on the samples sent to the laboratory, namely the water content, the grain size analysis, the sedimentometry, the consolidation, the Atterberg limits and the permeability.

8.4.3 Tests results

The on-site drilling and laboratory testing gave an accurate picture of the actual characteristics of the underlying overburden where the tailings pond is proposed to be built. Top soil was encountered in all holes averaging between 0.03 and 0.1 m. The thickness of this material reached 0.6 m and even 2.4 m in one hole. In all holes but one, the top soil is directly in contact with the clay horizon which thickness varies from 3.2 to 8.3 m. A silty horizon is then underlying the clay to a depth varying between 3.8 and 13.1 m. A final sandy to sand-gravel horizon forms the basement of the overburden layer encountered in the drilling campaign.

The water table was located at depth varying between 1.1 and 3.1 m underneath the surface. The underground drainage is apparently similar to the surface drainage being essentially radial from the watershed line. The in-situ measured horizontal permeability gives values in the range of 3×10^{-7} to 2×10^{-5} cm/s for clay and silty horizons. Since we have to know the vertical permeability of the clay horizon to evaluate the underground drainage potential of the tailings pond, laboratory tests were performed. Two samples were submitted to a series of 2 tests to determine their permeability under natural conditions and under confined pressures corresponding to those that the tailings pond will generate. The tests gave results such as 3×10^{-7} to 2×10^{-6} cm/s for the natural conditions, and 5×10^{-8} to 1×10^{-7} cm/s for the tailings operation conditions.

8.4.4 Tailings characteristics

A series of metallurgical tests were performed by Laboratoires LTM Inc., and the results of those show that the final products coming out of the mill complex will consist in Ag-Au ingots, Zn-Ag concentrate, pyrite concentrate and desulphurized tailings containing 0.24 % S. The pyrite concentrate represents 12 to 15 % of the total tailings.

The conception of the tailings pond is based on a series of parameters such as the total amount of material to handle. The tailings pond is considered to be able to safely contain about 6.5 Mt of tailings, of which 0.75 Mt will be a pyrite concentrate to be stored in a high-security cell, and the bulk of it (about 5.75 Mt) being neutral (generating no lixiviate, no acid and no cyanide). The high-security cell will eventually be closed during the production period and covered with the bulk of the neutral tailings material, thus facilitating the final reclamation of the area.

Starting in year 6, the volume of pyrite concentrate disposed of in the high-security cell will be minimized as most of it will be used in paste backfill for the underground operation. This approach will tend to diminish both the risks and costs of managing those residues on surface.

The tailings pond will be built in 3 steps in order to minimize the up-front capital costs:

- a first phase will see the starting dikes being built to the 10 003 m elevation including the high-security cell;
- a second phase will see the dikes raised 5.5 m after 2 years of production;
- the final third phase will consist in raising the central portion of the proposed tailings pond for another 5.5 m.

The respective cumulative total capacity of the tailings pond for each phase will be 1.1, 4.75 and 7 Mt.

It is expected that essentially all material to be used in the construction of the dikes for the tailings pond are present on site (or at very low trucking distances). For instance, clay is known to be present all across the area and non-acid generating waste rock is also stockpiled close to the tailings pond area. Additional waste will be produced by the open pit operations.

Golder's study mentions that considering the hydrogeological results for the clay layer's permeability, the tailings pond would match the low permeability criteria stated in the Government's Directive 019.

Dikes stability analysis were also performed by Golder's staff to qualify the safety of the construction concept. As part of this analysis, dike stability was confirmed under static and pseudo-static (seism) conditions. Soil liquefaction seems not problematical with a realistic design seism of 6.5 maximum magnitude. Future studies will have to confirm the appropriateness of the hypothesis. The results demonstrate the need to build a key against the shearing of the clay layer and moreover, the need for bulky berms to stabilize the dikes both to the east and west limits of the pond. These two areas are characterized by the adjoining presence of planned waste and overburden stockpiles that could easily serve as berms (as long as berms are essentially counterweights to stop any movement from occurring).

The typical dike structure will be constructed with permeable waste rock for the external part of it. Internally, fine waste rock will form the core of the dike with a transition sand layer covered by compacted crust clay over a 3-m thickness. The crest of the dike will be covered with sand and gravel. Since waste rock berms will essentially surround the whole tailings pond, access road will be built on top of the berms.

9. SITE WATER MANAGEMENT

Water that needs to be treated on site will be processed through a water treatment plant to be built close to the south-eastern section of the tailings pond in lot 30. The core of this plant will be the former water treatment plant that is currently installed on the Louvicourt Mine tailings site and that will be purchased and moved onto the Barraute site in the next few months.

9.1 Dewatering of existing pit

The actual open pit contains about 550 M imperial gallons of water. The recent characterisation of the water quality showed that the zinc content was too high when compared to the limits stated by Governmental authorities, thus a treatment will have to reduce the zinc concentration to acceptable limits.

During the dewatering process, the water will pass through the water treatment plant where the initial pH will be raised from its average 7.8 to 9.2 to precipitate zinc. The successive addition of a flocculent and of a coagulant will favour the settling of the zinc precipitate. This process was tested previously at the Quebec CRM to determine the proper parameters of both time and chemicals to optimize the process. These tests have indicated that lime, the coagulant and the flocculent proportions were 0.5 g/l, 0.25 mg/l and 0.25 mg/l respectively. The final settling will be performed in the settling pond northwest of the tailings pond. Upon final assaying, the good quality water will be discharged into the environment.

9.2 Mine water

During the mine operations, the mine water, either from underground work or open pits, will be pumped to the mill as fresh water or to the water treatment plant and then, to the tailings pond or in the ditch leading to the settling pond (northwest of the tailings pond) where solids will have time to precipitate.

9.3 Waste stockpiles runoff

The waste rock produced in the past was not acid generating. It is then not expected that the "new" waste rocks will generate any acid. Moreover, the recent rise in the price of zinc will allow the profitable exploitation of lower grade ore that was left in waste rock before, thus reducing the available contained sulphides to generate any acid from waste rock. For these reasons, it will not be necessary to collect and process the drainage water from the waste rock stockpiles. However, a follow-up will be made to insure that expectations are proving themselves true.

9.4 Ore stockpiles runoff

Water runoff from both ore and low grade ore stockpiles will be caught in ditches and directed to a sump where it will be pumped to the tailings box in the mill to receive the same treatment as in the water treatment plant. Then, the treated water will be mixed with the process residues and pumped into the tailings pond for further sedimentation.

9.5 Tailings pond

The tailings pond has been designed to receive a maximum precipitation of 100 mm within 24 hours (once every 100 years) equal to 59 400 m^3, to which a daily production of 2 085 m^3 with a 10-day retention period, for a total of 20 850 m^3, is added. The total potential volume of water to be managed within the tailings pond ends up at 80 250 m^3. The capacity of the tailings pond to contain water in the first phases of its time life is in the order of 160 000 m^3, which is twice the needed volume.

The operation of the tailings pond and the management of the water runoff should not cause any problem even considering the daily probable maximum precipitation. When needed, the water level within the tailings pond could be lowered using a floating barge carrying a pump.

The tailings pond concept is to keep the water pond in the middle of the tailings pond in order to minimize the hydraulic stress on the dikes. Finally, since the overall tailings pond system is located on top of the local topography, small ditches will suffice to control the external water flow.

Water in the tailings pond will generally be reused into the mill process. The overflow from the tailings pond during the production phase will be directed to the water treatment plant if required and discharged into a ditch leading to the settling pond located at the northwest limit of the tailings pond; otherwise, it will be discharged into the ditch leading to the settling pond without any treatment.

10. ENVIRONMENT

The Abcourt-Barvue project is located on a site that has already been used for mining activities. The environmental management proposed herein for the reopening of the open pit and underground mines was developed in accordance with all related regulations.

All required characterizations, verifications and studies regarding such a mining project will be integrated in the scope of work to be developed for the permitting phase of the project.

10.1 Abcourt-Barvue site characterization

No exhaustive characterization of the project mine site was performed for the present study. Prior to begin the development and construction phases of the project, it is recommended to proceed with surface water, groundwater and sediment sampling and analyzing, scientific fishing and fauna/flora surveys to establish the pre-project environmental conditions. Air and noise studies will have to be completed as part of the permitting phase of the project.

However, based on our regional environment knowledge and on the fact that the site has already been used for past mining activities, thus permits were received in the past for such activities, it is assumed that no exceptional characteristics regarding either water and sediments quality or fauna/flora rare or endangered species are present on the Abourt-Barvue property.

In the surrounding area, a yellow walleye spawning area is located nearby in the Laflamme river, upstream from the mine site (Golder, 1999).

10.2 Waste rock, ore and tailings characterization

The ore and the marginal ore contains most of the sulphur present in local rocks. They are considered as potentially acid generating materials and will be managed accordingly. Given that waste rock contains almost no sulphur, it is considered as non-acid generating or inert material which is confirmed by tests done on the existing waste stockpile.

A pyrite flotation circuit is part of the ore milling process to reduce the sulphur content under 0.3 % S in the tailings (0.24 % S in fact). The tailings are then inert material regarding acid generating potential. The pyrite concentrate is an acid generating residue because of its sulphur content and it will be managed accordingly.

Laboratory tests have confirmed that tailings will not contain any cyanide since its degradation occurs in the milling process inside the concentrator. In the mill, all the cyanides are destroyed by oxidation before the ore is treated by flotation. If they were not, the zinc would not float.

10.3 Environmental management

10.3.1 Waste rock

As soon as the presence of zinc and silver mineralization was observed or detected in drill hole cores, the material was considered as ore or marginal ore depending on its zinc equivalent content and the ore body was modeled accordingly. With the cut-offs used, the limits of the zone are quite sharp. There is practically no sulphides (pyrite) outside the ore outlines, thus rocks surrounding the ore body is considered as inert or non-acid generating.

No particular criterion was applied to waste rock dump design in regards to acid generating potential and metals content.

10.3.2 Ore

Ore and marginal ore will be stored in dedicated areas in two different stockpiles. The ore will be stockpiled for a very short period of time prior to its mixing and reclaiming and will be transferred to the primary crusher with a front-end loader. The marginal ore stockpile will be reclaimed at the end of the operations and the material will be treated in the concentrator.

During production period, the ditches around the ore mixing area and the marginal ore stockpiles will collect surface runoff from stockpiles and as mentioned before, the collected water will be directed to a sump where it will be pumped to the tailings box in the mill. There, a water treatment for pH control and metals precipitation purposes will be performed (addition of lime, coagulant and flocculent in the same proportions as in the water treatment plant) prior to its discharge into the tailings pond for sedimentation.

10.3.3 Pits and underground mine water

It is believed that pit and underground mine water will not be acid but will contain zinc in solution or suspension during both preproduction and production dewatering. For the same reasons as for collected ore stockpile runoffs, water will be pumped directly to the water treatment plant to be treated prior to its discharge into the ditch leading to the northwest settling pond, or used as fresh water in the mill.

10.3.4 Pyrite concentrate

The pyrite flotation will produce a pyrite concentrate and the remaining tailings will have 0.24 % S; tailings with less than 0.3 % S are considered as non-acid generating. On the other hand, the pyrite concentrate will be a potentially acid generating material but in a much smaller quantity than if the pyrite flotation had not been integrated in the ore milling process.

The pyrite concentrate will be disposed of in a high-security and impermeable sulphide cell, eventually encapsulated under other residues as the tailings pond will be developed, thus ensuring no overall potential for acid generation.

10.3.5 Tailings pond

As the water in the tailings pond will mostly be reused into the mill process, little overflow from the tailings pond will occur. When this happens and if required, the overflow will be directed to the water treatment plant, ditch and settling pond system; otherwise, the water treatment plant component of the system will be by-passed.

10.3.6 Mining traffic

Water will be sprayed on mine roads to prevent a potential increase in the level of dust particles caused by road traffic. Thus, dust control with water application will minimize the impact on air quality associated with mine traffic.

10.4 Conceptual closure and rehabilitation plan

10.4.1 Overview

The closure objectives of the conceptual rehabilitation plan are as follows:

- to dismantle and remove buildings and other infrastructures;
- to return the disturbed areas to pre-project land uses when feasible;
- to leave sites affected by the project in physically safe conditions (berms, fences and signs);
- to develop mine pit walls and dumps/stockpiles with respect to geotechnical stability objectives that will prevail during and after operations;
- to limit the potential for future erosion;

- to establish sustainable vegetation for long-term stabilisation of the land surface and for aesthetic purposes;
- to eliminate the need for long term care.

10.4.2 Buildings and infrastructures

Decommissioning, dismantling and demolition of all buildings and infrastructures, including the mill, will begin as soon as possible after the end of operations with the objective to facilitate salvage of equipment/materials, if no further potential use is indicated. These activities include the removal of constructed facilities as well as the shaping, rehabilitation and revegetation of disturbed areas. Potentially contaminated areas will be inspected and tested at closure to check that concentrations are at or below levels that would adversely impact soil, groundwater or vegetation. Where limits are exceeded, the area affected will be investigated and appropriately remedied, or the affected materials will be removed from the site to an approved disposal site. This will be followed by a period of monitoring and, if required, maintenance.

10.4.3 Open pits

At the end of production, approximately 37 ha of open-pit surfaces will have been mined. The open pits will progressively flood after the end of the underground operations. The existing Barvue pit overflow meets the environmental standards most of the time. When not, the discharge point is blocked and no more water flows into the environment. The occasional presence of zinc in water results from the presence of Zn-Ag mineralization exposed on about 50 % of the pit walls.

In the present design of open pits, final pit walls in the Zn-Ag bearing mineralization are minimal. Only 10-15 % of the overall perimeter of the ultimate pit walls will be located in that mineralization; the remaining walls will be located in waste rocks with very low concentrations or traces of metals. This will reduce to a minimum the input of dissolved and suspended metals in water. For this reason, it is believed that the metal content of the open pits' overflow should be at or below levels that would adversely impact the environment.

10.4.4 Underground openings

The main declines entrance will be blocked up with a pile of waste rock prior to eventually being flooded as the open pits will flood.

Concrete slabs will be installed on the old shaft and ventilation raises needed for underground mining.

10.4.5 Topsoil management

Sufficient soils will be available to re-establish vegetation at the mine site after the end of operations.

The total quantity of stripped topsoil over mining areas and under the impermeable sulphide cell in the tailings pond is estimated at 200 000 m^3, based on thicknesses of 0.5 m and 2.5 m over pits and under sulphide cell respectively. Approximately 30 cm of topsoil will be placed on areas to be rehabilitated, with a minor portion being placed progressively and the rest stockpiled for later rehabilitation use.

10.4.6 Dumps and piles

The 6.5-m wide horizontal setbacks or berms left between the crest of lower construction bench and the toe of the next 10-m lift of the waste rock dumps will be progressively revegetated as soon as they reach their final configuration. Trees will be planted on added overburden or topsoil for stabilization and aesthetic purposes.

As for the waste dumps, the overburden pile will be progressively revegetated, but by hydroseeding directly on 4H:1V slope for long-term stabilization and aesthetics purposes once again.

At or after closure, marginal ore, ore and construction rock stockpiles together with the topsoil pile will have been completely reclaimed or used. Ores will have been treated at the mill while construction rock and topsoil will have been used for construction and rehabilitation purposes respectively.

The footprints of ore stockpiles will be inspected and tested at closure to check that concentrations are at or below levels that would adversely impact soil, groundwater or vegetation. Where limits are exceeded, the area affected will be investigated and appropriately remedied, or the affected materials will be removed from the site to an approved disposal site.

10.4.7 Tailings pond

The sulphur content of the milling residues is reduced to 0.24 % S since the pyrite flotation was proven effective and has been integrated in the ore treatment process. For this reason, the processing residues are considered non-acid generating and the tailings pond will only be revegetated at closure by hydroseeding after overburden or topsoil addition, reshaping and embankment dam reinforcement to minimize the effect of surface water erosion, if required.

The pyrite concentrate will be disposed of in a high-security and impermeable sulphide cell encapsulated inside the tailings pond thus ensuring no potential for acid generating or used underground as paste or cemented backfill.

10.4.8 Water management installations

The water treatment plant required during preproduction and production periods will be dismantled after post-closure activities at the final release of the mine site from governmental authorities. At that time, no more water will be pumped from the open pits or the underground mine. Furthermore, water flowing out the non-acid generating tailings pond and surface runoff from ore stockpiles area, which will have been completely reclaimed and cleaned-up, will not need anymore water treatment prior to their release into the environment

Ditches around the ore mixing area, the marginal ore stockpiles and the tailings pond discharge / water treatment plant / northwest settling pond will be cleaned up and sediment removed will be transported to a suitable disposal site.

10.4.9 Mining equipment

Primary mining equipment, light vehicle, support equipment, pumps, lights and air fans from both open pits and underground mines will be sold on used market or sold for scrap after proper emptying and management of all fluids.

10.4.10 Chemicals, petroleum products and hazardous materials

All consumables labelled as chemicals, petroleum products and hazardous materials will be managed at the end of operations in order to be completely used. The remaining, if any, will be returned to suppliers.

10.4.11 Solid wastes and contaminated soils and materials

Soils contaminated with chemicals, petroleum products, hazardous materials will be inspected and tested at closure to check if concentrations are at or below levels that would adversely impact soil, groundwater or vegetation. Where limits are exceeded, the area affected will be investigated and appropriately remedied, or the affected materials will be removed from the site to an approved disposal site.

Materials such as concrete, pipes, tanks and other pieces of equipment contaminated with the same products as above will be cleaned up safely and recycled or disposed of as solid wastes or transferred to an approved disposal site if not cleanable.

10.4.12 Site safety

Inadvertent access by the general public to mined areas will be addressed by construction of rock barriers and installation of fences and warning signs.

10.4.13 Environmental controls

At closure and during rehabilitation work, environmental controls will be the same as those applied during the production period.

10.4.14 Post closure

The main post-closure activity involves monitoring water quality, revegetation progress and stability of pit walls and dumps/stockpiles. This will necessitate site and drainage maintenance and may include remedial work. Post-closure monitoring will continue until all closure objectives have been met, which could take up to 10 years.

11. CAPITAL EXPENDITURES

11.1 Summary

The total capital expenditure required for the Abcourt-Barvue project to plant start-up is estimated at 67.88 M$, plus a working capital of 3.38 M$ which will be entirely recovered at the end of the project. These costs are in third-quarter 2006 Canadian dollars, excluding taxes and duties. The estimate is summarized in the Table 11.1.1 and further detailed in this section. The totals for the mine, process plant and infrastructure cost centres contain both direct and indirect costs.

Table 11.1.1 Capital cost summary.

	$'000
Mine	
Equipment	6 580
Facilities	693
Overburden stripping	4 059
Waste stripping	8 480
Mine total	**19 812**
Process plant	
Equipment and installation	21 501
General and services	3 213
General steel works	5 920
General concrete works	3 423
Sub-total	34 057
EPCM (15 %)	5 109
Contingencies (15 %)	5 109
Process plant total	**44 275**
Infrastructure	
Road network	included
Site services	380
Power supply	25
Tailings pond	1 257
Water treatment plant	200
Infrastructure total	**1 862**
Owner's costs	**1 930**
Preproduction total	**67 879**
Working capital	**3 376**
On-going investment	**24 418**

11.1.1 Basis of estimate

Mine capital costs were estimated by GENIVAR with some assistance from specialized suppliers. Budget quotations were obtained for major mining equipment. The preproduction stripping costs were based on mine operating costs as described later.

Process plant capital costs were estimated by BUMIGEME and RWJ Consultants Miniers. Equipment costs were determined from vendor budget quotations supplemented by in-house data for similar projects. Building costs are based on a combination of direct cost factors and area/volume unit cost factors applied to the mill footprint. Other direct costs are factored on equipment costs using factors derived from historical projects. Indirect costs are also factored from historical data.

Given that some infrastructures are available on the mine site, their costs were developed on the basis of reshaping and/or improving them. For the other infrastructures, budget quotations from suppliers were obtained or Roche's cost figures in its 1999 Technical Validation Report were updated to 2006.

11.2 Estimate details

11.2.1 Mine capital cost estimate

Details of the mine capital cost estimate totalling 19.82 M$ are provided in Table 11.2.1. The production and support equipment unit costs are suppliers' quotations. It was estimated that used or rebuilt haul trucks, grader and bulldozers will be purchased at approximately 75 % of the price of new machines. The choice of used equipment for bulldozers and graders is based on the fact that they will not be utilized on a continuous basis giving time for maintenance and repairs. For haul trucks, there will always be at least one backup unit available which will be used during maintenance and repairs of others. A 30-t truck already owned by Abcourt will be adapted as water truck. This approach will lower the initial investment.

The total preproduction mining and support equipment cost is estimated at 6.58 M$. It should be noted that a second wheel loader (at the ore stockpile) and a fifth truck will be required at the beginning of production years 1 and 2 respectively for an additional cost totalling 1.3 M$, as shown in the on-going investment.

Because of the aforementioned existing or already owned mining facilities, namely haul roads, mine offices and pumping equipment, the related capital cost will be relatively low. Haul roads will be improved and/or constructed with preproduction waste rock, thus included in waste stripping costs. If not available from preproduction at the beginning, waste rock is easily available from the existing waste rock dump north of the former Barvue pit. The mining facilities capital cost estimate of 693 000 $ allows for pit dewatering before and during the preproduction pit development, together with site preparation of marginal ore stockpile and ore mixing areas.

Table 11.2.1 Mine preproduction capital cost estimate.

	Unit cost ($)	Qty	Cost ($)
Mining equipment			
Production			
Drill - 203 mm diam. (waste blast holes)	1 000 000	1	1 000 000
Drill - 76 mm diam. (ore blast holes)	250 000	1	250 000
Haul trucks (62-tonnes rebuilt)	600 000	4	2 400 000
Wheel loader (6.8 m^3)	700 000	1	700 000
Hydraulic shovel (3.5 m^3)	900 000	1	900 000
Support			
Bulldozers (240 kW rebuilt)	350 000	2	700 000
Grader (150 kW rebuilt)	200 000	1	200 000
Water truck (already owned; adapted)	20 000	1	20 000
Hydraulic rock breaker / scaling bar (used)	250 000	1	250 000
Pickup trucks	40 000	4	160 000
Total mining equipment			**6 580 000**
Mining facilities			
Pit dewatering			618 000
Marginal ore stockpile (site preparation)			75 000
Total mining facilities			**693 000**
Preproduction stripping and mining			
Staff costs			680 000
Labour costs			3 259 000
Equipment supplies and consumables			8 600 000
Total preproduction stripping			**12 539 000**
Total mine preproduction cost			**19 812 000**

Overburden stripping of both Barvue pit expansion and the east Abcourt pit (connected to the Barvue pit) and waste rock stripping of the Barvue pit will be performed during preproduction phase. The west Abcourt pit will be developed later in the mine schedule and the related overburden stripping cost will be included in the on-going investment.

Preproduction stripping of overburden and waste rock is required to expose sufficient ore to ensure a continuous supply to the mill after start-up. It is assumed that this work will be carried out using the owner's equipment and crews, so the estimate was based on operating costs. Preproduction costs for overburden and waste rock stripping are estimated at 4.06 M$ and 8.48 M$ respectively, for a total of 12.54 M$, and includes staff, labour and equipment supplies costs; the latter includes maintenance and repairs by the suppliers.

11.2.2 Process plant capital cost

The process plant capital cost estimate is based on quotations, cost factors and in-house data. Direct and indirect costs related to the concentrator and crushing section amounts to 44.27 M$. Details of the process plant capital cost estimate are provided in Table 11.2.2.

Table 11.2.2 Process plant and crushing section cost summary.

Description	Cost ($)
Equipment and Installation	
Ore Crushing	1 680 350
Ore Grinding	6 775 251
Zinc Flotation	1 522 377
Pyrite Flotation	907 052
On Stream Analyzer	589 836
Zinc Dewatering	1 708 348
Cyanydation	6 102 429
Reagents Facilities	859 672
Services Section	1 043 810
Laboratory	312 161
Sub-Total - Equipment and Installation	21 501 286
General and Services	
Instrumentation/Programmation	1 750 000
Main Power Supply Installation c/w MCC's	809 000
Main Sub-power Station	175 955
Re-circulated Water - 6" Line	134 960
Tailing Pipe - 8"	260 190
Tailing Pipe - 4"	82 440
Sub-Total - General and Services	3 212 545
General Steel Works	
Crusher Plant (Supports and Accessories)	441 925
Crusher Building	293 585
Process Plant (Supports and Accessories)	1 583 290
Process Building	3 507 590
1 800 t Crushed Ore Bin	93 825
Sub-Total - General Steel Works	5 920 215
General Concrete Works	
Crusher Plant (Foundations)	183 143
Crusher Building	240 872
Process Plant (Foundations)	1 796 930
Process Plant Building	840 958
Laboratory	143 435
1 800 t Crushed Ore Bin	218 045
Sub-Total - General Concrete Works	3 423 383
Sub-Total	**34 057 429**
EPCM 15 %	5 108 614
Contingencies 15 %	5 108 614
Total	**44 274 657**

The cost includes the equipment listed as per the flowsheet, the architectural and structural components, the concrete, the equipment transportation cost, the mechanical, electrical and piping costs, the EPCM (engineering, procurement and construction management) and 15 % for unforeseen and miscellaneous (Appendix 4).

Abcourt is presently in discussion with two mining companies to purchase/rent the ore processing equipment installed in either one of two non-operating mills in the Abitibi region. If a deal occurs, this could represent a potential economy of 8 to 10 M$ on the total capital cost of the process plant or a saving of about 12 % of the total initial capital cost of 67.88 M$. The impact on the profitability of the project is addressed in the Sensitivity analysis section of Chapter 13 (see figures 13.3.1 to 13.3.4).

11.2.3 Infrastructure capital cost

The estimated preproduction capital costs for the infrastructures required for this project is detailed in Table 11.2.3. The total cost is 1 862 000 $.

Table 11.2.3 Infrastructure preproduction capital costs estimate.

	Cost ($)
Road network	
Improvement and construction	Included in waste stripping
Total road network	**0**
Site services	
Reshaping of service building	150 000
Computers and telecommunications	130 000
Heavy equipment maintenance and repair shop	100 000
Total site services	**380 000**
Power line	
Improvement of existing 25 kV power line	25 000
Total power line	**25 000**
Tailings pond	
Starting dam (high-security sulphide cell included)	1 257 000
Total tailings pond	**1 257 000**
Water treatment plant	
Purchase of used equipment and construction of building	200 000
Total water treatment plant	**200 000**
Total infrastructure preproduction cost	**1 862 000**

The on-site road network is already well developed and its use will be maximized. However, some parts will have to be improved and others to be constructed. Waste rock from Barvue pit expansion will be used for road improvement and construction. The related costs are included in preproduction waste stripping.

Items under site services include reshaping the existing service building, construction of a new maintenance and repair shop for heavy mining equipment on an existing concrete pad, computers and telecommunication. The estimate is based on quotations and in-house data and amounts to 380 000 $.

The existing 25 kV power line with transformers and electrical distribution network are still available and operational on site. It will have to be checked and relocated where required. Reticulation network, transformers and sub-station to underground mine and mill plant are included under these items. A provision of 25 000 $ was made for work related to the main power line.

The quantity of material needed for the construction of the starting dike (3 m high) was estimated in Golder's 2007 report. Since all the waste rock and clay needed for the construction of this dike is available on the site or will be produced during the stripping operation and as the construction of the dike will be done by Abcourt personnel instead of contractor, the construction cost of the starting dike, including the installation of the high-security sulphide cell, is estimated at 1 257 000 $.

The water treatment plant constructed at the southeast limit of the tailings pond will consist of reused equipment purchased from another Abitibi mine (Louvicourt). A provision of 200 000 $ was made for the construction of a 54 m² building on a concrete slab and the purchase, dismantling, transportation and reinstallation of the equipment on the Abcourt-Barvue site.

11.2.4 Owner's cost

Owner's costs are indirect cost supported by Abcourt during construction and preproduction phase. In this case, these are general administration for one year (details in Operating costs section) and spare parts purchase for warehouse inventory. General administration for year –1 amounts to 1.43 M$ and warehouse inventory is estimated to 0.5 M$, for an owner's total cost of 1.93 M$.

11.2.5 Working capital

The working capital is equivalent to 2 months of operating costs and it amounts to 3.38 M$. This will be entirely recovered at the end of the project.

11.2.6 On-going investment

The usual accounting practice is to capitalize major expenditures during production. In this particular case, costs related to additional surface equipment, stripping of overburden, progressive rehabilitation (dumps, piles and tailings pond), mining and stockpiling of marginal ore, tailings pond management (dikes raising) and underground mine initial investment and on-going development will all be capitalized. Table 11.2.4 itemizes the estimated capital requirements for the Abcourt-Barvue project in specific years. The estimated on-going investment amounts to 24.42 M$.

Table 11.2.4 On-going capital expenditure estimate.

	Year -1 ('000 $)	Year 1 ('000 $)	Year 2 ('000 $)	Year 3 ('000 $)	Year 4 ('000 $)	Year 5 ('000 $)	Year 6 ('000 $)	Year 7 ('000 $)	Year 8 ('000 $)	Year 9 ('000 $)	Year 10 ('000 $)	Total ('000 $)
Surface and open pits												
Wheel loader (6.8 m³)		700.0										**700.0**
Haul truck (62-t rebuilt)			600.0									**600.0**
Overburden stripping (west pit)							578.2					**578.2**
Marginal ore mining&stockpiling		268.4	254.0	323.9	368.4	390.3	372.8	158.4	97.7	96.8	84.9	**2 415.6**
Tailings pond			2 269.0					2 041.0				**4 310.0**
Progressive rehabilitation		100.0	100.0	100.0	100.0	100.0	100.0	100.0	350.0	350.0	350.0	**1 750.0**
Sub-total		1 068.4	3 223.0	423.9	468.4	490.3	1 051.0	2 299.4	447.7	446.8	434.9	**10 353.8**
Underground mine												
Initial investment							1 712.0					**1 712.0**
On-going development							2 565.6	2 825.8	2 320.2	2 320.2	2 320.2	**12 352.0**
Sub-total							4 277.6	2 825.8	2 320.2	2 320.2	2 320.2	**14 064.0**
Total		**1 068.4**	**3 223.0**	**423.9**	**468.4**	**490.3**	**5 328.6**	**5 125.2**	**2 767.9**	**2 767.0**	**2 755.1**	**24 417.8**

Surface and open pits

Additional equipment will be required to achieve the planned production level in the open pits. Thus, a wheel loader (same model as the other unit at 700 000 $) will be purchased at year 1 to be assigned to the ore mixing area and another rebuilt 62-t haul truck will be bought for 600 000 $ at year 2. The overburden stripping cost of 578 200 $ for the west Abcourt pit will be capitalized at year 6.

A recurrent expenditure of 100 000 $ will occur each and every year of the Abcourt-Barvue project for progressive rehabilitation at surface (waste rock dumps, overburden pile, etc). The expense for the tailings pond rehabilitation amounts to 750 000 $ (60 ha at 12 500 $/ha) and it is spread over the last three years of the 10-year production schedule (250 000 $/year), but in reality these expenses will be incurred whenever the tailings pond is closed. The overall on-going rehabilitation cost if then 1.75 M$.

Costs related to mining and stockpiling of marginal ore (2.42 M$) and tailings pond (dams construction and/or raising for 4.31 M$) all over the life-span of the project will be considered as on-going investment for a total amount of 6.73 M$.

The estimated total cost is 10.35 M$ for open pit and surface on-going investment.

Underground mine

The preparation work for underground mining consists of re-installing the existing related surface installations removed to make room for the open pit. Even if Abcourt has most of the equipment needed for underground mining, a provision was made for any additional equipment purchase.

The estimated costs of preparation work for underground mining were updated from Roche's Technical Validation Report, 1999; and are as follows:

Relocation of the water tank	25 000 $
New shaft collar	12 500 $
Installation of pumping stations at bottom of stopes	215 000 $
Installation of two compressors and compressor room	200 000 $
Compressed air line	40 000 $
Relocation of power line and sub-stations	62 500 $
Installation of main ventilation fans and heating systems	75 000 $
Water and air lines on 90 m level	107 000 $
Extension of the 90 m sub-level in Abcourt and breakthrough of ramps in the pit	325 000 $
Additional equipment purchase	650 000 $
	1 712 000 $

Development unit costs come from detailed estimate made for Vendôme mine in 1999 and validated by Roche in its 1999 Technical Validation Report, adjusted by +25 % for inflation. The estimated costs of development work for Avoca stopes are as follows:

West stope (5010E to 5280E)			
Ramp No 3	500 m	at 1 800 $/m	900 000 $
Sub-levels	1 755 m	at 1 800 $/m	3 159 000 $
Loading and passing stations	100 m	at 1 800 $/m	180 000 $
Sumps	50 m	at 1 800 $/m	90 000 $
Stope raises	100 m	at 1 060 $/m	106 000 $
Fill and ventilation raises	300 m	at 1 060 $/m	318 000 $
Diamond drilling	1 500 m	at 40 $/m	60 000 $
Sampling and assaying			15 000 $
			4 828 000 $
Central stope (5295E to 5760E)			
Ramp No 2	475 m	at 1 800 $/m	855 000 $
Sub-levels	1 640 m	at 1 800 $/m	2 952 000 $
Loading and passing stations	100 m	at 1 800 $/m	180 000 $
Sumps	50 m	at 1 800 $/m	90 000 $
Stope raises	60 m	at 1 060 $/m	63 600 $
Fill and ventilation raises	180 m	at 1 060 $/m	190 800 $
Sampling and assaying			25 000 $
			4 356 400 $
East stope (5295E to 5760E and 15W to 135E)			
Ramp No 1	100 m	at 1 800 $/m	180 000 $
Sub-levels	1 500 m	at 1 800 $/m	2 700 000 $
Loading and passing stations	100 m	at 1 800 $/m	180 000 $
Sumps	50 m	at 1 800 $/m	90 000 $
Stope raises	160 m	at 1 060 $/m	169 600 $
Fill and ventilation raises	145 m	at 1 060 $/m	153 700 $
			3 473 300 $
Total development cost			**12 657 700 $**

Given that at the end of year 10, ore from the underground Avoca stopes will not be completely exhausted, total capitalized cost for the underground development is estimated at 12.35 M$; the remainder of investment occurring after year 10, thus subsequent to the first 10 years of the project.

The estimated total amounts to 14.06 M$ for underground mine initial investment and on-going development expenditures.

12. OPERATING COSTS

12.1 Summary

The estimated operating costs are in third-quarter 2006 Canadian dollars with no allowance for escalation. The total life-of-mine and average unit operating costs are summarized in Table 12.1.1 and further detailed in this section. The overall operating unit cost stands at 5.65 $/t mined or 31.16 $/t milled.

Table 12.1.1 Operating cost estimate summary.

Item	Life-of-mine costs			Unit costs	
	Labour (M$)	Supplies (M$)	Total (M$)	Mined ($/t)	Milled ($/t)
Mining[1]	31.23	67.37	98.60	2.77	15.30
Processing	17.29	70.13	87.42	2.45	13.46
G&A	5.71	8.59	14.30	0.40	2.22
Royalties	----	1.16	1.16	0.03	0.18
Total	54.23	147.25	201.48	5.65	31.16

[1] Supplies include maintenance and repairs by the suppliers for open pit equipment.

12.1.1 Basis of estimate

Salaries and hourly labour rates are based on similar operations in Abitibi region or on salaries posted in 1999 Roche's report and updated with published salary evolution again in the Abitibi region.

Supplies and parts costs were obtained from potential suppliers as budget quotations and from in house data for similar projects or, once again updated from 1999 Roche's report.

12.2 Personnel

12.2.1 Mine

Open pit mining crews will work 8-h shifts around the clock, 7 days per week and 52 weeks per year at peak annual waste to ore production ratio. Underground mining crews will work two 8-h shifts per day, 5 days per week and 52 weeks per year. Mining personnel requirements at peak and salaries, including fringe benefits (34 % for hourly paid employees and 23 % for staff employees), overtime and bonus, are listed in Table 12.2.1. Normally, there is an absence ratio of 10 % for various reasons. Therefore, replacement employees are needed for both open pit and underground operations and cost related to their salaries are covered in fringe benefits factors.

Table 12.2.1 Mine manpower requirement and salaries.

Description	Number of employees	Salary Cost	Description	Number of employees	Salary cost
OPEN PIT			*UNDERGROUND*		
Staff Employees		*($/year)*	*Staff Employees*		*($/year)*
Mine superintendent	1	115 000	Captain	1	105 000
Shift boss	4	68 000	Shift boss	2	80 000
Mining engineer	1	85 000	Maintenance foreman	1	80 000
Mining technician	1	56 000	Mining engineer	1	75 000
Geological technician	1	56 000	Geological technician	1	62 000
Surveyors	2	52 000	Surveyor	1	62 000
Environmental coordinator	1	60 000	Safety mine rescue	1	54 000
Sub total	11		Sub total	8	
Hourly Employees		*($/hour)*	*Hourly Employees*		*($/hour)*
Production loader operators	4	31.20	Long hole drillers	3	44.26
Shovel operators	3	31.20	Muckers	6	34.98
Truck drivers	16	29.48	Truck drivers	3	34.98
Drillers (ore)	4	35.00	Jumbo operators	6	44.26
Drillers (waste)	4	35.00	Jumbo helpers	6	38.79
Blaster	1	40.79	Blasters	3	44.26
Blaster helper	1	35.43	Mechanics	6	34.44
Stockpile loader operators	3	31.20	Electrician	1	32.80
Dozer operator	1	29.48	Sub total	34	
Dozer & Rock breaker op.	1	29.48	Spares	4	
Grader & Water truck op.	1	29.48	**Total**	**46**	
Sub total	39				
Spares	4				
Total	**54**				

12.2.2 Mill

The operating personnel has been based on an operation of three 8-hour shifts per day and the number of hourly paid employees has been increased by 10 % taking into consideration the requirement for additional employees needed for replacement during vacation, sickness and other reasons.

The total annual salary of hourly employees has been calculated by adding to the base salary 27.55 % as fringe benefits to the hourly paid employees and 15.14 % for the staff employees. Manpower costs are listed in Table 12.2.2.

Table 12.2.2 Mill manpower requirement and salaries.

Description	Number of Employees	Salary cost
Staff Employees		*($/year)*
Plant Superintendent/Metallurgist	1	120 897
General Foreman	1	77 144
Mechanic Foreman	1	70 235
Technician	1	51 813
Sub Total	4	
Hourly Employee		*($/hour)*
Crusher/Loader Operators	3	26.79
Concentrator Senior Operators	4	29.97
Concentrator Junior Operators	4	28.06
Concentrator Helper Operator	5	24.23
Maintenance Mechanic	4	28.06
Maintenance Electric/Instrumentation	1	28.06
Refiner Senior	1	29.97
Refiner Helper	1	24.23
Sub Total	23	
Security (contractor)	*1*	*18.00*
Sub Total	1	
Total manpower	**28**	
Miscellaneous	*Hours*	*($/year)*
Maintenance Overtime	1 250	27 500
Night Shifts Bonus	2 920	11 680
Weekend Bonus	1 248	9 984
Overtime Related to Holidays	144	4 618
Maintenance Contractor	250	11 250
Sub Total	5 812	

12.2.3 General and administration

General and administration (G&A) personnel requirements and salaries, including fringe benefits of 15.14 % are listed in Table 12.2.3.

Table 12.2.3 G&A personnel and salaries.

Description	Number of employees	Annual Salary ($)
Mine manager	1	180 000
Chief accountant	1	110 000
Accounting clerk	1	40 000
Storekeeper	1	64 000
Secretary	1	40 000
Total	**5**	**434 000**

12.3 Estimate details

12.3.1 Mine operating costs

The cost for open pit mining alone has been estimated for each year of production. The cost varies depending on which pits are in production, the mining depth, the waste dump height and haulage distances to stockpiles.

The 10-year average operating cost for the open pits is estimated to 2.34 $/t mined or 15.15 $/t milled for drilling and blasting, load and haul, support activities, pumping and water treatment and technical services (staff employees). The breakdown of the open pit mining cost per tonne mined is given in Table 12.3.1.

Table 12.3.1 Breakdown of open pit mining cost.

Activity	Unit cost ($/t mined)	Component	Unit cost ($/t mined)
Drilling and blasting	0.98	Labour	0.20
		Fuel	0.10
		Explosives	0.63
		Steel and bits	0.05
Load and haul	0.86	Labour	0.22
		Fuel	0.15
		Other supplies[1]	0.49
Production support	0.31	Labour	0.06
		Fuel	0.06
		Other supplies[1]	0.19
Pumping & water treatment	0.04	Labour	0.01
		Electricity	0.02
		Reagents and supplies	0.01
Technical services and Supervision	0.15	Staff employees salary	0.15
Total	**2.34**		

[1] Including maintenance and repairs by the suppliers.

The cost for underground mining alone comes from Roche's Technical Validation Report, 1999, and has been adjusted by +27 % to +30 % for inflation, depending on cost components.

The 10-year average operating cost for the underground mine is estimated to 20.93 $/t of ore mined from stope production activities. The development work to prepare stopes produces ore (23.5 % of underground ore tonnage) which cost is accounted in capital expenditures. Thus, the aforementioned unit cost decrease to 16.02 $/t of ore extracted or milled from underground mine, no matter its source (production or development activities). The breakdown of the underground mining cost per tonne of ore extracted is given in Table 12.3.2.

Table 12.3.2 Breakdown of the underground mining cost.

Component	Unit cost ($/t mined)	Item	Unit cost ($/t mined)
Stoping and ore Transportation	8.43	Drilling and blasting	2.54
		Mucking and transportation	4.33
		Stope backfilling[1]	1.09
		Mucking before blasting	0.47
Fixed costs	12.50	Mechanical services	5.75
		Electricity	1.38
		Propane	1.50
		Tech. serv. and supervision	3.87
Total	**20.93**		

1 Cost for broken waste muck backfill.

In combining overall total costs of open pit and underground mining, the 10-year average mine operating cost is estimated to 2.77 $/t mined or 15.30 $/t milled, as given in Table 12.3.3.

Table 12.3.3 Summary of overall mine operating costs.

Description	10-year total cost (M$)	Unit costs (OP & UG combined)	
		Mined ($/t)	Milled ($/t)
Open pit	80.9	2.27	12.55
Underground mine	17.7	0.50	2.75
Total	**98.6**	**2.77**	**15.30**

12.3.2 Ore processing costs

The 10-year processing cost is estimated to 87.42 M$ equivalent to 13.46 $/t milled or 2.45 $/t mined. The breakdown is given in Table 12.3.4.

Table 12.3.4 Breakdown of the ore processing cost.

Description	10-year total cost (M$)	Unit costs	
		Mined ($/t)	Milled ($/t)
Manpower	17.29	0.48	2.66
Processing reagents	37.34	1.05	5.75
Smelting (Ag precipitate)	1.64	0.05	0.25
Operating consumables	8.42	0.24	1.30
Hydro/Quebec charges	13.43	0.38	2.07
Other supplies and heating (gas)	9.30	0.25	1.43
Total	**87.42**	**2.45**	**13.46**

Ore processing cost includes all costs from crushing and receipt of ore through Zn-Ag concentrates and Ag-Au ingots production and thickening of tailings and pyrite concentrate prior to disposal. Details of operating cost item estimates are provided in Appendix 5.

12.3.3 General and administration costs

The fixed annual general and administration (G&A) cost is estimated to 1,43 M$, including staff salaries and other expenses (communications, fees, taxes, insurances, interests and bank fees, etc). The G&A unit cost is then equal to 0.40 $/t mined or 2.22 $/t milled.

12.3.4 Royalties

The royalties are payable to Terratech (0.25 $/st) on almost 80 % of ore tonnage mined by open pit, for a 10-year total cost of 1.16 M$. This life-of-mine total payment is equivalent to 0.03 $/t mined or 0.18 $/t milled.

13. ECONOMIC ANALYSIS

A pre-tax and an after-tax models have been developed for the Abcourt-Barvue project. All costs are in third-quarter 2006 Canadian dollars with no allowance for inflation or escalation.

The economic valuation of the project has been conducted with the Internal Rate of Return (IRR) and the Net Present Value (NPV) methods.

The NPV method converts all cash flows of investments and revenues occurring throughout the planning horizon of a project to an equivalent single sum at present time using a discount rate equals in this case to the commercial prime interest rate plus a risk premium (7 %) minus the inflation rate (2 %). The discount rate used in the analysis is then 5 %. According to the NPV method, a positive NPV represents a profitable investment where the initial investment plus any financing interest are recovered.

The IRR on an investment is defined as the rate of interest earned on the unrecovered balance of an investment. An investment can be accepted if IRR is greater than the minimum attractive rate of return of the project.

In the present analysis, in addition to the NPV and the IRR, the return on equity (ROE) and payback periods have been calculated from the pre-tax and after-tax cash flow statements.

The following analyses have been carried out:

- six scenarios have been developed to see the impact of changing metal prices for zinc, silver and gold and the exchange rate (C$/US$) on the economics of the project;
- the cash flow for various scenarios have been modelled on the assumptions of 40 % equity and 60 % debt at 7 % per year fixed interest rate;
- the cash flow for various scenarios have also been modelled with 100 % equity financing, with no debt, interest expense nor capital repayment;
- a sensitivity analysis was performed in order to examine the effect of variation of key parameters on the economic analysis results. The sensitivity to changes in total revenue (NSR) and capital and operating expenditures has been calculated from the cash flow statement and shown graphically.

13.1 Scenarios description

Six scenarios have been developed to study the impact of the zinc, silver and gold prices on the valuation of the project. Different values of exchange rate between the Canadian and American currencies have also been incorporated in the scenarios. A NSR value (C$ per tonne of ore) has thus been calculated for each scenario and has been used in the cash flow model. Table 13.1.1 summarizes the economic parameters for each scenario.

Table 13.1.1 Economic parameters for the scenarios modelled.

	Scen.1	Scen.2	Scen.3	Base Case	Scen.4	Scen.5	Scen.6
Au price (US$/oz)	370	450	545	560	600	625	650
Ag price (US$/oz)	4.80	8.00	9.50	9.54	10.00	11.75	13.50
Zn price (US$/lb)	0.40	0.60	1.00	1.15	1.40	1.80	2.20
Exchange rate (C$/US$)	1.33	1.22	1.16	1.15	1.12	1.12	1.12

13.2 Cash flow analysis

The results of the cash flow analysis for the six scenarios are presented in table 13.2.1. Capital and operating expenditures are constant for all the scenarios. Total revenue based on the NSR value varies for each scenario. The economic valuation of the project is presented for the two financing methods modelled.

The project becomes profitable with the parameters used in the scenario 3, for which the IRR after taxes is 11.59 %. More precisely, at constant metal prices and exchange rate over the 10-year period, the breakeven point on a 60 %/40 % loan to equity basis occurs at metal prices equal to 0.70 US$/lb Zn, 8.42 US$/oz Ag and 475 US$/oz Au and at an exchange rate of 1.20 C$/US$. On a 100 %-equity basis, the break even Zn price is then 0.66 US$/lb also considering capital expenditures. On a 100%-equity basis without considering any investment (EBITDA: earnings before interests, taxes, depreciation and amortization), the breakeven point occurs at a zinc price of 0.43 US$/lb.

13.3 Base case scenario

For the base case analysis, the most likely price scenario over the next 10 to 12 years has been used. This is based on the fact that demand for zinc, particularly from China and India, is expected to remain very strong over the next several years (8 to 10+ years). Therefore, the following zinc prices have been used: 1.36 US$/lb for the first six (6) years and 0.85 US$/lb for years 7 to 10 inclusively.

Table 13.2.1 Results of the cash flow analysis.

	Scénario 1	Scénario 2	Scénario 3	Base Case	Scénario 4	Scénario 5	Scénario 6
Au ($US/oz)	370	450	545	560	600	625	650
Ag ($US/oz)	4.80	8.00	9.50	9.54	10.00	11.75	13.50
Zn ($US/lb)	0.40	0.60	1.00	1.15	1.40	1.80	2.20
Exchange rate ($CAN/$US)	1.33	1.22	1.16	1.15	1.12	1.12	1.12
Total Revenue (NSR)	**188 845 900 $**	**272 423 400 $**	**392 827 900 $**	**433 537 500 $**	**497 561 000 $**	**627 638 300 $**	**757 774 100 $**
Total Capital Expenditures	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**	**(92 296 800 $)**
Total Operating Expenditures	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**	**(200 825 484 $)**
FINANCING METHOD : 60% DEBT, 40% EQUITY							
Financial Valuation before taxes							
Return on equity (ROE)	N/A	N/A	20.32%	39.66%	45.29%	79.36%	116.28%
NPV (5%)	(78 576 740 $)	(14 671 746 $)	77 574 445 $	114 704 209 $	157 859 241 $	257 477 065 $	357 139 396 $
IRR of the project	N/A	N/A	14.76%	25.00%	29.45%	46.22%	62.45%
Payback period (years)	NO PAY BACK	NO PAY BACK	6.78	4.00	3.40	2.16	1.60
Financial Valuation after taxes							
Taxes	0 $	0 $	(30 140 282 $)	(45 783 696 $)	(71 722 817 $)	(123 573 555 $)	(175 860 274 $)
Return on equity (ROE)	N/A	N/A	16.23%	33.12%	38.08%	68.15%	98.69%
NPV (5%)	(78 576 740 $)	(14 671 746 $)	57 835 865 $	82 575 331 $	108 335 000 $	169 382 050 $	229 458 990 $
IRR of the project	N/A	N/A	11.59%	20.02%	23.79%	37.66%	50.33%
Payback period (years)	NO PAY BACK	NO PAY BACK	8.63	5.00	4.20	2.66	1.99
FINANCING METHOD : 100% EQUITY							
IRR	N/A	N/A	17.04%	27.54%	31.68%	48.27%	64.18%
NPV (5%) (1)	(94 988 095 $)	(34 126 197 $)	53 727 318 $	89 088 999 $	130 189 030 $	225 063 147 $	319 979 653 $
EBITDA (2)	(11 979 584 $)	71 597 916 $	192 002 416 $	232 712 016 $	296 735 516 $	426 812 816 $	556 948 616 $
Taxes	0 $	0 $	(34 435 405 $)	(50 078 820 $)	(76 017 941 $)	(127 868 679 $)	(180 155 397 $)
Bare-Bones Valuation (3)	(11 979 584 $)	71 597 916 $	157 567 011 $	182 633 197 $	220 717 576 $	298 944 138 $	376 793 219 $

1 Non inflation discount rate: 7 % interest rate - 2 % inflation rate = 5 %.

2 EBITDA: earnings before interests, taxes, depreciation and amortization and as per definition, without considering any investment.

3 Bare-Bones Valuation: cash flow evaluations based on constant metal prices, constant dollars, no inflation, no debt, no interest, on a project basis and after tax.

In the meantime, the Ag price varies from 9.95 to 8.94 US$/oz and the Au price from 595 to 510 US$/oz, while the exchange rate increases from 1.13 to 1.19. The weighted average metal prices over the 10-year period are then 1.15 US$/lb for zinc, 9.54 US$/oz for silver and 560 US$/oz for gold. Always by financing the project with 60 % debt and 40 % equity, the pre-tax and after-tax IRR are 25 % and 20 % respectively with NPV of 115 M$ pre-tax and 83 M$ after-tax, at a discount rate of 5 %. The period of time required to recover the cost of the investment (payback period) is 4 years pre-tax and 5 years after-tax in this particular case.

13.4 Sensitivity analysis

The parameters analysed in the sensitivity analysis performed on the base case were chosen due to the expectation of a major impact on the outcome of economic valuation. These were:

- net revenue of production (REVENUE);
- capital expenditures (CAPEX);
- operating expenditures (OPEX).

The sensitivity calculations were performed on the project cash flow by applying a range of variations of ± 25 % to the parameter values both on a pre-tax and an after-tax basis. The results are presented in table 13.4.1. The effects on NPV and IRR before and after taxes are shown graphically in figures 13.4.1 to 13.4.4.

As illustrated on the figures, the project is highly sensitive to changes in total revenue, based on the NSR value of the ore, thus metal prices. At - 25 % of revenue, the NPV after taxes of the project would be 28 M$ while at + 25 %, it would be 135 M$.

Moreover, the project is moderately sensitive to the operating expenditures and to the capital expenditures. With OPEX and CAPEX 25 % lower to 25 % higher than the base case, the NPV after taxes of the project would range between 107 and 58 M$ for OPEX variations alone and between 91 and 74 M$ for CAPEX fluctuations alone.

Table 13.4.1 Sensitivity analysis results.

		-25 %	-10 %	0 %	+10 %	+ 25 %
Net revenue of production	NPV, 5.0% (before taxes)	30 258 245 $	80 925 824 $	114 704 209 $	148 482 595 $	199 150 173 $
	IRR (before taxes)	4.82%	17.54%	25.00%	32.03%	42.13%
	NPV, 5.0% (after taxes)	27 769 683 $	61 339 084 $	82 575 331 $	103 567 283 $	134 625 587 $
	IRR (after taxes)	4.08%	13.95%	20.02%	25.81%	34.07%
Capital expenditures (CAPEX)	NPV, 5.0% (before taxes)	130 856 779 $	121 080 946 $	114 704 209 $	108 222 109 $	98 551 640 $
	IRR (before taxes)	36.65%	29.01%	25.00%	21.57%	17.24%
	NPV, 5.0% (after taxes)	91 179 779 $	85 824 745 $	82 575 331 $	79 178 503 $	73 925 963 $
	IRR (after taxes)	30.09%	23.43%	20.02%	17.13%	13.51%
Operating expenditures (OPEX)	NPV, 5.0% (before taxes)	154 168 716 $	130 490 012 $	114 704 209 $	98 918 406 $	75 239 702 $
	IRR (before taxes)	33.59%	28.46%	25.00%	21.47%	16.04%
	NPV, 5.0% (after taxes)	107 104 894 $	92 353 592 $	82 575 331 $	72 801 072 $	57 647 522 $
	IRR (after taxes)	27.10%	22.85%	20.02%	17.16%	12.73%

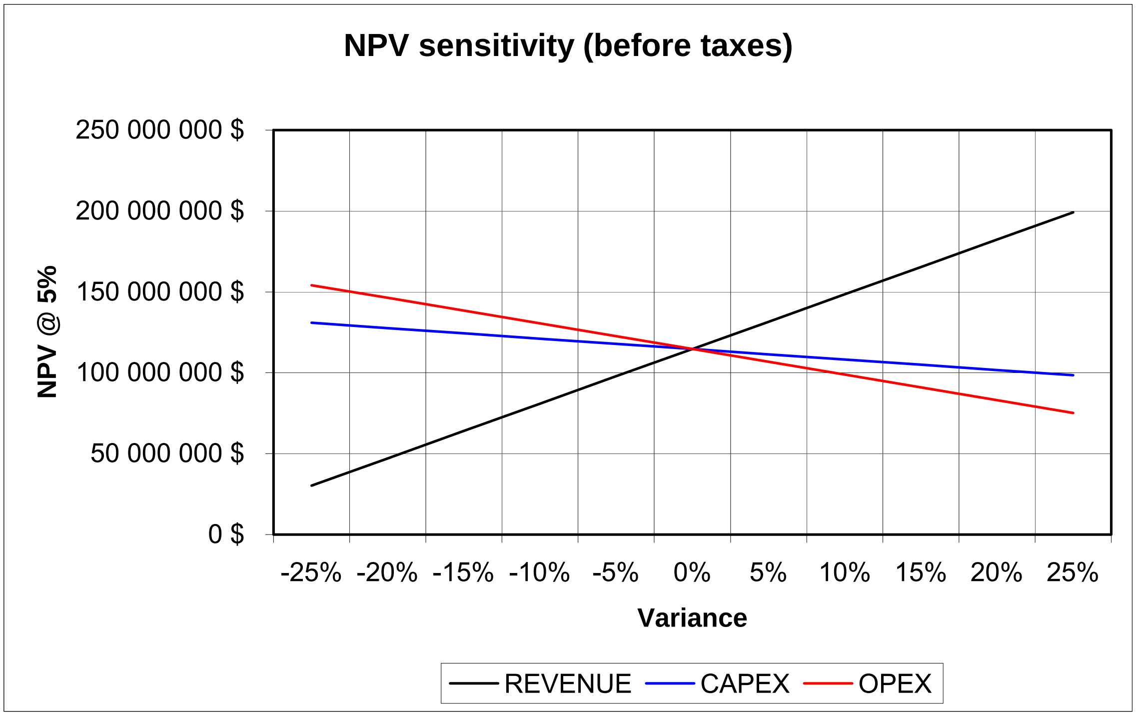


Figure 13.4.1 NPV sensitivity (before taxes).

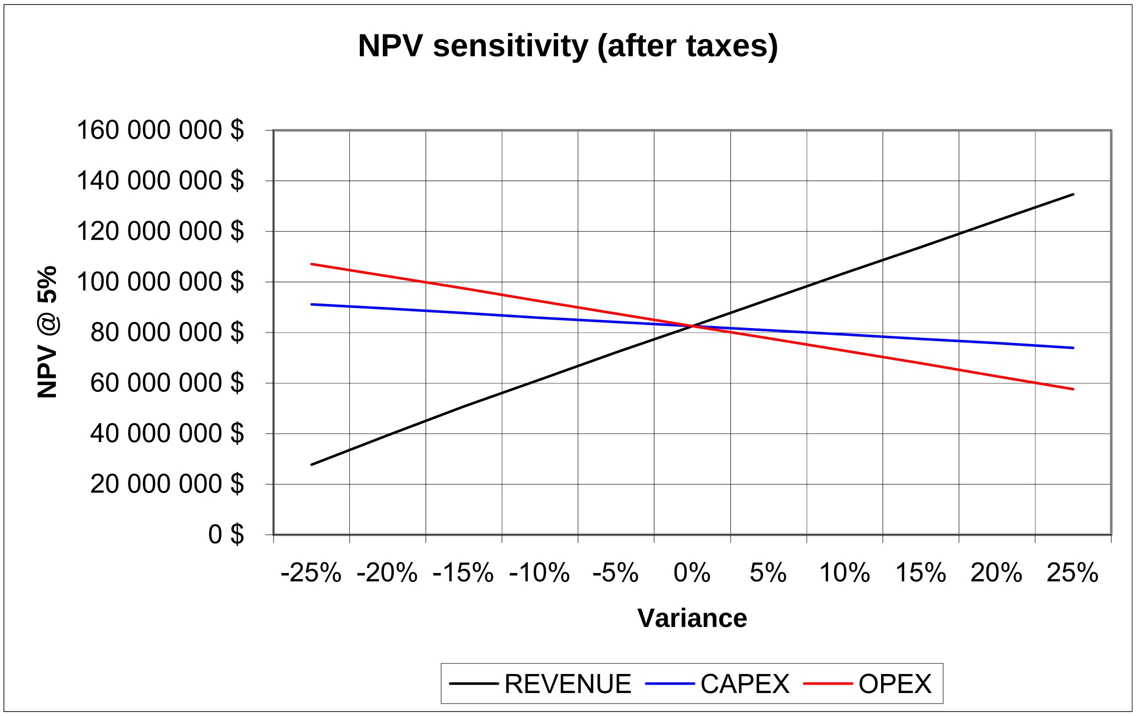


Figure 13.4.2 NPV sensitivity (after taxes).

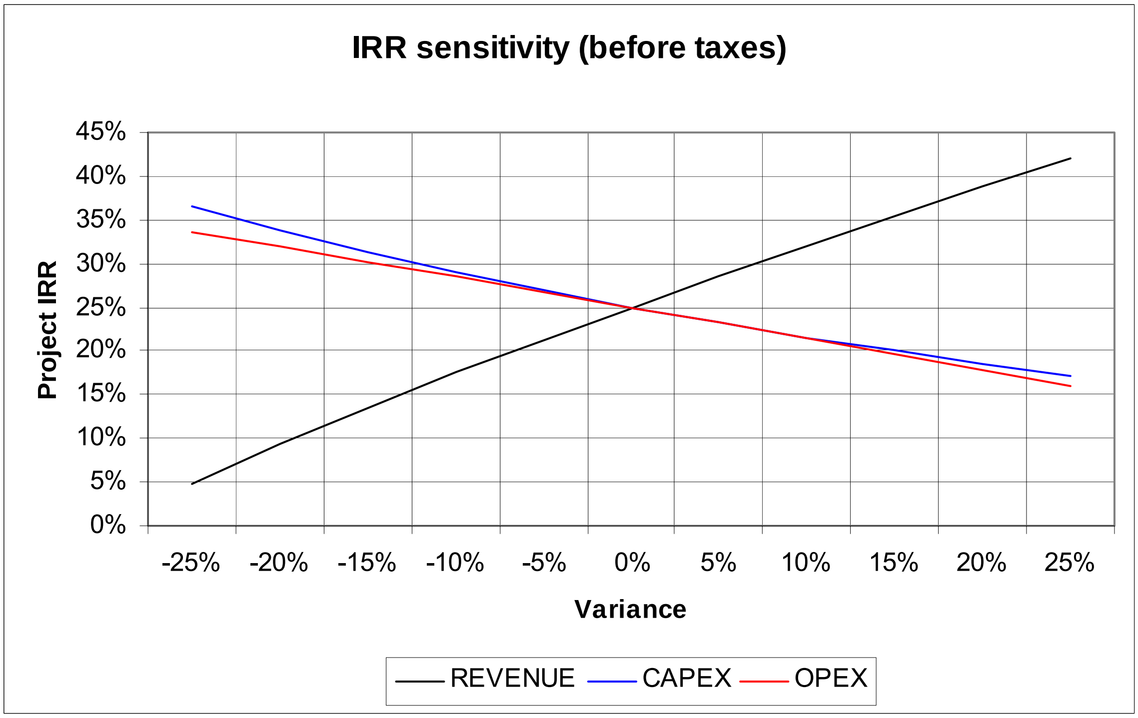


Figure 13.4.3 IRR sensitivity (before taxes).

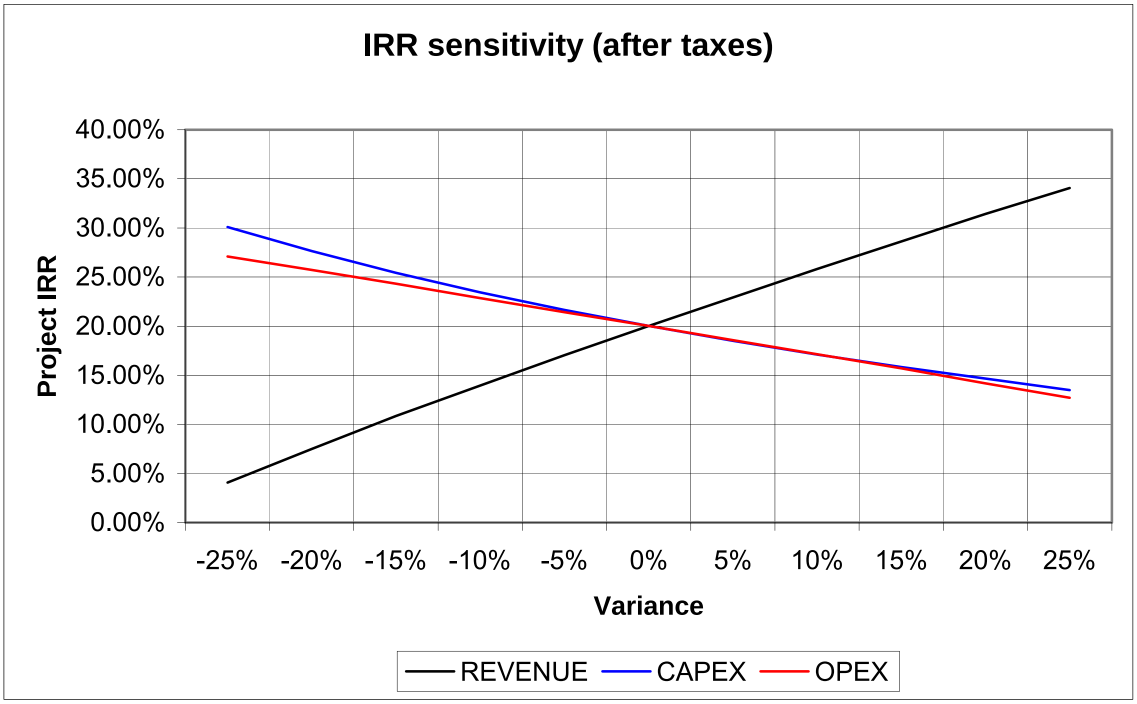


Figure 13.4.4 IRR sensitivity (after taxes).

14. PROJECT IMPLEMENTATION SCHEDULE

14.1 General

In the project schedule, the main engineering, procurement and construction activities are indicated. The schedule is based on information taken from supplier's quotes, in-house databases or estimated time frame for particular activities. The schedule presents the total duration of the construction project considering project design criteria stated in this report

It is assumed that:

- construction will be carried out over 5 days/week at regular 8 hours per day while other activities like overburden and waste stripping will occur at the mining rate of the open pit production schedule (7 days/week all around the clock);
- all concrete work will be done during the summer months (i.e. no winter allowances are included for civil works namely concrete).

14.2 Schedule of activities

14.2.1 Engineering

In Engineering, the important activities are the grinding mills (SAG and ball mills) specification preparation together with major equipment specification preparation (i.e. flotation cells, crushers, compressors, etc.). The general mechanical equipment specification preparation is also considered. It includes all other mechanical equipment. The service building and repair shop specification are also taken into account. The transformers specification preparation is next, followed by the general electrical and automation specification preparation (i.e. switchgear and motor control center etc.). The last activity is the detailed engineering that takes into account the project design criteria and all specific details from supplier's.

14.2.2 Procurement

The Procurement category details the previous activities and states the fabrication/rebuilt and expected delivery time to site.

14.2.3 Mine pre-stripping

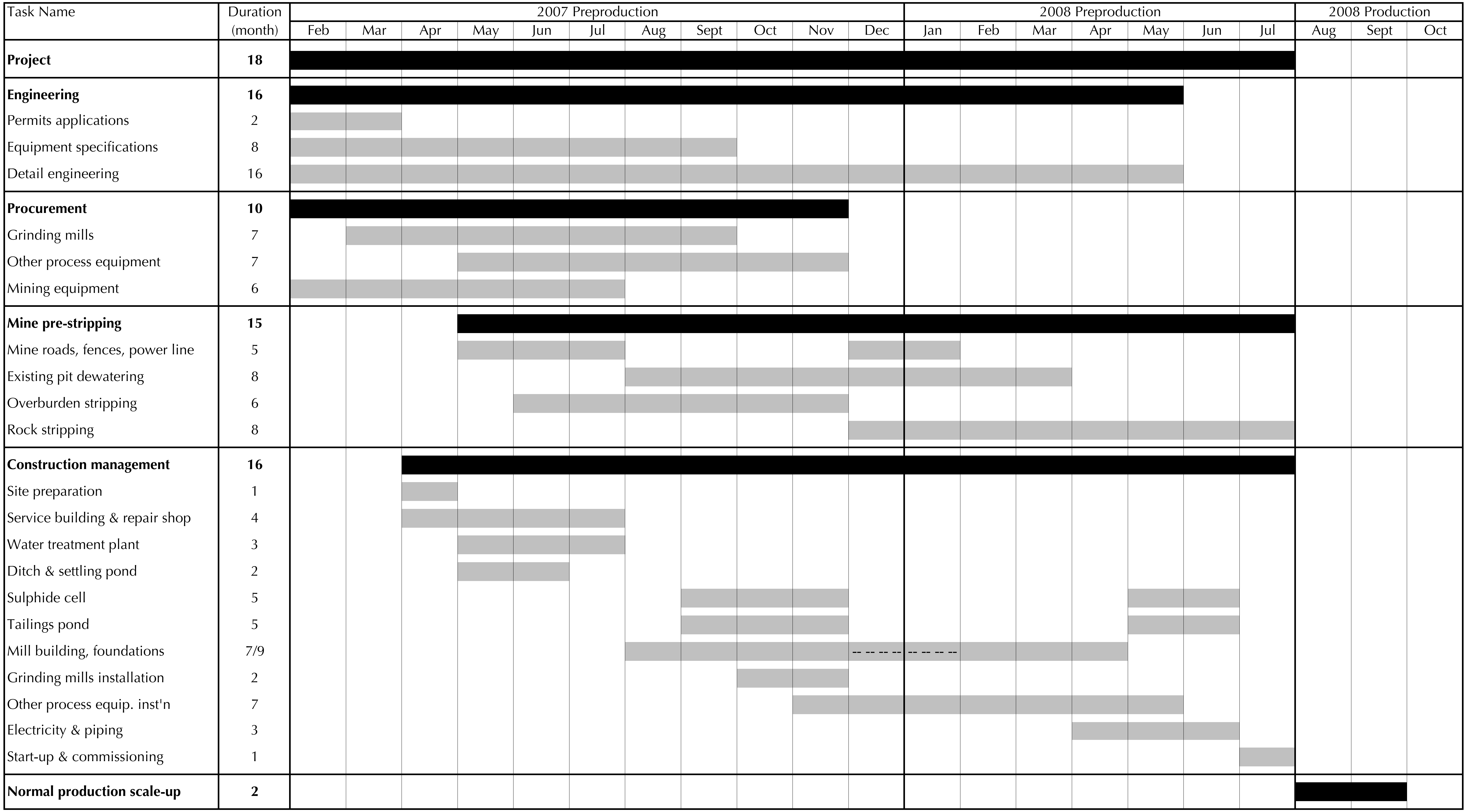
Mine Pre-Stripping indicates the duration required to have enough mine waste rock for construction and to expose sufficient ore in the pit before the start-up of the processing plant. Related activities are also indicated.

14.2.4 Construction

The Construction category indicates the duration for site-related activities which include site preparation, service building and repair shop installation, water treatment plant / ditch / settling pond system construction, tailings dams and sulphide cell construction, mill building construction (civil and concrete, structural steel, exterior and interior finishes), and mechanical, electrical and automation installation.

Based on the purchase and installation of reused grinding mills and new equipment for all other areas of the project, the estimated duration of the project implementation including start-up and commissioning is eighteen (18) months. The project implementation schedule is shown in figure 14.1.1.

Figure 14.1.1 Project implementation schedule



Task Name	Duration (month)
Project	**18**
Engineering	**16**
Permits applications	2
Equipment specifications	8
Detail engineering	16
Procurement	**10**
Grinding mills	7
Other process equipment	7
Mining equipment	6
Mine pre-stripping	**15**
Mine roads, fences, power line	5
Existing pit dewatering	8
Overburden stripping	6
Rock stripping	8
Construction management	**16**
Site preparation	1
Service building & repair shop	4
Water treatment plant	3
Ditch & settling pond	2
Sulphide cell	5
Tailings pond	5
Mill building, foundations	7/9
Grinding mills installation	2
Other process equip. inst'n	7
Electricity & piping	3
Start-up & commissioning	1
Normal production scale-up	**2**

15. CONCLUSIONS AND RECOMMENDATIONS

15.1 Conclusions

The project is located on an existing mine site with several readily useable infrastructures, in a very easily accessible area where all services may be obtained at competitive prices.

The economic mineral reserves which are planned to be mined by open pit and underground by the Avoca cut-and-fill method stand at 6 823 532 tonnes grading 3.11 % Zn and 57.76 g/t Ag for a zinc-equivalent grade of 4.31 % including dilution. The mineral reserves are subdivided as follows:

Mining method	Classification	Tonnage (t)	Grade		
			Ag (g/t)	Zn (%)	Zn EQ (%)
Open pit	Proven Mineral Reserves	5 338 731	44.79	3.15	4.09
Underground	Proven Mineral Reserves	1 169 662	105.19	2.87	5.06
	Probable Mineral Reserves	315 139	101.61	3.23	5.35
	Total Underground	1 484 801	104.43	2.95	5.12
Open pit and Underground	Proven Mineral Reserves	6 508 393	55.64	3.10	4.26
	Probable Mineral Reserves	315 139	101.61	3.23	5.35
Total		**6 823 532**	**57.76**	**3.11**	**4.31**

Because the project has been scheduled on a 10-year time horizon, tonnage extracted over that period is then 6 446 000 tonnes grading 3.11 % Zn and 54.96 g/t Ag for a zinc equivalent grade of 4.26 %.

Furthermore, there are slightly more than 3 additional years of production at the same milling rate with the remaining underground proven and probable reserves, the underground measured and indicated resources under the Barvue pit and in the Gs zone which can potentially become mineral reserves and the stockpiled proven marginal ore. After year 10, this represents a tonnage of about 2 Mt grading 47.88 g/t Ag and 2.43 % Zn for a zinc-equivalent grade of 3.43 %. This tonnage could be increased further with inferred resources after additional exploration and development.

All components of the Abcourt-Barvue project have been developed and cost to a degree of accuracy of ±15 % and then, the cash flow and sensitivity analyses have been performed. The Abcourt-Barvue project is viable using realistic long-term metal price variations (the base case for which metal prices decrease over the 10-year production schedule) with a 60 %/40 % debt to equity ratio on an after-tax basis. In this case, Zn prices vary from 1.36 to 0.85 US$/lb for an average of 1.15 US$/lb Zn while the silver and gold average prices are 9.54 and 560 US$/oz respectively during the same period. The IRR, the net cash flow and the NPV (5 %) of the project are then in the order 20 %, 108 M$ and 83 M$, for a payback period of 5 years.

At constant metal prices and exchange rate over the 10-year period, the break even price for zinc, 60 % loan / 40 % equity, occurs at 0.70 US$/lb Zn; the corresponding silver and gold prices are 8.42 US$/oz Ag and 475 US$/oz Au and the rate of exchange is 1.20 C$/US$. On a 100 %-equity basis, the break even Zn price is then 0.66 US$/lb also considering capital expenditures. On a 100%-equity basis without considering any investment (EBITDA), the breakeven point occurs at a zinc price of 0.43 US$/lb.

The project is highly sensitive to metal prices and moderately sensitive to capital and operating expenditures. Therefore, higher metal prices would have a tremendous positive effect on the project (see scenarios 4, 5 and 6 in Table 13.2.1). Inversely, lower metal prices would have a negative effect on the viability of the project. If this happens, lower metal prices could be mitigated by increasing the zinc-equivalent cut-off grade to increase the zinc and silver grades fed to the mill.

15.2 Recommendations

Given that the Abcourt-Barvue project contains an economic mineral reserve and based on the cash flow analysis using realistic long-term metal prices variations, it is worthy to continue the development of the project through permit applications, detailed engineering and construction to extract 1 800 tpd of mined/milled ore to produce Ag-Au ingots and Zn-Ag concentrates for sale.

The authors considers that, by carrying out an optimization test work program, it will be possible to increase the metallurgical efficiency of the Abcourt concentrator over the estimated numbers used in this report. The main components of this optimization program would be as follows:

- cyanidation tests to verify if wet grinding would result into a lower NaCN consumption rate;
- parameters optimization of the cyanidation section of the concentrator which could raise the Ag recovery from the estimated 70 % to approximately 78 %;
- Zn-Ag flotation tests to improve the characteristics of the froth (bubbles observed as being very “skinny”) to larger bubbles most likely resulting into a higher concentrate Zn grade to 56 % (from 54.5 %) while maintaining the Zn recovery at 96 %;

- addition of the collector (3477) to the 2 minutes conditioning stage with $CuSO_4$ as applied previously (Matagami cycle tests on Abcourt-Barvue ore as well as in the former Barvue Mill) to test the LTM primary flotation procedure.

A major economic improvement to the project would be the reduction in capital costs realized by the purchase of used equipment for the process plant. Such equipment could come from one of two existing non-operating mills in the Abitibi region. Also, the use of former mill foundations could represent another saving. The authors strongly recommend to intensify the current discussions between Abcourt and the owners of those mills and to maximize the use of existing foundations.

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17. CERTIFICATES

MARC LAVIGNE, Eng., M.Sc.

Certificate of Author

I, Marc Lavigne, Eng., M.Sc., Senior Mining Engineer and Project Manager of

GENIVAR Limited Partnership
5355 Des Gradins Blvd
Quebec, Quebec, G2J 1C8

do hereby certify that:

1. I am a mining consultant contracted by Abcourt Mines Inc.
2. I am a graduate of Laval University, with a Baccalaureate in mine engineering. I also obtained a Master's Degree in Geostatistics also at Laval University. I am a consultant in mine engineering who has worked on a continuous basis since 1989, mainly providing services to the mining industry.
3. I am a mining consultant currently licensed by the Ordre des ingénieurs du Québec (License No. 99190).
4. I am the author of sections 2 to 4, 8 to 10 and 13, and portions of sections 1, 7, 11, 12 and 14 to 16 of this technical report entitled "Technical feasibility study for the Abcourt-Barvue project" and dated February 15, 2007.
5. I have visited the Abcourt-Barvue project on October 31, 2005.
6. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
7. I am an independent of the issuer applying all of the tests in sect 1.4 of NI 43-101.
8. I am a "qualified person" for the purposes of NI 43-101 due to my experience and current affiliation with a professional organization (Ordre des ingénieurs du Québec) as defined in NI 43-101.

DATED this 15th day of February, 2007

Marc Lavigne, Eng., M.Sc.

Marc Lavigne
99190

FLORENT BARIL, Eng., M Sc

Certificate of Author

I, Florent Baril, B.Sc., Senior Metallurgical Engineer and President of:

Bumigeme Inc.
1140 De Maisonneuve Blvd W,. Suite 1060
Montreal, Quebec H3A 1M8

Do hereby certify that:

1. I reside at 624 Jean Deslauriers, Condo 17, Boucherville, Quebec J4B-8P5
2. I am a graduate from Laval University, Quebec with a B. Sc. Degree in Metallurgy (1954), and I have practiced my profession for over 50 years.
3. I am a member of the "Ordre des ingénieurs du Québec" (O.I.Q) (Quebec Order of Engineers) (Membership Number 6972)
4. I am the Owner and President of Bumigeme Inc, a firm of consulting engineers, which has been incorporated in 1994.
5. I am a Qualified Person for the purpose of NI 43-101 based on my experience with feasibility studies on mining projects and the preparation of technical reports.
6. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this report.
7. I am the author of sections 5 and 6 and I have collaborated to sections 1, 7, 11, 12, 14, 15 and 16.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Abcourt Mines Inc. or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Abcourt Mines Inc or any associated or affiliated companies.

Dated this 29th day of January, 2007

(Signed and Sealed)

FLORENT BARIL
6972

Florent Baril, Eng.

APPENDIX 1

LTM test program and Techni-Lab QA/QC program

LABORATOIRE LTM Inc.
C.P. 1783 Val d'or J9P 6C5 Tél: (819) 825 9415 Fax: (819) 825 9430

Matériel: Abcourt : AB-05-18 **Date:** 08-01-07
F12545 à F12547 **No:** AB-46
F12548 à F12549

Test: cyanuration + flottation du zinc
Échantillon broyé à 94,8 % passant 200 mailles

Objectifs et traitements préliminaires: Cyanuration suivit d'une filtration
On fait aéré le rejet de cyanuration 1 heure avant de flotter le zinc.

		cyanuration		Flottation du zinc			
Réactifs		chaux	NaCN	Chaux	CuSO4	3477	77
Quantitée		3,5	5g				
consom.			0,35 Kg/t	2,29 kg/t	0,57 kg/t	0,07 kg/t	0,07 kg/t
pH initial		Durée:	36 h	pH initial		Durée:	10 min
final	12,48			final	11,0	% solide	40%

	Poids	Teneurs					
		Au (g/t)	Ag (g/t	Cu (%)	Zn (%)		
Alim. cal.	978	0,021	216,431		3,181		
Alim. Ana.							
Rejet total	827	0,006	17,30		0,09		
conc. Zinc	151	0,025	515,60		20,02		
sol. mère	650	0,014	112,30		0,02		
sol. Lav 1	1020	0,003	45,60		0,01		
Recup. Zn	15,44%	18,06%	36,78%		97,16%		
Rejet		23,75%	6,76%		2,29%		
Recupé.	cyan.	58,19%	56,46%		0,55%		
récup total	cyan+flot	76,25%	93,24%		97,71%		

Remarque: Mousse très abondante au début mais qui devient de moins en moins abondante avec le temps. On le qualifie d'acceptable

LABORATOIRE LTM inc
C.P. 1783 Val d'or J9P 6C5 Tél: (819) 825 9415 Fax: (819) 825 9430

Matériel: Abcourt : AB-05-18
F12545 à F12547
F12548 à F12549

Date: 08-01-07
No: AB-47

Test: cyanuration + flottation du zinc
Échantillon broyé à 94,8 % passant 200 mailles

Objectifs et traitements préliminaires: Cyanuration suivit d'une filtration
On fait aéré le rejet de cyanuration 1 heure avant de flotter le zinc.

		cyanuration		Flottation du zinc			
Réactifs		chaux	NaCN	Chaux	CuSO4	3477	DOW 250
Quantitée		3,5	5g				
consom.			0,25 Kg/t	2,29 kg/t	0,57 kg/t	0,07 kg/t	0,07 kg/t
pH initial		**Durée:**	36 h	**pH initial**		**Durée:**	10 min
final	12,51			**final**	11,0	**% solide**	40%

	Poids	Teneurs					
		Au (g/t)	Ag (g/t	Cu (%)	Zn (%)		
Alim. cal.	1012	0,015	222,601		3,359		
Alim. Ana.							
Rejet total	799	0,007	12,30		0,07		
conc. Zinc	213	0,010	421,30		15,62		
sol. mère	610	0,008	120,80		0,01		
sol. Lav 1	1020	0,003	51,00		0,01		
Recup. Zn	21,05%	13,60%	39,83%		97,88%		
Rejet		35,71%	4,36%		1,72%		
Recupé.	cyan.	50,69%	55,80%		0,41%		
récup total	cyan+flot	64,29%	95,64%		98,28%		

Remarque: Mousse moins abondante que pour l'AÉROFROTH 77 mais persistante jusqu'à la fin. On le qualifie de bon moussant.

LABORATOIRE LTM inc.

C.P. 1783 Val d'or J9P 6C5 Tél: (819) 825 9415 Fax: (819) 825 9430

Matériel: Abcourt : AB-05-19 **Date:** 08-01-07
F12530 à F12534 No: AB-48

Test: cyanuration + flottation du zinc
Échantillon broyé à 95,3 % passant 200 mailles

Objectifs et traitements préliminaires: Cyanuration suivit d'une filtration
On fait aéré le rejet de cyanuration 1 heure avant de flotter le zinc.

		cyanuration		Flottation du zinc			
Réactifs		chaux	NaCN	Chaux	CuSO4	3477	73
Quantitée		3,5	5g				
consom.			0,45 Kg/t	2,29 kg/t	0,57 kg/t	0,07 kg/t	0,07 kg/t
pH initial		**Durée:**	36 h	**pH initial**		**Durée:**	10 min
final	12,49			**final**	11,0	**% solide**	40%

	Poids	**Teneurs**					
		Au (g/t)	Ag (g/t	Cu (%)	Zn (%)		
Alim. cal.	1023	0,023	68,416		5,968		
Alim. Ana.							
Rejet total	775	0,014	14,90		0,16		
conc. Zinc	248	0,020	102,40		23,98		
sol. mère	510	0,008	33,70		0,01		
sol. Lav 1	1300	0,003	12,20		0,02		
Recup. Zn	24,24%	20,85%	36,28%		97,41%		
Rejet		45,61%	16,50%		2,07%		
Recupé.	cyan.	33,54%	47,22%		0,52%		
récup total	cyan+flot	54,39%	83,50%		97,93%		

Remarque: Mauvaise qualité de mousse. Équivalent au MIBC

LABORATOIRE LTM inc.
C.P. 1783 Val d'or J9P 6C5 Tél: (819) 825 9415 Fax: (819) 825 9430

Matériel: Abcourt **Date:** 08-01-07
Test: cyanuration + flottation du zinc **No:** AB-49
Échantillon broyé à 95,3 % passant 200 mailles

Objectifs et traitements préliminaires: Cyanuration suivit d'une filtration
On fait aéré le rejet de cyanuration 1 heure avant de flotter le zinc.

		cyanuration		Flottation du zinc			
Réactifs		chaux	NaCN	Chaux	CuSO4	3477	65
Quantitée		3,5	5g				
consom.			0,55 Kg/t	2,29 kg/t	0,57 kg/t	0,07 kg/t	0,07 kg/t
pH initial		Durée:	36 h	pH initial		Durée:	10 min
final	12,4			final	11,0	% solide	40%

	Poids	Teneurs					
		Au (g/t)	Ag (g/t	Cu (%)	Zn (%)		
Alim. cal.	1009	0,022	69,378		6,534		
Alim. Ana.							
Rejet total	702	0,012	9,60		0,12		
conc. Zinc	307	0,021	104,40		21,12		
sol. mère	530	0,008	33,80		0,03		
sol. Lav 1	1090	0,003	12,20		0,01		
Recup. Zn	30,43%	28,81%	45,79%		98,35%		
Rejet		37,64%	9,63%		1,32%		
Recupé.	cyan.	33,56%	44,59%		0,33%		
récup total	cyen+flot	62,36%	90,37%		98,68%		

Remarque: Qualité de mousse excellente tout au long de la flottation

LABORATOIRE LTM inc.
C.P. 1783 Val d'or J9P 6C5 Tél: (819) 825 9415 Fax: (819) 825 9430

Matériel:	Abcourt :	AB-05-40 F12913 à F12918 F12933 à F12924	Date: No:	08-01-07 AB-50
Test:	cyanuration + flottation du zinc Échantillon broyé à 95,5 % passant 200 mailles			

Objectifs et traitements préliminaires: Cyanuration suivit d'une filtration
On fait aéré le rejet de cyanuration 1 heure avant de flotter le zinc.

		cyanuration		Flottation du zinc			
Réactifs		chaux	NaCN	Chaux	CuSO4	3477	76A
Quantitée		3,5	5g				
consom.			0,90 Kg/t	2,29 kg/t	0,57 kg/t	0,07 kg/t	0,07 kg/t
pH initial		Durée:	36 h	pH initial		Durée:	10 min
final	12,48			final	11,0	% solide	40%

	Poids	Teneurs					
		Au (g/t)	Ag (g/t	Cu (%)	Zn (%)		
Alim. cal.	1014	0,054	39,007		1,629		
Alim. Ana.							
Rejet total	875	0,011	5,30		0,06		
conc. Zinc	139	0,123	48,00		11,40		
sol. mère	640	0,032	27,10		0,01		
sol. Lav 1	1000	0,008	10,90		0,00		
Recup. Zn	13,71%	30,97%	16,87%		95,94%		
Rejet		17,44%	11,72%		3,39%		
Recupé.	cyen.	51,59%	71,41%		0,67%		
récup total	cyen+flot	82,56%	88,28%		96,61%		

Remarque: Mousse abondante mais peu chargé. Qualité de mousse excellente

LABORATOIRE LTM inc.
C.P. 1783 Val d'or J9P 6C5 Tél: (819) 825 9415 Fax: (819) 825 9430

Matériel: Abcourt : AB-05-40 Date: 08-01-07
F12913 à F12918 No: AB-51
F12933 à F12924

Test: cyanuration + flottation du zinc
Échantillon broyé à 95,5 % passant 200 mailles

Objectifs et traitements préliminaires: Cyanuration suivit d'une filtration
On fait aéré le rejet de cyanuration 1 heure avant de flotter le zinc.

		cyanuration		Flottation du zinc			
Réactifs		chaux	NaCN	Chaux	CuSO4	3477	88
Quantitée		3,5	5g				
consom.			1,10 Kg/t	2,29 kg/t	0,57 kg/t	0,07 kg/t	0,07 kg/t
pH initial		Durée:	36 h	pH initial		Durée:	10 min
final	12,35			final	11,0	% solide	40%

	Poids	Teneurs					
		Au (g/t)	Ag (g/t	Cu (%)	Zn (%)		
Alim. cal.	1011	0,051	35,609		1,552		
Alim. Ana.							
Rejet total	954	0,022	6,10		0,14		
conc. Zinc	57	0,221	58,30		24,80		
sol. mère	620	0,016	25,90		0,01		
sol. Lav 1	1000	0,008	10,80		0,01		
Recup. Zn	5,64%	24,46%	9,23%		90,08%		
Rejet		40,75%	16,16%		8,69%		
Recupé.	cyan.	34,79%	74,60%		1,23%		
récup total	cyan+flot	59,25%	83,84%		91,31%		

Remarque: Presqu'aucune mousse, on le qualifie donc de mauvais.

TECHNI-LAB
S.G.B. Abitibi inc.
Pyroanalyse, Géochimie, Environnement
Certifié ISO/CEI 17025

Corporate Profile

Techni-Lab S.G.B. Abitibi inc. is a private assay laboratory, specialised in gold and base metals assay, as well as environmental analysis.

Historical facts

In 1988, a group of local investors founded Techni-lab S.G.B. Abitibi inc.; their goal was to offer analytical services to the mining industry by performing gold and base metal assay. Over the years, an environmental section was added, in response to our clients need. Both the assay and environment sections had seen their expertise growing every year, adding gravimetric analysis (gold), titrimetric determinations (zinc and copper concentrates) for example. Most of our employees have 10 or more years of experience with the Techni-lab family.

Since 2001, a new owner permitted us to renew almost all of our analytical instruments, including two atomic absorption spectrometer and a graphite furnace. This new material helped us greatly in obtaininig a reputation for low detection limits, good turn around time and competitive prices.

Contacts

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Mrs Lucie Désaulniers (General manager)
M. Serge Larochelle ((Operation manager, geochemistry)
M. Marius Deac (Laboratory supervisor)
M. André Caouette (Operation manager (environment) and marketting)

Our activities

Our expertise include :

- *Gold (fire assay, AA finish)*
- *Gold (fire assay, gravimetric finish)*
- *Base metals (multi-acids digestion, AA or GFAA finish)*
- *Zinc and copper concentrates (titrimetric)*
- *Specific gravity of core samples, etc…*

Techni-Lab is ISO 17025 certified for many environmental analysis (water and wastewater).

Data transfer and confidentiality

Results can be delivered electronically or by fax, in the format of your choice. Every employee of Techni-Lab must sign, on his/her first day on the job, a non disclosure form, forbidding any transfer of information to a third party.

Technical expertise

15 to 25 employees, including 4 technicians and a chemist, acting as laboratory supervisor. The responsibility of the quality control falls on the section managers.

Certification

Techni-lab is accredited for environmental analysis with ''Le Ministère du Développement Durable, de l'Environnement et des Parcs''; we participate to the PTP-MAL since 1998, on a voluntary basis. We plan to proceed with the certification 17025 (Standards Counsil of canada) in 2007.

André Caouette *Operation manager*	*Lucie Désaulniers* *General manager*

TECHNI-LAB S.G.B. ABITIBI INC.

RÉFÉRENCES ET PROCÉDURES DU DÉPARTEMENT DE GÉOCHIMIE

TECHNI-LAB S.G.B. ABITIBI INC.

Mise à jour

Juin 2006

TABLE DES MATIÈRES

RÉCEPTION ET PRÉPARATION DES ÉCHANTILLONS

Voici les différentes étapes de manutention des échantillons avant l'analyse. Des procédures simples sont suivies pour prévenir les erreurs ou la perte d'échantillons. Des instructions sont également données pour éviter la contamination de ceux-ci.

Réception et concassage des échantillons

Lorsqu'un lot d'échantillon est reçu, ceux-ci sont classés et comptés. La liste ainsi produite, (feuille de projet) se voit attribuer un numéro d'entrée (# de projet). Cette liste est ensuite comparée à la demande d'analyse fournie par le client. ***Toute anomalie (par exemple : échantillon manquant ou surnuméraire, identification douteuse, contamination inter-échantillons) doit être immédiatement signalée au chef d'équipe et au superviseur. Ce dernier contactera le client concerné dans les plus brefs délais, afin de décider avec lui des mesures à prendre pour rectifier la situation.***

De plus, chaque échantillon doit être accompagné de deux étiquettes d'identification (TAG). La première accompagnera la portion d'échantillon pulvérisée (pulpe) et la seconde avec le reste de l'échantillon concassé (rejet).

- Les échantillons sont classés par ordre de priorité et disposés dans les casseroles par ordre numérique. Une table comprend 4 rangées de 12 casseroles numérotées de 1 à 48.

- Les échantillons humides sont séchés au four durant une heure.

- Les sacs destinés à recevoir les échantillons sont identifiés d'après le numéro de projet et de l'échantillon.

- Les échantillons sont concassés au complet. Le concasseur à mâchoires permet d'obtenir une grosseur de particules assez grossières (maximum 1/8). L'échantillon concassé est par la suite passé plusieurs fois sur un séparateur, afin de limiter la masse à broyer tout en homogénéisant l'échantillon.

- La masse d'échantillon concassé retenue pour la pulvérisation varie de 200 à 300 grammes.

Pulvérisation des échantillons

- Un sac de papier est identifié pour recevoir chaque échantillon.

- Les plats et les anneaux sont conditionnés avec la silice avant de commencer la pulvérisation ce qui permet de nettoyer le plat et les anneaux et ainsi, éviter les contaminations entre les échantillons.

- Chaque échantillon est pulvérisé de 2 à 3 minutes de façon à obtenir une pulpe très fine (environ 80 % à 200 mesh).

- L’échantillon peut ensuite être homogénéisé et soumis à la pyro-analyse.

Pyro-analyse des échantillons

Selon la nature de l’échantillon, le technicien peut devoir varier les quantités d’additifs.

- Un formulaire de données est rempli et les sacs de pulpes sont numérotés en suivant l’ordre indiqué sur le formulaire.
- Une série de 24 creusets est préparée incluant blanc, duplicata et étalon de référence qui seront répartis à intervalle de 7 échantillons.
- Les creusets sont remplis de 175 grammes de fondant #2 avec une cuillère de farine.
- Une portion de masse connue d’échantillon est pesée et ajoutée au fondant et à la farine dans les creusets. La masse d’échantillon pesée est de 15 grammes pour les analyses en grammes par tonnes et de 30 grammes pour les analyses en partie par milliard.
- Le mélange de chaque creuset doit ensuite être homogénéisé.
- Une solution de nitrate d’argent, composée de 25 grammes de nitrate d’argent dans 500ml d’eau distillée et déminéralisée, est ajoutée à raison de deux gouttes pour les analyses en parties par milliards (ppb) et cinq gouttes pour les analyses en grammes par tonnes. Le tout est recouvert de borax pour empêcher les éclaboussures durant la fusion.
- Les échantillons sont enfournés pour la fusion, par série de vingt-quatre. La fusion dure quarante-cinq minutes à une température de 1093ºC.
- Ensuite, les échantillons liquéfiés sont versés dans des lingotières et refroidis à l’air. Ils sont recouverts pour éviter les éclaboussures de scories.
- Le refroidissement terminé, il faut marteler les culots obtenus pour en séparer la scorie et en faire un cube qui pourra être envoyé en coupellation.
- Les coupelles d’os de moutons sont préchauffées durant dix minutes avant d’introduire les culots de forme cubique. La coupellation dure environ une heure à température de 954ºC.
- Lorsque la coupellation est terminée, les billes d’or et d’argent obtenues sont refroidies. Elles peuvent enfin être analysées par spectroscopie d’absorption atomique ou gravimétrie.

LES ANALYSES

La pyro-analyse sert à extraire l'or de la gangue séchée et pulvérisée. Suite au processus, l'or se présente alors sous forme d'une bille d'or et d'argent. Cette bille peut être attaquée pour être analysée gravimétriquement ou par spectroscopie par absorption atomique.

La concentration de l'or peut être exprimée en grammes par tonnes métriques (g/t), en onces par tonnes métriques (oz/t) ou en parties par milliards (ppb). Les masses d'échantillons utilisées pour les analyses en grammes par tonne sont habituellement de 15 grammes et pour les analyses en ppb, elles sont habituellement de 30 grammes. L'unité de masse arbitrairement utilisée dans l'industrie minière est «Assay/ton» qui équivaut à 30 grammes. Un demi «Assay/ton» équivaut à 15 grammes.

Les métaux peuvent être analysés directement par dissolution de la gangue séchée et pulvérisée. La masse d'échantillon normalement utilisée pour déterminer les métaux est approximativement de deux grammes quelquefois de un gramme et de un demi gramme pour les standards. La concentration des métaux est exprimée en parties par millions (ppm) ou en pourcentage (%).

LA PYRO-ANALYSE

La pyro-analyse sert à extraire l'or de la matrice rocheuse, pour pouvoir en déterminer la concentration. La méthode se résume à fusionner du minerai avec de l'oxyde de plomb et des agents réducteurs. Un alliage de plomb, contenant de l'or et de l'argent coule alors dans le fond de l'échantillon du creuset, la scorie vitreuse étant moins dense que le plomb. Le culot de plomb refroidi ainsi obtenu est dégagé de la scorie solidifiée et fusionnée dans une coupelle, qui absorbera le plomb en laissant une bille d'or et d'argent.

La fusion en creuset

Voici une liste des réactifs utilisés pour la fusion :

La litharge (PbO)

Oxyde de plomb fondu et cristallisé de couleur rouge-orangée. C'est un agent oxydant et désulfurant. Sa température de fusion est de 883°C. En se réduisant, la litharge fournie le plomb qui absorbera l'or et l'argent. Elle se combine facilement à la silice et l'alumine pour former des silicates et des aluminates fusibles.

$$2\ PbO + C \rightarrow 2\ Pb\downarrow + CO_2\uparrow$$
$$PbO + SiO_2 \rightarrow PbSiO_3$$
$$PbO + Al_2O_3 \rightarrow PbAl_2O_4$$

Le carbonate de sodium ($NaCO_3$)

Ce produit est communément appelé du soda. Il possède une température de fusion de 852°C. C'est un fondant basique qui réagit avec la silice et l'alumine pour former des silicates et des aluminates complexes avec les oxydes métalliques.

$$Na_2CO_3 + SiO_2 \rightarrow Na_2SiO_3 + CO_2\uparrow$$
$$Na_2CO_3 + Al_2O_3 \rightarrow Na_2Al_2O_4 + 2\ CO_2\uparrow$$

Le borax ($Na_2B_4O_7$)

Le borax est un fondant acide utilisé pour dissoudre et se combiner avec les constituants basiques présents dans la gangue et ainsi, former des borates complexes facilement fusibles. Il est à remarquer que certains constituants acides se dissolvent également en présence de borax notamment la silice.

$$Na_2B_4O_7 + 2\ CaO \rightarrow Na_2O{\bullet}2\ CaO{\bullet}2\ B_2O_3$$

La silice (SiO_2)

La silice est un fondant acide très efficace. Elle réagit avec les oxydes métalliques dont la litharge, et produit ainsi des silicates fusibles.

La farine et l'amidon

Ce sont des agents réducteurs, contenant du carbone, qui contribue à réduire la litharge en plomb.

Le fer (Fe)

Le fer est quelquefois utilisé comme agent réducteur et désulfurant. Il attaque les sulfures métalliques pour donner des métaux et du sulfure de fer.

Le nitrate de potassium (KNO_3)

Le nitrate de potassium est un agent oxydant. Il est ajouté lorsqu'il y a un trop grand excès de substances réductrices dans la gangue.

$$4\ KNO_3 + 5\ C + 2\ SiO_2 \rightarrow 2\ K_2SiO_3 + 5\ CO_2\uparrow + 2\ N_2\uparrow$$

La fluorine (CaF_2)

La fluorine améliore la fluidité de la scorie.

La nature des minerais

Minerai contenant des oxydes :

Avec du minerai contenant des oxydes, on ajoutera plus d'agents réducteurs pour obtenir le plomb et réduire les métaux.

Minerai sulfureux :

Pour des minerais sulfureux, il y aura une réduction de PbO car les sulfures sont des réducteurs. En fait, à cause de la grande quantité de sulfures, il est nécessaire d'ajouter un agent oxydant pour éviter une trop grande formation de plomb.

$$PbS + 3\ PbO + Na_2CO_3 \rightarrow 4\ Pb + Na_2SO_4 + CO_2\uparrow$$
$$ZnS + 4\ PbO + Na_2CO_3 \rightarrow 4\ Pb + Na_2SO_4 + ZnO + CO_2\uparrow$$
$$2\ FeS_2 + 15\ PbO + 4\ Na_2CO_3 \rightarrow 15\ Pb + Fe_2O_3 + 4\ Na_2SO_4 + 4\ CO_2\uparrow$$

La pyrite de fer, étant très réductrice, elle produira une trop grande quantité de plomb pour la cupellation ultérieure. Le nitrate de potassium est alors utilisé comme agent oxydant.

$$2\ KNO_3 + 6\ SiO_2 + 5\ Pb \rightarrow 5\ PbSiO_3 + K_2SiO_3 + N_2\uparrow$$
$$2\ FeS_2 + 2\ SiO_2 + 6\ KNO_3 \rightarrow Fe_2(SO_4)_3 + K_2SO_4 + 2\ K_2SiO_3 + 3\ N_2\uparrow$$

Mélange commun utilisé pour la fusion en creuset :

- Minerai (15g)
- Soda (25 à 35g)
- Borax (10 à 15g)
- Farine (varie selon la nature de la matrice)
- KNO_3 (varie selon la nature de la matrice)
- Litharge (60 à 75g)

Description de la fusion en creuset

Les mélanges d'échantillons et de réactifs sont contenus dans des creusets fait d'argile réfractaire. La fusion s'effectue dans un four à moufle ou dans un four d'essai. La chambre de fusion est constituée de briques réfractaires et d'une plaque d'enfournement en carbure de silicium. Ce réceptacle est ventilé par l'arrière et chauffé par des éléments de carbure de silicium, installés sous la plaque d'enfournement.

On traite une quantité connue de minerais, habituellement 15 ou 30 grammes, avec de la litharge et les autres réactifs nécessaires dans un creuset en argile réfractaire. Les réactifs sont choisis selon la nature de la matrice du minerai. Ils peuvent être sulfureux, acides, basiques, neutres ou contenir des oxydes. Il est donc nécessaire de bien connaître la nature de la matrice du minerai. Lors de la fusion, la litharge est réduite en plomb. L'or et l'argent sont alors absorbés par les gouttelettes de plomb fondu qui migrent vers le fond du creuset.

La fusion s'effectue à 1050°C. Au commencement, il y a réduction de la litharge, un début de réaction du nitrate de potassium ainsi que la réduction partielle des oxydes. Le mélange, qui a été placé dans le creuset et bien brassé, commence à fondre.

Ensuite, arrivent les réactions plus violentes. La farine, les sulfures et les autres réducteurs réduisent la litharge, les tellurures d'or et les sulfures d'argent en libérant les métaux qui sont entraînés vers le fond du creuset. Le carbonate de sodium et le borax réagissent pour produire la scorie dans laquelle les autres oxydes et l'alumine se dissolvent. Il y alors un violent dégagement de gaz contenant notamment du CO_2, CO, SO_2 et N_2.

Finalement, les réactions se terminent et la scorie se liquéfie davantage. Les petites gouttelettes de plomb peuvent migrer au fond du creuset en entraînant avec elles l'or et l'argent.

Le temps nécessaire à la fusion est de 40 à 55 minutes, pendant lesquelles la porte du four est fermée. La température doit être soigneusement maintenue puisque, si elle est trop haute, il y a danger de volatilisation des composés d'or et d'argent. Par contre, si la température est trop basse, le culot de plomb est trop petit, ce qui fait que l'or et l'argent n'auront pas été complètement collectés. Après la fusion, les creusets sont vidés dans des lingotières. Après refroidissement, la scorie est brisée et le culot de plomb est récupéré en le martelant pour éliminer les traces de scorie. Le culot peut enfin être envoyé en coupellation.

La coupellation

L'or et l'argent sont séparés du plomb dans une coupelle à base de phosphate de calcium, obtenu par la calcination d'os de mouton. Lorsque le culot de plomb est placé dans la coupelle, il est chauffé dans un four à moufle avec la porte initialement fermée. Lorsque la porte est ouverte, la litharge se reforme à partir du plomb, par oxydation. La température du four doit demeurer autour de 880°C. La litharge qui se forme, ne doit pas faire une croûte sur la surface de la coupelle, mais elle doit imbiber ses pores en restant fluide. Une croûte se forme lorsque la coupelle a été placée dans le four à une température trop basse.

Il faut donc préchauffer le four à 900°C durant 10 minutes avant l'introduction de la coupelle, pour éviter ce problème. Lorsque la fusion de la litharge s'effectue, et que celle-ci disparaît dans les pores de la coupelle, il faut descendre la température du four à 780°C, puisque l'oxydation du plomb est très exothermique, et que cela pourrait provoquer la volatilisation de l'or. La litharge semble donc disparaître dans la coupelle jusqu'à ce qu'il ne reste, au fond de la coupelle, qu'une petite bille métallique composée d'or et d'argent. Le temps de coupellation ne doit pas dépasser le point d'étincelle. C'est-à-dire, le point où la bille prend un aspect étincelante, car la bille d'or a tendance à se volatiliser quand il n'y a plus de plomb. Du bismuth peut laisser sur la coupelle un anneau d'apparence caractéristique. Du cuivre, bien que facilement oxydable, peut également se retrouver dans la bille.

ANALYSE DE L'OR PAR LA MÉTHODE GRAVIMÉTRIQUE

La gravimétrie consiste à déterminer la quantité d'or par des pesées successives après avoir obtenu la bille d'or et d'argent par la pyro-analyse (fire assay), puis en ayant séparé ses constituants par attaque à l'acide nitrique.

La séparation de l'or et de l'argent est effectuée par attaque à l'acide nitrique, qui transforme l'argent en nitrate d'argent soluble, mais qui reste inactif sur l'or. L'or forme alors un agglomérat qui peut être lavé et pesé. La séparation est bonne quand l'alliage contient au moins deux fois plus d'argent que d'or. Empiriquement, la meilleure concentration d'acide nitrique pour cette attaque a été déterminée comme étant une dilution par cinq. Plus concentré, la réaction serait trop violente et l'or serait pulvérisé, ce qui rendrait sa pesée difficile.

La séparation est effectuée dans des creusets de porcelaine, avec quelques millilitres d'acide. Après 20 minutes de réaction, la solution acide est décantée dans une casserole blanche pour éviter toute perte d'or. L'acide est éliminé et l'or est lavé trois fois avec de l'eau sans chlore. Après le chauffage et le refroidissement, l'or est pesé sur une balance de précision au cinq millièmes de milligrammes. La masse de l'or est alors déduite directement, et celle de l'argent, par la différence de masse avant et après l'attaque.

Il est à noter qu'à cause de l'effet de pépite, il y a normalement de fortes variations entre les résultats de plusieurs analyses sur le même échantillon.

Procédure expérimentale :

1. Après la pyro-analyse, il faut ramasser les billes dans les creusets et les aplatir délicatement avec un marteau.

2. Faire une digestion avec un volume de 5 ml d'acide nitrique à 20 % et chauffer sur une plaque pendant 30 minutes.

3. Aspirer la partie liquide, dans laquelle se trouve le nitrate d'argent, dans le creuset.

4. Rinser trois fois avec une solution d'ammoniaque dans de l'eau distillée et déminéralisée, dans un rapport un pour neuf.

5. Remettre sur la plaque chauffante pour sécher la bille d'or.

6. Passer la bille d'or à la flamme pour en réduire les oxydes.

7. Procéder à la pesée.

Calibration de la balance gravimétrique :

1. Lever les plateaux et enlever les disques métalliques des plateaux.
2. Baisser les plateaux et appuyer sur la touche «autotarer». Il y aura apparition de 4 chiffres après le point. L'appareil se tare automatiquement en affichant 0,000. Les chiffres disparaissent automatiquement et l'échelle de pesenteur change à 200 mg.
3. Lever les plateaux et mettre le poids de 100 milligrammes sur le plateau se situant à l'avant de la balance gravimétrique.
4. Sur le clavier de la balance, il faut inscrire le chiffre 100,00 mg et peser sur la touche «calibration».
5. Baisser les plateaux et attendre que le 100,00 mg disparaisse de l'écran digital.
6. Remonter les plateaux et enlever le poids de 100,00 mg et remettre les disques métalliques sur les plateaux. Automatiquement, l'échelle de pesenteur se fixe à 200 mg et le nombre de chiffres après le point est de trois (0,000 mg).
7. Peser sur la touche «autotarer» et peser les billes d'or.

Calcul en ppm ou g/t

Concentration en oz/t :

$$\frac{\text{Pesée de la bille (par gravimétrie) en mg X } 29{,}167}{\text{Masse de l'échantillon utilisé pour la fusion en g}}$$

Exemple :

$$\frac{0{,}042 \text{ mg X } 29{,}167}{15\text{g}} = 0{,}082 \text{ oz/t}$$

Calcul en ppm ou g/t

Concentration en ppm :

$$\frac{\text{Pesée de la bille (par gravimétrie) en mg X } 1000}{\text{Masse de l'échantillon utilisé pour la fusion en g}}$$

Exemple :

$$\frac{0{,}042 \text{ mg X } 1000}{15\text{g}} = 2{,}8 \text{ ppm}$$

ANALYSE DE L'OR PAR SPECTROSCOPIE AA

Suite à l'obtention de la bille par pyro-analyse, celle-ci est dissoute dans de l'acide nitrique et chlorhydrique. La détermination de la concentration en or est ensuite obtenue par lecture sur spectroscopie d'absorption atomique.

Teneur en ppb

1. La bille d'or et d'argent est introduite dans un tube de 5 ml.

2. 0,5 millilitre d'acide nitrique 50 % est ajouté. Le tout est chauffé dans un bain marie durant 30 minutes.

3. 1 millilitre d'acide chlorhydrique est ajouté. Le tout est chauffé de nouveau dans un bain marie durant 15 minutes.

4. Finalement, le volume est complété à 5 ml avec de l'eau du robinet, qui contient naturellement du calcium et du sodium. L'échantillon est mélanger, puis analysé par spectroscopie en absorption atomique sur flamme.

Note : La limite de détection de la méthode donne 5 ppb.

Calcul en ppb

Concentration en ppb :

$$\frac{\text{Absorbance X volume utilisé en ml X 1000}}{\text{Masse de l'analyse en g}}$$

Exemple :

$$\frac{0{,}5 \text{ X } 5 \text{ ml X } 1000}{30\text{g}} = 83 \text{ ppb}$$

Teneur en g/t

1. La bille d'or et d'argent est introduite dans un tube de 10 ml.

2. Un millilitre d'acide nitrique à 50 % est ajouté. Le tout est chauffé dans un bain marie durant 30 minutes.

3. 2 ml d'acide chlorhydrique sont ajoutés. Le tout est à nouveau chauffé dans un bain marie durant 15 minutes.

4. Le volume est finalement complété à 10 ml avec de l'eau du robinet, qui contient naturellement du calcium et du sodium. L'échantillon est finalement mélangé, puis analysé par spectroscopie en absorption atomique sur flamme.

Note : La limite de détection de la méthode donne 0,06 g/t.

Calcul en g/t

Concentration en g/t :

$$\frac{\text{Valeur de l'absorbance X volume utilisé en ml}}{\text{Masse de l'échantillon en g}}$$

Exemple :

$$\frac{1{,}0 \text{ X } 10 \text{ ml}}{15\text{g}} = 0{,}66 \text{ g/t}$$

Teneurs en oz/t

La procédure expérimentale est la même que celle utilisée pour la teneur en g/t. Le même calcul s'applique avec un facteur de conversion.

1 g/t = 0,0292 oz/t

L'exemple précédent donnera en oz/t : 0,66 g/t X 0,0292 = 0,019 oz/t

Note : La limite de détection de la méthode donne 0,002 oz/t.

DÉTERMINATION DES MÉTAUX AUTRES QUE L'OR

L'analyse des métaux autres que l'or s'effectue en attaquant le minerai pulvérisé et séché, par l'acide nitrique et chlorhydrique, puis en déterminant la concentration de cette solution par spectroscopie d'absorption atomique.

1. Une masse de 0,5 à 2 grammes d'échantillon est pesée dans un bécher de 250 ml. Une pesée de 0,5 gramme peut être suffisante pour les échantillons très concentrés.

2. Le minerai est digéré dans un mélange de 5 ml d'acide nitrique et 15 ml d'acide chlorhydrique. Il peut être nécessaire d'ajouter 1 ml de mixture de brome (brome + KBr) pour digérer complètement le minerai, si l'échantillon est très concentré.

3. Le mélange est couvert d'un verre de montre brassé et chauffé sur une plaque chauffante à feu moyen jusqu'à sécheresse, refroidir et ajouter 25 ml d'acide chlorhydrique et réchauffer 5 minutes.

4. Le mélange est ensuite transféré dans un ballon volumétrique en s'assurant de bien rincer complètement le bécher, le verre de montre avec de l'eau distillée et déminéralisée.

5. Le volume du ballon est complété avec de l'eau distillée et déminéralisée, puis remis dans le bécher.

6. Enfin, les métaux sont analysés par spectroscopie en absorption atomique sur flamme.

Note : La limite de détection de la méthode donne 0,5 ppm pour une masse d'échantillon de 2 g.

Calcul en ppm

Concentration en ppm :

$$\frac{\text{Valeur de l'absorbance X volume en ml}}{\text{Masse de l'échantillon en g}}$$

Exemple :

$$\frac{0{,}54 \text{ X } 100 \text{ ml}}{2{,}056} = 26 \text{ ppm}$$

PRÉPARATION DES STANDARDS DE CALIBRATION UTILISÉS EN SPECTROSCOPIE AA

Les solutions standard utilisées par la spectroscopie en absorption atomique sur flamme sont préparées en diluant un certain volume d'une solution plus concentrée dans des ballons de 100ml.

Standard (ppm)	Solution originale (ppm)	Volume à ajouter (ml)	Volume à compléter (ml)
100	1000	10	100
50	1000	5	100
20	100	20	100
10	100	10	100
5	100	5	100
3	100	3	100
1	10	10	100

Les solutions d'or :

Il faut ajouter 5 ml d'acide chlorhydrique dans les ballons de 100 ml et les compléter avec de l'eau froide du robinet.

Les solutions d'argent :

Il faut ajouter 25 ml d'acide chlorhydrique dans les ballons de 100 ml et les compléter avec de l'eau distillée et déminéralisée.

Les solutions de métaux :

Il faut ajouter 5 ml d'acide chlorhydrique dans les ballons de 100 ml et les compléter avec de l'eau distillée et déminéralisée.

Le blanc pour l'or :

Il suffi de faire une solution contenant de l'eau du robinet et de l'acide chlorhydrique.

Le blanc pour les métaux :

Il suffit de faire une solution contenant de l'eau distillée et déminéralisée avec un peu d'acide nitrique.

ANALYSE DE L'OR GROSSIER

L'analyse d'or grossier consiste à effectuer une mesure de l'or sur un échantillon préalablement fractionné en une partie grossière (+ 150 mesh) et une partie pulpe (- 150 mesh). Cette méthode permet de déterminer la proportion d'or natif et d'or disséminé dans l'échantillon. Il est possible d'utiliser un tamis différent au besoin (200 mesh, par exemple).

Procédure expérimentale

1. L'échantillon doit être pesé au complet, après le séchage, en tenant compte du poids du sac de papier et de la casserole.

2. L'échantillon doit ensuite être complètement broyé .

3. Une masse d'environ 350 grammes d'échantillon est prélevée. Il faut en mesurer la masse exacte.

4. Cette partie doit être pulvérisée de telle façon qu'environ 80 % de l'échantillon passe dans un tamis de 150 mesh. Cette partie est conservée dans un sac, sur lequel le numéro de l'échantillon y est inscrit avec un signe moins (-). Durant la pulvérisation, il faut prendre garde à limiter le plus possible, les pertes d'échantillons. Cette partie est la pulpe.

5. L'autre partie est transférée dans un sac, sur lequel est inscrit le numéro de l'échantillon avec un signe plus (+). Ceci est la partie métallique.

6. Les deux parties de l'échantillon sont alors pesées.

7. Finalement, l'échantillon est prêt pour procéder à la pyro-analyse, dans des creusets de 30 grammes A.P. Green. La partie métallique (+150 mesh) est analysée au complet tandis que la pulpe (-150 mesh) est analysée en double ou en quadruple selon la demande. La quantité d'échantillon requise pour la fusion est de 30 grammes.

Exemple de calcul

Poids de la pulpe = 300g
Poids de la partie métallique = 50g
Fusion moyenne de la pulpe = 1250 ppb
Fusion moyenne de la partie métallique = 1000 ppb

Le % de masse de la pulpe est de 100 X 300 / 350 = 85,7%
Le % de masse de la partie métallique est de 100 X 50 / 350 = 14,3%
La teneur pondérée pour la pulpe est de 0,857 X 1250 = 1071,25
La teneur pondérée pour la partie métallique est de 0,143 X 1000 = 143,00
La teneur pondérée de l'échantillon est de 1071 + 143 = 1214 ppb.

OR PAR ÉPONGE

Cette méthode, adaptée de la méthode Holtz, consiste à extraire l'or d'une solution par une réduction de plomb sous forme d'agglomérat dans celle-ci.

Matériel

- Béchers de 500 ml ou de 1 litre
- Cylindres gradués de 50 ou 500 ml
- Plaquettes d'aluminium
- Plaque chauffante

Réactifs

Solution de nitrate d'argent

Elle est composée de 25 grammes de nitrate d'argent dans 500 ml d'eau distillée et déminéralisée.

Tampon d'acétate de plomb

Le tampon est fait en solubilisant 500 grammes d'acétate de plomb dans 500 ml d'eau du robinet, en chauffant, au besoin. La solution obtenue est transférée dans une bouteille Winchester de deux litres et demie, complétée à 2,3 litres environ. Enfin, il faut ajouter 100 ml d'acide acétique glacial et bien agiter.

Procédure

1. Il faut mesurer, à l'aide du cylindre de 500 ml, 350 ml d'échantillon dans un bécher de cinq cent millilitres ou six cents millilitres d'échantillon dans un bécher de un litre.

2. Il faut ensuite ajouter dans l'ordre, cinq à six gouttes de solution de nitrate d'argent, quarante millilitres de tampon d'acétate de plomb, une plaquette d'aluminium, mélanger, ajouter dix millilitres d'acide chlorhydrique concentré et faire chauffer sur une plaque chauffante dont le thermostat est à huit ou neuf c'est-à-dire, à forte intensité.

3. La solution est chauffée jusqu'à ce que le plomb forme une éponge, à la surface de la solution, ou sur la plaquette d'aluminium, et que la solution devienne claire. Le mélange est alors retiré du feu.

4. Après refroidissement, l'éponge de plomb est recueillie, en prenant bien soin de porter des gants de latex essorés et déposés sur un papier absorbant. L'éponge est alors séchée sur une plaque chauffante ou à l'air.

5. Lorsque l'éponge est bien sèche, elle peut être envoyée en coupellation. Les résultats sont reportés en partie par milliards.

Calcul en ppb

Concentration d'or en µg/L ou ppb :

$$\frac{1000 \text{ X } V_2 \text{ X Lecture de l'appareil en ppb}}{V_1}$$

V_1 : Volume original d'échantillon utilisé, soit 350 ou 600 ml.
V_2 : Volume en millilitres de la solution obtenue par la dissolution de la bille.

LE CONTRÔLE DE LA QUALITÉ

L’or et les métaux sont analysés par série de 21 échantillons, accompagnés par un blanc dans son premier tiers, un double dans le second tiers et un standard dans le troisième tiers. La position de chacun est incrémentée d’une position, d’une série à l’autre et revient au début après la huitième série.

Le blanc sert à déceler une contamination. Le double sert à vérifier la reproductibilité de la méthode. Le standard est un échantillon de concentration connue.

Il y a trois types de standards utilisés pour l’or :

- Le standard en parties par milliards (Rocklab)
- Le standard en grammes par tonnes métrique (Rocklab)
- Un standard certifié CANMET pour les vérifications périodiques.

Il y a trois types de standards utilisés pour les métaux :

- Le standard maison pour les métaux.
- Le standard concentré, étalonné chez Techni-Lab.
- Le standard certifié CANMET pour les métaux.

La vérification des standards se fait à tous les mois pour l’or et les métaux sur une série de vingt-quatre échantillons. La série pour l’or comprend sept standards maison en g/t, sept standards maison en ppb, sept standards certifiés et trois blancs intercalés dans la série. La série pour les métaux comprend onze standards maison, onze standards certifiés et deux blancs intercalés dans la série.

Le calcul de chaque standard est calculé en faisant la moyenne des valeurs obtenues après avoir enlevé le plus grand et le plus petit des résultats. Le taux de récupération du standard certifié doit être supérieur à 90 %. Dans le cas contraire, une révision du standard ou de l’appareil peut être nécessaire afin de retrouver un taux de récupération acceptable.

La mesure est prise sur un spectrophotomètre AA à ionisation par flamme. Les solutions standard ci-dessous sont utilisées pour produire une courbe de calibration.

Tableau 1 : Solutions standard.

Élément	Concentrations (ppm)
Or	1 3 5 10 20 50 100
Argent	0,2 0,4 1,0 2,0 4,0
Cuivre	5 10 20 50 100
Zinc	5 10 20 50 100
Fer	5 10 20 50 100
Plomb	5 10 20 50 100

La courbe de calibration doit avoir un coefficient de corrélation au moins égal à 0,995. Dans le cas contraire, un remplacement des solutions standard utilisées ou une révision de l’appareil peut être effectuée.

L'écart acceptable des standards et duplicata est fonction de la méthode utilisée, ainsi que de la valeur mesurée. Un écart plus grand sera toléré sur une faible valeur, et sera refusé sur une valeur élevée. Par exemple, un standard d'or ayant une valeur théorique de 70 ppb aura un intervalle acceptable de ± 25%, alors qu'un standard de 1000 ppb devra se lire 1000 ± 10%.

Les séries d'échantillons qui n'auront pas rencontré ces normes seront réanalysés et une vérification des procédures sera effectuée.

La vaisselle utilisée est lavée à l'acide chlorhydrique quatre molaires, puis rincée à l'eau distillée et déminéralisée avant chaque analyse.

MESURE DE LA DENSITÉ DU MINERAI

1- Échantillon entier

La méthode pour mesurer la densité du minerai consiste à peser un échantillon de minerai dans deux milieux différents, tels l'air et l'eau. La densité du minerai peut être calculée à partir de la différence de poids dans les deux milieux.

Une casserole est posée sur la balance analytique précise au millième de gramme. Une chaudière trouée et suspendue à la balance par un câble d'acier, est immergée dans l'eau. La balance est tarée à zéro. L'échantillon est ajouté à la casserole, ce qui permet de mesurer sa masse dans l'air. Ensuite, l'échantillon est transféré dans la chaudière, ce qui permet de mesurer sa masse dans l'eau.

Calcul de la densité

$$\frac{\text{Masse de l'échantillon X densité de l'eau}}{\text{Différence de poids}}$$

Exemple :

Pesée dans l'air = 5,470g
Pesée dans l'eau = 4,400g
Masse de l'échantillon = 4,650g
Différence de masse entre les pesées = 5,470g – 4,400g = 1,070g

$$\frac{4{,}650\text{g X } 1\text{g/cm}^3}{1{,}070\text{g}} = 4{,}350\text{g/cm}^3$$

2- Pulpe

La densité peut être mesuré sur les pulpes de la manière suivante :

- Peser 20.00 g de pulpe, et transférer dans un cylindre gradué (verre) de 100ml.
- Compléter à la marque un ballon volumétrique de 50.0 ml avec de l'eau distillée et déminéralisée (important ! la température de l'eau doit être notée). Peser le ballon+eau et noter le poids obtenu.
- Verser environ 20ml d'eau dans le cylindre; agiter à l'aide d'une tige de verre, afin d'humecter complètement la pulpe, et enlever les bulles d'air présentes.
- À l'aide du reste de l'eau, rincer la tige de verre et les parois du cylindre de manière à ce que toute l'eau se retrouve dans le cylindre. Laisser reposer quelques minutes, au besoin, pour faciliter la lecture du volume.
- Peser le ballon vide; la différence de poids entre le ballon vide et plein correspond au volume d'eau ajouté au cylindre (après correction due à la température)
- À l'aide d'une pipette graduée de 10.0 ml, enlever le volume excédentaire de liquide, soit le volume d'eau déplacé par la pulpe.

$$\text{Densité de la pulpe} = \frac{Mp}{Ve}$$

Ou Mp = masse de la pulpe
Ve = volume excédentaire

ANALYSE DES MÉTAUX NOBLES (Pt, Pd, Rh,) PAR PYROANALYSE, FINITION AU FOUR GRAPHITE (GFAA)

La pyroanalyse permet d'extraire les métaux tel que le platine, palladium et rhodium de la matière rocheuse pour pouvoir en déterminer la concentration. La fusion du minerai avec de l'oxyde de plomb, des agents réducteurs ainsi que l'argent en solution provoque la migration des métaux nobles vers le plomb métallique formé lors de cette même fusion.

Une fois refroidie, la scorie vitreuse est écartée pour ne laisser qu'une boule de plomb (culot). Le culot est ensuite chauffé dans une coupelle qui absorbe le plomb fondu ne laissant qu'une bille d'argent et métaux précieux.

A) <u>Exploration</u>

1- Échantillon de départ : 30 grammes.
2- La bille d'argent produite est transférée dans une éprouvette graduée à 5.0 ml; 0.5 ml d'acide nitrique est ajouté, et une première digestion de 25 minutes est effectuée dans un bain-marie.
3- 1.0 ml d'acide chlorhydrique concentré sont ajouté pour une deuxième digestion de 15 minutes (bain-marie).
4- Après refroidissement, l'échantillon est complété à 5.0 ml, et homogénéisé.

B) <u>Catalyseurs (Pt ou Pd)</u>

1- Découper des carrés de 5 cm de côté dans une feuille de plomb métallique (1 par blanc/échantillon/étalon). Relever les côtés des carrés pour former de petites boîtes.
2- Peser environ exactement 0.5000g de catalyseur dans une «boîte». Ensuite, placer dans un creuset avec les ingrédients nécessaires et procéder à la fusion/cupellation.
3- Procéder comme pour l'exploration, en utilisant des tubes de 10.0 ml et des volumes doubles d'acide. Compléter à 10.0 ml.

C) <u>Four au graphite (spectrAA 64OZ – GTA 100 de Varian)</u>

1- Utiliser les méthodes enregistrées dans la mémoire de l'appareil.
2- Calculs :

- Exploration : $\text{Lecture en ppb X } \frac{5.0\text{ml}}{30\text{g}} = \text{concentration de l'échantillon}$
- Catalyseur : $\text{Lecture en ppb X } \frac{10.0\text{ml}}{\text{masse de échantillon}} = \text{concentration de l'échantillon}$

RENSEIGNEMENTS UTILES

Composition du fondant

Le fondant #2, fabriqué par notre fournisseur, Mines Assay Supplies à Kirkland Lake, est composé de :

Litharge (PbO)	57,4%
Carbonate de sodium (Na_2CO_3)	27,0%
Borax ($Na_2B_4O_7 \bullet 10H_2O$)	12,2%
Silice (SiO_2)	3,4%

Liste des équivalences
SOLIDES

1 % = 10000 g/t
1 g/t = 0,0001 %
1 g/t = 1 ppm
1 g/t = 1000 ppb
1 g/t = 0,029 oz/t
1 oz/t = 34,3 g/t
1 ppb = 0,001 g/t
1 ppb = 0,000029 oz/t

LIQUIDES

1 µg/ml = 1 mg/L
1 µg/ml = 1 ppm
1 µg/ml = 1000 ng/mL
1 ng/ml = 1 ppb

Toutes les unités sont exprimées en tonne métrique.
1 tonne métrique = 1000 kilogrammes = 2200 livres.

Les chiffres significatifs

<10	un chiffre après le point	8,45 = 8,5
entre 10-99	arrondir à l'unité	20,56 = 21
entre 100-999	arrondir au dixième	665 = 670 451 = 450
1000	arrondir au centième	1560 = 1600

Si le nombre est plus grand que 1000 ppm convertir en %

Caractéristiques des éléments

Or	peu soluble dans HNO_3, soluble dans HCl et insoluble dans H_2SO_4
Argent	soluble dans HNO_3 et H_2SO_4, insoluble dans HCl

GLOSSAIRE

Blende	Minerai naturel de sulfure de zinc.
Borax	Borate hydraté de sodium ($Na_2B_4O_7 \bullet 10\ H_2O$).
Calcite	Carbonate naturel de calcium cristallisé ($CaCO_3$) qui constitue la gangue de nombreux filons.
Chalcopyrite	Pyrite de soufre et de cuivre ($CuFeS_2$).
Chromite	Qui contient du chrome.
Dolomie	Carbonate naturel double de calcium et de magnésium ($MgCa(CO_3)_4$).
Galène	Sulfure naturel de plomb (PbS).
Gangue	Substance stérile mélangée aux minéraux utiles dans le minerai.
Inclusion	Introduction, étant d'une chose incluse, impureté dans la bille d'or.
Litharge	Oxyde de plomb (PbO) fondu et cristallisé de couleur rouge-orange.
Limonite	Oxyde ferrique naturel (Fe_2O_3) rouge : Oligiste. Brune : Limonite.
Molybdenite	Sulfure de molybdène (MoS_2).
Pyrite	Sulfure naturel de fer (FeS_2).
Pyrrohotine	Sulfure naturel de fer et de cuivre ($CuFeS_2$).
Quartz	Cristaux de silice pur (SiO_2 ou SiO_4).
Schiste	Roche sédimentaire et métamorphique.

LIMITES DE DÉTECTION

Matières organiques	0.01 %
Hydrocarbures C10-C50	100
Soufre total	0.01 %
Sulfate	0.01 %
Sulfure	0.01 %
Densité (échantillon complet)	0.02
Densité (pulpe)	0.1
Au (ppb)	5
Au (g/t)	0.06
Cyanures (ppm)	0.2
Mo (ppm)	1
Pt/Pd (ppb)	1 à 10
Be (ppm)	0.1
Al (ppm)	1
V (sous-traitance)	1
Hg (ppm)	0.04
U (sous-traitance)	0.5

	A.A.	G.F.A.A.
Ag	0.1	
As		0.1
Bi		1
Ca	5	
Cd	0.1	
Co	1	
Cr	1	
Cu	1	
Fe	10	
K	5	
Mg	5	
Mn	1	
Na	10	
Ni	1	
Pb	5	
Se		0.2
Zn	1	

Assurance de la qualité

Plusieurs procédures et contrôles sont utilisés pour assurer la qualité du travail effectué :

1. *Utilisation de blancs, duplicatas et étalons de références : chaque série d'échantillons, d'un nombre maximal de 21, doit obligatoirement être accompagnée d'au moins un blanc, duplicata et étalon de référence. Ces éléments de contrôle sont mobile, c'est à dire que leur position dans la série d'échantillon sera différente d'une série à l'autre. Cette approche permet à la fois de pouvoir identifier sans équivoque une série donnée, et de vérifier l'absence de contamination à l'intérieur des contenants (verrerie, creuset) utilisés.*

2. *Utilisation d'étalon de référence provenant de sources reconnues (CANMET, Rocklab). Dans certains cas, un ou des étalons maison sont utilisés après avoir été étalonné.*

3. *Granulométrie : un échantillon sur 20 est contrôlé pour la granulométrie, après concassage et pulvérisation, afin de répondre aux critères d'homogénéité et de reproductibilité des mesures.* ***Un échantillon dépassant 10% de >8 mesh subira une seconde étape de concassage. Un échantillon dépassant 10% de > 200 mesh subira une seconde étape de pulvérisation; ces étapes additionnelles permettent un meilleur contrôle de l'homogénéité des échantillons.***

4. *Un échantillon donnant des résultats non- reproductibles (analyse de l'or) sera ré-analysé selon la technique de l'or grossier; cette technique permettra de déterminer si la disparité des résultats provient de la nature même de l'échantillon, ou de la méthode utilisée pour les premières analyses.*

5. *Les résultats préliminaires transmis au client ne doivent pas inclure les valeurs originales des échantillons devant être ré-analysés. Les résultats des ré-analyses devront être vérifier et approuvés avant que ces résultats puissent être considérés comme officiels.*

6. *Toute anomalie, dérogation, erreur ou doute quant à la validité du travail doit être immédiatement consigné sur le formulaire prévu à cette fin. Une copie du formulaire est acheminée au chef analyste, qui prendra les mesures nécessaires pour régler la situation; le formulaire original sera joint aux documents relatifs au projet concerné.*

Critères d'acceptabilité des contrôles de la qualité

Blancs : les blancs doivent en tout temps être inférieurs à la limite de quantification de la méthode; leur valeur sera soustraite au besoin des résultats des échantillons. **Un blanc élevé peut entraîner une ré-analyse complète d'une série d'échantillons.**

Duplicata : la valeur acceptable d'un duplicata dépend de la limite de détection de la méthode employée et du résultat moyen échantillon/duplicata (voir tableau)

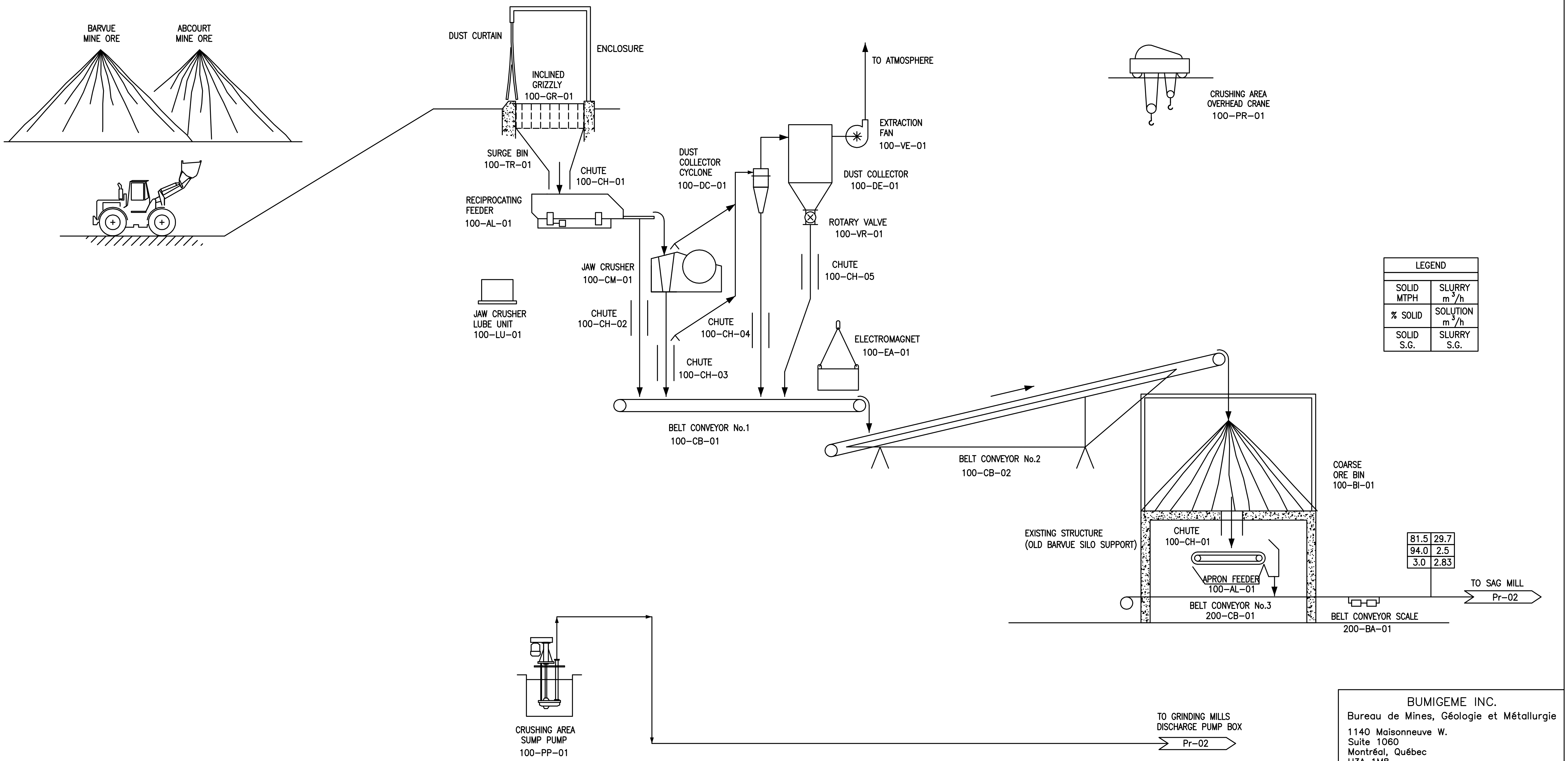
Valeur moyenne obtenue (duplicata/échantillon)	**Écart acceptable**
0 à 20 ppb	50 %
21 à 100 ppb	25 %
101 à 500 ppb	15 %
501 ppb et +	10 %
0 à 0.20 g/t	50 %
0.21 à 1 g/t	20 %
1.01 g/t et +	10 %

Étalons de référence (certifiés ou autre) : La valeur acceptable d'un étalon dépends de la méthode employée, ainsi que de l'importance de la valeur réelle :

Valeur obtenue (étalon)	**Écart acceptable**
200 à 1000 ppb	10 %
1001 et +	5%
0.80 à 2 g/t	10%
2 g/t et +	5%

APPENDIX 2

Flowsheets



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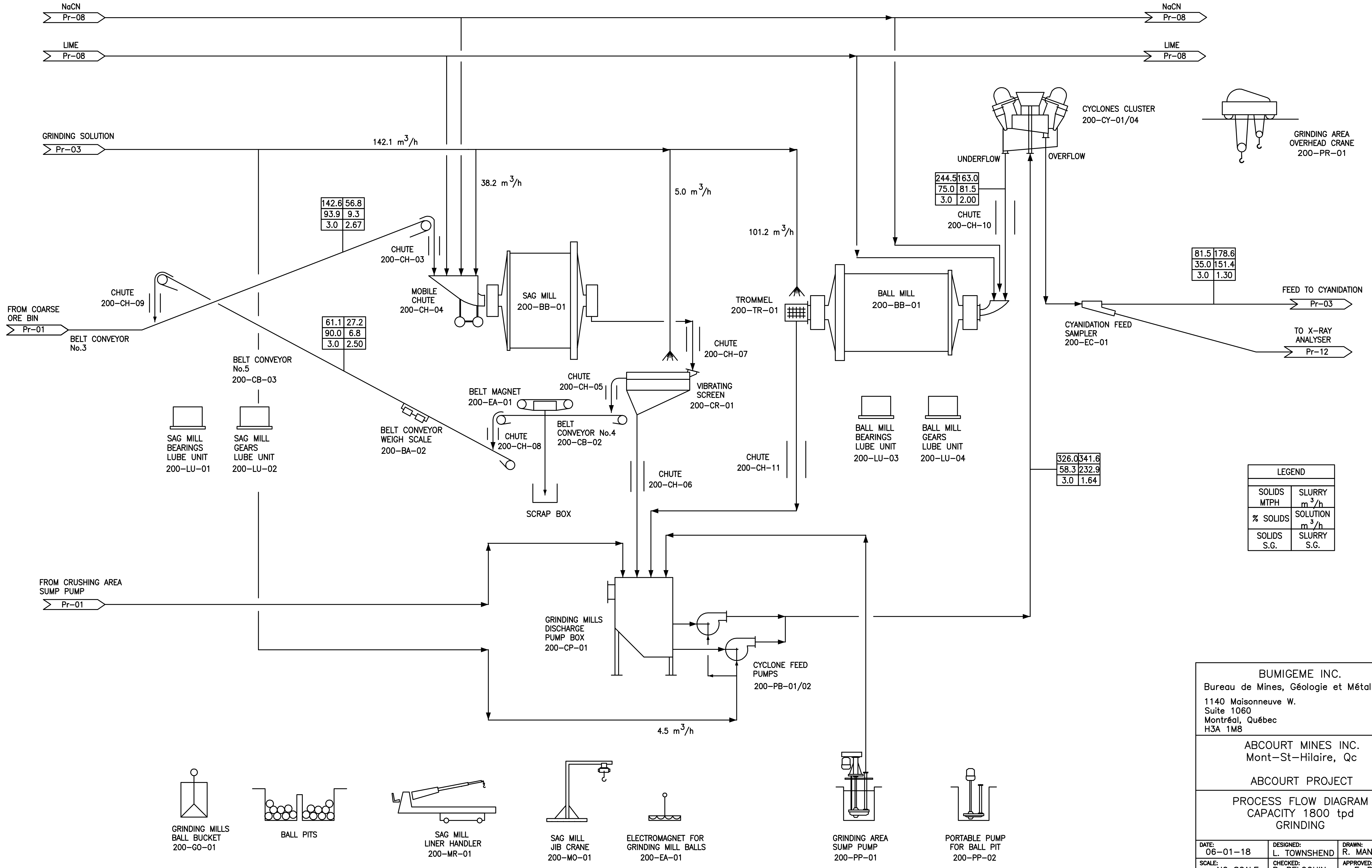
ABCOURT MINES INC.
Mont–St–Hilaire, Qc

ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
ORE CRUSHING AND RECLAIMING

DATE: 07–01–18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI	
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL	
PROJECT: B150–03–006	DRAWING: B15003006–Pr–01		REV: 01



LEGEND	
SOLIDS MTPH	SLURRY m^3/h
% SOLIDS	SOLUTION m^3/h
SOLIDS S.G.	SLURRY S.G.

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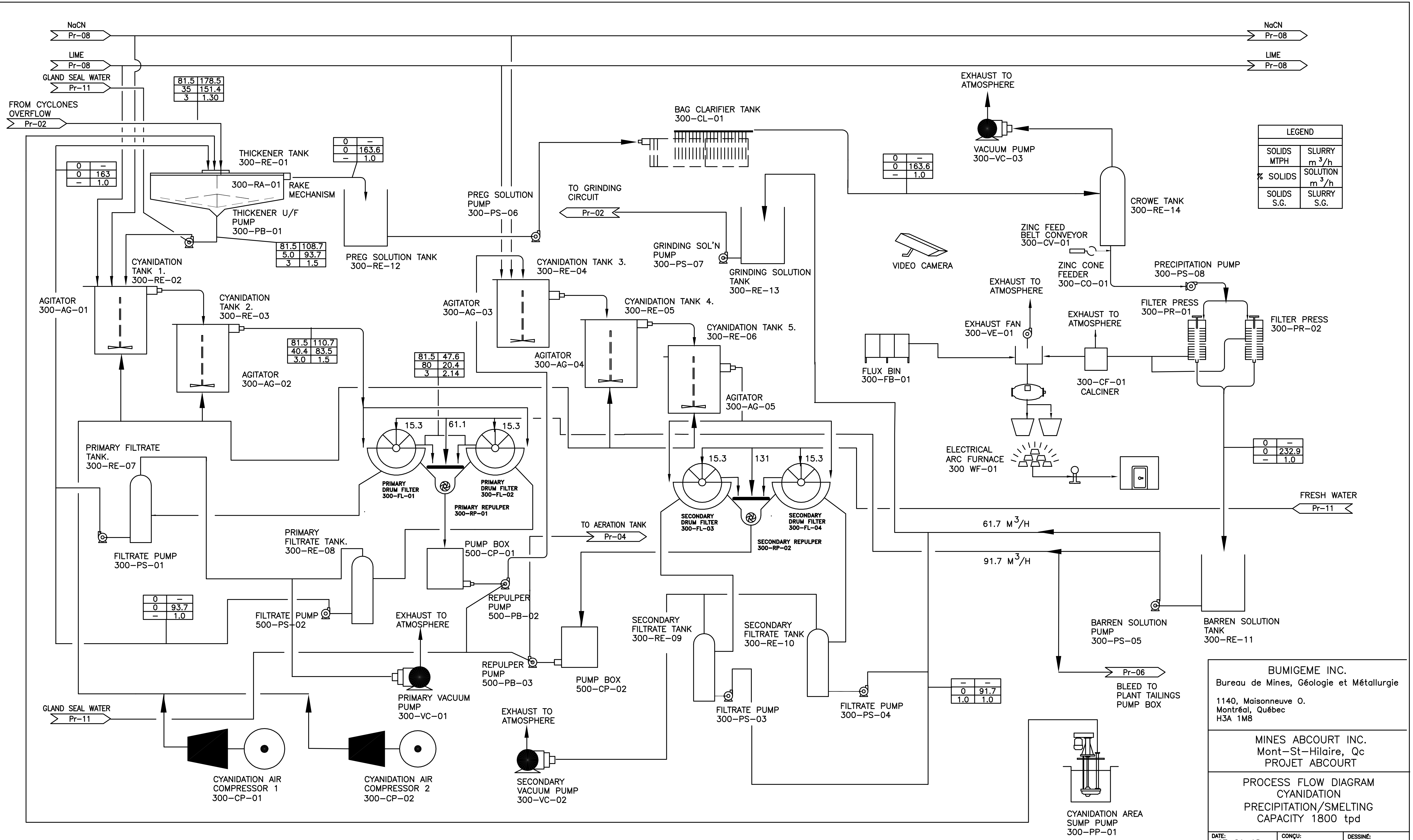
ABCOURT MINES INC.
Mont-St-Hilaire, Qc

ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
GRINDING

DATE: 06-01-18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150-03-006	DRAWING: B15003006-Pr-02	REV: 01



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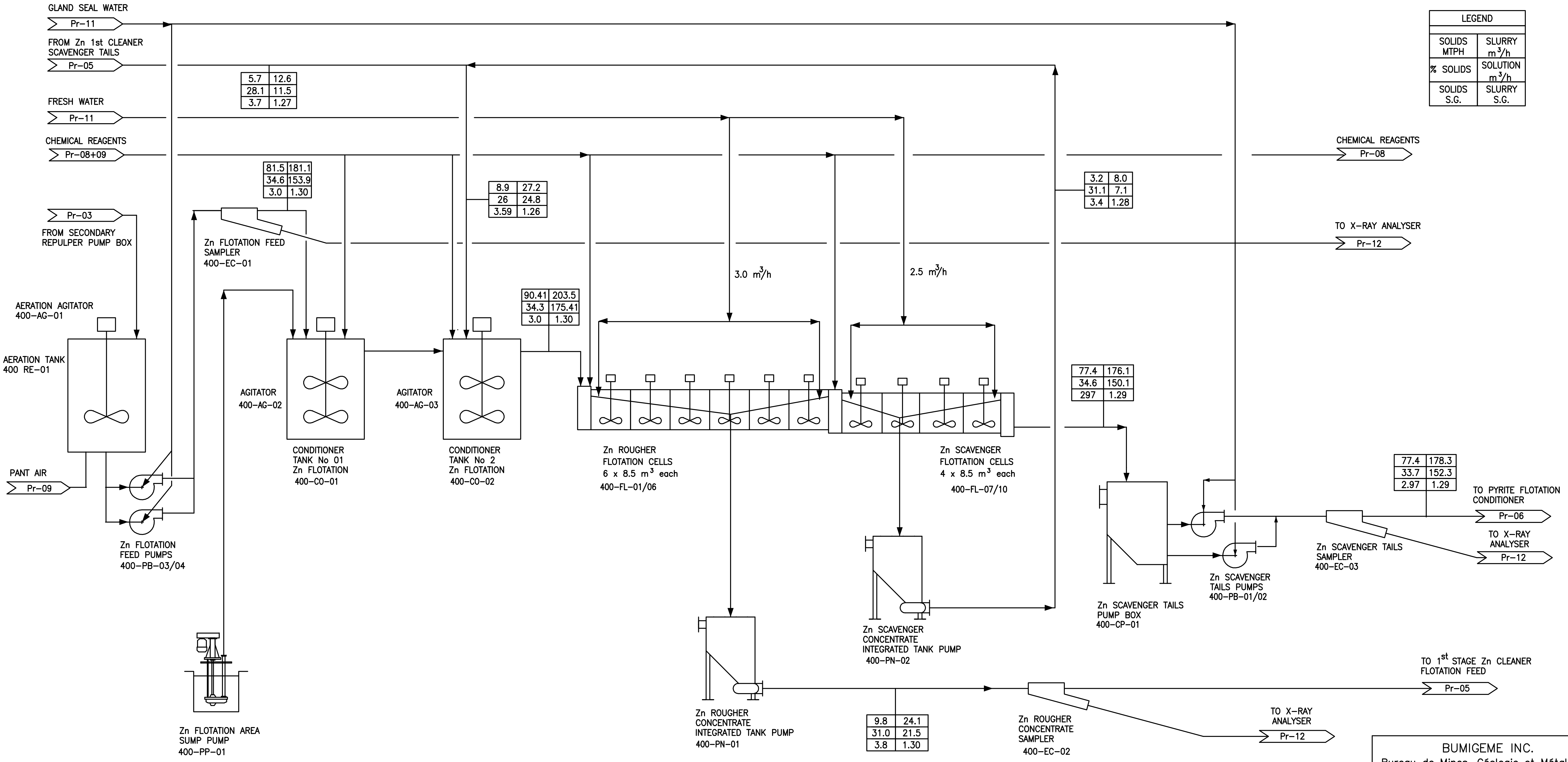
MINES ABCOURT INC.
Mont-St-Hilaire, Qc
PROJET ABCOURT

PROCESS FLOW DIAGRAM
CYANIDATION
PRECIPITATION/SMELTING
CAPACITY 1800 tpd

DATE: 07-01-18	CONÇU: L. TOWNSHEND	DESSINÉ: R. MANSOURI
ÉCHELLE: NO SCALE	VÉRIFIÉ: R. PELOQUIN	APPROUVÉ: F. BARIL
PROJET: B150-03-006	DESSIN: B15003006-Pr-03	REV: 01



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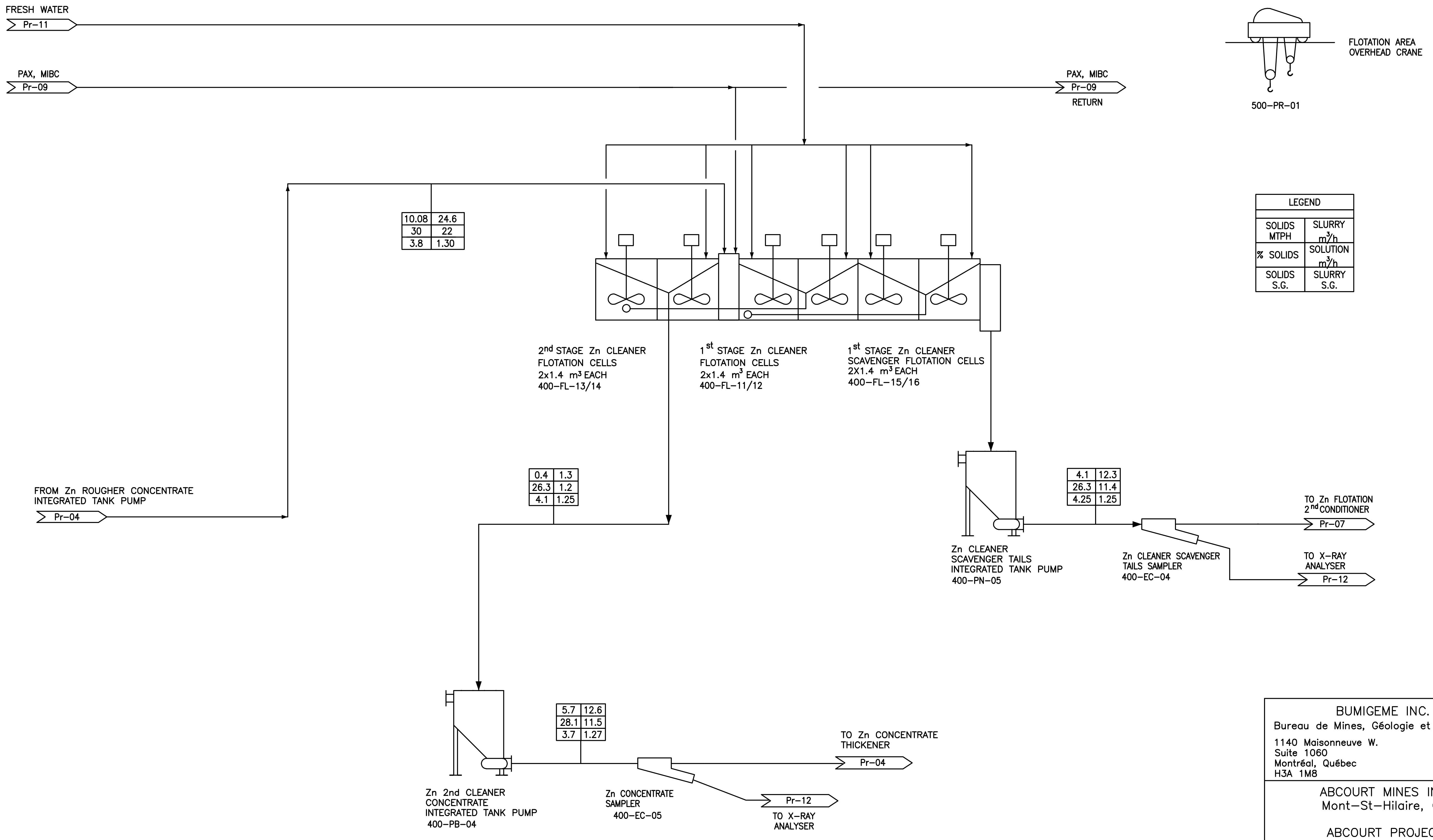
ABCOURT MINES INC.
Mont-St-Hilaire, Qc

ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
Zn ROUGHER/SCAVENGER FLOTATION

DATE: 07-01-18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150-03-006	DRAWING: B15003006-Pr-04	REV: 01



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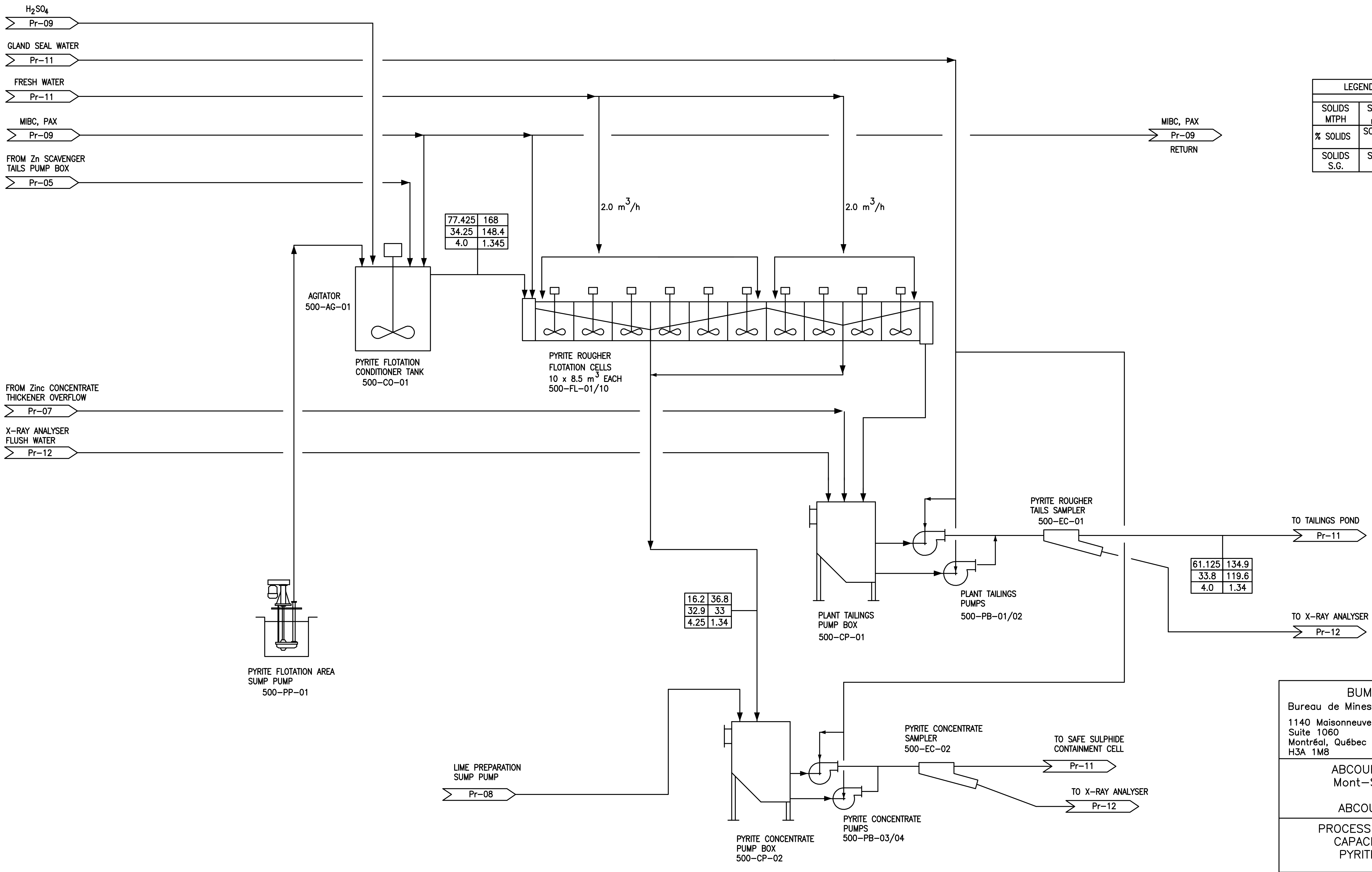
ABCOURT MINES INC.
Mont–St–Hilaire, Qc

ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
Zn CLEANER FLOTATION

DATE: 07–01–18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150–03–006	DRAWING: B15003006–Pr–05	REV: 01



LEGEND	
SOLIDS MTPH	SLURRY m^3/h
% SOLIDS	SOLUTION m^3/h
SOLIDS S.G.	SLURRY S.G.

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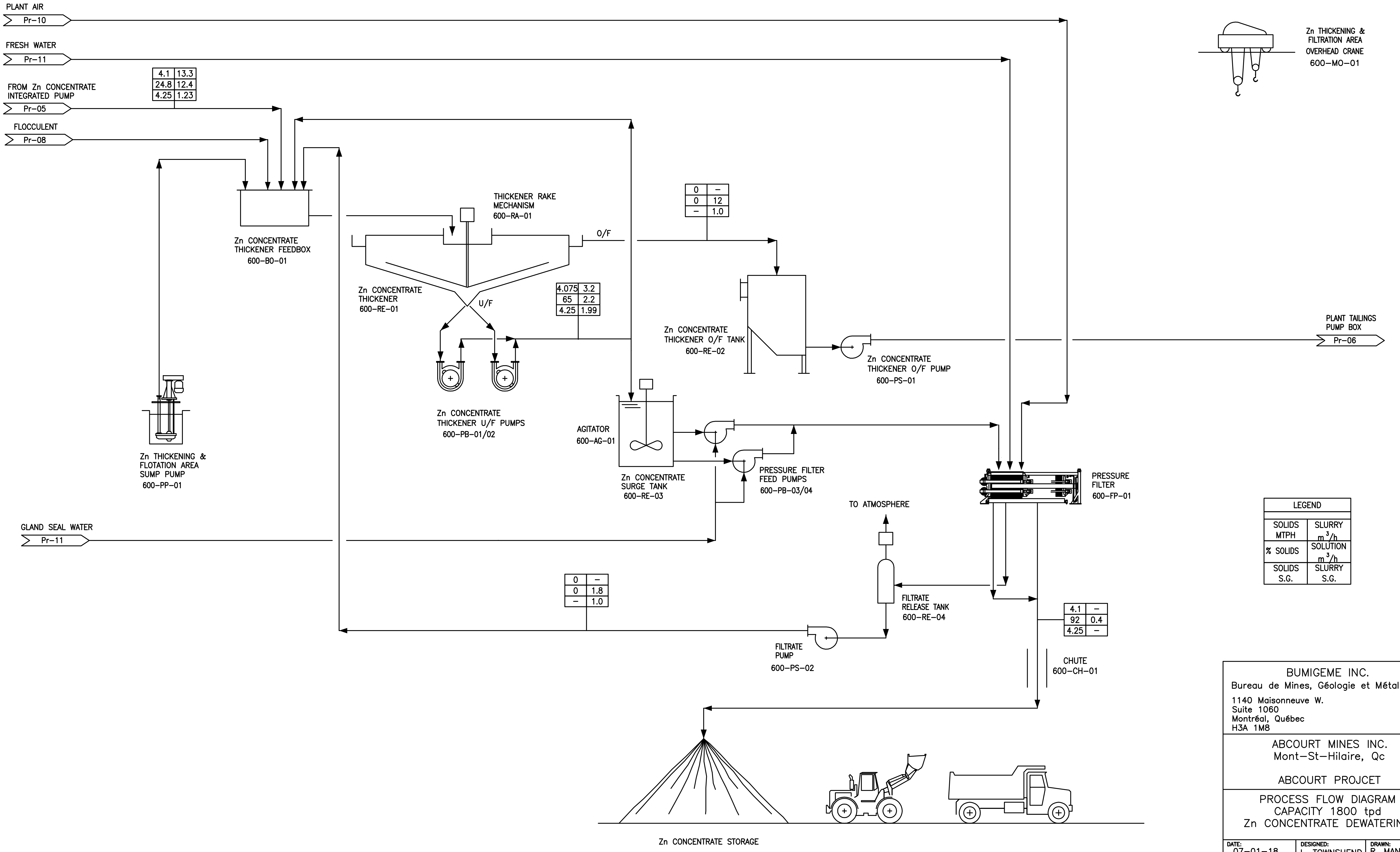
ABCOURT MINES INC.
Mont–St–Hilaire, Qc

ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
PYRITE FLOTATION

DATE: 07–01–18	DESIGNED: L. TOWNSHEND	DRAWN: M. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150–03–006	DRAWING: B15003006–Pr–06	REV: 02



LEGEND	
SOLIDS MTPH	SLURRY m^3/h
% SOLIDS	SOLUTION m^3/h
SOLIDS S.G.	SLURRY S.G.

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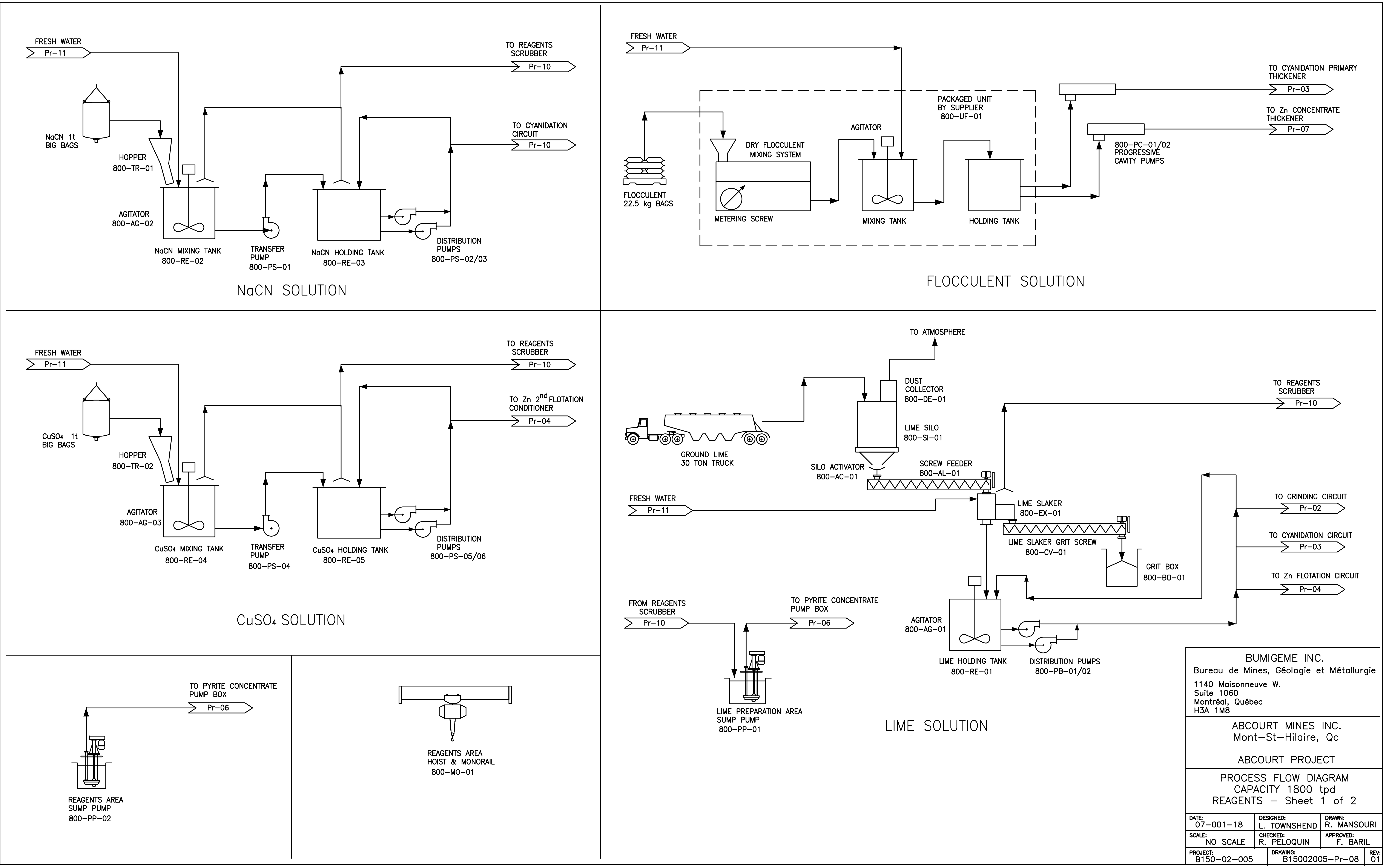
ABCOURT MINES INC.
Mont-St-Hilaire, Qc

ABCOURT PROJCET

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
Zn CONCENTRATE DEWATERING

DATE: 07-01-18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150-03-006	DRAWING: B15003006-Pr-07	REV: 01



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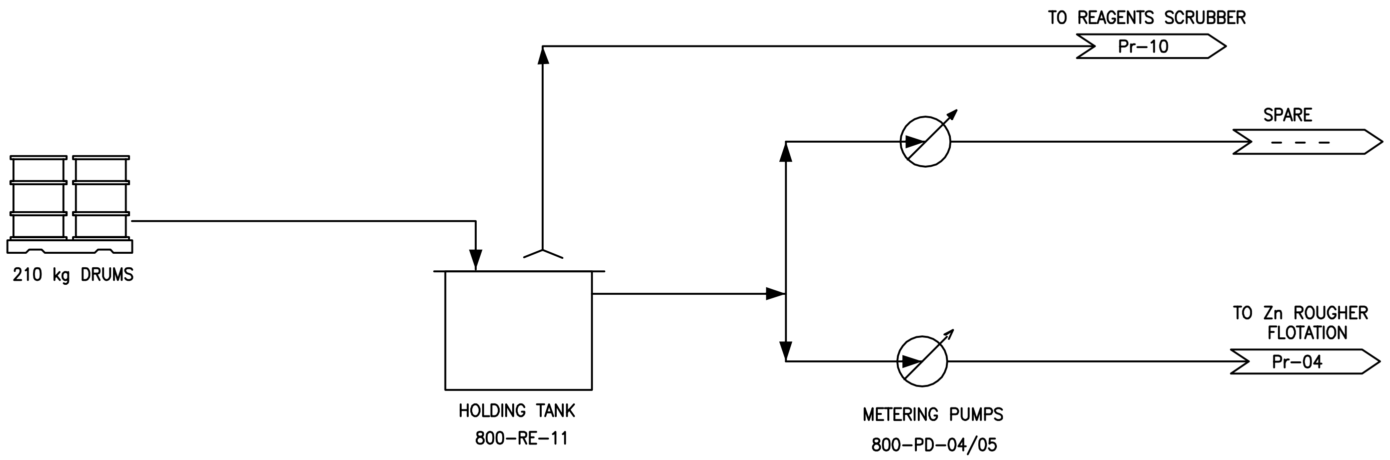
ABCOURT MINES INC.
Mont-St-Hilaire, Qc

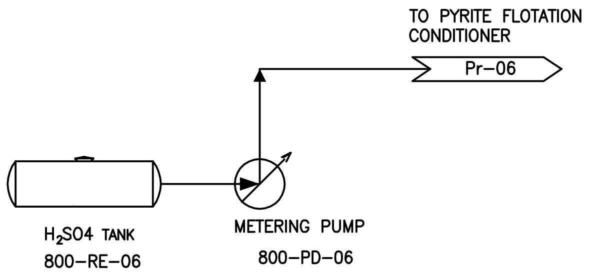
ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
REAGENTS – Sheet 1 of 2

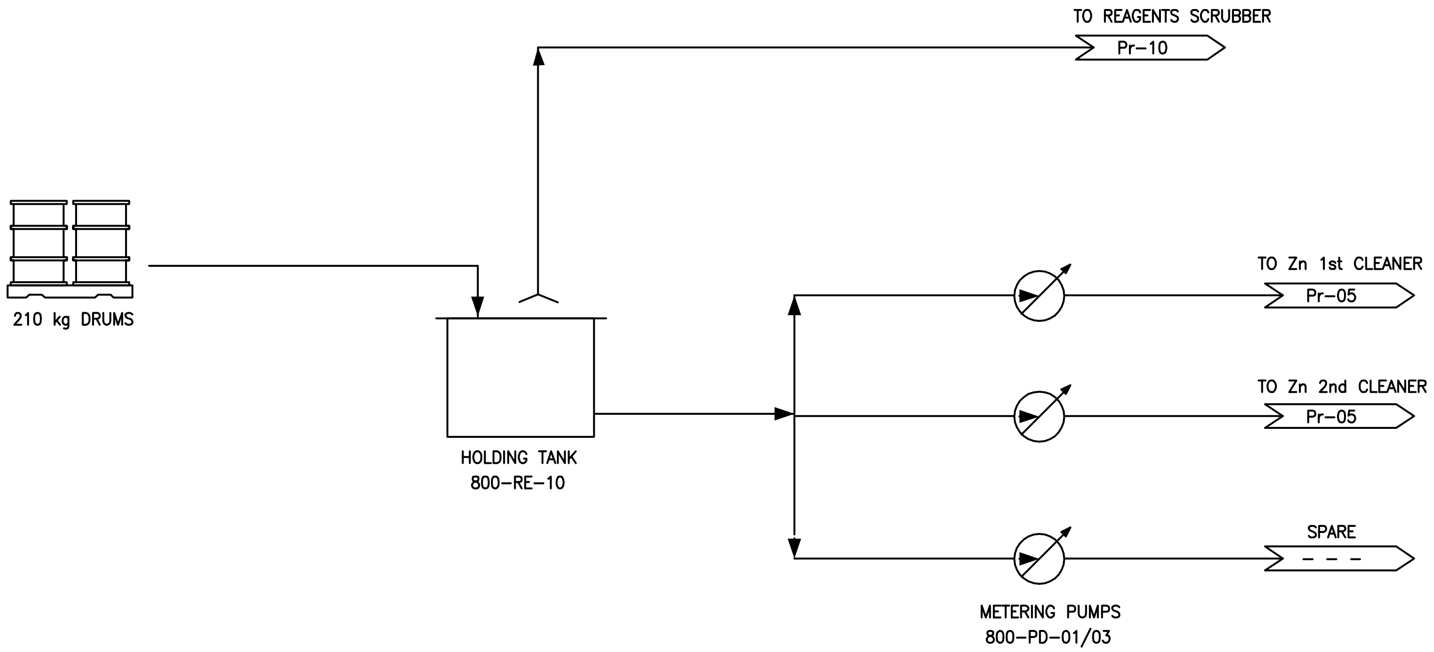
DATE: 07-001-18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150-02-005	DRAWING: B15002005-Pr-08	REV: 01



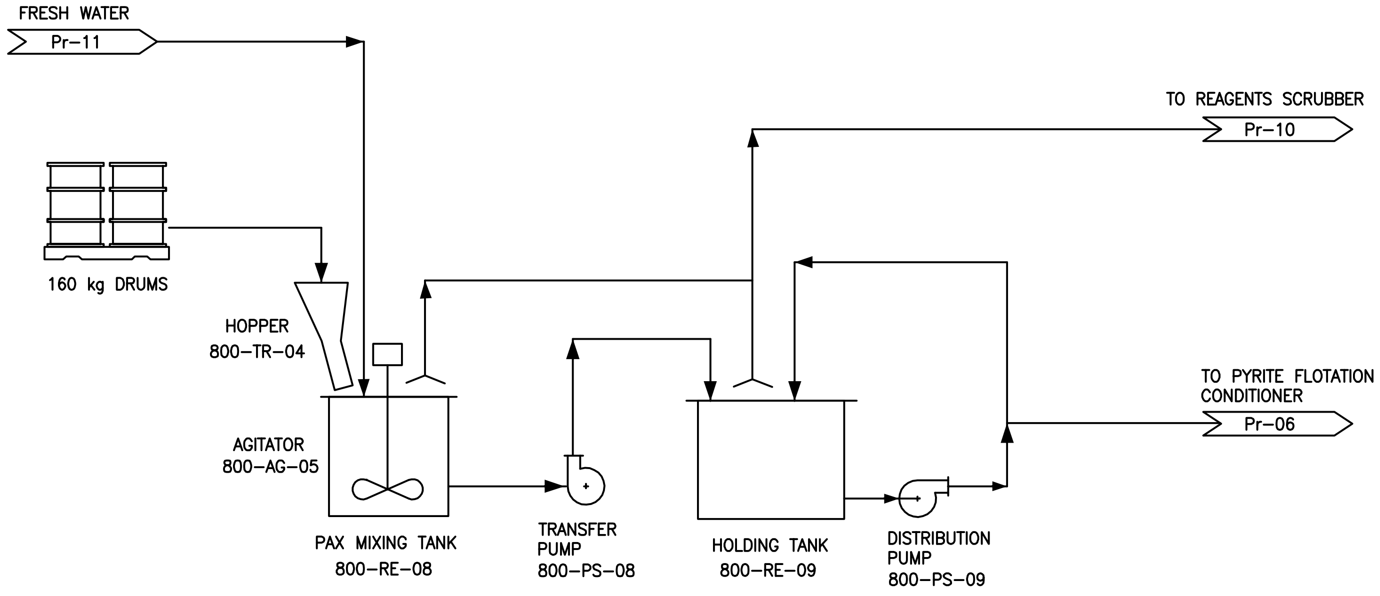
CYTEC 3477 SOLUTION



H_2SO4



MIBC SOLUTION



PAX SOLUTION

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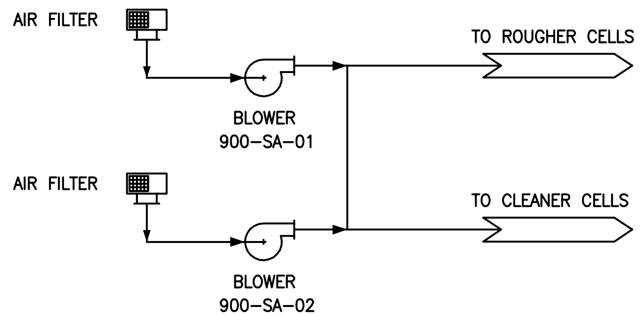
ABCOURT MINES INC.
Mont-St-Hilaire, Qc

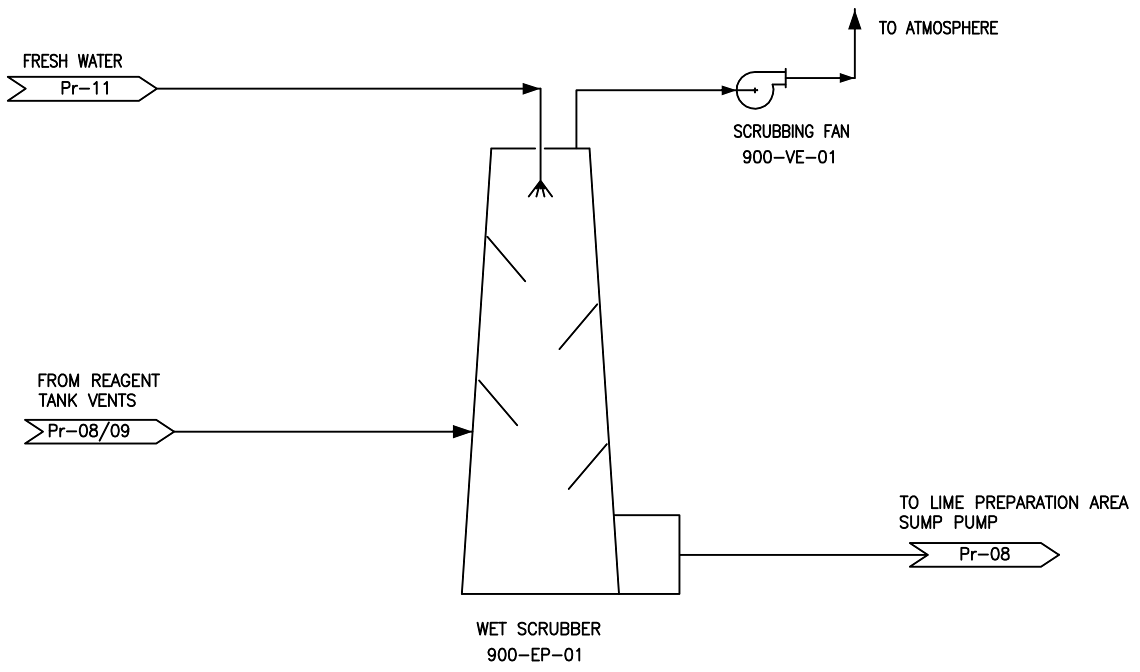
ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
REAGENTS – Sheet 2 of 2

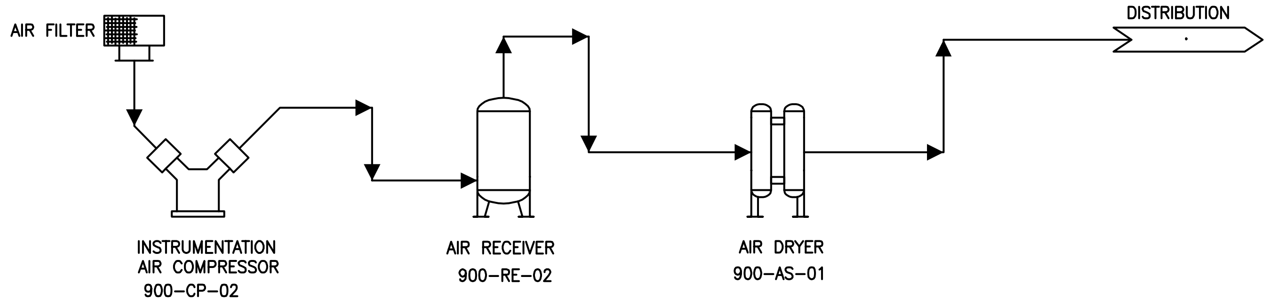
DATE: 07-01-18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150-03-006	DRAWING: B15003006-Pr-09	REV: 01



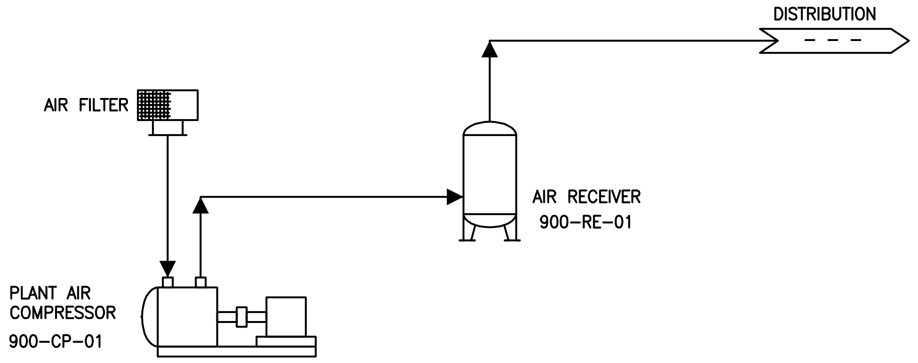
FLOTATION AIR



REAGENT SCRUBBER



INSTRUMENTATION AIR



PLANT AIR

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ABCOURT MINES INC.
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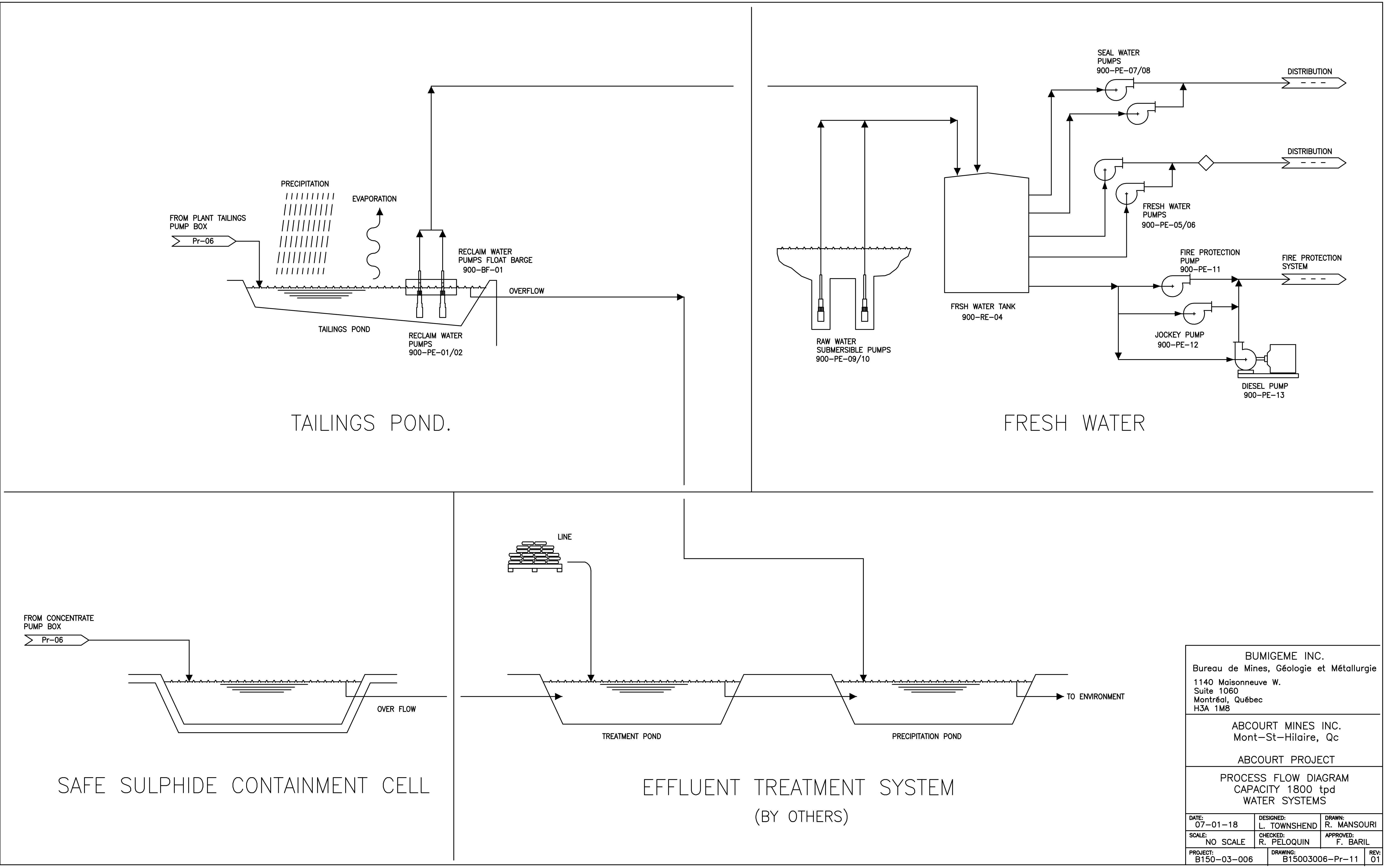
ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 mtpd
PLANT SERVICES

DATE: 07–01–18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150–03–006	DRAWING: B15003006–Pr–10	REV: 01



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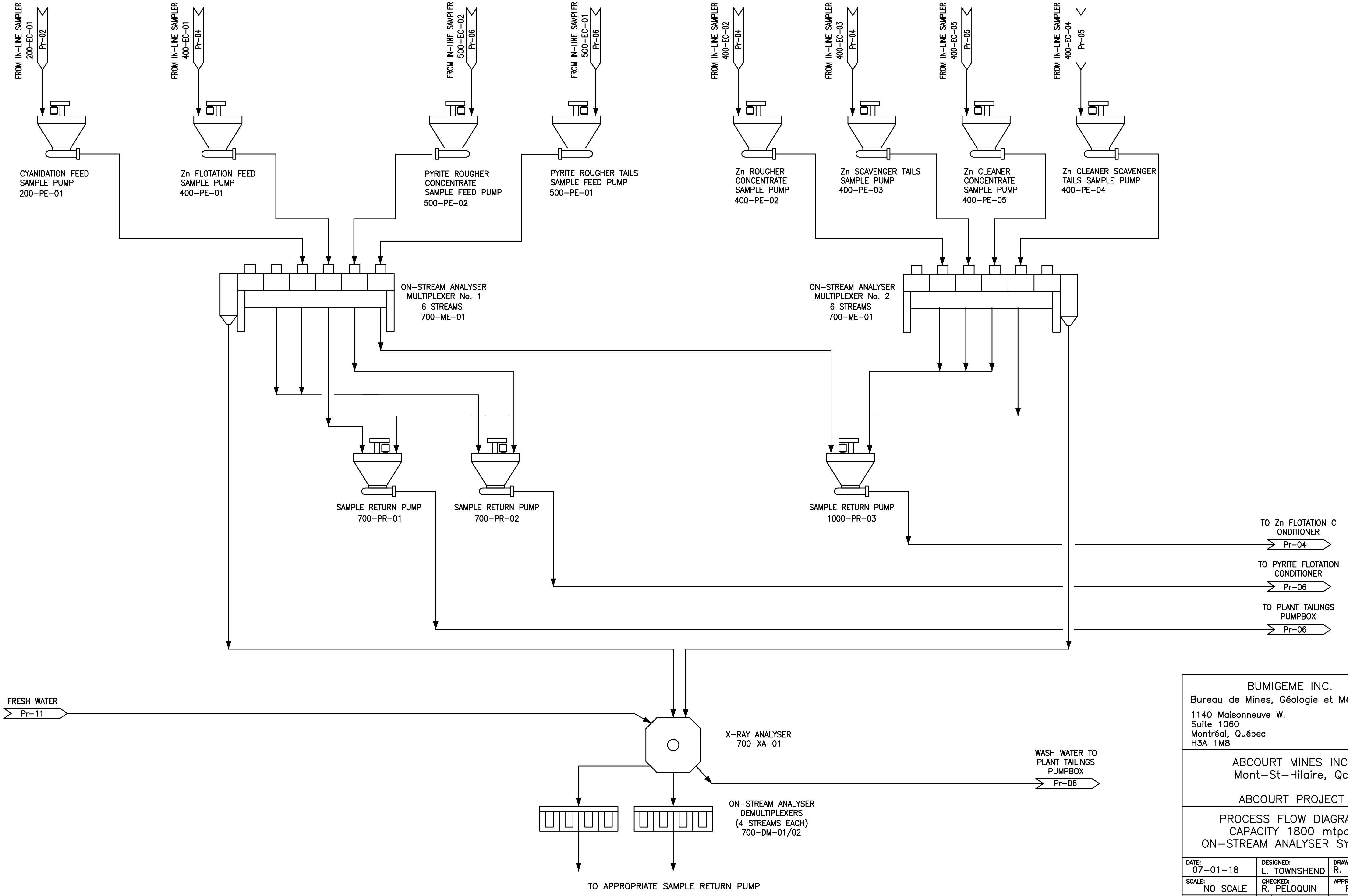
ABCOURT MINES INC.
Mont-St-Hilaire, Qc

ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 tpd
WATER SYSTEMS

DATE: 07-01-18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150-03-006	DRAWING: B15003006-Pr-11	REV: 01



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ABCOURT MINES INC.
Mont-St-Hilaire, Qc

ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 mtpd
ON-STREAM ANALYSER SYSTEM

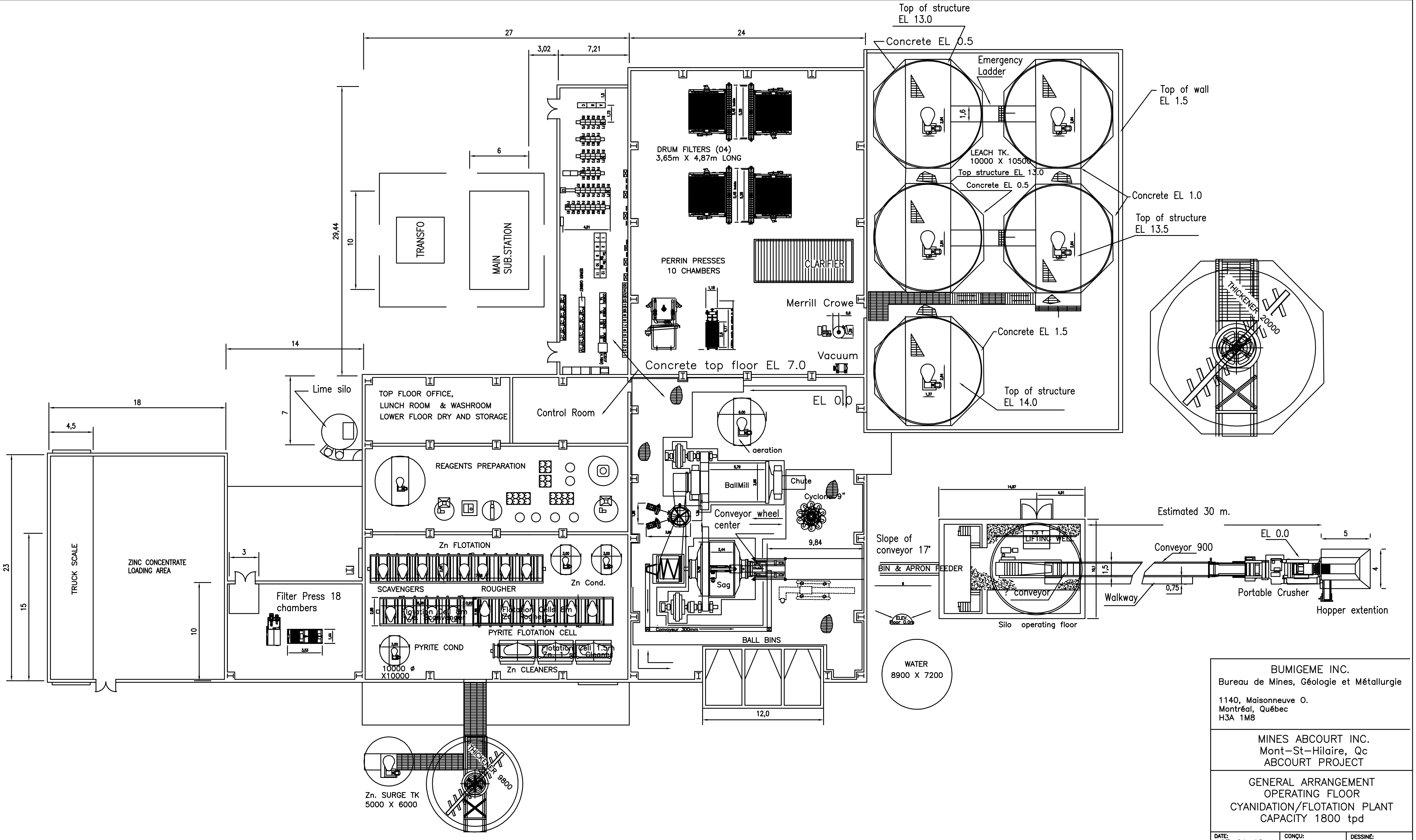
DATE: 07-01-18	DESIGNED: L. TOWNSHEND	DRAWN: R. MANSOURI
SCALE: NO SCALE	CHECKED: R. PELOQUIN	APPROVED: F. BARIL
PROJECT: B150-03-006	DRAWING: B15003006-Pr-12	REV: 01

APPENDIX 3

3D view of the mill building and the crusher
and general arrangement of the process plant

Mines Abcourt Inc.

1800 tpd concentrator



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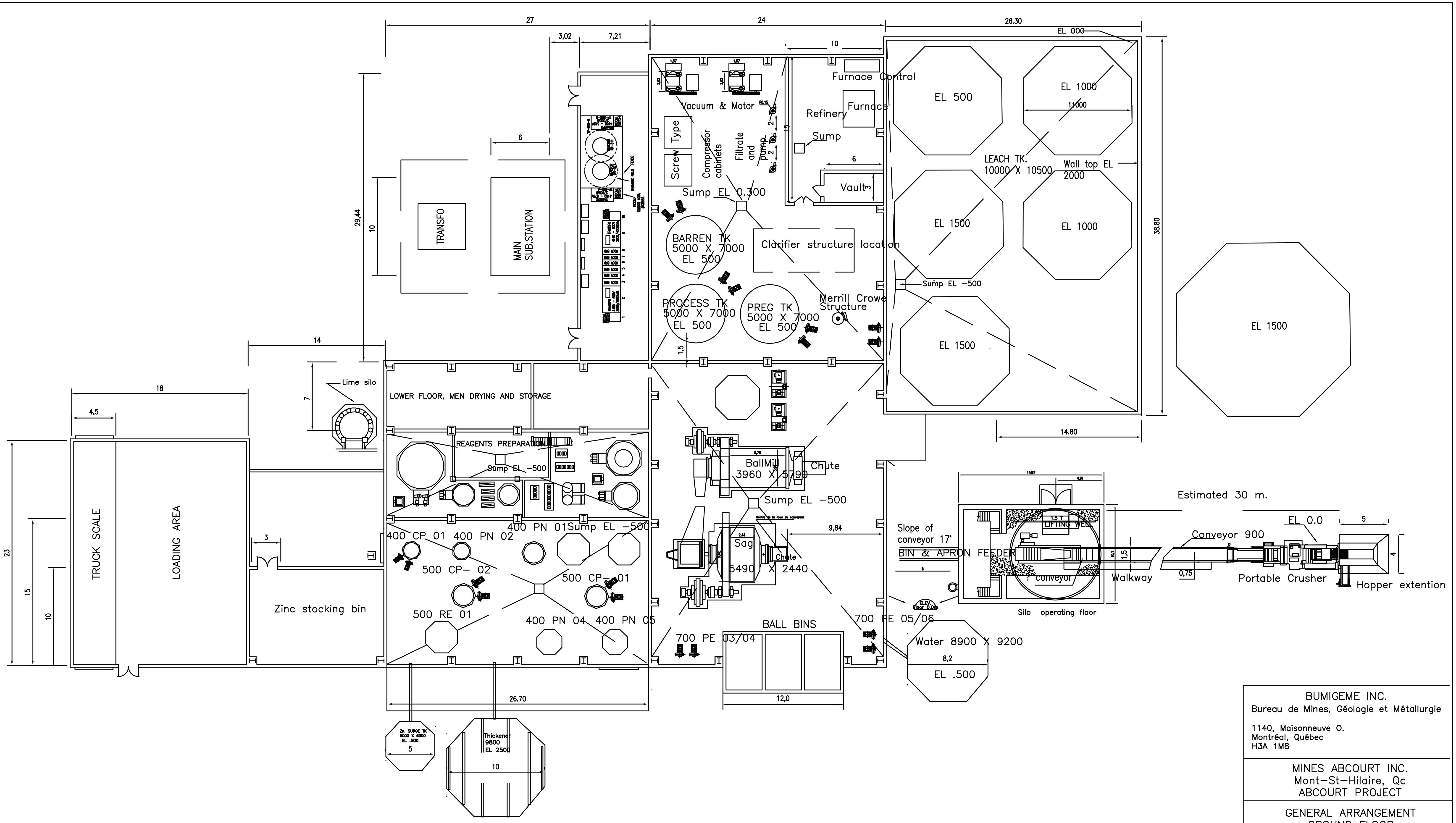
MINES ABCOURT INC.
Mont-St-Hilaire, Qc
ABCOURT PROJECT

GENERAL ARRANGEMENT
OPERATING FLOOR
CYANIDATION/FLOTATION PLANT
CAPACITY 1800 tpd

DATE: 07-01-18	CONÇU: L. TOWNSHEND	DESSINÉ: R. MANSOURI	
ÉCHELLE: NO SCALE	VÉRIFIÉ: R. PELOQUIN	APPROUVÉ: F. BARIL	
PROJET: B150-02-005	DESSIN: B15003006-Am-01		REV: 01



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MINES ABCOURT INC.
Mont-St-Hilaire, Qc
ABCOURT PROJECT

GENERAL ARRANGEMENT
GROUND FLOOR
CYANIDATION/FLOTATION PLANT
CAPACITY 1800 tpd

DATE: 07-01-18	CONÇU: L. TOWNSHEND	DESSINÉ: R. MANSOURI
ÉCHELLE: NO SCALE	VÉRIFIÉ: R. PELOQUIN	APPROUVÉ: F. BARIL
PROJET: B150-02-005	DESSIN: B15003006-Am-02	REV: 01

APPENDIX 4

List of equipment and detailed capital costs for the process plant

Mines Abcourt Inc.
Abcourt Project
Process Equipment Costs Including Installation

Appendix C

Equip. No.	Equipment List Description	Qty	Capacity or Dimensions	Power kW	Equipment cost	Total Electrical	Total Mechanical	Total Piping	Transport	Total cdn$
ORE CRUSHING AND STOCKPILING SECTION										
100-GR-01	Inclined Grizzly	1	500 mm sq. openings		8 000 $	- $	5 000 $		3 000 $	16 000 $
100-TR-01	Ore Surge Bin	1	50 t Capacity		incl.	- $	incl.		- $	
100-CH-01	Feed Chute	1			incl.	- $	incl.		- $	
100-AL-01	Reciprocating Feeder c/f with 140 mm Openings Grizzly	1	1143 mm large x 6100 mm	22,5	incl.	4 500 $	incl.		- $	4 500 $
100-CH-02	Undersized Ore Chute	1			incl.	- $	incl.		- $	
100-CM-01	Jaw Crusher Portable Unit	1	810 mm x 1220	150	497 655 $	19 000 $	45 000 $		8 000 $	569 655 $
100-LU-01	Jaw Crusher Lube Unit	1		2	incl.	2 100 $	incl.		- $	2 100 $
100-CH-03	Crusher Discharge Chute	1			incl.	- $	incl.		- $	
100-CB-01	Belt Conveyor No. 1	1	914 mmx 12 m	7,5	incl.	7 875 $	incl.		- $	7 875 $
100-EA-01	Electromagnet	1			19 355 $	4 820 $	14 000 $		1 500 $	39 675 $
100-DE-01	Dust Collector System	1		47,7	264 500 $	113 335 $	98 120 $	25 110 $	20 995 $	522 060 $
100-PR-01	Overhead Crane - Crusher Area	1	8 t capacity	5,5	21 000 $	6 955 $	NIA		3 500 $	31 455 $
100-PP-01	Crusher Area Sump Pump	1	50 mm	3,75	8 800 $	5 130 $	4 000 $	3 000 $	500 $	21 430 $
100-BI-01	Coarse Ore Bin	1	1800 tonne capacity		357 000 $	7 500 $			18 000 $	382 500 $
100-AL-02	Apron Feeder c/w Chute	1	1000 x 5000	10	53 600 $	8 500 $	15 000 $		6 000 $	83 100 $
	TOTAL CRUSHING			**249**	**1 229 910 $**	**179 715 $**	**181 120 $**	**28 110 $**	**61 495 $**	**1 680 350 $**
GRINDING SECTION										
200-CB-01	Belt Conveyor No. 3 c/w Transfer Chute	1	914 mm x 47 m	11,25	115 000 $	4 500 $	8 000 $		4 500 $	132 000 $
200-BA-01	Belt Conveyor Weigh Scale	1	0-150 t/h		12 000 $		6 000 $		1 000 $	19 000 $
200-BB-01	SAG Mill c/w Liner and Accessories	1	5490 mm dia. x 2135	550	2 936 864 $	74 360 $	169 500 $	26 000 $	50 500 $	3 257 224 $
200-TR-01	Trommel	1			14 000 $		2 000 $	2 500 $	3 000 $	21 500 $
200-CB-02	Belt Conveyor No. 4 c/w Transfer Chute	1	610 mm x 12 m	5	15 000 $	3 680 $	5 000 $		2 000 $	25 680 $
200-EA-01	Belt Magnet	1			14 000 $	1 800 $	4 000 $		500 $	20 300 $
200-CB-03	Belt Conveyor No. 5 c/w Transfer Chute	1	610 mm x 35 m	10	15 000 $	3 680 $	6 000 $		1 000 $	25 680 $
200-BA-02	Belt Conveyors Weigh Scale	1	0-50 t/h		10 000 $	3 500 $	4 000 $		500 $	18 000 $
200-MR-01	SAG Mill Liner Handler	1		18,6	235 000 $	2 687 $	3 000 $		7 000 $	247 687 $
200-CP-01	Grinding Mills Pump Box	1	16,8 m³		6 580 $		3 000 $	8 000 $	2 000 $	19 580 $
200-PB-01/2	Cyclone Feed Pumps (335,2 m^3/h)	2	254 mm x 203	100	56 000 $	10 562 $	7 500 $	24 000 $	2 000 $	100 062 $
200-CY-01/10	Cyclone Cluster	4	380 mm dia.		69 795 $		7 000 $	19 000 $	3 500 $	99 295 $
200-BB-02	Ball Mill c/w Liners and Accessories	1	3960 mm dia. x 5500	1400	2 046 352 $	84 360 $	160 000 $	14 000 $	56 500 $	2 361 212 $
200-CR-02	Vibrating Screen	1			167 538 $		2 000 $	2 500 $	3 000 $	175 038 $
200-PP-01	Sump Pump - Grinding Area	1	75 mm	18,6	8 800 $	2 687 $	2 500 $	7 000 $	1 000 $	21 987 $
200-PR-01	Overhead Crane - Grinding Area	1	15 t capacity	18,6	96 000 $	4 500 $	6 000 $		3 500 $	110 000 $
200-MQ-01	Jib Crane - Grinding Area	1	2 t capacity	5	14 000 $	3 500 $	3 000 $		500 $	21 000 $
200-EA-02	Electromagnet for Balls	1		2	31 434 $	3 680 $	5 000 $		500 $	40 614 $
200-GQ-01	Ball Bucket	1			3 500 $		*N/A*		500 $	4 000 $
200-EC-01	Sampler	1		1,5	16 000 $	3 680 $	2 000 $	4 000 $	500 $	26 180 $
200-PE-01	Integrated Tank Pump (Sample Pump)	1	32 mm		6 964 $	2 867 $	3 000 $	6 000 $	700 $	19 531 $
200-PP-02	Portable Sump Pump	1	32 mm	1,5	5 500 $	3 680 $	*N/A*		500 $	9 680 $
	TOTAL GRINDING			**2142**	**5 895 328 $**	**213 723 $**	**408 500 $**	**113 000 $**	**144 700 $**	**6 775 251 $**
CYANIDATION SECTION										
300-RE-01	Thickener Tank	1	20000 mm		208 869 $		4 000 $	9 000 $	7 000 $	228 869 $
300-RA-01	Thickener Rake Mechanism	1		2,25	541 000 $	3 680 $	9 000 $		2 500 $	556 180 $
300-PB-01	Thickener Underflow Pump	1	8" X 6" SRL	30	12 000 $	7 990 $	3 600 $	3 500 $	500 $	27 590 $
300-RE-02	# 1 Cyanidation Tank (1140 m³)	1	10000 X 10500		200 637 $		4 000 $	18 000 $	3 000 $	225 637 $
300-AG-01	# 1 Cyanidation Tank Agitator	1		37,2	93 000 $	9 390 $	10 000 $	6 000 $	1 500 $	119 890 $
300-RE-03	# 2 Cyanidation Tank (1140 m³)	1	10000 X 10500		200 637 $		4 000 $	18 000 $	3 000 $	225 637 $
300-AG-02	# 2 Cyanidation Tank Agitator	1		37,2	93 000 $	9 390 $	10 000 $	6 000 $	1 500 $	119 890 $
300-RE-04	# 2 Cyanidation Tank (1140 m³)	1	10000 X 10500		200 637 $		4 000 $	18 000 $	3 000 $	225 637 $
300-AG-03	# 2 Cyanidation Tank Agitator	1		37,2	93 000 $	9 390 $	10 000 $	6 000 $	1 500 $	119 890 $
300-RE-05	# 2 Cyanidation Tank (1140 m³)	1	10000 X 10500		200 637 $		4 000 $	18 000 $	3 000 $	225 637 $
300-AG-04	# 2 Cyanidation Tank Agitator	1		37,2	93 000 $	9 390 $	10 000 $	6 000 $	1 500 $	119 890 $

Mines Abcourt Inc.

Abcourt Project

Process Equipment Costs Including Installation

Appendix C

Equlp. No.	Equipment List Description	Qty	Capaclty or Dimensions	Power kW	Equipment cost	Total Electrical	Total Mechanical	Total Piping	Transport	Total cdn$
300-RE-06	# 2 Cyanidation Tank (1,140 m³)	1	10000 X 10500		200 637 $		4 000 $	18 000 $	3 000 $	225 637 $
300-AG-05	# 2 Cyanidation Tank Agitator	1		37,2	93 000 $	29 390 $	10 000 $	6 000 $	1 500 $	139 890 $
300-RE-07/08	Primary Filtrate Tanks	2	1200 X 2,500		37 500 $		5 000 $	8 000 $	500 $	51 000 $
300-PS-01/02	Primary Filtrate Pumps	2	5" X4" SRL	29,6	incl.	7 680 $	3 600 $	10 000 $	500 $	21 780 $
300-PV-01	Primary Vacuum Pump	1		150	200 000 $	9 000 $	8 000 $	12 000 $	3 000 $	232 000 $
300-FL-01	Primary Drum Filter (12' x 16')	1	3660 mm dia x 4880	3,75	281 200 $	3 993 $	25 000 $	10 000 $	7 000 $	327 193 $
300-FL-02	Primary Drum Filter (12' x 16')	1	3660 mm dia x 4880	3,75	281 200 $	3 993 $	25 000 $	10 000 $	7 000 $	327 193 $
300-RP-01	Primary Repulper	1		7,41	45 000 $	6 738 $	8 000 $	15 000 $	1 500 $	76 238 $
300-CP-01	Primary Repulper Pump Box	1	1600 X 2000		3 587 $		4 000 $	5 000 $	500 $	13 087 $
300-PB-02	Primary Repulper Pump	1	8" X 6" SRL	18,65	21 000 $	9 820 $	3 600 $	7 000 $	500 $	41 920 $
300-RE-09/10	Secondary Filtrate Tanks	1	1200 X 2,500		37 500 $		5 000 $	8 000 $	500 $	51 000 $
300-PS-03/04	Secondary Filtrate Pumps	1	5" X 4" SRL	29,6	incl.	3 680 $	3 600 $	10 000 $	500 $	17 780 $
300-PV-02	Secondary Vacuum Pump	1		150	200 000 $	9 000 $	8 000 $	12 000 $	3 000 $	232 000 $
300-FL-03	Secondary Drum Filter (12' x 16')	1	3660 mm dia x 4880	3,75	281 200 $	3 993 $	25 000 $	10 000 $	7 000 $	327 193 $
300-FL-04	Secondary Drum Filter (12' x 16')	1	3660 mm dia x 4880	3,75	281 200 $	3 993 $	25 000 $	10 000 $	7 000 $	327 193 $
300-RP-02	Secondary Repulper	1		7,41	45 000 $	6 738 $	8 000 $	15 000 $	7 000 $	81 738 $
300-CP-02	Secondary Repulper Pump Box	1	1600 X 2000		3 587 $		4 000 $	5 000 $	500 $	13 087 $
300-PB-03	Secondary Repulper Pump	1	8" X 6" SRL	18,65	21 000 $	9 820 $	3 600 $	7 000 $	500 $	41 920 $
300-PP-01	Cyanidation Area Sump Pump	1	3" Vertical	7,41	8 800 $	6 738 $	3 600 $	5 000 $	500 $	24 638 $
300-RE 11	Barren Solution Tank	1	5000 X 7000		37 825 $		3 000 $	12 000 $	1 500 $	54 325 $
300-PS-05	Barren Solution Pump	1	5" X 4" SRL	14,8	13 000 $	12 280 $	3 600 $	20 000 $	1 000 $	49 880 $
300-PS-06	Preg Solution Pump	1	5" X 4" SRL	14,8	13 000 $	12 280 $	3 600 $	20 000 $	1 000 $	49 880 $
300-RE-12	Preg Solution Tank	1	5000 X 7000		37 825 $		3 000 $	12 000 $	2 500 $	55 325 $
300-RE-13	Grinding Solution Tank	1	5000 X 7000		37 825 $		3 000 $	20 000 $	2 500 $	63 325 $
300-PS-07	Grinding Solution Pump	1	5" X 4" SRL	14,8	13 000 $	12 280 $	3 600 $	8 000 $	500 $	37 380 $
300-CL-01	Bag Clarifier Tank	1	30 bags		50 000 $		7 000 $	15 000 $	1 500 $	73 500 $
300-RE-14	Merrill-Crowe Tank	1	2000 X 5000		17 000 $		4 000 $	15 000 $	500 $	36 500 $
300-PSCV-01	Zinc Conveyor	1	300 X 2000	0,12						- $
300-CV-01	Zinc Feed Belt Conveyor	1	100 X 2000	0,12	3 500 $	1 000 $	800 $	- $	300 $	5 600 $
300-CO-01	Zinc Cone Feeder	1	18"X2"X24H		1 000 $		2 000 $	2 000 $	500 $	5 500 $
300-PS-08	Precipitation Pump	1	6" X 4" X 14"	37,2	22 800 $	7 090 $	3 600 $	15 000 $	500 $	48 990 $
300-PR-01	Perrin press c/w Hydraulic Closure System	1	36" X 36" X 40 Frames							- $
300-PR-02	Perrin Press c/w Hydraulic Closure System	1	36" X 36" X 40 Frames							- $
300-PR-01	Filter Press c/w Hydraulic Closure System	1	914 X 914 X 10 Frames		8 954,40 $	- $	5 000 $	2 000 $	750 $	16 704 $
300-PR-02	Filter Press c/w Hydraulic Closure System	1	914 X 914 X 10 Frames		8 954,40 $	- $	5 000 $	2 000 $	750 $	16 704 $
300-VC-03	Merrill-Crowe Vacuum Pump	1	300 SCFM	7,41	14 500 $	6 738 $	5 000 $	8 000 $	1 500 $	35 738 $
300-FB-01	Flux Bin	1			800 $	- $	1 000 $	- $	500 $	2 300 $
300-CF-01	Calciner (Silver Precipitate)	1		100	117 000 $	15 000 $	5 000 $	- $	10 000 $	147 000 $
300-CF-02	Calciner Accessories	1			25 000 $	- $	- $	- $	- $	25 000 $
300-CF-03	Piping and Ventilation	1			15 000 $	3 000 $	5 000 $	7 000 $	- $	30 000 $
300-CF-05	Dust Collector	1			62 000 $	3 000 $	15 000 $	8 000 $	- $	88 000 $
300-WF-01	Electric Arc Refining Furnace	1		300	166 863 $	30 000 $	8 000 $	1 000 $	3 000 $	208 863 $
300-VE-01	Furnace Exhaust Fan	1		7,5	5 000 $	3 680 $	4 000 $		1 000 $	13 680 $
300-CP-01	# 1 Cyanidation Compressor	1	2500 CFM low pressure	148,2	77 500 $	12 000 $	5 000 $	40 000 $	1 000 $	135 500 $
300-CP-02	# 2 Cyanidation Compressor	1	2500 CFM low pressure	148,2	77 500 $	12 000 $	5 000 $	40 000 $	1 000 $	135 500 $
	TOTAL CYANIDATION			**1445**	**4 803 675 $**	**294 154 $**	**354 800 $**	**538 500 $**	**111 300 $**	**6 102 429 $**
ZINC FLOTATION SECTION										
400-RE-01	Cyanidation Tails Aeration Tank	1			36 600 $	- $	20 000 $	15 000 $	6 000 $	77 600 $
400-AG-01	Cyanidation Tails Aeration Agitator	1		18,75	19 000 $	15 000 $	7 000 $	6 000 $	3 000 $	50 000 $
400-CO-01/02	Zinc Flotation Conditioner Tanks No. 1 et 2	2	3050 mm x 3350		9 287 $	0	8 000 $	10 000 $	2 500 $	29 787 $
400-EC-01	Zn Flotation Feed Sampler	1		1,5	6 000 $	3 000 $	1 500 $	3 000 $	500 $	14 000 $
400-AG-02/03	Zn Flotation Conditioner Agitators	2	1067mm dia.	11,25	30 600 $	11 798 $	6 000 $	6 000 $	2 500 $	56 898 $
400-FL-01/06	Zinc Rougher Flotation Cells	6	8,5 m³ each	56,2	275 448 $	50 375 $	12 000 $	25 000 $	2 500 $	365 323 $
400-FL-07/10	Zinc Scavenger Flotation Cells	4	8,5 m³ each	44,8	207 460 $	29 670 $	10 000 $	12 000 $	2 500 $	261 630 $
400-FL-11/14	Zn 1st and 2nd Cleaner Flotation Cells	4	1,4 m³ each	16	82 800 $	25 920 $	5 000 $	12 000 $	2 500 $	128 220 $
400-FL-15/16	Zn 1st Cleaner Scavenger Flotation Cells	2	1,4 m³ each	8	41 400 $	25 920 $	5 000 $	12 000 $	2 500 $	86 820 $
400-EC-05	Zinc Concentrate Sampler	1		1,5	6 000 $	3 000 $	1 500 $	3 000 $	500 $	14 000 $

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Equlp. No.	Equipment List Description	Qty	Capaclty or Dimensions	Power kW	Equipment cost	Total Electrical	Total Mechanical	Total Piping	Transport	Total cdn$
400-EC-04	1st Cleaner Scavenger Tails Sampler	1		1,5	6 000 $	3 000 $	1 500 $	3 000 $	500 $	14 000 $
400-CP-01	Zn Scavenger Tails Pump Box 3,8 m^3	1	1600 mm X 1700 mm		2 684 $	0	3 600 $	3 000 $	500 $	9 784 $
400-PB-01/02	Zn Scavenger Tails Pumps (150 m^3/h)	2	127mmx100mm	50	26 000 $	15 480 $	3 600 $	3 000 $	500 $	48 580 $
400-PB-03/04	Zn Flotation Feed Pumps (200 m^3/h)	2	200mmx150mm	50	27 000 $	15 480 $	3 600 $	3 000 $	500 $	49 580 $
400-PN-01	Zn Rougher Concentrate Integrated Tank Pump	1	100 mm	11,25	19 380 $	6 805 $	3 600 $	5 000 $	500 $	35 285 $
400-PN-02	Zn Scavenger Concentrate Integrated Tank Pump	1	50 mm	7,5	11 760 $	6 805 $	3 600 $	4 000 $	500 $	26 665 $
400-PN-03	Zn 1st Cleaner Concentrate Integrated Tank Pump	1	100 mm dia.	11,25	19 380 $	9 618 $	3 600 $	5 000 $	500 $	38 098 $
400-PN-04	Zn 2nd Cleaner Tails Integrated Tank Pump	1	50 mm dia.	7,5	11 760 $	6 805 $	3 600 $	5 000 $	500 $	27 665 $
400-PN-05	Zn 2nd Cleaner Concentrate Integrated Tank Pump	1	100 mm dia.	11,25	19 380 $	9 618 $	3 600 $	5 000 $	50 $	37 648 $
400-EC-03	Zn Scavenger Tails Sampler	1		1,5	6 000 $	3 993 $	3 000 $	4 000 $	300 $	17 293 $
400-EC-02	Zn Rougher Concentrate Sampler	1		1,5	6 000 $	3 993 $	3 000 $	4 000 $	300 $	17 293 $
400-PE-01/05	Integrated Tank Pumps for Sample Transfer	5	32 mm each	11,25	34 820 $	14 340 $	4 500 $	6 250 $	500 $	60 410 $
400-PP-01	Zinc Flotation Area Sump Pump	1	50 mm	3,75	7 500 $	3 993 $	4 000 $	7 000 $	500 $	22 993 $
400-PR-01	Flotation Area Overhead Crane	1	5 t capacity	5,5	21 000 $	5 305 $	5 000 $	0	1 500 $	32 805 $
	TOTAL ZINC FLOTATION			**332**	**933 259 $**	**269 918 $**	**125 800 $**	**161 250 $**	**32 150 $**	**1 522 377 $**
PYRITE FLOTATION SECTION										
500-CO-01	Pyrite Flotation Conditioner Tank	1	3050 mm x 3350		9 287 $	0	6 000 $	5 000 $	1 250 $	21 537 $
500-AG-01	Conditioner Tank Agitator	1	1067 mm dia.	5,6	17 500 $	5 380 $	5 000 $	4 000 $	1 250 $	33 130 $
500-FL-01/05	Pyrite Rougher Flotation Cells	5	8,5 m^3 each	33,9	259 325 $	29 895 $	8 000 $	15 000 $	2 500 $	314 720 $
500-FL-06/10	Pyrite Scavenger Flotation Cells	5	8,5 m^3 each	33,6	259 325 $	29 670 $	10 000 $	12 000 $	2 500 $	313 495 $
500-CP-01	Plant Tailings Pump Box 5,2 m^3	1	1800 mm X 2000 mm		3 587 $	0	4 000 $	3 000 $	500 $	11 087 $
500-PB-01/02	Plant Tailings Pumps (104.2 m^3/h)	2	127 mm x 100 mm	50	26 000 $	20 480 $	7 200 $	8 000 $	500 $	62 180 $
500-CP-02	Pyrite Concentrate Pump Box 5,2 m^3	1	1800 mm X 2000 mm		3 587 $	0	4 000 $	3 000 $	500 $	11 087 $
500-PB-03/04	Pyrite Concentrate Pumps (104.2 m^3/h)	2	127 mm x 100 mm	50	26 000 $	20 480 $	7 200 $	8 000 $	500 $	62 180 $
500-EC-01	Pyrite Rougher Tails Sampler	1		1,5	6 000 $	3 993 $	3 000 $	4 000 $	500 $	17 493 $
500-EC-02	Pyrite Concentrate Sampler	1		1,5	6 000 $	3 993 $	3 000 $	4 000 $	500 $	17 493 $
500-PE-01/02	Integrated Tank Pumps for Sample Transfer	2	32 mm each	3,75	6 964 $	3 993 $	3 600 $	5 000 $	500 $	20 057 $
500-PP-01	Pyrite Flotation Area Sump Pump	1	50 mm dia.	3,75	7 500 $	3 993 $	3 600 $	7 000 $	500 $	22 593 $
	TOTAL PYRITE FLOTATION			**184**	**631 075 $**	**121 877 $**	**64 600 $**	**78 000 $**	**11 500 $**	**907 052 $**
ON STREAM X-RAY ANALYSER SECTION										
700-XA-01	On Stream X-Ray Analyser	1			560 000 $					560 000 $
700-ME-01/02	On Stream Analyser Multiplexers	2			incl.					
700-DM-01/02	On Stream Analyser Demultiplexers	2			incl.					
700-PR-01/03	Sample Return Pumps	3	32 mm each	5,62	10 446 $	5 990 $	5 400 $	7 500 $	500 $	29 836 $
	TOTAL ON SREAM ANALYSER SYSTEM			**6**	**570 446 $**	**5 990 $**	**5 400 $**	**7 500 $**	**500 $**	**589 836 $**
Zinc CONCENTRATE DEWATERING SECTION										
600-BO-01	Zinc Concentrate Thickener Feed Box	1			1 178 $	- $	3 000 $	3 500 $	2 500 $	10 178 $
600-RE-01	Zinc Concentrate Thickener Tank	1	9800 mm dia.		51 874 $	- $	36 000 $	13 500 $	23 090 $	124 464 $
600-RA-01	Zinc Concentrate Thickener Rake Mechanism	1		2,5	262 000 $	3 055 $	8 000 $	5 000 $	4 500 $	282 555 $
600-PB-01/02	Thickener Underflow Pumps (Peristaltic Type)	2	50 mm x 50mm	5,6	15 462 $	6 560 $	3 600 $	6 000 $	2 500 $	34 122 $
600-RE-02	Thickener Overflow Tank 2,5 m^3	1	1500 mm X 3000 mm		3 819 $	- $	3 000 $	3 500 $	2 500 $	12 819 $
600-PS-01	Thickener Overflow Pump	1	100 mm x 75	7,5	8 900 $	6 805 $	3 600 $	5 000 $	2 500 $	26 805 $
600-RE-03	Zinc Concentrate Surge Tank	1	3600 mm dia. x 4000 mm		14 184 $	- $	3 000 $	3 500 $	3 500 $	24 184 $
600-AG-01	Surge Tank Agitator	1	1080 mm dia.	10	15 990 $	8 680 $	5 000 $	- $	2 500 $	32 170 $
600-PB-03/04	Pressure Filter Feed Pumps	2	50mmx50mm	5,6	5 900 $	6 560 $	3 600 $	5 500 $	2 500 $	24 060 $
600-FP-01	Zinc Concentrate Horizontal Pressure Filter	1	40 m^2		975 000 $	- $	16 000 $	40 000 $	8 500 $	1 039 500 $
600-CH-01	Pressure Filter Discharge Chute	1			incl.	- $	3 000 $	12 000 $	2 500 $	17 500 $

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Equlp. No.	Equipment List Description	Qty	Capaclty or Dimensions	Power kW	Equipment cost	Total Electrical	Total Mechanical	Total Piping	Transport	Total cdn$
600-RE-04	Filtrate Release Tank	1			incl.	- $	3 600 $	6 500 $	2 500 $	12 600 $
600-PS-02	Filtrate Pump	1	40 mm	2,25	incl.	2 868 $	5 000 $	- $	2 500 $	10 368 $
600-PP-01	Zinc Thickening and Filtration Area Sump Pump	1	50 mm	3,75	7 500 $	3 993 $	3 600 $	7 000 $	2 500 $	24 593 $
600-MO-01	Overhead Crane - Zn Thickening and Filtration Area	1	2 t capacity	1	20 000 $	1 930 $	6 000 $	- $	4 500 $	32 430 $
	TOTAL ZINC DEWATERING			**38**	**1 381 807 $**	**40 451 $**	**106 000 $**	**111 000 $**	**69 090 $**	**1 708 348 $**
CHEMICAL REAGENTS SECTION										
800-SI-01	Quick Lime Silo	1	4700 dia. x 18500		94 000 $	- $	2 500 $	- $	6 000 $	102 500 $
800-DE-01	Quick Lime Silo Dust Collector	1			9 500 $	- $	1 000 $	3 500 $	1 000 $	15 000 $
800-AC-01	Quick Lime Silo Activator	1		1,1	7 500 $	3 680 $	1 500 $	- $	500 $	13 180 $
800-AL-01	Screw Feeder	1	152 mm dia.	0,75	16 000 $	3 680 $	1 500 $	- $	500 $	21 680 $
800-EX-01	Quick Lime Slaker	1	640 kg/h	1,5	136 600 $	3 680 $	2 500 $	2 500 $	2 500 $	147 780 $
800-CV-01	Lime Slaker Grit Screw Conveyor	1	152 mm dia.	0,4	incl.	3 680 $	1 500 $	- $	- $	5 180 $
800-B0-01	Grit Box	1			incl.	- $	500 $	- $	- $	500 $
800-RE-01	Lime Holding Tank	1	4700 dia. x 5000		23 195 $	- $	1 500 $	4 000 $	500 $	29 195 $
800-AG-01	Lime Holding Tank Agitator	1	1410 mm dia.	5,6	17 500 $	5 380 $	2 000 $	- $	250 $	25 130 $
800-PB-01/02	Lime Distribution Pumps	2	32 mm x 25,4 mm	11,2	15 300 $	10 760 $	3 600 $	4 000 $	250 $	33 910 $
800-PP-01	Lime Preparation Area Sump Pump	1	50 mm	5,6	7 500 $	5 380 $	1 800 $	8 000 $	500 $	23 180 $
800-RE-06	Bulk Sulfuric Acid Storage Tank	1	4270 mm x 4570		70 000 $	- $	1 500 $	4 000 $	500 $	76 000 $
800-PD-06	Sulfuric Acid Metering Pump	1	38 mm x 38	3,75	10 000 $	4 555 $	1 500 $	4 000 $	500 $	20 555 $
800- TR-01	NaCN Hopper	1			4 046 $	- $	500 $	- $	250 $	4 796 $
800-RE-02	NaCN Mixing Tank	1	2800 mm x 3100		4 046 $	- $	1 000 $	4 000 $	250 $	9 296 $
800-AG-02	NaCN Mixing Tank Agitator	1	934 mm	1,1	7 000 $	3 680 $	1 000 $	- $	250 $	11 930 $
800-PS-01	NaCN Transfer Pump	1	38 mm x 38	1,5	3 070 $	3 680 $	1 800 $	3 500 $	250 $	12 300 $
800-RE-03	NaCN Holding Tank	1	3200 mm x 3500		4 046 $	- $	1 000 $	4 000 $	500 $	9 546 $
800-PS-02/03	NaCN Distribution Pumps	2	32 mm x 25,4 mm	2,4	3 070 $	4 860 $	1 500 $	5 000 $	250 $	14 680 $
800- TR-02	$CuSO_4$ Hopper	1			4 046 $	- $	500 $	- $	250 $	4 796 $
800-RE-04	$CuSO_4$ Mixing Tank	1	2250 mm x 2550		4 046 $	- $	1 000 $	4 000 $	250 $	9 296 $
800-AG-03	$CuSO_4$ Mixing Tank Agitator	1	754 mm	1	7 000 $	3 680 $	1 000 $	4 000 $	250 $	15 930 $
800-PS-04	$CuSO_4$ Transfer Pump	1	38 mm x 38	1,5	3 070 $	3 680 $	1 800 $	4 000 $	250 $	12 800 $
800-RE-05	$CuSO_4$ Holding Tank	1	2550 mm x 2850		4 046 $	- $	1 000 $	3 500 $	500 $	9 046 $
800-PS-05/06	$CuSO_4$ Distribution Pumps	2		1,2	3 070 $	4 860 $	1 500 $	4 000 $	250 $	13 680 $
800- TR-04	PAX Hopper	1			4 046 $	- $	500 $	- $	250 $	4 796 $
800-RE-08	PAX Mixing Tank	1	2250 mm x 2550		4 046 $	- $	1 000 $	4 000 $	500 $	9 546 $
800-AG-05	PAX Mixing Tank Agitator	1	754 mm	1	7 000 $	3 680 $	1 000 $	- $	250 $	11 930 $
800-PS-08	PAX Transfer Pump	1	38 mm x 38	1,5	3 070 $	3 480 $	1 800 $	4 000 $	250 $	12 600 $
800-RE-09	PAX Holding Tank	1	2550 mm x 2850		4 046 $	- $	1 000 $	3 500 $	250 $	8 796 $
800-PS-09	PAX Distribution Pump	1		1,2	1 535 $	3 680 $	1 500 $	4 000 $	250 $	10 965 $
800-RE-10	MIBC Holding Tank	1	1370 mm x 1500		2 519 $	- $	1 000 $	- $	500 $	4 019 $
800-PD-01/03	MIBC Metering Pumps	3		0,5	7 675 $	9 650 $	1 000 $	4 000 $	250 $	22 575 $
800-RE-11	Cytec 3477 Holding Tank	1	1370 mm x 1500		4 046 $	- $	1 000 $	- $	1 500 $	6 546 $
800-PD-04/05	Cytec 3477 Metering Pumps	3		0,3	3 300 $	3 430 $	1 000 $	4 000 $	250 $	11 980 $
800-UF-01	Flocculent Preparation Packaged Unit by Supplier	1		4	48 000 $	4 180 $	1 800 $	6 000 $	1 000 $	60 980 $
800-PC-01/02	Progressive Cavity Pumps	2		4,5	9 000 $	4 555 $	1 500 $	4 000 $	500 $	19 555 $
800-MO-01	Reagent Area Hoist Monorail	1	2 t capacity	0,7	6 500 $	3 705 $	2 000 $	- $	500 $	12 705 $
800-PP-02	Reagents Area Sump Pump	1	50 mm	3,75	7 500 $	3 993 $	1 800 $	7 000 $	500 $	20 793 $
	TOTAL REAGENTS FACILITIES			**51**	**569 934 $**	**105 588 $**	**54 400 $**	**106 500 $**	**23 250 $**	**859 672 $**
SERVICES SECTION										
900-CP-01	Plant Air Compressor	1	1900 m^3/h	112	85 000 $	11 200 $	4 000 $	35 000 $	2 631 $	137 831 $
900-RE-01	Air Receiver	1	8m^3		10 032 $	- $	4 000 $	15 000 $	1 000 $	30 032 $
900-CP-02	Instrumentation Air Compressor	1	360 m^3/h	37,5	45 000 $	8 220 $	4 000 $	20 000 $	2 000 $	79 220 $
900-RE-02	Instrumentation Air Receiver	1	1m^3		4 500 $	- $	3 000 $	5 000 $	1 500 $	14 000 $
900-AS-01	Instrumentation Air Dryer	1			6 500 $	3 500 $	3 000 $	5 000 $	1 500 $	19 500 $
900-SA-01/02	Flotation Air Blowers	2	7000 Nm^3/h each	320	90 000 $	21 600 $	4 000 $	60 000 $	3 000 $	178 600 $
900-EP-01	Wet Scrubber	1			2 500 $	- $	4 000 $	6 000 $	1 000 $	13 500 $

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900-VE-01	Scrubbing Fan	1		5	2 500 $	4 930 $	4 000 $	- $	1 000 $	12 430 $
900-BA-01	Truck Scale	1			30 000 $	4 500 $	3 600 $	- $	3 000 $	41 100 $
900-PE-11	Fire Protection Pump (115 m³/h)	1	125 mm x 100	20	18 000 $	9 680 $	3 600 $	10 000 $	1 000 $	42 280 $
900-PE-12	Jockey Pump (5,0 m³/h)	1	40mmx32	3,75	7 500 $	3 993 $	3 600 $	5 000 $	500 $	20 593 $
900-PE-13	Diesel Pump (115 m³/h)	1	125 mm x 100		32 000 $	- $	5 600 $	8 000 $	1 000 $	46 600 $
900-PE-09/10	Raw WaterVvertical Submersible Pumps	2	100 mm dia.	30	32 000 $	46 800 $	4 000 $	20 000 $	1 000 $	103 800 $
900-PE-01/02	Reclaim Water Pumps (135 m³/h each)	2	200 mm dia.	80	35 000 $	9 600 $	4 000 $	20 000 $	4 000 $	72 600 $
900-BF-01	Reclaim Water Pumps Float barge	1			6 000 $	- $	4 000 $	- $	1 000 $	11 000 $
900-RE-04	Fresh Water Tank (540 m³)	1	8,9 m x 9,2		102 362 $	- $	4 000 $	5 000 $	4 000 $	115 362 $
900-PE-05/06	Fresh water Pumps (170 m³/h each)	2	75mmx50	20	7 450 $	12 860 $	7 200 $	20 000 $	1 000 $	48 510 $
900-PE-07/08	Seal Water Pumps (10 m³/h each)	2	38 mm x 25	15	6 792 $	6 860 $	7 200 $	35 000 $	1 000 $	56 852 $
	TOTAL SERVICES			**643**	**523 136 $**	**143 743 $**	**76 800 $**	**269 000 $**	**31 131 $**	**1 043 810 $**
	TOTAL SUPPLY EQUIPMENT				**16 538 570 $**	**1 375 159 $**	**1 377 420 $**	**1 412 860 $**	**485 116 $**	**21 189 125 $**
LABORATORY SECTION										
1000-LA-01	Jaw Crusher	1		2,25	5 663 $	750 $	1 000 $	- $	500 $	7 913 $
1000-LA-02	Lab Crusher Gyroll	1		0,375	7 534 $	750 $	1 000 $	- $	500 $	9 784 $
1000-LA-03	Bico Pulvirizer	1		2,25	5 316 $	750 $	1 000 $	- $	500 $	7 566 $
1000-LA-04	Shift Shaker	1		0,125	1 848 $	500 $	300 $	- $	500 $	3 148 $
1000-LA-05	Wet Sieving Kit	1			308 $	- $	- $	- $	- $	308 $
1000-LA-06	Ohaus Plateform Balance	1			1 175 $	- $	300 $	- $	500 $	1 975 $
1000-LA-07	Ohaus Dialogram Balance	1			213 $	- $	300 $	- $	500 $	1 013 $
1000-LA-08	Chan Automatic Micro Balance	1			16 509 $	- $	1 000 $	- $	500 $	18 009 $
1000-LA-09	One Batch Vaccuum Filter	1			666 $	- $	- $	- $	- $	666 $
1000-LA-10	Miscellaneous Filter Equipment	1			69 $	- $	- $	- $	- $	69 $
1000-LA-11	Atomic Absorbtion c/w Accessories	1			35 495 $	500 $	500 $	500 $	500 $	37 495 $
1000-LA-12	Leeco Sulfur Analizer and Accessories	1			65 000 $	500 $	500 $	1 000 $	500 $	67 500 $
1000-LA-13	Fume Hood, Benches, Glassware, etc…	1			100 000 $	750 $	35 000 $	3 000 $	500 $	139 250 $
	Laboratory Electric Hardware and Installation	1				17 465 $				17 465 $
	TOTAL LABORATORY			**5**	**239 796 $**	**21 965 $**	**40 900 $**	**4 500 $**	**5 000 $**	**312 161 $**
	Total energy in Kw			5095						
	TOTAL SUPPLY EQUIPMENT + LABORATORY				**16 778 366 $**	**1 397 124 $**	**1 418 320 $**	**1 417 360 $**	**490 116 $**	**21 501 286 $**

Mines Abcourt Inc.

Abcourt Project

Structural and Architectural

Appendix C

Description	Units	Man-hours	Total $	Total $	TOTAL $
			Material	Man-hours	
CRUSHER PLANT					
Structural light steel	8 t	200	26 000 $	13 000 $	39 000 $
Inclined grizzly	5 t	60	15 500 $	3 900 $	19 400 $
Crusher hopper 50 tonnes	40 t	1040	120 000 $	67 600 $	187 600 $
Steel Grating	180 m2	275	27 000 $	17 875 $	44 875 $
Handrails	100 m	230	13 500 $	14 950 $	28 450 $
Stairs clw handrails	20 m	40	16 000 $	2 600 $	18 600 $
Stacker conveyor	22 t	500	71 500 $	32 500 $	104 000 $
Total Crushing			**443 400 $**	**222 950 $**	**441 925 $**
CRUSHER BUILDING					
Structural heavy steel	49 t	1050	151 900 $	68 250 $	220 150 $
Metal roof deck insulated	87 m2	87	4 785 $	5 655 $	10 440 $
Roof cladding un-insulated	117 m2	58	2 925 $	3 770 $	6 695 $
Wall cladding insulated	185 m2	100	10 175 $	6 500 $	16 675 $
Walls cladding un-insulated	185 m2	80	4 625 $	5 200 $	9 825 $
Men doors	5	60	5 500 $	3 900 $	9 400 $
Truck door	2	160	10 000 $	10 400 $	20 400 $
Total Crusher Building			**189 910 $**	**103 675 $**	**293 585 $**
PROCESS PLANT					
GRINDING AREA					
Sag feed conveyor	9,8 t	250	31 850 $	16 250 $	48 100 $
Conveyor belt cover	2 t	50	6 500 $	3 250 $	9 750 $
Conveyor belt cover	82 m2	12	266 500 $	780 $	267 280 $
Structurallight steel	15 t	400	48 750 $	26 000 $	74 750 $
Steel Grating	550 m2	800	82 500 $	52 000 $	134 500 $
Handrails	245 m	575	33 075 $	37 375 $	70 450 $
Stairs c/w handrails	70 m	150	56 000 $	9 750 $	65 750 $
Total Grinding			**525 175 $**	**145 405 $**	**670 580 $**
FLOTATION					
Structural light steel	6 t	75	19 500 $	4 875 $	24 375 $
Steel grating	682 m2	1025	102 300 $	66 625 $	168 925 $
Handrails	35 m2	75	4 725 $	4 875 $	9 600 $
Stairs clw handrails	30 m2	65	24 000 $	4 225 $	28 225 $
Total Flotation			**150 525 $**	**80 600 $**	**231 125 $**
REAGENT AREA					
Structurallight steel	22 t	575	68 200 $	37 375 $	105 575 $
Steel Grating	410 m2	620	61 500 $	40 300 $	101 800 $
Handrails	35 m	80	4 725 $	5 200 $	9 925 $
Stairs clw handrails	8m	20	6 400 $	1 300 $	7 700 $
Total Reagent			**140 825 $**	**84 175 $**	**225 000 $**

Mines Abcourt Inc.

Abcourt Project

Structural and Architectural

Appendix C

Description	Units	Man-hours	Total $ Material	Total $ Man-hours	TOTAL $
PRESSES FILTRATION					
Structurallight steel	4 t	125	13 000 $	8 125 $	21 125 $
Steel grating	12 m2	40	1 800 $	2 600 $	4 400 $
Handrails	26 m	60	3 510 $	3 900 $	7 410 $
Ladder	12 m2	24	1 440 $	1 560 $	3 000 $
Total Dewatering			**19 750 $**	**16 185 $**	**35 935 $**
CONCENTRATE STORAGE					
Structural light steel	21 t	550	68 250 $	35 750 $	104 000 $
Total Concentrate Storage			**68 250 $**	**35 750 $**	**104 000 $**
MERRILL-CROWE PROCESS					
Filters and presses area					
Structural light steel	20 t	500	62 000 $	32 500 $	94 500 $
Handrails	150 m	350	20 250 $	22 750 $	43 000 $
Stairs c/w handrails	40 m	100	32 000 $	6 500 $	38 500 $
Total Filters and Presses Area			**114 250 $**	**61 750 $**	**176 000 $**
LEACHING/THICKENER AREA					
Structural light steel	10 t	400	32 500 $	26 000 $	58 500 $
Steel grating	75 m2	250	11 250 $	16 250 $	27 500 $
Handrails	130 m	300	17 550 $	19 500 $	37 050 $
Ladder	12 m	24	1 440 $	1 560 $	3 000 $
Stairs clw handrails	15 m	40	12 000 $	2 600 $	14 600 $
Total Leaching			74 740 $	65 910 $	140 650 $
Total Process Plant					**1 583 290 $**
Process Building					
Grinding area structural heavv steel	104 t	2140	322 400 $	139 100 $	461 500 $
Flotation structural heavy steel	85 t	1965	263 500 $	127 725 $	391 225 $
Reagent area structural heavy steel	88 t	1875	286 000 $	121 875 $	407 875 $
Presses filtration structural heavy steel	20 t	415	62 000 $	26 975 $	88 975 $
Concentrate storage structural heavy steel	30 t	680	93 000 $	44 200 $	137 200 $
Merril Crowed structural heavy steel	140 t	2600	434 000 $	169 000 $	603 000 $
Leaching/thickener area structural heavy steel	25 t	800	77 500 $	52 000 $	129 500 $
Processing plant					
Metal roof deck insulated	2738 m2	2738	150 590 $	177 970 $	328 560 $
Roof cladding un-insulated	3422 m2	1711	85 550 $	111 215 $	196 765 $
Wall cladding insulated	3636 m2	3636	199 980 $	236 340 $	436 320 $
Walls cladding un-insulated	3636 m2	1818	90 900 $	118 170 $	209 070 $
Men doors	30	360	33 000 $	23 400 $	56 400 $
Truck doors	6	480	30 000 $	31 200 $	61 200 $
Total Processing Plant			**2 128 420 $**	**1 379 170 $**	**3 507 590 $**

Mines Abcourt Inc.

Abcourt Project

Structural and Architectural **Appendix C**

Description	Units	Man-hours	Total $ Material	Total $ Man-hours	TOTAL $
1,800 t CRUSHED ORE BIN					
Ore bin wall cladding insulated	470 m2	220	25 850 $	14 300 $	40 150 $
Ore bin wall cladding insulated	80 m2	100	4 400 $	6 500 $	10 900 $
Bin concrete wall cladding insulated	440 m2	220	24 200 $	14 300 $	38 500 $
Bin ladder	16 m	35	2 000 $	2 275 $	4 275 $
Total Ore Bin					**93 825 $**
GRAND TOTAL					**5 920 215 $**

Mines Abcourt Inc.

Abcourt Project

Concrete and Excavation Requirements

Appendix C

Description	Concrete	Excavation	Backfill	Total	Total $	Total $	Total $
	M³	M³	M³	Concrete	Excavation	Backfill	CA
CRUSHER PLANT							
Retaining Walls Base	100	340	240	68 000 $	1 870 $	2 160 $	72 030 $
Retaining Walls Stem	65			44 200 $			44 200 $
Crusher bin Grizzly	75			51 000 $			51 000 $
Belt conveyor No. 2	22	75	60	14 960 $	413 $	540 $	15 913 $
Total Crusher Plant							**183 143 $**
CRUSHING BUILDING							
Foundation Footings	75	250	150	51 000 $	1 375 $	1 350 $	53 725 $
Foundations Pedestals	22	75	25	14 960 $	413 $	225 $	15 598 $
Foundation Slab	200	200	200	136 000 $	1 100 $	1 800 $	138 900 $
Foundation Grade Beams	45	150		30 600 $	825 $		31 425 $
Anchor Bolts and Grout	1,8			1 224 $			1 224 $
Total Crusher Building							**240 872 $**
PROCESS PLANT:							
Belt conveyor No. 1	15	50	40	10 200 $	275 $	360 $	10 835 $
Foundation Footings	225	400	340	153 000 $	2 200 $	3 060 $	158 260 $
Foundations Pedestals	70	125	100	47 600 $	688 $	900 $	49 188 $
Foundations floors	400	600	250	272 000 $	3 300 $	2 250 $	277 550 $
Anchor Bolts and Grout	4,5			3 060 $			3 060 $
Pump Box and Pumps bases	90	157	130	61 200 $	864 $	1 170 $	63 234 $
Misc. Offices, Dry, Lunch Room, Wash Rooms.....	15			22 000 $			22 000 $
Concrete blocks Compressors, electric rooms......	55 t						88 000 $
Total Process Plant							**672 127 $**
GRINDING SECTION 200							
Sag mill feed conveyor	6	10	9	4 080 $	55 $	81 $	4 216 $
SAG mill	200	350	250	136 000 $	1 925 $	2 250 $	140 175 $
Ball mill	190	330	240	129 200 $	1 815 $	2 160 $	133 175 $
Ball bins	55	100	90	37 400 $	550 $	810 $	38 760 $
Total Grinding							**316 326 $**
ZINC FLOTATION & REGRIND SECTION 300							
Flotation cells	48	200	175	32 640 $	1 100 $	1 575 $	35 315 $
Regrind mill	84	150	100	57 120 $	825 $	900 $	58 845 $
Total Zinc Flotation							**94 160 $**

Mines Abcourt Inc.

Abcourt Project

Concrete and Excavation Requirements

Appendix C

Description	**Concrete**	**Excavation**	**Backfill**	**Total**	**Total $**	**Total $**	**Total $**
	M3	**M3**	**M3**	**Concrete**	**Excavation**	**Backfill**	**CA**
Zinc Dewatering							
Zinc Concentrate thickener	70	60	30	47 600 $	385 $	270 $	48 255 $
Zinc Concentrate surge tank	10	10	8	6 800 $		72 $	6 872 $
Pressure filter floor	70	100	30	47 600 $	550 $	270 $	48 420 $
Concentrate wall	156	100	25	106 080 $	550 $	225 $	106 855 $
Concentrate floor	46	70	30	31 280 $	385 $	270 $	31 935 $
Concentrate load-out	110	200	100	74 800 $	1 100 $	900 $	76 800 $
Total Zinc Dewatering							**319 137 $**

MERRILL-CROWE PLANT 500							
Concrete floors	550 m2	600			82 500 $	39 000 $	121 500 $
Foundations Footings	60	100	90	40 800 $	550 $	810 $	42 160 $
Foundations Pedestals	20	30	25	13 600 $	165 $	225 $	13 990 $
Foundations floors	275	200	50	187 000 $	1 100 $	450 $	188 550 $
Anchor Bolts and Grout	1,5			1 020 $			1 020 $
Pump Box and Pumps bases	80	110	90	54 400 $	605 $	810 $	55 815 $
Thickener tank 9800	70	60	10	47 600 $	330 $	90 $	48 020 $
5- Leach tanks 1140m3	660	300	450	448 800 $	1 650 $	4 050 $	454 500 $
Filtrate tank (2)	6			4 080 $			4 080 $
6- Drum filters	40	200	175	27 200 $	1 100 $	1 575 $	29 875 $
Barren solution tank	30	20	10	20 400 $	110 $	90 $	20 600 $
Preg solution tank	30	20	10	20 400 $	110 $	90 $	20 600 $
Process solution tank	30	20	10	20 400 $	110 $	90 $	20 600 $
Clarifier concrete floor	15			10 200 $			10 200 $
Perrin presses floor	15			10 200 $			10 200 $
Merrill-Crowe vacuum	4	2	1	2 720 $	11 $	9 $	2 740 $
Wabi furnace	6			4 080 $			4 080 $
Leaching compressors	15	8	4	10 200 $	44 $	36 $	10 280 $
Filters vacuum	8	4	2	5 440 $	22 $	18 $	5 480 $
Total Merrill-Crowe							**1 064 290 $**

CHEMICAL REAGENTS SECTION 600							
Lime silo and accessories	36	20	10	24 480 $	110 $	90 $	24 680 $
NaCN system	8	4	2	5 440 $	22 $	18 $	5 480 $
Cu804 system	8	4	2	5 440 $	22 $	18 $	5 480 $
802 tank *clw* accessories	20	8	4	13 600 $	44 $	36 $	13 680 $
PAX system	8	4	2	5 440 $	22 $	18 $	5 480 $
Flocculent preparation packaged unit by supplier	2			1 360 $			1 360 $
3418 A system	8	4	2	5 440 $	22 $	18 $	5 480 $
Total Reagents							**61 640 $**

Mines Abcourt Inc.
Abcourt Project
Concrete and Excavation Requirements

Appendix C

Description	Concrete	Excavation	Backfill	Total	Total $	Total $	Total $
	M3	M3	M3	Concrete	Excavation	Backfill	CA
Miscellaneous (Flotation)							
Compressors, Blower and Air Receivers	12	4	4	8 160 $	22 $	36 $	8 218 $
Vacuum pump	8	2	2	5 440 $	11 $	18 $	5 469 $
Propane tank	8	2		5 440 $	11 $	0 $	5 451 $
Truck scale	35	50	40	23 800 $	275 $	360 $	24 435 $
Process water tank (500 m3) Flotation	50	50	40	34 000 $	275 $	360 $	34 635 $
Concrete blocks for station secondary	15t						32 000 $
Total Micellaneaous							**110 208 $**
GRAND TOTAL							**2 637 888 $**

TOTAL: Process Plant Foundation = 1,796,930 $ Building = 840,958 $

Laboratory							
Laboratory							143 435 $
Total Laboratory				**0 $**	**0 $**	**0 $**	**143 435 $**

1800t Crushed Ore Bin							
Coarse ore	310	540	475	210 800 $	2 970 $	4 275 $	218 045 $
Total 1800t Crushed Ore Bin				**210 800 $**	**2 970 $**	**4 275 $**	**218 045 $**

GRAND TOTAL - General Concrete Works							**3 423 383 $**

Projet : B150-02-005_ABCOURT
Date : 31 octobre 2006
Révision :01

Estimation Budgétaire

Main power Supply Installation c/w MCC's

-4 transformateurs incluant les protections et le câblage	400 000.00$
-6 CCM'S	165 000.00$
-6 Breakers 600V 400-600A	24 000.00$
-Armoire blindé avec disjoncteur principal et les 4 INCOM. SWITCH	80 000.00$
-Bus et câblage	140 000.00$
Sous total :	809 000.00$
Total	**809 000.00$**

Notes :

- Les taxes TPS et TVQ applicables ne sont pas inclus.

APPENDIX 5

Details of mill operating costs

Appendix 5 Details of mill operating costs (Source: Bumigeme 2007 report).

Table 7.1 Summary of Detailed Operating Costs.

Description	Cost/Ton	Annual Cost
Staff Employees Salary	$ 0.49	$ 320,089
Hourly Employees Salary	$ 2.01	$ 1,306,622
Contractor (Security)	$ 0.06	$ 37,440
Miscellaneous Salary	$ 0.10	$ 65,032
Operating and Environmental Reagents	$ 5.75	$ 3,734,250
Smelting Ag. and Au. c/w Transport	$ 0.25	$ 164,176
Operating Consumables	$ 1.30	$ 842,200
Refinery Maintenances Supplies	$ 0.12	$ 78,000
Mechanical Maintenance Supplies	$ 0.60	$ 390,000
Electrical/Instrumentation Supplies	$ 0.30	$ 195,000
Piping Maintenance Supplies	$ 0.07	$ 45,500
Building Maintenance	$ 0.05	$ 32,500
Building Heating (Gas)	$ 0.16	$ 104,000
Plant Office Supplies	$ 0.04	$ 26,000
Health/Safety Supplies	$ 0.04	$ 26,000
Vehicles/Loader supplies	$ 0.05	$ 32,500
Power (Hydro-Quebec)	$ 2.07	$ 1,343,133
TOTAL COST	**$13.46**	**$ 8,742,442**

Appendix 5 (cont'd) Details of mill operating costs (Source: Bumigeme 2007 report).

Table 7.2 Manpower Costs.

Description	Annual Salary	Number of Employees	Total (Annual)	Cost per ton
Staff Employees				
Plant Superintendent/Metallurgist	$ 105,000	1	$ 120,897	$ 0.186
General Foreman	$ 67,000	1	$ 77,144	$ 0.119
Mechanic Foreman	$ 61,000	1	$ 70,235	$ 0.108
Technician	$ 45,000	1	$ 51,813	$ 0.080
Sub Total		**4**	**$ 320,089**	$ **0.492**
Hourly Employee	**Rate $/Hr**			
Crusher/Loader Operators	$ 21.00	3	$ 167,142	$ 0.257
Concentrator Senior Operators	$ 23.50	4	$ 249,386	$ 0.384
Concentrator Junior Operators	$ 22.00	4	$ 233,468	$ 0.359
Concentrator Helper Operator	$ 19.00	5	$ 252,039	$ 0.388
Maintenance Mechanic	$ 22.00	4	$ 233,468	$ 0.359
Maintenance Electric/Instrumentation	$ 22.00	1	$ 58,367	$ 0.090
Refiner Senior	$ 23.50	1	$ 62,346	$ 0.096
Refiner Helper	$ 19.00	1	$ 50,408	$ 0.078
Sub Total		**23**	**$1,306,622**	**$ 2.010**
Security (contractor)	$ 18.00	1	$ 37,440	$ 0.058
Sub Total		**1**	**$ 37,440**	**$ 0.058**
Miscellaneous	**Average/Rate**	**Hours**		
Maintenance Overtime	$ 22.00	1,250	$ 27,500	$ 0.042
Night Shifts Bonus	$ 4.00	2,920	$ 11,680	$ 0.018
Weekend Bonus	$ 8.00	1,248	$ 9,984	$ 0.015
Overtime Related to Holidays	$ 32.07	144	$ 4,618	$ 0.007
Maintenance Contractor	$ 45.00	250	$ 11,250	$ 0.017
Sub Total		**5,812**	**$ 65,032**	**$ 0.100**
TOTAL MANPOWER COST			**$1,729,183**	**$ 2.660**

Appendix 5 (cont'd) Details of mill operating costs (Source: Bumigeme 2007 report).

Table 7.3 Reagents and Consumable Costs.

Description	Consumption kg/ton	Reagents Costs $/kg	Total	Cost per ton
Processing Reagents				
Lime (Processing)	1.00	$ 0.16	$ 104.000	$ 0.16
Sodium Cyanide	1.00	$ 1.90	$1,235,000	$ 1.90
Cytec 3477	0.07	$ 5.50	$ 250,250	$ 0.39
MIBC	0.10	$ 3.20	$ 208,000	$ 0.32
Copper Sulphate	0.39	$ 2.53	$ 643,500	$ 0.99
PAX	0.44	$ 3.30	$ 949,000	$ 1.46
Caustic	0.02	$ 1.14	$ 13,000	$ 0.02
HCL Acid	0.05	$ 0.33	$ 13,000	$ 0.02
Sulfuric Acid	4.4	$ 0.07	$ 201,500	$ 0.31
Laboratory Supplies		$ 0.08	$ 52,000	$ 0.08
Flocculant	0.025	$ 3.80	$ 65,000	$ 0.10
Sub Total			**$3,734,250**	**$ 5.75**
Smelting (Ag Precipitate)				
Borax 40%	0.0460	$ 1.63	48,737	$ 0.07
Sodium Nitrate 30%	0.0344	$ 0.68	15,226	$ 0.02
Silica Sand 15%	0.0170	$ 0.25	2,763	$ 0.00
Soda Ash 8%	0.0920	$ 0.35	20,930	$ 0.03
Fluorspar 7%	0.0800	$ 0.51	26,520	$ 0.04
Au/Ag Ingots Transportation			50,000	$ 0.08
Sub Total			**164,176**	**$ 0.25**
Operating Consumables				
Jaw Crusher Liners (Sets)	1.0	$ 10,000/set	$ 10,000	$ 0.02
S.A.G. Liners (Sets)	1.6	$190,000/set	$ 304,000	$ 0.47
Ball Mill Liners (Set)	0.4	$ 78,000/set	$ 31,200	$ 0.05
S.A.G. Grinding Balls	300.0	$ 1,070/ton	$ 321,000	$ 0.49
Ball Mill Grinding Balls	200.0	$ 880/ton	$ 176,000	$ 0.27
Sub Total			**$ 842,200**	**$ 1.30**

Appendix 5 (cont'd) Details of mill operating costs (Source: Bumigeme 2007 report).

Table 7.4 Power Costs.

Hydro-Quebec Charges	
Cost of Power Demand:	$13.08/kW
Consumption Costs:	First 210,000 kWh = $ 0.042 Additional kWh = $ 0.0274
kW Total (Per Equipment List)	5,095 kW
Utilization	5,095 kW x 65% = 3,312 kW or 44.16 kWh/t
Based on 650,000 t/y	650,000 t/y x 44.16 kWh/t ÷ 12 = 2,392,000 kWh/month
Monthly Power Costs:	
Demand Charge	3,312 kW x $13.08 = $43,320.96/month
Consumption	First 210,000 x $ 0.042 = $ 8,820/month Balance 2,182,000 x $ 0.0274 =$ 59,786.80/month
Total per Month	$ 111,927.76
Cost per Ton	**$ 2.07**