UPDATE TO THE TECHNICAL FEASIBILITY REPORT ON THE ABCOURT-BARVUE DEPOSIT

Prepared for

ABCOURT MINES INC.
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By

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And

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1.0 SUMMARY

1.1 Introduction

Abcourt Mines Inc (Abcourt) is a Rouyn-Noranda based company contemplating to develop a mine-mill complex with open pit and underground operations on its Abcourt-Barvue property in the Abitibi region, Quebec, Canada. This property was in production by an open pit and an underground mine during two different periods. The project consists of three open pits, namely Barvue, Abcourt East and Abcourt West, and the development of an underground mine to produce a zinc-silver concentrate.

This National Instrument 43-101 Technical Report on the Abcourt-Barvue Project was prepared at the request of Abcourt to present the findings of an update to the Technical Feasibility Study Report On The Abcourt-Barvue Deposit dated February 15, 2007.

The Abcourt-Barvue Project is located in the Abitibi area, northwestern Quebec, Canada, 60 km north of the town of Val d'Or. 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.

Abcourt commissioned PRB Mining Services Inc. (PRB) and Bumigeme Inc. (Bumigeme) to update the 2007 feasibility study using current economic parameters, the Genivar 2007 pit design, and modifying the process flowsheet to eliminate the silver cyanidation circuit and produce solely a zinc-silver concentrate.

1.2 Land Tenure

The Abcourt-Barvue Project is owned 100% by Abcourt, consists of two mining concessions, two surface leases, and 103 claims covering 5,123.4 hectares. There are no options, royalties, or outstanding liens, encumbrances, or agreements on the proposed mining site.

1.3 Existing infrastructure

The property is located 60 km north of the mining community of Val-d'Or and 5.5 km from Barraute (Province of Quebec). It is easily reached all year round via highway 397 and by road 6-7 rang to the mine site. A commercial airport and rail service is located nearby. It is located in an active mining region where labour, supplies, and services are readily available.

1.4 History

The zinc-silver mineralization was discovered in 1950. The Abcourt-Barvue deposit was in operation during two periods: between 1952 and 1957 by Barvue Mines Limited and between 1985 and 1990 by Abcourt. In all 5,002,190 metric tonnes grading 38.74 g/t Ag and 2.98% Zn were mined from the Barvue open pit and 632,319 metric tonnes grading 131.65 g/t Ag and 5.04% Zn were mined from underground production. The tailings pond was recently rehabilitated by the Ministry of Natural Resources of the province of Quebec. Abcourt retains the mining concessions and surrounding cells to this day.

1.5 Geology and Mineralization

The property sits in the south-central part of the "Northern Volcanic Zone", and straddles the Figuery Group which hosts the Abcourt-Barvue volcanogenic deposit cluster. The geological units are dipping at 75° to the north with a well-developed E-W regional schistosity. The main deposit, which extends over 2 km, is located in a volcanoclastic sequence characterized by tuffs and agglomerates usually strongly silicified, carbonatized and sericitized. The Zn-Ag mineralization is located close to and partly within a major talc/sericite shear zone called Gray Schist (GS) by the Company's geologists.

1.6 Mineral Processing

Historical mineral recoveries during production were over 90% for zinc and 77% for silver. In 2017, metallurgical tests were performed in several laboratories. The cyclic flotation tests realized on the ore of Abcourt-Barvue have shown the possibility to recover 97.5% of the zinc and 77.8% of the silver in a Zn-Ag concentrate assaying 53.4% Zn and 740.6g/tm Ag.

1.7 Economic Analysis

Genivar Limited Partnership (Genivar) and Bumigeme Inc. produced a feasibility study in 2007 titled Technical Feasibility Study Report On The Abcourt-Barvue Deposit, dated February 15, 2007 (Genivar 2007 study). The mine plan developed in the study produced a total of 6,446,000 tonnes of ore grading 3.11% Zn and 54.96 gpt Ag, of which 5,338,700 tonnes to be extracted in open pit operations and 1,107,300 tonnes to be extracted from underground operations.

The open pit operation consisted of the expansion of the Barvue pit and the excavation of the Abcourt East and the Abcourt West pits over a period of 13 years. The pits will be excavated to depths of 166m, 72m, and 42m respectively. The underground operations consisted in mining stopes from a depth varying from 150m to 200 m below surface to the pit bottoms using the Avoca method. The underground work areas were accessed by excavating declines.

The processing plant was designed to process 650,000 tonnes per year. It had a cyanidation circuit and a flotation circuit producing silver ingots and a zinc-silver concentrate as sales products.

The Genivar 2007 study resulted in a mineral reserve of 6,446,000 tonnes grading 3.11% zinc and 54.96 gram per tonne silver.

A mineral resource estimate was produced by Jean-Pierre Bérubé in 2014 and described in a report titled NI 43-101 Mineral Resources Report For The Abcourt-Barvue

Property, dated February 28, 2014. The estimate returned measured and indicated resources (M&I) for the Abcourt-Barvue deposit totalling 8,086,000 tonnes grading 3.06% Zn and 55.38 g/t Ag of which the measured resources are 6,264,200 tonnes grading 43.93 g/t Ag and 3.09 % Zn and the indicated resources are 1,821,800 tonnes grading 94.75 g/t Ag and 2.98 % Zn.

The open pit resources were constrained within the Genivar 2007 pit shells. The measured and indicated resources increased significantly compared to the resource estimate used in the Genivar 2007 study because a lower cut-off grade was used reflecting better economic parameters. The increase in M&I mineral resource was most significant in the open pits and was insignificant underground.

A mine plan was developed for the 2018 mineral reserves using the Genivar 2007 pit design and underground mine design. The 2014 mineral resource diluted and recovered produced a total of 8,074,162 tonnes of mill feed grading 2.83% Zn and 51.79 gpt Ag, of which 6,589,361 tonnes were produced in open pit operations and 1,484,801 tonnes were produced in underground operations. The life of mine is 13 years.

The processing plant design remained at a mill feed of 650,000 tonnes per year but was modified by eliminating the cyanidation circuit to produce only a zinc-silver concentrate. Minor changes were brought to the surface infrastructure such as the installation of new 25kV power lines and the relocation of the waste rock stockpiles.

The project capital and operation costs were estimated for the new mine plan, modified process plant and site infrastructure using current costs. Mine rehabilitation costs were also estimated. Progressive rehabilitation costs were applied to years 3 to 13 with the remaining costs occurring in year 14. Project revenues were estimated using 1.10 US$/lb Zn, 16.50 US$/Oz Ag, 1.25 CA$/US$ exchange rate, and smelting terms.

An average of 32,000 tonnes per year of concentrate grading 52.7% Zn and 768 gpt Ag is produced. The project preproduction capital cost is estimated to 41.3M$ including 4.0M$ of working capital, and the sustaining capital cost is estimated to 18.1M$. The average operating cost is estimated to 39.94 $/t milled.

An economic analysis assuming 100% equity financing, a zinc price of 1.10 $/lb, a silver price of 16.50 $/Oz, an exchange rate of 1.25 CA$/US$, and an 8% discount rate resulted in a pre-tax cash flow of 170.0 million Canadian dollars, a pre-tax rate of return (IRR) of 26.1% and a pre-tax net present value (NPV) of 72.6 million Canadian dollars. Using a discount rate of 5%, as in the Genivar 2007 Report, the pre-tax NPV is estimated to 100.4 million Canadian dollars. The pre-tax payback period is 4.9 years. A sensitivity analysis on revenue, capital cost, and operating cost showed the project to be most sensitive to total revenue, in particular the price of zinc, followed by operating cost.

The economic analysis returned an after-tax cash flow of 106.7 million Canadian dollars, an after-tax rate of return (IRR) of 20.5% and an after-tax net present value (NPV) at 8% discount rate of 41.0 million Canadian dollars. The after-tax payback period is 5.3 years.

In comparison, the Genivar 2007 study's economic analysis used a zinc price of 1.15 $/lb, a silver price of 9.54 $/Oz, an exchange rate of 1.15 CA$/US$, and a 5% discount rate and, assuming 100% equity financing, returned a pre-tax cash flow of 138.7 million Canadian dollars, a pre-tax IRR of 27.1% and a pre-tax NPV at 5% discount rate of 87.6 million Canadian dollars.

2.0 INTRODUCTON

Abcourt Mines Inc. (Abcourt) is a Rouyn-Noranda based company contemplating to develop a mine-mill complex with open pit and underground operations on its Abcourt-Barvue property near Barraute in the Abitibi region, Quebec, Canada. In the past, this property was in production by an open pit and by an underground mine during two different periods. The project consists of three open pits, namely Barvue, Abcourt East and Abcourt West, and the development of an underground mine to produce a zinc-silver concentrate.

A feasibility study was produced in February 2007 by Genivar Limited Partership (Genivar) and Bumigeme Inc (Bumigeme) titled TECHNICAL FEASIBILITY STUDY REPORT ON THE ABCOURT-BARVUE DEPOSIT (the Genivar 2007 report). This report was later amended in November 2010 by Genivar and Bumigeme. The Genivar 2007 report based its mine plan on the resource estimate prepared by MRB and Associates in 2006.

The resource estimate was updated by Jean-Pierre Bérubé, Consulting Geologist, and was presented in a report titled NI 43-101 MINERAL RESSOURCES REPORT FOR THE ABCOURT-BARVUE PROPERTY dated February 28, 2014. This report is posted on Abcourt's Issuer Profile on Sedar's web site www.sedar.com.

Abcourt commissioned PRB Mining Services Inc. (PRB) and Bumigeme Inc. (Bumigeme) to update the Genivar 2007 feasibility study using current economic conditions, the Genivar 2007 pit design, and modifying the process flowsheet to eliminate the silver cyanidation circuits and produce solely a zinc-silver concentrate.

This technical report was prepared by PRB and Bumigeme with effective date January 15, 2019.

2.1 Qualified Persons

This technical report was prepared for Abcourt Mines Inc. by or under the supervision of Qualified Persons (QPs). PRB Mining Services Inc and Bumigeme Inc, are responsible for various sections of this report. The QPs responsible for this report, as defined by NI 43-101 and in compliance with Form 43-101F1 are as follows:

a) Mr. Florent Baril, ing., Bumigeme Inc., Montréal, Québec.
b) Mr. Paul Bonneville, ing., PRB Mining Services Inc., Val d'Or, Québec.

The QPs' areas of responsibility for the various Sections of the report are outlined in Table 2.1.

Table 2.1: Responsibilities of Qualified Persons

Qualified Person	Responsibility
Paul Bonneville	Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 15, 16, 18, 19, 20, 21, 22, 23, and portions of sections 1, 2, 3, 24, 25, 26, and 27 that are based on those sections.
Florent Baril	Sections 13, 17, and portions of sections 1, 2, 3, 21, 24, 25, 26, and 27 that are based on those sections.

2.2 Terms of Reference

Budgetary prices were obtained from various suppliers for several items including equipment, supplies, and infrastructure components. Other elements were compared to prices used in similar projects.

An exchange rate of 1.25 CA$ per US$ was assumed between the Canadian and the American dollars. The prices for zinc and silver used in the update were respectively 1.10 US$/lb and 16.50 US//Oz.

In evaluating updated revenues, typical industry smelting and refining terms were used.

The updated capital and operating costs were evaluated in 2018 Canadian dollars. An economic evaluation of the project was conducted using the Internal Rate of Return (IRR) and the Net Present Value (NPV) methods.

The updated revenues and costs presented in this report are compared to those of the 2007 Genivar report. The revenues and costs of the 2007 Genivar report were kept in 2007 Canadian dollars.

2.3 Sources of Information

Florent Baril, ing., Bumigeme, visited the site in the summer of 2018. Paul Bonneville, ing., PRB, visited the site in July 2018.

PRB prepared the report using data provided by Abcourt and the parties listed in Table 2.1. A portion of the information and technical data presented in this report came from technical reports listed below and previously filed on Sedar.

a) February 15, 2007: Technical Feasibility Study Report On The Abcourt-Barvue Deposit.
b) February 28, 2014: NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property.

WSP Canada acquired Genivar Limited in 2007. WSP Canada provided a copy of its archive files related to Genivar work on the Abcourt-Barvue property.

Sections 4 to 12 and section 14 were taken from the Mineral Resource Report of 2014. Abcourt provided information to update those sections and provided the current list of claims and drawings of the property limits.

Information regarding the tailings pond design were taken from Golder & Associates report titled ''Investigation Géotechnique et Hydrogéologique Détaillée et Conception d'un Parc à Résidus'' dated July 2008.

Historical metal prices and exchange rates were obtained from the London Metals Exchange, Kitco Metals, and the Bank of Canada web sites.

2.4 Units and Currency

All prices and costs are expressed in Canadian dollars (CA$ or $) unless otherwise specified. American dollars are expressed using the acronym US$. Quantities are generally stated in the Système International d'Unités (SI) metric units, the standard Canadian and international practice, metric tonnes (tonnes, t) for weight, and kilometre (km) and metres (m) for distance. In presenting numbers in digit form, thousands are separated by a comma (,) and decimal are separated by a period (.).

2.5 Abbreviations

°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
CA$	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)

3.0 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QP) who prepared this report relied on information provided by experts who are not QPs. The QP's who authored the sections in this report believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

3.1 Florent Baril

Florent Baril relied upon the work of M. Roger Jolicoeur, a consultant with significant experience in mining process projects. M. Jolicoeur performed all equipment and construction costs evaluation as well as the operating costs for the process plant described in section 21.

3.2 Paul Bonneville

Paul Bonneville relied on the work of M. R.Hinse, ing., of Abcourt for

- information regarding the smelting terms to use in estimating revenue,
- an update to the land tenure and property limits and environmental liabilities,
- and information contained in sections 16 and 20.

Paul Bonneville relied on reports from Golder & Associates regarding the design of the tailings pond.

Paul Bonneville assumed the engineering work by Genivar in 2007 was prepared in a professional manner.

4.0 PROPERTY DESCRIPTION AND LOCATION

The text of this section was taken from the report titled NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, dated February 28, 2014. The claims status and the environmental liabilities, the figure 4.4 showing the Abcourt-Barvue, Jonpol, and Vendome property outlines, and figure 4.2 showing the Barvue restored tailings site were updated with information provided by Abcourt.

4.1 Location

The Abcourt-Barvue project (Figure 4.3) is located in the Abitibi area, northwestern Québec, Canada. The property is 60 km north from the town of Val-d'Or and 11 km northwest from the little municipality of Barraute (pop: 1 980). It is accessible by highway 397 joining Val-d'Or to Barraute and highway 386 from Amos (Figure 4.4). The old Pershcourt vertical shaft, located in the central part of the orebody, lies within NTS sheet 32C/12, UTM coordinates Nad83, Zone 18: 301821 E, 5378277 N.

4.2 Claim Status

The current status of the mining titles was verified using GESTIM, from the Ministère des Ressources Naturelles et de la Faune du Québec (the "MRNF"). GESTIM, the latter is a claim management system accessible at the following internet address; https://gestim.mines.gouv.qc.ca. The ABCOURT database is under ID number "1722" on GESTIM's inquiry system.

The Abcourt-Barvue property consists of two mining concessions (MC), two surface leases, and 103 mining cells (CL, CDC). According to GESTIM's website the mineral holdings cover an area of 5,123.4 hectares in the Barraute and Landrienne townships including 141.6 hectares for CM 390 and 393. Finally, ABCOURT is also 100% owner of the surface rights of parts of Lots 24 to 34 in Range VII for approximately 250 ha. In addition, the property has 1.63 M$ of assessment work already filed at the MRNF, which is enough to renew most of the claims for a minimum period of six years. Since December 2013, under the rules of the new mining law (Bill 70), excess work amounts may be transferred to adjoining claims for a maximum period of 12 years.

All claims are 100% owned by ABCOURT and they are in good standing. The details about each of the claims are summarized in Appendix 1.

The Abcourt-Barvue property is subject to royalty payments as summarized in Table 4.1. Claim groups obtained from Messrs. Jean-Guy Barrette and Jack Stoch, which comprise the north half of lots 27 to 32 (Range VIII) were subject to a royalty of 1% NSR. Abcourt purchased this Royalty back from them. Terratech Resources inc., ("Terratech") had the right to receive a standard royalty of $0.25/st on ore extracted from the original Barvue property. The royalty was payable on MC 390, lot 27 in range VII, the south half of lot 32 in range VII, and the north half of lots 33 and 34 in range VI. Terratech has since dissolved and the royalty no longer applies.

Table 4.1 : Detailed royalties by claim title. The "Galahad" claim titles is for the 23 claims acquired by ABCOURT from Galahad Metals Inc. in September 2013

CLAIM TITLE	AREA	OWNERSHIP	ROYALTIES
CL 0434904	40,00	100% Abcourt	None
CL 4085774	40,00	100% Abcourt	None
CL 4085831	40,00	100% Abcourt	None
CM 390	80,94	100% Abcourt	None
CM 393	60,70	100% Abcourt	None
Galahad	820,00	100% Abcourt	1.5% NSR (Teck Resources)

In September 2013, the Company reached an agreement with Teck Resources Limited (“Teck”) under which Teck agreed to extinguish their pre-existing right of first refusal and a back-in right (the “Rights”) with respect to the Galahad Property. In consideration of the extinguishment of the Rights, Teck was granted a 1.5% net smelter royalty on the 23-claim property. Osisko Royalty has since acquired the 1.5% NSR on the Galahad Property from Teck Resources.

The two mining concessions on the Abcourt-Barvue property are not subject to royalties.

The Abcourt-Barvue property was legally surveyed prior to the issue of the mining concessions (CM), as required by the Québec’s Mining Act. The legal survey is filed at the Elder mine site in Rouyn-Noranda.

4.3 Environmental Liabilities

From 1952 to 1957, the extraction of silver-zinc mineralization from the Barvue mine created a 32-hectare tailings pond with acid draining lixiviates. For this reason, the mining concession related to the old Barvue tailings storage area was returned to the Crown and is not included in the Abcourt-Barvue property. There is no liability related to this tailings storage area for the Company. The old Barvue pit waste pile area is on an Abcourt-Barvue mining concession. A letter from the MERN releases Abcourt from any responsibility.

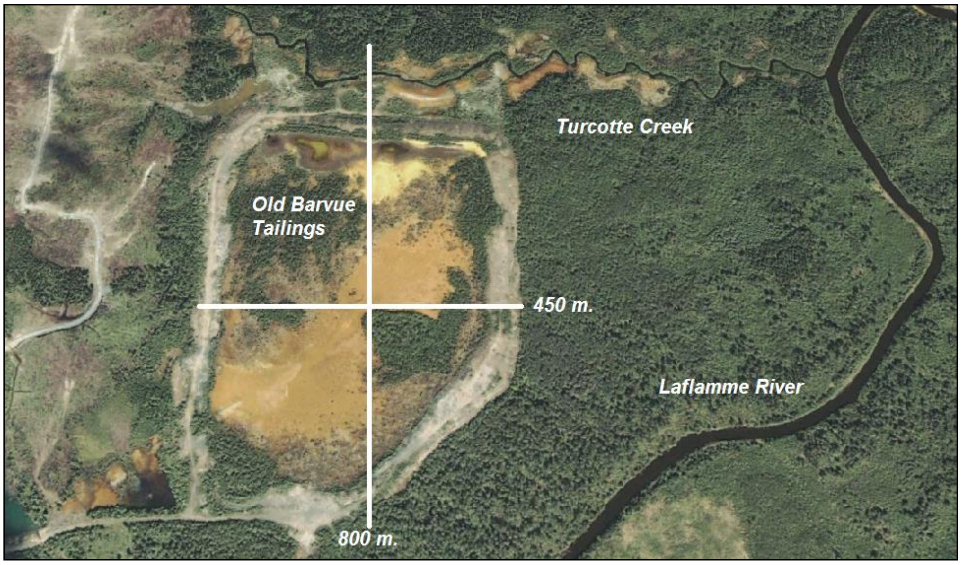

Figure 4.1: Aerial view of the Barvue tailings in 2003 (Source: Google Earth historic imagery)

Figure 4.2 : Restored Area along the Turcotte Creek on the Barvue's tailing site (source: MERN Web Site, 2018)

The MRNF restored the old Barvue tailings area at a cost of 40 million dollars. This cost also covered the transportation of 13,000 tonnes of mineralized rock from the Vendome property to the Barvue site.

A settling pond formerly used by ABCOURT was restored by the Company, but as a small amount of lixiviate seeps from the bottom, ABCOURT is currently collecting this seepage and processing it with a portable treatment plant installed in a 20-ft container. Water samples are sent to an assay lab on a regular basis.

To date, the sites selected for the new tailings and the location of the processing plant were approved by the MRNF. Moreover, the Company has received from the MDDEFP a certificate of authorization for the dewatering of the mine, the construction of a water treatment plant and the development of a settling basin. Finally, an application for a mining lease covering part of the mining claims CL 337382 and 434911 will be required to mine the Abcourt West pit.

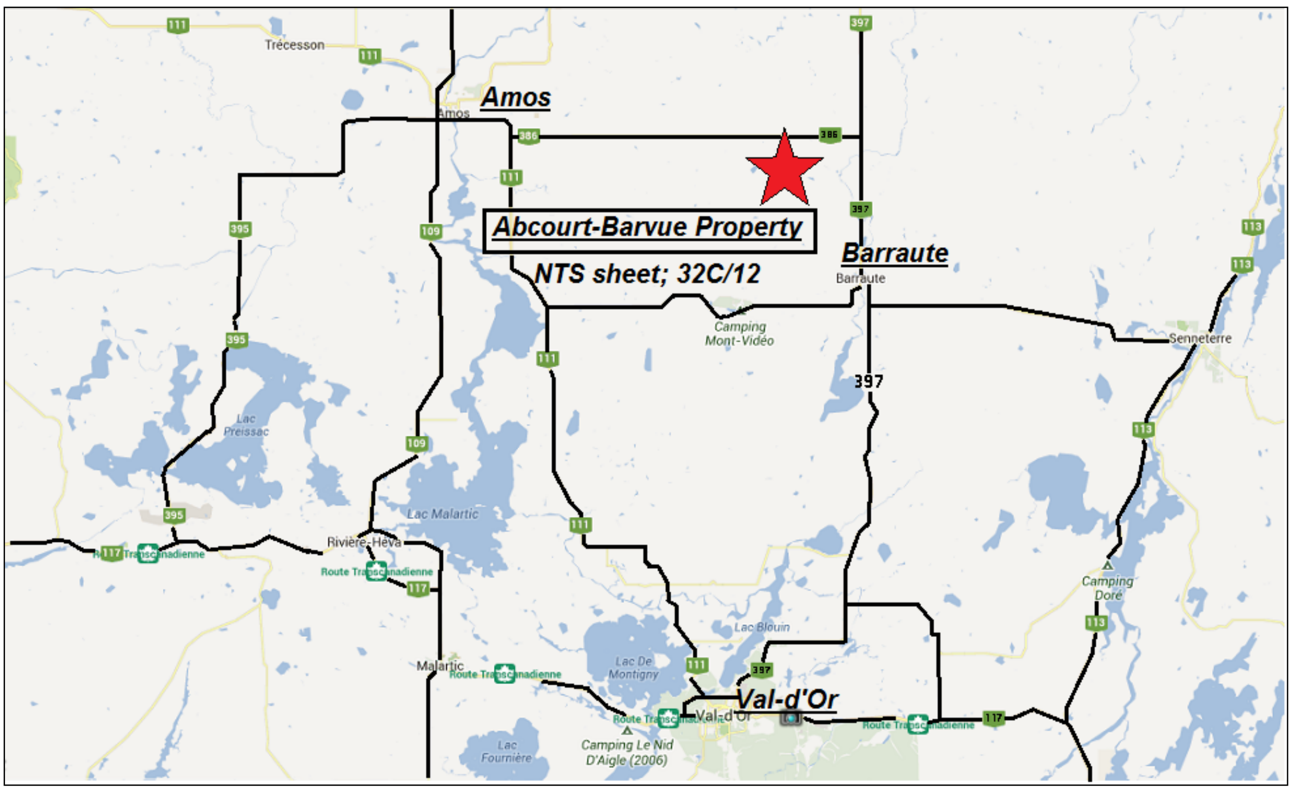

Figure 4.3 : Regional location map (from Google Earth basic map)

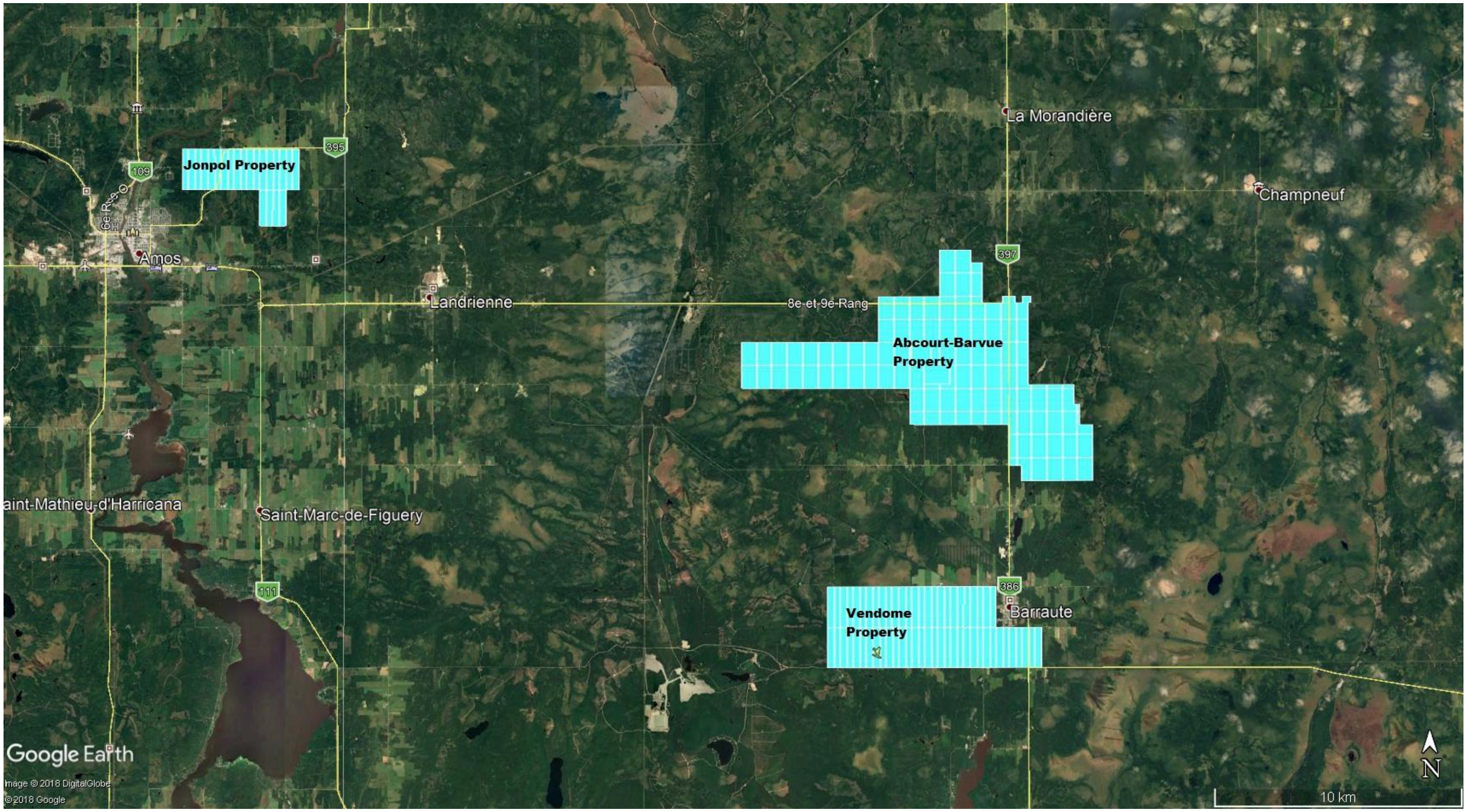

Figure 4.4: Satellite view of both the Abcourt-Barvue, Jonpol, and Vendome property outlines (source: Sigeom and Google Earth)

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014. The list of equipment in table 5.1 was updated with information provided by Abcourt.

The property is located 60 km north of the mining community of Val-d'Or and 5.5 km from Barraute (Province of Quebec). It is easily reached all year round via highway 397 and by range road 6-7 to the mine site.

The area is relatively flat and lies within the great "Clay Belt" of northern Ontario and Québec. 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 315 metres above sea level with a maximum of 472 metres (Mont Video).

This region is characterized by a continental climate with winter temperature lows in the -10°C to -35°C range with a total snow cover of 3.17 m and summer temperature highs in the range of 10°C to 23°C with 95 mm of rain per month. Access to water is available near the east end of the property at the Laflamme River which runs north to the Bell River.

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 and material (Table 5.1). It also includes milling and mining equipment such as Jumbo drills and Scoops (see Table 5.1).

In 2008, the Company bought used mill equipment from Osisko Exploration Ltd. and Legault Metal Inc. (Company's Press releases of Feb. 01, Feb. 07 and April 28). Most of this equipment came from the former 1 800 t/d East Malartic mill plant. A list of the mill equipment, surface infrastructures and rolling mine units is presented in the following table.

Material and human resources needed for exploration and mining activities are available in the town of Val-d'Or and the neighbouring area.

Table 5.1 : List of the equipment and infrastructures located at the mine site in 2013

MILLING EQUIPMENT	INFRASTRUCTURES
16 ft. Ø SAG Mill (1)	22 x 65 m. Main Office & Garage
10 ft. Ø 600 HP Ball Mill (2)	18 x 22 m. Garage
4 x 5 ft. 250 HP Crusher (1)	Ventilation Shaft
2.5 ft. Knelson Concentrators	50 x 130 m. Concrete Mill Basement
Cyclones, Pumps, Conveyors,	**ROLLING EQUIPMENT**
Silos and Mixing Tanks	3-boom Jumbo Drills (3)
Overhead Crane	5-cubic yard Scoop (1)
Lime and Reagents Circuits	4-cubic yard Scoops (2)
Electric Circuits and Appliances	8-cubic yard Scoops (4)
Spare Parts	Compressors 1700 cfm (2)
Water Treatment (Lime) Circuit	Ventilation fans

Figure 5.1 : From left to right; External view of the main building, 4 yard Scoop parked in the garage, a ball mill shell being delivered to Abcourt-Barvue, and, in the background, concrete basement of the old mill.

6.0 HISTORY

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014. The photos of the Barvue mine in 1953 were removed. The property limits shown on figure 6.1 were updated by Abcourt.

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). 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 mineral resources of 15 400 000 t grading 3.26% Zn and 39.0 g/t Ag (Vachon, 1994). 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 tonnage 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 MRNF, over 1,325 diamond drill holes have been drilled in the Barraute Township by past owners and ABCOURT on the Abcourt-Barvue property since then. 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 price.

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 price. 1968-69: Pershcourt and Frebert Mines Limited 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 Corporation, 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 Exploration) 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 series) 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).

The resources 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 estimated 204 ounces of gold was recovered from the silver concentrate.

Figure 6.1 : Sigeom's generated compilation map showing the distribution of surface holes (green dots) drilled on the property. Note the lack of exploration work in the western part of the land holding

1990-1993: NOREX optioned the Abcourt-Barvue property 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 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).

1997-2008: ABCOURT remained active on its property and carried out 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 (AB99-01 and AB99-02),
- 2003: Ten (10) holes totalling 530 metres (AB03-01 to AB03-10 series),
- 2004: Twenty-four (24) holes totalling 1 169 metres (AB04-01 to AB04-24 series),
- 2005: Forty-six (46) holes totalling 5 879 metres (AB05-01 to AB05-46 series),
- 2007: Ten (10) holes totalling 2 080 metres (AB07-01 to 10 series),
- 2008: One (1) 179 metres hole (AB08-01) on Lot 26, Range 7.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014. The property limits shown on figure 7.1 were updated by Abcourt.

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 area of 85 000 km2, 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.

7.1 Regional-Scale Geological Setting

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 Mya and the Southern Volcanic Zone is interpreted as a younger arc segments, 2705-2698 Mya (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 7.1). 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.

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 7.3). 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 thickness of the Héva Formation varies from 3 to 5 km for over more than 100 km on an EW to ESE-WNW regional trend. This formation, poorly exposed on surface, 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 outline 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),

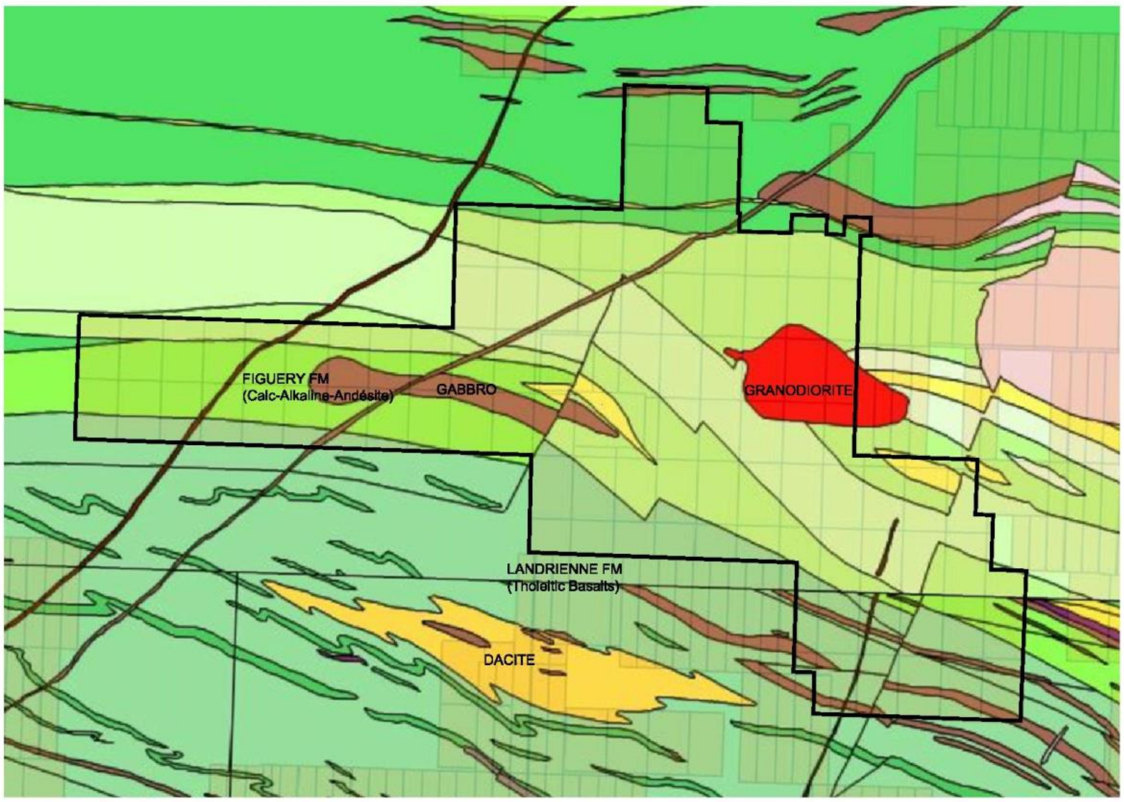

Figure 7.1: Regional geology of the property (modified from Sigeom

(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 breccia. It also includes units of ash tuffs, lapilli tuffs and block 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.

7.2 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 and related base metal occurrences are located in the Figuery 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. Chlorite is usually restricted to intermediate to mafic volcanics rocks (Vachon, 1994). 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 intensely carbonatized and sericitized. This extensive regional shear zone carries high above background zinc values in a zone up to 175 metres in width extending along the strike of the zone of shearing far beyond the limits of ore mineralization (Cornwall, 1956). U-Pb age dating of a dacite tuff intersected by borehole AB05-35 indicates that it has the same age (2706 ±3.3 Mya) than lithologies topping the Malartic Group in the Val-d'Or area (RP 2007-01).

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). The mineral 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 southwest sides of a granodioritic pluton, with some gravimetric anomalies and several electromagnetic (EM) anomalies indicate a potential for massive sulphides 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.

7.2.1 Geological Units of the Abcourt-Barvue Deposit

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.

Figure 7.2: Close-up views of typical host rocks (from left to right); finely bedded "Marker" tuff horizon and felsic coarse fragmental volcanoclastic rocks, mineralized felsic volcanoclastic rocks (agglomerate), finely bedded "Marker" tuff horizon

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.

Gélinas et al. (1984) subdivided the volcanites of the Blake River Group into stratigraphic units from tholeiitic to calc-alkaline. These are, from the base of the volcanic pile to the top; Tholeiitic unit of Rouyn-Noranda, Transitional unit of Duprat-Montbray, Tholeiitic unit of Pelletier, Calc-alkaline unit of Dufault, Tholeiitic unit of Trémoy, Calc-alkaline unit of Cléricy, Tholeiitic unit of Destor, Calc-alkaline unit of Reneault and, Tholeiitic unit of Dufresnoy.

The tholeiitic units are dominated by mafic lavas (basalts and andesites) with minor rhyolites. These units are composed of magnesium-rich basalts poorly developed at the bottom of the pile (Pelletier Unit) that grade to iron and titanium-rich andesites at the top (Dufresnoy Unit). This enrichment in iron is accompanied by an increase in K, Ba, Rb, Sr, Zr, Y, Nb and REE (Gélinas et al., 1984).

The calc-alkaline units are characterized by bi-modal volcanism in which the andesites and the rhyolites are very abundant. Most of the rhyolites are explosive in nature and are composed of fragments of variable sizes (volcaniclastites). The calc-alkaline andesites can be differentiated from the tholeiitic andesites by their higher Rb/Sr, Zr/Y, La/Sm and La/Yb ratios (Gélinas et al., 1984).

7.2.2 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. 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.

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

- iron-poor, honey brown and dark colour sphalerite (ZnFeS2), 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 (CuFeS2), native silver (Ag) and proustite (Ag3AsS3).

The thickness of the mineralized horizon tends to increase eastward, most of the sulphides being located in the Barvue area. Generally speaking, 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).

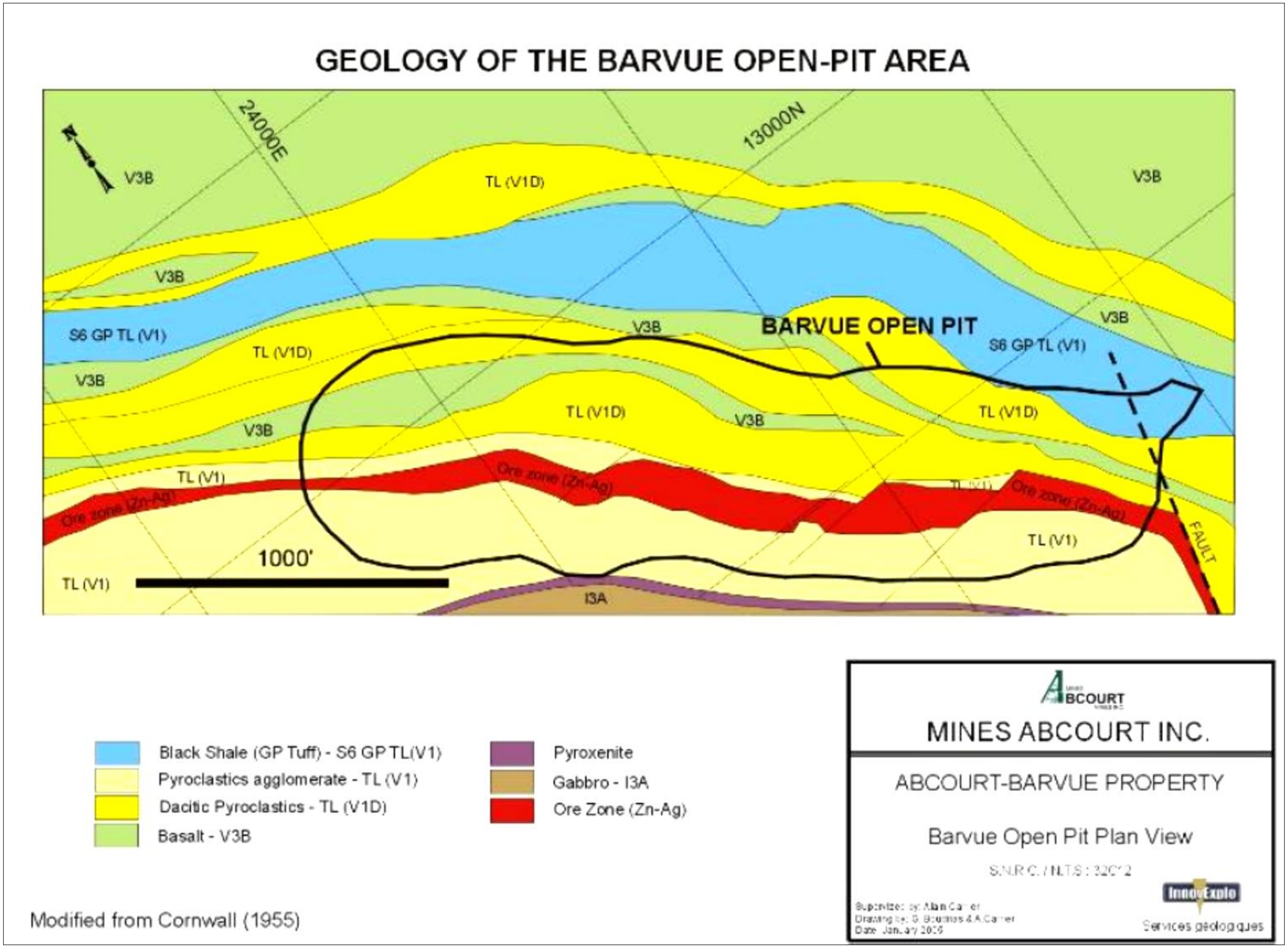

Figure 7.3: Geological plan view of the old Barvue open pit area. (Source: Innovexplo's Technical Report, March 2005)

8.0 DEPOSIT TYPE

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014.

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. Québec 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 Mya), to deep seated porphyry-style and syenite-associated disseminated gold systems (2682-2672 Mya), and deeper settings for orogenic lode gold deposits (< 2670 Mya) 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 zinc and silver 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).

9.0 EXPLORATION

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014.

There is no relevant exploration work other than drilling to report.

10.0 DRILLING

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014.

ABCOURT carried out a 72-hole diamond drilling program totalling 18,103 metres from October 2010 to January 2012. The holes were drilled at shallow depth (125 to 300 vertical metres from surface) along the west portion of the Abcourt-Barvue deposit. This area was conveniently called the "Abcourt Area" by the Author in this report. The main objective of the Company was to carry-out in-fill drilling around silver-zinc intersections already known from previous diamond drilling programs.

The set-up and drilling pattern of each hole was laid-out by Mr. Renaud Hinse, Mining Eng. and President of ABCOURT. Mr. Eugène Gauthier, Ing., geological consultant, supervised the diamond drilling program until October 2011 when he left the Company. He logged and sampled 71% of the core brought by the drillers to the core logging facilities installed at the mine site. Mr. Gauthier is a senior geologist with 30 years of experience in exploration and production for gold. Miss Adriana Jerez, a junior geologist, was fully involved in the final phase of the program from September 2011 until January 2012.

The BQ size core was recovered by Forage Spektra Inc (formerly Forage Mercier) of Val-d'Or, and each hole was spotted in the field with a Geneq SXBlue Mapper unit having a precision of 1 metre. The "in-the-hole" surveys were performed with a rented Flex-it instrument capable of measuring the azimuth, dip and magnetic field (in nanotesla) for each 30-metre depth intervals. As the area is swampy and poorly drained, all the holes were plugged and cemented to avoid excessive water infiltrations from surface. When possible, casing rods were removed and replaced by a 3 metre long "PVC" plastic tube identified with an aluminum tag wearing the borehole ID. The mineralized intervals were kept in their original wood boxes and stored in a garage at the Abcourt-Barvue's mine site.

The proportion of sampled core versus un-sampled core is varying from 6 to 32% for an average of 19%. The core recovery is generally very good. Based on the Author's observations, the sample quality is good and the samples are generally representative of the mineralized intervals.

A longitudinal view and representative examples of drill sections through the mineral deposit are presented in Appendix 2.

Table 10.1: Borehole parameters from the 2010-2011 diamond drilling program

Hole ID	UTM Coordinates (NAD 83)			Az	Dip	Depth	Hole ID	UTM Coordinates (NAD 83)			Az	Dip	Depth
	East	North	Elev.	(°)	(°)	(m)		East	North	Elev.	(°)	(°)	(m)
AB 10-01	301611	5378391	317	180	-67	195	AB 11-37	301373	5378483	317	174	-67	279
AB 10-02	301580	5378395	317	180	-68	228	AB 11-38	301372	5378457	317	175	-67	240
AB 10-03	301582	5378410	317	180	-68	195	AB 11-39	301639	5378393	318	175	-67	219
AB 10-04	301554	5378415	317	180	-67	213	AB 11-40	301640	5378413	318	175	-67	249
AB 10-05	301524	5378444	317	180	-67	243	AB 11-41	301671	5378408	316	175	-67	249
AB 10-06	301522	5378414	317	180	-67	207	AB 11-42	301685	5378397	316	175	-67	223
AB 10-07	301197	5378565	318	180	-67	249	AB 11-43	301702	5378438	317	175	-67	276
AB 10-08	301194	5378508	317	180	-67	171	AB 11-44	301701	5378412	315	175	-67	249
AB 10-09	301138	5378555	320	177	-67	246	AB 11-45	301715	5378386	315	175	-67	228
AB 10-10	301134	5378523	320	177	-67	204,8	AB 11-46	301732	5378433	316	175	-67	270
AB 10-11	301077	5378607	316	176	-67	279	AB 11-47	301731	5378404	317	176	-67	234
AB 10-12	301076	5378571	317	176	-67	246	AB 11-48	301342	5378459	318	175	-67	210
AB 10-13	301075	5378541	316	177	-67	192	AB 11-49	301327	5378435	317	176	-67	180
AB 10-14	301046	5378605	316	176	-67	279	AB 11-50	301315	5378510	318	175	-67	240
AB 10-15	301045	5378564	315	177	-67	225	AB 11-51	301284	5378527	316	175	-67	264
AB 10-16	301015	5378575	315	176	-67	261	AB 11-52	301282	5378495	316	175	-67	213
AB 10-17	301014	5378545	315	177	-67	201	AB 11-53	301283	5378471	318	176	-67	192
AB 10-18	300987	5378590	314	175	-67	270	AB 11-54	301255	5378550	318	175	-67	269
AB 10-19	300986	5378554	315	176	-67	223,3	AB 11-55	301255	5378516	318	175	-67	240
AB 10-20	300956	5378597	315	175	-67	267	AB 11-56	301253	5378455	319	177	-67	165
AB 10-21	300954	5378552	316	176	-67	204	AB 11-57	301227	5378578	316	174	-67	300
AB 11-22	300927	5378596	314	175	-67	264	AB 11-58	301226	5378525	317	175	-67	235
AB 11-23	300926	5378571	314	175	-67	210	AB 11-59	301224	5378489	318	177	-67	201
AB 11-24	300925	5378550	316	177	-67	177	AB 11-60	300871	5378681	318	174	-67	336
AB 11-25	300898	5378596	319	170	-67	240	AB 11-61	300868	5378627	319	176	-67	267
AB 11-26	300897	5378571	318	175	-67	195	AB 11-62	300867	5378596	322	177	-70	231
AB 11-27	300897	5378549	317	172	-67	171	AB 11-63	300866	5378706	317	172	-78	404
AB 11-28	301433	5378453	317	175	-67	270	AB 11-64	300633	5378760	319	172	-74	78
AB 11-29	301461	5378455	317	174	-67	291	AB 11-64A	300633	5378760	319	172	-63	417
AB 11-30	301461	5378433	317	175	-67	261	AB 11-65	300605	5378657	320	175	-67	399
AB 11-31	301461	5378408	318	175	-67	239	AB 11-66	300607	5378632	321	177	-60	273
AB 11-32	301492	5378452	318	174	-67	285	AB 11-67	300928	5378627	318	176	-70	306
AB 11-33	301489	5378410	318	175	-67	222	AB 11-68	300930	5378672	322	176	-72	390
AB 11-34	301403	5378480	317	174	-67	294	AB 11-69	300989	5378657	318	176	-69	402
AB 11-35	301403	5378455	316	175	-67	267	AB 11-70	301048	5378655	316	176	-72	381
AB 11-36	301402	5378427	316	175	-67	238	AB 11-71	301108	5378623	316	175	-72	371

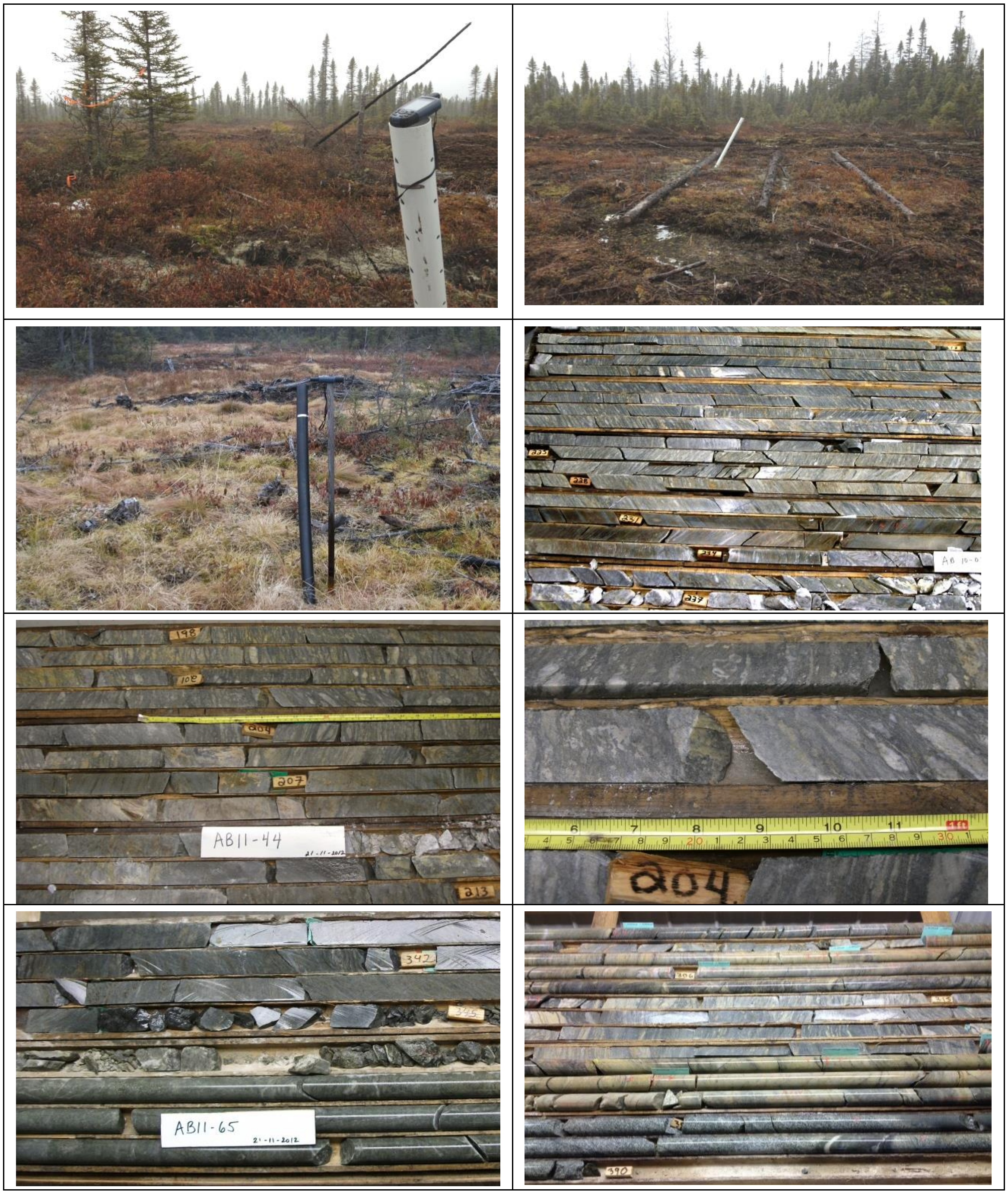

Figure 10.1: From upper left to bottom right; PVC casing of hole AB11-65 beside casing of historic hole FS-06 (red flag to the left), drill set-up of hole AB11-64, drill set-up of hole AB11-46, mineralized interval in hole AB10-07, mineralized interval in hole AB11-44, close-up of lapilli tuff (agglomerate?) in hole AB11-44, mineralized zone in hole AB11-65 followed by a unit showing a poor RQD and, mineralized interval of hole AB11-68 showing complementary samples taken by the Author in May 2013.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014.

11.1 Sampling Method

The core logging and sampling was done by Mr. Eugène Gauthier and Miss Adrianna Jerez at the Abcourt-Barvue mine site.

Sampling intervals are determined by the geologist depending on the nature of alteration and the presence of sulphide mineralization. Zinc-bearing horizons are generally circumscribed to silicified and sericitized zones having 2 to 20 metres in width along the core.

The sampling method is straightforward. After logging, the sections to be assayed are identified in the core box. The core is then split at the mine site using a diamond core saw, bagged, tagged and stored before being collected by the lab's employee and shipped directly to Ste-Germaine-Boulé. The other half of the core is kept into the wooden box for further analysis or as reference. As a general procedure, ABCOURT is only keeping the core sections that are split along with the first and the last box of each hole. The remaining (full) core is kept a little time after the reception of the assay results and thrown out in the back yard. As the core is not photographed, there is no record of the geological units intersected before or between mineralized zones other than the core description itself.

A total of 2 669 samples were assayed for silver (in ppb) and zinc (in ppm) by Techni-Lab S.G.B. Abitibi Inc. ("Techni-Lab") a wholly-owned member of the Activation Laboratories Ltd. ("Act-Lab" or "ACT") Group in Ste-Germaine-Boulé.

All samples received at Techni-Lab are dried and weighed prior to being processed. Sample material is crushed in a jaw and/or roll crusher (70% passing 9 meshes). Ground material is split with a rifle splitter to obtain a 250 gram sub-sample. The sub-samples are pulverized in a "flying disk" or a "ring and puck" type grinding mill to give a pulp (85% passing 200 meshes). The remaining of the crushed sample (reject) is returned into the original plastic bag.

The silver and zinc content of each sample were determined by inductively coupled plasma atomic emission spectroscopy (ICP-AES) under code number TMT-G5F. Assay results grading over 1 000 g/t were re-analyzed by the gravimetric method. Assay results are sent electronically to R. Hinse, president of ABCOURT, and the geologist in charge of the diamond drilling program. Assay certificates are sent by post mail at ABCOURT's main office in Rouyn-Noranda (Évain).

11.2 Quality Control Program

The conformity of Techni-Lab to the standards of its own industry was verified by the Standards Council of Canada under CAN-P-1579 "Guidelines for the Accreditation of Mineral Analysis Testing Laboratories" and CAN-P-4E "General Requirements for the Competence of Testing and Calibration Laboratories (Adoption of ISO/IEC 17025)" specifications.

ABCOURT shipped all its core samples to Techni-Lab. This lab follows QA/QC procedures which consist in the introduction of mineralized standards, trace elements (blank) and field duplicates for each batch of twenty-four (24) samples. Each set comprises one blank in the first third, one duplicate in the second third and three standards 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 any contamination during the sample preparation. The pulp duplicate is used to determine the reproducibility of the assay results and the standard is used to monitor the precision of the equipment.

Techni-Lab is currently using three (3) certified standards and one (1) certified very low grade "blank" for each batch of samples sent by ABCOURT to their laboratory:

1. GBM908-5, 10 and 14 which are certified reference materials prepared and sold by Geostats Pty Ltd in Australia,
2. SU-1b is a "blank" (6.36 ppb Ag and 235 ppm Zn) certified reference material issued in 2009 by CANMET.

The Author didn't observed any significant analytical bias in the standard samples assayed by Techni-Lab.

ABCOURT did not put in place a quality program to secure the validity of its assay results during the diamond drilling program. The reason invoked by ABCOURT is; "…as the lab can identify the standards (a standard is a pulp sample, not a core sample), this becomes a futile operation (R. Hinse)." The Company prefers to check the assay results after the completion of the diamond drill program in sending the pulp samples of significant intersections to two other laboratories. As both the pulps and the rejects are stored at the mine site, this operation can be carried out a long time after the reception of the original assay results.

12.0 DATA VERIFICATION

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014.

During this NI 43-101 technical report, the Author reviewed all data available which was collected and placed at its disposal by Mr. Renaud Hinse, President of the Company. The Author visited the Abcourt-Barvue property in September 12-13 of 2012, in November 21-22 of 2012 and in January 10, April 11 and May 14 of 2013. As the technical data of the property is located at the Elder mine, the Author had to work several days in Rouyn-Noranda. These site visits included;

- Selected core review of mineralized and un-mineralized intervals of the Abcourt Area including a survey of some drill set-up, complementary sampling and lithological observations,
- Review of the internal reports, sets of old diamond drill sections, plan views of the geology and sampling data from previous operations,
- Selection of pulp samples from the original assay lab to be compared with two other independent labs for accuracy and repetitivity,
- Validation of the drill hole database; during this step, the geological data related to the boreholes is checked and compared to its representation on sections and longitudinal plans,
- Definition of the size of mineralized blocks following the precepts of the CIM Definition Standards of 2010. The mineralized outline was taken from a section to section geological interpretation carried out by ABCOURT which is based on past production and field experience,
- Categories of mineral resources allocation based on the amount of information provided by the old stopes, composite muck or chip samples, surface diamond drilling and underground definition drilling intersections.

12.1 <u>Visit of Drill Set-ups</u>

Each of the holes drilled on the Abcourt-Barvue property was surveyed in-house by Rachel Charette, surveyor and draft person, with a GENEC Recon (Pocket PC) GPS having a precision of ± 1 metre. The UTM Nad 87 system of coordinates was used and the data was transferred in mines coordinates from permanent field stations already surveyed by Corriveau et Associés Inc in October 2010. In November 2012, the Author has visited 30% of the set-up drilled on the west side of the Abcourt-Barvue deposit. With the use of his handheld Garmin GPSmap 60cx unit, the Author has observed that;

- Most of the drill set-up had a plastic PVC pipe in place of the regular casing used to penetrate the overburden. Measurements are indicating that they were almost all drilled to the south at steep angles (60-70°),

- An aluminum tag was clamped to each of the PVC pipes with the borehole identification. On my way to the drill set-up of borehole AB11-44, the tag was indicating the one of the 43. That same anomaly was observed for holes AB11-41, 42 and 43.

12.2 Deviation of the Holes in Azimuth and Dip

Moreover, 30 metre-spaced "Flex-it" azimuth, dip and magnetic measurements were taken by the drilling company's operators for all the holes. These measurements are indicating that the holes are generally dipping from 0.6 to 1.1° for a lateral deviation of 1.3 to 2.0° for every 30 metre in a 250-metre deep hole. These deviations are acceptable considering the relatively shallow depth of the holes. The Author considers that, if exploration below 300 vertical metres is justified, the use of NQ core size would help maintain straighter borehole deviations.

12.3 Core Observations

At the time of the visits it was possible to see the core drilled by ABCOURT in 2010 and 2011. The Author has made a comparison between the unit's description (log) and what he was seeing in the core. The descriptions are generally concise making possible the correlation of the main units from section to section. However, it was observed that the quality of work was inconsistent for the last logs. The mine term "Gray Schist" or "GS" was not used by the geologists despite they clearly identified a high sericite and, to a lesser extent, chlorite alteration zone where the GS was expected to be intersected. On the other hand, there are few mentions of the "Marqueur" horizon (a highly silicified unit marking the footwall of the main Ag-Zn mineralization). Even if the faults and main schistosity are well described, the logs are suffering from a lack of information on the core recovery and rock quality designation (RQD) measurements for almost all the holes. From what I observed myself, the RQD was good (75-90%) in the mineralized zones and in the average (50-75%) in the Gray Schist.

After having done a random examination of the diamond drill logs, it was found that the core recovery was excellent (95%). However, it was also observed that 10 to 15% of the mineralized zone was lost in borehole AB11-65 (Figure 12.1). As this particular area is highly fractured, a better core recovery would have been possible with a NQ size core diameter. Nevertheless, the Author believes that the majority of the samples taken are generally representative of the mineralized intervals.

Figure 12.1: Mineralization of borehole AB11-65 locally showing a poor RQD and 10-15% of lost core

12.4 Visit at the Assay Laboratory's Installations

The Author has visited the Techni-Lab (Act-Lab) installations at Ste-Germaine-Boulé on February 8, 2013. The place was considered clean and well maintained despite its exiguity. The lab can process up to 15 000 samples in a month. The assay sheets are manually filled, scanned and stored in a safe place located outside the lab for a minimum period of five years. The visit was led by Mr. André Caouette, a competent person having more than 30 years of experience in the mineral industry. A Bachelor in chemistry was also present during my visit. As a professional, his main responsibility is to maintain the quality of the assay results for Techni-Lab which is a subsidiary of Activation Laboratories Ltd near Hamilton, Ontario.

12.5 Presentation of the Assay Results

The Author has used the assay sheets and assay certificates delivered by Techni-Lab to validate most of the assay results appearing in the log reports. They consist in Excel spreadsheets containing sample numbers and assay results. The laboratory uses a relatively standard file format. Assay certificates contain results of standards (3), duplicates and blanks.

(a) (b)

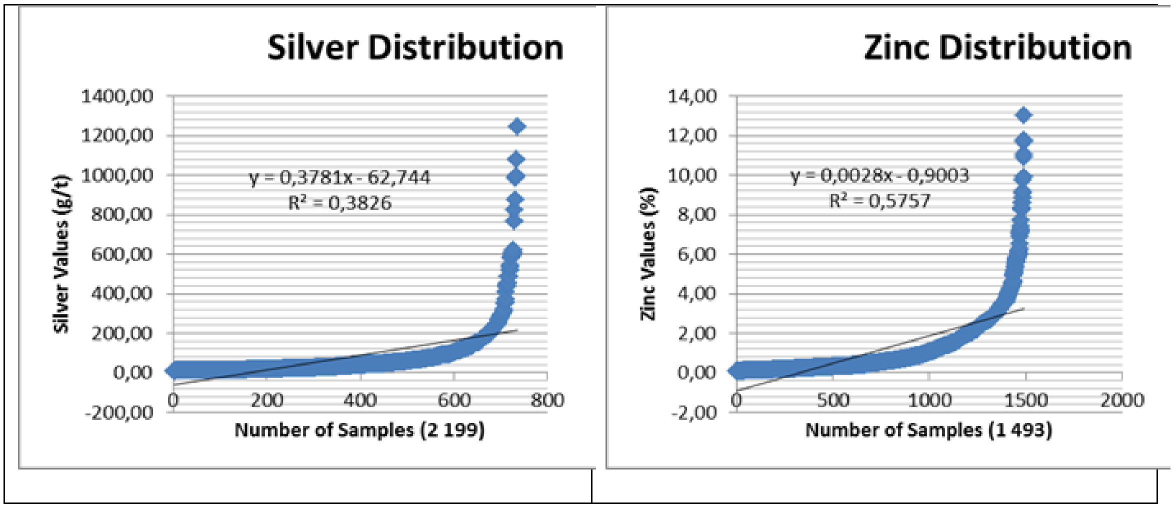

Figure 12.2: (a) Distribution of silver for 736 assay results grading more than 10 g/t Ag; (b) Distribution of zinc for 1 492 samples grading more than 0.10% Zn

Most of the silver values (97%) are grading under 150 g/t. The zinc values are generally grading about 3% Zn. Figure 12.2(a) shows the distribution of silver values (x=2 199) from 10 to 1 244 g/t Ag for an average of 27,49 g/t. Figure 12.2(b) shows the distribution of zinc values (x=1 493) from 0.10 to 13% Zn for an average of 1.17% Zn. The average length of the 2 669 samples taken is 1.29 m.

12.6 <u>Quality Control on Selected Samples</u>

In order to validate the quality of the assay results provided by Techni-Lab, the Author has selected 54 pulp samples from holes AB10-07 (no. 3317 to 3336), AB11-34 (no. 85407 to 85414, 85417 to 85421), AB11-44 (no. 85807 to 85814), AB11-65 (no. 18807 to 18812), AB11-67 (no. 18873 to 76, 18878 and 18879). From these 54 samples, 21 were grading less than 2% ZnEq (ZnEq; the combined values of silver and zinc), 27 were grading between 2 and 10% ZnEq, and 6 were grading over 10% ZnEq. The pulp samples were re-assayed by ALS-Chemex in Val-d'Or, Québec and Accurassay in Thunder Bay, Ontario. ALS-Chemex is a Standards Council of Canada certified laboratory (certification n° 689). Accurassay is an ISO 9001:2000 certified laboratory that routinely performs assaying for junior mining companies.

The graphs below show plots of this data (Figure 12.3 and Figure 12.4).

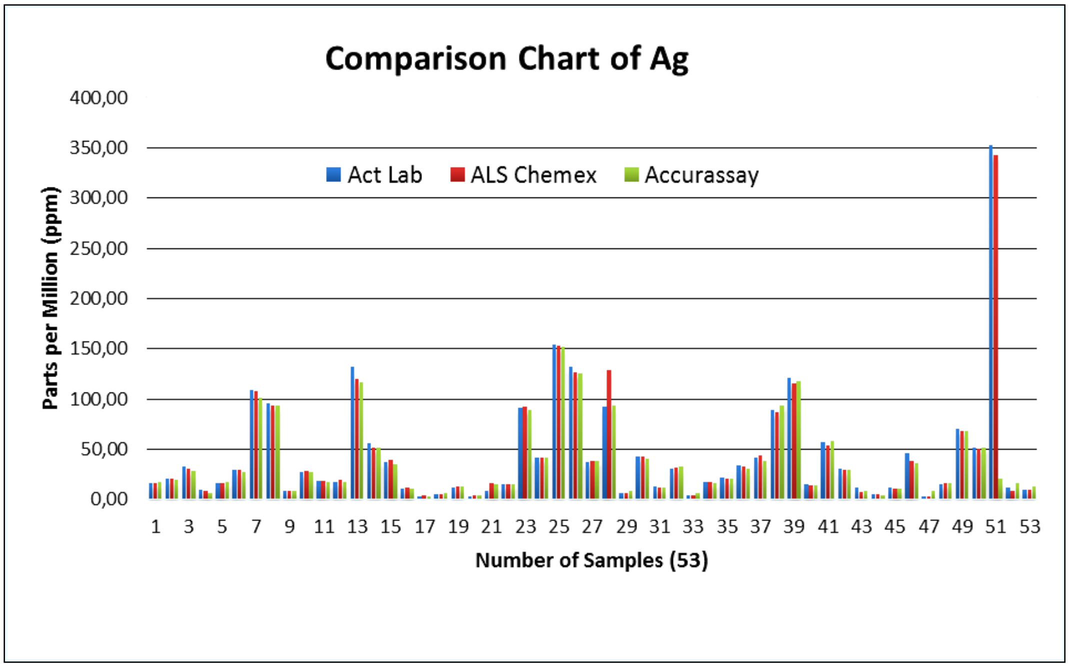

Figure 12.3: Comparison chart of silver values from Techni-Lab (blue), ALS-Chemex (red) and Accurassay (green). Overall, the value are quite similar despite Techni-Lab's slight deviation for higher grades

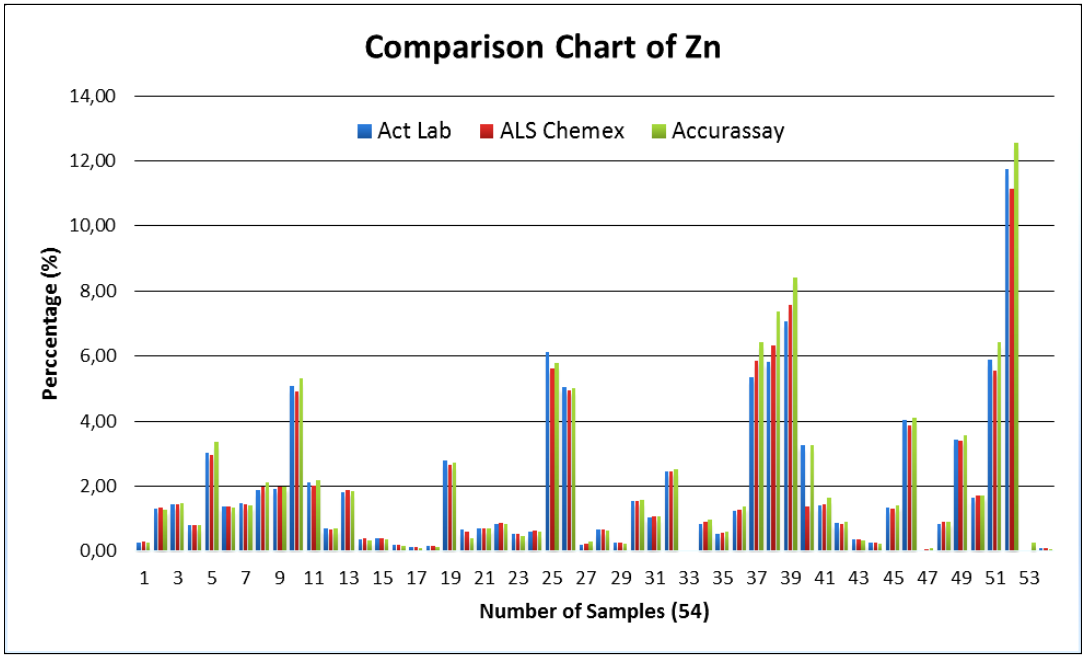

Figure 12.4: Comparison chart of zinc values from Techni-Lab (blue), ALS-Chemex (red) and Accurassay (green). The values are in the same range with a slight deviation towards higher grades for Accurassay

There are seven (7) sampling results where Accurassay was slightly off the other labs for the zinc results but nothing that indicates an out-of-control situation nor an inability to provide accurate data. Nevertheless, Accurassay have completed re-assays on these seven samples deemed to differ enough to warrant such effort. In all instances, the re-assays verify the initial results and, in all but one case, the re-assay results are somewhat closer to the other labs' results. The Author believes that the degree of variation is within the realm expected for assays on large numbers of samples. Indeed, the other labs show deviances on several occasions as well. It is showing the importance to compare results as a whole and not individually.

With the exception of sample no. 19202, which is far over the cut-off grade of 514 g/t Ag used by ABCOURT, Techni-Lab's assay results for both silver and zinc compare well with the two other laboratories. From these 53 samples, a group of 27 samples shows assay results ranging from 2 to 10% ZnEq which represents "ore grade" mineralization (Table 12.1). In this group, the assay results are quite similar for the three laboratories. In fact, the average grades of Techni-Lab for silver and zinc are within a range of 5% from the two other labs.

Moreover, the 21 samples selected in mineralization grading less than 2% ZnEq are showing an average difference of 9% for silver and 2% for zinc. If taken solely, some major differences (+10%) can be observed from lab to lab in assay results. These should be taken more as material differences (scarcity of metal and statistical representativity of a single 30-gram pulp sample) than inaccuracies in the labs results.

The same comments apply for the five samples grading more than 10% ZnEq which were re-assayed by ALS-Chemex and Accurassay. In this case, it is also materially difficult to reproduce the values initially obtained by Techni-Lab because of a higher than the average metal content in the pulp sample. Despite this, the bias between the three labs was quite acceptable; 3% for silver and 10% for zinc.

Sample 19202 is an outlier grading 4 696 g/t Ag (151 oz/t) over 1.30 metre which was intersected in hole AB11-67. A second pulp was prepared by Techni-Lab from the rejects and it was sent to the other labs. The high grade values were confirmed by the two labs for both silver and zinc. This high silver value is quite uncommon as, from the 25 658 samples collected from surface and underground holes, only 193 of them are grading more than 514 g/t Ag. From that population, only two other samples (H-14453 in borehole 85ST-192 and H-17929 in borehole 87ST-59) have better silver metal content by core length.

Table 12.1: List of the 27 samples grading between 2 and 10% Zinc Equivalent (ZnEq) showed in increasing order. The averages of Techni-Lab are somewhere between the two other labs for both silver and zinc.

ORIGINAL	VALUES ACT		VALUES	NEW	VALUES ALS		VALUES ACC	
SAMPLE	Ag	Zn	In ZnEq	SAMPLE	Ag	Zn	Ag	Zn
85413	37,90	0,20	2,08	19165	38,91	0,22	38,06	0,29
3331	37,20	0,39	2,23	19186	39,34	0,41	35,50	0,36
3318	20,70	1,29	2,31	19173	21,38	1,36	19,85	1,28
3325	8,20	1,91	2,32	19180	8,20	1,98	8,70	1,98
18807	31,06	0,86	2,40	19192	29,33	0,83	29,59	0,89
85410	41,30	0,61	2,65	19162	41,59	0,62	41,99	0,60
3322	29,70	1,37	2,84	19177	29,86	1,37	27,39	1,32
85809	34,20	1,24	2,93	19153	33,13	1,26	30,28	1,36
3327	18,50	2,10	3,02	19182	18,53	2,02	17,08	2,20
3319	33,30	1,43	3,08	19174	30,83	1,46	28,46	1,48
3330	56,30	0,36	3,15	19185	51,87	0,39	52,20	0,34
3335	12,30	2,78	3,39	19190	12,77	2,66	12,62	2,74
85418	42,60	1,53	3,64	19168	43,22	1,54	40,75	1,58
3321	16,50	3,01	3,83	19176	16,69	2,95	17,54	3,37
85420	31,20	2,44	3,98	19170	32,27	2,45	33,17	2,50
85813	15,60	3,27	4,04	19157	13,79	1,39	14,47	3,28
85814	56,90	1,40	4,22	19158	54,02	1,44	58,16	1,65
18875	51,70	1,66	4,22	19200	50,64	1,70	51,20	1,72
85409	91,60	0,53	5,07	19161	92,55	0,53	88,86	0,46
85414	92,50	0,67	5,25	19166	129,21	0,66	93,14	0,64
18811	46,40	4,02	6,32	19196	38,83	3,87	36,33	4,12
3326	27,50	5,09	6,45	19181	29,03	4,90	26,96	5,32
3324	96,30	1,87	6,64	19179	94,07	1,98	93,87	2,13
3323	109,00	1,46	6,86	19178	107,49	1,45	101,41	1,40
18874	70,00	3,43	6,90	19199	68,62	3,39	67,93	3,55
85810	41,40	5,35	7,40	19154	43,92	5,87	38,86	6,42
3329	132,70	1,80	8,37	19184	120,12	1,89	116,78	1,84
Average	**47,50**	**1,93**			**47,79**	**1,87**	**45,23**	**2,03**

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

In 2006, Bumigeme Inc. was mandated by Abcourt Mines Inc. to develop a flowsheet based on laboratory and mill tests on the Abcourt-Barvue deposit in Barraute in the Abitibi area followed by the estimation of the Capex and Opex costs for the construction of a concentrator of 1800 metric tonnes of ore per day.

In collaboration with other professionals, Bumigeme has developed a process consisting in the cyanidation of the ore followed by flotation to produce a Zn- Ag concentrate and to finish by the flotation of the pyrite concentrate to reduce the percentage of S in the final tails to less than 0.30%.

The report entitled “CONSTRUCTION AND OPERATING COSTS OF A 1800 TPD CONCENTRATOR FOR THE ABCOURT – BARVUE PROJECT” was issued in January 2007 by Bumigeme and this section was reproduced in the NI43-101 “TECHNICAL FEASIBILITY STUDY REPORT ON THE ABCOURT-BARVUE DEPOSIT” prepared by Genivar Limited and Bumigeme in partnership, dated November 2010.

13.1 Metallurgical History

13.1.1 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 in Table 13.1.

Table 13.1: Metallurgical balance

Period		**Heads**		**Concentrate**			
From January	**To September**	**Zn %**	**Ag oz/t**	**Zn %**	**Ag oz/t**	**% Recovery**	
						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 lead content in the ore.

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 (Ref. 8, 9, 15).

13.1.2 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.

Table 13.2: Metallurgical balance calculated from May 1985 to June 1990

	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 (Ref. 16)

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.

13.1.3 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:

Table 13.3: Some of the characteristics of the sample

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:

Table 13.4: Best results at a grind of 93.8% -200 mesh

Test No.	Product	Weight %	Zn %	Ag g/t	% Recovery	
					Zn	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:

Table 13.5: Results after 40 hours of cyanidation period

Test	Grind % -200 mesh	% Extraction	
		Ag	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:

Table 13.6: Some Characteristics of composite 1

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:

Table 13.7: Best results from an Ag pre-flotation followed by Zn flotation test conducted on composite 1 ground to 73% -200 mesh

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:

Table 13.8: The characteristics of the composite

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):

Table 13.9: Typical results of a cycle test

Product	Weight %	Zn %	Ag g/t	% Recovery	
				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:

Table 13.10: Some of the main characteristics of the samples

Composite	**Zn %**	**Ag g/t**	**S %**	**Cu %**	**Pb %**	**S.G.**	**Work Index kWh/st**
60% Barvue, 40% Abcourt	5.65	148.8	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 of 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 Mines.

  Representative samples of ore from the Abcourt, Barvue and Vendôme properties were prepared by Abcourt and delivered to the Louvicourt Mines 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.

13.2 Characteristics of the Abcourt-Barvue Ore

13.2.1 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.

13.2.2 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 (Ag3 As S3) 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.

13.2.3 Mill Feed

Based on the geological interpretation and on the estimate of the “in place” mineral resources, carried out by MRB & Associates, Roche Ltd. developed the mining plan and the corresponding mining schedule shown in Table 13.11.

Table 13.11: Ore and Waste Mining Schedule for the Abcourt-Barvue Mines

Year	Annual					Cumulative Summary				
	Ore (t)	Grade			Waste (t)	Ore (t)	Grade			Waste (t)
		Ag (g/t)	Zn (%)	Zn Eq (%)			Ag (g/t)	Zn (%)	Zn Eq (%)	
-1					3,770,355					3,770,355
1	596,000	62.42	2.73	4.04	5,233,309	596,000	62.42	2.73	4.04	9,003,664
2	650,000	58.10	2.76	3.97	5,239,566	1,246,000	60.17	2.75	4.00	14,243,230
3	650,000	52.42	2.87	3.97	5,209,170	1,896,000	57.51	2.79	3.99	19,452,400
4	650,000	37.93	3.37	4.17	5,189,816	2,546,000	52.51	2.94	4.04	24,642,216
5	650,000	34.34	3.43	4.15	3,180,305	3,196,000	48.82	3.04	4.06	27,822,521
6	650,000	38.95	3.37	4.18	2,501,895	3,846,000	47.15	3.09	4.08	30,324,416
7	650,000	57.65	3.26	4.46	1,488,027	4,496,000	48.67	3.12	4.13	31,812,443
8	650,000	69.17	3.10	4.55	791,720	5,146,000	51.26	3.12	4.19	32,604,163
9	650,000	67.46	3.06	4.47	312,364	5,796,000	53.07	3.11	4.22	32,916,527
10	650,000	71.75	3.12	4.62	230,162	6,446,000	54.96	3.11	4.,26	33,146,689
Total	**6,446,000**	**54.96**	**3.11**	**4.26**	**33,146,689**					

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.

13.3 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 of the ore 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:

- float a pyrite concentrate to be stored in a "unit-cell" (safe sulphide containment cell) according to the directives of the Ministry of Environment;

- reject a tailing assaying less than 0.3% S to be stored into a conventional tailings pond.

Having in mind the good results obtained by Barvue Mines Ltd in their operations from 1952 to 1957 from their 4500 short tons per day concentrator in Barraute that was producing a good Zn-Ag concentrate, the management of Abcourt Mines decided to review the project with a similar flowsheet (see section 13.1.1 for some production result of Barvue Mines)

In this new approach, a more traditional flowsheet consisting in rougher-scavenger-cleaners circuit, eliminating the Capex and Opex costs of the cyanidation circuit proposed in the 2006 study could result in substantial costs reduction, while remaining financially attractive.

So, in 2017, the management of Abcourt Mines Inc. mandated the URSTM of the University of Québec in Abitibi-Temiscamingue (UQAT) to realize new tests to simplify the flowsheet as follow :

- Direct flotation of a Zn-Ag concentrate to be shipped to a zinc smelter;
- Float a Pyrite concentrate from the flotation tailings of above, to be stocked in a "unit-cell" according to the directives of the Ministry of Environment;
- Reject a final tailings (non-acid generating) assaying less than 0.3%S to be stored in a conventional tailings pond.

Following this series of new tests a simplified new flowsheet has been developed and the capital and operating costs have been actualized followed by a revised financial analysis.

This 2018 new study of the Abcourt project is based on the new flowsheet, but for the interested readers, the previous laboratory and milling test, can be consulted in the report NI43-101 Feasibility Study on the Abcourt deposit prepared by Genivar (WSP) and Bumigeme in november 2010. This report has been filed on SEDAR.

13.4 New Process Development (2017-2018)

13.4.1 Objective

The principal objective of those tests was to maximize the recovery of silver in a zinc concentrate and to evaluate the economical feasibility of the recovery of silver by a zinc smelter.

Table 13.12: Summary of the tests done

Test No	Description
AB-62	Kinetic flotation test on the second ore sample to evaluate the flotation time
AB-63	Cyclic flotation test (6 cycles) on the second ore sample
AB-64	Cyclic flotation test (4 cycles) on the second ore sample with flotation flowsheet slightly modified
AB-64-CN-A-4	Cyanidation test done on the pyrite concentrate of the AB-64 flotation without regrind
AB-64-CN-B-4	Cyanidation test done on the pyrite concentrate of the AB-64 flotation with regrind
AB-64-CN-A-3	Cyanidation test done on the pyrite concentrate of the AB-64 flotation(3 cycles) without regrind
AB-64-CN-B-3	Cyanidation test on the pyrite concentrate of the AB-64 (3cycles) with regrind

The above tests have been realized in August and October 2017 in the laboratories of the CEGEP of l'Abitibi Temiscamingue par M. Jean Lelièvre, Eng and M.Sc, for the URSTM.

The analyses were done by Le Laboratoire Expert of Rouyn-Noranda, Multilab of Rouyn-Noranda, Actlabs of Ste Germaine-Boulé and Actlabs of Ancaster, Ontario.

13.4.2 Sample Description

The samples from Abcourt-Barvue's core drilling were shipped to the laboratories of URSTM and crushed and splited in lots of 2 kg and 1 kg each. The procedure is shown below :

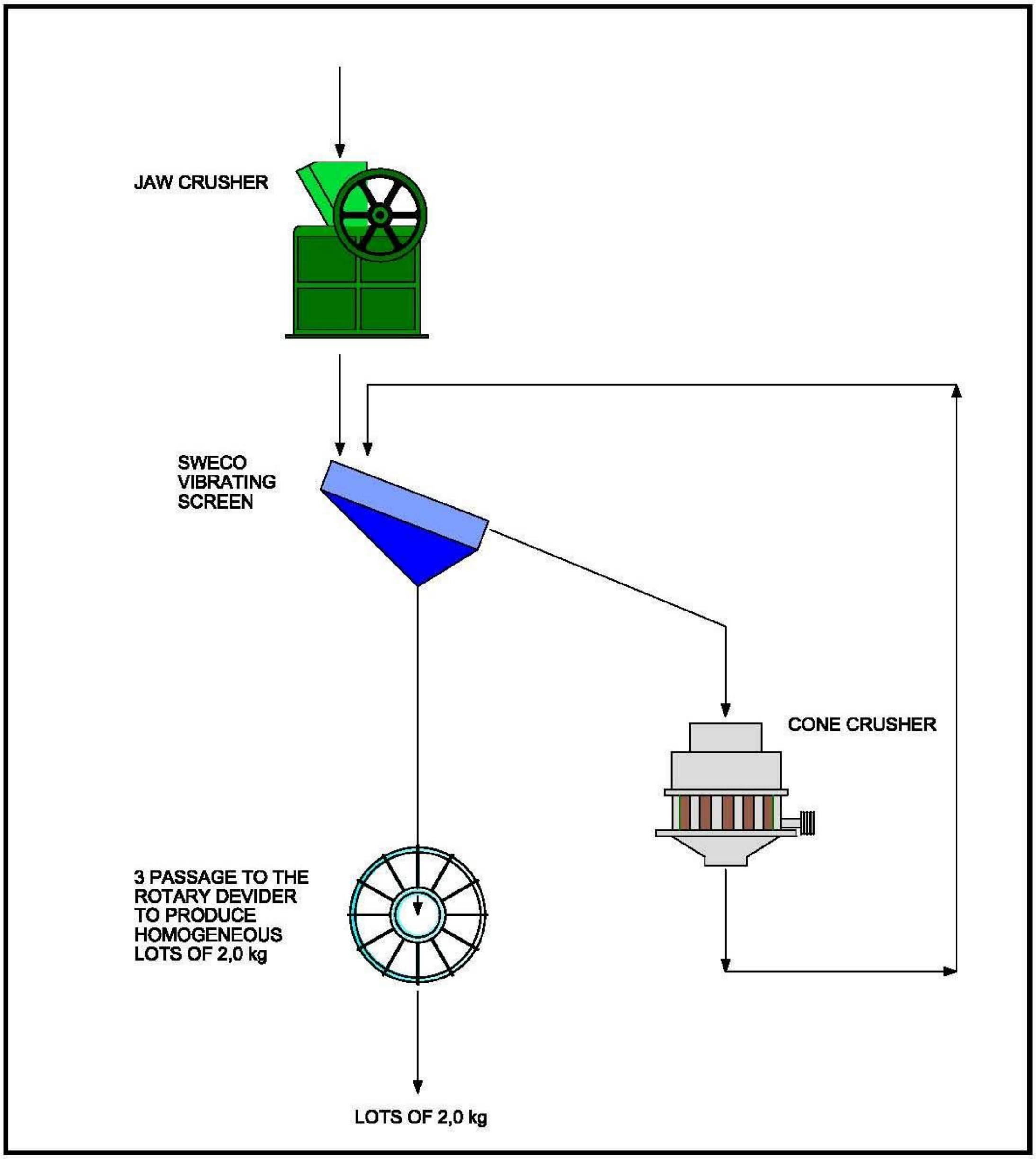

Figure 13.1: Crushing and Separation of Lots of 2kg

13.4.3 <u>Flotation Tests</u>

13.4.3.1 Cyclic Tests AB-49 (4 Cycles)

The AB-49 cycle test was done on the first ore sample received from Abcourt-Barvue based on the flotation data in the project PU-2007-12-356, realized by URSTM in 2007.

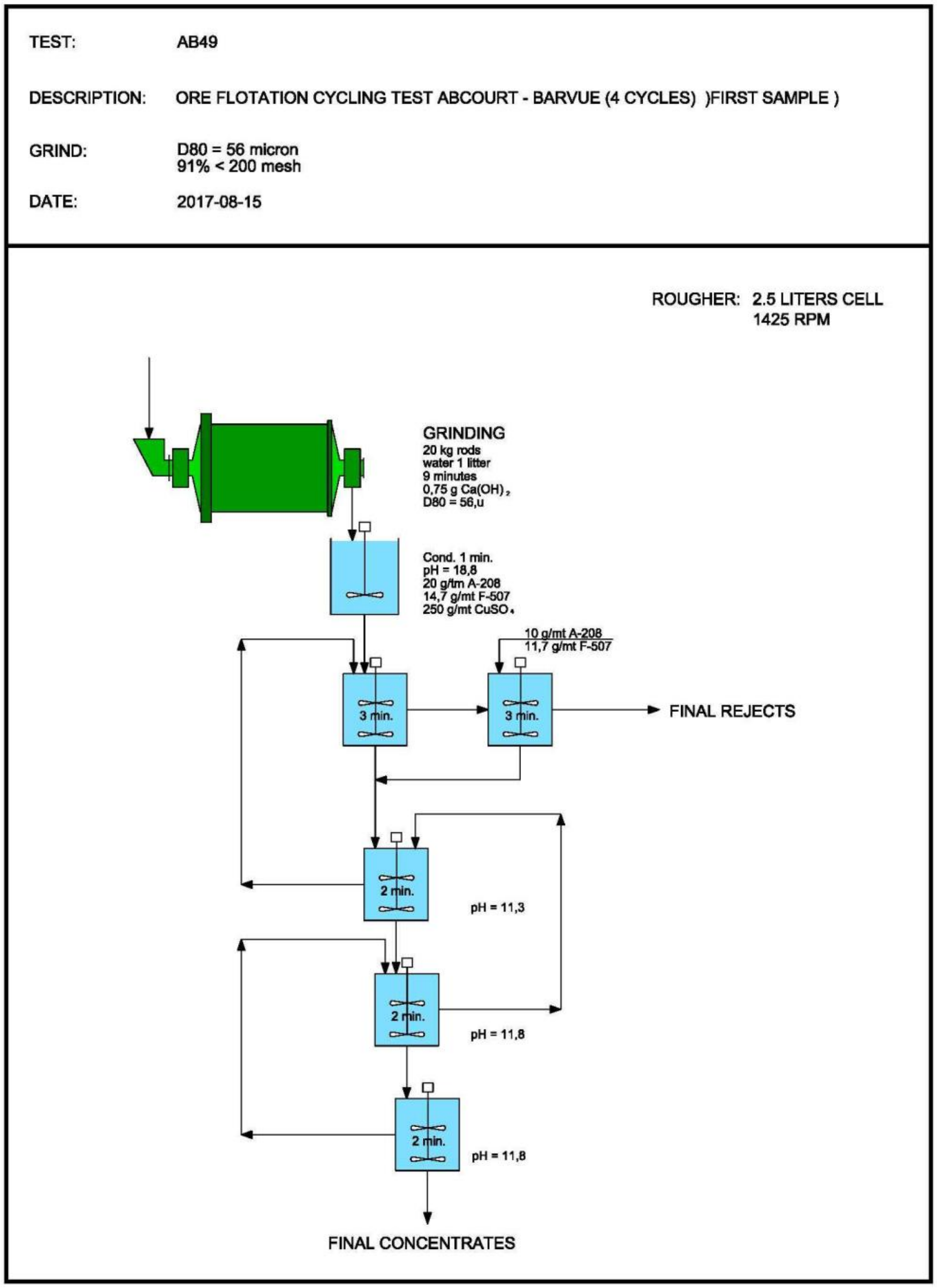

Figure 13.2: Cyclic test AB-49 (4 cycles)

Table 13.13: Metallurgical balance of cycle test AB-49 (Cycle 3 and 4 average)

	Weight (g)	% mass	Grade		Units		% distribution	
			Ag (g/tm)	% Zn	mg Ag	g Zn	Ag	Zn
Final Concentrate Zn-Ag	61.5g	6.1%	631.1	51.0%	38.81	31.4	**69.6%**	**96.5%**
Final tailings	942.8g	93.9%	18.0	0.12%	16.9	1.1	**30.4%**	**3.5%**
Calc. feed	1 004.3g	100.0%	55.5	3.24%	55.73	32.50	100.0%	100.0%

The silver recovery is only 69.6% in the Zinc-silver concentrate which is inferior of the recovery obtained in 2007. In the project PU-2007-12-356, that reached 79%. One possible reason could be that the 6 minutes flotation time is insufficient for the silver recovery.

13.4.3.2 Kinetic Flotation Test AB-62

As the silver recovery was not satisfactory in cyclic test AB-49, a kinetic flotation test was conducted on the ore in order to estimate the optimum flotation time necessary to obtain the best recovery of zinc and silver in the Zn-Ag concentrate. Test detail on the following page.

TEST:	AB62
DESCRIPTION:	FLOTATION KINETIC TEST
FLOTATION REJECTS:	K80 = 74 micron
DATE:	2017-09-07

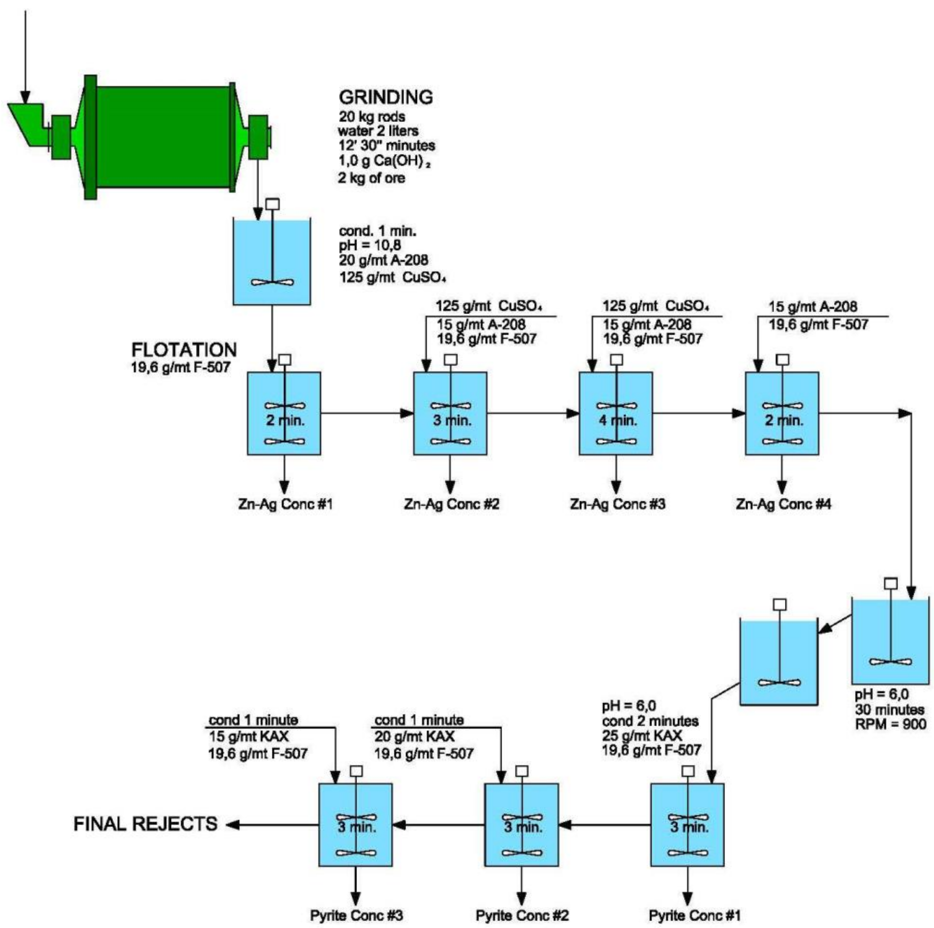

Figure 13.3: Kinetic flotation test AB-62

Table 13.14: Metallurgical balance (Ag) - Test AB-62

	Weight(g)	%Mass	Grade (g Ag/tm)	mg Ag	% Distribution	% Cumul. distribution	Cumul. Flotation Time (Minutes)	Analysis Code
Concentrate Zn-Ag #1	212.4	10.65%	449.75	95.53	68.8%	68.8%	2.0	S-107
Concentrate Zn-Ag #2	89.9	4.51%	130.15	11.70	8.4%	77.2%	5.0	S-106
Concentrate Zn-Ag #3	48.3	2.42%	84.75	4.09	2.9%	80.2%	9.0	S-105
Concentrate Zn-Ag #4	31.3	1.57%	62.40	1.95	1.4%	81.6%	11.0	S-104
Sulfide concentrate #1	99.2	4.97%	58.85	5.84	4.2%	85.8%	3.0	S-103
Sulfide concentrate #2	51.2	2.57%	31.55	1.62	1.2%	86.9%	6.0	S-102
Sulfide concentrate #3	34.8	1.74%	24.70	0.86	0.6%	87.6%	9.0	S-101
Final Tailings	1428.0	71.58%	12.10	17.28	12.4%	100.0%		S-100
Calculated Feed	1995.1	100.00 %	69.60	138.87	100.0%			

Table 13.15: Metallurgical balance (Zn) - Test AB-62

	Weight(g)	%Mass	Content (%Zn)	g Zn	% Distribution	% Cumul. distribution	Cumul. Flotation Time (Minutes)	Analysis Code
Concentrate Zn-Ag #1	212.4	10.65%	25.00%	53.10	82.2%	82.2%	2.0	S-107
Concentrate Zn-Ag #2	89.9	4.51%	9.50%	8.54	13.2%	95.4%	5.0	S-106
Concentrate Zn-Ag #3	48.3	2.42%	1.58%	0.76	1.2%	96.5%	9.0	S-105
Concentrate Zn-Ag #4	31.3	1.57%	0.68%	0.21	0.3%	96.9%	11.0	S-104
Sulfide concentrate #1	99.2	4.97%	0.23%	0.23	0.4%	97.2%	3.0	S-103
Sulfide concentrate #2	51.2	2.57%	0.25%	0.13	0.2%	97.4%	6.0	S-102
Sulfide concentrate #3	34.8	1.74%	0.27%	0.09	0.1%	97.6%	9.0	S-101
Final Tailings	1428.0	71.58%	0.11%	1.57	2.4%	100.0%		S-100
Calculated Feed	1995.1	100.00 %	3.24%	64.64	100.0%			

Figure 13.4 represents the curve of the cumulative recovery of zinc and silver in a Zn-Ag concentrate in function of the flotation time in minutes. This graph shown that the flotation kinetic of zinc is faster than the one of silver.

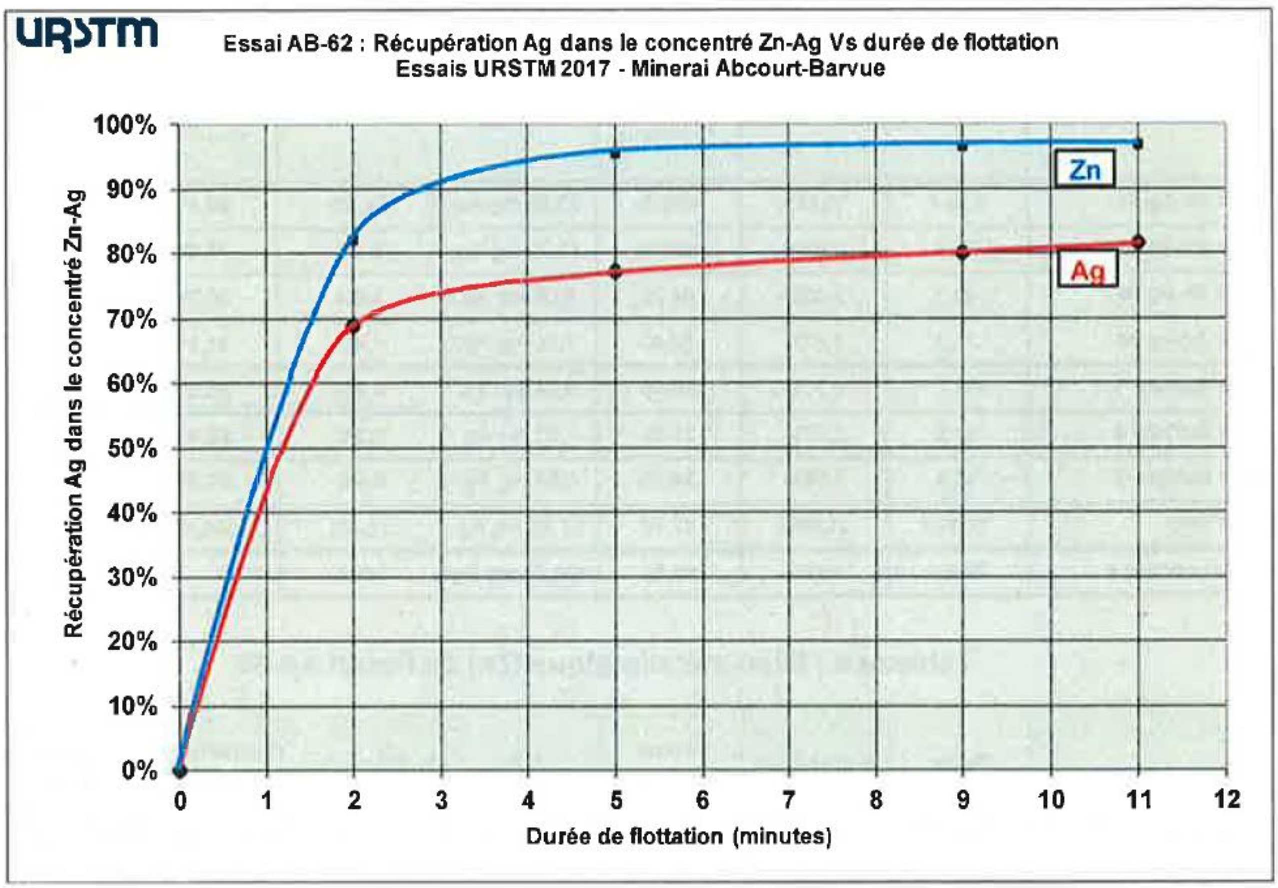

Figure 13.4: Graph of the cumulative recovery of zinc and silver in a Zn-Ag concentrate VS the flotation time - Test AB-62

Figure 13.5 presents the curve of the concentrate silver grade versus flotation time. It shows the silver grade of the Zn-Ag concentrate produced at a specific time during the flotation period. It does not show the cumulative silver grade in the Zn-Ag concentrate. The curve in Figure 13.5 indicates that a flotation time of 10.2 minutes is sufficient for silver. This is normally the time when the silver grade of the concentrate produced at that time equals the silver grade of the mill feed.

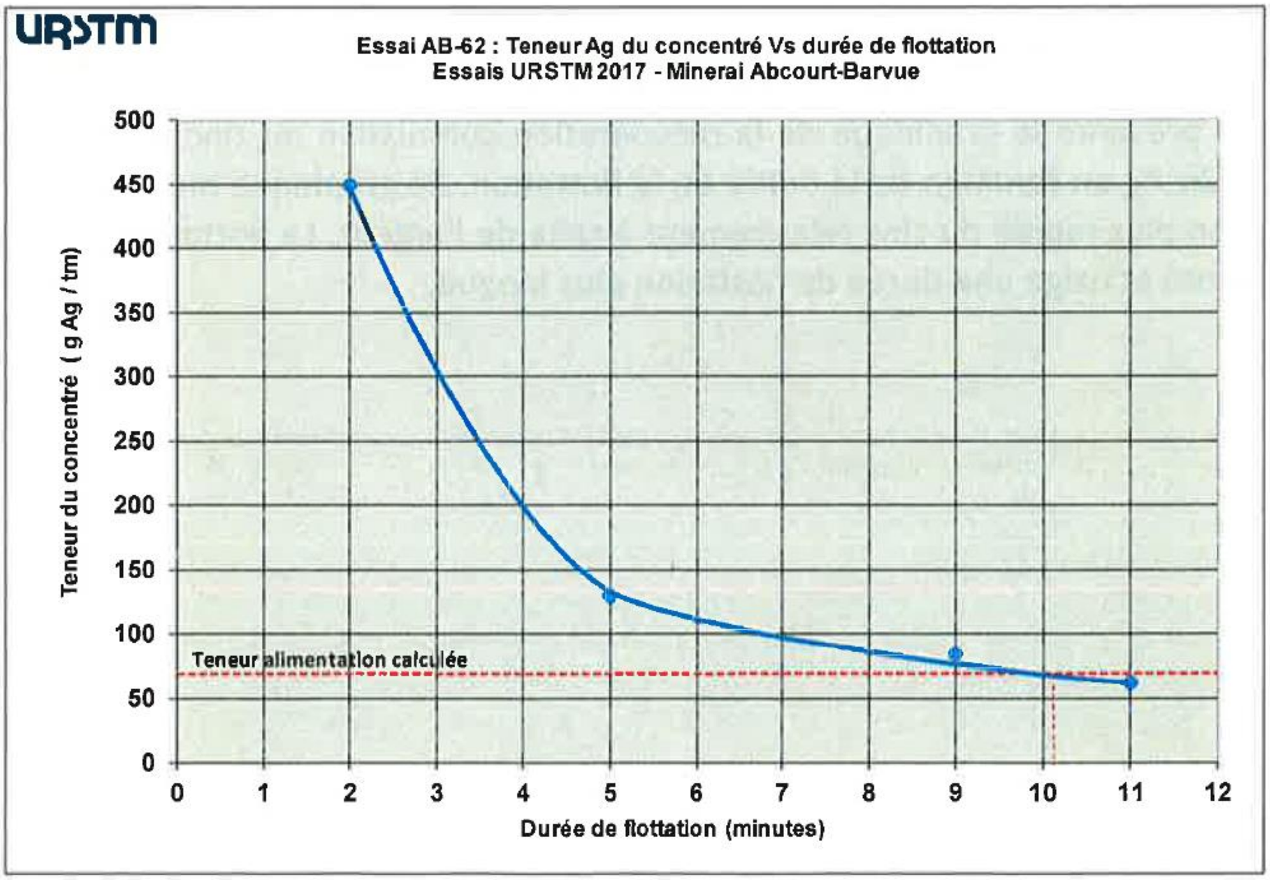

Figure 13.5: Ag grade of the concentrate VS flotation time (Test AB-62)

For the zinc, a flotation time 7.9 minutes seems sufficient. However to maximize the combined recovery of silver and zinc, the flotation time of 11 minutes was retained for test AB-63 and AB-64.

With a flotation time of 11 minutes for test AB-62 the silver recovery obtained was 81.6% and 96.9% for zinc (See Table 13.14 and Table 13.15)

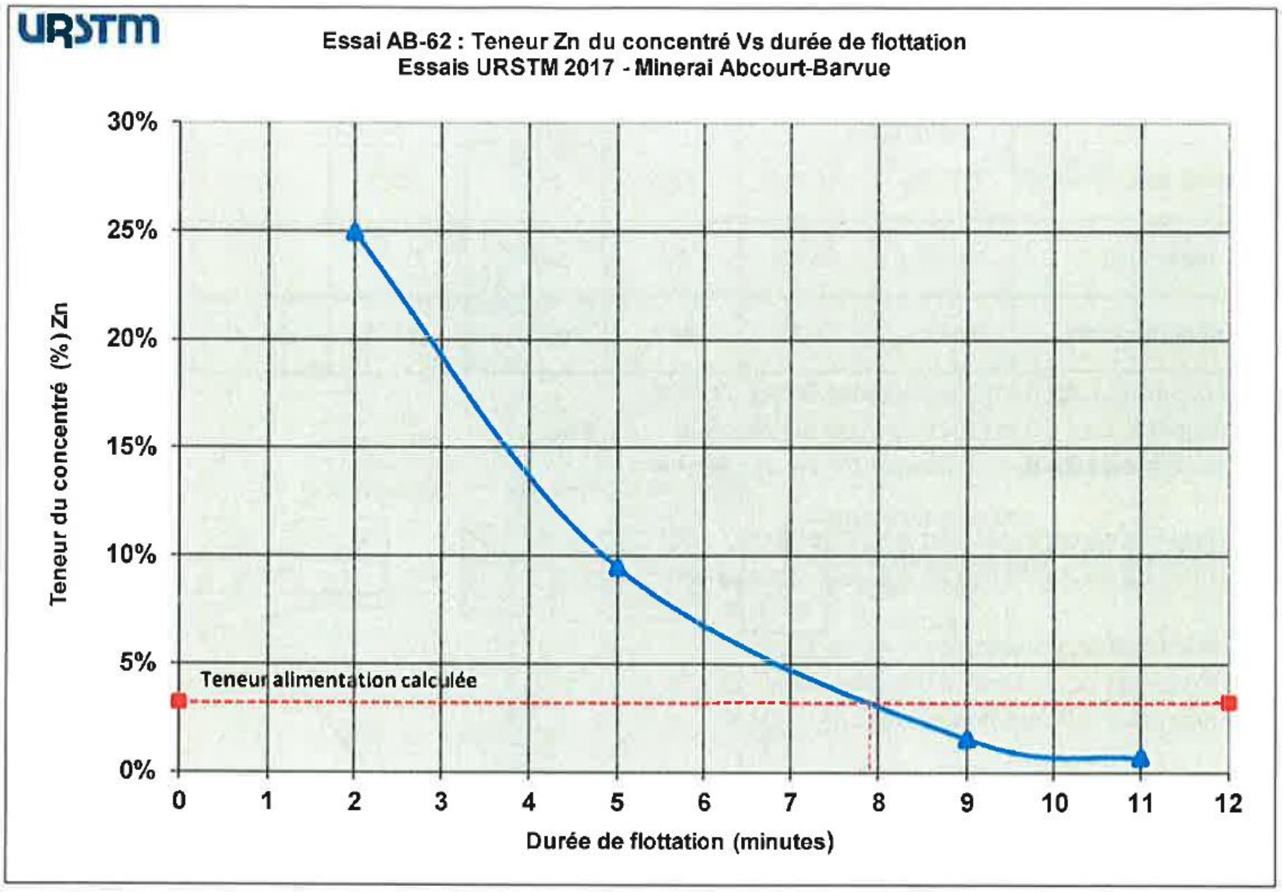

Figure 13.6: Zn grade of the Concentrate VS Flotation time (Test AB-62)

13.4.3.3 Cyclic Flotation Test AB-63 (6 Cycles)

A total of two cycles tests was done in this program AB-63 and AB-64.

The test AB-63 was based on the flotation flowsheet of test N°15 done in 2007 in the project PU-2007-12-356. and with only 6 minutes of flotation time.

With the Abcourt Barvue 2017 ore sample the results of test AB-62 suggest to increase the flotation time to 11 minutes to maximize the recovery of silver. For this a cleaner-scavenger stage was added to the flowsheet. Figure 13.7 represents the flowsheet used for cyclic test AB-63.

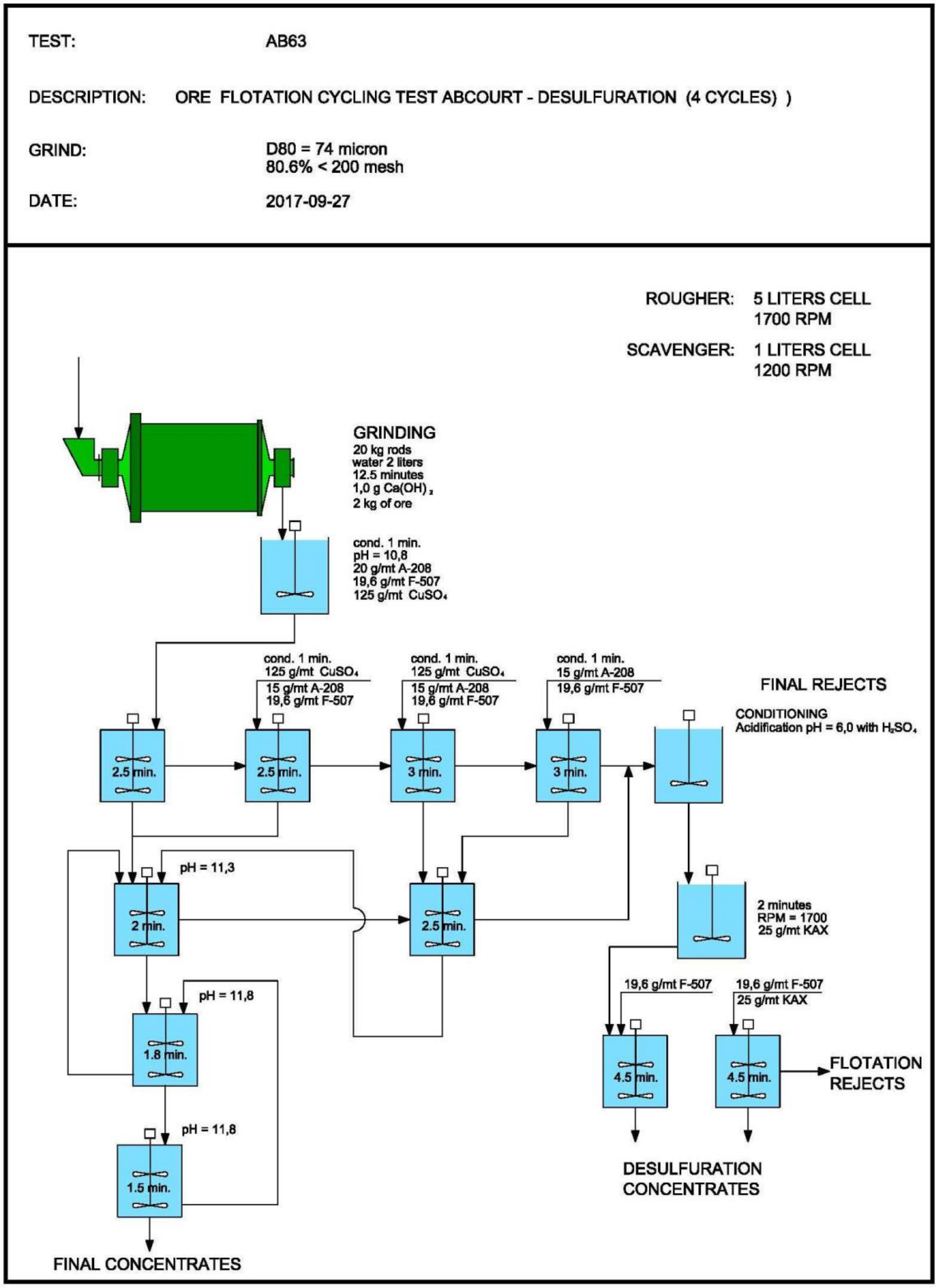

Figure 13.7: Cyclic Flotation (Test AB-63)

This test has given a silver recovery of 77.8% in the Zn-Ag concentrate which is closed to the recovery of 2007 for the test AB-15, at 79%

To note that the sample of 2007 had a silver grade of 31.2g Ag/tm versus 64.1gAg/tm in the sample of 2017 (see results in the Table 13.16)

The grade of the zinc in that test is low at 37.5%Zn, while the expected grade is at least 50%Zn. This result can be explained by the flotation time too long at the cleaner stage. For this reason the flotation times in the cleaning stages have been reduced substantially.

Table 13.16: Metallurgical balance - Cyclic test AB-63 (Average of cycles 4, 5 and 6)

	Weight (g)	% mass	Grade		Units		% distribution	
			Ag (g/tm)	% Zn	mg Ag	g Zn	Ag	Zn
Final Concentrate Zn-Ag	160.1g	8.1%	618.2	37.5%	98.95	60.0	77.8%	96.6%
Concentrate desulphurization	260.8g	13.1%	62.6	0.3%	16.32	0.8	12.8%	1.4%
Final tailings	1563.3g	78.8%	7.6	0.08%	11.85	1.3	9.3%	2.1%
Calc. feed	1984.2g	100.0%	64.1	3.13%	127.12	62.14	100.0%	100.0%

13.4.3.4 Cyclic Flotation Test AB-64 (4 Cycles)

The cycle test AB-64 has been realized with less float time in the cleaning stages as follows :
N°1 cleaner : 2.0 minutes
N°2 cleaner : 1.0 minute
N°3 cleaner : 0.75 minute
The grinding has been increased to a D_{80} at 64µ versus a D_{80} of 74µ for the test AB-63.

TEST: AB64

DESCRIPTION: ORE FLOTATION CYCLING TEST ABCOURT - DESULFURATION (4 CYCLES))

GRIND: D80 = 64 micron
86.6% < 200 mesh

DATE: 2017-09-27

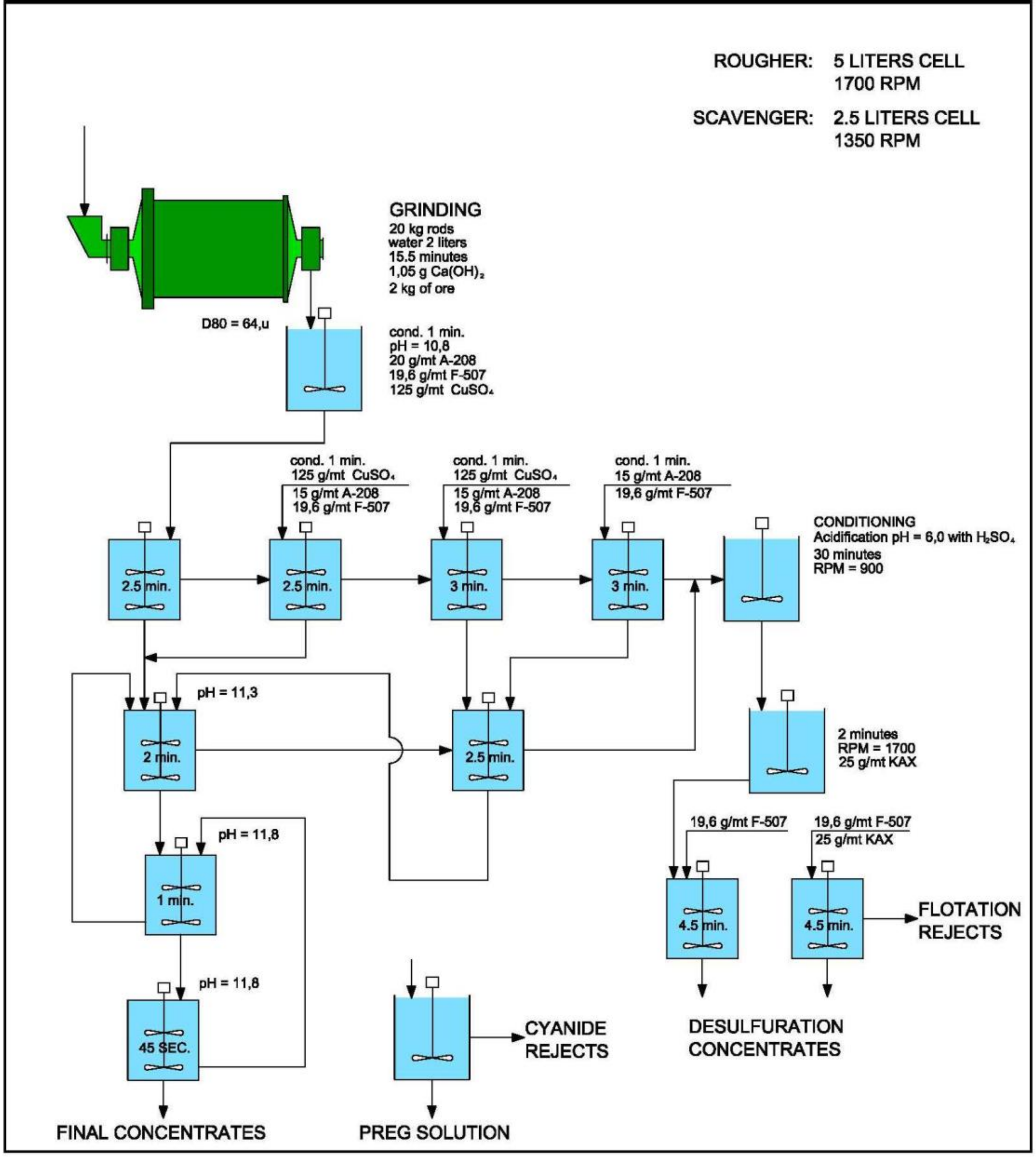

Figure 13.8: Cyclic flotation Test AB-64 (4 Cycles)

Table 13.17: Metallurgical balance - Cyclic flotation AB-64 (Average of cycles 3 and 4)

	Weight (g)	% mass	Grade		Units		% distribution	
			Ag (g/tm)	% Zn	mg Ag	g Zn	Ag	Zn
Final Concentrate Zn-Ag	132.4g	6.7%	740.6	53.4%	98.02	70.7	77.8%	97.5%
Concentrate desulphurization	302.3g	15.2%	58.6	0.3%	17.71	1.0	14.0%	1.4%
Final tailings	1555.1g	78.2%	6.7	0.05%	10.34	0.8	8.2%	1.1%
Calc. feed	1989.7g	100.0%	63.4	3.64%	126.07	72.51	100.0%	100.0%

Generally speaking the results are similar to those of cyclic test AB-63 excepted for the grades of the final concentrate. The reduction of flotation time in the cleaners stages has increased the grade of Zn to 53.4% and the silver to 740.6gAg/tm in the final concentrate, which corresponds to the objective of the tests.

The silver recovery at 77.8% is closed to the 79% obtained in the Zn-Ag concentrate of the project PU-2007-12-356.

The zinc recovery at 97.5%is slightly higher than the 96.6% in the test AB-63.

The sulfur (S) percentage after the pyrite flotation is low at 0.136% in the final tailing.

13.4.3.5 Chemical Analysis of the Final Concentrate of Test AB-64

Chemical analysis has been completed on the above final Zn-Ag concentrate. Those analysis have been done by the following laboratories:

Expert Laboratory for zinc and silver;

Actlabs of Ste Germaine-Boulé for silver;

Actlabs of Ancaster on the whole rock and on the multi-element analysis.

Table 13.18: Chemical analysis - Zn-Ag concentrate

Ag ppm	As ppm	Ba ppm	C	Ca	Cd ppm	Cl	Co ppm	Cu ppm	f	Fe	Ge ppm	Hg ppm
710	1200	10	0.33%	0.12%	>1000	0.01%	63	3350	0.02%	8.67%	<0.1	6.21
Mg	Mn ppm	Ni ppm	Pb ppm	Sb ppm	Se ppm	Te ppm	Ti ppm	S	Zn	SiO_2	AlO_3	
0.05%	1310	20.4	4310	403	16.1	0.1	0.16	31.4%	54.1%	6.89%	1.86%	

13.4.3.6 Synthesis of the Flotation Tests Done

The Table 13.19 resumed the results obtained from the three flotation tests done.

Table 13.19: Resume flotation test (AB-49, AB-63 and AB-64)

Cyclic flotation Tests	D_{80}	Zn-Ag Final concentrate				Desulphurisation Concentrate				Final tailings				Grade feed calculated	
		Grade		%Distribution		Grade		%Distribution		Grade		%Distribution			
		Ag g/tm	Zn	Ag	Zn	Ag g/tm	Zn	Ag	Zn	Ag g/tm	Zn	Ag	Zn	Ag g/tm	Zn
AB-49	56µ	631.1	51.0%	69.6%	96.5%					18.0	0.12%	30.4%	3.5%	55.5	3.2%
AB-63	74µ	618.2	37.5%	77.8%	96.6%	62.2	0.33%	12.8%	1.4%	7.6	0.08%	9.3%	2.1%	64.1	3.1%
AB-64	64µ	740.6	53.4%	77.8%	97.5%	58.6	0.34%	14.0%	1.4%	6.7	0.05%	8.2%	1.1%	63.4	3.6%

13.4.4 Conclusions

The cyclic flotation tests realized on the ore of Abcourt-Barvue have shown the possibilities to reach recoveries of 77.8% for silver and 97.5% for zinc in a Zn-Ag concentrate assaying 740.6g/tm Ag and 53.4%Zn.

Those results have been obtained in the cyclic flotation test AB-64.

The sulfur (S) content in the final tailings is low at 0.136% and those tailings should not be acid generating.

Those results have been obtained based on the flotation data of test AB-15 and using the same collectors than those used in project PU-2007-12-356. Other reagents could be investigated to try to increase the silver recovery.

In this present test program no attention was given to gold as its content in this ore is low and erratic.

14.0 MINERAL RESOURCE ESTIMATE

The text of this section was taken from the NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, February 28, 2014.

In order to update the mineral resource evaluation of the Abcourt-Barvue property, the Author used the results of the resource estimate carried-out by MRB & Associates in May 2006. This report, entitled "NI 43-101 Resources Evaluation Report for the Abcourt-Barvue Project", was compliant with regulations of the NI 43-101 and its Form 43-101F1 at the time of preparing such technical report.

The reader should be aware that a mineral resource evaluation is not a precise calculation, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. For these reasons, reporting of tonnage and grade figures are generally referred to as an evaluation, not a calculation. Moreover, mineral resources exploration and development is highly speculative and involves a high degree of risk, which even a combination of careful evaluation, experience and knowledge may not be able to avoid.

According to the CIM Standing Committee on Reserves Definitions (CIM Definition Standards, Nov. 27, 2010)

> *"A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals 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."*

14.1 Geological Interpretation

The drill data was plotted with the AutoCad software on 15 to 30-metre spaced cross-sections and mineralized zones were allocated to the appropriate mineralized structure. The geological interpretation is concordant with the different units and fits well with the geometry of the deposit. Lateral and at-depth geological continuity of the mineralization was confirmed by the Barvue open pit (5 Mt extracted), the geological mapping and sampling of underground workings (0.7 Mt extracted) and by 677 surface and underground diamond drill holes spread over 2.1 km of continuous mineralization.

The interpretation made by the Author shows the Gray Schist (GS) unit as a major sheared structure characterised by a strong sericite-talc alteration generally striking at 270° (Abcourt side) to 305° (Barvue side) and dipping 75-65° to the north. The west side of the deposit (Abcourt) shows mineralized layers parallel to each other mainly located in the hanging-wall of the G.S. unit. These zones are referred as the Hanging Wall Zones by the Author. A zinc-silver enriched horizon is also well defined in the footwall of the G.S. unit, between sections 5070 E and 5250 E and between sections 5625 E and 5745 E. This zone has been referred to as the Footwall Zone by the Author.

In plan view, metric-scale (6 to 15 m) lateral displacements of the mineralized zone along E-W (N090°) and SW-NE (N045°) were caused by a set of vertical brittle faults. These post-mineralization faults do not affect the global resources estimate but will have to be taken into account in future mine planning. These structures are well known by ABCOURT's management as the limits of the stopes mined between 1986 and 1990 were delineated by these transverse faults. Another major N-S fault, located at the eastern limit of the Barvue open pit, caused an apparent 300 m horizontal displacement of the main mineralization to the south. Finally, a major late-stage NNE-trending fault also displace the main mineralization at the west end of what is known from the Abcourt zone on section 5 000 E.

Considering that; 1) both the Footwall and the Hanging Wall Zones are in fact the zinc-silver enriched walls of the Abcourt-Barvue orebody, 2) they are close enough to share common underground openings and, 3) distinctive symbols were used by ABCOURT to represent both zones on a longitudinal view (circles for the hanging-wall and squares for the footwall zones), the Author agrees with ABCOURT on the principle of showing one longitudinal view of the overall mineral resources instead of two. The Author's only concern is that, in doing such a composite representation of the two zones, the longitudinal view is not showing each of the ore outline properly.

14.2 Methodology

The polygonal method was used in previous estimation work completed by Roche (1999), ABCOURT (2004), InnovExplo (2005) and MRB (2006). The Abcourt-Barvue deposit was entirely reviewed, re-evaluated and classified into measured, indicated and inferred mineral resources.

Polygons have been traced using two different approaches: (i) on cross-section for measured, indicated and part of the inferred mineral resources and (ii) on the longitudinal view for deep seated, inferred mineral resources.

The estimated resources blocks were delineated in two (2) steps. First, the measured and indicated resources were delimited on cross-sections using the parameters defined in Section 14.3 for areas having a high amount 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 borehole intercepts for defining the resources blocks. Grades of the estimated blocks were obtained from weighted averages of borehole 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 borehole 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. Tonnages for each block was calculated in multiplying the lateral influence of the intersection (7.5, 15, or 30 m) by the area of the block and by a specific gravity of 2.91 t/m^3.

Inferred mineral resources were determined from polygons drafted on the longitudinal view in areas characterized by fewer borehole 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 65 m distance from drill holes intercepts was used until reaching a mid-distance between borehole intercepts or reaching the limit defined by the measured and indicated resources. The area covered by each inferred polygon was calculated with AutoCad. The weighted averages were calculated from borehole intercepts with their corresponding horizontal widths. They were multiplied by their respective areas as represented on the longitudinal view to evaluate the volume of each block.

The open pit outlines were provided by ABCOURT based on the pit design prepared by Genivar in 2007. On the Barvue side, Genivar is considering the deepening of the old open pit including a shallower extension going well into the Abcourt side as far as section 5200 E. A shallower open pit, centered on section 5030 E, is also planned in the westernmost portion of the deposit.

This mineral resources estimate is presented with a cut-off grade of 1.99% Zn (Ag expressed in Zn equivalent + % Zn) for the Abcourt and Barvue open pits, and 4.51% Zn for the underground operations.

14.3 Resources Categories Definition

Resources categories for the Abcourt-Barvue property are following the recommendations of the CIM Standing Committee on Resources and 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."*

14.3.1 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 resources category, the Author has used the following parameters:

• Blocks are within a range of up to 50 metres from a drill hole, generally at 30-metre centres, and/or mineralization is located at a range of between 30 and 50 metres from stopes, drifts, raises or other sampled faces; and
• Blocks defined on a longitudinal view are limited by the area covered by the indicated and measured resources blocks.

The Author is considering 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 should not be used to evaluate reserves as further drilling (or sampling) is requested. Given that inferred resources have a greater uncertainty associated with them, a Preliminary Feasibility Study relying on inferred resources is going to have a much higher degree of uncertainty associated with it than a Feasibility Study which use measured and indicated resources for reserves estimation.

14.3.2 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, the author has used the following parameters:

• Blocks are generally not located between two levels;
• Blocks are within 30 metres of diamond drill holes which are drilled at 30-metre centres and/or mineralization located between up to 30 metres from stopes, drifts, raises or other sampled faces; and
• Lateral influence of blocks are defined on a longitudinal view.

14.3.3 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 the Abcourt-Barvue property, the nature and character of the geology and the grade continuity of the main mineralization is sufficiently confirmed by closely spaced drill holes, drifts, and stopes. The database contains the information about 667 surface and 775 underground holes drilled at a 15 to 30-metre line spacing along with 5.4 km of drifts and sub-levels with silver and zinc assay results.

For the measured resource category, the Author has used the following parameters:

- Blocks are located between two levels where a drift runs along the main mineralization on both the upper and lower levels;
- Blocks are within 15 metres of diamond drill holes which are drilled at 30-metre centres and/or mineralization is extrapolated up to 15 metres from stopes, drifts, raises or other sampled faces;
- Blocks are within the limits of the open pit design prepared by Genivar in 2007; and
- Lateral influence of blocks defined on a longitudinal view.

14.4 Specific Gravity

A specific gravity of 2.91 t/m^3 was used for the present evaluation. This number is based on several test works taken by mill's laboratory in Matagami and Louvicourt (Table 14.1). It is noteworthy that a specific gravity of 3.00 t/m^3 was used in the previous mineral resources evaluation on the basis of two (2) bulk sample measurements instead of six (6).

The Author carried out a theoretical calculation assuming the presence of 10% sulphides in a felsic tuff. With a silver-zinc mineralization composed of 5.3% pyrite (SG: 5.025 t/m^3), 4.7% sphalerite (SG: 6.38 t/m^3) and 0.05% silver (SG: 10.49 t/m^3) associated with 90% of felsic tuff (SG: 2.60 t/m^3), it is estimated that the specific gravity would be around 2.91 t/m^3. This estimation fits well with the specific gravity measured in mill tests.

Table 14.1: Historic Specific Gravity tests taken from mill labs

Date	Location	Area	S.G.
Jan-81	Matagami	Abcourt (A)	2,84
May-81	Matagami	Barvue (B)	2,94
Apr-84	Matagami	Abcourt (A)	3,02
May-84	Matagami	Barvue (B)	2,88
May-84	Matagami	40%A/60%B	2,81
Dec-04	Louvicourt	50%A/50%B	2,99
Average:			**2,91**

14.5 Minimum Width

A minimum horizontal width of 1.5 metre was used to determine the minimum mineralized intervals in drill holes or underground chip samples. When lower than 1.5 metre, the interval was diluted with low grade material from rock adjacent to the interval. The mineralized intervals on Abcourt (west side) and Barvue (east side) are respectively averaging 5.5 metres and 9.3 metres in thickness.

14.6 Maximum Cut-Off Grade

The high silver grades were cut to 514 g/t Ag for 15 of the holes drilled in 2010 and 2011 (Table 14.2

A total of 192 samples grading over 514 g/t Ag were cut to that grade for the whole database. This procedure was used by ABCOURT in previous grade reconciliations to reduce the risks of overestimating the silver content of the deposit.

No maximum cut-off was applied to the high zinc values as in previous mineral resources evaluations. In fact, from the 2 667 core samples collected in the whole diamond drilling program of 2010-2011, only five (5) samples graded more than 10% Zn. It means that 0,18% of the samples were higher than 10% Zn, the highest being 13.01% Zn over 0.90 metre in borehole AB10-10.

Table 14.2: Silver values grading higher than 514 g/t intersected in holes drilled in 2010-2011

Section	Hole	From (m)	To (m)	Length (m)	Sample ID	Ag (g/t)	Zn (%)
5670	AB 10-04	198,70	200,10	1,40	3214	584,10	2,79
5250	AB 10-09	202,40	203,20	0,80	3399	605,00	1,20
5250	AB 10-10	166,60	167,50	0,90	3374	1244,00	13,01
5190	AB 10-11	239,60	240,40	0,80	3426	531,60	3,49
5190	AB 10-11	238,80	239,60	0,80	3425	580,60	1,03
5190	AB 10-12	204,50	205,50	1,00	3454	877,00	5,83
5130	AB 10-16	208,60	209,50	0,90	10631	620,86	3,39
5130	AB 10-16	197,50	198,50	1,00	10620	1080,95	6,51
5070	AB 10-20	219,30	220,60	1,30	10801	823,85	8,88
5040	AB 11-24	139,40	140,60	1,20	10941	986,90	7,21
5010	AB 11-27	136,60	138,00	1,40	85052	590,30	9,16
5580	AB 11-29	227,10	228,20	1,10	85149	544,20	3,01
5850	AB 11-47	195,40	196,50	1,10	85912	768,30	3,34
5460	AB 11-48	169,50	171,00	1,50	85956	603,95	3,64
5340	AB 11-58	210,00	210,40	0,40	84419	515,30	5,59
5340	AB 11-58	203,30	204,30	1,00	84412	607,30	1,95
5040	AB 11-67	243,40	244,70	1,30	18877	4696,00	11,75
5220	AB 11-71	294,50	296,00	1,50	19064	995,00	3,85

14.7 Minimum Cut-Off Grade (as estimated by ABCOURT)

The first notion to understand is that the Abcourt-Barvue's mineralization is polymetallic, silver and zinc being the most abundant elements. So, silver values have to be expressed in zinc equivalent to get a zinc grade equivalence (identified by the "Eq" symbol in our tables). For the

Abcourt-Barvue deposit, the grade equivalent is the sum of the zinc (%) and silver (g/t) contents for each mineralized intersection:

Grade equivalence = ((g/t Ag / 31.105) x (silver vs zinc ratio)) + % 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 determined by calculations who take into account several parameters amongst which we have;

- Estimated zinc and silver grades from resources evaluation,
- Milling processes to use and expected metal recoveries,
- Zinc rate of concentration,
- Silver grade in zinc concentrate,
- N.S.R. for one tonne of zinc concentrate at a price of x US$/lb for zinc, y US$/oz for Ag and z US$/oz for gold at a given exchange rate for the Canadian and American currencies,
- Smelting, transportation and other costs.

At the time the mineral resources evaluation was initiated (October 2012), the 3-year average price on the LME for gold, silver and zinc were respectively standing at 1 360 US$ per ounce, 27.55 US$ per ounce and 0.96 US$ per pound. The exchange rate was almost the same for the two currencies (1 US$ = 1 Can$). The minimum cut-off grade was based upon; tonnes and grades expected to be extracted according to MRB's resources evaluation of 2006, operating and mining costs as described in Genivar's Feasibility Study (2007) and by metallurgical tests and other technical reports available at the Elder mine site.

The following is the detailed presentation of the parameters used by Abcourt to determine the minimum cut-off grades for both the open pit and the underground operations;

<u>Silver Payment in the Zinc Concentrate</u>

For,

Grade of silver milled: 55.09 g/t Ag
Zn concentration: 19:1
Ag recovery at the mill: 80 %

The silver grade in the zinc concentrate should be:

55.09 x 19 x 0.8 = 837.37 g/t Ag or, 26.93 oz/t Ag

For,

3-year average Ag price: 27.55 $/oz.
Penalty: 25 %
Other fees: 100 g/t Ag

The silver payment in the zinc concentrate should be:

(((837.37 g/t-100)/31.1)*75%*27.55$)/19 = 25.78 $/t of ore or, 14.56$ per ounce of Ag.

Zinc Payment in the Zinc Concentrate

For,

Grade of zinc milled: 3.11%
Pounds of zinc /t: 68.42 lbs.
Zn recovery at the mill: 96 %
3-year average Zn price: 0.96 $/lbs.
Penalty: 15%

The zinc value in the zinc concentrate should be:

(68.42 lbs. x 96% x 0.96$/lb x 85%) = 53.60 $/t of ore

Minus,

Smelting costs [(1)] :
200 $/t / Zn concentration (19) = -10.53 $/t

Minus,

Transportation, custom fees and analysis:
60.50 $/t / Zn concentrate (19) = -3.18 $/t

The zinc payment in the zinc concentrate should be:

53.60 $ - (10.53 $ + 3.18 $) = 39.89 $/t of ore or, 12.83 $ for 1 % Zn.

Gold and Silver Payments in the Pyrite Concentrate

For,

Grade of silver milled:	1.77 oz/t of ore
Ag recovery by cyanidation:	9 %
Recoverable silver:	99 % (1% retained by the Royal Canadian Mint)
3-year average Ag price:	27.55 $/oz.

The silver payment by cyanidation should be:

(1.77 x 0.09 x 0.99 x 27.55 $) = 4.39 $/t of ore

For,

Grade of gold milled [(2)]:	0.0035 oz/t
Au recovery by cyanidation:	15 %
Recoverable gold:	97 %
3-year average Au price:	1 360 $/oz.

The gold payment by cyanidation should be:

(0.0035 x 0.15 x 0.97 x 1 360 $) =0.73 $/t of ore

So, the silver and gold payments in the pyrite concentrate should be:

(4.39 + 0.73) = 5.12 $/t of ore

From the above calculations, it is possible to determine what will be the silver and gold equivalence expressed in a percentage of zinc.

$$1 \text{ ounce Ag} = \frac{\text{Value of Ag in Zn conc.} + \text{Value of Ag+Au in Pyrite conc.}}{\text{Value of zinc in Zn concentrate}}$$

$$= \frac{14.56 + 5.12}{12.83} \qquad = \frac{19.68}{12.83} \qquad = 1.54\ \%\ \text{Zn}$$

Knowing that, it is now possible to evaluate the minimum cut-off grades for the open pit and the underground operations.

Minimum cut-off grade for the open pit operations

For,

Mining costs of ore[4]:	2.34 $/t
Mining cost of ore + waste[5]:	12.68 $/t
Milling cost:	10.48 $/t
General + Administration:	2.22 $/t
Royalties:	0.18 $/t
Total:	25.56 $/t

The minimum cut-off grade for the open pit operations should be:

25.56 $ / 12.83 $ = **1.99 % Zinc Equivalent**

Minimum cut-off grade for the underground operations

For,

Mining cost[7]:	45.08 $/t
Milling cost (from Genivar):	10.42 $/t
Administration (from Genivar):	2.22 $/t
Royalties:	0.18 $/t
Total[6] :	57.90 $/t

The minimum cut-off grade for the underground operations should be:

57.90 $ / 12.83 $ = **4.51 % Zinc Equivalent**

So, the minimum cut-off grades of **1.99 % Zn and 4.51 % Zn Equivalent** were respectively applied on open pit and underground operations for the current resources evaluation.

Notes: (1) Number based on a document presented by Teck-Cominco to investors, (2) Average from the mill recoveries of ore shipped by ABCOURT from 1985 to 1990 (3) Genivar 2007, p. 58 (4) Genivar 2007, p. 148 (5) Based on a waste/ore ratio of 4.42:1 (6) Based on a document presented by Industrielle-Alliance (7) Based on a 22% production from development drifts.

14.8 Summary of the Resources Evaluation

The measured and indicated resources for the Abcourt-Barvue deposit are totalling 8 086 000 tonnes grading 55.38 g/t Ag and 3.06 % Zn at a minimum cut-off grade of 1.99 and 4.51 % Zn(Eq) for the open pit and underground operations respectively. The measured resources are 6 264 200 tonnes grading 43.93 g/t Ag and 3.09 % Zn. The indicated resources are standing at 1 821 800 tonnes grading 94.75 g/t Ag and 2.98 % Zn (Table 14.3).

Table 14.3: Summary of the Measured and Indicated resources

Category	Tonnes	Ag (g/t)	Zn (%)
Measured	6 264 198	43,93	3,09
Indicated	1 821 799	94,75	2,98
M + I	8 085 998	55,38	3,06
Inferred	2 041 146	114,32	2,89

When compared to the 7 018 969 tonnes estimated in 2006, these numbers are in line with what should be expected from the use of a lower minimum cut-off grade (an increase in the mineral inventory) combined with the 2010-2011 in-fill diamond drilling program (an increase of the indicated resources to the expense of the inferred ones). The increase in the indicated resources is mainly explained by the large number of holes drilled on the Abcourt side between the -150 and -300m elevations where limited open pit operations where planned by Genivar (2007). Narrower but higher grade mineralized intersections located on either sides of a talc-rich shear zone namely, the Gray Schist horizon or the "GS", were intersected in this area.

Table 14.4: Comparisons of the Measured and Indicated resources from 2006 (gray) to 2014 (blue)

Category	Tonnes	Ag (g/t)	Zn (%)	Tonnes	Ag (g/t)	Zn (%)
Mesured	6 515 863	58,32	3,33	6 284 198	43,93	3,09
Indicated	503 106	98,35	3,44	1 821 799	94,75	2,98
M + I	7 018 969	61,19	3,33	8 085 998	55,38	3,06
Inferred	1 505 687	120,53	2,98	2 041 146	114,32	2,89

Other factors having contributed to increase the mineral inventory are:

- Some inferred blocks displayed on the longitudinal section were previously calculated using the vertical instead of the horizontal width of mineralized intersections. In these cases, the tonnage of some blocks were increased by up to 20-25%,
- A few holes drilled in 2010-2011 in the west end of the deposit have intersected significant zinc and silver values (AB11-69, 70).
- The contribution of new mineralization intersected in the footwall of the Gray Schist (AB11-18, 19, 20, 22, 23 and 25).

The measured resources represent 67% of the mineral inventory as compared to 33% for the indicated and inferred categories. The Author has classified most of the mineral resources in the measured category for the following reasons:

- **Tight drilling pattern**: The 15 to 30 m drill spacing used by ABCOURT is distributed quite uniformly along the strike length of the deposit. In all, 677 surface holes and 775 underground holes were drilled, logged, sampled and assayed for Ag and Zn,
- **Simple geometry of the deposit**: Apart from a north-south crosscutting fault system having displaced the mineralization over metric scale distances, the geometry of the deposit is predictable and well understood over a strike length of 2 km,
- **Presence of a "marker horizon"**: The low-grade Ag-Zn mineralization is located in a coarse-grained tuffaceous unit and it is limited to the vicinity of a regional-scale shear zone called the Gray Schist (GS),
- **Proven historical open pit mining**: Open pit production of 5 002 190 metric tonnes grading 38.74 g/t Ag and 2.98% Zn from 1952 to 1957 to a maximum depth of 75 metres right over the mineral resources drilled by ABCOURT (Barvue side),
- **Development work in ore**: Drifting of sub-levels in ore at 15 m intervals from a decline excavated between the 76 and the 152 m levels. In all, 5.4 km were driven in the mineralization from a total of 13.8 km of underground development,
- **Proven historical underground mining**: Underground production of 632 319 metric tonnes grading 131.65 g/t Ag and 5.04% Zn from 1985 to 1989 using the Avoca mining method,
- **Feasibility of a large open pit operation**: Genivar’s technical report (2007) stated that more than 80% of the Ag-Zn mineralization can be extracted by open pit mining to a maximum depth of 166 metres (Summary Section, page iii).

If the open pit outline designed in 2007 by Genivar remains unchanged, it is estimated that 77% of the measured and indicated mineral resources could be extracted from an open pit and 23% from underground operations.

Table 14.5: Distribution of the tonnes by bench in open pit (shaded green) and from underground (shaded blue).

ABCOURT ZONE					BARVUE ZONE				
BENCH NO.	MEASURED	INDICATED	INFERRED	SHAFT	BENCH NO.	MEASURED	INDICATED	INFERRED	**TOTAL**
BENCH 1	137 518	36 875	0		BENCH 1	145 672	17 283	6 494	343 842
BENCH 2	385 292	13 196	0		BENCH 2	287 694	8 371	26 518	721 071
BENCH 3	322 819	21 016	0		BENCH 3	336 696	52 091	5 141	737 763
BENCH 4	150 645	65 918	0		BENCH 4	467 669	31 642	11 235	727 109
BENCH 5	0	16 635	0		BENCH 5	770 415	18 617	14 645	820 312
					BENCH 6	750 352	31 923	309	782 584
					BENCH 7	768 981	30 734	3 367	803 082
					BENCH 8	622 670	28 904	9 643	661 218
					BENCH 9	409 841	12 733	2 906	425 480
					BENCH 10	246 639	0	0	246 639
	229 482					231 814			461 296
		1 124 388					311 475		1 435 863
			1 554 563					406 326	1 960 889
4 058 347	1 225 756	1 278 028	1 554 563		**6 068 799**	5 038 442	543 773	486 584	**10 127 146**

14.9 Marginal Mineralization

Some blocks are located within the main mineralization but, as they are grading less than the minimum cut-off, this mineralization is considered sub-economic. Although this marginal mineralization cannot be included in a classic mineral resources inventory, it remains that it is grading between 1.62 and 1.84% ZnEq. So, further studies should be carried out to evaluate the possibility to process them at the mill instead of stockpiling them and managing this potentially acid generating material.

The marginal mineralization was calculated in removing them from the global inventory as this low grade material is generally located between two layers of higher grade measured and indicated resources. The blocks which were classified in this sub-economic mineralization are identified with a red circled **M** (for **M**arginal) on sections.

Table 14.6: Marginal mineralization

Operation	Category	Tonnes	Ag	Zn	ZnEq
ABCOURT	MESURED	64 563	8,56	0,76	1,18
	INDICATED	6 369	9,97	0,99	1,48
	INFERRED	0	0,00	0,00	0,00
MARGINAL ABCOURT:		70 932	8,69	0,78	1,21
BARVUE	MESURED	263 185	10,92	1,18	1,72
	INDICATED	83 609	11,36	1,01	1,61
	INFERRED	69 408	12,94	1,04	1,68
MARGINAL BARVUE:		416 202	11,47	1,12	1,69
TOTAL MARGINAL :		487 134	11,06	1,07	1,62

15.0 MINERAL RESERVE ESTIMATE

Mines Abcourt Inc mandated PRB Mining Services to update the 2007 feasibility study prepared by Genivar dated February 15 2007 using the 2014 resource estimate and current economic parameters. The 2007 feasibility study prepared by Genivar is titled Technical Feasibility Study Report on the Abcourt-Barvue Deposit and is dated February 15, 2007. It is posted on the Sedar website. Sections 15, 16, 18, 19, and 21 are written in comparison to the 2007 feasibility report.

15.1 Overview

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 S49ºE direction). The dip is 75º to 90º to the north.

In the Barvue sector, the mineralization 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.

The mineral resource estimate was updated in 2014 by Jean-Pierre Bérubé, Consulting Geologist, as described in section 14. The 2014 resource estimate has significant change in the open pits, namely in the Barvue pit, but has insignificant change in the underground portion.

The 2014 in-pit resource was diluted and an open pit reserve was estimated. The underground reserve remained that of the 2007 feasibility. The 2007 and 2018 reserve estimates are discussed in this section and presented in Table 15.3.

15.2 Ressources

15.2.1 Feasibility Study 2007 Resource

The measured and indicated (M&I) resources estimated in 2006 by MRB & Associates and described in the 2007 feasibility study total 7,018,969 tonnes grading 3.33% Zn and 61.19 gpt Ag as presented in Table 15.1. The estimation of net smelter return (NSR) values assumed prices of 1.00US$/lb zinc, 10.00US$/Oz silver, and an American to Canadian currency exchange rate of 1.18 US$ per CA$ (0.85 CA$ per US$) as well as process parameters and smelting and associated costs.

The estimated NSR values were as follows:

- Abcourt open pit:
  The NSR for 1% zinc = 12.83 US$ and for 1 Oz/t silver = 8.29 US$.
- Barvue open pit:
  The NSR for 1% zinc = 12.86 US$ and for 1 Oz/t silver = 8.08 US$.
- Underground:
  The NSR for 1% zinc = 12.83 US$ and for 1 Oz/t silver = 8.29 US$.

The silver zinc equivalence was set at 0.65%Zn per Oz/t silver.

The cut-off grades in estimating resources were as follows:

- Abcourt open pit: 2.40% Zinc equivalent,
- Barvue open pit: 2.55% zinc equivalent,
- Underground: 3.20% zinc equivalent.

15.2.2 2014 Resource Estimate

In comparison, the 2014 measured and indicated (M&I) resource estimated by Jean-Pierre Bérubé, described in chapter 14, totals 8,085,998 tonnes grading 3.06% Zn and 55.38 gpt Ag.

The 2014 estimation of net smelter return (NSR) values assumed prices of 0.96 US$/lb zinc, 27.55 US$/Oz silver, and an American to Canadian currency exchange rate of 1.00 US$ per CA$, as well as process parameters and smelting and associated costs.

The 2014 estimated NSR values for open pits and underground were 12.83 US$ for 1% zinc and 19.69 US$ for 1 Oz/t silver. The silver zinc equivalence was calculated to 1.54% Zn per Oz/t silver.

The cut-off grades in estimating resources were as follows:

- Open pit: 1.99% zinc equivalent,
- Underground: 4.51% zinc equivalent.

Table 15.1: M&I Resource Estimate 2007 vs 2014 by Category

	2006 Resource Estimate			2014 Resource Estimate		
	Tonnage	Ag	Zn	Tonnage	Ag	Zn
Classification	(t)	(g/t)	(%)	(t)	(g/t)	(%)
Measured ressource	6,515,863	58.32	3.33	6,264,198	43.93	3.09
Indicated ressource	503,106	98.35	3.44	1,821,799	94.75	2.98
M&I ressources	7,018,969	61.19	3.33	8,085,998	55.38	3.06

In March 2018, M. Bérubé verified the cut-off grades with recent economic and metallurgical factors and reported that the 2014 resource estimate may be used for an update to the 2007 Genivar study.

The 2014 estimate has 1,067,031 tonnes more of measured and indicated resource than the 2006 estimate. As shown in Table 15.2, the difference is all in the open pits and the underground resource is virtually unchanged.

Table 15.2: M&I Resource Estimate 2007 vs 2014 by Sector

	2006	2014	Difference	
Sector	Total M&I	Total M&I	Total M&I	
Barvue Pit	4,116,978	5,038,927	921,949	22%
Abcourt Pit	1,009,151	1,149,914	140,763	14%
Underground	1,892,840	1,897,159	4,319	0%
Total	7,018,969	8,086,000	1,067,031	15%

15.3 Open Pit Model Preparation

Genivar describes the open pit model preparation as follows:

In 2007, the geological interpretation prepared by Abcourt, Gescad, and Innovexplo and updated by MRB on a 15-metre vertical section pattern was 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 zinc and silver 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 were be used during the open pit design process to avoid permanent pit wall location in this geological unit for stability reasons.

15.4 Underground Model Preparation

Genivar describes the underground model preparation as follows:

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.*

15.5 Reserve

15.5.1 2007 Open Pit Mineral Reserve Estimate

The 2007 open pit reserve was estimated as described in the following text in italic taken from the Genivar 2007 report.

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.

The resulting open pit mineral reserve estimate totaled 5,338,731 tonnes grading 3.15% zinc and 44.79 gpt silver.

15.5.2 2018 Open pit mineral reserve estimate

In estimating an open pit mineral reserve, the in-pit 2014 resource was similarly diluted as follows.

A dilution factor of 5% was applied to the Barvue open pit resource. The diluting material is composed of waste material grading 0% Zn and 9 g/t Ag.

A dilution factor of 10% was applied to the Barvue open pit resource. The diluting material is composed of waste material grading 0% Zn and 13 g/t Ag.

In both cases marginal and sub-marginal resource was not considered in the dilution grade.

The resulting open pit mineral reserve totals 6,589,361 tonnes grading 2.80% zinc and 39.93 gpt silver. The economic analysis hereafter will be applied to this open pit reserve.

15.5.3 Underground Mineral Reserve Estimate

Since there is no difference between the 2007 and the 2014 underground resource estimates, the underground mineral reserve of the 2007 technical feasibility report will be used for the analysis. The underground reserve includes a dilution factor of 20 % for development and 10 % in stopes, and a dilution grade of 0% Zn and 31.1 g/t Ag. The silver grade was estimated from drill hole intersections found on both sides of the ore in contiguous rock.

15.6 Mineral Reserve

15.6.1 Mineral Reserve Classification, Category, and Definition

Mineral Reserve

According to CIM Definitions Standards, a Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

Probable Mineral Reserve

A Probable Mineral Reserve is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

Proven Mineral Reserve

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

15.6.2 Mineral Reserve Estimate

The mineral reserve estimate including dilution is presented in Table 15.3. The 2007 mineral reserve is also presented in table 15.3 for comparison.

Table 15.3: Mineral Reserve

		2007 Estimate				2018 Estimate			
		Tonnage	Grade			Tonnage	Grade		
Mining Method	Classification		Ag	Zn	Zn EQ		Ag	Zn	Zn EQ
		(t)	(g/t)	(%)	(%)	(t)	(g/t)	(%)	(%)
Open Pit	Proven Mineral Reserves	5,338,731	44.79	3.15	3.15	6,180,510	39.72	2.83	3.61
	Probable Mineral Reserves	0	0.00	0.00	0.00	408,851	43.01	2.36	3.20
	Total Open Pit	5,338,731	44.79	3.15	3.15	6,589,361	39.93	2.80	3.58
Underground	Proven Mineral Reserves	1,169,662	105.19	2.87	2.87	1,169,662	105.19	2.87	4.93
	Probable Mineral Reserves	315,139	101.61	3.23	3.23	315,139	101.61	3.23	5.22
	Total Underground	1,484,801	104.43	2.95	2.95	1,484,801	104.43	2.95	4.99
Open Pit and Underground	Proven Mineral Reserves	6,508,393	55.64	3.10	3.10	7,350,172	50.14	2.84	3.82
	Probable Mineral Reserves	315,139	101.61	3.23	3.23	723,990	68.52	2.74	4.08
	Total	**6,823,532**	57.76	3.11	3.11	**8,074,162**	51.79	2.83	3.84

Notes: 1) Zn Eq grades are calculated with 2018 parameters for this table.
2) Silver zinc equivalence : 0.61 %Zn / 1 Oz Ag

15.6.3 Marginal Ore

Marginal ore is mineralized material excavated from the open pits in the course of accessing the mineral reserve but with a value below the cut-off grade. It is not included in the mineral reserve estimation. It is stockpiled separately from waste rock as it may potentially be processed at a later date when prices are higher or at the end of the life of mine when operating costs are significantly less than during open pit excavation. The marginal ore for the Abcourt-Barvue pits is presented in table 15.4.

Table 15.4: Marginal Ore

		2007 Estimate				2018 Estimate			
		Tonnage	Grade			Tonnage	Grade		
	Classification		Ag	Zn	Zn EQ		Ag	Zn	Zn EQ
		(t)	(g/t)	(%)	(%)	(t)	(g/t)	(%)	(%)
Open Pit Marginal Ore	Proven Marginal Ore	1,151,502	17.65	1.58	1.58	303,463	10.64	1.09	1.30
	Probable Marginal Ore	0	0.00	0.00	0.00	82,262	11.48	1.00	1.22
	Total Marginal Ore	1,151,502	17.65	1.58	1.58	385,725	10.82	1.07	1.28

Notes: 1) Marginal ore is mineralized material with a value below the cut-off grade. It is not included in the mineral reserve estimation.

16.0 MINING METHODS

This section is written in comparison with Genivar's technical feasibility study dated February 15, 2007.

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 S49ºE direction). The dip is 75º to 90º to the north.

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

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 will proceed underground under the pit using the Avoca cut-and-fill method to a depth of 150m to 200m with trackless equipment. Three declines will access the underground operation; the west, the central, and the east decline.

16.1 Open Pit Mining

Open pit mining will be carried out using conventional truck and shovel method. Drilling will be performed by in-the-hole production drills. Blasting operations will use an emulsion ANFO (ammonium-nitrate fuel oil) explosive and a non-electric delay initiation system. An hydraulic shovel and a front-end loader will be used to load rigid haulage trucks.

For most of the open pit tonnages, haulage trucks will exit from Abcourt and Barvue pits onto a common haul road approximately 500m west of the mill site. Waste rock will be hauled to three (3) different storage areas. The West waste pad and the East waste pad are located approximately 1.7 km from the East Abcourt and the Barvue pit exits.

Over the first 5 years, open pit mining will occur at a maximum annual production rate of 6 Mt of rock per year of which 650 000 tonnes will be ore and the remaining tonnes will be waste rock and marginal ore. More details are given on the following pages.

16.1.1 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.

a) *"The objective 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."

b) *"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 60o slope and more. Similarly, the excavation of the Abcourt pit at a slope of 60o 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 60º wall slopes were used by GENIVAR in the design of the Abcourt and Barvue open pits. The Genivar pit design is used in the economic update.

16.1.2 <u>Pit Water Inflow</u>

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 per day. According to the Genivar report, water inflow rates in the Abcourt pits will probably account for an additional 50 imperial gallons per minute or 330 m^3 per day.

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

16.1.3 Open Pit Design

This section is taken from the Genivar report and is shown in italic font. It describes the open pit design of 2007.

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 16.1*. A typical cross-section of the Barvue open pit is illustrated in* Figure 16.2*.*

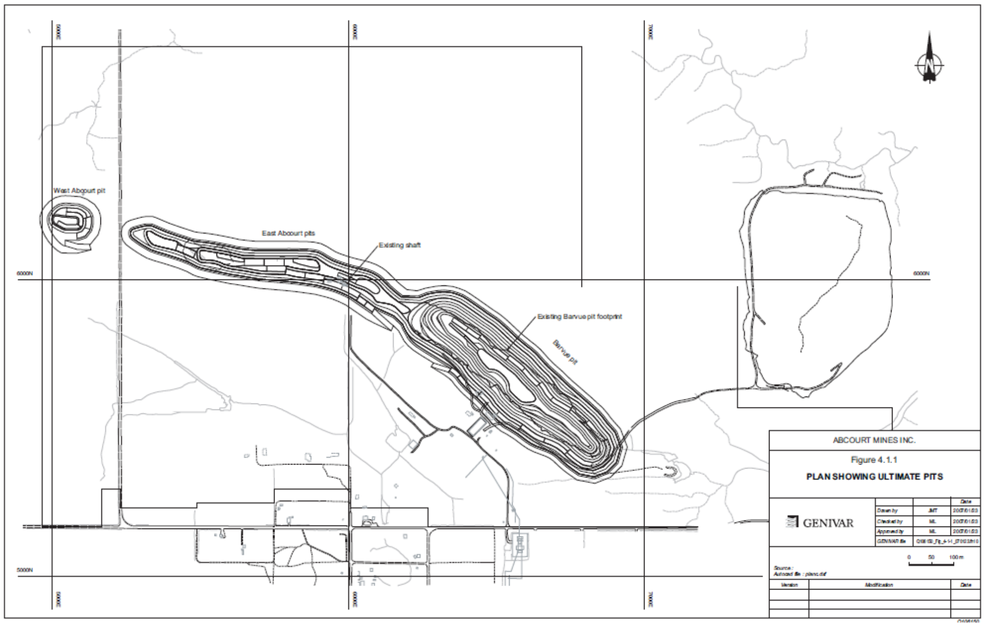

Figure 16.1: Plan showing ultimate pits.

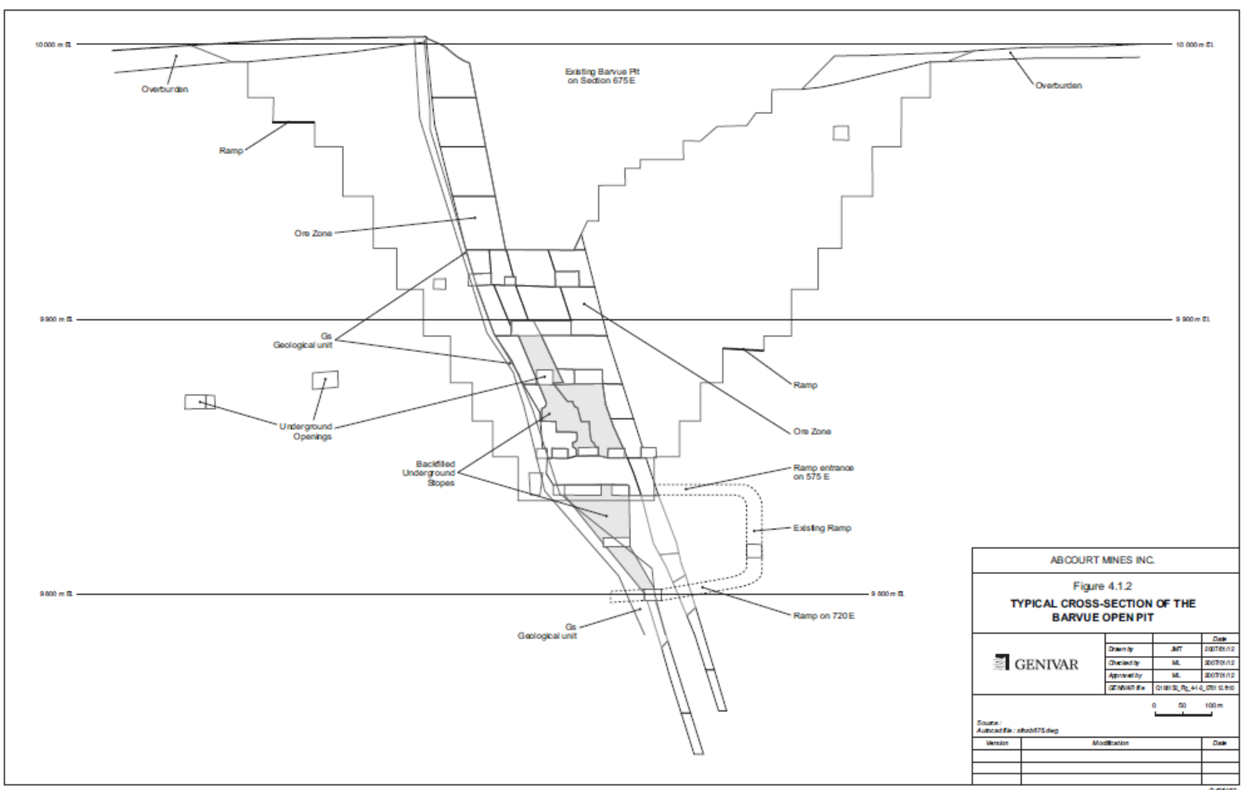

Figure 16.2: Typical cross-section of the Barvue open pit.

16.2 Underground Mining

16.2.1 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.

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.

16.2.2 Avoca Cut-And-Fill method

The underground resource of the Abcourt-Barvue deposit is found generally in the hanging wall of a marker tuff where ground conditions are good. The Avoca cut-and-fill method will be used for the underground 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 ore zone with sub-levels developed at 15 to 20 metre vertical intervals. All mining will be trackless.

Jumbos will be used for lateral development. The stopes will be mucked out with remote controlled LHD. Abcourt Mines owns several 8 yard^3 and 5 yard^3 scooptrams and one 4 yd^3 scooptram. The 8 yard^3 and 5 yard^3 scoops will be used for mucking purposes while the 4 yard^3 scoop will be used for servicing. The stopes will be drilled with pneumatic long hole machines. The main declines will be driven with a minus 12% to 15% slope.

Blasting operations will use ANFO explosives. A gel cartridge and nonel detonation system will be used for both development and production purposes. Trucks with 42 tonne loads will tram the ore to the surface.

Fresh air for underground mine ventilation purposes will be pushed through ventilation raises. Mine air will exhaust by the ramps. A manway will be installed in the ventilation raises and serve as a second egress.

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 on each sub-level. The 1.5-m x 1.5-m slot raises will be drilled with a long hole machine.

Stopes will be backfilled with waste rock from mine development and the open pits. During underground production, trucks will return with a load of waste rock to be used as rockfill if needed. Where possible, development waste rock will be hauled directly to a stope being filled.

Figure 16.3 is a schematic view of the Avoca mining method with summary description of the steps of a production cycle. Concurrently with the production cycle, development crews will be driving additional sub-levels above the top sill. The longitudinal view of the underground mine and the schematic view of the avocat cut-and-fill method are illustrated in Figure 16.4 and Figure 16.2.

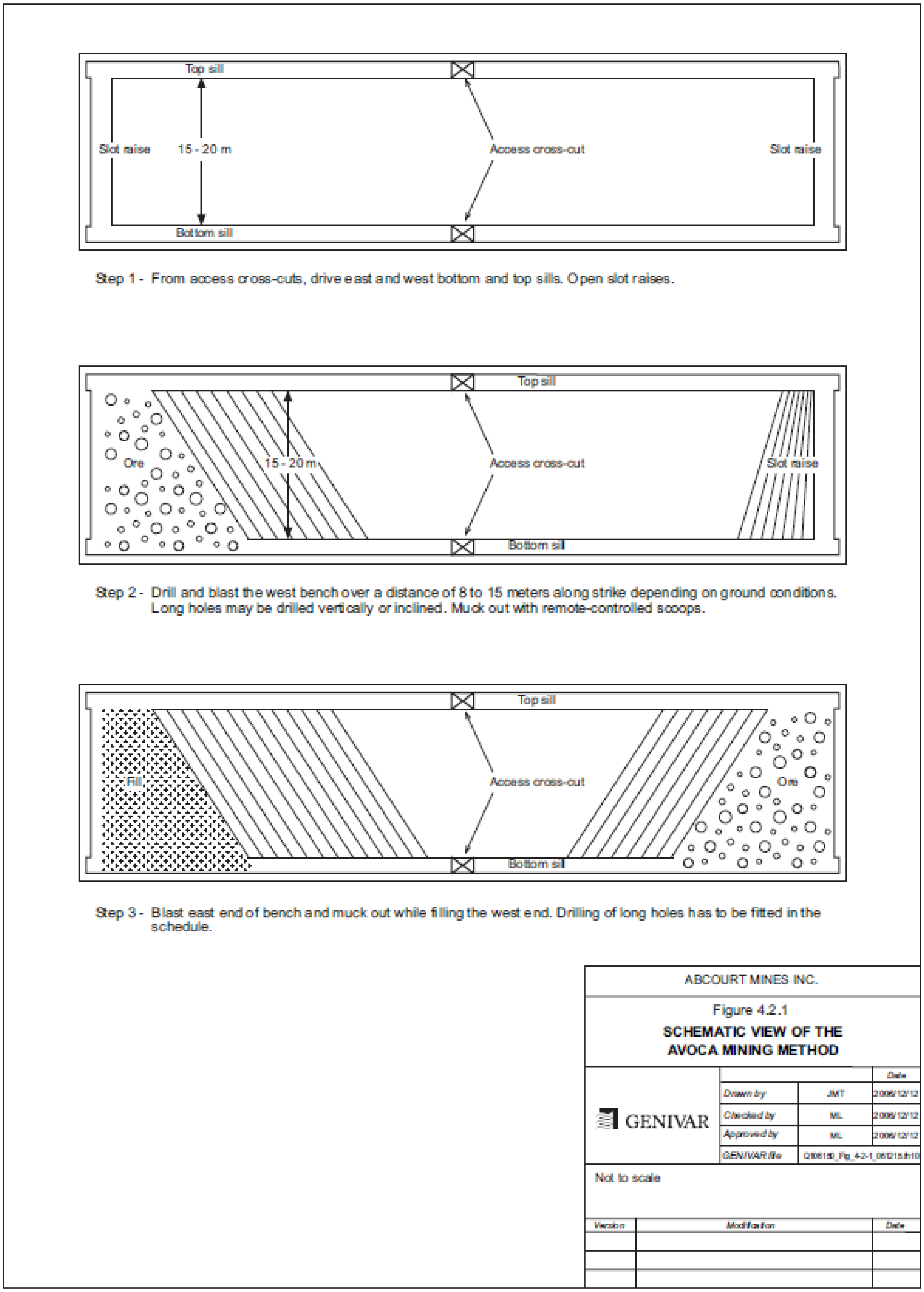

Figure 16.3: Schematic view of the avoca mining method

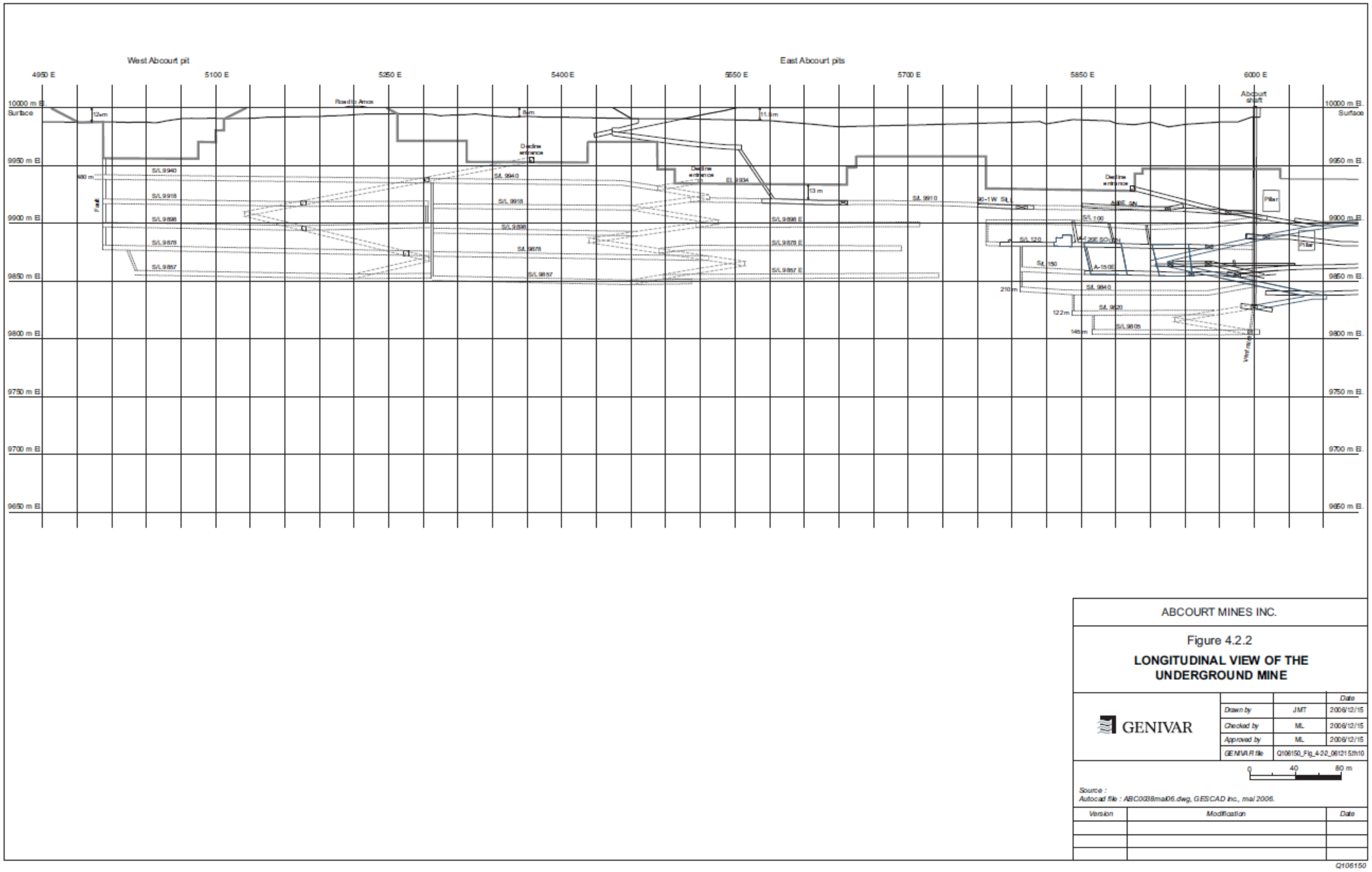

Figure 16.4: Longitudinal view of the underground mine

16.2.3 Underground Mining Proposal for the Abcourt-Barvue Deposit

This section describes how Genivar developed the underground mining program.

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 planned ore tonnage is located west of the limit separating Barvue from Abcourt.

Mining will progress from the bottom up. The west and central stopes will start at a depth of about 150 metres and finish in the pit bottom at a depth of about 55 metres. The east stope will start at a depth of about 200 metres and will be completed at a depth of about 65 metres, also in pit bottom. Each stope will typically 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.

16.3 Mine Production Schedule

The 2007 mine plan is discussed here in italic font.

> *The 2007 mine plan milled a total of 6,446,000 tonnes over a period of 10 years as shown in* Table 16.1*; that is 5,338,731 tonnes from open pit and 1,107,269 tonnes from underground. At the end of the 10 years, 377,532 tonnes of ore were left underground untouched to be mined in subsequent underground operation.*
>
> *At first, production came from open pits. Underground production started supplementing mill feed on year 6 as open pit production was reduced. In total, 83 % of production came from open pits and 17 % came from underground stopes. Haulage trucks from both the open pit and underground mine dumped directly to the ore mixing area near the crusher.*
>
> *The average waste and marginal ore to ore ratio for the open pits was 6.4:1. The mining of 3.87 Mt of waste and marginal ore and 128 000 t of ore was scheduled as preproduction. The average waste to ore ratio for the 10 year production period was 5.7:1. The preproduction ore was stockpiled apart on the marginal ore stockpile area to be reclaimed during the first two years of production.*

Table 16.1: Genivar 2007 Production Schedule

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

a) [1] Including waste rock (33.15 Mt) and marginal ore (1.15 Mt).
b) Silver zinc equivalence: 1 Oz Ag = 0.65% Zn.

The updated reserve estimate totals 8,074,162 tonnes, 6,589,361 tonnes of which are in open pits and 1,485,201 tonnes are underground. Under the updated mine plan, the milling rate will remain the same at 650,000 tonnes per year and production will span over a 13 year period as shown on Table 16.2.

Under the updated plan, production will begin in the West Abcourt and Barvue pits. The West Abcourt pit will be completed during year -1. The East Abcourt pit production will begin in year 1 and will be completed in year 5. The Barvue pit production will be completed during year 13. This sequence was selected to have better grade ore in the early years of production as the West Abcourt pit has a better grade than the East Abcourt pit which has better grades than the Barvue pit. Underground production will supplement mill feed from year 7 to year 13. In total, 82 % of production will be from open pits and 18 % will be from underground operations. At the end of year 13 all reserve will be mined out. Ore will be mixed on the ore pad when necessary.

The average waste and marginal ore to ore ratio for the open pits 5.0:1. Preproduction excavation is reduced to minimize capital expenditures. Preproduction excavations will prepare the Barvue pit for production and will stockpile two months of mill feed for mill start-up. It is assumed that the lower grade Barvue ore will be utilized for mill commissioning until target recoveries and concentrate quality are attained. Preproduction excavation will consist of 436,000 m^3 of overburden, 903,902 tonnes of waste, 17,360 tonnes of marginal ore, and 122,164 tonnes of ore. Of this, 886,562 tonnes of waste, 11,360 tonnes of marginal ore, and 118,164 tonnes ore will be hauled out of the pits. The preproduction waste will be utilized to upgrade roads and construct the tailings storage facility. A detailed production schedule is presented in appendix 3.

Table 16.2: Updated Production Schedule (2018)

Description	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Open Pit Production									
Overburden									
Total overburden	m3	436,000	1,234,000	0	250,000	0	0	0	0
Rock Mined									
Waste Mined	tonnes	903,902	4,703,826	4,296,654	4,780,270	5,341,322	3,956,472	3,242,733	2,162,125
Marginal Ore	tonnes	17,360	72,056	73,101	55,731	48,450	37,159	23,165	18,226
Total Waste & Marginal Ore	tonnes	921,261	4,775,883	4,369,755	4,836,001	5,389,772	3,993,631	3,265,898	2,180,351
Ore Mined	tonnes	122,164	510,894	660,421	583,860	650,000	649,983	633,038	630,000
Total Rock Mined	tonnes	1,043,426	5,286,776	5,030,176	5,419,860	6,039,772	4,643,615	3,898,936	2,810,351
Waste to Ore Ratio		7.5	9.3	6.6	8.3	8.3	6.1	5.2	3.5
Rock Hauled Out of Pits									
Waste Hauled	tonnes	886,562	4,623,811	4,222,976	4,947,664	5,341,322	3,956,472	3,242,733	2,162,125
Marginal Ore Hauled	tonnes	11,360	58,806	64,851	51,231	48,450	37,159	23,165	18,226
Total Waste & Marginal Ore Hau	tonnes	897,922	4,682,617	4,287,827	4,998,894	5,389,772	3,993,631	3,265,898	2,180,351
Ore Hauled	tonnes	118,164	498,394	646,421	650,000	650,000	649,983	633,038	630,000
Total Rock Hauled	tonnes	1,016,086	5,181,010	4,934,248	5,648,894	6,039,772	4,643,615	3,898,936	2,810,351
Ore Stockpile									
To Ore Stockpile	tonnes	118,164	0	0	0	0	0	0	0
From Ore Stockpile	tonnes	0	97,606	3,579	0	0	17	16,962	0
Ore Stockpile Inventory End	tonnes	118,164	20,558	16,979	16,979	16,979	16,962	0	0
Underground production									
Development	tonnes	0	0	0	0	0	0	0	20,000
Production	tonnes	0	0	0	0	0	0	0	0
Total underground	tonnes	0	0	0	0	0	0	0	20,000
Mill Production									
Ore Milled	tonnes	0	596,000	650,000	650,000	650,000	650,000	650,000	650,000
Zinc Grade	% Zn	0.00	2.53	2.49	2.45	2.60	2.94	2.98	2.98
Silver Grade	gpt Ag	0.00	61.84	51.60	48.08	49.07	32.80	32.06	32.74
Zinc Equivalent	% Zn Eq	0.00	3.74	3.51	3.39	3.57	3.59	3.61	3.62

Continued

Description	Unit	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Total	Preprod'n	Prod'n
Open Pit Production										
Overburden										
Total overburden	m3	0	0	0	0	0	0	1,920,000	436,000	1,484,000
Rock Mined										
Waste Mined	tonnes	1,079,274	668,666	458,977	445,305	465,271	160,678	32,665,476	903,902	31,761,575
Marginal Ore	tonnes	17,777	12,479	14,164	14,020	14,037	0	417,725	17,360	400,365
Total Waste & Marginal Ore	tonnes	1,097,051	681,145	473,142	459,325	479,308	160,678	33,083,201	921,261	32,161,940
Ore Mined	tonnes	534,731	326,000	326,000	326,000	385,600	215,030	6,553,721	122,164	6,431,556
Total Rock Mined	tonnes	1,631,782	1,007,145	799,142	785,325	864,908	375,708	39,636,922	1,043,426	38,593,496
Waste to Ore Ratio		2.1	2.1	1.5	1.4	1.2	0.7	5.0	7.5	5.0
Rock Hauled Out of Pits										
Waste Hauled	tonnes	1,079,274	668,666	458,977	445,305	465,271	160,678	32,661,836	886,562	31,775,274
Marginal Ore Hauled	tonnes	17,777	12,479	14,164	14,020	14,037	0	385,725	11,360	374,365
Total Waste & Marginal Ore Hau	tonnes	1,097,051	681,145	473,142	459,325	479,308	160,678	33,047,561	897,922	32,149,640
Ore Hauled	tonnes	534,731	326,000	326,000	326,000	385,600	215,030	6,589,361	118,164	6,471,196
Total Rock Hauled	tonnes	1,631,782	1,007,145	799,142	785,325	864,908	375,708	39,636,922	1,016,086	38,620,836
Ore Stockpile										
To Ore Stockpile	tonnes	0	0	0	0	0	400	118,564	118,164	400
From Ore Stockpile	tonnes	0	0	0	0	0	0	118,165	0	118,165
Ore Stockpile Inventory End	tonnes	0	0	0	0	0	400			
Underground production										
Development	tonnes	60,000	60,000	60,000	60,000	400	0	260,400	0	260,400
Production	tonnes	55,269	264,000	264,000	264,000	264,000	113,532	1,224,801	0	1,224,801
Total underground	tonnes	115,269	324,000	324,000	324,000	264,400	113,532	1,485,201	0	1,485,201
Mill Production										
Ore Milled	tonnes	650,000	650,000	650,000	650,000	650,000	328,162	8,074,162	0	8,074,162
Zinc Grade	% Zn	3.03	2.98	2.95	2.95	2.97	3.03	2.83	0.00	2.83
Silver Grade	gpt Ag	43.86	67.96	68.25	67.49	60.36	64.13	51.79	0.00	51.79
Zinc Equivalent	% Zn Eq	3.89	4.31	4.29	4.27	4.16	4.29	3.85	0.00	3.85

Notes: 1) Ore hauled is greater than ore mined because ore from some ore of Barvue upper benches fall to existing pit bottom and is diluted with marginal ore and waste.
2) Silver zinc equivalence : 1 Oz Ag = 0.61% Zn.

16.4 Mining Operations

16.4.1 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.

16.4.1.1 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. Waste will be blasted in full benches 13 to 18 metres high. Ore will be blasted in half-benches 6 to 9 metres high. Drilling in waste will consist of 165mm (6 ½") diameter holes on a 5.5m x 5.5m pattern. Drilling in ore will consist of 114mm (4 ½") holes on a 3.5m x 3.5m pattern. Perimeter drilling on the final walls for pre-shearing will be 114mm diameter holes spaced 1.6m. Drilling will be with conventional in-the-hole hammer rigs. Two drill rigs will be necessary.

Production holes will be loaded with bulk emulsion explosives delivered to site by a UMF truck. Perimeter holes will be loaded with 38mm diameter pre-splitting cartridge explosive. Holes will be blasted using non-electric type detonators.

The design powder factor will be 0.269 kg/tonne in waste and 0.277 kg/tonne in ore. Pre-shearing holes will have 1.0 kg of explosives per meter drilled. After mining commences, fragmentation will be evaluated and the drill and blast parameters may be refined to optimize results.

An explosives storage facility will be installed on the mine site. It will consist of two 20,000 kg ISO containers, a UMF truck garage and maintenance shop, and storage magazines for packaged explosives and detonators. This facility will be supplied and installed by the supplier.

16.4.1.2 Load and Haul

The waste and the ore will be loaded as separate operations. The main loading equipment will be two 3.5 m^3 hydraulic shovels in ore and waste. A 6.8-m^3 front-end loader will handle ore mixing and transfer to the crusher.

Genivar's 2007 mine plan had a fleet of up to five 62 tonne capacity trucks to haul ore and waste. Current 62 tonne truck models are wider than previous models and are not amenable to the ramp width planned for the Barvue pit. Instead 50 tonne capacity trucks will be used. A fleet of five trucks will be acquired. Year 1 and year 4 will require 7 trucks. The two extra trucks will be hired externally. At the time of procurement, used trucks of higher capacity will be considered to increase productivity and reduce costs.

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 place the ore from the individual piles in a mixing stockpile and will keep occasional boulders apart for later re-blasting in a pit. 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.

16.4.1.3 Support Equipment

The production equipment will be supported by one bulldozer (240 kW), one grader (150 kW), one hydraulic shovel (2-m^3 equivalent) mounted with either bucket or a scaling bar (pit wall securing), one front end loader and one water truck for dust suppression. Abcourt may choose to have the mine support services provided by a contractor.

16.4.1.4 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.

16.4.2 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 their work places underground. All equipment for underground production will be diesel powered or have electric batteries.

16.4.2.1 Drilling and Blasting

Drilling and blasting will be required for development and production work. The waste will be produced from development of declines, access cross-cuts, etc. Ore will be produced from sub-level development and production stopes.

Abcourt Mines owns three pneumatic jumbos which will be used for the lateral development.

Abcourt Mines owns two pneumatic long hole drills which will be used for production drilling. Production drilling will consist of 76 mm (3 inch) diameter holes on a 1.8m x 1.8m pattern. Holes will be added to the stope walls to limit dilution.

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

Explosives for underground operations will be stored in the open pit magazines on surface.

16.4.2.2 Load and Haul

Development waste rock and ore will be mucked with 5 yard^3 and 8 yard^3 LHD and hauled to surface with 42 tonne trucks. Waste rock will be hauled to a rockfill transfer point near a stope to be backfilled or to a surface stockpile. Ore will be hauled directly to the surface ore pad.

Avoca stopes will be muck out with remote controlled 8 and 5 yard^3 LHD. Ore will be hauled directly to the mill with 42 tonne trucks. Trucks will be loaded at the sub-level access cross-cuts. From the stope, ore will be dumped directly in the truck or in a remuck station.

The stopes will be backfilled with un-cemented rock fill. During underground production, trucks will return with a load of waste rock to be used as backfill when needed.

16.4.2.3 Ventilation Raises

Ventilation raises will be excavated from mine bottom to surface or pit bottom with a mechanical raise climber and hand held pneumatic drills (stoppers & jacklegs). Their dimensions will be 3.0m x 3.0m. Ground support will be installed all four walls after the raise excavation is completed. A manway will be installed with steel grating landings on a timber frame every 7 meters of raise and timber ladders.

16.4.2.4 Grade Control

Grade control will be performed by a geological technician. Each ore drift round face will be sampled and assayed and a test hole will be drilled in alternating walls. Based on the assay results and a visual inspection, the technician will mark the ore limits in the face. This information and diamond drilling results will be used to design stope limits. Muck samples will be taken by the muckers.

16.4.2.5 Ground Support

Ground support in declines, cross-cuts, remucks, etc will generally consist of 2.1m long rock bolts in the back on a 1.2m x 1.2m pattern, 1.5m long rock bolts on a 1.2m x 1.2m pattern in the walls, and welded wire mesh no.6 in the back and walls down to 1.5m from the floor. In fractured ground, resined rebars will be used instead of rock bolts.

Ground support in the ore drifts will generally consist of 2.1m long rock bolts on a 1.2m x 1.2m pattern in the back, 1.5m long rock bolts on a 1.2m x 1.2m pattern in the walls, and welded wire mesh no.6 in the back and walls down to 1.5m from the floor.

Ground support in the escape ways will consist of 1.5m long rock bolts on a 1.2m x 1.2m pattern and galvanized no.9 chain link screen in the walls.

16.4.2.6 Ventilation

Development of each stope sector (west, central, and east) will require 75,000 cfm of fresh air. Development and production operations together will require 160,000 cfm of fresh air. Each stope sector will be separate and have its own ventilation system. The west and central stopes will be the first to be developed and mined. As mining of the west stope diminishes, mining will gradually transfer to the east sector.

The development work of a stope level will consist of the excavation of the decline and sub-level accesses to the mine bottom followed by the excavation of the bottom sub-level and the ventilation raise. When the ventilation raise is completed, the mine air ventilation system will be installed over it.

The ventilation system for the excavation of the decline, lower sub-levels, and the ventilation raise will consist of a 150 hp fan with a capacity of 80,000 cfm and total pressure of 10"H_2O. Mine air will be heated with a 6 MBTU capacity propane heater. Propane will be supplied from a 38,000 litre reservoir. On 54" diameter plastic fan pipe will be installed in the decline to bottom. Plastic fan pipes have less friction and air loss, and are more resistant than steel or fiberglass ventilation tubing. When the permanent ventilation system is in place, this fan pipe will be dismantled and may be used elsewhere.

The sublevels will be ventilated using smaller fans capturing mine air flowing up the ramp, and 30" diameter flexible vinolon fan pipe in the ore drifts.

The ventilation system installed at the ventilation raise collar for production and development operations will consist of two 150 hp fan and 6 MBTU heating units working in parallel. The total capacity will be 160,000 cfm and 12 MBTU. The heating systems modulate heat output according to outside temperature and the set downstream temperature. The downstream temperature will be set to 5ºC.

When the development in the west stope is completed, one 150 hp fan and 6MBTU heater units will be transferred to the east stope to initiate development in that sector. In all, four 150 hp fan and 6 MBTU heating units will be required. The decline will remain the mine air exhaust.

16.4.2.7 Dewatering

Each stope sector will have its own dewatering circuit. The main pumping station will be located near ramp bottom. It will consist of a 75 hp centrigugal pump installed in a skid mounted 2000 gallon reservoir with water level controls and starter. It will pump the mine water directly to surface.

Mud in suspension will be handled by a 7 foot diameter separation cone system like the Mudwizard. It will be installed in a bay near the 2000 gallon reservoir/pump unit. Mine water will be directed to the ramp bottom. A 15 hp submersible pump will pump water from the ramp bottom through the separation cone system and on to the skid mounted reservoir/pump unit. The mud collected in the separation cone will flow to a mud bay excavated nearby and retained by a muck filtering wall and allowed to drain. The mud will periodically be mucked out and hauled to surface.

The pump line will consist of 4" diameter steel pipes with grooved ends.

16.4.2.8 Compressed Air

Abcourt Mines owns two 1700 cfm air compressors with a glycol heat exchange system. They will be installed in a building and located near the central stope decline portal. Compressed air will be distributed to the west, central, and east stopes from there. The compressed air line will consist of a 6” diameter steel pipe in the declines and a 4” diameter steel pipe on the sub-levels.

16.4.2.9 Industrial Water

Abcourt Mines owns a 10,000 gallon tank which will be used to store industrial water for the underground operations. The industrial water will be distributed underground through a 4” diameter steel pipe in the declines and a 2” diameter steel pipe on the sub-levels. The water will come from the mine exhaust.

16.4.2.10 Electrical Distribution

The electrical capacity requirement of each stope sector is about 1.8 MW. A 25 kV electrical line will be installed from the main electrical station near the mill to the top of the East Abcourt pit above the central stope ramp. A 25 kV to 5 kV transformer will be installed at the end of this power line. From there a 5 kV electrical line will be installed down the pit wall to the west, central, and east ramp portals.

A portable sub-station will be installed at each ramp portal to provide power to the top sub-levels. Two other portable sub-stations will be installed underground in each stope sector to supply power to the mid and lower sub-levels. The sub-stations will be supplied with power at 4160 volts with 5 kV cables installed in the declines.

A blasting line will be installed to all work places underground. Blasting will be controlled at the mine dry using a tag system. Radio communication with the surface network will be provided underground with a leaky feeder system installed in the declines to the sub-levels.

16.5 Mine Support Facilities

16.5.1 Logistical Support

Pickup trucks will be used by engineering, geology and management personnel for travelling on site. Pick-up trucks and personnel carriers will transport personnel to their work sites. Tractors will be fitted to travel underground. Ambulance services are available such that an ambulance is not required on site.

16.5.2 Dewatering

The existing Barvue pit will be dewatered during the preproduction period. Dewatering will be performed with four 140 HP submersible pumps installed on a frame and lowered into the pit along with pipes. According to Abcourt, water tests indicate the water quality satisfies the regulations. Water will initially be pumped to a stream to the east of the pit until tests show it needs treatment at which time it will be pumped to the water treatment plant. Electrical power will be provided from the local power grid.

Each pit will have submersible pump installed on the first bench to intercept water from surface. Water from ground seepage and rainfall will be stage-pumped from pit bottom to surface. A 74 HP diesel pump setup on a trailer for easy transfer will be installed at the pit bottom. An intermediate 60 HP submersible pump will be installed in a reservoir on a bench of the final pit wall to transfer water from pit bottom to surface.

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

16.5.2.1 Mine Dry

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

17.0PROCESS DESIGN

17.1 Design Criteria

The design criteria used for development of the mill flowsheet was based on the latest results of the test AB-64 performed at URSTM laboratory (See Appendix 4) and the other tests information available and also BUMIGEME's in-house experience for similar operations. Process flowsheets are provided in Appendix 5.

17.1.1 General

Annual ore production	650,000	t
Mill head grade	2.83%	Zn
	51.79 g/t	Ag
Recovery of zinc	97.5%	Zn
Recovery of silver	77.8%	Ag
Silver content in Zn-Ag concentrate	740.6g/tm	Ag
%Zinc in Zn-Ag concentrate	53.4%	Zn
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	2.91	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	53.4%	Zn
Zn concentrate weight recovery	5.68	%

17.1.2 Coarse Ore Storage

Storage crushed stockpile	1000	t
Coarse ore storage bin live capacity	600	t
Coarse ore storage bin capacity (in treatment hours)	8	h
Ore moisture	3.0	%

17.1.3 Grinding

Fineness of grinding (K_{80})	64	μm
Fineness of grinding (% passing 74 μm)	86.6	%
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})	64	µm
Ball mill energy consumption	12.96	kWh/t
Ball mill circulating load	300	%
Ball mill discharge pulp solids density	75	%

17.1.4 Flotation Circuits

Zinc flotation times-kinetic test AB-64 (4 cycles)

Conditioning time	11.6	min.
Rougher	16.6	min.
Scavenger	17.8	min.
1st Cleaner	24.8	min.

Pyrite flotation times (lab locked cycle test x 2)

Conditioning time	25	min.
Rougher flotation time	16.9	min.
Scavenger flotation time	11.4	min

Flotation pH

Zinc rougher/scavenger circuit	10.8	--
Zinc cleaner circuit	11.8	--
Pyrite flotation	6.0	--

Flotation pulp solids density

In Zn rougher and scavenger circuit	34	%
In Zn 1st Cleaner	27 – 29	%
In Pyrite rougher circuit	28	%

17.1.5 Zinc Concentrate Thickening

Unit area	0.4	m²/mt/d
Feed pulp solids density	14	%
Underflow pulp solids density	65	%

17.1.6 Zinc Concentrate Filtration

Filter type	pressure	--
Filtration rate	n/a	kg/m²/h
Filter cake moisture content	8.0	%

17.1.7 Zinc Storage Capacity

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

17.1.8 Chemical Reagent Consumption

Sulfuric acid	Pyrite flotation	4.4 kg/t
Copper sulfate	Zinc and pyrite flotation	0.395 kg/t
KAX	Pyrite flotation	0.018 kg/t
Percol 338	Primary and Zinc concentrate thickeners	0.025 kg/t
A-208	Zinc flotation	0.05kg/t
F-507	Zinc and Pyrite flotation	0.118kg/t

17.2 Process Description

17.2.1 General

The main process steps for treating the Abcourt and Barvue ores are primary crushing and stockpiling, ore bin, SAG/ball grinding, zinc flotation, 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.

17.2.2 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 750 mm (30”) square openings grizzly with the undersize dropping into a lined 75 t hopper. Mixing the ore will regulate the grade fed to the processing plant. The ore is extracted from the feed hopper by a pan vibrating feeder.

The crusher size was selected based on its ability to handle the expected rock size as determined by the grizzly openings (750 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 (36”) 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 cone stockpile. From there a 914 mm (36”) belt conveyor carries the ore from the crushing area to the coarse ore storage bin.

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

A 600 t coarse ore bin will provide a buffer of 8 hours of plant operation between the crusher and feed to the SAG mill. The 6.8 m diameter by 10m 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.

17.2.3 Grinding

The ore is withdrawn from the coarse ore storage bin by two 1,000 mm x 5,000 mm (3’4”x16’8”) apron feeders. The feeder will discharge on a 914 mm (36”) belt conveyor carrying the ore to a semi-autogenous (SAG) mill 4.0 m X 5.484 m long driven by a 1450 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 in a pump box equipped with two 203 mm X 203 mm (8”x8”) AC variable control pumps into a 20” cyclone (two installed, one in operation). The underflow is recycled in the SAG and the overflow is sent to a secondary pump box equipped with two 203 mm X 254 mm (8”x10”) variable control pumps that feeds a 10 cyclones-pack. The 8 cyclones (in operation) 250 mm (10”) operated in close circuit with the two Ball Mills 3.048 m X 3.66 m long, driven by 590 kW electric motors. The underflow of these cyclones are recycled in the mills and the overflow pass on a Delkor linear screen 1500 mm X 2750 mm (5’x9’) to retain trash. The undersize of the screen is pumped by a 203 mm X 254 mm pump to the flotation conditioner tank, the over size (Trash) is pumped to the tailing pump box with a integrated pump). A Jib crane will be provided to facilitate SAG mill liner replacement. An overhead crane 35/5 tonnes 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.

17.2.4 Zinc Flotation

Based on the required conditioning time in an active volume of 35 m^3 conditioner, the pulp is fed to a 50 m^3 rougher cell. The rougher tails are transferred to a 50 m^3 scavenger cell through an automatic air controlled valve. The flotation concentrate is transferred by integrated pumps to a 10 m^3 flotation cleaner cells. The scavenger concentrate can also be transferred back to the conditioner tank ahead of the circuit. The cleaner cells concentrate is pumped to a 6.0 m, diameter thickener with two integrated pumps. The underflow discharge of the thickener is pumped with peristaltic pumps to a stock tank feeding the dewatering circuit. The cleaner cells tails are sent back to conditioner tank ahead of the circuit with integrated pumps.

17.2.5 Pyrite (or Residual Sulphide) Flotation

The scavenger tails of the zinc/silver circuit are transferred with a variable speed pump to the 70 m^3 conditioner. By gravity the pulp is transferred to a 50 m^3 rougher cell.

The rougher tails are pumped to a three 10 m^3 cells in series; the tails of the last cell become the main mill tailings. These tails are pumped to a 10 m diameter thickener. The overflow is reclaimed water and the underflow is pumped to the tailing pond.

Both rougher and the three scavenger bank of cells float a pyrite concentrate that is pumped to a secure tailing pond. This operation is necessary to produce a final main tail with less than 0.3% S giving non-acid generating materials.

17.2.6 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 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.

17.2.7 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.

17.2.8 Process Plant Layout

The processing plant houses the grinding, flotation, services, reagent preparation and electrical room. The dimensions of the building are 200' length x 140' width. General arrangement drawings for the mill building are appended in Appendix 6 and water treatment unit arrangement is shown in Appendix 7.

17.2.9 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. The effluent will be treated by water treatment plant.

17.2.10 Reagent Handling Systems

17.2.10.1 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 results 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 is 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.

17.2.10.2 Zinc Flotation Collector (A-208)

Total consumption of the zinc collector will be 0.065 g/t of mill feed.

A-208 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. A-208 will be distributed to the individual addition points by metering pumps. One standby pump will be provided.

17.2.10.3 Pyrite Flotation Collector (KAX)

The xanthate KAX 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 9m^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 KAX 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%.

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

17.2.10.4 Zinc and Pyrite Flotation Circuit Frother (F-507)

F-507 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. F-507 will be distributed to the individual addition points by metering pumps. One standby pump will be provided.

17.2.10.5 Quicklime

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

17.2.10.6 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.

17.2.10.7 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 tons 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.

17.2.10.8 Flocculant

One flocculent 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 flocculent 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% flocculent solution strength.

The flocculent 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.

17.2.11 <u>Water and Mechanical Service Systems</u>

17.2.11.1 Fresh and Fire Water

The water requirement for processing is estimated at 180 m^3/hr. About 55 m^3/h of water from the mine; reclaimed from the tailings thickener is estimated at 42 m^3/h and the balance will be either from tailing pond or fresh water pond. Vertical pumps are used and the water is pumped into a process water tank located near the concentrator. The processing water tank has a 340m^3 capacity and is 8.1 m. in diameter x 8.1 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.

17.2.11.2 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.

17.2.11.3 Potable Water

The drinking water will be supplied by bottle refrigerated systems.

17.2.11.4 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 1900 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.

17.2.11.5 Flotation Air

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

17.2.11.6 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.

17.2.11.7 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.

Figure 17.1: Abcourt's crane and flotation equipment

18.0 PROJECT INFRASTRUCTURE

18.1 <u>Introduction</u>

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."

Abcourt Mines has maintained the buildings and roads in good condition to this day. A new surface electrical distribution will be necessary.

18.2 <u>Site Layout</u>

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).

Most of the tailings pond area, the stockpiles/dumps, and the Abcourt pits footprint areas are not covered by trees. Tree-cutting is required over some minor areas of the surface. Topsoil thickness varies from 0.5 to 3 metres while overburden thickness ranges from 2 to 10 metres.

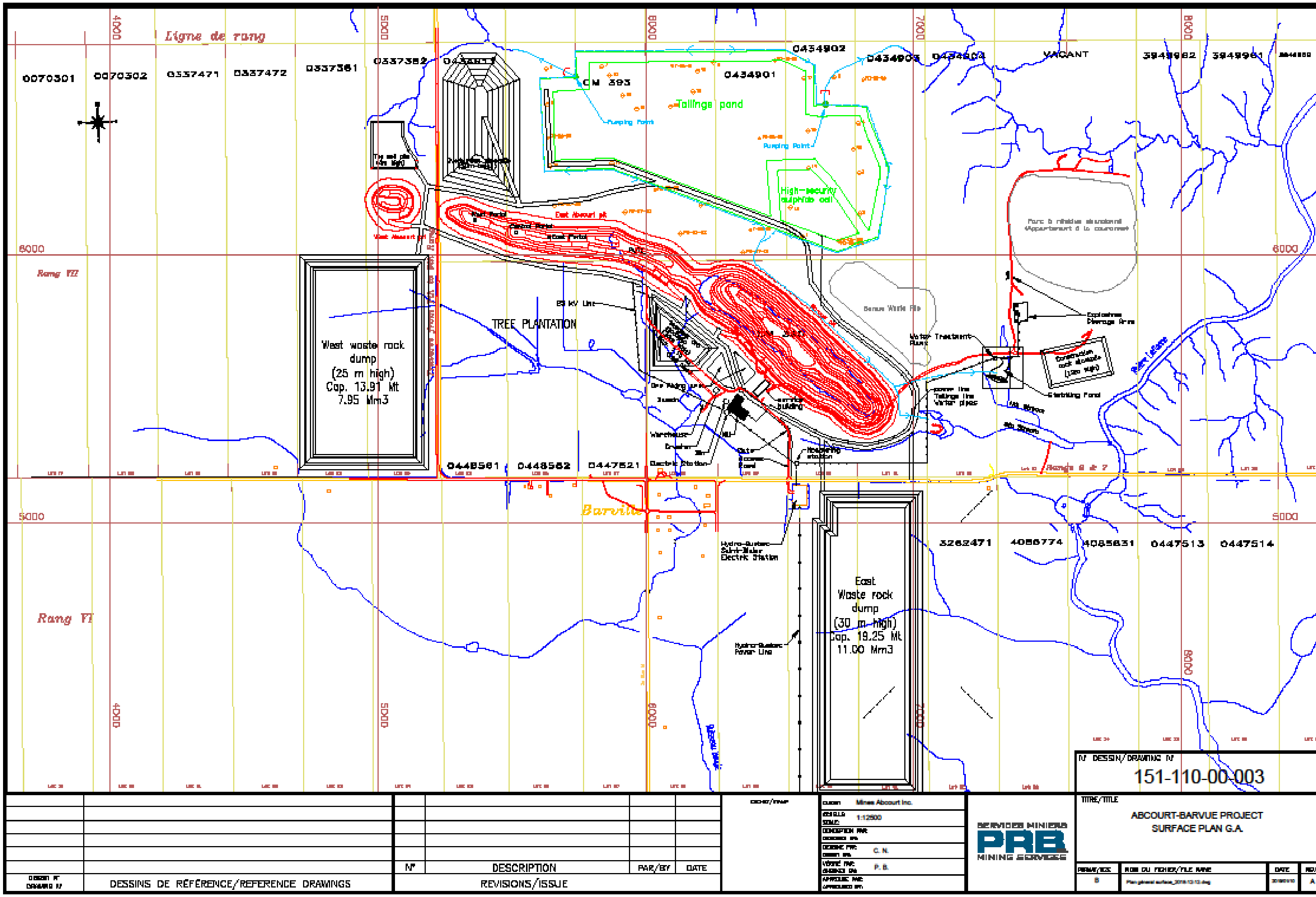

Figure 18.1: Abcourt-Barvue Mine Site Plan View

18.3 Roads

18.3.1 Access Road

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

18.3.2 Site Roads

Site roads will be required to access different parts of the property, namely the topsoil stockpile, the overburden stockpile, the tailings pond, the waste rock pads 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 Barvue and Abcourt preproduction pit excavation. Inert waste rock will be used for this purpose.

18.3.3 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 complete with a phone booth.

The road accessing the East waste pad crosses a public secondary road. A remote controlled gate will be placed on both sides of the public road to restrict access. The truck dRivers will control these gates.

Fencing will be restricted to the main gate, the main sub-station, 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.

18.3.4 Parking Area

The existing parking area will be used.

18.3.5 Service Building

The existing service building is in good condition. Only minor improvement and reshaping will be required for the project. The electrical heating system of the shop and storage areas will be replaced by a propane system. The warehouse was moved to make room for the engineering office. The service building will contain the mine office (staff and technical services), the mine dry, the mine warehouse, the mine rescue room, and a mechanical workshop for light vehicles, drills and other equipment of limited size.

Heavy equipment such as trucks, front-end loaders, and hydraulic shovels will be maintained in a shop in the mill.

18.4 Services

18.4.1 Fuel Storage and Refuelling Station

Abcourt Mines owns a diesel fuel reservoir and functional fuelling station which will be used for the open pit operations. A concrete pad will need to be poured in front of it.

18.4.2 Explosives Storage Facility

The explosives storage facility will consist of a 20,000 kg capacity bulk emulsion explosives transfer site, a magazine for packaged explosives, and a magazine for detonators and accessories. The transfer site and magazines will be erected and operated by the explosives supplier.

18.4.3 Water Systems

The site water systems include: the fresh water supply 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 needs of the mine/mill complex. 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 currently used to provide fresh water to sustain the needs of on-site employees and staff.

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 when required. Otherwise, mine water will be pumped to the tailings pond.

Tailings pond excess water will flow to the water treatment plant and settling pond.

The industrial water needed in the underground mine, when development begins at year 7 of production schedule, will be supplied from a 10,000 gallon filled with mine water as done in the past.

18.4.4 Drinking Water

The artesian well presently servicing the service building provides fresh water for showers and toilets but not drinking water. Bottled water will be provided for drinking.

18.4.5 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 except in the offices which have electric heating.

18.5 Electrical Power

The service building is currently supplied with electrical power from Hydro-Québec's Saint-Blaise station located approximately 300 metres from the process plant. A new electrical distribution network will be required to supply power to the mill, the tailings pond, the water treatment plant, the open pits, and the ramp portals.

The power demand is estimated to 3.8 MW for the mill and 3.5 MW for underground operations. The average power consumption is estimated to 22.0GWh per year for the process plant and 5.4 GWh per year for the underground operations.

The measuring station needs to be replaced.

A 25 kV power line will be erected to the East Abcourt pit above the central stope portal to service underground operations. A 25 kV to 5 kV step-down transformer will be erected at the end of this line. A 5 kV line will be installed down the pit wall to distribute power to the west, central, and east ramp portals.

A 25 kV power line will be constructed to the Barvue pit, the water treatment plant and the tailings pond.

18.6 Waste Management

18.6.1 Sanitary waste water

A septic system is in place and operational. It consists of a network of underground collection piping, a concrete septic tank located near the fence and a septic drainfield. The sanitary water system at the mill building will be connected to the existing septic system.

18.6.2 Domestic Garbage Disposal

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

18.6.3 Other Wastes

Metal scrap will be sold to a scrap dealer and industrial waste/hazardous materials will be managed by an approved contractor. Construction waste rock will be used as needed or sold for land and road fill as requested.

18.7 Telecommunications and Computers

18.7.1 Telecommunications

The service building is currently connected to a commercial 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.

18.7.2 Computers

The computer systems of the mine will include a network platform, a server, and personal computers. The software that will be available on-site will comprise an industry standard office applications package including accounting, warehouse, and technical systems for geology, mine planning and production control.

18.8 Topsoil and Overburden Stockpiles

The topsoil and overburden stockpiles were designed by Genivar in 2007 and described in this section.

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 and the tailings pond footprints.

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.

18.9 Marginal Ore Stockpile

The marginal ore is mineralized rock coming from open pits with grade lower than 1.99% Zn equivalent and a sulphide content greater than 0.3%. Over the 13-year life span of the project, at least 386,000 tonnes of marginal ore will be produced at average grades of 1.07% Zn and 10 gpt Ag, and stockpiled. The marginal ore will be processed at the end of the mine life.

Designed by Genivar in 2007, the marginal ore storage area has a capacity of 1.7 Mt which may be increased 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.

18.10 Waste Rock Stockpile

The waste rock is mineral material coming from open pits and without Zn-Ag bearing mineralization and a sulphide content less than 0.3%. The waste rock of the Abcourt-Barvue project is non-acid generating or inert. The waste produced from the gabbro sill which will be excavated will be utilized for construction purposes at surface (pads, roads, dikes, embankments, fill) for this reason. Under the updated mine plan, the preproduction and the 13-year open pit production phases, will excavate 32.7 Mt of waste rock. Waste rock will be disposed of onto three different storage areas.

Two waste rock dumps and one 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.

Designed by Genivar in 2007, 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. The 2007 study had two waste rock dumps; one to the north-east of the Barvue pit and the other to the south of the East Abcourt pit. Their design is described in the following text in italic font.

> *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).*

The west dump site has been turned into a tree plantation to cut noise and dust to neighbouring residences. The east dump has been relocated to provide space for a future tailings pond expansion.

The update has designed two waste rock dumps using Genivar's 2007 design parameters. Together they have a capacity of 33.16 Mt. The east rock dump is located south of the secondary public road to the south-east of the Barvue pit. It has capacity of 19.25 Mt and a height of 30m. The west waste rock dump is located south of the West Abcourt pit. It has a capacity of 13.91 Mt and a height of 25m. The waste rock from the west and east Abcourt pits will be hauled to the west waste rock dump. The waste from the Barvue pit will be hauled mainly to the east dump.

A ditch will be excavated around both waste rock dumps and a runoff monitoring post will be installed where water may be sampled for quality control.

18.11 Tailings Pond

The Genivar 2007 report provides the following discussion on the tailings pond.

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.*

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.

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 x 10-7 to 2 x 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 x 10-7 to 2 x 10-6 cm/s for the natural conditions, and 5 x 10-8 to 1 x 10-7 cm/s for the tailings operation conditions.

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.

Since then, Golder & Associates Ltd produced a preliminary report on the stability of the overburden located on the north side of the projected Barvue and East Abcourt pits next to the retained tailings pond location. The purpose of this study was to assess the stability of the overburden slopes between the pits and the tailings pond. The preliminary report is dated May 2007.

In 2008, Golder & Associates LTD conducted a detailed geotechnical and hydrogeological investigation of the proposed site and designed a tailings pond. Golder produced a preliminary report dated in July 2008.

The tailings pond as designed by Golder in 2008 has a total capacity of 6.0 Mtonnes of tailings. It is designed in the same perspective as described by Genivar. It is to be constructed in three phases. The first phase consists in the construction of a starter dam and an inner cell for the pyrite concentrate. It is to be followed by two rises. The first rise will be constructed in year 2 and the second in year 6. The updated mine plan will produce 7.6 Mtonnes of tailings and a third rise or an expansion will be necessary in year 9. The tailings pond design will need to be revised to accommodate a greater capacity.

18.12 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 was installed on the Louvicourt Mine tailings site and that was purchased and moved to the Abcourt-Barvue site.

18.12.1 Dewatering the Existing Pit

According to Abcourt Mines, presently the pit water overflow, tested every week, respects environmental regulations and flows to the Laflamme River without treatment. Abcourt Mines plans to dewater the pit initially by pumping directly to the Laflamme River without treatment. The water quality will be monitored. As the water level goes down, water quality may deteriorate. At some point, it is expected to direct pump water to a treatment plant for pH adjustment, precipitation of matter in suspension, and metals in solution.

18.12.2 Mine Water

During the mine operations, the mine water, either from underground work or open pits will be pumped to the tailings pond or the mill.

18.12.3 Waste Stockpiles Runoff

The waste rock will be non-acid generating. Water runoff from the waste rock stockpiles will be collected in ditches to enable quality control.

18.12.4 Ore Stockpile Runoff

Water runoff from both ore and marginal ore stockpiles will be captured in ditches and directed to a sump where it will be pumped to the tailings box in the mill.

18.12.5 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.

19.0 MARKET STUDIES AND CONTRACTS

The zinc concentrate will be sold to a smelter. There are a few smelters in North America and many around the world that may take the concentrate. Typically, a contract is signed with a smelter for a period of six months or one or more years. Abcourt Mines obtained smelting terms from a Canadian smelter for zinc concentrates. The smelting terms include the price of zinc and silver on a specific stock exchange, smelting and refining costs, zinc and silver deductions, zinc and silver payable, and penalties for contaminants if any. These terms are usually in line with those agreed to by the world's major players in contracts negotiated every year.

The London Metal Exchange inventory has steadily dropped from 900,000 tonnes in December 2013 to less than 200,000 tonnes in December 2018 as shown on Figure 19.1. Over the last three years, the price of zinc fluctuated between a low 0.80 US$/lb in 2016 and a high of 1.60 US$/lb in 2018 as shown on Figure 19.2. The average price of zinc during that period was 1.18 US$/lb. The low world zinc inventory suggests that the price of zinc will remain strong in the next few years. Abcourt chose to use a conservative zinc price of 1.10 US$ for the economic analysis.

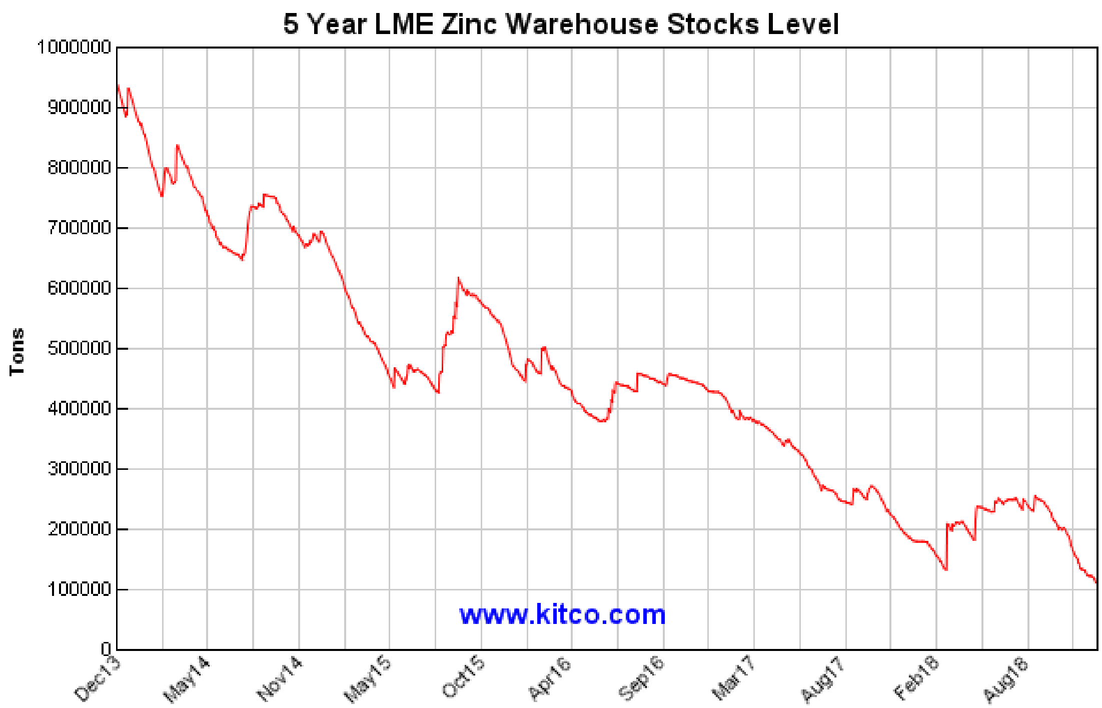

Figure 19.1: 5 Year London Metal Exchange (LME) Zinc Warehouse Stocks Level

Figure 19.2: 3 Historical Zinc Prices from London Metal Exchange (LME)

The price of silver fluctuated between 14.00 US$/Oz and 20.00 US$/Oz over the last three years. The average price of silver during that period was 16.60 US$/Oz. Abcourt chose to use a silver price of 16.50 US$/Oz for the economic analysis.

Figure 19.3: 3 Year Historical Silver Prices (Kitco)

The Canadian to American dollar exchange rate fluctuated between 1.22 CA$/US$ and 1.42 CA$/US$ over the last three years as shown on Figure 19.4. The average exchange rate during that period was 1.30 CA$/US$. Abcourt chose a conservative exchange rate of 1.25 CA$/US$ for the economic analysis.

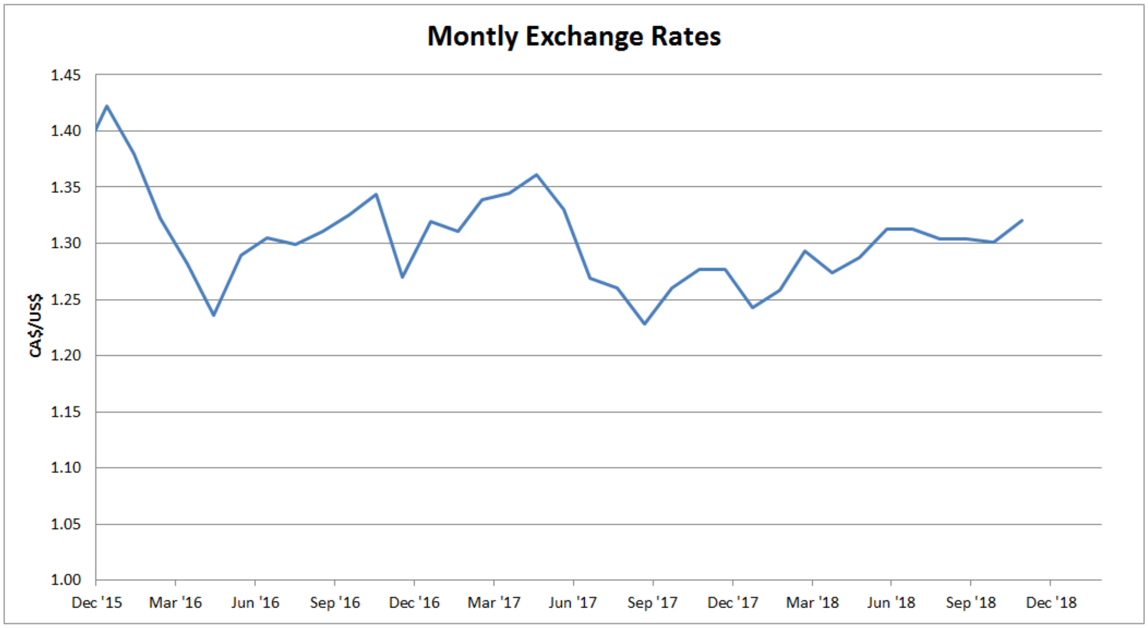

Figure 19.4: Canadian dollar per American dollar (data from Bank of Canada)

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

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.

20.1 Abcourt-Barvue site characterization

Abcourt indicated that a certificate may be obtained from the ministry of environment to the effect that no endangered species are present in the vicinity of the project.

Based on regional environment knowledge and the fact that the project was previously permitted for past mining activities, it is assumed that permitting will not be problematic.

Air and noise studies will have to be completed as part of the permitting phase of the project.

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

The wells of the neighbouring residences were verified in the past and will be verified again.

20.2 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 representative core samples.

A pyrite flotation circuit is part of the ore milling process to reduce the sulphur content under 0.3 % S in the tailings (0.13% to 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. It will be stored in an impervious cell in the tailings pond. It may also be used as backfill in underground stopes. Tests at the URSTM laboratory investigating its potential as paste fill returned positive results using flyash from the power co-generation power station in Senneterre, 45 km away, as a binder instead of cement.

20.3 Environmental management

20.3.1 Waste rock

When the presence of zinc and silver mineralization was observed or detected in drill hole core, 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. The waste will contain less than 0.3% S.

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

20.3.2 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 during or at the end of the operations and processed in the concentrator.

During the production period, the ditches around the ore mixing area and the marginal ore stockpiles will collect surface runoff from stockpiles and the collected water will be directed to a sump from where it will be pumped to the tailings box in the mill.

20.3.3 Open pits and underground mine water

It is believed that open pit and underground mine water will not be acid but will contain zinc in solution or suspension. Mine water will be pumped to the tailings pond.

20.3.4 Pyrite concentrate

The pyrite flotation will produce a pyrite concentrate and the remaining tailings will have 0.13% to 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 thus ensuring no overall potential for acid generation. It may also be stored in underground stopes as paste fill.

20.3.5 Tailings pond

Water in the tailings pond basin will be returned to the mill as process water. The tailings pond overflow will be directed to the water treatment plant.

20.3.6 Mining traffic

Water will be sprayed on mine roads to dust particles in the air caused by road traffic.

20.4 Conceptual closure and restoration plan

20.4.1 Overview

The closure objectives of the conceptual restoration plan to be attained within five years of definite closure are as follows:

- to dismantle and remove buildings and other infrastructures if not used for other purposes;
- 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.

20.4.2 Buildings and infrastructures

If no further potential use is indicated, decommissioning, dismantling and demolition of all buildings and infrastructures, including the mill, will be completed within 5 years after the end of operations with the objective to facilitate salvage of equipment/materials. These activities include the removal of constructed facilities as well as the shaping, rehabilitation and revegetation of prescribed limits . 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 maintenance as required.

20.4.3 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 operations. Presently, the existing Barvue pit overflow meets the environmental standards.

In the present design of open pits, there will be no Zn-Ag bearing mineralization left in the walls. 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. The water treatment plant will remain on site until the end of the monitoring period.

20.4.4 Underground openings

The decline portals will be blocked with waste rock at closure.

Concrete slabs will be constructed over all ventilation raises.

20.4.5 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.

20.4.6 Dumps and piles

The final slopes of the waste rock dumps will be gradually vegetated over the mine life.

The overburden stock pile will be progressively revegetated, by hydroseeding directly on 4H:1V slope for long-term stabilization and aesthetics.

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, or sold.

The footprint of ore stockpiles will be inspected and tested at closure to verify that concentrations respect regulations. Where limits are exceeded, the area affected will be investigated and appropriately remedied.

20.4.7 Tailings pond

The sulphur content of the milling residues is reduced to 0.13% to 0.24 % S since the pyrite flotation was proven effective and has been integrated in the ore treatment process. For this reason, the process tailings are considered non-acid generating. The tailings pond will be revegetated after closure by hydroseeding.

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 generation, or used as paste fill.

20.4.8 Water management installations

The water treatment plant will be dismantled after post-closure activities at the final release of the mine site from governmental authorities.

Ditches around the ore mixing area, the marginal ore stockpiles and the tailings pond will be cleaned up and sediment removed will be transported to a suitable disposal site.

20.4.9 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.

20.4.10 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.

20.4.11 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.

20.4.12 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.

20.4.13 Environmental controls

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

20.4.14 Post-restoration

The main post-restoration 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 5 years.

21.0 CAPITAL AND OPERATING COSTS

The updated cost estimates presented in this section are in 2018 Canadian dollars with no allowance for escalation. The Genivar 2007 costs presented in this section are in 2006 Canadian dollars with no allowance for escalation. According to the Bank of Canada, the consumer price index rose by 23% between 2006 and 2018.

21.1 Capital Costs

21.1.1 Summary

The total capital expenditure required for the Abcourt-Barvue project to mill start-up is estimated to 41.3 M$, including a working capital of 4.0 M$. This excludes a mine restoration deposit of 1.85 M$ required to get environmental permits. The sustaining capital is estimated to 18.1 M$. These costs are in 2018 Canadian dollars, excluding taxes and duties. This compares with a Genivar 2007 estimate of capital expenditure of 71.2 M$ including a pre-production expenditure of 67.88 M$ and a working capital of 3.38 M$. The 2018 estimate is summarized in Table 21.1 and further detailed in this section. The totals for the mine, process plant and infrastructure cost centres contain both direct and indirect costs. Genivar's 2007 estimate is presented in Table 21.2.

Table 21.1: Capital cost summary

	$
Mine	
Equipment	2,185,000
Facilities	849,000
Overburden stripping	865,426
Waste stripping	4,871,206
Mine total	**8,770,632**
Process plant	
Equipment and installation	9,810,264
General and services	3,243,545
General steel works	3,974,162
General concrete works	1,698,492
Sub-total	18,726,463
EPCM (8 %)	1,498,117
Contingencies (15 %)	3,033,687
Process plant total	**23,258,267**
Infrastructure	
Road network	0
Site services	359,000
Power supply	298,000
Tailings pond	2,300,000
Water treatment plant	0
Infrastructure total	**2,957,000**
Owner's costs	**2,350,000**
Preproduction sub-total	**37,335,899**
Working capital	**4,000,000**
Preproduction total	**41,335,899**
Pre-production mine restoration deposit	**1,850,000**
Sustaining Capital	**18,079,593**

Table 21.2: Genivar 2007 Capital Cost Summary

	$
Mine	
Equipment	6,580,000
Facilities	693,000
Overburden stripping	4,059,000
Waste stripping	8,480,000
Mine total	**19,812,000**
Process plant	
Equipment and installation	21,501,286
General and services	3,212,545
General steel works	5,920,215
General concrete works	3,423,383
Sub-total	34,057,429
EPCM (15 %)	5,108,614
Contingencies (15 %)	5,108,614
Process plant total	**44,274,657**
Infrastructure	
Road network	0
Site services	380,000
Power supply	25,000
Tailings pond	1,257,000
Water treatment plant	200,000
Infrastructure total	**1,862,000**
Owner's costs	**1,930,000**
Preproduction total	**67,878,657**
Working capital	**3,376,000**
Sustaing Capital	**24,418,000**

21.1.2 Basis of Estimate (2018)

The mine capital costs were estimated by PRB 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.

The process plant and water treatment capital costs were estimated by BUMIGEME.

The tailings pond was designed by Golder & Associates Ltd which was the basis of the construction costs estimate. Equipment working hours were estimated to which labour and equipment costs were applied.

Genivar's 2007 basis of estimate is described here in italic font as follows.

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.

21.1.3 Capital Estimate Details (2018)

21.1.3.1 Mine Capital Cost

The mine capital cost estimate totals 8.77M$ as presented in Table 21.3. This compares with Genivar's 2007 estimate of 19.8M$ as presented in Table 21.4

The capital mining and support equipment cost is estimated to 2.19M$. Much of the open pit mining equipment will be acquired under a purchase rental agreement reducing significantly the capital cost. The rental cost of this equipment is estimated in the operating cost section. Equipment will be rented for a period of 60 months and purchased at the end of this period. The purchase cost at the end of the 60 month period is in the sustaining cost estimate. This compares with the Genivar 2007 estimate of 6.58M$ where all required equipment was purchased. Abcourt Mines will examine the possibility of purchasing used equipment. The price of reconditioned open pit equipment is typically 50% to 80% of new equipment.

The existing mining facilities, namely haul roads and mine offices also help reduce the capital expenditure. Haul roads will be improved and/or constructed with preproduction waste rock, thus included in waste stripping costs. The mining facilities capital cost estimate of 849,000 $ allows for pit dewatering during the preproduction pit development, together with preparation of the marginal ore stockpile and the ore mixing area, and preparation of explosives storage facilities. This compares with the Genivar 2007 estimate of 693,000 $.

Overburden and waste rock stripping the Barvue pit expansion and the excavation of the West Abcourt pit will be performed during preproduction phase. Preproduction West Abcourt and Barvue ore will be stockpiled ready for mill commissioning and ramp-up.

Preproduction stripping of overburden and waste rock is required to expose sufficient ore to ensure a continuous supply to the mill after start-up. This work will be carried out using the owner's equipment and crews. The estimate was based on open pit operating costs.

Preproduction costs for overburden stripping and rock excavation are estimated at 0.86M$ and 4.87M$ respectively, for a total of 5.74M$, and includes staff, labour, equipment supplies and maintenance costs. This compares with Genivar's 2007 preproduction stripping estimate of 12.5M$. The updated preproduction mine plan excavates 436,000 m3 of overburden and 1,043,426 of rock while the Genivar 2007 plan excavated 1,684,000 m3 of overburden and 4,000,000 tonnes of rock.

Table 21.3: Mine Capital Cost (2018)

	Unit cost ($)	Qty	Cost ($)
Production Equipment			
Production drill - 203 mm dia. (waste blastholes)			
Production drill - 115 mm dia. (ore blastholes)	930,000	1	930,000
Haulage truck 50 tonnes	725,000	1	725,000
Pickup trucks	45,000	4	180,000
Dewatering Equipment			
Submersible Pump (30hp)	30,000	3	90,000
Submersible Pump (60hp)	40,000	3	120,000
Diesel Pump 74hp	70,000	2	140,000
Total mining equipment			2,185,000
Mining Facilities			
Pit dewatering	687,000	1	687,000
Marginal ore stockpile (site preparation)	20,000	1	20,000
Explosives Magazines	142,000	1	142,000
Total mining facilities			849,000
Preproduction stripping and mining			
Overburden Stripping			865,426
Mining			4,871,206
Total preproduction stripping			5,736,632
Total mine preproduction cost			8,770,632

Table 21.4: Genivar 2007 Mine Capital Cost

	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 m³)	700,000	1	700,000
Hydraulic shovel (3.5 m³)	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**

21.1.3.2 Process Plant Capital Cost (2018)

The 2018 Capital cost of the concentrator and crushing sections amounts to 23,258,267 $ as shown in Table 21.5.

The cost includes the equipment listed as per flowsheet, the architectural and structural components, the concrete, the equipment repaired and reconditioned, the equipment transportation cost, the mechanical, electrical and piping costs, the EPCM and 15% for unforeseen and miscellaneous.

Table 21.5: Process Plant and Crushing Capital Cost Summary (2018)

Description	$
Equipment and Installation	
Ore Crushing	1 044 250 $
Ore Grinding	2 747 136 $
Zinc/Silver Flotation	643 305 $
Pyrite Flotation	928 985 $
On Stream Analyser	931 500 $
Zinc Concentrate Dewatering	588 450 $
Reagents Distribution Facilities	582 340 $
Service Section	1 549 923$
Water Plant Treatment	469 055 $
Laboratory	325 320 $
Sub-Total - Equipment and Installation	**9 810 264 $**
General and Services	
Instrumentation/Programmation	1 900 000 $
Main Sub-power Station	250 955 $
Main Power Supply Installation c/w MCC's	509 000 $
Re-circulated Water - 6" Line	190 960 $
Zinc Flotation Tailing Piping	290 190 $
Pyrite Tailing Piping	102 440 $
Sub-Total - General and Services	**3 243 545 $**
General Steel Works	
Crusher Plant (Supports and Accessories)	215 600 $
Crusher Building	122 130 $
Process Plant (Supports and Accessories)	1 900 206 $
Process Building	1 414 675 $
Water Plant Treatment	98 025 $
Laboratory	141 250 $
Main electric Sub-Station	82 276 $
Sub-Total - General Steel Works	**3 974 162**
General Concrete Works	
Crushing Plant (Foundation & Conveyor)	195 048 $
Crusher Building	39 200 $
Process Plant (equipment Foundations)	732 557 $
Piling required for grinding mills, lime bins and overhead structure	37 500 $
Process Plant Building	565 614 $
Main Electric Station	34 500 $
Water Treatment	52 700 $
Laboratory	41 373 $
Sub-Total - General Concrete Works	**1 698 492 $**
SUB-TOTAL-1	**18 726 463 $**
EPCM 8%	1 498 117 $
SUB-TOTAL-2	**20 224 580 $**
Contingencies 15%	3 033 687 $
SUB-TOTAL-3	**3 033 687 $**
TOTAL	**23 258 267 $**

Process equipment costs were estimated with either budgetary quotations or equipment costs database. Minor equipment costs such as small bind or pumps were estimated based on historical data or with an allowance.

The process facilities equipment installation, freight process piping and process electrical and instrumentation costs are shown in Appendix 8. Structural, architecture and concrete costs are shown in Appendix 9.

The present updated capital cost of 23 258 267 $ is lower than the capital cost estimated in the Genivar 2007 report which was evaluated to 44 274 657 $ shown in Table 21.6. The significant reduction in the capital cost is mainly due to the reduction in the direct costs by the use of refurbished used equipment already on site and the elimination of the cyanidation stage which led to a reduction in equipment, concrete and steel costs. The indirect costs were also impacted by the direct costs reduction because of the use of factors applied on the direct costs for EPCM and contingency estimation.

Table 21.6: Genivar 2007 Plant and Crushing Capital 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**

21.1.3.3 Infrastructure Capital Cost (2018)

The 2018 estimated capital cost for the infrastructure required for this project is estimated to 2.96M$ as shown in Table 21.3. This includes the costs for roadwork improvements, site services, power lines and electrical distribution on site, the tailings pond starter dam and internal cell, and the water treatment plant and management system. This cost compares with the Genivar 2007 estimate of 1.82M$ as shown in Table 21.4

Table 21.7: Infrastructure Capital Cost

Description	Total Cost $
Road Network	
Improvement and construction	
Sub-Total - Road network	**0**
Site Services	
Refurbish service building	183,000
Computers and Communications	158,000
Diesel Reservoir & Distribution	18,000
Sub-Total - Site services	**359,000**
Power Line	
25kV Power line to Mill	26,000
25kV Power line from mill to Water Treatment Plant	272,000
Sub-Total - Power line	**298,000**
Tailings Pond	
Starter Dam and Internal Cell	2,300,000
Sub-Total - Tailings pond	**2,300,000**
Total	**2,957,000**

Table 21.8: Genivar 2007 Infrastructure Cost

	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**

a) Road Network

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.

b) Site Services

Items under site services include refurbishing the existing service building, computers and telecommunication, and the installation of a diesel fueling station concrete pad. The costs for refurbishing the existing maintenance shop, and the computers and telecommunication were estimated in 2007 based on quotations and in-house data. A factor of 22% was applied to these to account for inflation. The cost of installing the diesel fueling station accounts for a concrete pad and hookup. The total site services cost is estimated to 359,000$. This compares to the Genivar 2007 estimate of 380,000$.

c) Power Line

The existing 25 kV power line with transformers and electrical distribution network are no longer available on site and will be replaced.

The capital cost for the power line and electrical distribution is estimated to 298,000$. This compares with the genivar 2007 estimate of 25,000$ when an existing power line was in place.

Repairs to the measuring station is estimated to cost of 26,000$.

A 25kV power line will be installed from the measuring station to the mill, the East Abcourt pit, the Barvue pit, the tailings pond, and the water treatment plant at a cost of 272,000$.

d) Tailings Pond

The quantity of materials required for the construction of the starter dike was provided by Golder and Associates in their 2007 report. All the waste rock and clay needed for the construction of this dike is available on site or will be produced during the preproduction stripping operation. The sand and gravel will be hauled in from a borrow pit 5 km from the the tailings pond. The tailings pond starter dike and inner cell will be constructed by Abcourt personnel and equipment. The cost is estimated to 2.3M$. The cost includes stripping and grubbing, transfer and placement of clay, haulage and placement of sand and gravel, the water management circuit between the mill and the tailings pond, and instrumentation. The cost of placement of the waste rock is included in the pit excavation costs.

This compares to the Genivar 2007 estimate of 1.26M$. The difference is due to the use of sand and gravel from the borrow pit and higher operating costs.

e) Water Treatment and Water Distribution

The water treatment plant constructed at the southeast limit of the tailings pond will consist of used equipment purchased from Louvicourt Mine in Abitibi and refurbished by Abcourt Mines. The cost of the water treatment plant is included in the mill construction cost. The cost for the construction of a settling pond is included in the cost for the tailings pond starter dam. The Genivar 2007 estimate allowed a provision of 200 000 $ for the construction of a 54 m2 building on a concrete slab and installation of the equipment on the Abcourt-Barvue site.

21.1.3.4 Ownwer's Costs

Owner's costs are indirect cost supported by Abcourt during the construction and preproduction phase. In this case, these are general administration costs for one year (details in Operating costs section) and spare parts purchase for warehouse inventory. The Genivar 2007 estimation of general administration for one year was 1.43 M$ and warehouse inventory was 0.5 M$, for a total owner's cost of 1.74 M$. This estimate was revised by applying a 2% inflation over 10 years to the Genivar 2007 numbers. The updated estimate for owner's costs is 2.35 M$.

21.1.3.5 Mine Restoration

The total estimated mine restoration cost for the life of mine is 3.7M$. According to the environmental regulations of the Province of Québec instituted in 2016, 50% of this amount is to be paid to a reserve fund to obtain a mining lease. The remaining 50% is to be paid to the fund in two annual instalments of 25% each in the first two years of operation. Abcourt Mines has two mining concessions and two surface leases in good standing for the Abcourt-Barvue project. A mine restoration deposit of 1.85M$ will be required in the preproduction period followed by two mine restoration deposits of 925,000$ each in years 1 and 2.

The mine restoration activities will occur within five years of the definite mine closure. The cash flow in section 22, however, presents the mine closure costs occurring in year 14. The cost will be offset by the sale of assets resulting in a net estimated cost of 0.0M$.

This compares with the 2007 Genivar total estimate of 1.75M$ for mine rehabilitation.

21.1.3.6 Working Capital

The working capital is equivalent to 2 months of operating costs and amounts to 4.0M$. This will be entirely recovered at the end of the project. This compares with the Genivar 2007 estimate of 3.38M$.

21.1.3.7 Sustaining Capital

The usual accounting practice is to capitalize major expenditures during production. Costs related to additional surface equipment, stripping of remaining overburden, progressive rehabilitation, raising of the tailings pond and initial underground mine investment will be capitalized. Underground excavation of declines, loading stations, sumps, and ventilation raises will be capitalized whereas sublevels and stope raises will be accounted as operating costs. Table 21.9 itemizes the estimated sustaining capital cost for the Abcourt Barvue project. The total sustaining capital cost is estimated to 18.1M$. This compares to the Genivar 2007 estimate of $24.4M$ as shown in Table 21.10.

Table 21.9: Sustaining Capital Expenditure (2018)

	Year 1 ($)	Year 2 ($)	Year 3 ($)	Year 4 ($)	Year 5 ($)	Year 6 ($)	Year 7 ($)
Surface and Open Pit							
Mining Equipment	0	0	0	0	1,677,645	1,007,762	0
Overburden stripping	2,449,395	0	496,231	0	0	0	0
Tailings pond	0	500,000	0	0	0	490,000	0
Mine rehabilitation	0	0	0	0	0	0	0
Sub-total	2,449,395	500,000	496,231	0	1,677,645	1,497,762	0
Underground mine							
Initial Investment	0	0	0	0	0	0	0
UG Equipment	0	0	0	0	0	0	0
Capitalized Development	0	0	0	0	0	0	0
Sub-total	0	0	0	0	0	0	0
Total	**2,449,395**	**500,000**	**496,231**	**0**	**1,677,645**	**1,497,762**	**0**

Cont'd	Year 8 ($)	Year 9 ($)	Year 10 ($)	Year 11 ($)	Year 12 ($)	Year 13 ($)	Year 14 ($)	Total ($)
Surface and Open Pit								
Mining Equipment	0	0	0	0	0	0	0	2,685,407
Overburden stripping	0	0	0	0	0	0	0	2,945,625
Tailings pond	0	490,000	0	0	0	0	0	1,480,000
Mine rehabilitation	0	0	0	0	0	0	0	0
Sub-total	0	490,000	0	0	0	0	0	7,111,032
Underground mine								
Initial Investment	1,192,000	0	0	0	0	0	0	1,192,000
UG Equipment	2,433,537	1,880,537	1,435,737	30,000	0	0	0	5,779,811
Capitalized Development	3,007,703	404,548	584,500	0	0	0	0	3,996,750
Sub-total	6,633,240	2,285,085	2,020,237	30,000	0	0	0	10,968,561
Total	**6,633,240**	**2,775,085**	**2,020,237**	**30,000**	**0**	**0**	**0**	**18,079,593**

Table 21.10: Genivar 2007 Sustaining capital Expenditure

	Year -1 ($)	Year 1 ($)	Year 2 ($)	Year 3 ($)	Year 4 ($)	Year 5 ($)
Surface and open pits						
Wheel loader (6.8 m³)		700,000				
Haul truck (62-t rebuilt)			600,000			
Overburden stripping (west pit)						
Marginal ore mining&stockpiling		268,400	254,000	323,900	368,400	390,300
Tailings pond			2,269,000			
Progressive rehabilitation		100,000	100,000	100,000	100,000	100,000
Sub-total		1,068,400	3,223,000	423,900	468,400	490,300
Underground mine						
Initial investment						
On-going development						
Sub-total						
Total		**1,068,400**	**3,223,000**	**423,900**	**468,400**	**490,300**

(Continued)	Year 6 ($)	Year 7 ($)	Year 8 ($)	Year 9 ($)	Year 10 ($)	Total ($)
Surface and open pits						
Wheel loader (6.8 m³)						**700,000**
Haul truck (62-t rebuilt)						**600,000**
Overburden stripping (west pit)	578,200					**578,200**
Marginal ore mining&stockpiling	372,800	158,400	97,700	96,800	84,900	**2,415,600**
Tailings pond		2,041,000				**4,310,000**
Progressive rehabilitation	100,000	100,000	350,000	350,000	350,000	**1,750,000**
Sub-total	1,051,000	2,299,400	447,700	446,800	434,900	**10,353,800**
Underground mine						
Initial investment	1,712,000					**1,712,000**
On-going development	2,565,600	2,825,800	2,320,200	2,320,200	2,320,200	**12,352,000**
Sub-total	4,277,600	2,825,800	2,320,200	2,320,200	2,320,200	**14,064,000**
Total	**5,328,600**	**5,125,200**	**2,767,900**	**2,767,000**	**2,755,100**	**24,417,800**

a) Surface and open pits

The 2018 total sustaining capital estimate for surface and open pit is 8.01$M. The open pit production and support equipment on rental will be purchased in years 5 and 6 at a cost of 2.68M$. Excavation of the remaining overburden during years 1 and 3 is estimated to cost 2.95M$.

The tailings pond will be raised in years 2, 6, and 9 at a total cost of 1.48M$.

The Genivar 2007 report explained the surface and open pit sustaining capital as follows:

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.

b) Underground mine

The underground sustaining capital component consists of preparation work, mining equipment, and development work.

The preparation work for underground mining consists of installing related surface installations such as ventilation, mine air heating, compressed air, water supply, etc. The estimated preparation work totals 1.19M$ as shown in Table 21.11.

Table 21.11: Preparation Work for Underground Mining (2018)

Relocation of the water tank	31,000 $
New shaft collar	16,000 $
Installation of two compressors and compressor room	244,000 $
Compressed air line	49,000 $
Relocation of power line and sub-stations	0 $
Installation of main ventilation fans and heating systems	92,000 $
Air & Water lines on 90 m level	131,000 $
Extension of the 90 m sub-level in Abcourt and breakthrough of ramps in the pit	397,000 $
Power line to ramp portals	232,000 $
Total	**1,192,000 $**

Abcourt Mines has a number of underground equipment from former operations such as jumbos, scooptrams, fans, compressors, water tank, pumps, etc. The equipment purchase requirement will consist of low profile trucks, support equipment, main ventilation fans and heating systems, pumping equipment and is estimated to total 5.78M$ as shown in Table 21.12. In comparison, the Genivar 2007 estimate for purchase of underground equipment totaled 650,000$.

Table 21.12: Underground Equipment Purchase Schedule (2018)

Description	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Total
UG Equipment Expenditure Schedule								
Development								
Truck 42 tonnes	1,239,737	0	0	0	0	0	0	1,239,737
Scissor lift truck	200,000	0	0	0	0	0	0	200,000
Tractor	85,000	0	0	0	0	0	0	85,000
Production								
Truck 42 tonnes	0	1,239,737	1,239,737	0	0	0	0	2,479,474
Support								
Tractor w/ forks	0	90,000	0	0	0	0	0	90,000
Boom truck	0	200,000	0	0	0	0	0	200,000
Ventilation								
Mine air propane heater 6 MBTU	180,000	180,000	0	0	0	0	0	360,000
Fan skid unit & cones 80,000 cfm each	75,800	75,800	0	0	0	0	0	151,600
Dewatering								
Réservoir 2000 gallons sur skis	42,000	0	0	0	0	0	0	42,000
Pompes submersibles (75hp)	180,000	0	90,000	0	0	0	0	270,000
Séparateur Mudwizard (7 pieds)	64,000	0	0	0	0	0	0	64,000
Pompe submersible (30hp)	0	0	30,000	30,000	0	0	0	60,000
Pompe submersible (15hp)	30,000	10,000	0	0	0	0	0	40,000
Electricity								
Transformer 25kV/5kV/600V	45,000	0	0	0	0	0	0	45,000
Electrical Sub station 1000 kva 1000 kva	152,000	0	76,000	0	0	0	0	228,000
Electrical Sub station 750 kva 750 kva	0	0	0	0	0	0	0	0
Electrical Sub station 500 kva 500 kva	0	0	0	0	0	0	0	0
Miscellaneous								
Toyota for transp workers, technical serv, etc.	0	85,000	0	0	0	0	0	85,000
Mobile refuge (or trailor) + toilet	40,000	0	0	0	0	0	0	40,000
Mine Rescue Equipment (gears, clothes, Ocenc	100,000	0	0	0	0	0	0	100,000
Total Mining Equipment Expenditure	2,433,537	1,880,537	1,435,737	30,000	0	0	0	5,779,811

The Genivar 2007 estimate for preparation work for underground operation totals 1.71M$ including equipment purchase as shown in Table 21.13.

Table 21.13: Genivar 2007 Preparation Work for Underground Mining

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 $

The 2018 development unit costs were estimated from base principals using Abcourt personnel and equipment. They include labour, equipment maintenance and operating costs, consumables, and materials. The total estimated cost of development work is 16.7M$ as shown in Table 21.14 The declines, loading stations, sumps ventilation raises, and diamond drilling will be capitalized at an estimated cost of 5.88M$. The sub-levels and stope raises will account as an operating expenditure at an estimated total cost of 10.86M$.

Table 21.14: Underground Development Cost (2018)

	Qty	Unit Cost	Total Cost	Capital Expenditure	Operating Expenditure
West stope (5010E to 5280E)					
Ramp	500 m	at 2730 $/m	$1,365,000	$1,365,000	
Sub-levels	1,755 m	at 2150 $/m	$3,773,250		$3,773,250
Loading and passing stations	100 m	at 2090 $/m	$209,000	$209,000	
Sumps	50 m	at 2050 $/m	$102,500	$102,500	
Stope raises	100 m	at 1040 $/m	$104,000		$104,000
Ventilation raise / Escapeway	300 m	at 3010 $/m	$903,000	$903,000	
Diamond drilling	1,500 m	at 85 $/m	$127,500	$127,500	
			$6,584,250	**$2,707,000**	**$3,877,250**
Central stope (5295E to 5760E)					
Ramp	475 m	at 2730 $/m	$1,296,750	$1,296,750	
Sub-levels	1,640 m	at 2150 $/m	$3,526,000		$3,526,000
Loading and passing stations	100 m	at 2090 $/m	$209,000	$209,000	
Sumps	50 m	at 2050 $/m	$102,500	$102,500	
Stope raises	60 m	at 1040 $/m	$62,400		$62,400
Ventilation raise / Escapeway	180 m	at 3010 $/m	$541,800	$541,800	
			$5,738,450	**$2,150,050**	**$3,588,400**
East stope (5295E to 5760E and 15W to 135E)					
Ramp	100 m	at 2730 $/m	$273,000	$273,000	
Sub-levels	1,500 m	at 2150 $/m	$3,225,000		$3,225,000
Loading and passing stations	100 m	at 2090 $/m	$209,000	$209,000	
Sumps	50 m	at 2050 $/m	$102,500	$102,500	
Stope raises	160 m	at 1040 $/m	$166,400		$166,400
Ventilation raise / Escapeway	145 m	at 3010 $/m	$436,450	$436,450	
			$4,412,350	**$1,020,950**	**$3,391,400**
Total development cost			**$16,735,050**	**$5,878,000**	**$10,857,050**

This compares with the Genivar 2007 total estimate for underground development work of 7.83M$, as shown in Table 21.15, which was all capitalized.

Table 21.15: Genivar 2007 UG Development Cost

West stope (5010E to 5280E)			
Ramp No 3	500 m	at 1800 $/m	900,000 $
Sub-levels	1,755 m	at 1800 $/m	3,159,000 $
Loading and passing stations	100 m	at 1800 $/m	180,000 $
Sumps	50 m	at 1800 $/m	90,000 $
Stope raises	100 m	at 1060 $/m	106,000 $
Fill and ventilation raises	300 m	at 1060 $/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 1800 $/m	855,000 $
Sub-levels	1,640 m	at 1800 $/m	2,952,000 $
Loading and passing stations	100 m	at 1800 $/m	180,000 $
Sumps	50 m	at 1800 $/m	90,000 $
Stope raises	60 m	at 1060 $/m	63,600 $
Fill and ventilation raises	180 m	at 1060 $/m	190,800 $
Sampling and assaying			25,000 $
			4,356,400 $
East stope (5295E to 5760E and 15W to 135E)			
Ramp No 1			
Sub-levels	100 m	at 1800 $/m	180,000 $
Loading and passing stations	1,500 m	at 1800 $/m	2,700,000 $
Sumps	100 m	at 1800 $/m	180,000 $
Stope raises	50 m	at 1800 $/m	90,000 $
Fill and ventilation raises	160 m	at 1060 $/m	169,600 $
	145 m	at 1060 $/m	153,700 $
			3,473,300 $
Total development cost			**7,829,700 $**

21.2 Operating Costs

21.2.1 Summary

The 2018 operating costs are summarized in Table 21.16 and further detailed in this section. The total operating cost over the 13 year mine life is estimated to 322.5M$, for an average of 8.04 $/t mined or 39.94 $/t milled. This compares with the Genivar 2007 estimate for the 10 year mine life of 201.4M$ for an average of 5.65 $/t mined and 31.16 $/t milled as shown in Table 21.17 The updated mine plan processes 8.07M tonnes ore whereas the Genivar 2007 mine plan processed 6.44M tonnes of ore.

Table 21.16: Operating Cost Summary (2018)

Item	Life-of-mine costs Total (M$)	Unit costs Mined ($/t)	Unit costs Milled ($/t)
Mining	186.7	4.66	23.13
Processing	111.1	2.77	13.76
Environment	2.0	0.05	0.25
G&A	22.62	0.56	2.80
Royalties	0	0.00	0.00
Total	322.5	8.04	39.94

Table 21.17: Genivar 2007 Operating Cost Summary

Item	Life-of-mine costs Total (M$)	Unit costs Mined ($/t)	Unit costs Milled ($/t)
Mining	98.6	2.77	15.3
Processing	87.42	2.45	13.46
G&A	14.3	0.4	2.22
Royalties	1.16	0.03	0.18
Total	201.48	5.65	31.16

21.2.2 <u>Basis of the 2018 Estimate</u>

Salaries and hourly labour rates are based on similar operations in the Abitibi region and Abcourt Mines current costs at other operations.

Supplies and parts costs were obtained from potential suppliers as budget quotations and from in house data for similar projects.

In a few cases Genivar 2007 costs were used with adjustment for inflation.

The following text in italic describes the Genivar 2007 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.*

21.2.3 <u>Labour</u>

A) Mine

The 2018 work schedule will be the same as in the Genivar 2007 report. The open pit will work on a schedule of 8 hours per shift, 3 shifts per day, 7 days per week, and 52 weeks per year. The work schedule will be reduced as the total rock production drops in later years. The underground operation will work 8 hours per shift, 2 shifts per day, 5 days per week, and 52 weeks per year. Mining personnel requirement and costs are shown in Table 21.18. The costs include salaries, bonus, and fringe benefits of 20%. The salaries and conditions reflect of current Abcourt Mines operations.

Table 21.18: Mine Manpower Requirement (2018)

Description	Employees on payroll	Salary & Benefits Cost
OPEN PIT		
Staff Employees		*($/year)*
Mine superintendent	1	165,600
Shift boss	4	108,222
Mining engineer	1	126,000
Mining technician	1	91,020
Geological technician	1	91,020
Surveyors	2	91,020
Maintenance foreman	1	108,222
Environmental coordinator	1	72,000
Sub total	12	
Hourly Employees		*($/hour)*
Production loader operators	4	45.50
Shovel operators	3	45.50
Truck drivers	26	40.30
Drillers (ore)	4	43.73
Drillers (waste)	4	43.73
Blaster	1	43.73
Blaster helper	1	38.23
Stockpile loader operators	3	40.30
Dozer operator	1	44.54
Dozer & Rock breaker op.	1	44.54
Grader & Water truck op.	1	44.54
Mechanic	3	48.25
Mechanic helper	3	45.50
Sub total	55	
Total	**67**	

Description	Employees on payroll	Salary & Benefits cost
UNDERGROUND		
Staff Employees		*($/year)*
Captain	1	180,000
Shift boss	2	156,000
Maintenance foreman	1	144,000
Mining engineer	1	180,000
Geological technician	1	91,020
Surveyor	1	91,020
Safety mine rescue	1	144,000
Sub total	8	
Hourly Employees		*($/hour)*
Long hole drillers	3	65.25
Muckers	6	60.90
Truck drivers	3	50.75
Jumbo operators	6	65.25
Jumbo helpers	6	60.90
Blasters	3	58.00
Mechanics	6	60.90
Electrician	2	60.90
Sub total	35	
Total	**43**	

Table 21.19 shows the Genivar 2007 manpower requirement. The Genivar 2007 labour costs included fringe benefits of 34% for the hourly employees and 23% for the staff employees.

Table 21.19: Genivar 2007 Mine Manpower Requirement

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	31.2	Long hole drillers	3	44.3
Shovel operators	3	31.2	Muckers	6	35.0
Truck drivers	16	29.5	Truck drivers	3	35.0
Drillers (ore)	4	35.0	Jumbo operators	6	44.3
Drillers (waste)	4	35.0	Jumbo helpers	6	38.8
Blaster	1	40.8	Blasters	3	44.3
Blaster helper	1	35.4	Mechanics	6	34.4
Stockpile loader operators	3	31.2	Electrician	1	32.8
Dozer operator	1	29.5	Sub total	34	
Dozer & Rock breaker op.	1	29.5	Spares	4	
Grader & Water truck op.	1	29.5	**Total**	**46**	
Sub total	39				
Spares	4				
Total	**54**				

B) Mill

The 2018 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 20.77% as fringe benefits to the hourly paid employees and 17.48% for the staff employees. The manpower requirement is shown on Table 21.20 and Table 21.21

In comparison, the Genivar 2007 estimate also increased the number of employees by 10% for replacement during vacation and other reasons but used a fringe benefit of 27.55% for hourly paid employees and 15.14% for staff employees. The Genivar 2007 manpower requirement is shown on Table 21.22.

Table 21.20: Mill Manpower Requirement and Costs (2018)

Description	Annual Salary	Employees		Total Annual	Cost per Tonne
Staff Employees:					
Plant Superintendent/Metallurgist	149 000 $		1	149 000 $	0,229 $
General Forman	95 000 $		1	95 000 $	0,146 $
Mechanic Forman	95 000 $		1	95 000 $	0,146 $
Technician	64 500 $		1	64 500 $	0,099 $
Sub Total			**4**	**403 500 $**	**0,621 $**
Hourly Employee:	**Rate $ / Hr.**				
Crusher/Loader Operators	29,18 $	65 363,20 $	3	196 090 $	0,302 $
Concentrator Senior Operators	33,96 $	76 070,40 $	4	304 282 $	0,468 $
Concentrator Junior Operators	30,24 $	67 737,60 $	4	270 950 $	0,417 $
Concentrator Helper Operator	27,59 $	61 801,60 $	4	247 206 $	0,380 $
Maintenance Mechanic	33,96 $		4	271 680 $	0,418 $
Maintenance Electric/Instrumentation	33,96 $		1	67 920 $	0,104 $
Security (contractor)	22,00 $		4	176 000 $	0,271 $
Sub Total			**24**	**1 534 128 $**	**2,360 $**
Miscellaneous:	**Average/Rate**		**Hours**		
Maintenance Overtime	50,94 $		1 250	63 675 $	0,098 $
Night Shifts Bonus 261 Days X 3 E.X12Hr	8,00 $		9 396	75 168 $	0,116 $
Weekend Bonus (Average at 1/2)	15,12 $		3 744	56 609 $	0,087 $
Overtime related to holidays	25,47 $		864	22 006 $	0,034 $
Maintenance Contractor (Not included in benefits)	85 $		250	21 250 $	0,033 $
Sub Total				**238 708 $**	**0,367 $**
Total Salaries				**2 176 336 $**	3,348 $
Salaries vacation (Staff not included)	1 751 586 $		6,00%	105 095 $	0,162 $
Employees Benefits:	2 260 181 $		17,48%	395 080 $	0,608 $
Total Manpower Cost				**2 676 511 $**	
Average Processing Manpower Cost Per Metric Tonnes					**4,118 $**

Table 21.21: Mill Employees Benefits Costs (2018)

Description	Hourly %	Maximum Annual	Staff %
Québec Pension Plan	5,40%	55 900,00 $	3,21%
Federal Employment Insurance	1,30%	51 700,00 $	0,71%
Quebec Health Insurance	2,30%		2,30%
Quebec CNESST	6,80%		6,80%
RQAP	2,30%	74 000,00 $	1,81%
Quebec Commission des Normes du Travail	0,07%	72 500,00 $	0,05%
Company Group Insurance	1,70%		1,70%
Compagny. Pension Plan	0,90%		0,90%
Total Benefits cost	**20,77%**		**17,48%**
Total Payroll		**2 260 181 $**	

Table 21.22: Genivar 2007 Mill manpower Requirement and Costs

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	

C) General and Administration (2018)

General and administration (G&A) personnel requirements are listed in Table 21.23. The unit labour costs include salaries, bonus, and fringe benefits of 20%.

Table 21.23: G&A Personnel (2018)

Description	Number of employees	Annual Salary ($)
Mine manager	1	204,000
Chief accountant	1	108,000
Accounting clerk	1	84,000
Storekeeper	1	96,000
Secretary	1	78,000
Total	**5**	**570,000**

Genivar's 2007 estimate for general and administration personnel requirements and salaries, including fringe benefits of 15.14 % are listed in Table 21.24.

Table 21.24: Genivar 2007 G&A Personnel

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**

21.2.4 Operating Estimate Details

21.2.4.1 Mine Operating Costs

The cost for open pit mining was estimated on a per tonne basis using average requirements for the life of mine and applied to yearly tonnages.

Drilling will be performed by Abcourt personnel. Unit drilling costs were estimated for ore, waste, and perimeter holes. A compounded average was estimated to determine the average drilling cost per tonne.

Blasting will be performed by the supplier's personnel or by drillers and supplier's equipment. A cost was estimated for ore, waste, and perimeter blasting. A compounded average cost was estimated to determine the blasting cost per tonne.

Load and haul equipment (trucks, shovels, and loaders) operating hours for overburden, waste rock, ore, and marginal ore were estimated using tonnes and travel distances and speeds for each bench. Hourly equipment and labour costs were applied to the operating hours and an average cost per tonne was calculated for total rock loaded and hauled. The load and haul equipment will be acquired on a rental/purchase basis. The equipment will be rented for 5 years and purchased at a residual sales price. The equipment 5 year rentals were estimated as a separate cost item. A diesel fuel price of $1.00 per litre was assumed based on suppliers' quotes.

Production support includes the operation of a grader, a bulldozer, a front-end loader, a water truck, a hydraulic shovel, and 2 pickup trucks working 8 hours per day. It also includes operators and mechanics.

Dewatering includes the cost of pumps, materials, electricity, and labour. Electrical power will be provided by Hydro-Quebec. An average cost of $0.0525 per kwh was assumed.

The supervision and technical services include a mine superintendent, four mine supervisors, one maintenance foreman, one environmental coordinator, one mine engineer, one mine technician, one surveyor, and one geological technician.

The open pit operating cost is estimated to 3.66 $/t mined or 22.04 $/t milled for drilling and blasting, load and haul, support activities, dewatering, technical services supervision, equipment rental, and a 10% contingency. The breakdown of the open pit mining cost per tonne mined is given in Table 21.25 This compares with the Genivar 2007 estimate of 2.34 $/t mined and 15.15 $/t milled as shown in Table 21.26

The main differences are due to the following:

a) The load and haul has more trucks than the Genivar 2007 estimate because 50 tonne trucks are used instead of 63 tonne trucks as explained in section 16.
b) Production support includes mechanics. Under the Genivar 2007 estimate, all equipment maintenance was by the supplier and was included in the equipment operating costs.
c) Cost of labour is higher than in the Genivar 2007 estimate.
d) The pit operating costs include rental of most equipement for five years whereas no equipment was rented in the Genivar 2007 estimate.

Table 21.25: Open Pit Mining Costs (2018)

Activity	Unit cost ($/t mined)	Component	Unit cost ($/t mined)
Drilling and blasting	1.03	Labour	0.08
		Fuel	0.09
		Explosives	0.42
		Bits & Steel	0.07
		Services	0.23
		Maintenance	0.14
Load and haul	1.13	Labour	0.42
		Fuel	0.33
		Maintenance	0.38
Production support	0.68	Labour	0.37
		Fuel	0.14
		Other supplies	0.17
Dewatering	0.06	Labour	0.00
		Electricity	0.01
		Fuel	0.01
		Maintenance	0.04
		Other supplies	0.01
Technical services and Supervision	0.48	Technical Service	0.18
		Supervision	0.28
		Other Supplies	0.02
Equipment Rental	0.28	Load & Haul	0.18
		Production support	0.10
Total	**3.66**		

Table 21.26: Genivar 2007 Open Pit Mining Costs

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 underground stope production cost is estimated to 27.38 $/t of ore as shown in Table 21.27. This cost excludes development excavations. This compares to the Genivar 2007 estimate of 20.93 $/t of ore as shown in Table 21.28.

Table 21.27: Underground Production Cost (2018)

Component	Unit cost ($/t mined)	Item	Unit cost ($/t mined)
Stoping and ore transportation	15.92	Drilling and blasting	4.28
		Mucking and transportation	7.52
		Stope rockfilling	3.74
		Mucking before blasting	0.39
Fixed costs	11.46	Mechanical services	4.09
		Electricity	0.93
		Propane	1.61
		Tech. serv. and supervision	4.82
Total	**27.38**		

Table 21.28: Genivar 2007 Underground Production 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**		

In combining the overall 2018 total costs of open pit and underground mining, the 13-year average mine operating cost is estimated to 4.52 $/t mined or 18.08 $/t milled, as given in Table 21.29 as shown in table 21.2.12. This compares to the Genivar 2007 estimate of 2.77 $/t mined and 15.30 $/t milled as shown in Table 21.30.

Table 21.29: Summary of Total Mine Operating Costs (2018)

Description	13-year total cost (M$)	Unit costs (OP & UG combined)	
		Mined ($/t)	Milled ($/t)
Open pit	140.5	3.50	17.39
Underground mine	46.3	1.15	5.73
Total	**186.7**	**4.66**	**23.12**

Table 21.30: Genivar 2007 Summary of Total 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**

21.2.4.2 Processing Operating Costs (2018)

The operating costs are based on processing 1800tm of ore per day or an average of 650 000 tonnes per year. The operating costs are based on budgetary costs for new equipments, reagents, actual wages in the area, and actualized costs for other items like mechanical maintenance, building maintenance, office supply, safety supply, etc, as detailed in the report study of 2007, prepared by Bumigeme and RWJ Consultant.

The process operating cost is estimated at 13.76 Canadian dollars per tonne milled and consists of the manpower, energy, consumable reagents, spares and others required for the operation of the concentrator (process plant).

In comparison, the Genivar 2007 estimate for the processing operating cost including cyanidation of ore and pyrite concentrate was 13.46 Canadian dollars per tonne milled.

Table 21.31: Summary of detailed Operating Cost (2018)

Description	**Cost /Ton**	**Annual cost**
Staff Employees Salary	0,621 $	403 500 $
Hourly Employees Salary	2,360 $	1 534 128 $
Miscellaneous	0,367 $	238 708 $
Employees Vacations 6%	0,162 $	105 095 $
Employees Benefits Cost	0,608 $	395 080 $
Operating and environmental Reagents	2,030 $	1 319 728 $
Operating Consumable	3,648 $	2 371 000 $
Mechanical Maintenance Supply	0,692 $	450 000 $
Electrical/Instrumentation Supply	0,323 $	210 000 $
Piping Maintenance Supply	0,101 $	65 500 $
Building Maintenance	0,068 $	44 000 $
Building Heating (Gas)	0,191 $	124 000 $
Custom Laboratory (Assay's)	0,251 $	163 000 $
Plant Office Supply	0,049 $	32 000 $
Health / Safety Supply	0,083 $	54 000 $
Vehicules/loader supplies	0,095 $	62 000 $
Consultants Fees	0,077 $	50 000 $
Power (Hydro Quebec)	2,035 $	1 322 750 $
Total Operating Cost:	**13,76 $**	**8 944 489 $**

Table 21.32: Reagents and Consumable Costs (2018)

Processing reagents	Consumption Kg/Ton	Price Kg	Total	Cost per ton
Lime (processing)	0,5	0,27 $	87 750 $	0,14 $
A-208	0,065	4,50 $	190 125 $	0,29 $
F-507	0,078	3,60 $	182 520 $	0,28 $
Copper sulphide	0,039	3,80 $	96 330 $	0,15 $
Kax	0,018	4,20 $	49 140 $	0,08 $
F-357	0,01	4,10 $	26 650 $	0,04 $
Sulfuric Acid	0,4	0,75 $	195 000 $	0,30 $
Concentrator lab.supplies		0,12 $	78 000 $	0,12 $
Copper sulphide (Pyrite treatment)	0,065	3,80 $	160 550 $	0,25 $
Aerofloat	0,03	3,80 $	74 100 $	0,11 $
Lime (environment	0,5	0,27 $	87 750 $	0,14 $
Flocculent	0,025	3,05 $	49 563 $	0,08 $
Coagulant	0,025	2,60 $	42 250 $	0,07 $
Sub total			**1 319 728 $**	**2,03 $**
Operating Consumables				
Jaw Crusher Liners (sets)	1	32 000 $	32 000 $	0,05 $
S.A.G. liners (sets)	1,6	300 000 $	480 000 $	0,74 $
Ball mill liners (set)	0,3	130 000 $	39 000 $	0,06 $
Ball mill liners (set)	0,3	130 000 $	39 000 $	0,06 $
S.A.G. Grinding Balls	0,8	1,60 $	832 000 $	1,28 $
Ball mill Grinding Balls	0,5	1,46 $	474 500 $	0,73 $
Regrind Grinding Balls	0,5	1,46 $	474 500 $	0,73 $
Sub total			**2 371 000 $**	**3,65 $**
Total cost:			**3 690 728 $**	**5,68 $**

Table 21.33: Power costs (2018)

Hydro Québec Charges	
Demand	14,46 $/kW
Consumption:	for the first 210 000 kWh $/kWh 0,0499 $
	for the additional kWh $/kWh 0,037 $
Rebated $ per kW of demand for user's transformation (Included)	
Power connected	5 245 H.P. X ,746 = 3 913 kW
Utilization Evaluation 3 913 X factor 70%	2 739 kW or 33,9 kWh/metric ton
Base on 650 000 t/y	650 000 t/y X 33,9 kWh = 22 035 000 / 12 months = 1 836 250 kWh/month

Estimated monthly invoice:			
Monthly			
Demand kW	2 739	14,4600 $	39 606 $
First	210 000	0,0499 $	10 479 $
Balance	1 626 250	0,0370 $	60 171 $
Total per month			110 256 $
Cost per ton			**2,035 $**

Table 21.34: Genivar 2007 Process Plant Operating 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**

21.2.4.3 Environment

A provision of 75,000$ in each of years 1 and 5 was allocated to monitoring of aquatic life downstream from the project. Regular monitoring testing will cost 144,000$ per year. The total environmental operating cost estimate is 2.0M$. This represents 0.05 $/t mined and 0.25 $/t milled.

An environmental cost component was not specified in the Genivar 2007 estimate.

21.2.4.4 General and Administration Costs

The general and administration (G&A) cost is estimated to 1.74M$ per year. This represents a unit cost of 0.56 $/t mined and 2.80 $/t milled.

Genivar 2007 report described the general and administration cost as follows.

> *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.*

21.2.4.5 Royalties

The project is no longer subject to Royalties.

The Genivar 2007 report contained royalties as per the following description.

> *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.*

22.0 ECONOMIC ANALYSIS

22.1 Introduction

An economic model was prepared for the Project to estimate annual cash flows and assess sensitivities to certain economic parameters. The economic results of this report are based upon the engineering performed by Bumigeme Inc., PRB Mining Services, and Abcourt Mines Inc.

The Project includes an open pit mine, an underground mine, a process plant to recover a zinc concentrate, surface infrastructure to support the mine and mill operations, and a tailings storage facility. The analysis used a zinc price of 1.10 US$/lb, a silver price of 16.50 US$/Oz, and an exchange rate of 1.25 CA$/US$.

The Project indicates a pre-tax cash flow of $170.0 million Canadian dollars, a pre-tax NPV @ 8% discount rate of $72.6 million Canadian dollars, and a pre-tax IRR of 26.1%. The project pre-tax payback period is 4.9 years. Using a discount rate of 5% (as in the Genivar 2007 report), the pre-tax net present value is $100.4 million Canadian dollars. The project is most sensitive to revenue, in particular to the price of zinc.

The Project indicates an after-tax cash flow of $106.7 million Canadian dollars, an after-tax NPV @ 8% of $41.0 million Canadian dollars and a pre-tax IRR of 20.5%. The project after-tax payback period is 5.3 years.

Table 22.1summarizes the Economic Analysis results.

Table 22.1: Summary of Economic Analysis Results, Base Case

Item	Units	Value
Production		
Mine life	years	13
Total mill feed	tonnes	8,074,162
Average mill feed grade		
Zinc	%Zn	2.83%
Silver	gpt Ag	52
Zinc Equivalent	%ZnEq	3.85%
Commodity Prices		
Zinc	US$/lb	1.10
Silver	US$/Oz	16.50
Exchange rate	CA$/US$	1.25
Project Costs		
Mining Cost	$/t milled	23.13
Milling Cost	$/t milled	13.76
Environment Cost	$/t milled	0.25
G&A Cost	$/t milled	2.80
Total Operating Cost	$/t milled	39.94

Item	Units	Value
Project Economics		
Net Revenue	$M	547.9
Total Operating Cost Estimate	$M	322.5
Total Sustaining Capital Cost Estimate	$M	18.1
Total Capital Cost Estimate	$M	41.3
Duties and Taxes	$M	63.3
Average Annual EBITDA	$M	18.3
Pre-Tax Cash Flow	$M	170.0
After-Tax Cash Flow	$M	106.7
Discount Rate	%	8%
Pre-Tax Net Present Value @ 8%	$M	72.6
Pre-Tax Internal Rate of Return	%	26.1%
Pre-Tax Payback Period	years	4.9
After-Tax Net Present Value @ 8%	$M	41.0
After-Tax Internal Rate of Return	%	20.5%
After-Tax Payback Period	years	5.3

The Genivar 2007 base case analysis used a zinc price of 1.15 US$/lb, a silver price of 9.54 US$/Oz, a gold price of 560 US$/Oz, an exchange rate of 1.15 CA$/US$, and a discount rate of 5% (based on a risk premium of 7% minus an inflation rate of 2%). Based on 100% equity, the Genivar 2007 base case analysis estimated a pre-tax cash flow of 138.7 million Canadian dollars, a pre-tax net present value @ 5% discount rate of 87.6 million Canadian dollars, and an internal rate of return of 27.1%.

22.2 Cautionary Statement

The results of the Economic Analysis are based on forward looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements in this section include, but are not limited to, statements with respect to:

- Future prices of zinc and silver;
- Currency exchange rate fluctuations;
- Estimation of Mineral Reserves;
- Realization of Mineral Reserve estimates; and
- Estimated costs and timing of Capital and Operating expenditures.

22.3 Principal Assumptions

The cash flow estimate includes only revenue, costs, taxes, and other factors applicable to the Project. Corporate obligations, financing costs, sunk costs, and taxes at the corporate level are excluded.

The model was prepared from mining schedule estimated on an annual basis. The cash flow model was based on the following:

- All costs are reported in Canadian dollars (CAN$) and referenced as '$', unless otherwise stated.
- One hundred percent (100%) equity basis.
- No cost escalation beyond 2018.
- No provision for effects of inflation.
- Constant 2018 dollar analysis.
- The economic analysis consists of the technical assumptions outlined in the previous sections, together with the economic assumptions and estimated Capital and Operating costs described in Section 21.
- The economic analysis is based on Abcourt Mines' preferred scenario:
    - Process – Flotation Zn-Ag concentrate:
        - Zinc recovery 97.5%
        - Silver recovery 77.8%
        - Concentrate zinc grade 52.65%
    - Commodity prices:
        - Zinc 1.10 US$/lb
        - Silver 16.50 US$/Oz
    - Exchange rate 1.25 CA$/US$
- Any additional project study costs have not been included in the analysis.
- Mine rehabilitation costs at the end of the project were included in the economic analysis with the assumption that sales of assets would offset a portion of those costs.
- The cost of financing the working capital was not included in the analysis.

22.4 Taxes and Royalties

22.4.1 Duties and Taxes

The Project has been evaluated on an after-tax basis. It must be noted that there are many potential complex factors that affect the taxation of a mining project. The taxes, depletion, and depreciation calculations in the FS economic analysis are simplified and only intended to give a general indication of the potential tax implications.

The Project will be subject to the following taxes as they relate to the Project:

- A federal income tax rate of 15%.
- A provincial corporate income tax rate of 11.5% (in 2020 and thereafter).
- A provincial mining tax rate from 16% to 24% depending on the profit margin of the year.

Processing Allowance

A company is entitled to deduct a processing allowance in the calculation of its mining profit. Basically, this deduction corresponds to 10% of the original value of an asset used in the ore processing.

Depreciation Allowance

A company may claim a depreciation allowance on an asset used in the mining operations at the declining rate of 30%.

22.4.2 Royalties

The project is not subject to Royalties.

22.5 Economic Results, Base Case

The results are derived from the production schedule presented in Section16, the recovery method discussed in Section 17, and the Capital and Operating costs presented in Section 21. The cash flow model is presented in Table 22.2.

Table 22.2: Cash Flow Model, Base Case

Description	Unit	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Mill Feed																	
Ore Milled	tonnes	8,074,162	0	596,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	328,162	0
Zinc Grade	%Zn	2.83%	0.00%	2.53%	2.49%	2.45%	2.60%	2.94%	2.98%	2.98%	3.03%	2.98%	2.95%	2.95%	2.97%	3.03%	0.00%
Silver Grade	gpt Ag	52	0	62	52	48	49	33	32	33	44	68	68	67	60	64	0
Zinc Equivalent	% Zn Eq	3.85%	0.00%	3.74%	3.51%	3.39%	3.57%	3.59%	3.61%	3.62%	3.89%	4.31%	4.29%	4.27%	4.16%	4.29%	0.00%
Zinc Concentrate Production																	
Concentrate produced	dmt	423,429	0	27,934	30,015	29,452	31,335	35,413	35,921	35,827	36,460	35,871	35,503	35,497	35,776	18,426	0
Zinc grade	%Zn		0.00%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	0.00%
Silver grade	gpt		0	1,026	869	826	792	468	451	462	608	958	972	961	853	887	0
Zinc recovered	tonnes	222,936	0	14,707	15,803	15,506	16,498	18,645	18,912	18,863	19,196	18,886	18,692	18,689	18,836	9,701	0
Silver recovered	Oz	10,459,043	0	921,842	838,872	781,661	797,835	533,281	521,252	532,306	713,065	1,104,884	1,109,593	1,097,325	981,367	525,757	0
Revenues																	
Prices																	
Exchange rate	CA$/US$		0.00	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	0.00
Zinc Price	US$/lb		0.00	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	0.00
Silver Price	US$/Oz		0.00	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	0.00
Net Smelter Return	M$	547.9	0.0	39.8	40.4	39.0	40.9	40.4	40.6	40.7	44.2	49.9	49.7	49.5	47.9	25.0	0.0
Costs																	
Operating Costs	M$	322.5	0.0	30.3	29.8	31.5	33.8	27.3	24.0	21.0	21.7	23.9	23.5	23.5	21.3	10.8	0.0
Capital Costs	M$	37.3	37.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Costs	M$	18.1	0.0	2.4	0.5	0.5	0.0	1.7	1.5	6.6	2.3	2.5	0.0	0.0	0.0	0.0	0.0
Working Capital	M$	0.0	4.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-4.0	0.0
Duties and Taxes	M$	63.3	0.0	0.4	0.4	0.4	0.4	3.1	5.2	5.9	7.1	8.5	8.8	9.1	9.5	4.6	0.0
Total Costs	M$	441.2	41.3	33.2	30.7	32.4	34.2	32.1	30.7	33.5	31.1	34.9	32.4	32.5	30.8	11.4	0.0
Cash Flow Results																	
EBITDA	M$	225.4	0.0	9.5	10.6	7.5	7.1	13.0	16.6	19.7	22.5	26.0	26.2	26.0	26.5	14.2	0.0
Pre-Tax Cash-Flow	M$	170.0	-41.3	7.1	10.1	7.0	7.1	11.4	15.1	13.1	20.2	23.5	26.1	26.0	26.5	18.2	0.0
Cumulative Pre-Tax Cash Flow	M$		-41.3	-34.3	-24.2	-17.3	-10.1	1.2	16.3	29.4	49.7	73.1	99.3	125.3	151.8	170.0	170.0
After-Tax Cash-Flow	M$	106.7	-41.3	6.7	9.7	6.6	6.7	8.3	9.9	7.2	13.1	15.0	17.3	16.9	17.1	13.6	0.0
Cumulative After-Tax Cash-Flow	M$		-41.3	-34.6	-25.0	-18.4	-11.6	-3.4	6.5	13.7	26.8	41.8	59.1	76.0	93.1	106.7	106.7

Table 22.3 summarizes the economic indicators, both pre-tax and after-tax, for the estimated cash flow model in Table 22.2.

Table 22.3: Economic Indicators, Base Case

Economic Indicator	Units	Pre-Tax	After-Tax
Payback Period (from start of production)	years	4.9	5.3
Cash Flow	M$	170.0	106.7
Internal Rate of Return	%	26.1%	20.5%
Net Present value @ 5%	M$	100.4	59.8
Net Present value @ 8%	M$	72.6	41.0

22.6 Sensitivity Analysis, Pre-Tax Basis

The pre-tax cash flow was evaluated for sensitivity to revenue, capital expenditures, and operating costs. All sensitivities were analyzed as mutually exclusive variations.

The project's pre-tax NPV was most sensitive to revenue, in particular to the fluctuation in the price of zinc, followed by operating costs (opex) as shown in Figure 22.1 and Figure 22.2.

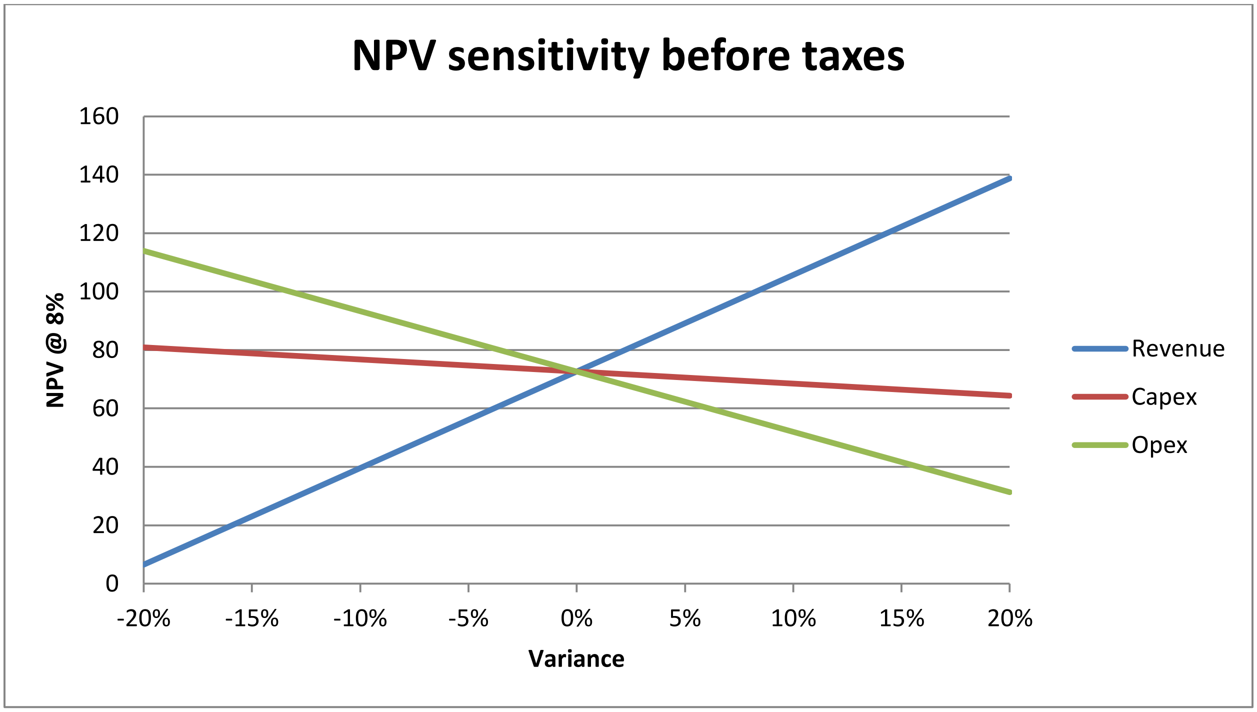

Figure 22.1: Pre-Tax Sensitivity Analysis on $NPV_{8\%}$

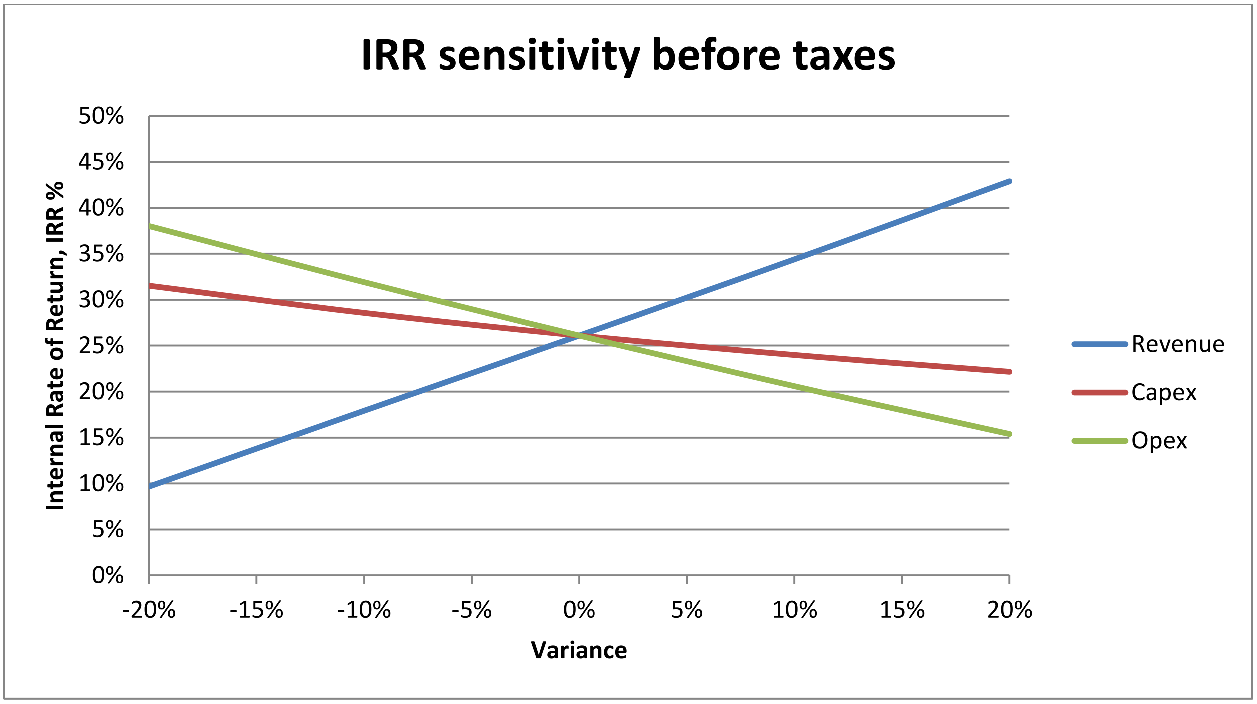

Figure 22.2: Pre-Tax Sensitivity on IRR

23.0 ADJACENT PROPERTIES

The text of this section is taken from the report titled NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property dated February 28, 2014.

Although, the Barraute area is hosting numerous mineral occurrences, there is no deposit or producing mine in the vicinity of the Abcourt-Barvue property. The Company owns the Vendome, Barvallée and Belfort Zn-Ag deposits (The "Vendome property") and a group of gold occurrences possibly associated with a regional scale NNE trending structure called the Laflamme Fault. Notable gold-bearing structures like the Bartec and Swanson deposits are also related to the Laflamme Fault (Figure 23.1).

23.1 Zinc-Silver Deposits

The Vendome property is located 11 km south of the Abcourt-Barvue property. This 59-claim property, wholly owned by ABCOURT, covers three zinc-silver-copper-gold deposits;

The Vendôme (Mogador) zinc-copper deposit is located 2.4 km southwest of the municipality of Barraute in Fiedmont Township. The mineralization consists mainly in sphalerite with smaller amounts of chalcopyrite, galena, native silver and gold disseminated in semi massive pyrite and pyrrhotite. The hosting rock is a rhyolite of the Kinojevis Group located close to a granodioritic stock. The orebody was probably generated by syn-volcanic gold and base metals-bearing sulphides. A NI 43-101 compliant resources evaluation was prepared by the Author in February 2013 in which he estimated that a total of 675 554 tonnes grading 7.50% Zn, 0.51% Cu, 58.50 g/t Ag and 1.10 g/t Au were in place. Most of the mineralization is accessible from drifts developed on levels 76, 114 and 152.

The Barvallée zinc-copper deposit is located 1.6 km west of Mogador deposit in Barraute Township. The ore zone consists in massive sulphides associated with a rhyolitic horizon intercalated in andesite and basalts of the Kinojevis Group. The massive and disseminated mineralization is composed of pyrite, pyrrhotite, sphalerite, chalcopyrite, silver and gold. The NI 43-101 compliant mineral resources are estimated at 275 923 tonnes grading 4.38% Zn, 0.87% Cu, 48.06 g/t Ag and 1.31 g/t Au in the indicated and the inferred categories.

The Belfort (Roymont) zinc-silver-copper ore body is located in the south-east corner of the Barraute Township (Lot 28, Range I). The massive sulphides mineralization is hosted by rhyolites of the Kinojevis Group located near the edges of a granodioritic stock. The mineralization is massive and disseminated and consists in pyrite, pyrrhotite, sphalerite, galena, gold, silver and chalcopyrite. The zone is in the westerly extension of the Barvallée and Vendôme deposits. NI 43-101 compliant mineral resources were estimated at 66 625 tonnes grading 5.71% Zn, 19.22 g/t Ag, 0.18% Cu and 1.05 g/t Au in the inferred category.

23.2 Gold Deposits

The Bartec (Ontex) gold deposit is 8.9 km north of Barraute and is adjacent to the eastern limit of the Abcourt-Barvue property. It consist in quartz-carbonate-sericite veins associated with a shear zone crosscutting andesitic flows and pyroclastic rocks of the Figuery Formation. Mineralization consists in disseminated pyrite, native gold and silver, chalcopyrite and mariposite. The historical resources, as indicated in the Quebec's government SIGEOM web site, were estimated at 113 400 tonnes grading 7.9 g/t Au.

Additionnal gold showings are found not far from the Laflamme fault on the Abcourt-Barvue property.

The Swanson gold deposit is located 8.7 km south-east of La Morandière and 1.4 km east from Laflamme River. According to press releases delivered in December 2003 by Phoenix Matachewan Mines, the deposit contains NI 43-101 compliant resources of 1 108 642 tonnes grading 3.17 g/t Au (Source: MRB & Associates). The gold mineralization is associated with disseminated pyrite and quartz veins and veinlets with several free gold intersections which may also contain Cu-Pb-Zn low values. The deposit is associated with the syenite of Swanson and the adjacent basic volcanic rocks of the Amos Group.

North Occurrences

Zn-Ag

Frebert W Ext.;	Zn showing (no work)
Frebert;	Zn-Ag deposit (worked)
Pershcourt;	Zn-Ag deposit (worked)
Mine Barvue;	Zn-Ag mine (closed)
Bar-Manitou;	Zn-Ag deposit (worked)
Ruisseau Blin;	Zn showing (no work)

Au

Bartec;	Au deposit (113 400 t.)
Jackson;	Au deposit (worked)
Malbar/G.L.;	Au occurrence (few work)
Swanson;	Au deposit (1,1 Mt @ 3.2 g/t)

South Occurrences

Zn-Ag

Vendome;	Zn deposit with tonnage
Barvallée;	Zn deposit with tonnage
Belfort;	Zn deposit with tonnage
Absam:	Zn-Ag occurrence (worked)

Au

Venus (Barexor); Au showing (29 700 t.)

Tonnes : NI 43-101 compliant, Tonnes : Not NI 43-101

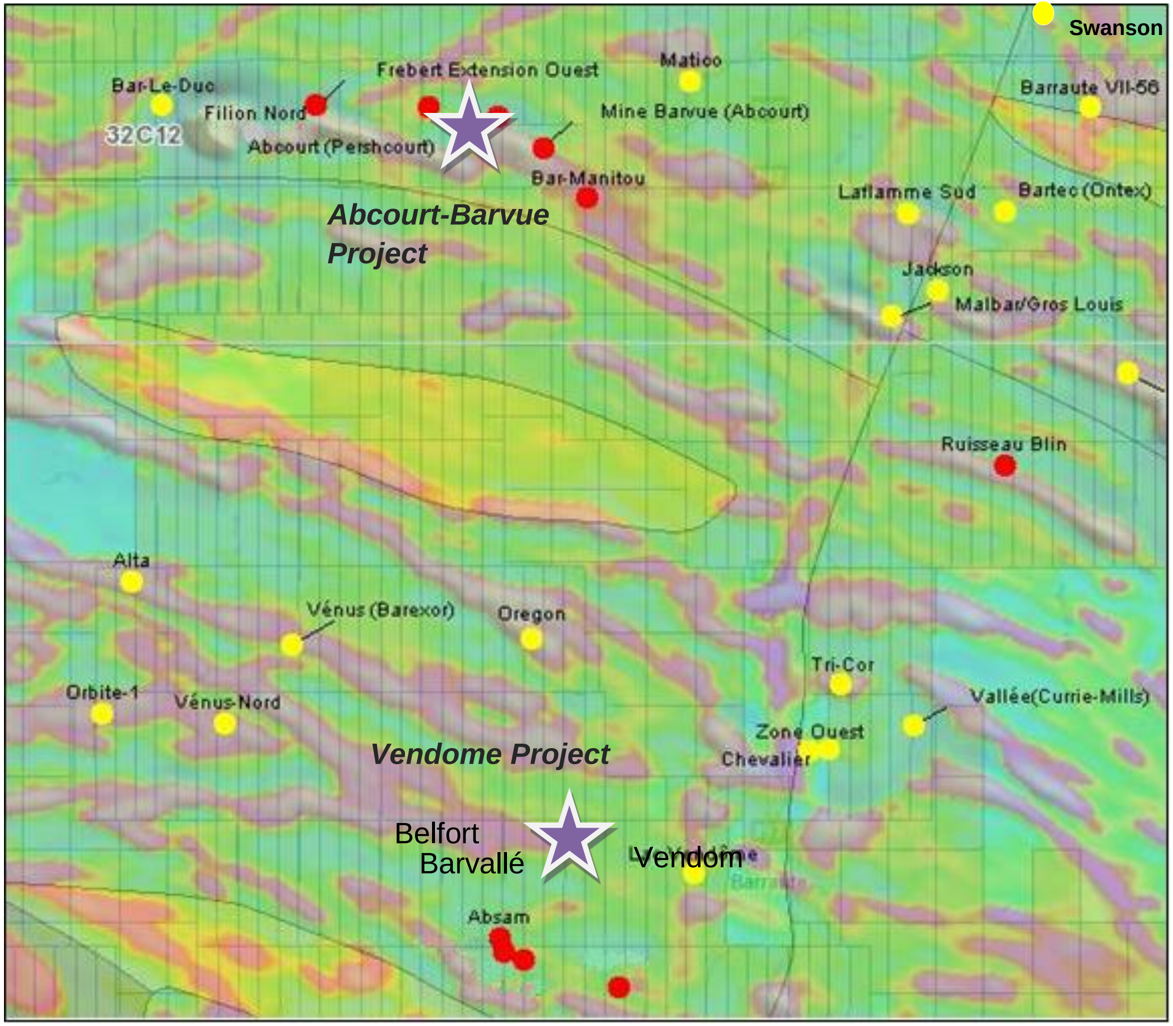

Figure 23.1: Compilation of major deposits and mineralized occurrences in the Barraute Area as compiled from Sigéom's database.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Implementation

The Project implementation schedule covers all the areas of the Project and includes the engineering, procurement, permitting, construction, and commissioning of the facilities, and pre-production excavations. The facilities include the main electrical transfer station, 25 kV power lines, the process plant, and site infrastructure.

The Project schedule reflects the schedule contained in the Genivar 2007 report. It assumes environmental certificates of authorization and the mining lease for the Abcourt West pit will be obtained in due time. The planned mill start-up is planned for month 20 of the implementation schedule as shown on figure 24.1.

Abcourt Mines will have an Owner's team to manage the detailed engineering, procurement, and construction. It will contract consultants to conduct the detailed engineering for each discipline, as required.

The Project will not require a camp as it is in located in an area with good infrastructure and services.

Mine construction priority will be given to detailed engineering, procurement, and site preparation to initiate the mill construction as early as possible.

Construction work and pre-stripping activities will be carried out on a 5 days/week, regular 8 hours per day schedule,

It is assumed that all concrete work will be carried out during the summer months.

Figure 24.1: Project implementation schedule

25.0 INTERPRETATION AND 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.

Genivar Limited Partnership (Genivar) and Bumigeme Inc. produced a feasibility study in 2007 titled Technical Feasibility Study Report On The Abcourt-Barvue Deposit and dated February 15, 2007 (Genivar 2007 study). The mine plan developed in the study produced a total of 6,446,000 tonnes of ore grading 3.11% Zn and 54.96 gpt Ag, of which 5,338,700 tonnes were produced in open pit operations and 1,107,300 tonnes were produced from underground operations.

The open pit operation consisted of the expansion of the Barvue pit and the excavation of the Abcourt East and the Abcourt West pits over a period of 10 years. The pits were excavated to a depth of 166m, 72m, and 42m respectively. The underground operations consisted in the mining stopes from a depth varying from 150m to 200 m below surface to the pit bottoms using the Avoca method. The underground work areas were accessed by excavating declines.

The processing plant was designed to process 650,000 tonnes per year. It had two cyanidation circuits and a flotation circuit producing silver ingots and a zinc-silver concentrate as sales products.

The Genivar 2007 study resulted in a mineral reserve of 6,446,000 tonnes grading 3.11% zinc and 54.96 gram per tonne silver.

A mineral resource estimate was produced by Jean-Pierre Bérubé in 2014 titled NI 43-101 Mineral Resources Report For The Abcourt-Barvue Property, dated February 18, 2014. It used the Genivar 2007 pit shells and increased significantly the resource estimate used in the Genivar 2007 study. The increase in mineral resource was most significant in the open pits and was insignificant underground.

A mine plan was developed for the 2014 mineral resource estimate using the Genivar 2007 pit design and underground mine design. The 2014 mineral resource diluted and recovered produced a total of 8,074,162 tonnes of mill feed grading 2.83% Zn and 51.79 gpt Ag, of which 6,589,361 tonnes (81.6%) were produced in open pit operations and 1,484,801 tonnes (18.4%) were produced in underground operations. The life of mine is 13 years. There are good possibilities of increasing the life of mine by converting inferred resources into indicated or measured resources and by finding new resources with additional exploration.

The processing plant design remains at a mill feed of 650,000 tonnes per year but was modified by eliminating the cyanidation circuits to produce only a zinc-silver concentrate. Minor changes were brought to the surface infrastructure such as the installation of new 25kV power lines and the relocation of the waste rock stockpiles.

The project capital and operation costs were estimated for the new mine plan, modified process plant and site infrastructure using current costs. Mine rehabilitation costs were also estimated.

Rehabilitation costs were applied to year 14 at the end of production. Project revenues were estimated using 1.10 US$ per pound zinc, 16.50 US$ per ounce silver, an exchange rate of 1.25 CA$ per US$, and smelting & refining terms.

An average of 32,000 tonnes of concentrate grading 52.7% Zn and 768 gpt Ag is produced annually. The project preproduction capital cost is estimated to 41.3M$ including working capital of 4.0M$, and the sustaining capital cost is estimated to 18.1M$. The average operating cost is estimated to 39.69 $/t milled.

An economic analysis with metal prices and rate of exchange indicated above, assuming 100% equity financing, resulted in a pre-tax cash flow of 170.0 million Canadian dollars, a pre-tax rate of return (IRR) of 26.1% and a pre-tax net present value (NPV) of 72.6 million Canadian dollars using an 8% discount rate. Using a discount rate of 5% as in the Genivar 2007 report, the pre-tax NPV is estimated to 100.4 million Canadian dollars. A sensitivity analysis on revenue, capital cost, and operating cost shows the project is most sensitive to total revenue (price of zinc and rate of exchange) followed by operating cost.

In comparison, the Genivar 2007 study's economic analysis used a zinc price of 1.15 $/lb, a silver price of 9.54 $/Oz, and an exchange rate of 1.15 CA$/US$ and, assuming 100% financing, returned a pre-tax cash flow of 140.4 million Canadian dollars, a pre-tax IRR of 27.5% and a pre-tax NPV at 5% discount rate of 89.1 million Canadian dollars.

26.0 RECOMMENDATIONS

26.1 General

Given that the Abcourt-Barvue project contains an economic mineral reserve and based on the cash flow analysis using realistic long-term metal price 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 zinc-silver concentrates for sale.

26.2 Project Infrastructure

The tailings pond designed by Golder and Associates has a capacity of 6.0 M tonnes. The mine plan developed for this study produces 7.6 M tonnes of tailings over its 13 year mine life. It is recommended to review the tailings pond design to accommodate the increased tailings production after year 9.

The East and West waste rock dumps were designed to a height of 30 meters and 25 meters respectively. They were designed without knowledge of the bearing capacity of the soil beneath them. It is recommended that the soil in the East and West waste rock dumps be characterized and the design reviewed accordingly.

27.0 REFERENCES

27.1 Accessibility, Climate, Local Ressources, Infrastructure, and Physiography

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27.2 Geological Setting and Mineralization

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27.3 Mineral Resource Estimate

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27.4 Mining Methods

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27.5 Project Infrastructure

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ROCHE Ltd Consulting Group, 1999; Barvue-Vendôme-Abcourt project (BARVENAB) - Technical Validation Report, May 1999.

27.6 Market Studies and Contracts

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KITCO, 2018; Monthly Silver Chart, https://www.kitco.com/charts/historicalsilver.html> (December 13, 2018).

LME, 2018; Daily Prices and Monthly Averages, < https://www.lme.com/Market-Data/Reports-and-data/Monthly-averages> (December 13, 2018).

LME, 2018; Warehouse and Stock Reports, < https://www.lme.com/en-GB/Market-Data/Reports-and-data/Warehouse-and-stocks-reports> (December 10, 2018).

28.0 CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF QUALIFIED PERSON

Paul Bonneville, Eng.

I, Paul Bonneville, Eng, of Val d'Or Québec do hereby certify that:

- I reside at 190 boul Dennison, Val d'Or, Quebec, Canada J9P 2K7.
- I am a graduate from Queen's University, Kingston, Ontario, Canada with a B.Sc. Degree in Mining Engineering (1980), and I have practiced my profession for over 30 years.
- I am a Professional Engineer registered with the Ordre des Ingénieurs du Québec (OIQ) with membership number 36248.
- This certificate applies to the report entitled *Update to the Technical Feasibility Report on the Abcourt-Barvue Deposit* (the Technical Report) dated January 15, 2019.
- As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101). I visited the Abcourt-Barvue site in July 2018.
- 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.
- I am responsible for sections 4 to 12, sections 14, 15, and 16, and sections 18 to 23 and portions of sections 1, 2, 3, 24, 25, 26, and 27 that are based on those sections.
- I have read the sections of the technical report for which I am responsible and which have been prepared in compliance with Instrument NI 43-101.
- I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Abcourt-Barvue Project or securities of Abcourt Mines Inc. or any related subsidiary.
- I operate under the business name PRB Mining Services Inc.
- As of the effective date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 15th day of January 2019 at Val d'Or, Québec.

Paul Bonneville, Eng.
President
PRB Mining Services Inc.

MINES ABCOURT INC.

UPDATE TO THE TECHNICAL REPORT ON THE ABCOURT-BARVUE DEPOSIT

CERTIFICATE OF QUALIFIED PERSON

I, Florent Baril, B.Sc., Senior Metallurgical Engineer and President of:

Bumigeme Inc.
615 René-Lévesque West, Room 750
Montreal, Quebec H3B 1P5

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 60 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. This certificate applies to the technical report entitled **Update to the technical report on the Abcourt-Barvue deposit**.(The Technical Report)
7. I have visited the property.
8. I have prior involvement with the property that is the subject of the technical Report (Technical feasibility study report on the Abcourt-Barvue deposit, dated November 23, 2010).
9. 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.
10. I am the author of sections 13 and I have collaborated to sections 3, 17, 21, and 26 of this technical report.
11. I have read the Instrument and the sections of the Technical report that I am responsible for that have been prepared in compliance with the Instrument.
12. 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 Mines Abcourt Inc. or any associated or affiliated entities.
13. Neither I, nor any affiliated entity of mine own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Mines Abcourt Inc. or any associated or affiliated companies.
14. As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 15th day of January, 2019.at Montreal (Quebec)

Original document signed and stamped by

INGÉNIEUR - ENGINEER
FLORENT BARIL
6972
QUÉBEC

Florent Baril, P.Eng.
President
Bumigeme Inc.

615 René-Lévesque West, Room 750
Montreal, Quebec H3B 1P5
514-843-6565

APPENDICES

APPENDIX 1

Abcourt-Barvue Property Claims (ref. Section 4)

Cells Abcourt Barvue 32C12/32C05							
No titre	Statut du titre	Date d'expiration	Superficie (Ha)	Excédents	Travaux requis	Droits requis	Détenteur(s) (Nom, Numéro et Pourcentage)
CM 390	Actif		84.66				
CM 393	Actif		63.90				
CDC 2430540	Actif	13/06/2019 23:59	57.03	19,537.30	1,813.34	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430543	Actif	13/06/2019 23:59	40.24	13,785.40	1,279.48	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430544	Actif	13/06/2019 23:59	46.72	16,005.31	1,485.52	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430545	Actif	13/06/2019 23:59	57.03	16,101.24	1,494.42	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430547	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430549	Actif	13/06/2019 23:59	57.04	19,540.71	1,813.66	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430550	Actif	13/06/2019 23:59	57.08	19,554.42	1,814.93	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430551	Actif	13/06/2019 23:59	57.04	19,540.71	1,813.66	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430553	Actif	13/06/2019 23:59	57.07	19,551.00	1,814.61	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430554	Actif	13/06/2019 23:59	57.03	19,537.29	1,813.34	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430556	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC2430557	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430558	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430559	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430561	Actif	13/06/2019 23:59	56.56	19,376.28	1,798.39	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430562	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430563	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430564	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430565	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430566	Actif	13/06/2019 23:59	7.45	2,552.21	236.89	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430567	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430568	Actif	13/06/2019 23:59	54.57	18,694.55	1,735.12	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430569	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430571	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430573	Actif	13/06/2019 23:59	57.09	1,065.42	98.89	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430574	Actif	13/06/2019 23:59	3.06	1,048.29	97.30	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430575	Actif	13/06/2019 23:59	45.04	15,429.77	1,432.10	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430577	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430578	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430579	Actif	13/06/2019 23:59	29.90	3,381.26	313.83	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430580	Actif	13/06/2019 23:59	57.08	19,554.42	1,814.93	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430581	Actif	13/06/2019 23:59	57.08	3,165.43	293.80	32.77	Mines Abcourt inc. (1722) 100 % (responsable)

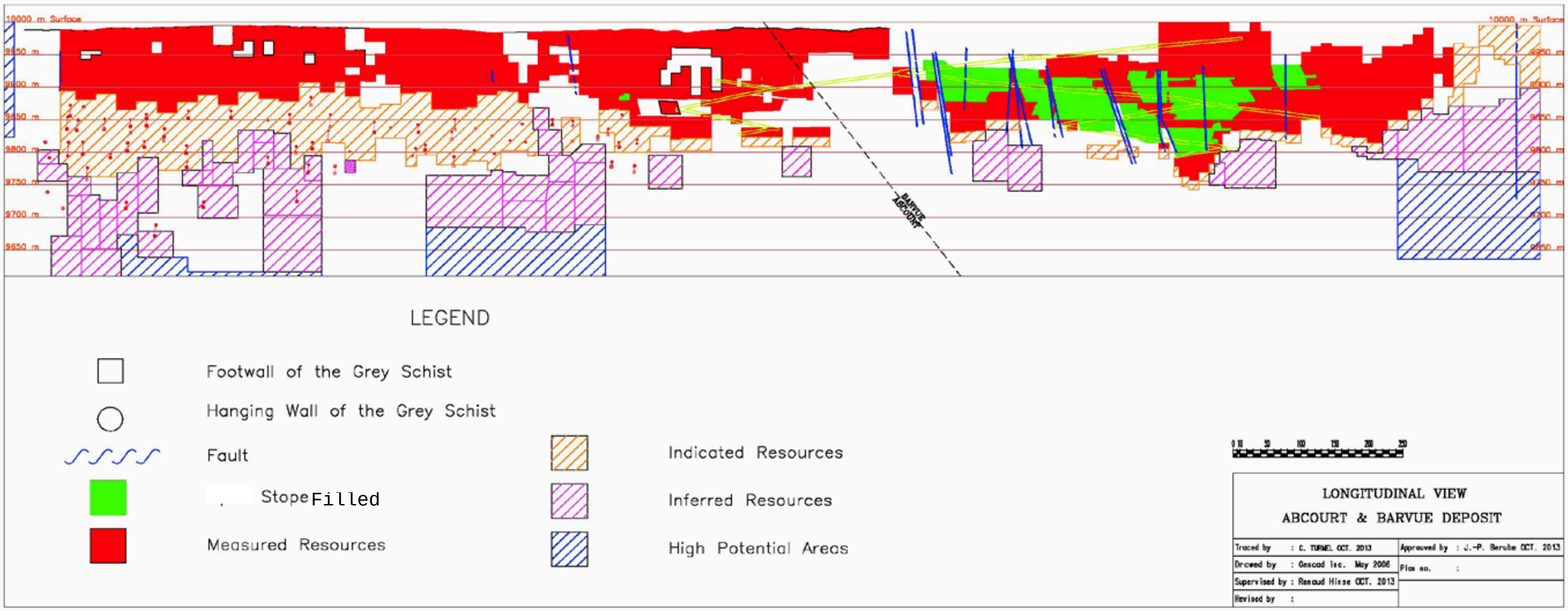
No titre	Statut du titre	Date d'expiration	Superficie (Ha)	Excédents	Travaux requis	Droits requis	Détenteur(s) (Nom, Numéro et Pourcentage)
CDC 2430583	Actif	13/06/2019 23:59	57.03	19,537.29	1,813.34	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430584	Actif	13/06/2019 23:59	57.09	6,745.38	626.07	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430585	Actif	13/06/2019 23:59	57.03	19,537.29	1,813.34	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430588	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430590	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430591	Actif	13/06/2019 23:59	54.74	18,752.79	1,740.52	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430592	Actif	13/06/2019 23:59	57.02	19,533.88	1,813.01	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430593	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430594	Actif	13/06/2019 23:59	57.07	19,551.00	1,814.61	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430595	Actif	13/06/2019 23:59	57.03	19,537.30	1,813.33	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430596	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430598	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430599	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430600	Actif	13/06/2019 23:59	57.03	19,537.30	1,813.33	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430601	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430602	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430603	Actif	13/06/2019 23:59	33.05	11,322.25	1,050.86	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430604	Actif	13/06/2019 23:59	57.01	13,607.25	1,262.94	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430605	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430606	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430607	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430608	Actif	13/06/2019 23:59	32.77	11,226.32	1,041.96	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430609	Actif	13/06/2019 23:59	57.02	6,228.10	578.05	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430610	Actif	13/06/2019 23:59	57.03	19,537.30	1,813.33	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430612	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430614	Actif	13/06/2019 23:59	57.08	19,554.43	1,814.92	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430616	Actif	13/06/2019 23:59	32.44	11,113.27	1,031.47	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430618	Actif	13/06/2019 23:59	41.47	14,206.77	1,318.58	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430622	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430623	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430624	Actif	13/06/2019 23:59	32.73	11,212.63	1,040.68	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430625	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430626	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430628	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430632	Actif	13/06/2019 23:59	57.08	19,554.43	1,814.92	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430633	Actif	13/06/2019 23:59	47.21	16,173.17	1,501.09	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430634	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)

No titre	Statut du titre	Date d'expiration	Superficie (Ha)	Excédents	Travaux requis	Droits requis	Détenteur(s) (Nom, Numéro et Pourcentage)
CDC 2430636	Actif	13/06/2019 23:59	20.55	7,040.01	653.41	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430637	Actif	13/06/2019 23:59	44.45	15,227.66	1,413.33	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430638	Actif	13/06/2019 23:59	9.89	3,388.11	314.46	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430639	Actif	13/06/2019 23:59	53.79	18,427.34	1,710.31	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430640	Actif	13/06/2019 23:59	32.82	11,243.45	1,043.55	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430641	Actif	13/06/2019 23:59	57.09	6,539.85	606.98	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430642	Actif	13/06/2019 23:59	10.49	3,593.66	333.54	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430643	Actif	13/06/2019 23:59	57.03	15,577.08	1,445.77	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430644	Actif	13/06/2019 23:59	46.07	15,782.63	1,464.85	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430646	Actif	13/06/2019 23:59	9.97	3,415.51	317.01	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430648	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430651	Actif	13/06/2019 23:59	57.09	6,697.43	621.61	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430652	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430653	Actif	13/06/2019 23:59	28.67	9,821.75	911.59	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430654	Actif	13/06/2019 23:59	32.29	11,061.89	1,026.69	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430655	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430656	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430658	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430659	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430660	Actif	13/06/2019 23:59	25.13	8,609.02	799.03	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430661	Actif	13/06/2019 23:59	32.63	11,178.37	1,037.50	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430663	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430664	Actif	13/06/2019 23:59	57.01	8,677.53	805.40	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430666	Actif	13/06/2019 23:59	43.72	14,977.57	1,390.12	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430667	Actif	13/06/2019 23:59	57.07	19,551.00	1,814.61	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430668	Actif	13/06/2019 23:59	57.06	19,547.57	1,814.29	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430669	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430670	Actif	13/06/2019 23:59	9.71	3,326.45	308.74	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430671	Actif	13/06/2019 23:59	43.70	6,680.30	620.02	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430674	Actif	13/06/2019 23:59	57.07	19,551.00	1,814.61	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430675	Actif	13/06/2019 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2430676	Actif	13/06/2019 23:59	57.04	19,540.72	1,813.65	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2435830	Actif	13/06/2017 23:59	13.76	4,713.90	437.51	32.77	Mines Abcourt inc. (1722) 100 % (responsable)
CDC 2435831	Actif	13/06/2017 23:59	57.05	19,544.15	1,813.97	64.09	Mines Abcourt inc. (1722) 100 % (responsable)
103 cells			5,123.37	1,628,957.65	151,190.04		
2 mining concessions			148.56				

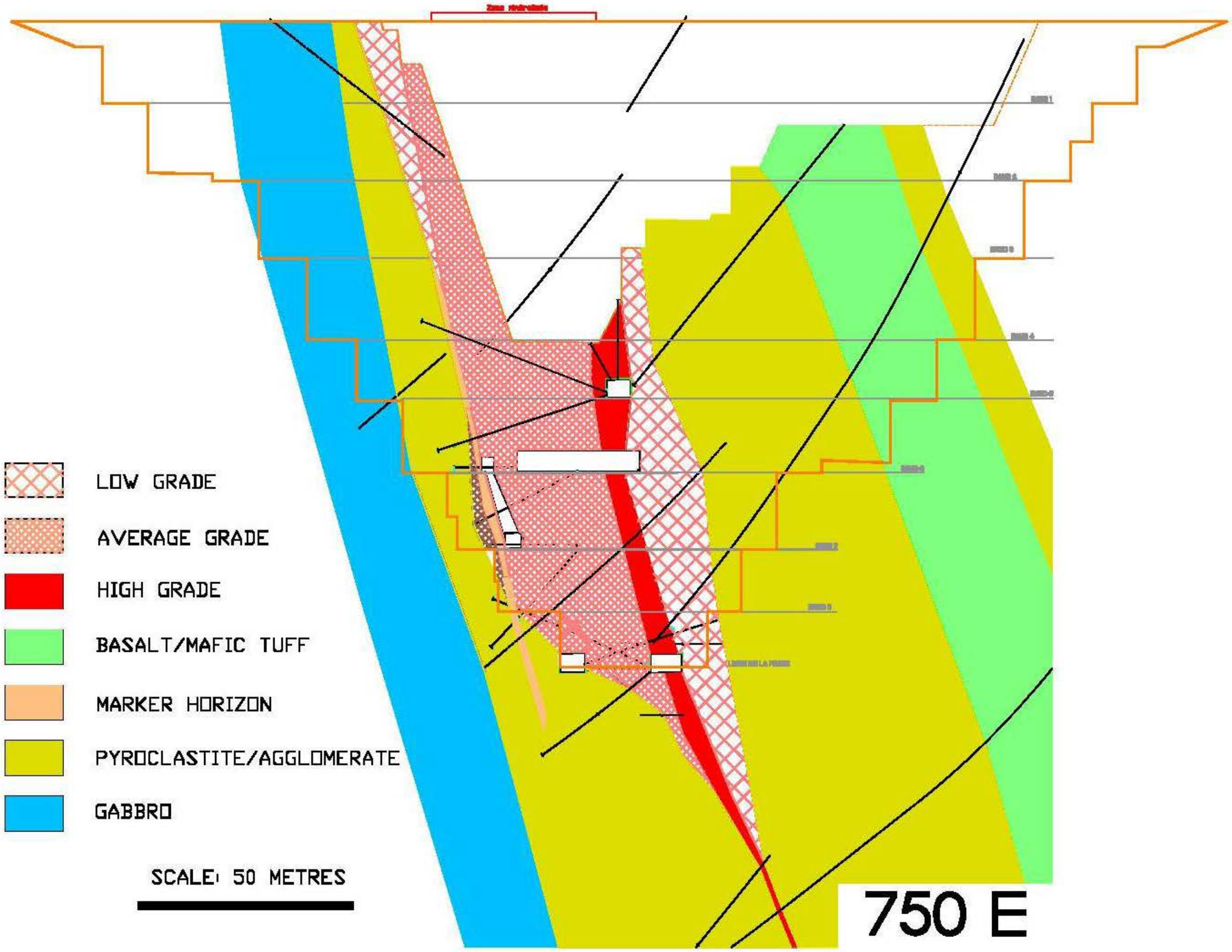
claims abcourt-barvue 08 nov 2018

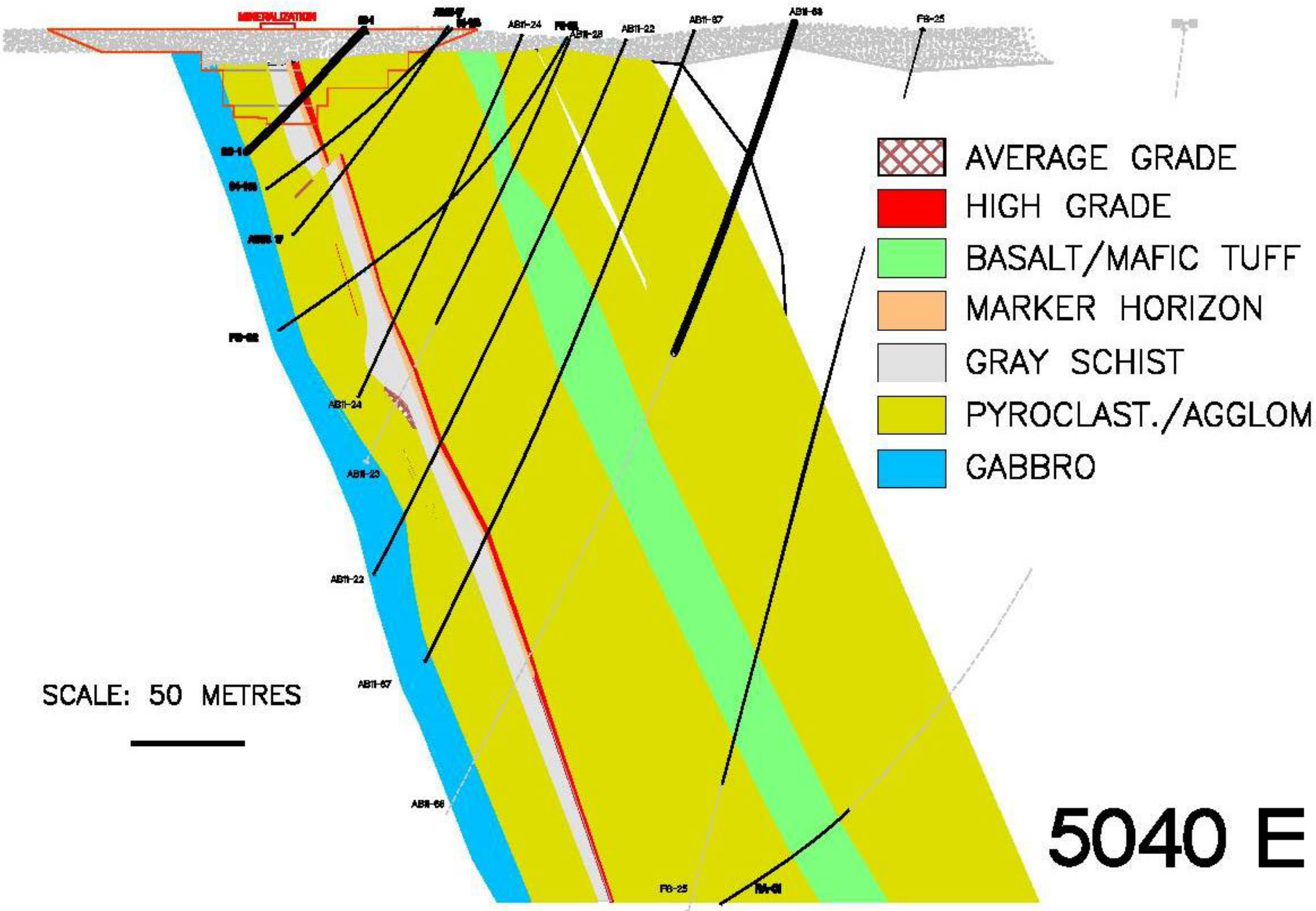
APPENDIX 2

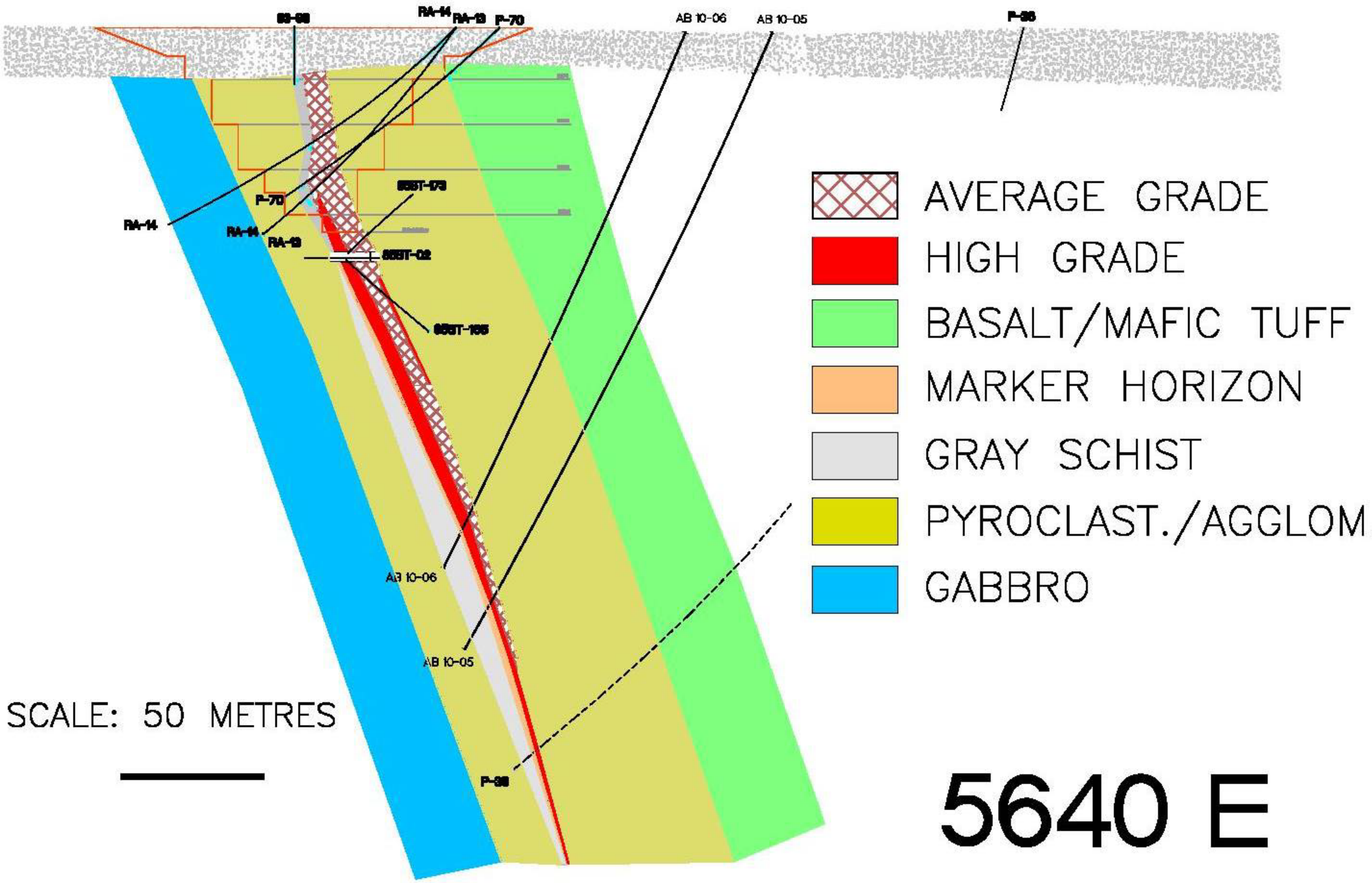
Longitudinal view and drill sections through the mineralize deposit (ref. Section 10)



10000 m Surface
9950 m
9900 m
9850 m
9800 m
9750 m
9700 m
9650 m
LEGEND
Footwall of the Grey Schist
Hanging Wall of the Grey Schist
Fault
Stope Filled
Measured Resources
Indicated Resources
Inferred Resources
High Potential Areas
LONGITUDINAL VIEW
ABCOURT & BARVUE DEPOSIT
Traced by : C. TURMEL OCT. 2013
Drawed by : Gescad inc. May 2006
Supervised by : Renaud Hinse OCT. 2013
Revised by :
Approved by : J.-P. Berube OCT. 2013
Plan no. :


LOW GRADE
AVERAGE GRADE
HIGH GRADE
BASALT/MAFIC TUFF
MARKER HORIZON
PYROCLASTITE/AGGLOMERATE
GABBRO
SCALE: 50 METRES
750 E


AVERAGE GRADE
HIGH GRADE
BASALT/MAFIC TUFF
MARKER HORIZON
GRAY SCHIST
PYROCLAST./AGGLOM.
GABBRO
SCALE: 50 METRES
5040 E
AB11-24
AB11-22
AB11-67
FS-25



RA-14
RA-13
P-70
AB 10-06
AB 10-05
AVERAGE GRADE
HIGH GRADE
BASALT/MAFIC TUFF
MARKER HORIZON
GRAY SCHIST
PYROCLAST./AGGLOM.
GABBRO
SCALE: 50 METRES
5640 E

APPENDIX 3

Production schedule (ref. Section 16)

Projet Abcourt Barvue
FS 2007 Update
Calendrier de production
Tonnes des fosses avec sommaire du sous terre

Banc #	Élévation plancher (m)	Année -1 Minerai (t)	Année -1 Minerai marginal (t)	Année -1 Stériles (t)	Année 1 Minerai (t)	Année 1 Minerai marginal (t)	Année 1 Stériles (t)	Année 2 Minerai (t)	Année 2 Minerai marginal (t)	Année 2 Stériles (t)	Année 3 Minerai (t)	Année 3 Minerai marginal (t)	Année 3 Stériles (t)
Mort-Terrain													
Fosse Barvue (m3)				200,000									250,000
Fosse Abcourt Est (m3)							1,234,000						
Fosse Abcourt Ouest (m3)				236,000									
Total Mort-Terrain				436,000			1,234,000			0			250,000
Production des fosses													
Fosse Barvue (5%)													
1	9990	60,433	17,360	420,372									428,139
	9981				110,669	28,307	1,321,279						1,146,663
2	9972				154,976	25,665	1,215,628						
	9963							155,893	27,424	1,362,307			
3	9954							197,966	28,355	1,206,397			
	9945							0	0	0	210,260	33,625	1,343,886
4	9936										69,411	7,630	439,371
	9926										0	0	0
5	9918												
	9911												
6	9902												
	9894												
7	9885												
	9877												
8	9870												
	9863												
9	9857												
	9850												
10	9842												
	9834												
Fosse Abcourt Est (10%)													
1	9986				63,984	5,747	899,127						
	9982				113,570	7,651	843,672						
2	9974				67,695	4,687	424,120	135,389	9,374	848,241			
	9966							101,173	5,485	609,430	101,173	5,485	609,430
3	9958							70,000	2,463	270,279	123,613	4,350	477,286
	9950										79,403	4,641	335,496
4	9942												
	9934												
5	9928												
Fosse Abcourt Ouest (10%)													
1	9986	5,748	0	121,747									
	9982	8,531	0	109,647									
2	9974	17,676	0	125,954									
	9966	15,231	0	88,244									
3	9958	14,546	0	37,937									
	9950												
Total - Excavé des Fosse		122,164	17,360	903,902	510,894	72,056	4,703,826	660,421	73,101	4,296,654	583,860	55,731	4,780,270
Fosse Barvue (5%)		60,433	17,360	420,372	265,645	53,972	2,536,907	353,859	55,778	2,568,704	279,671	41,255	3,358,059
Fosse Abcourt Est (10%)		0	0	0	245,248	18,084	2,166,919	306,562	17,322	1,727,950	304,189	14,476	1,422,211
Fosse Abcourt Ouest (10%)		61,731	0	483,530	0	0	0	0	0	0	0	0	0
Total stériles & marginal				921,261			4,775,883			4,369,755			4,836,001
Total roc tout venant				1,043,426			5,286,776			5,030,176			5,419,860
Roche à récupérer au fond actuel de la fosse Barvue													
1	9990	4,000	6,000	17,340	0	0	0	0	0	0	0	0	17,660
	9981	0	0	0	6,000	9,000	41,224	0	0	0	0	0	35,776
2	9972	0	0	0	6,500	4,250	38,792	0	0	0	0	0	0
	9963	0	0	0	0	0	0	6,500	4,750	39,874	0	0	0
3	9954	0	0	0	0	0	0	7,500	3,500	33,803	0	0	0
	9945	0	0	0	0	0	0	0	0	0	15,500	4,500	34,577
Total à récupérer		4,000	6,000	17,340	12,500	13,250	80,016	14,000	8,250	73,678	15,500	4,500	88,014
Dilution additionnelle 5%		187	280	-467	583	618	-1,202	653	385	-1,038	723	210	-933
Total à récupérer - dilué		4,187	6,280	16,873	13,083	13,868	78,814	14,653	8,635	72,639	16,223	4,710	87,080
Total récupéré au fond - dilué 5% - Minerai et Minerai Marginal Combiné											81,640		255,407
Total Excavé & récupéré - Fosse Barvue		118,164	11,360	886,562	498,394	58,806	4,623,811	646,421	64,851	4,222,976	650,000	51,231	4,947,664
Total stériles & marginal				897,922			4,682,617			4,287,827			4,998,894
Pit Ore Stockpile													
Ore inventory @ Period Start		0	0		118,164	11,360		20,558	70,166		16,979	135,017	
Ore to stockpile		118,164	11,360		0	58,806		0	64,851		0	51,231	
Ore from stockpile to mill		0	0		97,606	0		3,579	0		0	0	
Ore Inventory @ Period End		118,164	11,360		20,558	70,166		16,979	135,017		16,979	186,247	
Total à l'usine venant des Fosses													
Venant des excavation des fosses		0	0		498,394	0		646,421	0		568,360	0	
Récupéré du fond de la fosse Barvue		0	0		0	0		0	0		81,640	0	
Venant du Stockpile		0	0		97,606	0		3,579	0		0	0	
Total des fosses à l'usine		0	0		596,000	0		650,000	0		650,000	0	
Production Souterraine													
Développement													
Production													
Total Sous-terre													
Total Excavé et récup. Fosse et Sous-terre		118,164	11,360	886,562	498,394	58,806	4,623,811	646,421	64,851	4,222,976	650,000	51,231	4,947,664
Alimentation à l'usine													
Fosses		0	0		596,000	0		650,000	0		650,000	0	
Sous-Terre		0	0		0	0		0	0		0	0	
Total		0	0		596,000	0		650,000	0		650,000	0	

Projet Abcourt Barvue
FS 2007 Update
Calendrier de production
Tonnes des fosses avec sommaire (

		Année 4			Année 5			Année 6			Année 7		
Banc #	Élévation plancher (m)	Minerai (t)	Minerai marginal (t)	Stériles (t)	Minerai (t)	Minerai marginal (t)	Stériles (t)	Minerai (t)	Minerai marginal (t)	Stériles (t)	Minerai (t)	Minerai marginal (t)	Stériles (t)
Mort-Terrain													
Fosse Barvue (m3)													
Fosse Abcourt Est (m3)													
Fosse Abcourt Ouest (m3)													
Total Mort-Terrain				0			0			0			0
Production des fosses													
Fosse Barvue (5%)													
1	9990												
	9981												
2	9972			1,040,652									
	9963			1,097,572									
3	9954			970,615									
	9945						1,026,940						
4	9936	120,000	13,191	759,595			924,863						
	9926	221,134	18,355	770,653	113,732	9,440	396,356			951,507			
5	9918	0	0	0	500,000	23,574	1,544,400	13,153	620	40,627			
	9911							315,331	16,487	1,588,256			
6	9902							304,554	6,058	662,343	292,018	5,809	635,079
	9894										224,817	8,740	1,319,729
7	9885										113,166	3,677	207,317
	9877												
8	9870												
	9863												
9	9857												
	9850												
10	9842												
	9834												
Fosse Abcourt Est (10%)													
1	9986												
	9982												
2	9974												
	9966												
3	9958	0	0	0									
	9950	88,601	5,178	374,360	0	0	0						
4	9942	150,266	8,557	215,514	0	0	0						
	9934	70,000	3,169	112,361	17,954	813	28,818						
5	9928				18,298	3,332	35,095						
Fosse Abcourt Ouest (10%)													
1	9986												
	9982												
2	9974												
	9966												
3	9958												
	9950												
Total - Excavé des Fosse		650,000	48,450	5,341,322	649,983	37,159	3,956,472	633,038	23,165	3,242,733	630,000	18,226	2,162,125
Fosse Barvue (5%)		341,133	31,546	4,639,088	613,732	33,014	3,892,559	633,038	23,165	3,242,733	630,000	18,226	2,162,125
Fosse Abcourt Est (10%)		308,867	16,904	702,234	36,252	4,145	63,913	0	0	0	0	0	0
Fosse Abcourt Ouest (10%)		0	0	0	0	0	0	0	0	0	0	0	0
Total stériles & marginal				5,389,772			3,993,631			3,265,898			2,180,351
Total roc tout venant				6,039,772			4,643,615			3,898,936			2,810,351
Roche à récupérer au fond actuel de la fosse B													
1	9990	0	0	0	0	0	0	0	0	0	0	0	0
	9981	0	0	0	0	0	0	0	0	0	0	0	0
2	9972	0	0	33,208	0	0	0	0	0	0	0	0	0
	9963	0	0	32,126	0	0	0	0	0	0	0	0	0
3	9954	0	0	27,197	0	0	0	0	0	0	0	0	0
	9945	0	0	0	0	0	26,423	0	0	0	0	0	0
Total à récupérer		0	0	92,531									
Dilution additionnelle 5%		0	0	0									
Total à récupérer - dilué		0	0	92,531									
Total récupéré au fond - dilué 5% - Minerai et				92,531									
Total Excavé & récupéré - Fosse Barvue		650,000	48,450	5,341,322	649,983	37,159	3,956,472	633,038	23,165	3,242,733	630,000	18,226	2,162,125
Total stériles & marginal				5,389,772			3,993,631			3,265,898			2,180,351
Pit Ore Stockpile													
Ore inventory @ Period Start		16,979	186,247		16,979	234,697		16,962	271,856		0	295,021	
Ore to stockpile		0	48,450		0	37,159		0	23,165		0	18,226	
Ore from stockpile to mill		0	0		17	0		16,962	0		0	0	
Ore Inventory @ Period End		16,979	234,697		16,962	271,856		0	295,021		0	313,247	
Total à l'usine venant des Fosses													
Venant des excavation des fosses		650,000	0		649,983	0		633,038	0		630,000	0	
Récupéré du fond de la fosse Barvue		0	0		0	0		0	0		0	0	
Venant du Stockpile		0	0		17	0		16,962	0		0	0	
Total des fosses à l'usine		650,000	0		650,000	0		650,000	0		630,000	0	
Production Souterraine													
Développement											20,000		
Production													
Total Sous-terre								0			20,000		
Total Excavé et récup. Fosse et Sous-terre		650,000	48,450	5,341,322	649,983	37,159	3,956,472	633,038	23,165	3,242,733	650,000	18,226	2,162,125
Alimentation à l'usine													
Fosses		650,000	0		650,000	0		650,000	0		630,000	0	
Sous-Terre		0	0		0	0		0	0		20,000	0	
Total		650,000	0		650,000	0		650,000	0		650,000	0	

Projet Abcourt Barvue
FS 2007 Update
Calendrier de production
Tonnes des fosses avec sommaire c

Banc #	Élévation plancher (m)	Année 8 Minerai (t)	Année 8 Minerai marginal (t)	Année 8 Stériles (t)	Année 9 Minerai (t)	Année 9 Minerai marginal (t)	Année 9 Stériles (t)	Année 10 Minerai (t)	Année 10 Minerai marginal (t)	Année 10 Stériles (t)	Année 11 Minerai (t)	Année 11 Minerai marginal (t)	Année 11 Stériles (t)
Mort-Terrain													
Fosse Barvue (m3)													
Fosse Abcourt Est (m3)													
Fosse Abcourt Ouest (m3)													
Total Mort-Terrain				0			0			0			0
Production des fosses													
Fosse Barvue (5%)													
1	9990												
	9981												
2	9972												
	9963												
3	9954												
	9945												
4	9936												
	9926												
5	9918												
	9911												
6	9902												
	9894												
7	9885	398,793	12,956	730,581									
	9877	135,938	4,821	348,692	191,804	6,802	491,994						
8	9870				134,196	5,677	176,672	212,403	8,986	279,634			
	9863							113,597	5,178	179,343	223,957	10,209	353,577
9	9857										102,043	3,811	91,728
	9850												
10	9842												
	9834												
Fosse Abcourt Est (10%)													
1	9986												
	9982												
2	9974												
	9966												
3	9958												
	9950												
4	9942												
	9934												
5	9928												
Fosse Abcourt Ouest (10%)													
1	9986												
	9982												
2	9974												
	9966												
3	9958												
	9950												
Total - Excavé des Fosse		534,731	17,777	1,079,274	326,000	12,479	668,666	326,000	14,164	458,977	326,000	14,020	445,305
Fosse Barvue (5%)		534,731	17,777	1,079,274	326,000	12,479	668,666	326,000	14,164	458,977	326,000	14,020	445,305
Fosse Abcourt Est (10%)		0	0	0	0	0	0	0	0	0	0	0	0
Fosse Abcourt Ouest (10%)		0	0	0	0	0	0	0	0	0	0	0	0
Total stériles & marginal				1,097,051			681,145			473,142			459,325
Total roc tout venant				1,631,782			1,007,145			799,142			785,325
Roche à récupérer au fond actuel de la fosse B													
1	9990	0	0	0	0	0	0	0	0	0	0	0	0
	9981	0	0	0	0	0	0	0	0	0	0	0	0
2	9972	0	0	0	0	0	0	0	0	0	0	0	0
	9963	0	0	0	0	0	0	0	0	0	0	0	0
3	9954	0	0	0	0	0	0	0	0	0	0	0	0
	9945	0	0	0	0	0	0	0	0	0	0	0	0
Total à récupérer													
Dilution additionnelle 5%													
Total à récupérer - dilué													
Total récupéré au fond - dilué 5% - Minerai et													
Total Excavé & récupéré - Fosse Barvue		534,731	17,777	1,079,274	326,000	12,479	668,666	326,000	14,164	458,977	326,000	14,020	445,305
Total stériles & marginal				1,097,051			681,145			473,142			459,325
Pit Ore Stockpile													
Ore inventory @ Period Start		0	313,247		0	331,024		0	343,503		0	357,668	
Ore to stockpile		0	17,777		0	12,479		0	14,164		0	14,020	
Ore from stockpile to mill		0	0		0	0		0	0		0	0	
Ore Inventory @ Period End		0	331,024		0	343,503		0	357,668		0	371,688	
Total à l'usine venant des Fosses													
Venant des excavation des fosses		534,731	0		326,000	0		326,000	0		326,000	0	
Récupéré du fond de la fosse Barvue		0	0		0	0		0	0		0	0	
Venant du Stockpile		0	0		0	0		0	0		0	0	
Total des fosses à l'usine		534,731	0		326,000	0		326,000	0		326,000	0	
Production Souterraine													
Développement		60,000			60,000			60,000			60,000		
Production		55,269			264,000			264,000			264,000		
Total Sous-terre		115,269			324,000			324,000			324,000		
Total Excavé et récup. Fosse et Sous-terre		650,000	17,777	1,079,274	650,000	12,479	668,666	650,000	14,164	458,977	650,000	14,020	445,305
Alimentation à l'usine													
Fosses		534,731	0		326,000	0		326,000	0		326,000	0	
Sous-Terre		115,269	0		324,000	0		324,000	0		324,000	0	
Total		650,000	0		650,000	0		650,000	0		650,000	0	

Projet Abcourt Barvue
FS 2007 Update
Calendrier de production
Tonnes des fosses avec sommaire (

Banc #	Élévation plancher (m)	Année 12 Minerai (t)	Année 12 Minerai marginal (t)	Année 12 Stériles (t)	Année 13 Minerai (t)	Année 13 Minerai marginal (t)	Année 13 Stériles (t)	Total Minerai (t)	Total Minerai marginal (t)	Total Stériles (t)
Mort-Terrain										
Fosse Barvue (m3)										450,000
Fosse Abcourt Est (m3)										1,234,000
Fosse Abcourt Ouest (m3)										236,000
Total Mort-Terrain				0			0			1,920,000
Production des fosses										
Fosse Barvue (5%)										
1	9990							60,433	17,360	848,511
	9981							110,669	28,307	2,467,942
2	9972							154,976	25,665	2,256,280
	9963							155,893	27,424	2,459,879
3	9954							197,966	28,355	2,177,012
	9945							210,260	33,625	2,370,826
4	9936							189,411	20,822	2,123,829
	9926							334,865	27,795	2,118,516
5	9918							513,153	24,194	1,585,028
	9911							315,331	16,487	1,588,256
6	9902							596,572	11,867	1,297,422
	9894							224,817	8,740	1,319,729
7	9885							511,959	16,633	937,898
	9877							327,742	11,623	840,686
8	9870							346,599	14,664	456,306
	9863							337,554	15,388	532,920
9	9857	140,066	5,231	125,907				242,109	9,042	217,635
	9850	201,593	8,806	318,445				201,593	8,806	318,445
10	9842	43,941	0	20,920	125,610	0	59,801	169,550	0	80,721
	9834				89,420	0	100,877	89,420	0	100,877
Fosse Abcourt Est (10%)										
1	9986							63,984	5,747	899,127
	9982							113,570	7,651	843,672
2	9974							203,084	14,061	1,272,361
	9966							202,346	10,970	1,218,860
3	9958							193,613	6,813	747,565
	9950							168,004	9,819	709,855
4	9942							150,266	8,557	215,514
	9934							87,954	3,981	141,179
5	9928							18,298	3,332	35,095
Fosse Abcourt Ouest (10%)										
1	9986							5,748	0	121,747
	9982							8,531	0	109,647
2	9974							17,676	0	125,954
	9966							15,231	0	88,244
3	9958							14,546	0	37,937
	9950							0	0	0
Total - Excavé des Fosse		385,600	14,037	465,271	215,030	0	160,678	6,553,721	417,725	32,665,476
Fosse Barvue (5%)		385,600	14,037	465,271	215,030	0	160,678	5,290,872	346,794	26,098,718
Fosse Abcourt Est (10%)		0	0	0	0	0	0	1,201,118	70,931	6,083,228
Fosse Abcourt Ouest (10%)		0	0	0	0	0	0	61,731	0	483,530
Total stériles & marginal				479,308			160,678			33,083,201
Total roc tout venant				864,908			375,708			39,636,922
Roche à récupérer au fond actuel de la fosse B										
1	9990	0	0	0	0	0	0	4,000	6,000	35,000
	9981	0	0	0	0	0	0	6,000	9,000	77,000
2	9972	0	0	0	0	0	0	6,500	4,250	72,000
	9963	0	0	0	0	0	0	6,500	4,750	72,000
3	9954	0	0	0	0	0	0	7,500	3,500	61,000
	9945	0	0	0	0	0	0	15,500	4,500	61,000
Total à récupérer								46,000	32,000	351,577
Dilution additionnelle 5%								2,147	1,493	-3,640
Total à récupérer - dilué								48,147	33,493	347,937
Total récupéré au fond - dilué 5% - Minerai et								81,640	0	347,937
Total Excavé & récupéré - Fosse Barvue		385,600	14,037	465,271	215,030	0	160,678	6,589,361	385,725	32,661,836
Total stériles & marginal				479,308			160,678			33,047,561
Pit Ore Stockpile										
Ore inventory @ Period Start		0	371,688		0	385,725		0	0	
Ore to stockpile		0	14,037		0	0		118,164	385,725	
Ore from stockpile to mill		0	0		0	0		118,165	0	
Ore Inventory @ Period End		0	385,725		0	385,725		0	385,725	
Total à l'usine venant des Fosses										
Venant des excavation des fosses		385,600	0		215,030	0		6,389,556	0	
Récupéré du fond de la fosse Barvue		0	0		0	0		81,640	0	
Venant du Stockpile		0	0		0	0		118,165	0	
Total des fosses à l'usine		385,600	0		215,030	0		6,589,361	0	
Production Souterraine										
Développement		400			0			260,400		
Production		264,000			113,132			1,224,401		
Total Sous-terre		264,400			113,132			1,484,801		
Total Excavé et récup. Fosse et Sous-terre		650,000	14,037	465,271	328,162	0	160,678	8,074,162	385,725	32,661,836
Alimentation à l'usine										
Fosses		385,600	0		215,030	0		6,589,361	0	
Sous-Terre		264,400	0		113,132	0		1,484,801	0	
Total		650,000	0		328,162	0		8,074,162	0	

APPENDIX 4

Flotation and cyanidation tests on Abcourt-Barvue ore

By Jean Lelièvre, ing. MSc.

URSTM

2018

Rapport final
PU-2017-07-1144-B

Partie B :

Essais de flottation et de cyanuration sur le minerai Abcourt-Barvue

Présenté à :

M. Renaud Hinse, président
Mines Abcourt
506, rue des Falaises
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MARS 2018

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1. Mandat

Ce rapport fait suite à une demande de M. Renaud Hinse, président de Mines Abcourt, afin de réaliser des essais de flottation du zinc et de l'argent sur le minerai Abcourt-Barvue de même que des essais de cyanuration sur le concentré de pyrite. Le gisement Abcourt-Barvue est situé en Abitibi, à environ 40 km à l'est de la ville d'Amos (Qc).

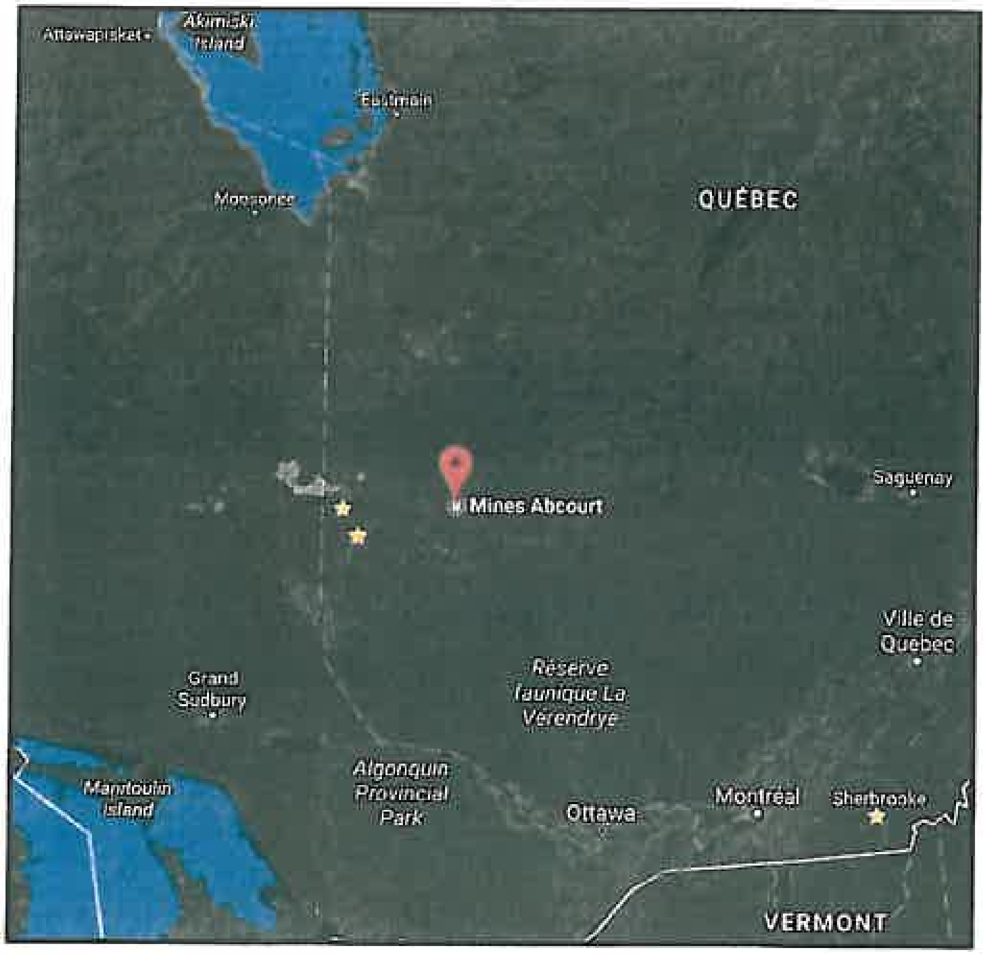


Figure 1 : Localisation du projet **Abcourt-Barvue**

2. Objectif

Le principal objectif des essais de flottation était de maximiser la récupération de l'argent dans le concentré de zinc et de produire un concentré de sulfures (pyrite-Ag) sur lequel pourrait être effectuée une cyanuration. Cette démarche s'inscrit dans un processus visant à évaluer la faisabilité économique de la récupération de l'argent dans le concentré, par un affineur de zinc. Le *tableau 1* présente un aperçu des essais réalisés.

Tableau 1 : Sommaire des essais réalisés dans le présent projet

Essais	Description
AB-62	Essai de flottation cinétique sur le 2e échantillon de minerai de façon à préciser les durées de flottation
AB-63	Essai de flottation cyclique (6 cycles) sur le 2e échantillon de minerai
AB-64	Essai de flottation cyclique (4 cycles) sur le 2e échantillon de minerai avec schéma de flottation légèrement modifié
AB-64-CN-A-4	Essai de cyanuration effectué sur le concentré de pyrite de l'essai de flottation cyclique AB-64 (cycle 4) (sans rebroyage)
AB-64-CN-B-4	Essai de cyanuration effectué sur le concentré de pyrite de l'essai de flottation cyclique AB-64 (cycle 4) (avec rebroyage)
AB-64-CN-A-3	Essai de cyanuration effectué sur le concentré de pyrite de l'essai de flottation cyclique AB-64 (cycle 3) (sans rebroyage)
AB-64-CN-B-3	Essai de cyanuration effectué sur le concentré de pyrite de l'essai de flottation cyclique AB-64 (cycle 3) (avec rebroyage)

Les essais métallurgiques ont été réalisés en août et octobre 2017 dans les laboratoires du Cégep de l'Abitibi-Témiscamingue, par Jean Lelièvre, ing., M. Sc., pour l'Unité de recherche et de service en technologie minérale (URSTM) de l'Université du Québec en Abitibi-Témiscamingue (UQAT). Les analyses ont été effectuées par le Laboratoire Expert de Rouyn-Noranda (Qc), Multilab de Rouyn-Noranda (Qc), Actlabs de Ste-Germaine-Boulé (Qc) et d'Actlabs de Ancaster (On).

3. Description des échantillons reçus

Les échantillons Abcourt-Barvue, sous forme de carottes de forage, ont été reçus aux laboratoires de l'URSTM. Ils ont ensuite été soumis à une procédure standard de concassage et de division pour produire finalement des lots de 2 kg et de 1 kg de minerai. Le schéma de la *figure 2* donne une vue d'ensemble du protocole utilisé pour la préparation du minerai.

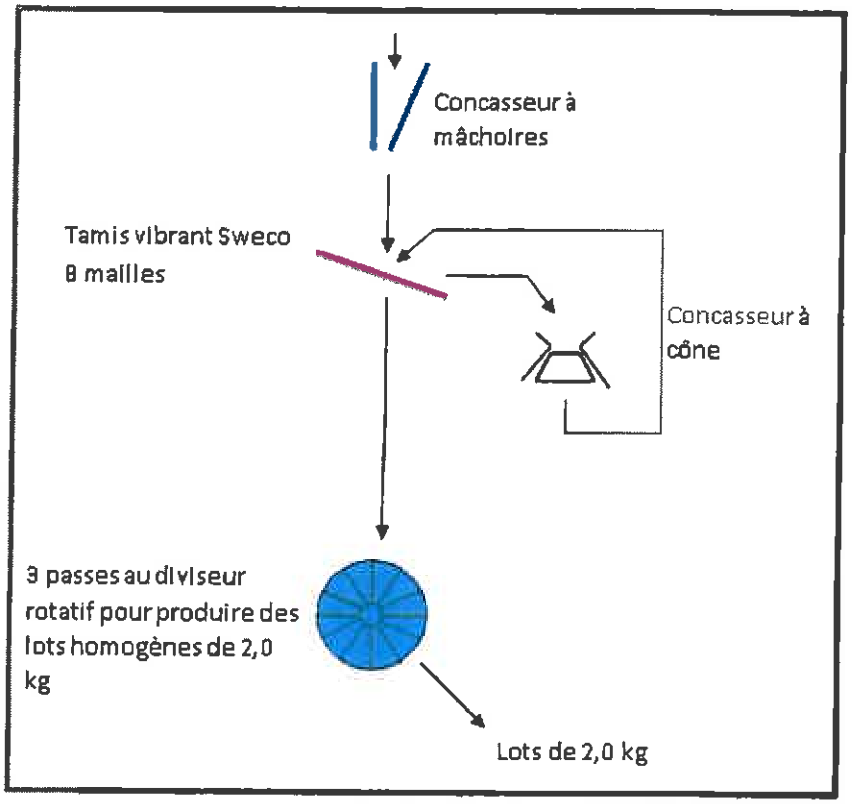


Figure 2 : Schéma de concassage et de division des lots de 2 kg

4. Essais de flottation

4.1 Essai cyclique de flottation AB-49 (4 cycles)

L'essai cyclique de flottation AB-49 a été réalisé sur le premier échantillon reçu de minerai Abcourt-Barvue en s'inspirant du protocole de flottation utilisé dans le projet PU-2007-12-356, réalisé par l'URSTM en 2007.

Essai:	**AB-49**
Description:	Essai cyclique de flottation du minerai Abcourt-Barvue (4 cycles) (1er échantillon)
Granulométrie:	D80 = 56 microns 91% < 200 mailles
Date:	15-août-2017

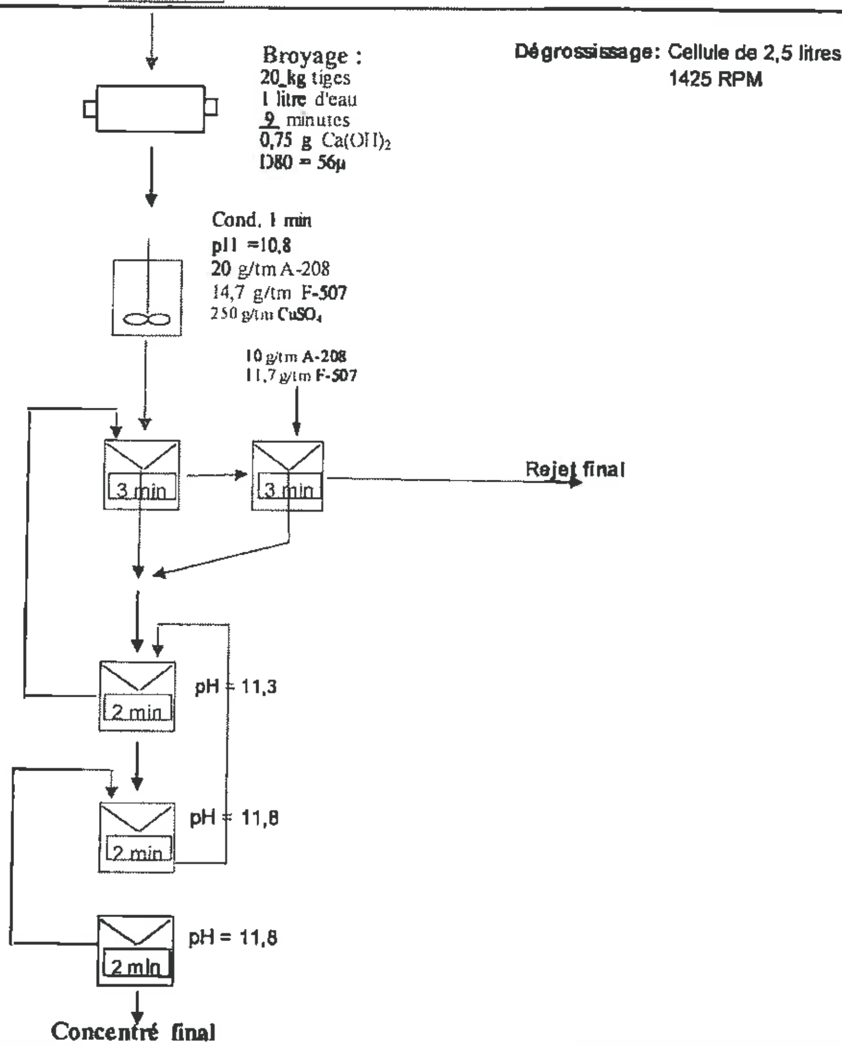


Figure 3 : Schéma de l'essai cyclique de flottation AB-49 (4 cycles)

Tableau 2 : Bilan métallurgique de l'essai cyclique AB-49 (moyenne des cycles 3 et 4)

	Masse (g)	% massique	Teneur		Unités		% distribution	
			Ag (g/tm)	% Zn	mg Ag	g Zn	Ag	Zn
Concentré final Zn-Ag	61,5 g	6,1%	631,1	51,0%	38,81	31,4	**69,6%**	**96,5%**
Rejet final	942,8 g	93,9%	18,0	0,12%	16,92	1,1	**30,4%**	**3,5%**
Alimentation calc.	1004,3 g	100,0%	55,5	3,24%	55,73	32,50	100,0%	100,0%

- Récupération Ag dans le concentré Zn-Ag : 69,6 %
- Récupération Zn dans le concentré Zn-Ag : 96,5 %

- Teneur Zn du concentré Zn-Ag : 51,0 % Zn
- Teneur Ag du concentré Zn-Ag : 631,1 g Ag / tm

- % massique au concentré Zn-Ag : 6,1 %

On constate une récupération de seulement 69,6 % de l'argent dans le concentré Zn-Ag, ce qui est inférieur aux attentes et aux récupérations obtenues en 2007 dans le projet PU-2007-12-356 où une récupération de 79 % avait été obtenue.

Ces résultats laissent supposer que la durée de flottation utilisée de 6 min n'est pas suffisante pour atteindre une récupération en argent satisfaisante.

4.2 Essai cinétique de flottation AB-62

Compte tenu des récupérations en argent non satisfaisantes obtenues dans l'essai cyclique AB-49, un essai de flottation cinétique a été réalisé sur le minerai de façon à préciser la durée ultime de flottation permettant de maximiser la récupération du zinc et de l'argent dans le concentré Zn-Ag. Cet essai était nécessaire afin de tenter de hausser la récupération en argent et de valider les durées de flottation sur le deuxième échantillon de minerai Abcourt-Barvue que reçu.

Essai: AB-62

Description: **Essai cinétique de flottation**

Rejet flottation: K80 = 74 microns

Date: 7 septembre 2017

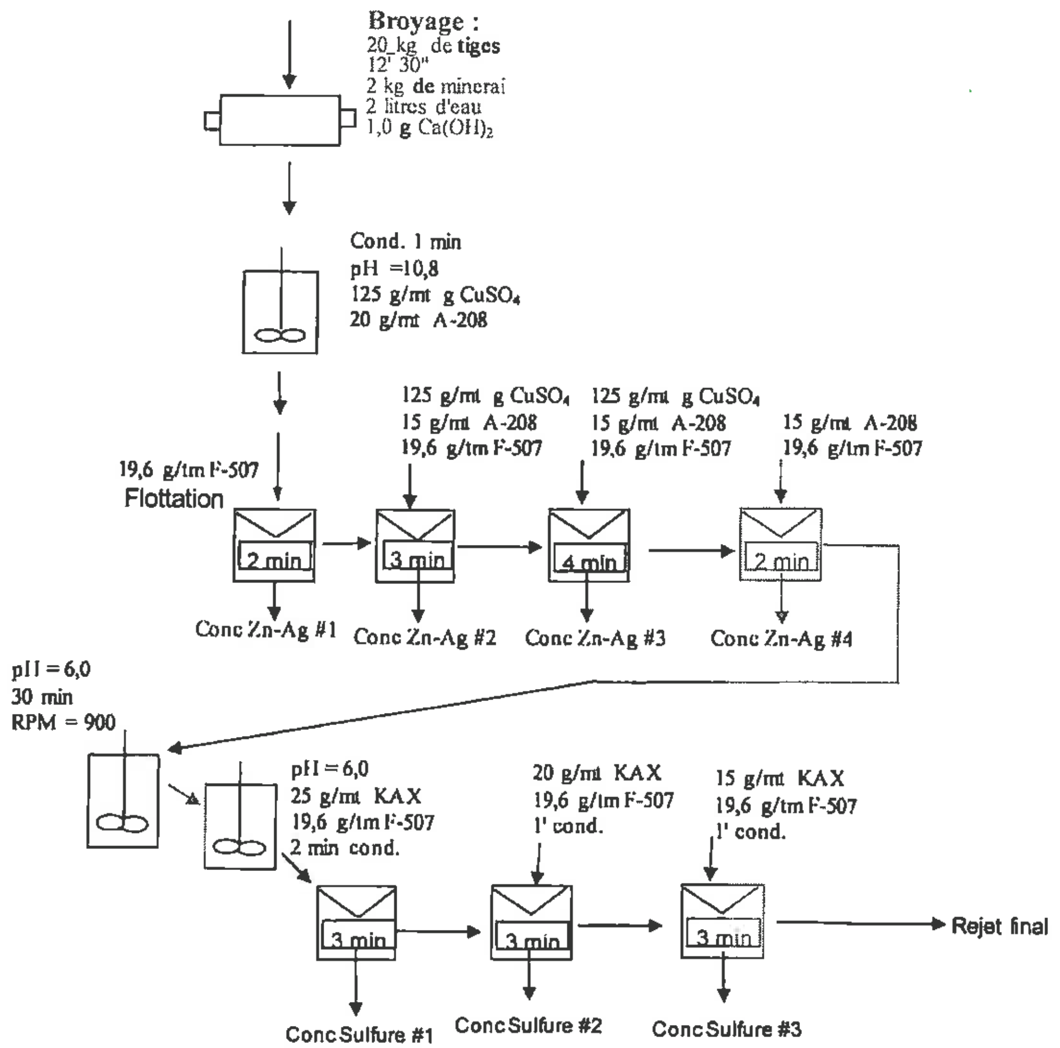


Figure 4 : Schéma de flottation de l'essai AB-62

Tableau 3 : Bilan métallurgique (Ag) de l'essai AB-62

	Masse	% massique	Teneur (g Ag/tm)	mg Ag	% distribution	% distribution cumulatif	Durée cumulative flottation (minutes)	Code analyse
Concentré Zn-Ag #1	212,4	10,65%	449,75	95,53 mg Ag	68,8%	68,8%	2,0	S-107
Concentré Zn-Ag #2	89,8	4,51%	130,15	11,70 mg Ag	8,4%	77,2%	6,0	S-106
Concentré Zn-Ag #3	48,3	2,42%	84,75	4,09 mg Ag	2,9%	80,2%	9,0	S-105
Concentré Zn-Ag #4	31,3	1,57%	62,40	1,95 mg Ag	1,4%	81,6%	11,0	S-104
Concentré Sulfure 1	99,2	4,97%	58,85	5,84 mg Ag	4,2%	85,8%	3,0	S-103
Concentré Sulfure 2	51,2	2,57%	31,55	1,62 mg Ag	1,2%	86,9%	6,0	S-102
Concentré Sulfure 3	34,8	1,74%	24,70	0,86 mg Ag	0,6%	87,6%	9,0	S-101
Rejet final	1428,0	71,58%	12,10	17,28 mg Ag	12,4%	100,0%		S-100
Alimentation calculée	1995,1	100%	69,60	138,87 mg Ag	100%			

Tableau 4 : Bilan métallurgique (Zn) **de l'essai** AB-62

	Masse	% massique	Teneur (% Zn)	g Zn	% distribution	% distribution cumulatif	Durée cumulative flottation (minutes)	Code analyse
Concentré Zn-Ag #1	212,4	10,65%	25,00%	53,10 g Zn	82,2%	82,2%	2,0	S-107
Concentré Zn-Ag #2	89,9	4,51%	9,50%	8,54 g Zn	13,2%	95,4%	5,0	S-106
Concentré Zn-Ag #3	48,3	2,42%	1,58%	0,76 g Zn	1,2%	96,6%	9,0	S-105
Concentré Zn-Ag #4	31,3	1,57%	0,68%	0,21 g Zn	0,3%	96,9%	11,0	S-104
Concentré Sulfure 1	99,2	4,97%	0,23%	0,23 g Zn	0,4%	97,2%	3,0	S-103
Concentré Sulfure 2	51,2	2,57%	0,25%	0,13 g Zn	0,2%	97,4%	6,0	S-102
Concentré Sulfure 3	34,8	1,74%	0,27%	0,09 g Zn	0,1%	97,6%	9,0	S-101
Rejet final	1428,0	71,58%	0,11%	1,57 g Zn	2,4%	100,0%		S-100
Alimentation calculée	1995,1	100%	3,24%	64,64 g Zn	100%			

La *figure 5* présente le graphique de la récupération cumulative du zinc et de l'argent dans le concentré Zn-Ag en fonction de la durée de la flottation. Ce graphique montre bien la cinétique de flottation plus rapide du zinc relativement à celle de l'argent. La flottation de l'argent est un peu plus lente et exige une durée de flottation plus longue.

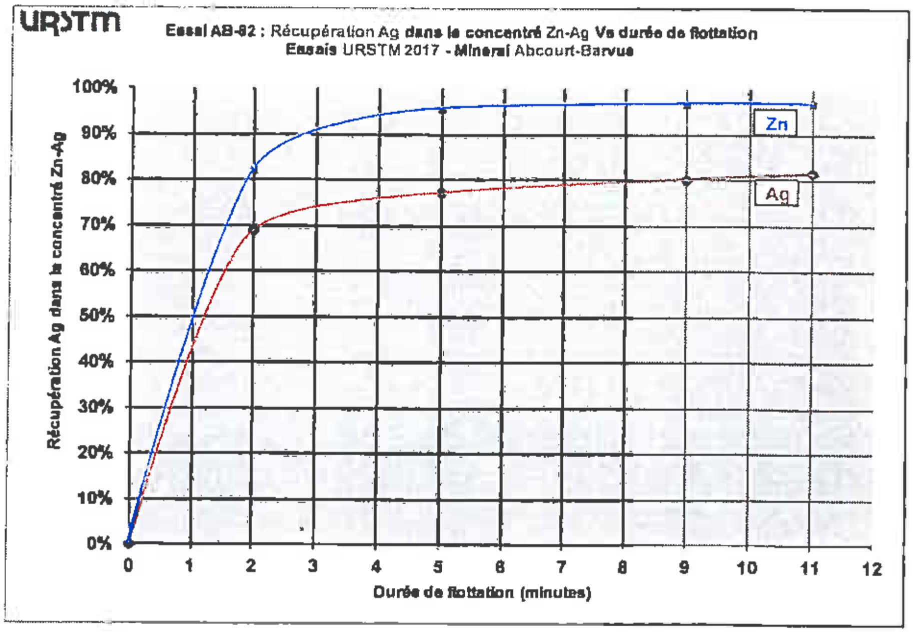


Figure 5 : Graphique de la récupération du zinc et de l'argent Vs durée de flottation pour l'essai AB-62

Les données de la *figure 6* permettent, pour l'argent, d'estimer la durée de flottation à 10,2 min. Cette durée de flottation est normalement celle où la teneur du concentré atteint la teneur de l'alimentation calculée.

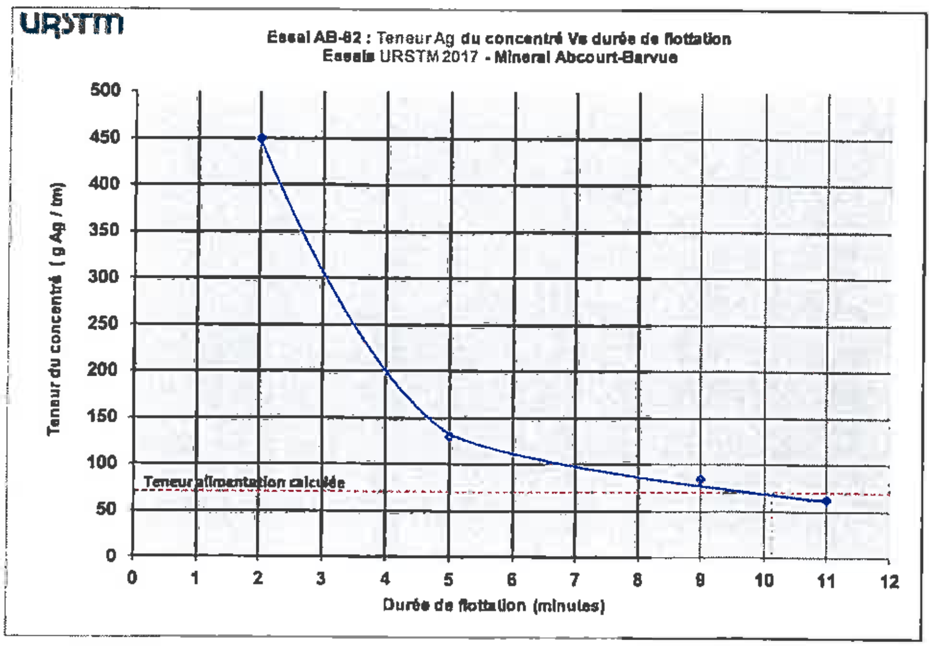


Figure 6 : Graphique de la teneur (Ag) du concentré Vs la durée de flottation (Essai AB-62)

Pour le zinc, on voit qu'une durée de flottation de 7,9 min serait correcte. Cependant, afin de maximiser la récupération combinée de l'argent et du zinc, la durée de flottation a été fixée à 11 min pour les essais AB-63 et AB-64.

Avec une flottation de 11 min pour l'essai AB-62, on obtient une récupération de 81,6 % pour l'argent et de 96,9 % pour le zinc.

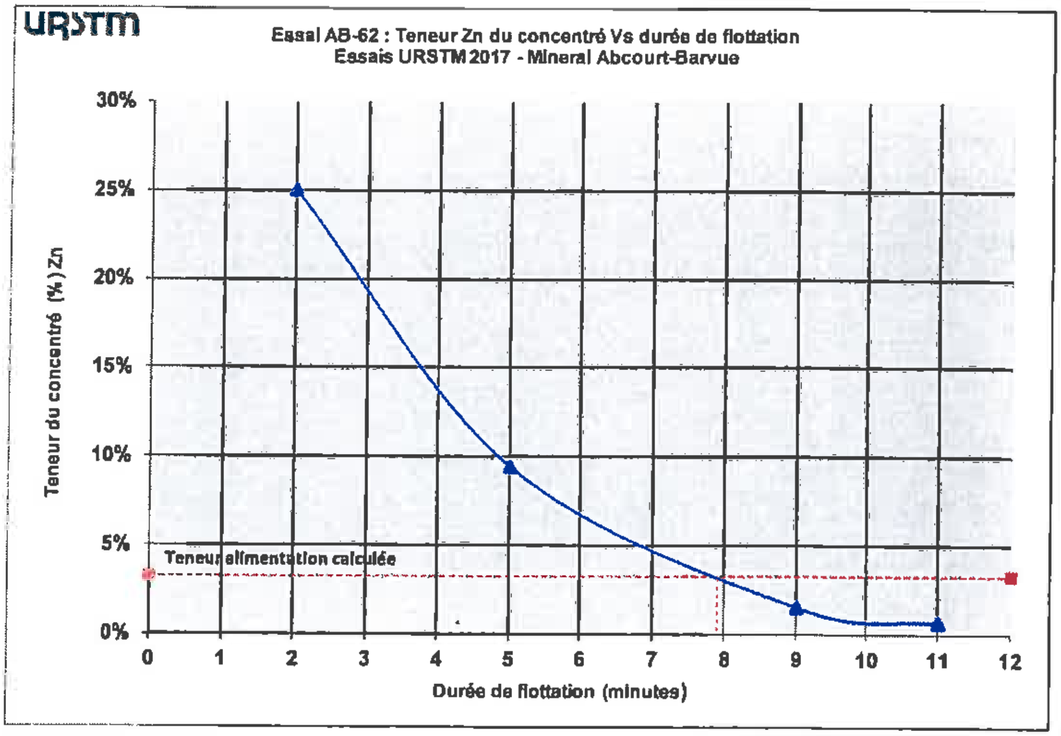


Figure 7 : Graphique de la teneur (Zn) du concentré Vs la durée de flottation (Essai AB-62)

4.3 Essai cyclique de flottation AB-63 (6 cycles)

Un total de deux essais cycliques a été effectué dans le cadre de ce projet, soit AB-63 et AB-64. L'essai AB-63 s'est inspiré du schéma de flottation de l'essai AB-15 réalisé en 2007 dans le projet PU-2007-12-356. Cependant, l'essai AB-15 avait été exécuté avec seulement 6 min de flottation. Avec l'échantillon de minerai Abcourt-Barvue 2017, les données de l'essai AB-62 suggère d'augmenter la durée de flottation à 11 min pour maximiser la récupération de l'argent. Pour tenir compte de cette durée de flottation augmentée, une étape de nettoyeur-épuiseur a été insérée dans le schéma. La *figure 8* présente le schéma utilisé pour l'essai de flottation cyclique AB-63.

Essai:	**AB-63**
Description:	Essai cyclique de flottation du minerai Abcourt avec désulfuration (6 cycles)
Granulométrie:	D80 = 74 microns 80,6% < 200 mailles
Date:	27-sept-2017

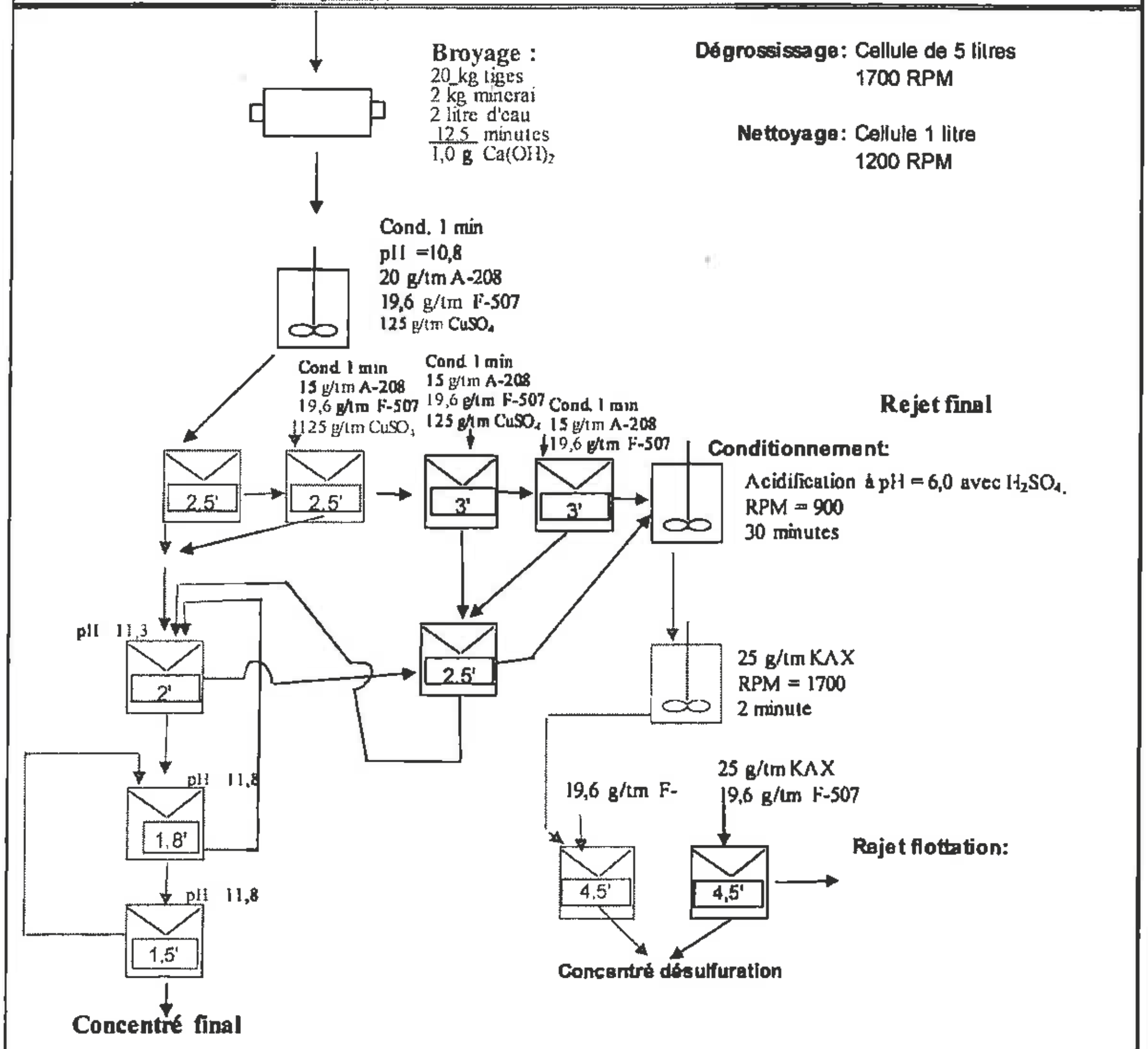


Figure 8 : Schéma de flottation de l'essai cyclique AB-63

On constate une récupération de l'argent de 77,8 % dans le concentré Zn-Ag, ce qui est similaire à la récupération en argent obtenue antérieurement en 2007 par l'essai AB-15, qui avait obtenu 79,0 %. L'échantillon de 2007 avait toutefois une teneur en argent de seulement 31,2 g Ag/ tm Vs 64,1 g/tm de l'échantillon 2017.

Le principal problème de ces résultats concerne la teneur du concentré de 37,5 % Zn qui est inférieure à la teneur anticipée d'au moins 50 % Zn. Ces résultats s'expliquent par les durées de flottation trop longues dans les étapes de nettoyage. C'est pour cette raison que les durées de flottation des étapes de nettoyage de l'essai AB-64 ont été réduites de manière significative.

Tableau 5 : Bilan métallurgique de l'essai cyclique AB-63 (Moyenne des cycles 4, 5 et 6)

	Masse (g)	% massique	Teneur		Unités		% distribution	
			Ag (g/tm)	% Zn	mg Ag	g Zn	Ag	Zn
Concentré final Zn-Ag	160,1 g	8,1%	618,2	37,5%	98,95	60,0	77,8%	96,6%
Concentré désulfuration	260,8 g	13,1%	62,6	0,3%	16,32	0,8	12,8%	1,4%
Rejet final	1563,3 g	78,8%	7,6	0,08%	11,85	1,3	9,3%	2,1%
Alimentation calc.	1984,2 g	100,0%	64,1	3,13%	127,12	62,14	100,0%	100,0%

- Récupération Ag dans le concentré Zn-Ag : 77,8 %
- Récupération Ag dans le concentré désulfuration : 12,8 %
- Récupération Zn dans le concentré Zn-Ag : 96,6 %

- Teneur Zn du concentré Zn-Ag : 37,5 %
- Teneur Ag du concentré Zn-Ag : 618,2 g Ag / tm

- % massique au concentré Zn-Ag : 8,1 %
- % massique du concentré désulfuration : 13,1 %
- % massique du rejet final désulfuré : 78,8 %

4.4 Essai de flottation cyclique AB-64 (4 cycles)

L'essai cyclique AB-64 a été réalisé en diminuant les durées de flottation pour les étapes de nettoyage de la façon suivante :

- Nettoyeur n° 1 : 2,0 minutes;
- Nettoyeur n° 2 : 1,0 minute;
- Nettoyeur n° 3 : 0,75 minute.

Le degré de broyage a été augmenté à un D_{80} de 64 µ versus un D_{80} de 74 µ pour l'essai AB-63.

Essai:	**AB-64**	
Description:	Essai cyclique de flottation du minerai Abcourt avec désulfuration **(4 cycles)**	
Granulométrie:	D80 =	64 microns 86,6% < 200 mailles
Date:	27-sept-2017	

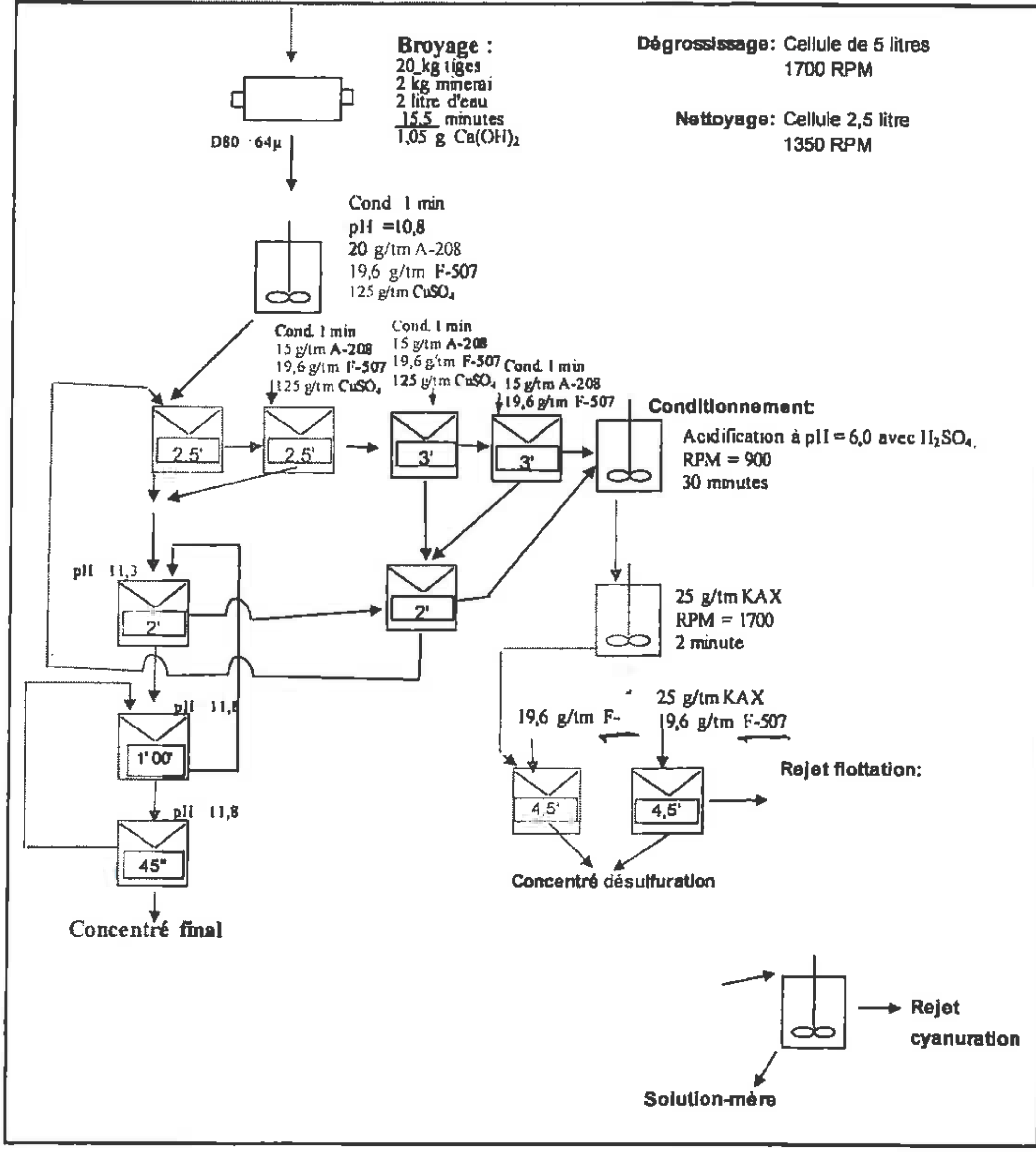


Figure 9 : Schéma de flottation de l'essai cyclique AB-64 (4 cycles)

Tableau 6 : Bilan métallurgique de l'essai cyclique de flottation AB-64 (moyenne des cycles 3 et 4)

	Masse (g)	% massique	Teneur		Unités		% distribution	
			Ag (g/tm)	% Zn	mg Ag	g Zn	Ag	Zn
Concentré final Zn-Ag	132,4 g	6,7%	740,6	53,4%	98,02	70,7	77,8%	97,5%
Concentré désulfuration	302,3 g	15,2%	58,6	0,3%	17,71	1,0	14,0%	1,4%
Rejet final	1555,1 g	78,2%	6,7	0,05%	10,34	0,8	8,2%	1,1%
Alimentation calc.	1989,7 g	100,0%	63,4	3,64%	126,07	72,51	100,0%	100,0%

- Récupération Ag dans le concentré Zn-Ag : 77,8 %
- Récupération Ag dans le concentré désulfuration : 14,0 %
- Récupération Zn dans le concentré Zn-Ag : 97,5 %

- Teneur Zn du concentré Zn-Ag : 53, 4% Zn
- Teneur Ag du concentré Zn-Ag : 740,6 g Ag / tm

- % massique au concentré Zn-Ag : 6,7 %
- % massique du concentré désulfuration : 15,2%
- % massique du rejet final désulfuré : 78,2 %
- Teneur en soufre du rejet final : 0,136 % S

De façon générale, on obtient des résultats similaires à l'essai cyclique AB-63 sauf en ce qui concerne la teneur du concentré final. La diminution des durées de flottation dans les étapes de nettoyage a permis d'augmenter de façon importante les teneurs des concentrés finaux. La teneur moyenne du concentré final Zn-Ag obtient 53,4 % Zn et 740,6 g Ag/tm, ce qui correspond aux objectifs de l'essai.

La récupération en argent de 77,8 % est également assez similaire à celle obtenue en 2007 dans le projet PU-2007-12-356 qui était alors de 79 % de récupération pour l'argent dans le concentré Zn-Ag.

La récupération en zinc de 97,5 % pour l'essai AB-64 est légèrement supérieure à la récupération obtenue pour l'essai AB-63, qui est de 96,6 %. Il faut prendre en considération les erreurs expérimentales des analyses chimiques et, également, le fait que cet essai cyclique ne comportait que quatre cycles. D'autres essais seraient requis si on désire valider davantage la récupération moyenne du minerai Abcourt-Barvue.

La teneur en soufre du rejet final, quant à elle, est très basse avec seulement 0,136 % soufre.

4.5 Analyses chimiques sur le concentré final de l'essai AB-64

Des analyses chimiques complètes ont été réalisées sur le concentré final Zn-Ag. Ces analyses ont été effectuées dans différents laboratoires, soit :

- Laboratoire Expert pour le zinc et l'argent;
- Actlabs de Ste-Germaine-Boulé pour l'argent;
- Actlabs de Ancaster pour les analyses de roche totale ainsi que l'analyse multiélément.

Les certificats d'analyses complets se retrouvent à l'*annexe 5*.

Tableau 7 : Analyses chimiques sur le concentré final Zn-Ag

Ag ppm	As ppm	Ba ppm	C	Ca	Cd ppm	Cl	Co ppm	Cu ppm	F	Fe	Ge ppm	Hg ppm
710	1200	10	0,33%	0,12%	>1000	0,01%	63	3350	0,02%	8,67%	< 0,1	6,21
Mg	**Mn ppm**	**Ni ppm**	**Pb ppm**	**Sb ppm**	**Se ppm**	**Te ppm**	**Tl ppm**	**S**	**Zn**	**SiO2**	**Al2O3**	
0,05%	1310	20,4	4310	403	16,1	0,1	0,16	31,4%	54,1%	6,89%	1,86%	

4.6 Synthèse des essais de flottation réalisés

Le *tableau 8* présente un résumé des principaux résultats obtenus lors de la réalisation des trois essais de flottation.

Tableau 8 : Synthèse des essais de flottation (AB-49, AB-63 et AB-64)

Essais cycliques de flottation	D80	Concentré final Zn-Ag				Concentré désulfuration				Rejet final				Teneur alimentation calculée	
		Teneur		% distribution		Teneur		% distribution		Teneur		% distribution			
		Ag g / tm	Zn	Ag	Zn	Ag g / tm	Zn	Ag	Zn	Ag g / tm	Zn	Ag	Zn	Ag g / tm	Zn
AB-49	56 µ	631,1	51,0%	69,6%	96,5%					18,0	0,12%	30,4%	3,5%	55,5	3,2%
AB-63	74 µ	618,2	37,5%	77,8%	96,6%	62,6	0,33%	12,8%	1,4%	7,6	0,08%	9,3%	2,1%	64,1	3,1%
AB-64	64 µ	740,6	53,4%	77,8%	97,5%	58,6	0,34%	14,0%	1,4%	6,7	0,05%	8,2%	1,1%	63,4	3,6%

4.7 Essais de cyanuration sur le concentré de pyrite de l'essai AB-64

Un total de quatre essais de cyanuration a été effectué sur des échantillons du concentré de pyrite provenant de l'essai de flottation cyclique AB-64. Ci-dessous, les paramètres de cyanuration utilisés :

- Essais en bouteille;
- 30 % solide;
- Durée de cyanuration = 48 h;
- Concentration initiale en NaCN : 2000 ppm;
- Pas d'ajout de nitrate de plomb.

Le *tableau 9* donne un aperçu des résultats obtenus de ces essais de cyanuration.

Tableau 9 : Résumés des résultats obtenus des essais de cyanuration sur le concentré de pyrite

	Essais cyanuration sur le concentré de pyrite	D80	% récupération		Teneur du rejet de cyanuration		Teneur alimentation calculée		Consommation NaCN		Consommation $Ca(OH)_2$	
			Ag	Au	Ag g / tm	Au g / tm	Ag g / tm	Au g / tm	kg NaCN / tm concentré	kg NaCN/ tm minerais	kg $Ca(OH)_2$ / tm concentré	kg $Ca(OH)_2$ / tm minerais
Sans rebroyage	AB-64-CN-A-4	23 µ	73,9%		17,4		66,6		1,38	0,21	5,03	0,76
	AB-64-CN-A-3	25 µ	56,8%	19,4%	24,2	0,07	56,0	0,08	1,31	0,20	5,21	0,79
	Moyenne (sans rebroyage):	**24 µ**	**65,3%**	**19,4%**	**20,8**	**0,07**	**61,3**	**0,08**	**1,34**	**0,20**	**5,12**	**0,78**
Avec rebroyage	AB-64-CN-B-4	19 µ	82,9%		11,8		68,8		1,82	0,28	6,52	0,99
	AB-64-CN-B-3	19 µ	69,1%	42,9%	16,7	0,06	54,0	0,10	2,09	0,32	7,95	1,21
	Moyenne (avec rebroyage):	**19 µ**	**76,0%**	**42,9%**	**14,2**	**0,06**	**61,4**	**0,10**	**1,96**	**0,30**	**7,24**	**1,10**

- La récupération moyenne obtenue de l'argent sur le concentré de pyrite sans rebroyage est de 65,3 %.
- La récupération moyenne obtenue de l'argent sur le concentré de pyrite avec rebroyage est de 76,0 %.

On peut également observer que la granulométrie du concentré de désulfuration est très fine même avant rebroyage avec une granulométrie D_{80} de 23 µ. Cependant, un léger rebroyage à 19 µ permet tout de même d'augmenter la récupération de façon significative.

Le *tableau 10* présente une projection du bilan métallurgique global du traitement du minerai Abcourt-Barvue combinant la flottation d'un concentré de zinc et la flottation d'un concentré de pyrite qui serait ensuite cyanuré. Le bilan métallurgique de l'essai AB-64 a été utilisé pour faire cette projection en appliquant la récupération moyenne obtenue pour les essais de cyanuration AB-64-CN-A-3 et AB-64-CN-A-4 (sans rebroyage). Pour cette projection, les données massiques ont été réajustées en fonction d'une alimentation de 1 tm de minerai.

Tableau 10 : Bilan métallurgique projeté (Flottation + cyanuration du concentré de pyrite sans rebroyage)

	Masse (tm)	% massique	Teneur		Unités		% distribution	
			Ag (g/tm)	% Zn	g Ag	kg Zn	Ag	Zn
Concentré final Zn-Ag	0,067 tm	6,7%	740,6	53,4%	49,26	35,53	77,8%	97,5%
Concentré pyrite	0,152 tm	15,2%	58,6	0,34%	8,90	0,51	14,0%	1,4%
Solution calculée de la cyanuration du concentré de pyrite					5,82	0,03	9,2%	0,1%
Rejet solide calculé de cyanuration du concentré de pyrite	0,152 tm	15,2%	20,3	0,31%	3,08	0,48	1,3%	1,3%
Rejet final de flottation	0,782 tm	78,2%	6,7	0,05%	5,20	0,40	8,2%	1,1%
Alimentation calc.	1,000 tm	100,0%	63,4	3,6%	63,36	36,44	100,0%	100,0%

On peut constater que la récupération combinée de l'argent dans le concentré de zinc et dans le concentré de pyrite atteint 87,0 % (77,8 % + 9,2 %) sans rebroyage du concentré de pyrite.

Tableau 11 : Bilan métallurgique projeté (Flottation + cyanuration du concentré de pyrite avec rebroyage)

	Masse (tm)	% massique	Teneur		Unités		% distribution	
			Ag (g/tm)	% Zn	g Ag	kg Zn	Ag	Zn
Concentré final Zn-Ag	0,067 tm	6,7%	740,6	53,4%	49,26	35,53	77,8%	97,5%
Concentré pyrite	0,152 tm	15,2%	58,6	0,34%	8,90	0,51	14,0%	1,4%
Solution calculée de la cyanuration du concentré désulfuration					6,76	0,03	10,7%	0,1%
Rejet solide calculé de cyanuration du concentré de pyrite	0,152 tm	15,2%	14,1	0,31%	2,14	0,48	3,4%	1,3%
Rejet final de flottation	0,782 tm	78,2%	6,7	0,05%	5,20	0,40	8,2%	1,1%
Alimentation calc.	1,000 tm	100,0%	63,4	3,6%	63,36	36,44	100,0%	100,0%

% récupération Ag dans le concentré de flottation Zn-Ag :	77,8%
% récupération Ag dans la solution de cyanuration du concentré de désulfuration :	10,7%
Récupération Ag combinée (flottation + cyanuration du concentré désulfuration) :	88,5%

On peut constater que la récupération combinée de l'argent dans le concentré de zinc et dans le concentré de pyrite atteint 88,5 % avec rebroyage du concentré de pyrite. Donc le rebroyage du concentré de pyrite permet d'augmenter la récupération globale de l'argent de 1,5 %.

5. Bilans métallurgiques projetés

Le *tableau 12* présente un bilan métallurgique estimatif qui prend en considération les teneurs réelles du minerai Abcourt-Barvue, telles qu'évaluées dans le rapport d'évaluation des ressources[1] en 2014. Le *tableau 13*, quant à lui, montre le bilan estimatif de cyanuration du concentré de pyrite avec rebroyage.

Tableau 12 : Bilan flottation projeté pour le minerai Abcourt-Barvue

	Masse (tm)	% massique	Teneur		Unités		% distribution	
			Ag (g/tm)	% Zn	g Ag	kg Zn	Ag	Zn
Alimentation minerai	1,000 tm	100,0%	55,0	3,0%	55,00	30,00	100,0%	100,0%
Concentré zinc	0,055 tm	5,5%	781,4	53,4%	42,79	29,25	77,8%	97,5%
Concentré pyrite	0,125 tm	12,5%	61,8	0,34%	7,73	0,42	14,0%	1,4%
Rejet final de flottation	0,820 tm	82,0%	5,5	0,04%	4,48	0,33	8,2%	1,1%

- Récupération Ag dans le concentré de flottation du zinc : 77,8 %
- Récupération Ag dans le concentré de flottation de la pyrite : 14,0 %
- La récupération en zinc est de 97,5 %.

Tableau 13 : Bilan cyanuration projeté pour le concentré de pyrite rebroyé

	Masse (tm)	% massique	Teneur		Unités		% distribution	
			Ag (g/tm)	% Zn	g Ag	kg Zn	Ag	Zn
Concentré pyrite	0,125 tm	12,5%	61,8	0,34%	7,73	0,42	14,0%	1,4%
Solution-mère					5,87	0,03	10,7%	0,1%
Rejet de cyanuration du concentré pyrite	0,125 tm	12,5%	14,1	0,31%	1,85	0,39	3,4%	1,3%

La cyanuration du concentré de pyrite après rebroyage ne récupère que 5,87 g d'argent par tonne de minerai. En assumant un prix de 20 $CND/oz, on obtient une valeur brute de l'argent récupéré de seulement 3,77$/tonne de minerai.

Cela ne semble pas suffisant pour couvrir le coût de construction des circuits de rebroyage et de cyanuration, au moins au début des opérations. Cette situation pourrait changer en cours d'opération par l'optimisation du circuit de flottation du zinc qui pourrait entraîner un déplacement d'une plus grande partie de la pyrite, de l'or et de l'argent vers le circuit de flottation de la pyrite.

[1] NI-43-101 Mineral Resources Report for the Abcourt-Barvue Property, Feb. 2014.

6. Méthode de traitement envisagée pour le traitement du minerai Abcourt-Barvue

6.1 Présentation de la méthode

La principale méthode envisagée pour le traitement du minerai Abcourt-Barvue consiste à procéder tout d'abord à une flottation du minerai, pour ensuite expédier le concentré de flottation à une affinerie de zinc, dans le but de récupérer conjointement le zinc et l'argent. Le rejet de cette flottation est ensuite désulfuré, et le concentré de pyrite résultant est cyanuré pour récupérer l'argent. Le rejet de cette flottation de la pyrite est alors désulfuré et est considéré non générateur d'acide (voir *figure 10*).

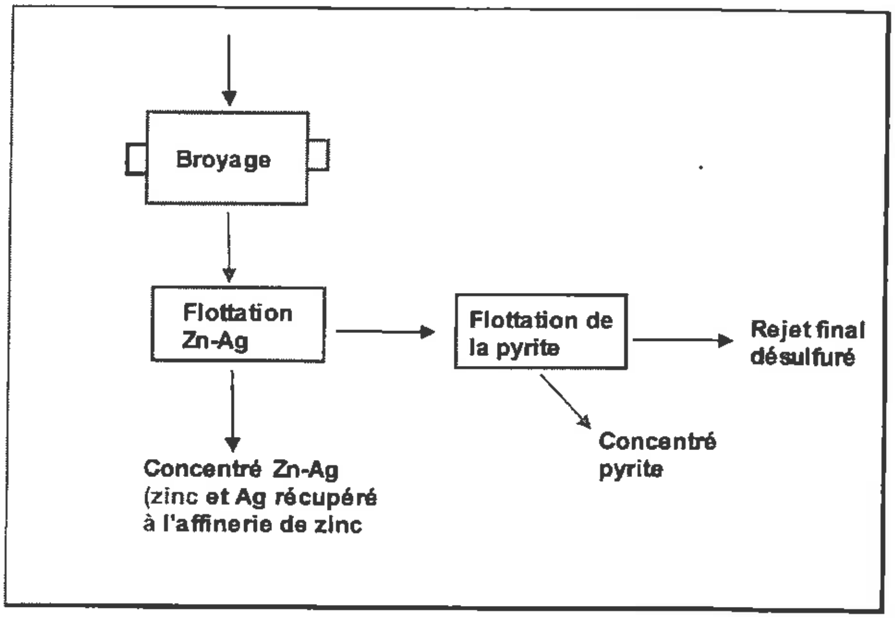


Figure 10: Méthode de traitement envisagée pour le traitement du minerai Abcourt-Barvue

6.2 Estimation de la récupération globale en argent

La méthode de traitement envisagée consiste essentiellement à flotter un concentré combiné Zn-Ag et de le faire traiter à l'affinerie de zinc pour récupérer l'argent. Cette récupération dans un concentré de zinc implique cependant des coûts qui se traduisent finalement en équivalent de perte de récupération. La cyanuration du concentré de pyrite est ensuite effectuée pour récupérer l'argent.

Les hypothèses suivantes ont été posées :

- L'affinerie de zinc impose une pénalité initiale de -3 oz Ag/tm;
- L'affinerie de zinc paie ensuite 77,5 % de la valeur de l'argent;
- Teneur en argent du minerai = 55,0 g Ag/tm

Le *tableau 14* présente les calculs permettant d'établir la récupération réelle en argent après raffinage. Les données sont issues de l'essai cyclique de flottation AB-64.

Tableau 14 : Calcul de la récupération effective de l'argent après raffinage
(méthode de traitement envisagée avec rebroyage du concentré de désulfuration)

	Masse (g)	% massique	Teneur	Unités	% distribution
			Ag (g/tm)	g Ag	Ag
Concentré flottation final Zn-Ag	0,058 tm	5,8%	**740,6**	43,06	**77,8%**
pénalité (- 3 oz Troy / tm)	0,058 tm	5,8%	93,3	5,42	9,8%
Concentré flottation final Zn-Ag - pénalité de -3 oz	0,058 tm	5,8%		37,63	68,0%
Teneur économique du concentré après frais de raffinage (Concentré flottation final Zn-Ag - pénalité de -3 oz) * 77,5% valeur de l'argent	0,058 tm	5,8%	501,7	29,17	52,7%
Concentré pyrite	0,133 tm	13,3%	**58,6**	7,78	**14,0%**
Solution cyanuration du concentré de pyrite				5,91	10,7%
Rejet de cyanuration du concentré de pyrite	0,133 tm	13,3%	14,1	1,87	3,4%
Rejet final de flottation	0,809 tm	80,9%	6,7	4,54	8,2%
Rejet final combiné	0,942 tm	94,2%	6,8	6,41	11,6%
Alimentation calc.	1,000 tm	100,0%	**55,00**	55,38	100,0%

Il indique également que la récupération réelle de l'argent, suivant le schéma de traitement envisagé, est de 52,7 % après raffinage. Cette récupération est très faible, mais possède au moins l'avantage d'éviter les coûts d'opération et les investissements reliés à l'opération d'un circuit de cyanuration. La cyanuration du concentré de désulfuration avec rebroyage du concentré permet d'ajouter 10,7 % de récupération supplémentaire pour l'argent. La récupération globale en argent obtenue après raffinage serait alors 63,3 % (52,7 % + 10,7 %).

Tableau 15 : Calcul de la récupération effective de l'argent après raffinage (méthode de traitement envisagée sans rebroyage du concentré de désulfuration

	Masse (g)	% massique	Teneur Ag (g/tm)	Unités g Ag	% distribution Ag
Concentré flottation final Zn-Ag	0,058 tm	5,8%	740,6	43,06	77,8%
pénalité (- 3 oz Troy / tm)	0,058 tm	5,8%	93,3	5,42	9,8%
Concentré flottation final Zn-Ag - pénalité de -3 oz	0,058 tm	5,8%		37,63	68,0%
Teneur économique du concentré après frais de raffinage (Concentré flottation final Zn-Ag - pénalité de -3 oz) * 77,5% valeur de l'argent	0,058 tm	5,8%	501,7	29,17	52,67%
Concentré pyrite	0,133 tm	13,3%	58,6	7,78	14,0%
Solution cyanuration du concentré de pyrite				5,08	9,18%
Rejet de cyanuration du concentré de pyrite	0,133 tm	13,3%	20,3	2,70	4,9%
Rejet final de flottation	0,809 tm	80,9%	6,7	4,54	8,2%
Rejet final combiné	0,942 tm	94,2%	7,7	7,24	13,1%
Alimentation calc.	1,000 tm	100,0%	55,00	55,38	100,0%

Sans rebroyage du concentré de désulfuration, la récupération globale de l'argent après raffinage serait de 61,8 % (52,7 % + 9,2 %).

6.3 Récupération de l'or

Le calcul de la récupération de l'or n'avait pas été planifié au départ dans ce projet. Cependant, il s'est avéré pertinent de réaliser trois essais de cyanuration supplémentaires pour tenter d'obtenir un ordre de grandeur de la récupération de l'or.

Un essai de cyanuration AB-65-CN a été réalisé sur le concentré Zn-Ag provenant de l'essai AB-62[2], et deux essais de cyanuration (AB-64-CN-A-3 et AB-64-B-3) l'ont été sur le concentré de désulfuration provenant de l'essai AB-64 (cycle 3).
La récupération en or par cyanuration obtenue sur ce concentré Zn-Ag a été de 97,3 % (Essai AB-65-CN). Pour le concentré de pyrite (concentré de désulfuration), les récupérations en or ont été respectivement de 19,4 % sans rebroyage et de 42,1 % avec rebroyage.

[2] Étant donné qu'il ne restait plus de concentré final provenant de l'essai cyclique AB-64, l'essai de cyanuration sur le concentré Zn-Ag a été réalisé sur le concentré de flottation provenant de l'essai AB-62.

En fonction de ces données, un bilan métallurgique estimatif a été réalisé pour l'or. Pour ce faire, une analyse de l'or a été effectuée sur un des échantillons de minerai Abcourt-Barvue provenant d'un des lots précédemment préparés et une autre, sur le rejet de flottation de l'essai AB-64. La teneur du concentré en or du concentré Zn-Ag a conséquemment été calculée.

Tableau 16 : Bilan métallurgique estimatif de l'or (avec rebroyage du concentré de pyrite)

	Masse (g)	% massique	Teneur	Unités	% distribution	Code analyse
			Au (g/tm)	g Au	Au	
Concentré flottation final Zn-Ag	0,058 tm	5,8%	1,12	0,065	84,5%	Note[1]
Concentré pyrite	0,132 tm	13,2%	0,09	0,012	15,5%	Note[2]
Solution calculée de la cyanuration du concentré de pyrite				0,005	6,7%	Note[3]
Rejet solide calculé de la cyanuration du concentré de pyrite	0,132 tm	13,2%	0,06	0,007	8,5%	S-156 S-158
Rejet final de flottation	0,810 tm	81,0%	<0,03	0,000	0,0%	S-142 S-142-B
Alimentation calc.	1,000 tm	100,0%	0,077	0,077	100,0%	

Note[1] : Teneur en or du concentré Zn-Ag tirée de l'essai AB-65-CN.

Note[2] : Teneur moyenne en or du concentré de pyrite (Essais AB-64-CN-A-3 et AB-64-CN-B-3). Les % massique du tableau précédent ont été utilisés.

Note[3] : Le nombre de grammes d'argent récupéré a été calculé à partir de la récupération de 42,9 % de l'essai AB-64-CN-B-3 (avec rebroyage du concentré de pyrite).

Bien qu'estimatif, le bilan métallurgique permet d'obtenir un ordre de grandeur de la valeur possible de l'or récupéré dans le concentré de désulfuration. Selon ce bilan, la récupération de l'or serait de 0,005 g Au/tm de minerai. En fonction d'un prix de l'or de $1 350 US/oz et d'un taux de change de $0,75 US pour 1 $CDN, on arrive à une valeur de l'or récupéré de l'or pour la cyanuration du concentré de désulfuration de 0,405 $CDN/tm de minerai (voir *équation 1*).

$$\frac{0{,}005\ g\ Au/tm \times \$1350\ US \times 1\ oz\ Troy \times 1\ \$CDN}{31{,}1\ g \times \$0{,}75\ US} = 0{,}29\ \$CDN/tm\ de\ minerai \qquad [1]$$

Cette valeur a été établie à partir d'un échantillon de minerai avec une teneur d'alimentation calculée de 0,077 g Au/tm. Cette teneur est un peu inférieure à celle indiquée dans l'évaluation des ressources[3] réalisée en 2014 qui utilisait une teneur du minerai de 0,1 g Au/tm. Cette faible quantité d'or récupéré pourrait augmenter un peu avec le déplacement d'une partie de la pyrite flottée dans le concentré de zinc vers le concentré de pyrite lors de l'optimisation du procédé.

[3] NI-43-101 Mineral Resources Report for the Abcourt-Barvue Property, Feb. 2014

7. Conclusions

Les essais cycliques de flottation réalisés sur le minerai Abcourt-Barvue ont permis d'atteindre une récupération maximale de 77,8 % de l'argent et 97 % du zinc. La teneur du concentré final était de 740,6 g Ag/tm et de 53,4 % Zn/tm, ce qui est très acceptable. Ces résultats sont issus de l'essai cyclique AB-64. D'autres essais seraient requis si on désire valider davantage la récupération moyenne du minerai Abcourt-Barvue.

La teneur en soufre du rejet final désulfuré est très basse (0,136 %). La réalisation d'un essai de génération d'acide devrait confirmer le caractère non générateur d'acide de ce rejet désulfuré.

Ces résultats ont été obtenus en s'inspirant du protocole de flottation AB-15 et en utilisant les mêmes collecteurs que ceux utilisés dans le projet PU-2007-12-356). Il pourrait être envisageable d'augmenter le degré de récupération en argent en procédant à une évaluation d'autres collecteurs. Cependant, il reste très probable qu'une bonne partie de l'argent résiduel soit associée à la pyrite et que les gains de récupération en argent soient peu importants.

L'évaluation de la récupération de l'argent, selon la méthode de traitement envisagée et en tenant compte des déductions de l'affineur, donne une récupération effective de l'argent de 61,8 % (sans rebroyage du concentré de désulfuration.) Cette méthode consiste essentiellement à flotter conjointement le zinc et l'argent dans un concentré qui serait ensuite expédié à une affinerie pour en récupérer le zinc et l'argent. Le rejet de cette première flottation serait ensuite soumis à une seconde étape de flottation, qui aurait pour objectif de produire un concentré de pyrite-argent, qui serait ensuite cyanuré pour en récupérer une partie de l'argent résiduel. Le rejet final de flottation serait désulfuré et considéré non générateur d'acide.

Cette récupération de l'argent de 61,8 % pourrait être supérieure si on effectuait plutôt une cyanuration en usine du concentré de flottation Zn-Ag. Selon l'estimation présentée en *annexe 2*, il serait possible d'atteindre dans ce cas une récupération globale de l'argent de 74,5 %. Cependant, la cyanuration du concentré Zn-Ag implique des investissements importants, qui ne sont pas envisagés pour le moment.

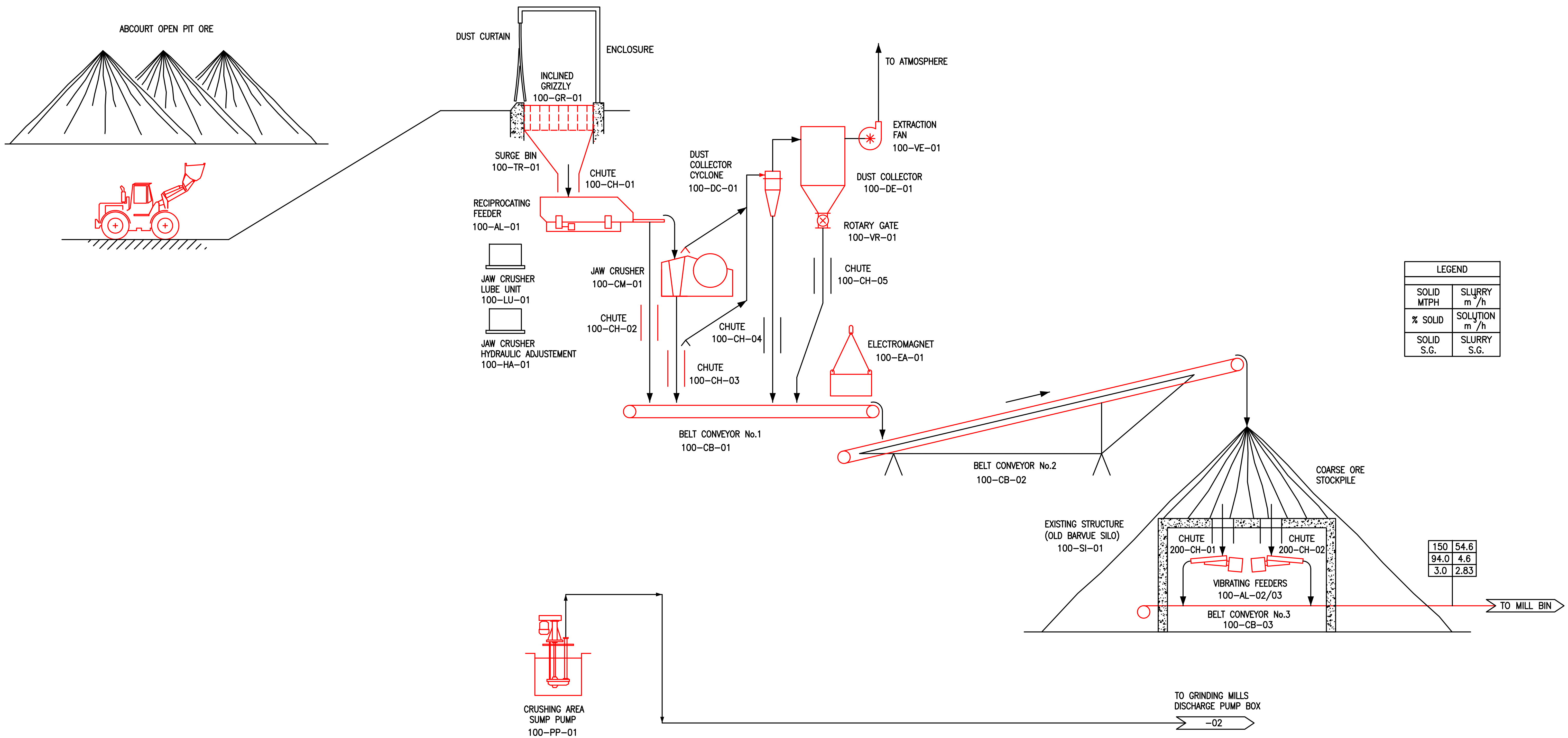
Même si les teneurs en or ne sont pas importantes, il serait pertinent d'inclure l'or dans certains essais futurs puisque sa valeur pourrait contribuer à augmenter celle des métaux récupérés.

Jean Lelièvre, ing. M. Sc.

INGÉNIEUR · ENGINEER
Jean Lelièvre
38387
QUÉBEC

APPENDIX 5

Process Plant Flow Sheet (ref. Section 17)



PROJECT
BARVUE

CLIENT
MINES ABCOURT

LOCATION: BARRAUTE

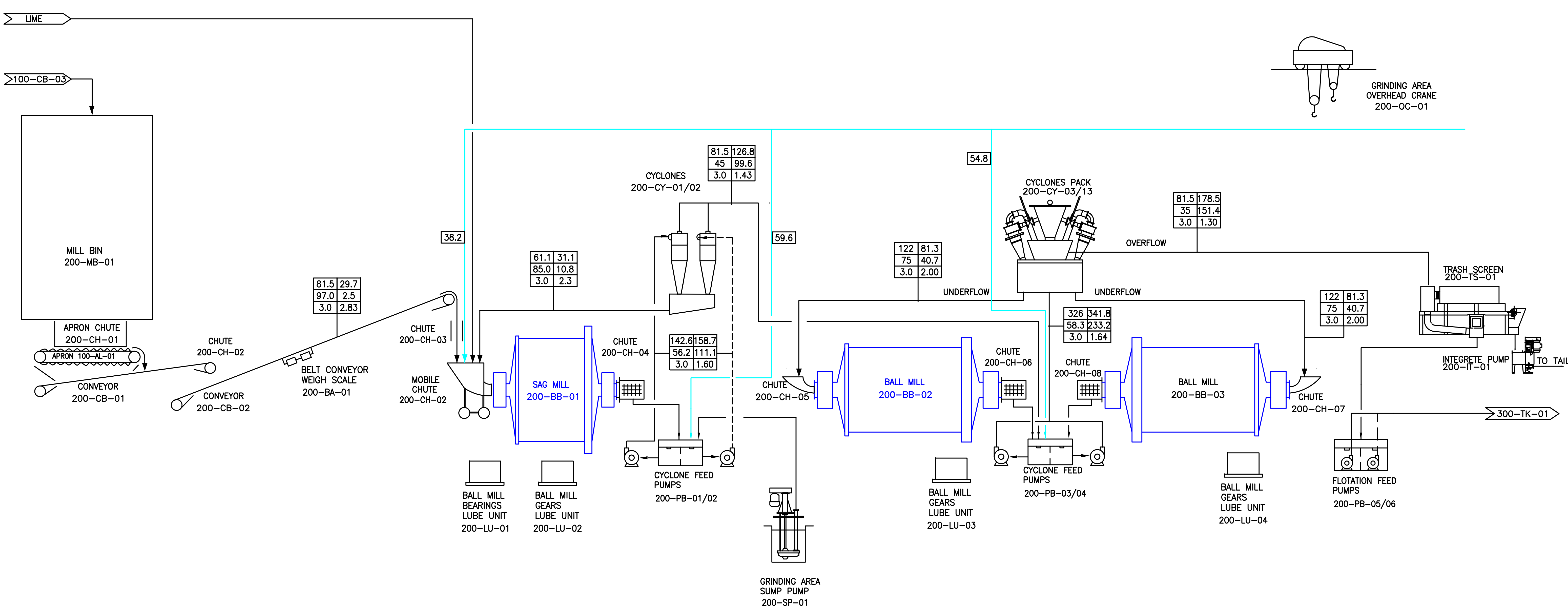
TITLE
CRUSHING FLOWSHEET
1 800 TONNES/DAY

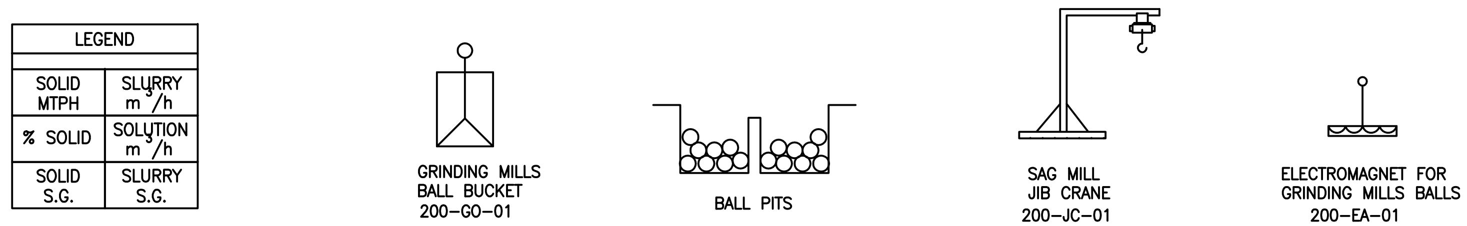
DEVELOP	R.W. J. CONS,	NOV. 2018
DESIGN	R.W.J. cons.	NOV 2018
APPROVED	-	-
SCALE	-	

PROJECT NO. AB2018

DRAWING NO. 200-10

REV 01





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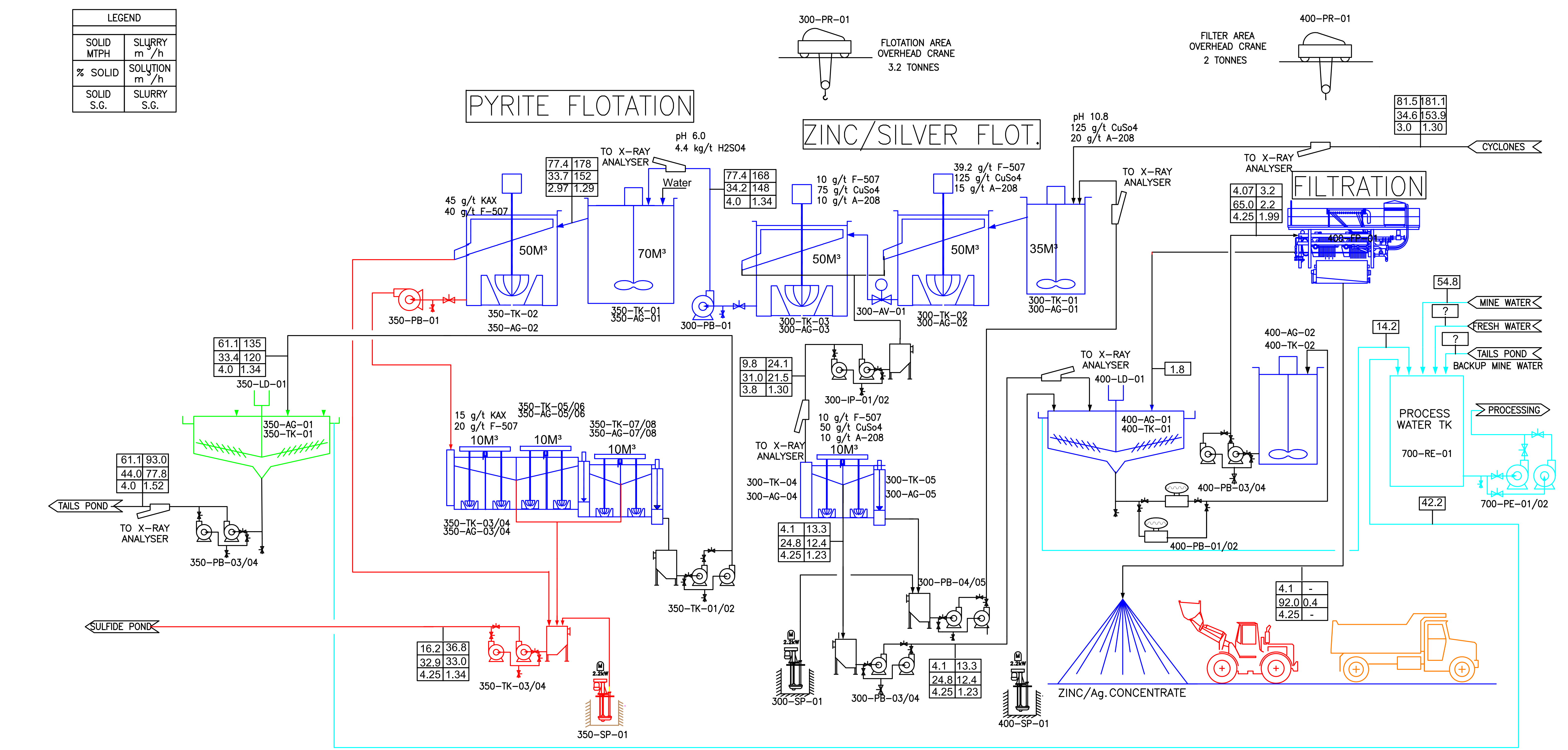
		DESIGN R.W.J. cons. — NOV. 2018	
		APPROVED – — –	DRAWING NO. 200–10
		SCALE –	REV 01



PROJECT: BARVUE

CLIENT: **MINES ABCOURT**

LOCATION: BARRAUTE

TITLE: FLOTATION/FILTRATION FLOWSHEET
1 800 TONNES/DAY

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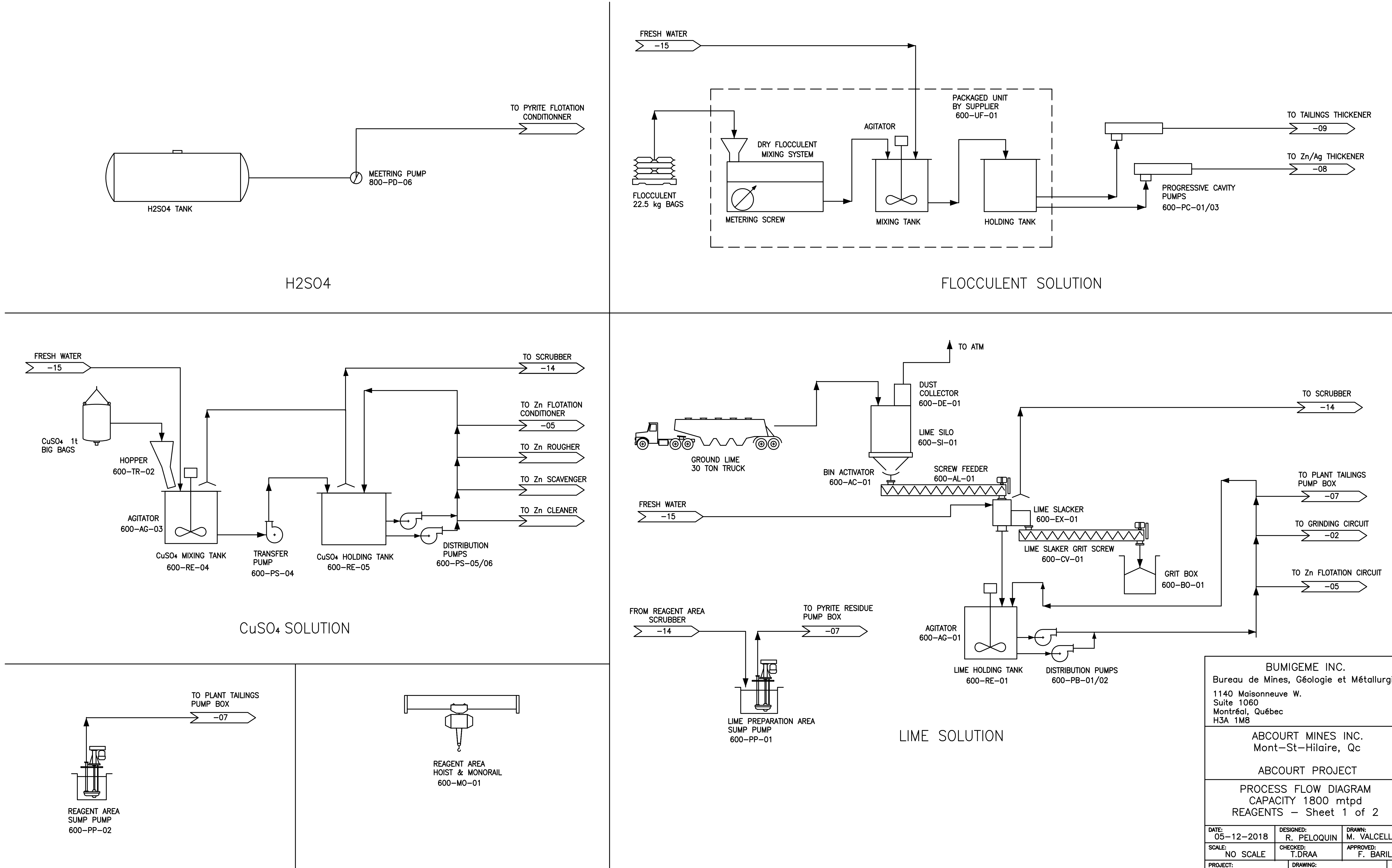
DESIGN	R.W.J. cons.	NOV. 2018	
APPROVED	-	-	DRAWING NO. 200-11
SCALE	-		REV 02



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1140 Maisonneuve W.
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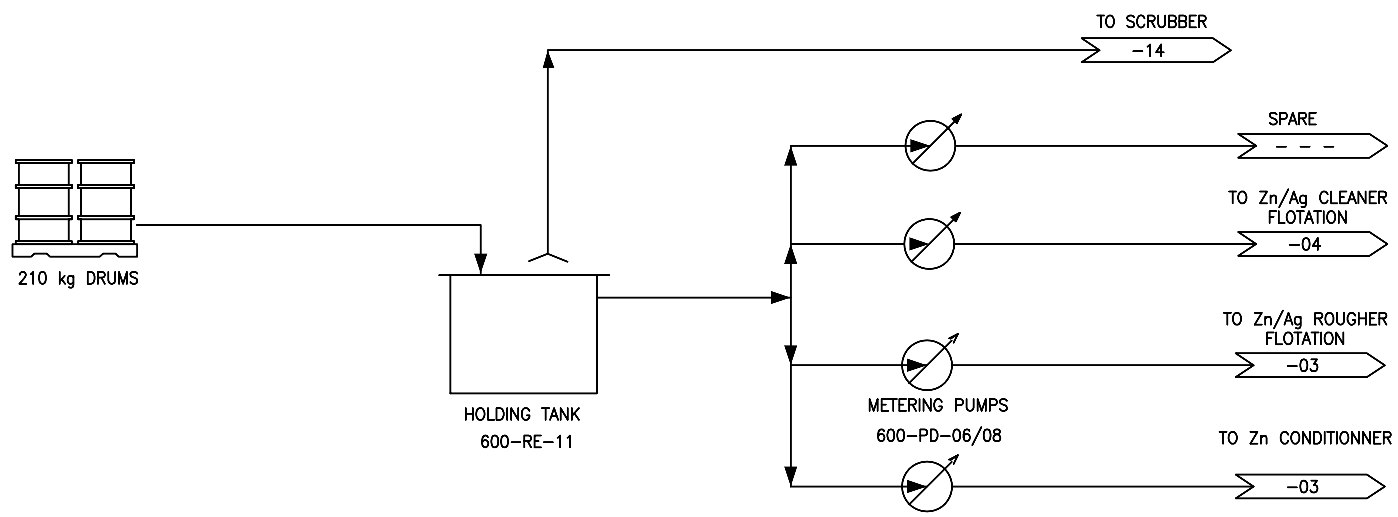
Montréal, Québec
H3A 1M8

ABCOURT MINES INC.
Mont-St-Hilaire, Qc

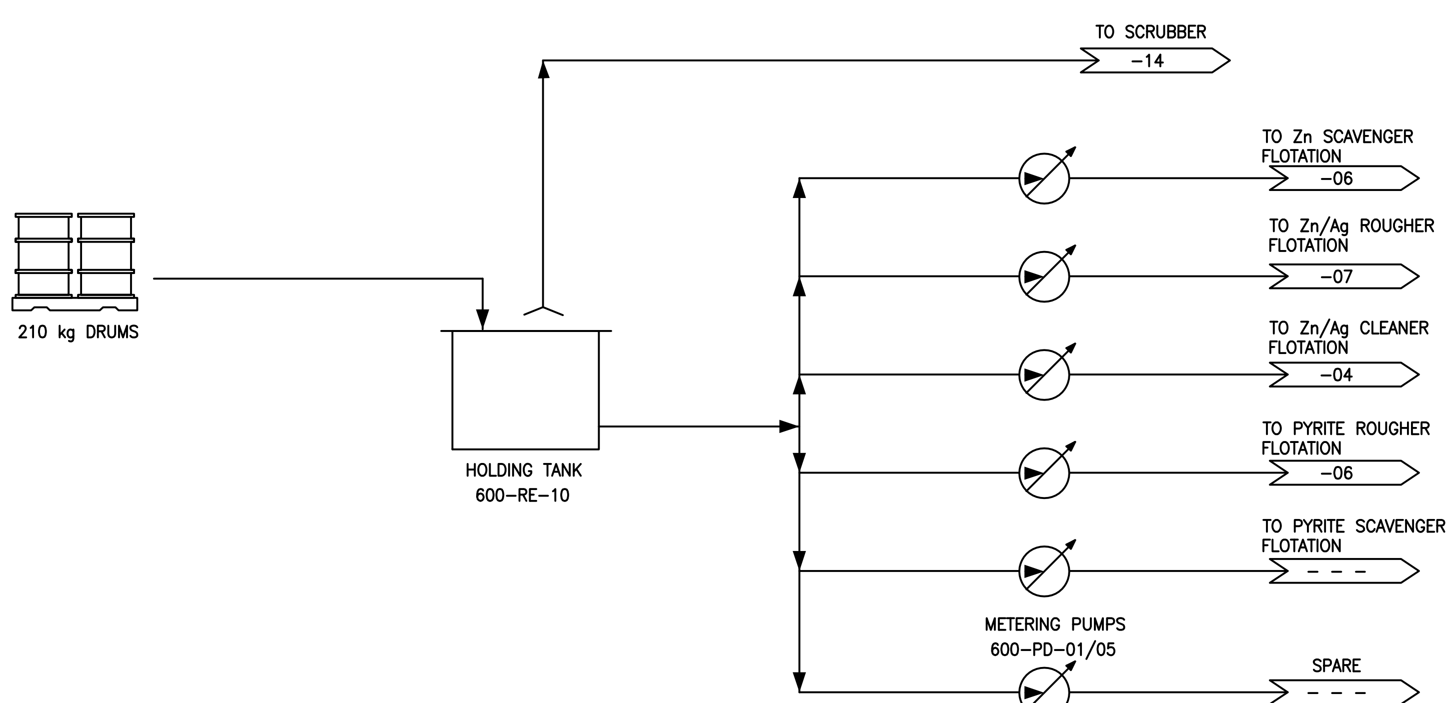
ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 mtpd
REAGENTS – Sheet 1 of 2

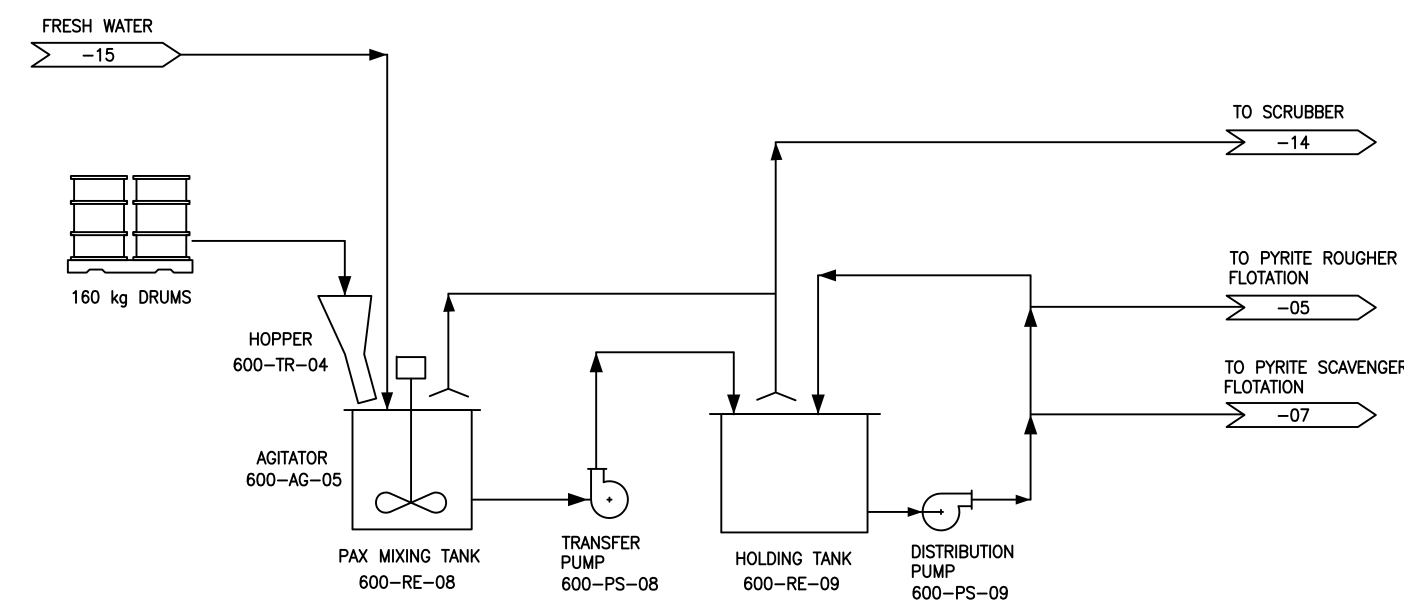
DATE: 05-12-2018	DESIGNED: R. PELOQUIN	DRAWN: M. VALCELLI
SCALE: NO SCALE	CHECKED: T.DRAA	APPROVED: F. BARIL
PROJECT: B150-02-005	DRAWING: B15002005-12	REV: 01



A–208 SOLUTION



F–507 SOLUTION



KAX SOLUTION

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ABCOURT PROJECT

PROCESS FLOW DIAGRAM
CAPACITY 1800 mtpd
REAGENTS – Sheet 2 of 2

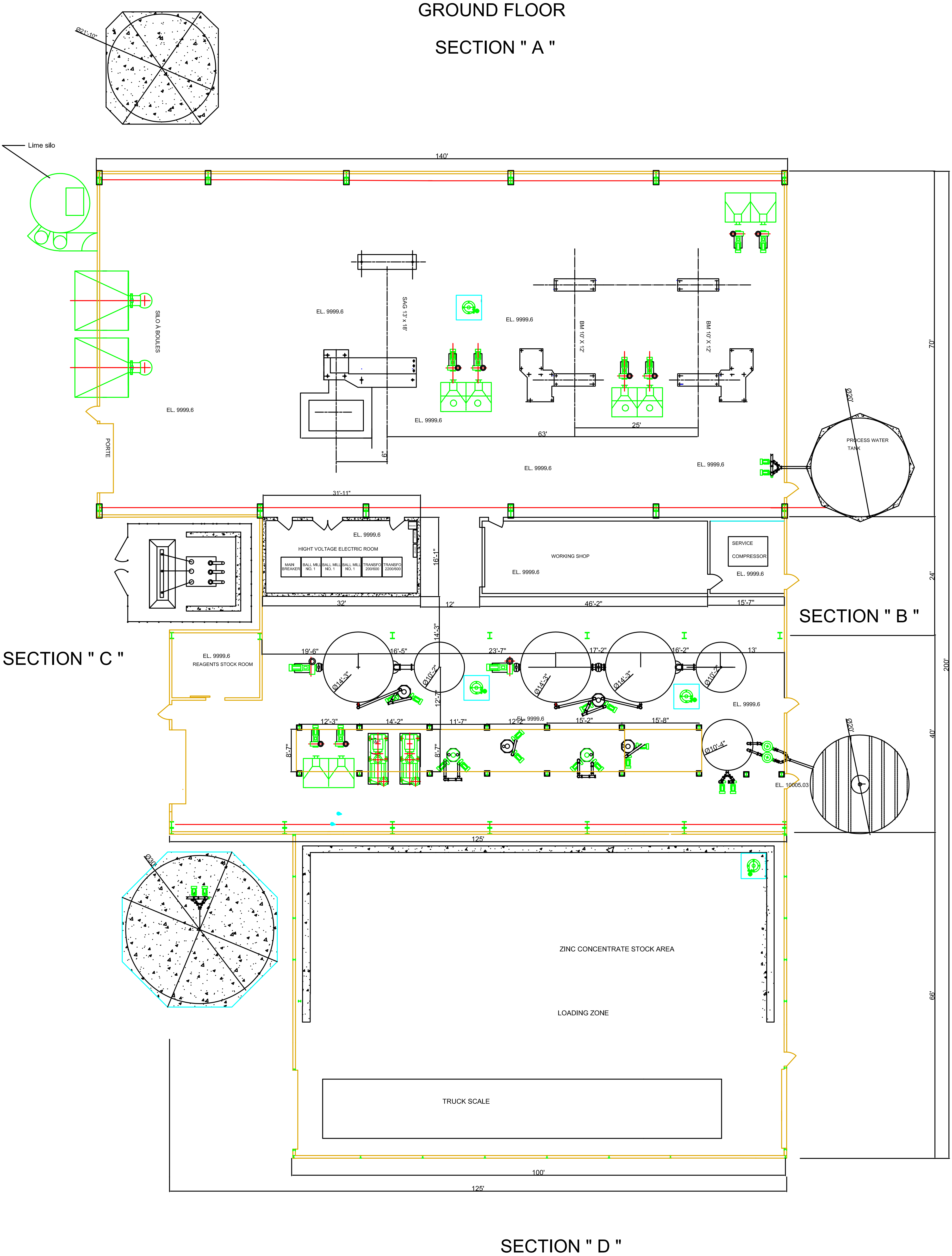
DATE: 06–12–2018	DESIGNED: R. PELOQUIN	DRAWN: M. VALCELLI	
SCALE: NO SCALE	CHECKED: T.DRAA	APPROVED: F. BARIL	
PROJECT: B150–02–005	DRAWING: B15002005–13		REV: 01

APPENDIX 6

Process Plant General Arrangement (ref. Section 17)

PLAN VIEW
GROUND FLOOR

SECTION " A "



SECTION " D "

PROJECT	CLIENT	TITLE	
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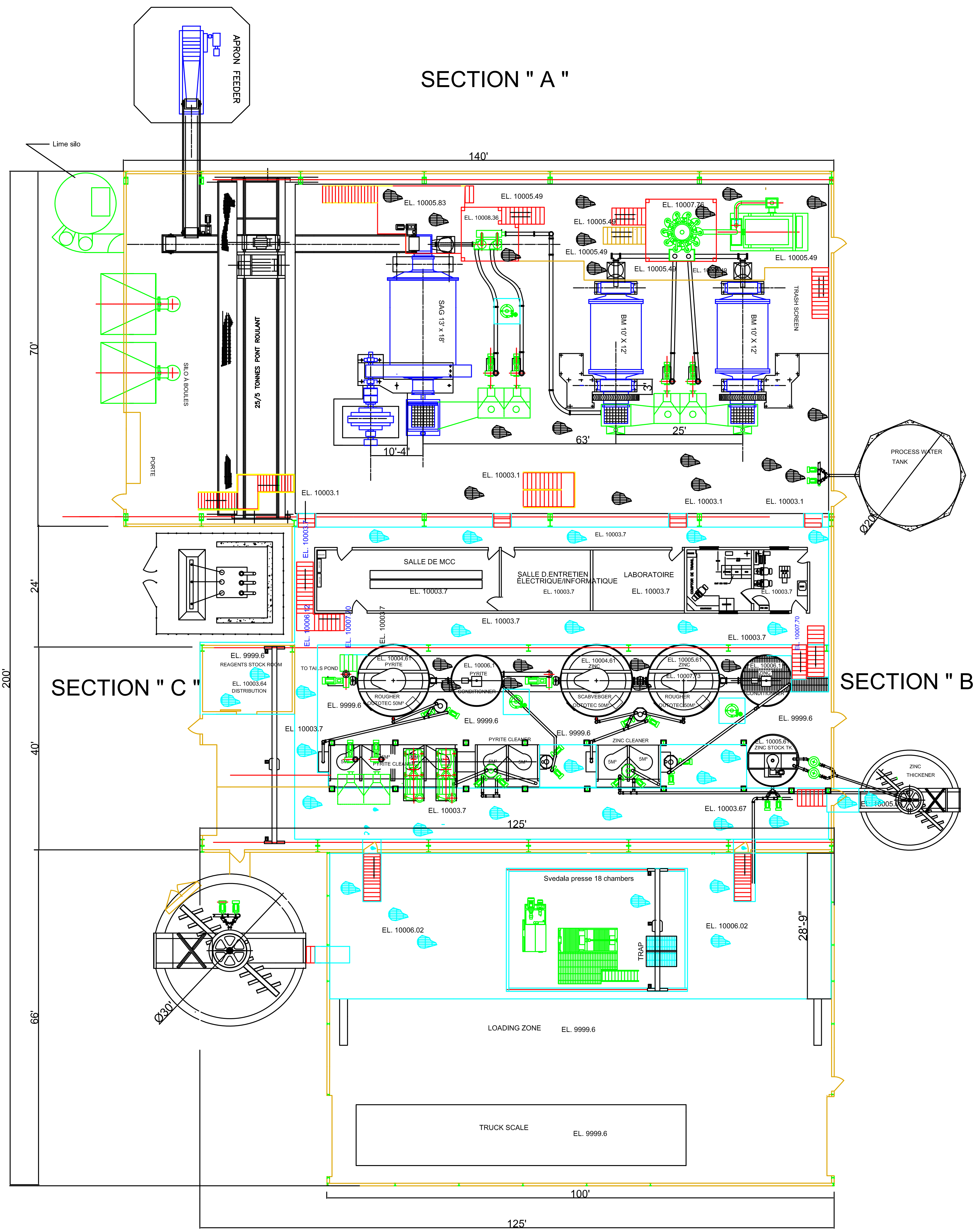
		DESIGN R.W.J. cons. Nov. 2018	
		APPROVED - -	DRAWING NO. 200-01
		SCALE -	REV 03

PLAN VIEW
TOP FLOOR



SECTION " D "

PROJECT	CLIENT	TITLE		
BARVUE	MINE ABCOURT	PLANT VIEW 1 800 TONNES/DAY		
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		DESIGN R.W.J. cons.	Nov. 2018	
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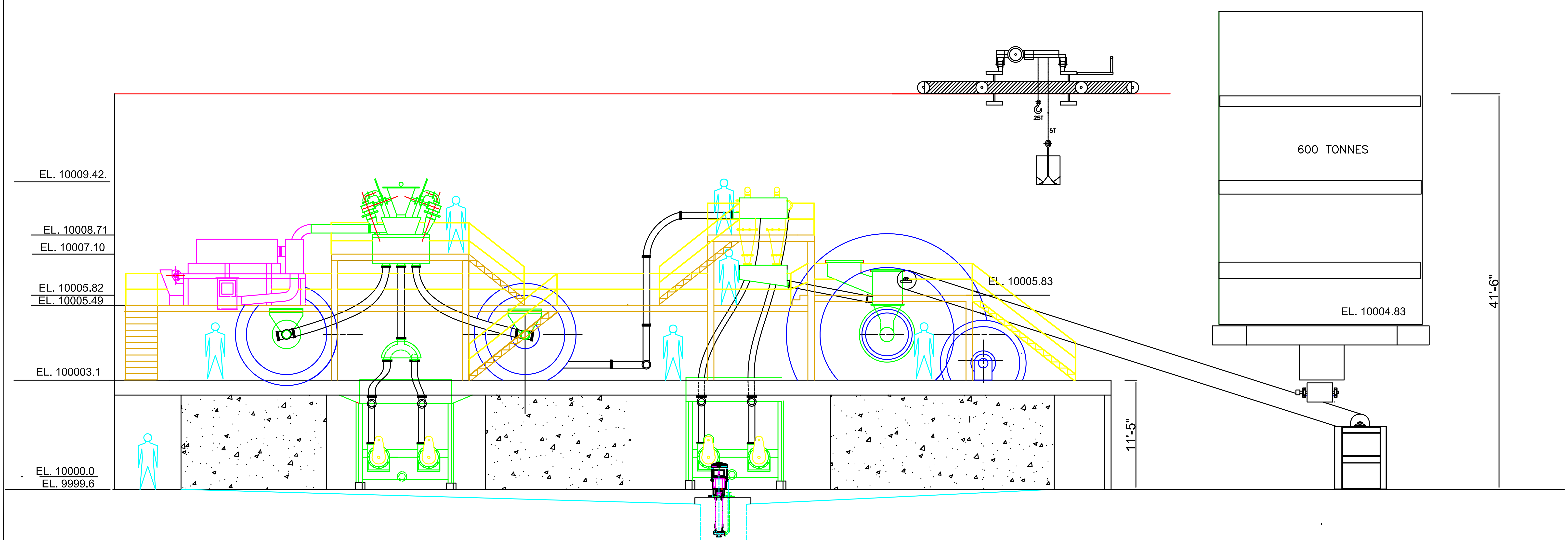
		SCALE -		REV 03

SECTION " A "



PROJECT: BARVUE	MINE ABCOURT	TITLE: SECTION "A" 1 800 TONNES/DAY			
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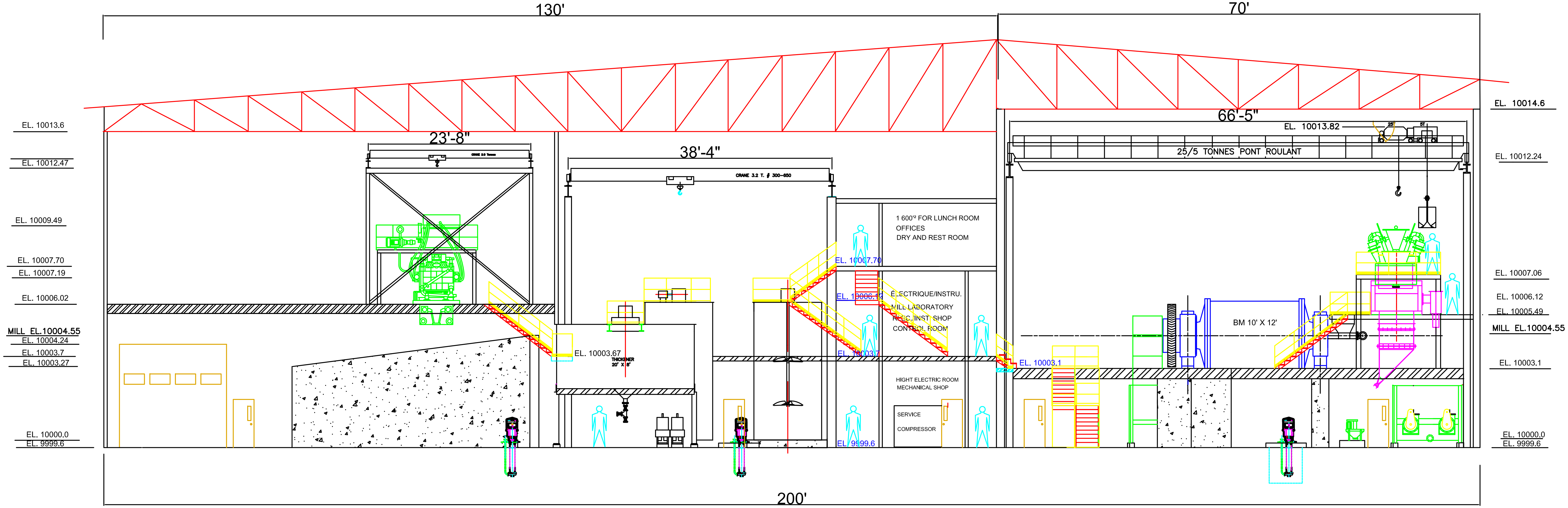
		APPROVED -	-	DRAWING NO. 200-04	REV 02
		SCALE -			

SECTION " B "



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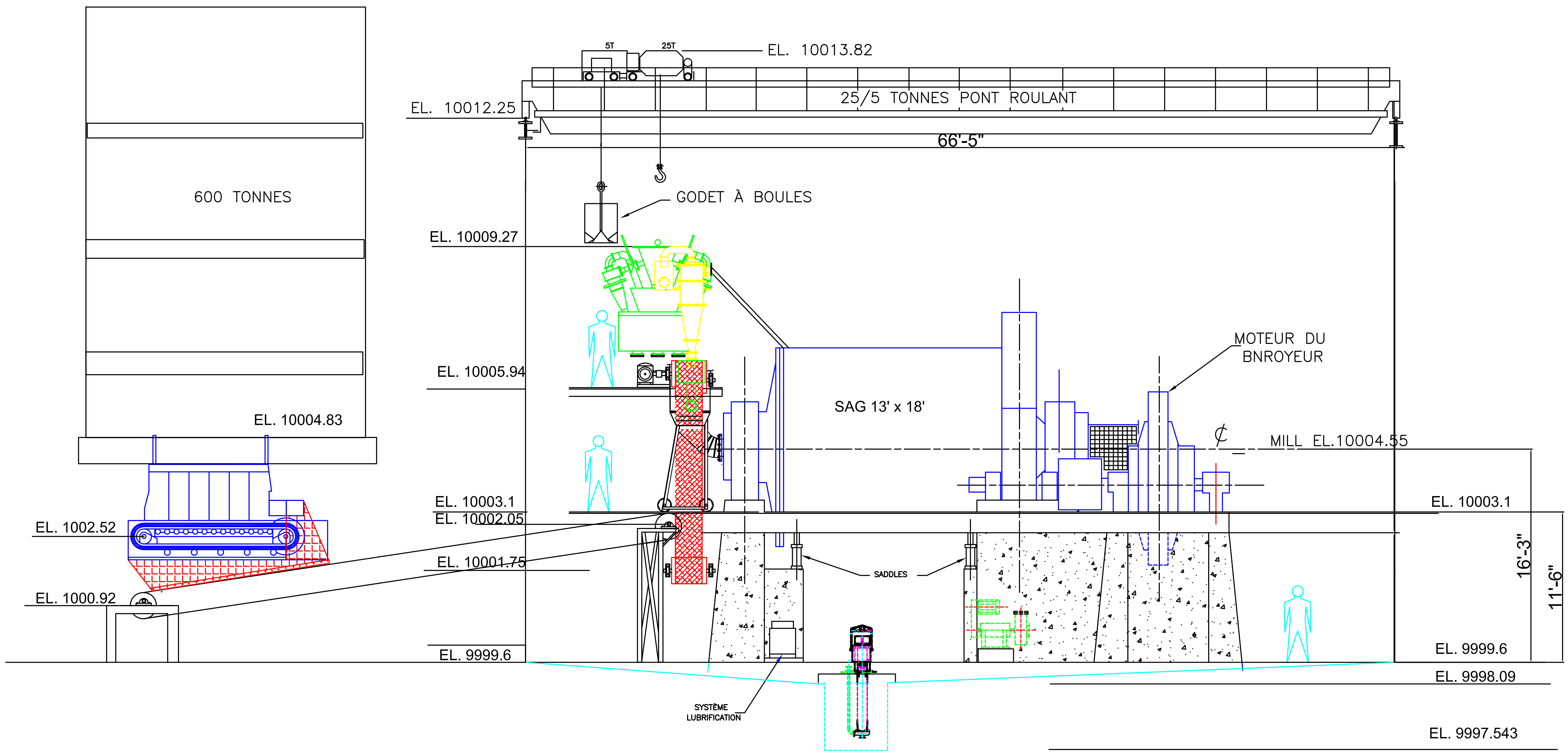
SECTION " C "



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		DESIGN R.W.J. cons.	DEC. 2018		
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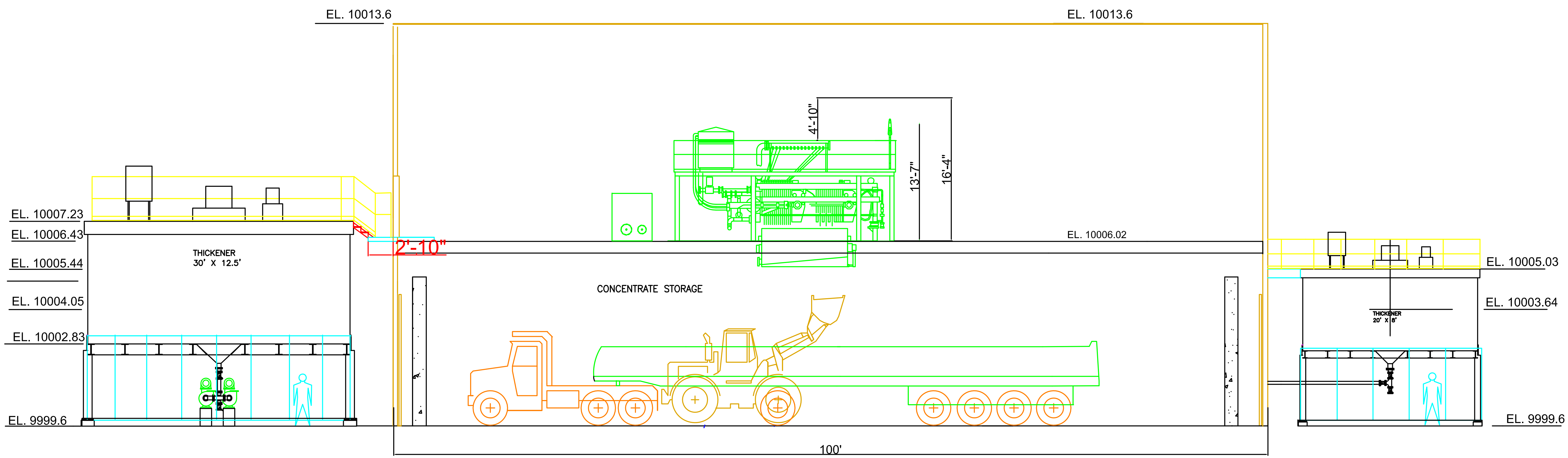
		SCALE -			

SECTION " D "



PROJECT	CLIENT	TITLE		
BARVUE	MINE ABCOURT	FILTRATION SECTOR 1 800 TONNES/DAY		
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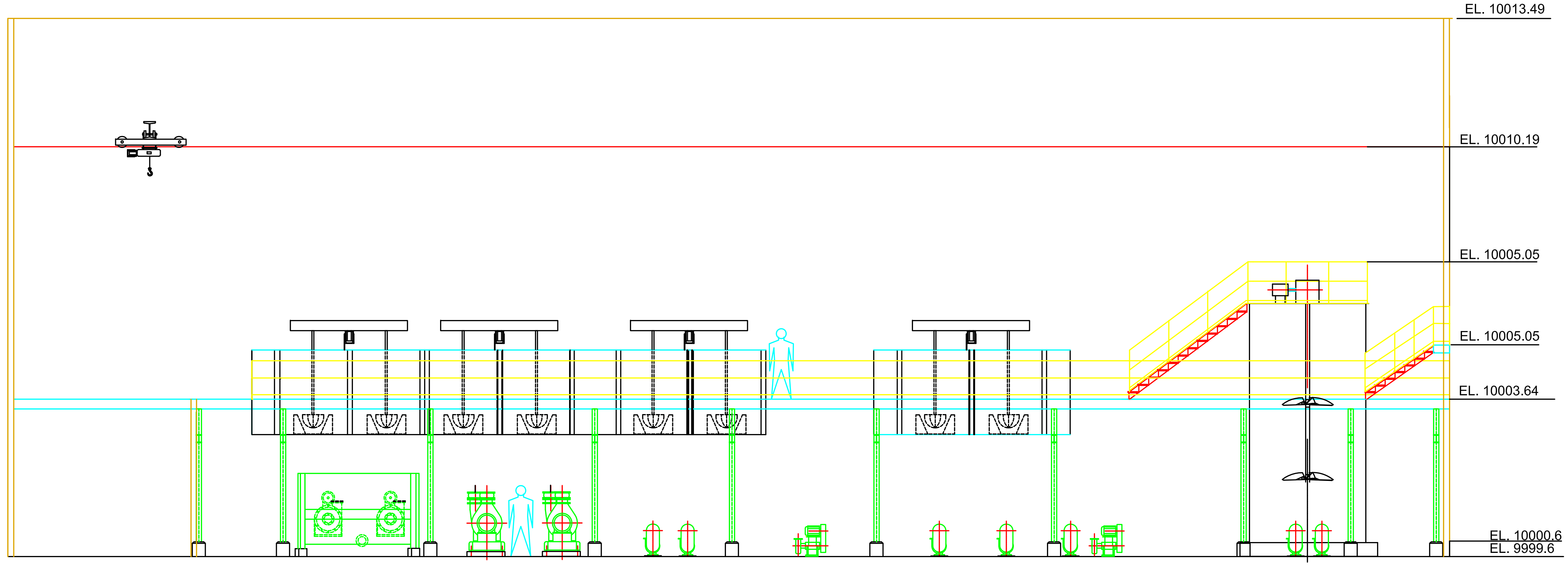
		DESIGN R.W.J. cons.	APRIL 2018	
	LOCATION: BARRAUTE	APPROVED -	-	DRAWING NO. 200-08 / REV 02
		SCALE -		

CLEANER SIDE VIEW



PROJECT	CLIENT	TITLE		
BARVUE	MINE ABCOURT	CLEANER SIDE VIEW 1 800 TONNES/DAY		
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		DESIGN R.W.J. cons.	APRIL 2018	
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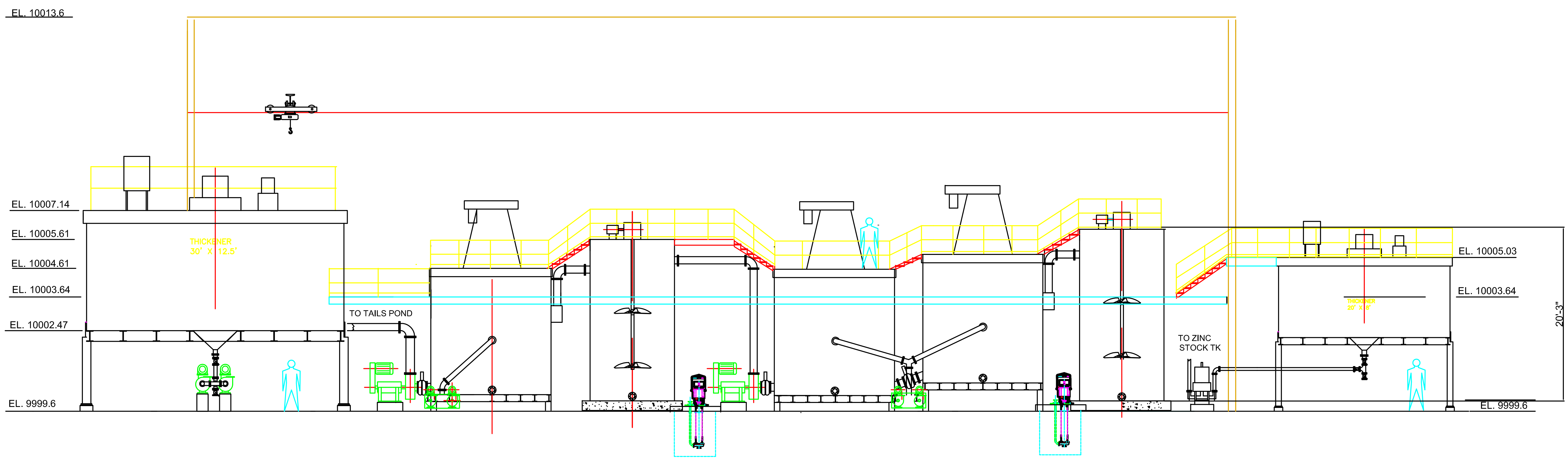
		SCALE -		REV 02

FLOTATION SIDE VIEW



PROJECT	CLIENT	TITLE		
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		DESIGN R.W.J. cons.	APRIL 2018	
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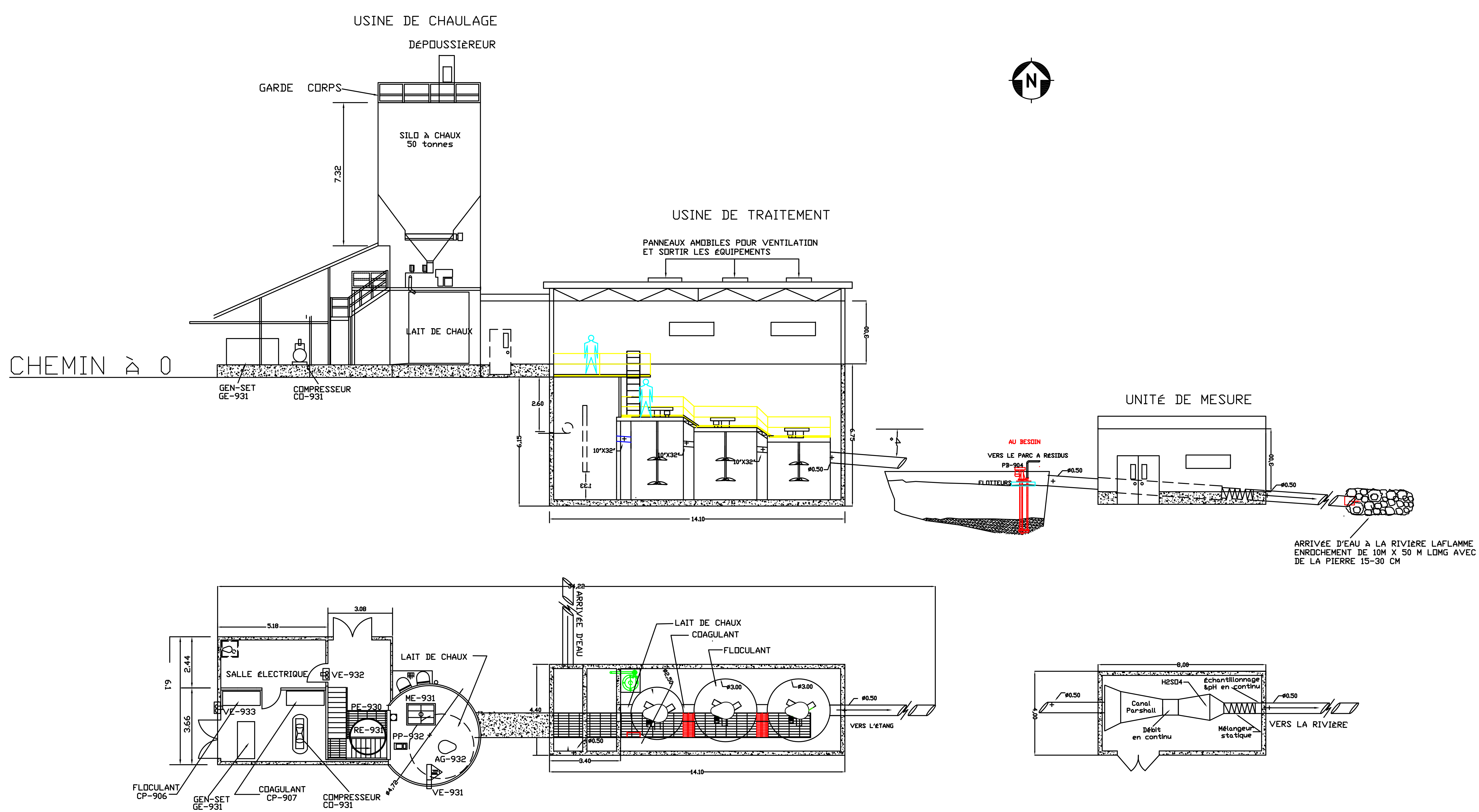
APPENDIX 7

Water Treatment Plant (ref. Section 17)



NOTE:

Toutes les dimensions devront être vérifiées avec les relevés pris lors du démantelement de l'usine de Mine Louvicourt.

MINES ABCOURT MINES INC.	CLIENT: MINES ABCOURT INC.	TITRE: TRAITEMENT DE L'EAU DE LA FOSSE		
		CONÇU Roger Jolicoeur	MAI 2018	PROJET NO. USAB 01
		DESSINE R.W.J. consultants	MAI 2018	
	ENDROIT: BARRAUTE QC.	VÉRIFIÉ Renaud Hinse Ing.	NOV. 2018	DESSIN NO. 003 — REV 03
		ECHELLE AUCUNE		

APPENDIX 8

Process Equipment List (ref section 21)

MINES ABCOURT MINES INC.

Process Equipments Cost including installation 2018

Equipment List

Equip. No.	Description	Qty	Capacity or Dimension	Power	Used Equipments			New	Installation				
				in kW	On	Purchased	Refurbishing	Equipments:	Electrical	Mechanical	Piping:	Transport:	Total:
ORE CRUSHING AND STOCKPILING SECTION:					Hand	Used							
100-GR-01	Grizzly	1	3m X 3m					8 000 $		5 000 $		1 500 $	14 500 $
100-TR-01	Surge bin	1	75 tonnes					30 000 $				4 000 $	34 000 $
100-AL-100	Panne/grizzly feeder	1	1,5m X 4m			10 000 $	10 000 $		2 680 $				22 680 $
100-CM-01	Jaw crusher	1	1220 mm x 1525	186,5	X		25 000 $		19 000 $	45 000 $			89 000 $
100-LU-01	Jaw crusher lube unit	1	--	2	X		2 000 $		2 100 $	2 000 $			6 100 $
100-HA-01	Jaw crusher hydraulic adjustement system				X		2 000 $						2 000 $
100-CH-03	Crusher discharge chute	1	--	--				20 000 $		3 000 $			23 000 $
100-CB-01	Belt conveyor No. 1 (Crusher discharge)	1	914 mm x 12 m	7,5				30 000 $	9 195 $	10 000 $		2 000 $	51 195 $
100-EA-01	Electromagnet	1	--	--				15 000 $	4 820 $	14 000 $		1 500 $	35 320 $
100-CH-04	Electromagnet trolley and chute	1	--	--				16 000 $		5 000 $		1 500 $	22 500 $
100-CB-02	Belt conveyor No. 2 (Stock pile feed)	1	914 mm x 50 m	14,9				125 000 $	16 320 $	30 000 $		4 500 $	175 820 $
100-TC-01	Cone stock pile												
100-CF-01	Under cone feeders	2	Vibrating	7,5				50 000 $	5 820 $	8 000 $		1 500 $	65 320 $
100-CB-03	Belt conveyor No. 3 (Stock pile to transfert)	1	914 mm x 53 m	14,9				132 500 $	16 320 $	30 000 $		4 500 $	183 320 $
100-CB-04	Convoyeur mill bin feed	1	914 mm x 78m	37,3				195 000 $	16 320 $	30 000 $		4 500 $	245 820 $
100-DC-01	Dust collector - Cyclone	1	--					18 000 $		6 000 $	1 500 $	1 500 $	27 000 $
100-DE-01	Dust collector - Crusher	1	180 m²	7,5				8 000 $	9 055 $	3 500 $	2 500 $	1 200 $	24 255 $
100-VR-01	Rotary gate	1	--	0,2				3 500 $	2 160 $				5 660 $
100-CH-05	Dust collector chute	1	--	--				2 500 $		2 000 $			4 500 $
100-VE-01	Extraction fan	1	--	15				6 500 $	3 760 $	2 000 $			12 260 $
TOTAL CRUSHING:				**293,3**		**10 000 $**	**39 000 $**	**660 000 $**	**107 550 $**	**195 500 $**	**4 000 $**	**28 200 $**	**1 044 250 $**
GRINDING SECTION:													
200-OB-01	Coarse ore bin	1	600 tonnes 6,8m X 10,0					250 000 $				4 500 $	254 500 $
200-CH-01	Coarse ore chute	1	--	--				12 000 $		5 000 $		1 500 $	18 500 $
200-CH-02	Coarse ore chute	1	--	--				12 000 $		5 000 $		1 500 $	18 500 $
200-AL-01	Apron feedre c/w discharge chute	1	1000 x 5000	14,9				125 000 $	3 680 $	15 000 $		2 500 $	146 180 $
200-CB-01	Belt conv. No. 4 transfer apron to sag feed conv.	1	914 mm x 7,75 m	11,25				35 000 $	2 019 $	5 000 $		1 500 $	43 519 $
200-CH-02	Chute of head conveyor # 4							4 500 $		2 500 $		500 $	7 500 $
200-CB-02	Sag feed belts # 5		914 mm x 14,5 m					50 000 $	3 180 $	5 000 $		1 500 $	59 680 $
200-BA-01	Belt conveyor weigh scale	1	0 - 150 t/h	--				14 000 $	2 680 $	6 000 $		1 000 $	23 680 $
200-CH-0	SAG mill feed chute	1	--	--				14 000 $		5 000 $	4 000 $	2 000 $	25 000 $
200-BB-01	SAG mill	1	4000 mm dia. x 5,484	1450	X		20 000 $		67 000 $	160 000 $			247 000 $
200-LU-01	SAG mill bearings lube unit	1	--	3,75	X		8 000 $		3 680 $	4 000 $	2 500 $		18 180 $
200-LU-02	SAG mill gears lube unit	1	--	3,75	X		8 000 $		3 680 $	2 000 $	2 500 $		16 180 $
200-BL-01	Sag mill liners	1	Polymet					300 000 $		60 000 $		2 500 $	362 500 $
200-TS-01	Trommel screen	1	1500 mm x 2500	7,5				18 000 $	3 608 $	4 000 $	2 500 $	1 000 $	29 108 $

Process Equipments Cost including installation 2018

Equipment List

Equip. No.	Description	Qty	Capacity or Dimensions	Power	Used Equipments			New	Installation				
				in kW	On	Purchased	Refurbishing	Equipments:	Electrical	Mechanical	Piping:	Transport:	Total:
200-CH-04	Sag mill discharge chute	1	--	--				8 500 $		3 500 $	4 500 $	2 500 $	19 000 $
200-CP-01	Sag discharge pumps box	1	10,0 m^3	--				15 000 $		3 000 $	8 000 $	2 000 $	28 000 $
200-PB-01	Cyclone feed pumps	2	203 mm x 203	37,3	X				4 511 $	7 500 $	24 000 $	2 000 $	38 011 $
200-PB-02	Cyclone feed pumps	2	203 mm x 203	37,3	X				4 511 $	7 500 $	24 000 $	2 000 $	38 011 $
200-CY-01	Sag recirculating cyclones	2	500 mm					12 000 $		4 000 $	10 000 $	1 500 $	27 500 $
200-CY-02	Sag recirculating cyclones	2	500 mm					12 000 $		4 000 $	10 000 $	1 500 $	27 500 $
200-CH-05	Ball mill feed chute	1	--	--				5 000 $		2 000 $	4 000 $		11 000 $
200-BB-02	Ball mill	1	3 048 mm dia. x 3,66	590	X		15 000 $		77 000 $	135 000 $		55 000 $	282 000 $
200-BL-02	Ball mill liners							130 000 $		40 000 $		3 500 $	173 500 $
200-LU-03	Ball mill gears lube unit	1	--	3,75			3 000 $		3 680 $	2 000 $	5 000 $		13 680 $
200-TS-02	Trommel screen	1	--	--				12 000 $		2 000 $	3 000 $	3 000 $	20 000 $
200-CH-06	Ball mill discharge chute	1	--	--				6 000 $		4 000 $	4 000 $	1 500 $	15 500 $
200-CH-07	Ball mill feed chute	1	--	--				5 000 $		2 000 $	4 000 $	1 500 $	12 500 $
200-BB-03	Ball mill	1	3 048 mm dia. x 3,66	590	X		15 000 $		77 000 $	135 000 $		55 000 $	282 000 $
200-BL-03	Ball mill liners							130 000 $		40 000 $		1 500 $	171 500 $
200-LU-04	Ball mill gears lube unit	1	--	3,75			3 000 $		3 680 $	2 000 $	5 000 $		13 680 $
200-TS-02	Trommel screen	1	--	--				12 000 $		2 000 $	2 500 $	3 000 $	19 500 $
200-CH-06	Ball mill discharge chute	1	--	--				6 000 $		4 000 $	4 000 $	1 500 $	15 500 $
200-CP-02	Ball Mills discharge pumps box	1	10,0 m^3	--				6 000 $		3 000 $	8 000 $	2 000 $	19 000 $
200-PB-03	Cyclone feed pumps (335,2 m^3/h)	1	203 mm x 254	56	X				6 026 $	7 500 $	24 000 $	2 000 $	39 526 $
200-PB-04	Cyclone feed pumps (335,2 m^3/h)	1	203 mm x 254	56	X				6 026 $	7 500 $	24 000 $	2 000 $	39 526 $
200-CY-03/13	Cyclo-Pack c/w ajustable rubber- ni-hard vortex	10	250 mm dia.	--				11 000 $		7 000 $	19 000 $	3 500 $	40 500 $
200-TS-01	Delkor linear trash screen c/w chute		1500mm X 2750mm		X		5 000 $		2 680 $	3 000 $	5 000 $	1 500 $	17 180 $
200-IP-05	Integrate Trash pump		Sala		X				2 680 $	1 500 $	5 000 $		9 180 $
200-PB-05	Flotation feed pumps		203 mm x 254	26,6	X				4 180 $	7 500 $	12 000 $		23 680 $
200-PB-06	Flotation feed pumps		203 mm x 254	22,4	X				4 180 $	3 000 $	3 000 $		10 180 $
200-SP-01	Sump pump - Grinding area	1	75 mm	11,2				14 000 $	2 087 $	2 500 $	7 000 $		25 587 $
200-OC-01	Overhead crane - Grinding area	1	25 tonnes 5 tonnes	18,6	X			8 000 $	2 687 $	6 000 $			16 687 $
200-JG-01	Sag mill jig crane							8 000 $					8 000 $
200-EM-01	Electromagnet for balls	1	--	2				4 500 $	3 680 $	5 000 $		500 $	13 680 $
200-GO-01	Ball bucket	1	--	--				5 500 $				500 $	6 000 $
TOTAL GRINDING:				**2946**			**77 000 $**	**1 245 000 $**	**294 136 $**	**731 500 $**	**230 500 $**	**169 000 $**	**2 747 136 $**
ZINC/SILVER FLOTATION SECTION													
300-TK-01	Zn/Ag Conditioner tank	1	35m^3 3,0m X 6,2m	--				21 500 $		6 000 $	5 000 $	1 500 $	34 000 $
300-AG-01	Zn/Ag Conditioner agitator	1		11,2				24 000 $	9 580 $	5 000 $	3 000 $	1 000 $	42 580 $
300-TK-02	Zn/Ag Rougher flotation cell tank	3	50m^3		X		75 000 $			7 000 $	15 000 $	2 500 $	99 500 $
300-AG-02	Zn/Ag Rougher flotation cell agitator			33,6	X		25 000 $		11 095 $	5 000 $		1 500 $	42 595 $

MINES
ABCOURT
MINES INC.

Process Equipments Cost including installation 2018

Equipment List

Equip. No.	Description	Qty	Capacity or Dimensions	Power in kW	Used Equipments On	Used Equipments Purchased	Used Equipments Refurbishing	New Equipments:	Installation Electrical	Installation Mechanical	Installation Piping:	Transport:	Total:
300-AV-01	Auto oparating valve						5 000 $	5 000 $	3 000 $	2 000 $	3 000 $	2 500 $	20 500 $
300-TK-03	Zn/Ag Scanvenger flotation cell tank		50m³		X		75 000 $			7 000 $	15 000 $	2 500 $	99 500 $
300-AG-03	Zn/Ag Scavenger flotation cell agitator			33,6	X		25 000 $		11 095 $	5 000 $		1 500 $	42 595 $
300 IP-01/02	Rougher and scanvenger to cleaners with pumps	2	3 X 3 SRL	7,5	X				3 680 $	2 500 $	5 000 $		11 180 $
300-PB-01	Scanvenger tailing to pyrite conditionner		70m³	14,92				42 000 $		5 000 $	5 000 $	2 000 $	54 000 $
300-AG-04	Conditionner agitator							32 000 $	11 095 $	5 000 $		1 500 $	49 595 $
300-TK-04	Zn/Ag 1st cleaner flotation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
300-AG-05	1 st cleaner agitator	1		14,8	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
300-TK-05	Zn/Ag 2st cleaner flotatation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
300-AG-05	2 st cleaner agotator	1		7,5	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
300-PB-03	1 st and 2 st cleaning to concentrate thick. pumps	1	3 X 3 SRL	7,5	X				3 180 $	2 000 $	3 000 $	1 000 $	9 180 $
300-PB-04	1 st and 2 st cleaning to concentrate thick. pumps	1	3 X 3 SRL	7,5	X				3 180 $	2 000 $	3 000 $	1 000 $	9 180 $
300-PB-05	2 st cleaner tails to zinc to cond. tank pumps	1	3 X 3 SRL	7,5	X				3 180 $	2 000 $	3 000 $	1 000 $	9 180 $
300-PB-06	2 st cleaner tails to zinc to cond.tank pumps	1	3 X 3 SRL	7,5	X				3 180 $	2 000 $	3 000 $	1 000 $	9 180 $
300-SP-01	Zinc flotation area sump pump	1	75mm	11,2				15 000 $	3 180 $	2 000 $	3 000 $	1 000 $	24 180 $
TOTAL ZINC FLOTATION::				**164**			**255 000 $**	**139 500 $**	**71 805 $**	**75 500 $**	**76 000 $**	**25 500 $**	**643 305 $**
PYRITE FLOTATION SECTION													
350-PB-01	Pyrite conditionner feed pump	1	203 mm x 254	29,6	X				3 180 $	3 000 $	5 000 $		11 180 $
350-TK-01	Pyrite flotation conditioner tank	1	70m³	--	X			75 000 $		7 000 $	15 000 $	2 500 $	99 500 $
350-AG-01	Conditioner tank agitator	1	1067 mm dia.	18,7	X			25 000 $	11 095 $	5 000 $		1 500 $	42 595 $
350-TK-02	Pyrite Rougher flotation cells	1	50m³		X		75 000 $			8 000 $	15 000 $	2 500 $	100 500 $
350-AG-02	Rougher cell agitator			33,9	X		25 000 $		3 680 $	4 000 $			32 680 $
350-TK--03	Pyrite Cleaner flotation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
350-AG-03	Pyrite Cleaner flotation cell agitator			14,8	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
350-TK--04	Pyrite Cleaner flotation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
350-AG-04	Pyrite Cleaner flotation cell agitator			14,8	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
350-TK--05	Pyrite Cleaner flotation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
350-AG-05	Pyrite Cleaner flotation cell agitator			14,8	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
350-TK--06	Pyrite Cleaner flotation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
350-AG-06	Pyrite Cleaner flotation cell agitator			14,8	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
350-TK--07	Pyrite Cleaner flotation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
350-AG-07	Pyrite Cleaner flotation cell agitator			14,8	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
350-TK--08	Pyrite Cleaner flotation cell tank	1	10m³		X		15 000 $			5 000 $	4 000 $	1 000 $	25 000 $
350-AG-08	Pyrite Cleaner flotation cell agitator			14,8	X		10 000 $		3 180 $	3 000 $	1 000 $	1 000 $	18 180 $
350-PB-1/02	Integrated tank pumps tails to pyrite thickener	2	5 X 5 SRL	22,2	X				6 360 $		5 000 $		11 360 $
350-PB-03/04	Integrated tank pumps pyrite to sulfur tailing pond	2	5 X 5 SRL	22,2	X				6 360 $		5 000 $		11 360 $
350-FW-01	Tailing thickener feed well							3 500 $		3 000 $	3 500 $	500 $	10 500 $

MINES ABCOURT MINES INC.

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Equip. No.	Description	Qty	Capacity or Dimensions	Power	Used Equipments			New	Installation				
				in kW	On	Purchased	Refurbishing	Equipments:	Electrical	Mechanical	Piping:	Transport:	Total:
350-TK-02	Tailing thickener tank							120 000 $		4 000 $	6 000 $	2 000 $	132 000 $
350-AG-02	Tailing thickener rakes drive	1		2,2				40 000 $	2 830 $	8 000 $		1 500 $	52 330 $
350-LD-02	Thickener linting divise	1		2,2				30 000 $	2 830 $	4 000 $		1 000 $	37 830 $
350-PB-03/04	Pyrite tailings pumps (104.2 m^3/h)	1	127 mm x 100 mm	74,6	X				20 480 $	7 200 $	8 000 $	500 $	36 180 $
350-PB-03/04	Pyrite tailings pumps (104.2 m^3/h)	1	127 mm x 100 mm	74,6	X				20 480 $	7 200 $	8 000 $	500 $	36 180 $
350-PP-01	Pyrite Flotation area sump pump	1	50 mm dia.	7,5				21 000 $	6 805 $	3 600 $	7 000 $	500 $	38 905 $
300-PR-01	Overhead crane - Flotation area	1	3,2 t capacity	5,5	X		5 000 $		5 305 $	5 000 $		1 500 $	16 805 $
TOTAL PYRITE FLOTATION:				**352,4**			**255 000 $**	**314 500 $**	**108 485 $**	**117 000 $**	**107 500 $**	**26 500 $**	**928 985 $**
Zn/Ag CONCENTRATE DEWATERING SECTION													0 $
400-FW-01	Concentrate thickener feed well	1	--	--			2 000 $			3 000 $	3 500 $	500 $	9 000 $
400-TK-01	Concentrate thickener tank	1	6500 mm dia.	--	X		15 000 $			4 000 $	6 000 $	2 000 $	27 000 $
400-AG-01	Thickener Hydraulic drive mechanism c/w rakes	1	**Tank not Included**	2,5	X		5 000 $		3 055 $	8 000 $		1 500 $	17 555 $
400-LD-01	Thickener lifting divise			2,5	X		2 000 $		2 680 $	4 000 $		1 000 $	9 680 $
400-PB-01/02	Thickener U/F pumps Peristaltic c/w motor and drive	2	32 mm x 25,4 mm	11,2				45 000 $	10 760 $	3 600 $	6 000 $	500 $	65 860 $
400-TK-02	Zn/Ag Concentrate surge tank	1	2,5 m dia. x 2,9 m	--				10 032 $		3 000 $	3 500 $	1 000 $	17 532 $
400-AG-02	Surge tank agitator	1	750 mm dia.	11,2				18 900 $	9 580 $	5 000 $		500 $	33 980 $
400-PB-03/04	Pressure filter feed pumps	2	32 mm x 25,4 mm	5,6				5 900 $	6 560 $	3 600 $	5 500 $	500 $	22 060 $
400-FP-01	Metso Horizontal Pressure filter req. Ag. Ag. Concen	1	13,2 m² METSO	--		215 800 $	25 000 $		5 180 $	16 000 $	40 000 $	3 500 $	305 480 $
400-RE-03	Filtrate release tank	1	--	--		Included				3 000 $	12 000 $	500 $	15 500 $
400-PS-01	Filtrate pump	1	32 mm dia.	2,25		Included			2 780 $	3 600 $	6 500 $	500 $	13 380 $
400-CH-01	Pressure filter discharge chute	1	--	--		Included				5 000 $		500 $	5 500 $
400-SP-01	Concentrate area sump pump	1	50 mm	3,75				15 000 $	3 993 $	5 000 $	7 000 $	500 $	31 493 $
400-PR-01	Overhead crane - filtration area	1	2 t capacity	1	X		5 000 $		1 930 $	6 000 $		1 500 $	14 430 $
TOTAL ZINC DEWATERING:				**40**		**215 800 $**	**54 000 $**	**94 832 $**	**46 518 $**	**72 800 $**	**90 000 $**	**14 500 $**	**588 450 $**
CHEMICAL REAGENTS SECTION													
600-SI-01	Lime silo	1	4700 dia. x 16500	--	X		10 000 $			2 500 $		6 000 $	18 500 $
600-DE-01	Lime silo dust collector	1	--	--	X		2 000 $			1 000 $	3 500 $	1 000 $	7 500 $
600-AC-01	Lime silo agitator	1	--	1,1				9 500 $	3 680 $	1 500 $		500 $	15 180 $
600-AL-01	Screw feeder	1	152 mm dia.	0,75				16 000 $	3 680 $	1 500 $		500 $	21 680 $
600-EX-01	Lime slacker	1	640 kg/h	1,5				35 000 $	3 680 $	2 500 $	2 500 $	500 $	44 180 $
600-CV-01	Lime slacker grit screw	1	152 mm dia.	0,4				included	3 680 $	1 500 $			5 180 $
600-BO-01	Grit box	1	--	--				included		500 $			500 $
600-RE-01	Lime holding tank	1	4700 dia. x 5000	--				33 195 $		1 500 $	4 000 $	500 $	39 195 $
600-AG-01	Lime holding tank agitator	1	1410 mm dia.	5,6				17 500 $	5 380 $	2 000 $		250 $	25 130 $
600-PB-01/02	Lime distribution pumps	2	32 mm x 25,4 mm	7,5				15 300 $	7 985 $	3 600 $	4 000 $	250 $	31 135 $
600-PP-01	Lime preparation area sump pump	1	50 mm	2,3				12 500 $	2 905 $	1 800 $	8 000 $	500 $	25 705 $
600-TR-02	$CuSO_4$ hopper	1	--	--				6 500 $		500 $		250 $	7 250 $

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600-RE-04	$CuSO_4$ mixing tank	1	2000 mm x 2200	--				500 $		1 000 $	4 000 $	250 $	5 750 $
600-AG-03	$CuSO_4$ mixing tank agitator	1	667 mm	0,75				9 000 $	3 680 $	1 000 $	4 000 $	250 $	17 930 $
600-PS-04	$CuSO_4$ transfer pump	1	38 mm x 38	1,5				4 000 $	3 680 $	1 800 $	4 000 $	250 $	13 730 $
600-RE-05	$CuSO_4$ holding tank	1	2000 mm x 2000	--				5 000 $		1 000 $	3 500 $	500 $	10 000 $
600-PS-05/06	$CuSO_4$ distribution pumps	2	--	2,2				4 000 $	4 860 $	1 500 $	4 000 $	250 $	14 610 $
600-RE-06	SO_2 tank c/w accessories	1	--	--				20 000 $		4 000 $	6 000 $	500 $	30 500 $
600-TR-04	PAX hopper	1	--	--				5 000 $		500 $		250 $	5 750 $
600-RE-08	PAX mixing tank	1	2000 mm x 2200	--				5 000 $		1 000 $	4 000 $	500 $	10 500 $
600-AG-05	PAX mixing tank agitator	1	667 mm	0,75				9 000 $	3 680 $	1 000 $		250 $	13 930 $
600-PS-08	PAX transfer pump	1	38 mm x 38	1,1				4 000 $	3 480 $	1 800 $	4 000 $	250 $	13 530 $
600-RE-09	PAX holding tank	1	2000 mm x 2000	--				5 000 $		1 000 $	3 500 $	250 $	9 750 $
600-PS-09	PAX distribution pump	1	--	1,2				2 500 $	3 680 $	1 500 $	4 000 $	250 $	11 930 $
600-RE-10	MIBC holding tank	1	1370 mm x 1500	--				5 000 $		1 000 $		500 $	6 500 $
600-PD-01/05	MIBC metering pumps	5	--	0,5				9 000 $	9 650 $	1 000 $	4 000 $	250 $	23 900 $
600-RE-11	3418 A holding tank	1	1370 mm x 1500	--				5 000 $		1 000 $		1 500 $	7 500 $
600-PD-06/08	3418 A metering pumps	3	--	0,3				4 000 $	3 430 $	1 000 $	4 000 $	250 $	12 680 $
600-UF-01	Flocculent preparation packaged unit by supplier	1	--	4				57 000 $	4 180 $	1 800 $	6 000 $	1 000 $	69 980 $
600-PC-01/03	Progressive cavity pumps	3	--	3,7				11 000 $	3 955 $	1 500 $	4 000 $	500 $	20 955 $
600-MO-01	Reagent area hoist & monorail	1	2 t capacity	0,7				7 500 $	3 705 $	2 000 $		500 $	13 705 $
600-PP-02	Reagent area sump pump	1	50 mm	7,46				12 000 $	6 775 $	1 800 $	7 000 $	500 $	28 075 $
TOTAL REAGENTS FACILITIES:				**43**			**12 000 $**	**328 995 $**	**85 745 $**	**48 600 $**	**88 000 $**	**19 000 $**	**582 340 $**
SERVICE SECTION													0 $
700-CR-01	Courier Outotec c/w with sampler and return pumps	6	Sampling points					800 000 $	75 000 $	30 000 $	25 000 $	1 500 $	931 500 $
700-CP-01	Service air compressor	1	1900 m³/h	112				10 000 $	11 200 $	4 000 $	35 000 $	1 000 $	61 200 $
700-RE-01	Air receiver	1	8 m³	--				12 000 $		4 000 $	15 000 $	500 $	31 500 $
700-CP-02	Instrumentation air compressor	1	360 m³/h	37,4				48 000 $	8 220 $	4 000 $	20 000 $	750 $	80 970 $
700-RE-02	Instrumentation air receiver	1	1 m³	--				6 000 $		3 000 $	5 000 $	500 $	14 500 $
700-AS-01	Instrumentation air dryer	1	--	--				8 000 $	3 500 $	3 000 $	5 000 $	500 $	20 000 $
700-SA-01	Flotation air blowers	2	7000 Nm³/h each	37,1				110 000 $	21 600 $	4 000 $	60 000 $	1 000 $	196 600 $
700-SA-02	Flotation air blowers	2	7000 Nm³/h each	37,1				110 000 $	21 600 $	4 000 $	60 000 $	1 000 $	196 600 $
700-BA-01	Truck scale	1	--	--				40 000 $	4 500 $	3 600 $		1 000 $	49 100 $
700-PE-11	Fire protection pump (115 m^3/h)	1	125 mm x 100	20				2 200 $	9 680 $	3 600 $	10 000 $	500 $	25 980 $
700-PE-12	Jockey pump (5,0 m^3/h)	1	40 mm x 32	3,75				9 000 $	3 993 $	3 600 $	5 000 $	250 $	21 843 $
700-PE-13	Diesel pump (115 m^3/h)	1	125 mm x 100	--				34 000 $		5 600 $	8 000 $	500 $	48 100 $
700-PE-09	Raw water pumps	2	203 mm dia.	22,2				34 000 $	18 450 $	4 000 $	20 000 $	500 $	76 950 $
700-PE-10	Raw water pumps	2	203 mm dia.	22,2				34 000 $	18 450 $	4 000 $	20 000 $	500 $	76 950 $
700-BF-01	Raw water pumps float barge	1	--	--				7 500 $		4 000 $		500 $	12 000 $

MINES ABCOURT MINES INC.

Process Equipments Cost including installation 2018

Equipment List

Equip. No.	Description	Qty	Capacity or Dimensions	Power	Used Equipments			New	Installation				
				in kW	On	Purchased	Refurbishing	Equipments:	Electrical	Mechanical	Piping:	Transport:	Total:
700-PE-01/02	Recycled water pumps (180 m^3/h each)	2	254 mm dia.	29,8				42 000 $	9 600 $	4 000 $	20 000 $	1 500 $	77 100 $
700-BF-02	Recycled water pumps float barge	1	--	29,8				7 500 $		4 000 $		500 $	12 000 $
700-RE-04	Process water tank (500 m^3)	1	8,9 m x 9,2	--				135 000 $		4 000 $	5 000 $	1 500 $	145 500 $
700-PE03	Process water pumps (120 m^3/h each)	2	125 mm x 100	14,8				9 000 $	8 945 $	7 200 $	20 000 $	500 $	45 645 $
700-PE-04	Process water pumps (120 m^3/h each)	2	125 mm x 100	14,8				9 000 $	8 945 $	7 200 $	20 000 $	500 $	45 645 $
700-RE-05	Fresh water tank (380 m^3)	1	8,0 m x 8,1	--				90 000 $		4 000 $		1 500 $	95 500 $
700-PE-05	Freshwater pumps (70 m^3/h each)	2	75 mm x 50	14,9				9 000 $	12 860 $	7 200 $	20 000 $	500 $	49 560 $
700-PE-06	Freshwater pumps (70 m^3/h each)	2	75 mm x 50	14,9				9 000 $	12 860 $	7 200 $	20 000 $	500 $	49 560 $
700-PE-07	Seal water pumps (25m^3/h each)	2	50 mm x 40	11,9				9 000 $	6 860 $	7 200 $	35 000 $	500 $	58 560 $
700-PE-08	Seal water pumps (25m^3/h each)	2	50 mm x 40	11,9				9 000 $	6 860 $	7 200 $	35 000 $	500 $	58 560 $
TOTAL SERVICES:				435				**1 593 200 $**	**263 123 $**	**143 600 $**	**463 000 $**	**18 500 $**	**2 481 423 $**
SUB TOTAL:	**Sub -Total: Connected**			**4274**	**0 $**	**225 800 $**	**692 000 $**		**977 361 $**	**1 384 500 $**	**1 059 000 $**	**301 200 $**	**9 015 888 $**
	Spare équipements connected			**-386,8**									
	Base consumption in kW			**3887**									
WATER TREATMENT PLANT													
Lime plant													
1200-LB-01	Lime bin	1	75 tonnes		X		15 000 $			6 000 $		30 000 $	51 000 $
1200-BV-01	Bin vibrator	1	Air		X		500 $			200 $			700 $
1200-DC-01	Dust Collector	1	BV-500-245		X		1 500 $			2 000 $			3 500 $
1200-SV-01	Screw conveyor	1		2,20	X				1 500 $	1 000 $			2 500 $
1200-AD-01	Air dryer	1	16l/s		X				1 500 $	2 000 $			3 500 $
1200-AC-01	Air compressor	1	Piston 50'³/min	7,46	X				2 500 $	3 000 $			5 500 $
1200-AG-01	Lime mixer	1	2-Hélices	1,12	X					2 500 $	6 000 $		8 500 $
1200-R1-01	Lime tank	1	3,m X 3,7M H	3,73	X		1 500 $						1 500 $
1200-PP-01	Sump pump	1	50mm	7,46	X				2 500 $	1 500 $	2 500 $		6 500 $
1200-PE-01	Surpression pump	1	60 USGPM	3,73	X		1 500 $		2 500 $	1 500 $	3 000 $		8 500 $
1200-RE-02	Water tank	1	1,5m X 1,8m		X		2 000 $			1 000 $	6 000 $		9 000 $
1200-PE-02	Fresh water pump	1	75mmX50mm	7,46	X		1 500 $		2 500 $	2 500 $	4 000 $		10 500 $
1200-GS-01	Diesel generator	1	DG6OP3,54. 5KW		X		3 000 $		1 500 $				4 500 $
1200-RE-013	Fuel tank	1	500 gallons		X					500 $	500 $		1 000 $
1200-EF-01/04	Exaust fans	4	24"	0,75	X				1 000 $	2 800 $			3 800 $
Total:							26 500 $		15 500 $	26 500 $	22 000 $		90 500 $
Water treatment section:													
1200-RE-01	Floculation tank	1	3m X 3,75					14 200 $		5 000 $	2 500 $	1 500 $	23 200 $
1200-AG-01	Mixer	1		7,46				22 000 $	6 800 $	8 000 $		1 000 $	37 800 $
1200-RE-01	Coaculation tank	1	3m X 3,25					10 200 $		5 000 $	2 500 $	1 500 $	19 200 $
1200-AG-01	Mixier	1		7,46				22 000 $	6 800 $	8 000 $		1 000 $	37 800 $
1200-RE-01	Settelment tank	1	3m X 2,75					10 200 $		5 000 $	2 500 $	1 500 $	19 200 $

MINES ABCOURT MINES INC.

Process Equipments Cost including installation 2018

Equipment List

Equip. No.	Description	Qty	Capacity or Dimensions	Power	Used Equipments			New	Installation				
				in kW	On	Purchased	Refurbishing	Equipments:	Electrical	Mechanical	Piping:	Transport:	Total:
1200-AG-01	Mixer	1		7,46				22 000 $	6 800 $	8 000 $		1 000 $	37 800 $
1200-PP-01	Pompe puisard	1	50mm	7,46				21 000 $	6 805 $	3 600 $	7 000 $	500 $	38 905 $
1200-SF-01	Floculent system	1	Skid mounted	0,25				49 000 $	1 500 $	1 000 $	500 $	500 $	52 500 $
1200-PD-01	Floculent pump	1	Meetering VS	0,10				6 200 $		500 $	500 $	250 $	7 450 $
1200-CS-01	Coaculent system	1	Skid mounted	0,25				7 800 $	1 500 $	1 000 $	500 $	500 $	11 300 $
1200-PD-02	Coaculent pump	1	Meetering VS	0,10				6 200 $		500 $	500 $	250 $	7 450 $
1200-PD-03	Acid pump	1	Meetering VS	0,10				6 200 $		500 $	500 $	250 $	7 450 $
1200-FM-01	Flowmeter	1	0-2000gal./minute					18 000 $	3 000 $	2 000 $	Pip. Section	500 $	23 500 $
1200-PC-01	Parshall Canal	1	600mm					24 000 $	1 500 $	4 000 $	Pip. Section	1 500 $	31 000 $
1200-SP-00	Water Sampler	1	Online					8 500 $	3 000 $	2 500 $	Pip. Section	500 $	14 500 $
1200-PH-00	online pH meter	2	Online					5 000 $	1 500 $	500 $	Pip. Section	250 $	7 250 $
1200-SM-01	Satic mixer	1	300mm							1 500 $		750 $	2 250 $
	Total water treatment plant							252 500 $	39 205 $	56 600 $	17 000 $	13 250 $	378 555 $
Total for water treatment equipments installation								**252 500 $**	**54 705 $**	**83 100 $**	**39 000 $**	**13 250 $**	**469 055 $**
LABORATORY SECTION													
1000-LA-01	Jaw Crusher	1		2,25				5 000,0 $	1 680,0 $	1 200,0 $		500,0 $	8 380 $
1000-LA-02	Lab Crusher Gyroll	1		0,375				7 000,0 $	1 680,0 $	1 200,0 $		500,0 $	10 380 $
1000-LA-03	Bico Pulvirizer	1		2,25				5 000,0 $	1 680,0 $	1 200,0 $		500,0 $	8 380 $
1000-LA-04	Shift Shaker	1		0,125				2 000,0 $	1 680,0 $	500,0 $		500,0 $	4 680 $
1000-LA-05	Wet Sieving Kit	1						400,0 $					400 $
1000-LA-06	Ohaus Plateform Balance	1						2 000,0 $		500,0 $		500,0 $	3 000 $
1000-LA-07	Ohaus Dialogram Balance	1						500,0 $		1 000,0 $		500,0 $	2 000 $
1000-LA-08	Chan Automatic Micro Balance	1						14 000,0 $				500,0 $	14 500 $
1000-LA-09	One Batch Vaccuum Filter	1						1 000,0 $					1 000 $
1000-LA-10	Miscellaneous Filter Equipment	1						100,0 $	1 000,0 $	1 000,0 $			2 100 $
1000-LA-11	Atomic Absorbtion c/w Accessories	1						35 000,0 $	1 000,0 $	1 000,0 $		500,0 $	37 500 $
1000-LA-12	Leeco Sulfur Analizer and Accessories	1						65 000,0 $	1 500,0 $	35 000,0 $	500,0 $	500,0 $	102 000 $
1000-LA-13	Fume Hood, Benches, Glassware, etc…	1						85 000,0 $	28 000,0 $		1 000,0 $	500,0 $	113 500 $
	Laboratory Electric Hardware and Installation	1							17 500,0 $		3 000,0 $		17 500 $
	TOTAL LABORATORY			**110**				**222 000,0 $**	**55 720,0 $**	**42 600,0 $**	**4 500,0 $**	**5 000,0 $**	**325 320 $**
Grand trotal for equipments instaled for concentrator and water treatment						**225 800 $**	**692 000 $**	**474 500 $**	**1 072 286 $**	**1 483 700 $**	**1 080 500 $**	**319 450 $**	**9 810 263 $**

APPENDIX 9

Process Plant Concrete and Steel (ref. Section 21)

Abcourt-Barvue Project 1 800 metric tonnes pes day:

Structural and Architectural

Description	Units		Men/hours	Total $ Material	Total $ Men/hours	Total $
CRUSHING PLANT						
Structural heavy steel suronding crusher	15	t	340	75 000 $	28 900 $	103 900 $
Structural light steel (With equipment)						
Inclined grizzly (With equipments)						
Crusher hopper 50 tonnes (With equipments)						
Steel Grating	100	m²	275	22 500 $	23 375 $	45 875 $
Handrails	100	m	230	22 500 $	19 550 $	42 050 $
Stairs c/w handrails	12	m	40	14 400 $	3 400 $	17 800 $
Conveyor # 1,2 and 3 with equipements						
Bin ladder	16	m	35	3 000 $	2 975 $	5 975 $
Ore bin is with equipments list						
Total Crushing				137 400 $	78 200 $	215 600 $
GRINDING AREA						
Sag feed conveyor (With equipments)						
Conveyor belt cover 2, 3 and 4	9,8	t	250	3 920 $	21 250 $	25 170 $
Conveyor belt cover	82	m²	320	32 800 $	27 200 $	60 000 $
Structural heavy steel	80	t	1500	400 000 $	127 500 $	527 500 $
Structural light steel	10	t	300	50 000 $	25 500 $	75 500 $
Steel Grating	200	m²	700	45 000 $	59 500 $	104 500 $
Handrails	100	m	450	25 000 $	38 250 $	63 250 $
Stairs c/w handrails	70	m	200	84 000 $	17 000 $	101 000 $
Total Grinding				640 720 $	316 200 $	956 920 $
FLOTATION						
Structural heavy steel	60	t	1750	300 000 $	148 750 $	448 750 $
Structural light steel	6	t	75	30 000 $	6 375 $	36 375 $
Steel Grating	200	m	350	45 000 $	29 750 $	74 750 $
Handrails	35	m	75	8 750 $	6 375 $	15 125 $
Stairs c/w handrails	30	m	65	36 000 $	5 525 $	41 525 $
Total Flotation				419 750 $	196 775 $	616 525 $
REAGENT AREA						
Structural heavy steel	10	t	250	50 000 $	21 250 $	71 250 $
Structural light steel	5	t	100	25 000 $	8 500 $	33 500 $
Steel Grating	25	m²	60	5 625 $	5 100 $	10 725 $
Handrails	10	m	40	2 500 $	3 400 $	5 900 $
Stairs c/w handrails	8	m	20	9 600 $	1 700 $	11 300 $
Total Reagent				92 725 $	39 950 $	132 675 $
PRESSES FILTRATION						
Structural heavy steel	20	t	415	100 000 $	35 275 $	135 275 $
Structural light steel	4	t	125	20 000 $	10 625 $	30 625 $
Stell deck (Concrete floor)	278	m²	32	7 506 $	2 720 $	10 226 $
Handrails	40	m	60	9 000 $	5 100 $	14 100 $
Ladder	12	m	16	2 500 $	1 360 $	3 860 $
Total Dewatering				139 006 $	55 080 $	194 086 $
CONCENTRATE STORAGE						
Maded with concrete						
Building is included with plant building						

Abcourt-Barvue Project 1 800 metric tonnes pes day:

Structural and Architectural

Description	Units		Men/hours	Total $ Material	Total $ Men/hours	Total $
Total Concentrate Storage						
CRUSHER BUILDING						
Metal roof deck insulated	84	m²	90	4 620 $	7 650 $	12 270 $
Roof cladding un-insulated	84	m²	60	2 100 $	5 100 $	7 200 $
Wall cladding insulated	222	m²	100	12 210 $	8 500 $	20 710 $
Walls cladding un-insulated	222	m²	80	5 550 $	6 800 $	12 350 $
Men doors	2		60	1 400 $	5 100 $	6 500 $
Truck door	1		120	10 000 $	10 200 $	20 200 $
Bin concrete wall cladding insulated	440	m²	220	24 200 $	18 700 $	42 900 $
Total crusher Building				60 080 $	62 050 $	122 130 $
PROCESSING PLANT BUILDING						
Metal roof deck insulated	2250	m²	2600	162 000 $	221 000 $	383 000 $
Roof cladding un-insulated	2250	m²	1600	74 250 $	136 000 $	210 250 $
Wall cladding insulated	3225	m²	3000	232 200 $	255 000 $	487 200 $
Walls cladding un-insulated	3225	m²	1500	106 425 $	127 500 $	233 925 $
Men doors	6		180	4 200 $	15 300 $	19 500 $
Truck doors	4		480	40 000 $	40 800 $	80 800 $
Total Processing plant Building				619 075 $	795 600 $	1 414 675 $
LABORATORY BULDING						
Metal roof deck insulated	240	m²	250	17 280 $	21 250 $	38 530 $
Roof cladding un-insulated	240	m²	160	7 920 $	13 600 $	21 520 $
Wall cladding insulated	225	m²	175	16 200 $	14 875 $	31 075 $
Walls cladding un-insulated	225	m²	150	7 425 $	12 750 $	20 175 $
Men doors	3		90	2 100 $	7 650 $	9 750 $
Truck doors	1		120	10 000 $	10 200 $	20 200 $
Total Laboratory Building				60 925 $	80 325 $	141 250 $
WATER TREATMENT PLANT						
Metal roof deck insulated	120	m²	125	8 640 $	10 625 $	19 265 $
Roof cladding un-insulated	120	m²	80	3 960 $	6 800 $	10 760 $
Wall cladding insulated	175	m²	135	12 600 $	11 475 $	24 075 $
Walls cladding un-insulated	175	m²	120	5 775 $	10 200 $	15 975 $
Men doors	3		90	2 100 $	7 650 $	9 750 $
Truck doors (3m x 3m)	2		120	8 000 $	10 200 $	18 200 $
Total Water treatment plant				41 075 $	56 950 $	98 025 $
MAIN ELECTRIC SUB-STATION						
Metal roof deck insulated	60	m²	60	4 320 $	5 100 $	9 420 $
Roof cladding un-insulated	77	m²	35	2 541 $	2 975 $	5 516 $
Wall cladding insulated	192	m²	192	13 824 $	16 320 $	30 144 $
Walls cladding un-insulated	192	m²	96	6 336 $	8 160 $	14 496 $
Men doors	2		60	1 400 $	5 100 $	6 500 $
Truck door	1		120	6 000 $	10 200 $	16 200 $
Tota Main Sub				34 421 $	47 855 $	82 276 $
Grand Total						3 974 162 $

ABCOURT MINES INC.

Abcourt -Barvue Project

CONCRETE AND EXCAVATION REQUIREMENT

Description	Concrete	Excavation	Backfill	Total	Total $	Total $	Total $
	M³	M³	M³	Concrete	excavation	Backfill	CAN
CRUSHING PLANT 100							
Jaw crusher hopper and retainning walls	65	125	150	57 200 $	1 500 $	2 400 $	61 100 $
Jaw crusher and retaning wall	70	125	25	61 600 $	1 500 $	400 $	63 500 $
Slab, foutage and walls button	70	200	200	33 600 $	2 400 $	3 200 $	39 200 $
Anchor Bolts and Grout	2,6			2 288 $			2 288 $
Electric rooms wall blocks	10t						18 000 $
Total Crusher area							184 088 $
Belt conveyor No. 1	8	30	20	7 040 $	360 $	320 $	7 720 $
Belt conveyor No. 2	22	75	60	19 360 $	900 $	960 $	21 220 $
Belt conveyor No. 3	22	75	60	19 360 $	900 $	960 $	21 220 $
Total Crushing							50 160 $
PROCESS PLANT:							
Foundation Footings	60	400	340	52 800 $	4 800 $	5 440 $	63 040 $
Foundations Pedestals	70	125	100	61 600 $	1 500 $	1 600 $	64 700 $
Foundations floors	185	600	250	162 800 $	7 200 $	4 000 $	174 000 $
Anchor Bolts and Grout	4,5			3 960 $			3 960 $
Tanks, pump Box and Pumps bases	90	157	130	79 200 $	1 884 $	2 080 $	83 164 $
Misc. Offices, Dry, Lunch Room, Wash Rooms…..	15t						33 750 $
Misc. Offices, Dry, Lunch Room, Wash Rooms…..	20 t						45 000 $
Concrete blocks Compressors, electric rooms……	55 t						98 000 $
Total Process Plant							565 614 $
GRINDING SECTION 200							
Coarse ore	30	50	50	26 400 $	600 $	800 $	27 800 $
Sag mill feed conveyor	6	10	9	5 280 $	120 $	144 $	5 544 $
SAG mill	70	150	70	61 600 $	1 800 $	1 120 $	64 520 $
Ball mill	55	150	70	48 400 $	1 800 $	1 120 $	51 320 $
Ball mill	55	150	70	48 400 $	1 800 $	1 120 $	51 320 $
Ball bins	50	100	90	44 000 $	1 200 $	1 440 $	46 640 $
Total Grinding							247 144 $
ZINC/Ag. PYRITE FLOTATION 300 & 350							
Flotation cells	48	150	100	42 240 $	825 $	900 $	43 965 $
Total Zinc Ag Pyrite Flotation							43 965 $
ZINC CONCENTRATE DEWATERING SECTION 400							
Zinc Concentrate thickener	15	60	30	13 200 $	720 $	480 $	14 400 $
Zinc Concentrate surge tank	10	10	8	8 800 $	120 $	128 $	9 048 $
Tailing thickener	25	80	40	22 000 $	960 $	640 $	23 600 $
Pressure filter floor	40			35 200 $	0 $	0 $	35 200 $
Concentrate wall	40	80	25	35 200 $	960 $	400 $	36 560 $
Concentrate floor	78	75	30	68 640 $	900 $	480 $	70 020 $
Concentrate load-out	40	150	40	35 200 $	1 800 $	640 $	37 640 $
Total Zinc Dewatering							226 468 $
CHEMICAL REAGENTS SECTION 600							
Lime silo and accessories	15	20	10	13 200 $	240 $	120 $	13 560 $

Abcourt -Barvue Project

CONCRETE AND EXCAVATION REQUIREMENT

Description	Concrete	Excavation	Backfill	Total	Total $	Total $	Total $
	M³	M³	M³	Concrete	excavation	Backfill	CAN
$CuSO_4$ system	8	4	2	7 040 $	48 $	24 $	7 112 $
$H2SO_4$ tank c/w acessories	20	8	4	17 600 $	96 $	48 $	17 744 $
Kax system	8	4	2	7 040 $	48 $	24 $	7 112 $
F-357 system	8	4	2	7 040 $	48 $	24 $	7 112 $
A-208 system	8	4	2	7 040 $	48 $	24 $	7 112 $
F-507 system	8	4	2	7 040 $	48 $	24 $	7 112 $
Flocculent preparation packaged unit by supplier	2	4	2	1 760 $	48 $	24 $	1 832 $
Total Reagents							59 752 $
Miscellaneous (flotation)							
Zn Flotation Thickner	35	90	20	30 800 $	1 080 $	240 $	32 120 $
Compressors, Blower and Air Receivers	12	4	4	10 560 $	48 $	48 $	10 656 $
Vacuum pump	8	2	2	7 040 $	24 $	24 $	7 088 $
Propane tank	8	2	10	7 040 $	24 $	120 $	7 184 $
Truck scale	8	100	70	7 040 $	1 200 $	840 $	9 080 $
Process water tank (500 m^3) Flotation	20	50	40	17 600 $	600 $	480 $	18 680 $
Concrete blocks for station secondary	15t						41 000 $
Total Micellaneaous							125 808 $
Main electric station	50	50	25	34 000 $	275 $	225 $	34 500 $
Laboratory	60	55	30	40 800 $	303 $	270 $	41 373 $
Lime silo and accessories	39	200	200	26 520 $	1 100 $	1 800 $	29 420 $
Water process plant	70	600	200	47 600 $	3 300 $	1 800 $	52 700 $
Pillings required for grinding mills, lime bins and structure	250mm Dia		150m L	$250,/m			37 500 $
Total							195 493 $
Grand Total							1 698 492 $

APPENDIX 10

Detailed Cash Flow (ref. Section 22)

Services Miniers PRB Inc.
Projet Abcourt Barvue
FS 2007 Update
Cashflow - Final
rL - 2019 01 23

	Description	Unit	Parameter	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Total	Preprod'n	Prod'n
1.0	**Mine Production**																				
1.1	**Open Pit Mining**																				
	Overburden																				
	Barvue Pit	m3		200,000	0	0	250,000	0	0	0	0	0	0	0	0	0	0		450,000	200,000	250,000
	Abcourt East Pit	m3		0	1,234,000	0	0	0	0	0	0	0	0	0	0	0	0		1,234,000	0	1,234,000
	Abcourt West Pit	m3		236,000	0	0	0	0	0	0	0	0	0	0	0	0	0		236,000	236,000	0
	Total	m3		436,000	1,234,000	0	250,000	0	0	0	0	0	0	0					1,920,000	436,000	1,484,000
	Rock Mined																				
	Waste Mined	tonnes		903,902	4,703,826	4,296,654	4,780,270	5,341,322	3,956,472	3,242,733	2,162,125	1,079,274	668,666	458,977	445,305	465,271	160,678		32,665,476	903,902	31,761,575
	Marginal Ore	tonnes		17,360	72,056	73,101	55,731	48,450	37,159	23,165	18,226	17,777	12,479	14,164	14,020	14,037	0		417,725	17,360	400,365
	Total Waste & Marginal Ore	tonnes		921,261	4,775,883	4,369,755	4,836,001	5,389,772	3,993,631	3,265,898	2,180,351	1,097,051	681,145	473,142	459,325	479,308	160,678		33,083,201	921,261	32,161,940
	Ore Mined	tonnes		122,164	510,894	660,421	583,860	650,000	649,983	633,038	630,000	534,731	326,000	326,000	326,000	385,600	215,030		6,553,721	122,164	6,431,556
	Total Rock Mined	tonnes		1,043,426	5,286,776	5,030,176	5,419,860	6,039,772	4,643,615	3,898,936	2,810,351	1,631,782	1,007,145	799,142	785,325	864,908	375,708		39,636,922	1,043,426	38,593,496
	Waste & Marginal to Ore Ratio			7.5	9.3	6.6	8.3	8.3	6.1	5.2	3.5	2.1	2.1	1.5					5.0		
	Rock Hauled Out of Pits																				
	Waste Hauled	tonnes		886,562	4,623,811	4,222,976	4,947,664	5,341,322	3,956,472	3,242,733	2,162,125	1,079,274	668,666	458,977	445,305	465,271	160,678		32,661,836	886,562	31,775,274
	Marginal Ore Hauled	tonnes		11,360	58,806	64,851	51,231	48,450	37,159	23,165	18,226	17,777	12,479	14,164	14,020	14,037	0		385,725	11,360	374,365
	Total Waste & Marginal Ore Hauled	tonnes		897,922	4,682,617	4,287,827	4,998,894	5,389,772	3,993,631	3,265,898	2,180,351	1,097,051	681,145	473,142	459,325	479,308	160,678		33,047,561	897,922	32,149,640
	Ore Hauled	tonnes		118,164	498,394	646,421	650,000	650,000	649,983	633,038	630,000	534,731	326,000	326,000	326,000	385,600	215,030		6,589,361	118,164	6,471,196
	Total Rock Hauled	tonnes		1,016,086	5,181,010	4,934,248	5,648,894	6,039,772	4,643,615	3,898,936	2,810,351	1,631,782	1,007,145	799,142	785,325	864,908	375,708		39,636,922	1,016,086	38,620,836
	Ore Stockpile																				
	To Ore Stockpile	tonnes		118,164	0	0	0	0	0	0	0	0	0	0	0	0	0		118,164	118,164	0
	From Ore Stockpile	tonnes		0	97,606	3,579	0	0	17	16,962	0	0	0	0	0	0	0		118,165	0	118,165
	Ore Stockpile Inventory	tonnes		118,164	20,558	16,979	16,979	16,979	16,962	-0	-0	-0	-0	-0	-0	-0	-0				
1.2	**Underground Mining**																				
	Underground Development																				
	Ramp	m									941	34	100	0	0	0	0		1,075	0	1,075
	Sub-levels	m									299	1,078	930	1,216	1,234	138	0		4,895	0	4,895
	Loading and passing stations	m									100	100	100	0	0	0	0		300	0	300
	Sumps	m									50	50	50	0	0	0	0		150	0	150
	Stope raises	m									0	33	31	114	88	54	0		320	0	320
	Fill Raise	m									0	0	0	0	0	0	0		0	0	0
	Ventilation raise / Escapeway	m									0	480	145	0	0	0	0		625	0	625
	Diamond drilling	m									1,500	0	0	0	0	0	0		1,500	0	1,500
	Underground Production																				
	Development Ore	tonnes									20,000	60,000	60,000	60,000	60,000	400	0		260,400	0	260,400
	Production Ore	tonnes									0	55,269	264,000	264,000	264,000	264,000	113,132		1,224,401	0	1,224,401
	Total Ore Mined	tonnes									20,000	115,269	324,000	324,000	324,000	264,400	113,132		1,484,801	0	1,484,801

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	Description	Unit	Parameter	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Total	Preprod'n	Prod'n
2.0	**Processing**																				
2.1	**Mill Feed**																				
	Ore Milled	tonnes			596,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	650,000	328,162		8,074,162	0	8,074,162
	Zinc Grade	%Zn			2.53%	2.49%	2.45%	2.60%	2.94%	2.98%	2.98%	3.03%	2.98%	2.95%	2.95%	2.97%	3.03%		2.83%	0.00%	2.83%
	Silver Grade	gpt Ag			61.84	51.60	48.08	49.07	32.80	32.06	32.74	43.86	67.96	68.25	67.49	60.36	64.05		51.79	0.00	51.79
	Zinc Equivalent	% Zn Eq	0.61 %Zn/OzAg		3.74%	3.51%	3.39%	3.57%	3.59%	3.61%	3.62%	3.89%	4.31%	4.29%	4.27%	4.16%	4.29%		3.85%	0.00%	3.85%
2.2	**Recoveries**																				
	Zinc Recovery		97.5%		97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%				
	Silver Recovery		77.8%		77.8%	77.8%	77.8%	77.8%	77.8%	77.8%	77.8%	77.8%	77.8%	77.8%	77.8%	77.8%	77.8%				
2.3	**Zinc Concentrate Production**																				
	Concentration ratio				21:1	22:1	22:1	21:1	18:1	18:1	18:1	18:1	18:1	18:1	18:1	18:1	18:1				
	Humidity		5.0%		5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%				
	Concentrate produced	dmt			27,934	30,015	29,452	31,335	35,413	35,921	35,827	36,460	35,871	35,503	35,497	35,776	18,426		423,429	0	423,429
		wmt	5.0% humidity		29,330	31,516	30,924	32,902	37,183	37,717	37,618	38,283	37,665	37,278	37,272	37,565	19,348		444,601	0	444,601
	Zinc grade	%Zn			52.7%	52.7%	52.7%	52.7%	52.7%	52.7%	52.7%	52.7%	52.7%	52.7%	52.7%	52.7%	52.7%		52.7%		52.7%
	Silver grade	gpt			1,026	869	826	792	468	451	462	608	958	972	961	853	887		768		768
	Zinc recovered	tonnes			14,707	15,803	15,506	16,498	18,645	18,912	18,863	19,196	18,886	18,692	18,689	18,836	9,701		222,936	0	222,936
	Silver recovered	Oz			921,842	838,872	781,661	797,835	533,281	521,252	532,306	713,065	1,104,884	1,109,593	1,097,325	981,367	525,757		10,459,043	0	10,459,043
3.0	**Revenues**																				
3.1	**Concentrate**																				
	Quantity	dmt			27,934	30,015	29,452	31,335	35,413	35,921	35,827	36,460	35,871	35,503	35,497	35,776	18,426		423,429	0	423,429
		wmt			29,330	31,516	30,924	32,902	37,183	37,717	37,618	38,283	37,665	37,278	37,272	37,565	19,348		444,601	0	444,601
	Zinc Grade	%Zn			52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%	52.65%				
	Silver Grade	gpt			1,026	869	826	792	468	451	462	608	958	972	961	853	887				
	Zinc Contained	tonnes			14,707	15,803	15,506	16,498	18,645	18,912	18,863	19,196	18,886	18,692	18,689	18,836	9,701		222,936	0	222,936
	Silver Contained	Oz			921,842	838,872	781,661	797,835	533,281	521,252	532,306	713,065	1,104,884	1,109,593	1,097,325	981,367	525,757		10,459,043	0	10,459,043
3.2	**Prices**																				
	Exchange rate	CAD/USD	1.25 CAD/USD		1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25				
	Zinc Price	USD/lb	1.10 USD/lb		1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10				
		CAD/lb	1.38 CAD/lb		1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38				
	Silver Price	USD/Oz	16.50 USD/Oz		16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50				
		CAD/Oz	20.63 CAD/Oz		20.63	20.63	20.63	20.63	20.63	20.63	20.63	20.63	20.63	20.63	20.63	20.63	20.63				
3.3	**Metal Payable**																				
	Zinc Contained	lbs			32,355,593	34,766,337	34,113,702	36,295,520	41,018,583	41,607,181	41,498,412	42,231,666	41,549,458	41,122,583	41,116,403	41,439,461	21,343,277		490,458,175	0	490,458,175
	Zinc Payable	lbs	85.0%		27,502,254	29,551,386	28,996,647	30,851,192	34,865,795	35,366,104	35,273,650	35,896,916	35,317,039	34,954,196	34,948,943	35,223,542	18,141,785		416,889,449	0	416,889,449
	Silver Contained	Oz			921,842	838,872	781,661	797,835	533,281	521,252	532,306	713,065	1,104,884	1,109,593	1,097,325	981,367	525,757		10,459,043	0	10,459,043
	Silver Deduction	Oz	3.0 Oz/t		83,801	90,045	88,355	94,005	106,238	107,763	107,481	109,380	107,613	106,508	106,492	107,328	55,279		1,270,288	0	1,270,288
	Silver Less Deduction	Oz			838,041	748,827	693,307	703,830	427,043	413,490	424,825	603,685	997,271	1,003,086	990,833	874,038	470,478		9,188,755	0	9,188,755
	Silver Payable	Oz	77.5%		649,482	580,341	537,313	545,468	330,958	320,454	329,239	467,856	772,885	777,391	767,896	677,380	364,620		7,121,285	0	7,121,285
	Zinc Revenue	CA$	1.38 $/lb		37,815,599	40,633,156	39,870,389	42,420,389	47,940,468	48,628,392	48,501,268	49,358,260	48,560,929	48,062,019	48,054,796	48,432,370	24,944,955		573,222,992	0	573,222,992
	Silver Revenue	CA$	20.63 $/Oz		13,395,565	11,969,533	11,082,078	11,250,282	6,826,019	6,609,372	6,790,561	9,649,531	15,940,757	16,033,699	15,837,853	13,970,958	7,520,296		146,876,504	0	146,876,504
	Total	CA$			51,211,165	52,602,690	50,952,467	53,670,671	54,766,487	55,237,765	55,291,829	59,007,792	64,501,686	64,095,717	63,892,649	62,403,328	32,465,250		720,099,496	0	720,099,496
3.4	**Charges**																				
	Smelting Cost	$	212.50 $/dmt		5,935,909	6,378,181	6,258,449	6,658,722	7,525,208	7,633,192	7,613,237	7,747,759	7,622,602	7,544,288	7,543,154	7,602,422	3,915,606		89,978,729	0	89,978,729
	Rail transportation	$	160.00 $/dmt		4,692,860	5,042,515	4,947,856	5,264,308	5,949,341	6,034,712	6,018,936	6,125,287	6,026,339	5,964,425	5,963,529	6,010,385	3,095,632		71,136,125	0	71,136,125
	Handling to rail, assays, etc	$	25.00 $/dmt		733,259	787,893	773,103	822,548	929,585	942,924	940,459	957,076	941,616	931,941	931,801	939,123	483,692		11,115,020	0	11,115,020
	Total Charges	$			11,362,028	12,208,588	11,979,408	12,745,578	14,404,134	14,610,827	14,572,631	14,830,122	14,590,557	14,440,655	14,438,485	14,551,930	7,494,930		172,229,874	0	172,229,874
3.5	**Net Smelter Return**	$			39,849,136	40,394,102	38,973,059	40,925,093	40,362,353	40,626,938	40,719,198	44,177,670	49,911,129	49,655,063	49,454,164	47,851,398	24,970,320		547,869,623	0	547,869,623

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	Description	Unit	Parameter	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Total	Preprod'n	Prod'n
4.0	**Opex**																				
4.1	**Opex - Open Pit Mining**																				
	Overburden stripping	$	1.98 $/m3	865,426	2,449,395	0	496,231	0	0	0	0	0	0	0	0	0	0		3,811,052	865,426	2,945,625
	Drilling and Blasting	$	1.03 $/t mined	1,074,472	5,444,079	5,179,844	5,581,123	6,219,480	4,781,781	4,014,945	2,893,970	1,680,334	1,037,112	822,919	808,691	890,643	386,887		40,816,279	1,074,472	39,741,808
	Load & Haul	$	1.13 $/t mined	1,149,832	5,862,979	5,583,735	6,392,449	6,834,778	5,254,846	4,412,147	3,180,273	1,846,571	1,139,714	904,331	888,696	978,755	425,162		44,854,267	1,149,832	43,704,436
	Transfer from stockpile to mill	$	0.71 $/t transfer	0	69,112	2,534	0	0	12	12,010	0	0	0	0	0	0	0		83,668	0	83,668
	Production support	$	0.68 $/t mined	706,027	3,577,260	3,403,633	3,667,310	4,086,769	3,142,069	2,638,187	1,901,604	1,104,134	681,478	540,733	531,385	585,234	254,220		26,820,043	706,027	26,114,016
	Dewatering	$	0.06 $/t mined	67,000	339,475	322,998	348,020	387,826	298,176	250,359	180,458	104,780	64,671	51,315	50,427	55,538	24,125		2,545,168	67,000	2,478,167
	Technical services and Supervision	$	0.48 $/t mined	498,706	2,526,819	2,404,176	2,590,426	2,886,714	2,219,419	1,863,499	1,343,209	779,911	481,366	381,950	375,347	413,384	179,570		18,944,497	498,706	18,445,791
	Equipment Rental	$		1,375,170	2,359,057	2,109,327	2,109,327	2,570,367	734,157	0	0	0	0	0	0	0	0		11,257,405	1,375,170	9,882,235
	S.Total	$		5,736,632	22,628,174	19,006,247	21,184,887	22,985,934	16,430,461	13,191,147	9,499,515	5,515,730	3,404,341	2,701,249	2,654,546	2,923,553	1,269,963		149,132,379	5,736,632	143,395,746
	Transfer to Capex - Overburden stripping	$		-865,426	-2,449,395	0	-496,231	0	0	0	0	0	0	0	0	0	0		-3,811,052	-865,426	-2,945,625
	Transfer to Capex - Mining	$		-4,871,206															-4,871,206	-4,871,206	0
	Total Open Pit Opex	$		0	20,178,779	19,006,247	20,688,656	22,985,934	16,430,461	13,191,147	9,499,515	5,515,730	3,404,341	2,701,249	2,654,546	2,923,553	1,269,963		140,450,121	0	140,450,121
4.2	**Opex Underground Mining**																				
	Underground Development																				
	Ramp	$	2,730 $/m	0	0	0	0	0	0	0	2,568,703	93,047	273,000	0	0	0	0		2,934,750	0	2,934,750
	Sub-levels	$	2,150 $/m	0	0	0	0	0	0	0	642,850	2,318,023	1,999,070	2,614,647	2,652,467	297,193	0		10,524,250	0	10,524,250
	Loading and passing stations	$	2,090 $/m	0	0	0	0	0	0	0	209,000	209,000	209,000	0	0	0	0		627,000	0	627,000
	Sumps	$	2,050 $/m	0	0	0	0	0	0	0	102,500	102,500	102,500	0	0	0	0		307,500	0	307,500
	Stope raises	$	1,040 $/m	0	0	0	0	0	0	0	0	34,320	32,240	118,907	91,520	55,813	0		332,800	0	332,800
	Fill Raise	$	1,710 $/m	0	0	0	0	0	0	0	0	0	0	0	0	0	0		0	0	0
	Ventilation raise / Escapeway	$	3,010 $/m	0	0	0	0	0	0	0	0	1,444,800	436,450	0	0	0	0		1,881,250	0	1,881,250
	Diamond drilling	$	85 $/m	0	0	0	0	0	0	0	127,500	0	0	0	0	0	0		127,500	0	127,500
	S.total	$		0	0	0	0	0	0	0	3,650,553	4,201,690	3,052,260	2,733,554	2,743,987	353,007	0		16,735,050	0	16,735,050
	Capital Development to Capex	$		0	0	0	0	0	0	0	-3,007,703	-404,548	-584,500	0	0	0	0		-3,996,750	0	-3,996,750
	S. Total UG Development Opex	$		0	0	0	0	0	0	0	642,850	3,797,143	2,467,760	2,733,554	2,743,987	353,007	0		12,738,300	0	12,738,300
	Underground Production																				
	Development Ore	$	included in development costs		0	0	0	0	0	0	0	0	0	0	0	0	0		0	0	0
	Production Ore	$	27.38 $/t	0	0	0	0	0	0	0	0	1,513,392	7,228,925	7,228,925	7,228,925	7,228,925	3,097,813		33,526,903	0	33,526,903
	S.total UG Production Opex	$		0	0	0	0	0	0	0	0	1,513,392	7,228,925	7,228,925	7,228,925	7,228,925	3,097,813		33,526,903	0	33,526,903
	Total UG Mining Opex	$		0	0	0	0	0	0	0	642,850	5,310,534	9,696,685	9,962,478	9,972,911	7,581,931	3,097,813		46,265,203	0	46,265,203
4.3	**Opex Processing**																				
	Processing	$	13.76 $/t	0	8,200,960	8,944,000	8,944,000	8,944,000	8,944,000	8,944,000	8,944,000	8,944,000	8,944,000	8,944,000	8,944,000	8,944,000	4,515,508		111,100,468	0	111,100,468
4.4	**Environment**																				
	ESEE	$			75,000				75,000										150,000	0	150,000
	Testwork	$			144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000		1,872,000	0	1,872,000
	Total Environment			0	219,000	144,000	144,000	144,000	219,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000		2,022,000	0	2,022,000
4.5	**Opex G&A**																				
	G&A	$	1.74 M$/yr		1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000	1,740,000		22,620,000	0	22,620,000
4.6	**Opex Total**	$		0	30,338,739	29,834,247	31,516,656	33,813,934	27,333,461	24,019,147	20,970,365	21,654,264	23,929,025	23,491,727	23,455,457	21,333,484	10,767,284		322,457,792	0	322,457,792
		$/t milled			50.90	45.90	48.49	52.02	42.05	36.95	32.26	33.31	36.81	36.14	36.09	32.82	32.81		39.94		39.94
4.7	**Operating Cashflow**			0	9,510,397	10,559,854	7,456,403	7,111,160	13,028,892	16,607,791	19,748,833	22,523,405	25,982,104	26,163,335	25,998,707	26,517,914	14,203,036		225,411,831	0	225,411,831

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	Description	Unit	Parameter	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Total	Preprod'n	Prod'n
5.0	**Capex**																				
5.1	**Open Pit Mining Equipment**																				
	Production Equipment																				
	Production drill - 203 mm dia. (waste blas	$		930,000	0	0	0	0	0	0	0	0	0	0	0	0	0		930,000	930,000	0
	Production drill - 115 mm dia. (ore blasth	$		725,000	0	0	0	0	0	0	0	0	0	0	0	0	0		725,000	725,000	0
	Haulage truck 50 tonnes	$	rental/purchase	0	0	0	0	0	491,364	737,046	0	0	0	0	0	0	0		1,228,410	0	1,228,410
	Wheel loader 11.4 tonnes	$	rental/purchase	0	0	0	0	0	270,715	0	0	0	0	0	0	0	0		270,715	0	270,715
	Hydraulic shovel 7.8 tonne	$	rental/purchase	0	0	0	0	0	176,288	0	0	0	0	0	0	0	0		176,288	0	176,288
	Support Equipment																				
	Wheel loader 11.4 tonnes			0	0	0	0	0	0	270,715	0	0	0	0	0	0	0				
	Bulldozer	$	rental/purchase	0	0	0	0	0	265,930	0	0	0	0	0	0	0	0		265,930	0	265,930
	Grader	$	rental/purchase	0	0	0	0	0	143,053	0	0	0	0	0	0	0	0		143,053	0	143,053
	Water truck	$	rental/purchase	0	0	0	0	0	165,148	0	0	0	0	0	0	0	0		165,148	0	165,148
	Hydraulic shovel	$	rental/purchase	0	0	0	0	0	165,148	0	0	0	0	0	0	0	0		165,148	0	165,148
	Pickup trucks	$		180,000	0	0	0	0	0	0	0	0	0	0	0	0	0		180,000	180,000	0
	Dewatering Equipment																				
	Pompe submersible (30hp)	$		90,000	0	0	0	0	0	0	0	0	0	0	0	0	0		90,000	90,000	0
	Pompe submersible (60hp)	$		120,000	0	0	0	0	0	0	0	0	0	0	0	0	0		120,000	120,000	0
	Pompe diesel 74hp	$		140,000	0	0	0	0	0	0	0	0	0	0	0	0	0		140,000	140,000	0
	Total Mining Equipment Expenditure	$		2,185,000	0	0	0	0	1,677,645	1,007,762	0	0	0	0	0	0	0		4,870,407	2,185,000	2,685,407
5.2	**Mining Facilities**																				
	Pit Dewatering	$		687,000															687,000	687,000	0
	Marginal Stockpile Preparation	$		20,000															20,000	20,000	0
	Explosives Magazines	$		142,000															142,000	142,000	0
	Total Mining Facilities	$		849,000	0	0	0	0	0	0	0	0	0	0	0	0	0		849,000	849,000	0
5.3	**Capital Stripping and Mining (from 4.1)**															0					
	Overburden Stripping	$		865,426	2,449,395	0	496,231	0	0	0	0	0	0	0	0	0	0		3,811,052	865,426	2,945,625
	Mining	$		4,871,206	0	0	0	0	0	0	0	0	0	0	0	0	0		4,871,206	4,871,206	0
	Total Stripping and Mining	$		5,736,632	2,449,395	0	496,231	0	0	0	0	0	0	0	0	0	0		8,682,258	5,736,632	2,945,625
5.4	**Infrastructure**																				
	Road Network																				
	Improvement and construction	$		0															0	0	0
	Site services																				
	Refurbish service building	$		183,000															183,000	183,000	0
	Computers and Communications	$		158,000															158,000	158,000	0
	Refurbish Maintenance Shop	$		0															0	0	0
	Diesel Reservoir & Distribution	$		18,000															18,000	18,000	0
	Power Line																				
	Repairs to measuring station	$		26,000															26,000	26,000	0
	Power line from measuring station to Mill	$		272,000															272,000	272,000	0
	Tailings Pond																				
	Starter Dam and Internal Cell	$		2,300,000															2,300,000	2,300,000	0
	Rise - Phase 2	$				500,000													500,000	0	500,000
	Rise - Phase 3	$								490,000									490,000	0	490,000
	Rise - Phase 4	$											490,000						490,000	0	490,000
	Total Infrastructure	$		2,957,000	0	500,000	0	0	0	490,000	0	0	490,000	0	0	0	0		4,437,000	2,957,000	1,480,000
5.5	**Underground Capital Expenditures**																				
	Initial Investment	$									1,192,000								1,192,000	0	1,192,000
	UG Equipment	$		0	0	0	0	0	0	0	2,433,537	1,880,537	1,435,737	30,000	0	0	0		5,779,811	0	5,779,811
	Capitalized Development (from 4.2)	$		0	0	0	0	0	0	0	3,007,703	404,548	584,500	0	0	0	0		3,996,750	0	3,996,750
	Total UG Capital Expenditures	$		0	0	0	0	0	0	0	6,633,240	2,285,085	2,020,237	30,000	0	0	0		10,968,561	0	10,968,561
5.6	**Process Plant**																				
	Process Plant Construction Cost	$		23,258,267															23,258,267	23,258,267	0
5.7	**Owner's Costs**																				
	Owner's Costs	$		2,350,000															2,350,000	2,350,000	0
	Total Owner's Costs	$		2,350,000	0	0	0	0	0	0	0	0	0	0	0	0	0		2,350,000	2,350,000	0
5.8	**Mine Closure and Restoration**																				
	Mine Rehabilitation	$					0	0	0	0	0	0	0	0	0	0	0	3,700,000	3,700,000	0	3,700,000
	Sales of assets	$					0	0	0	0	0	0	0	0	0	0	0	-3,700,000	-3,700,000	0	-3,700,000
	Total Mine Closure and Restoration	$		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5.10	**Total Capex**	$		37,335,899	2,449,395	500,000	496,231	0	1,677,645	1,497,762	6,633,240	2,285,085	2,510,237	30,000	0	0	0	0	55,415,493	37,335,899	18,079,593
5.9	**Working Capital**																				
	Working Capital	$		4,000,000													-4,000,000		0	4,000,000	-4,000,000
5.11	**Total All-in Cost**	$		41,335,899	32,788,134	30,334,247	32,012,887	33,813,934	29,011,106	25,516,909	27,603,605	23,939,349	26,439,262	23,521,727	23,455,457	21,333,484	6,767,284	0	377,873,284	41,335,899	336,537,385
		$/t milled			55.01	46.67	49.25	52.02	44.63	39.26	42.47	36.83	40.68	36.19	36.09	32.82	20.62		46.80		41.68

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	Description	Unit	Parameter		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Total	Preprod'n	Prod'n
6.0	**Pre-tax Cashflow**																					
	Gross Revenue	$			0	39,849,136	40,394,102	38,973,059	40,925,093	40,362,353	40,626,938	40,719,198	44,177,670	49,911,129	49,655,063	49,454,164	47,851,398	24,970,320	0	547,869,623	0	547,869,623
	Net Revenue	$			0	39,849,136	40,394,102	38,973,059	40,925,093	40,362,353	40,626,938	40,719,198	44,177,670	49,911,129	49,655,063	49,454,164	47,851,398	24,970,320	0	547,869,623	0	547,869,623
	Operating Costs	$			0	30,338,739	29,834,247	31,516,656	33,813,934	27,333,461	24,019,147	20,970,365	21,654,264	23,929,025	23,491,727	23,455,457	21,333,484	10,767,284	0	322,457,792	0	322,457,792
	Capital Costs	$			37,335,899															37,335,899	37,335,899	0
	Sustaining capital costs	$			0	2,449,395	500,000	496,231	0	1,677,645	1,497,762	6,633,240	2,285,085	2,510,237	30,000	0	0	0	0	18,079,593	0	18,079,593
	Mine restoration deposit	$																				
	Working Capital	$			4,000,000	0	0	0	0	0	0	0	0	0	0	0	0	-4,000,000	0	0	4,000,000	-4,000,000
	Total Pre-tax Cash Flow	$			-41,335,899	7,061,002	10,059,854	6,960,172	7,111,160	11,351,247	15,110,029	13,115,593	20,238,321	23,471,867	26,133,335	25,998,707	26,517,914	18,203,036	0	169,996,338	-41,335,899	211,332,238
	Cumulative Pre-Tax Cashflow	$			-41,335,899	-34,274,897	-24,215,043	-17,254,871	-10,143,711	1,207,536	16,317,565	29,433,158	49,671,479	73,143,346	99,276,681	125,275,388	151,793,302	169,996,338	169,996,338			
7.0	**Duties & Taxes**																					
	Mining Tax	$			0	370,167	375,116	360,906	380,426	1,415,566	2,799,864	3,174,327	3,896,550	4,715,520	4,920,466	4,999,245	5,199,686	2,380,000	0	34,987,839	0	34,987,839
	Income Tax	$			0	0	0	0	17,276	1,678,905	2,378,418	2,745,372	3,229,351	3,792,059	3,923,848	4,057,528	4,264,956	2,226,736	0	28,314,450	0	28,314,450
	Total Tax	$			0	370,167	375,116	360,906	397,702	3,094,471	5,178,282	5,919,699	7,125,901	8,507,579	8,844,314	9,056,773	9,464,642	4,606,736	0	63,302,289	0	63,302,289
8.0	**After-tax Cashflow**																					
	After-tax Cashflow	$			-41,335,899	6,690,835	9,684,738	6,599,266	6,713,458	8,256,776	9,931,747	7,195,894	13,112,420	14,964,287	17,289,022	16,941,933	17,053,272	13,596,300	0	106,694,049	-41,335,899	148,029,948
	Cumulative After-Tax Cashflow	$			-41,335,899	-34,645,064	-24,960,326	-18,361,060	-11,647,602	-3,390,826	6,540,921	13,736,816	26,849,235	41,813,523	59,102,544	76,044,478	93,097,750	106,694,049	106,694,049			
9.0	**Economic Indicators**																					
9.1	**Pre-Tax Cash Flow**	$			-41,335,899	7,061,002	10,059,854	6,960,172	7,111,160	11,351,247	15,110,029	13,115,593	20,238,321	23,471,867	26,133,335	25,998,707	26,517,914	18,203,036	0			
	Discounting Rate	$			8%																	
	Discounting Period				0	1	2	3	4	5	6	7	8	9	10	11	12	13	14			
	Discounting Factor				1.00	0.93	0.86	0.79	0.74	0.68	0.63	0.58	0.54	0.50	0.46	0.43	0.40	0.37	0.34			
	Discounted Pre-Tax Cash Flow	$			-41,335,899	6,537,965	8,624,704	5,525,209	5,226,915	7,725,468	9,521,881	7,652,823	10,934,135	11,741,777	12,104,791	11,150,400	10,530,629	6,693,219	0			
	Cumulative Discounted Cash Flow	$			-41,335,899	-34,797,935	-26,173,231	-20,648,022	-15,421,107	-7,695,639	1,826,242	9,479,065	20,413,200	32,154,977	44,259,768	55,410,168	65,940,796	72,634,015	72,634,015			
	Pre-Tax Payback	years		4.9																		
	Pre-Tax Internal Rate of Return	%		26.1%																		
	Discount Rate Sensitivity																					
	Pre-Tax NPV @ 5%	M$	5%	100.4																		
	Pre-Tax NPV @ 8%	M$	8%	72.6																		
9.2	**After-Tax Cash Flow**	$			-41,335,899	6,690,835	9,684,738	6,599,266	6,713,458	8,256,776	9,931,747	7,195,894	13,112,420	14,964,287	17,289,022	16,941,933	17,053,272	13,596,300	0			
	Discounting Rate	$			8%																	
	Discounting Period				0	1	2	3	4	5	6	7	8	9	10	11	12	13	14			
	Discounting Factor				1.00	0.93	0.86	0.79	0.74	0.68	0.63	0.58	0.54	0.50	0.46	0.43	0.40	0.37	0.34			
	Discounted After-Tax Cash Flow	$			-41,335,899	6,195,218	8,303,102	5,238,710	4,934,592	5,619,423	6,258,685	4,198,735	7,084,232	7,485,869	8,008,162	7,266,105	6,772,089	4,999,331	0			
	Cumulative Discounted Cash Flow	$			-41,335,899	-35,140,682	-26,837,580	-21,598,869	-16,664,277	-11,044,854	-4,786,169	-587,434	6,496,799	13,982,668	21,990,830	29,256,935	36,029,024	41,028,355	41,028,355			
	After-Tax Payback Period	years		5.3																		
	After-Tax Internal Rate of Return	%		20.5%																		
	Discount Rate Sensitivity																					
	After-Tax NPV @ 5%	M$	5%	59.8																		
	After-Tax NPV @ 8%	M$	8%	41.0																		
10.0	**EBITDA**																					
	Net Revenue	$			0	39,849,136	40,394,102	38,973,059	40,925,093	40,362,353	40,626,938	40,719,198	44,177,670	49,911,129	49,655,063	49,454,164	47,851,398	24,970,320	0	547,869,623		547,869,623
	Operating Costs	$			0	30,338,739	29,834,247	31,516,656	33,813,934	27,333,461	24,019,147	20,970,365	21,654,264	23,929,025	23,491,727	23,455,457	21,333,484	10,767,284	0	322,457,792		322,457,792
	EBITDA	$			0	9,510,397	10,559,854	7,456,403	7,111,160	13,028,892	16,607,791	19,748,833	22,523,405	25,982,104	26,163,335	25,998,707	26,517,914	14,203,036	0	225,411,831		225,411,831
	Average Annual EBITDA	$																		18,336,218		18,336,218